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A STUDY OF THE IRON AND STEEL INDUSTRY IN LATIN AMERICA

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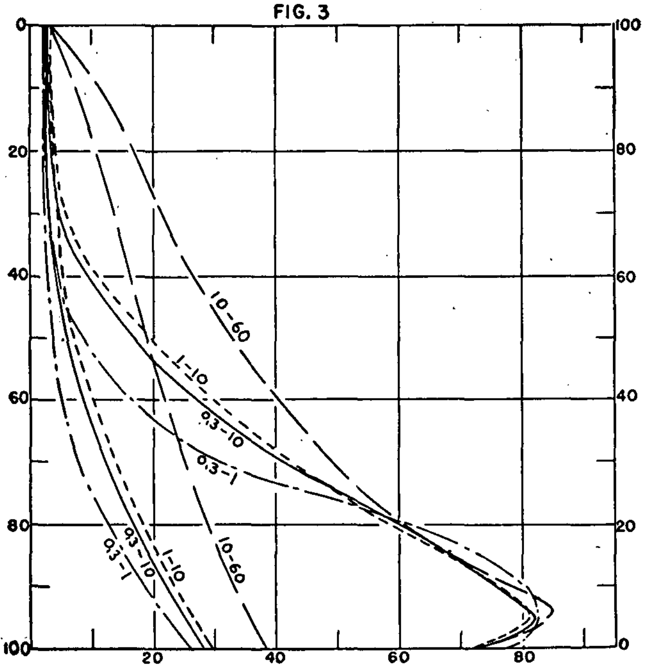
UNITED NATIONS

FIG. 3



up to 12.5), and that the total "small size" should be treated in a "PIC washing tube", and in a centrifuge. The raw 10–60 is treated in magnetite troughs according to a special process with two liquid circuits, since the "drewboy" process proves too expensive in this case owing to the small capacity of the installations.

WASHING IN OTHER WEST-EUROPEAN COUNTRIES

Great Britain. The most frequently used process is the Baum jig, treating the run-of-mine. It has already been explained why this process is suitable only for coals which are easily washable. When the coal is difficult, the British use a heavy-media process (Chance, Barvoys, Ridley, Scholes, etc.) for the minus 10 mm.

Great Britain attaches great importance to the flotation process; whereas in France attempts are made to lower the screen limit of other processes (feldspar-jig 0.5 mm.—cyclone 0 mm.), the British apparently try to raise the upper limit of the flotation process.

British coals are, on an average, very easy to wash since they are very little mixed. When the raw coal is dirty, this is due to stones brought in by mechanization. It is thus understandable that imperfect processes give good results there and that the experts have taken less care to study the imperfection of this or that particular process or installation. On the other hand, the new English plants are of a studied simplicity.

Belgium has the same problem as France, and the same techniques are used to solve them.

CLEANING OF LATIN-AMERICAN COALS

In a great number of cases, the lightest products have a constitutionally high ash content, which sometimes reaches 15%, therefore the hope of obtaining clean washed coals must apparently be abandoned.

Case of the "large size"

As indicated earlier, only the heavy-media processes are of interest, and study can therefore be based on the washability curves.

In Europe, a large part of this goes to domestic use. The most rational treatment in that case consists first in eliminating all the pieces which cannot burn, i.e. all those having an ash content in excess of, say, 25%. Logically, this large size should be treated in a heavy-media machine adjusted to the density of the portions with 25% ashes (in France that density varies from 1.5–1.6; it can be as high as 1.8 in the case of La Mure "per-anthracites". In France, the remainder is treated in a second bath with a density of about 2.0 and the middlings are either crushed (if there is a liberation of coal and shales) or sent to a colliery power station.

The use of a jig or any other imperfect process is not advisable for eliminating these incombustible portions; according to the partition curve, some fragments will always go astray to the prepared coal, and be noticed by the customers.

According to the washability curve of the raw coal, the float made up of fragments below 25% ash will be of a highly variable percentage. In France, it is generally 5 to 8%, for the large coalfields.

Further study is necessary, in order to select one of the following three solutions:

(*a*) Burn the raw coal under suitable boilers;

(*b*) Crush the raw coal and wash it if crushing has freed light fractions;

(*c*) Pulverize the raw coal and send it to a boiler burning pulverized coal.

Case of the "small size"

Washability curves should be drawn in this case in order to select the appropriate process. As an example, these have been established for the 0.5–8 mm. variety of the Brazil Jafet Group (see Pull-out).

The ash content of the raw coal is 28%.

If it is desired to make French-type coking small size at 8% ash content, the theoretical yield is 36%. The operation, in the author's opinion, is therefore not feasible. If a not-so-clean product is acceptable, the yields in weight are the following:

Ash percentage of the washed fines		*12%*	*15%*	*18%*
Imperfection of the fractions...	I = 0......	57	73	83
	I = 0.06....	53	70	82
	I = 0.12....	48	67	80
	I = 0.15....	44	65	79
	I = 0.30....		53	75

Evidently, then:

(*a*) In no case is a rheowasher suitable (I = 0.30);

(*b*) For an 18% product a well-adjusted jig is acceptable;

(*c*) For a 15% product there is not yet much difference between a very good modern jig (I = 0.12) and a heavy-media device;

(*d*) If it is necessary to wash down to 12%, only the heavy-media process will do.

These details are interesting because they show that no over-all answer can be given to the question of washing coal from various sources.

V. SELECTION OF COAL FOR CHAR PRODUCTION

It has been proved that, at least with certain high-volatile coals found in France and in Colorado, the char need not be made from the same coking coal as is later used in the blend for charging into the coke oven. The use of very weakly coking or non-coking coals for this purpose has been demonstrated both to cause a less difficult retort operation and to result in a better physical grade of final coke than when the use of a high-grade coking coal is attempted. Low-temperature retort operating difficulties make the use of coals of the latter type practically impossible.

West, in his patent application, states that the partially devolatilized char is "a fused and gummy mass", and this statement is no exaggeration. In fact, Frederick coal when used alone in a low-temperature retort results in the formation of a mass so "gummy" that it cannot be handled. It therefore becomes necessary to use some means of decreasing the tendency of the coal toward "gumminess" and several expedients have been suggested. Among these are the following:

(*a*) Adding an inert material such as coke breeze to the coal;

(*b*) Re-cycling fine char from the process;

(*c*) Pre-oxidizing or "roasting" the coal before charging;

(*d*) Adding certain proportions of high-oxygen, non-coking coal.

Of these, the last one only was found to give both the desired retort action and quality of char. It appears, therefore, that the selection of the coal for the production of the char, and the blending of coals where necessary for this purpose, is one of the prime investigations to be undertaken.

A coal for char production, then, should meet as many as possible of the following requirements:

(1) It must yield the proper quality of char from the viewpoint of coke improvement;

(2) It should be low in ash, so as not to require washing;

(3) It should be of such a nature as to avoid excessive gumminess during heating or should be susceptible to treatment by blending to minimize this tendency;

(4) It must be low in other objectionable impurities, such as sulfur, phosphorus, chlorides, etc.;

(5) It should be low in moisture content to prevent the retort's capacity being reduced through the need of pre-drying;

(6) It should be locally available so that its delivered cost will be economically sound.

VI. REQUIRED CHARACTERISTICS OF CHAR

The primary function of the char under review is its ability to improve the quality of metallurgical coke. Other uses of char, such as domestic fuels, are important, but for coke production the char must possess certain characteristics which may not be important in other circumstances. Past work has indicated that the following must be considered in order to produce a satisfactory char:

(1) The selection of coals, which has already been discussed;

(2) The amount of char to be used in the final mixture. In general, it has been found that slightly more char is required than low-volatile coal in order to secure comparable results. But, whereas low-volatile coal may be used in as high proportions as the expansion properties of the final mixture will allow, char may only be used in proportions up to approximately 20% of the final mixture. Above that amount the coke appears grainy in structure and loses strength;

(3) The degree of pulverization of the char. Pulverization to all pass ⅛ in. (3.0 mm.) will usually be sufficient although pulverization to pass 20 mesh (0.8 mm.) has been indicated to give somewhat better results. No information is available about the effect of very fine pulverization (100 mesh or 0.15 mm.) or of the presence of substantial amounts of this fine char in the product;

(4) The volatile matter content of the char is critical and the optimum point will usually be found to fall within the limits of 15 to 20%. Variation of more than 1.0% plus or minus the selected point is likely to impair the effectiveness of the char. The same is true of the volatile matter content of the individual pieces of char, and, regardless of their size, they should not vary outside the 2.0% range which has been demonstrated as best;

(5) The physical properties of the char are important. This appears largely a matter of the coal used, of the temperature of production and of the time of retention in the retort. Especially is the surface condition of the char important. Char which has been so produced as to form cenospheres or which has been given a distorted surface condition usually gives a better final coke than a char of unaltered appearance. In general, the use of higher retort temperatures, approaching 750° C, give a better char than the lower limit of 500°.

VII. CONCLUSION

As in all other problems of coke production, there is no general rule of char production and use which can be applied to all base coals. Each such coal, or mixtures, must be made a matter of individual study and a solution worked out for each particular case. Since this paper is concerned with the improvement of coke, no consideration has been given to the by-products of the low-temperature process. These are a matter of vital concern as affecting the economics of char production, and their recovery, use and value must be studied along with the main investigation of the production of char. While char as a blending agent for coke production on a commercial scale has to date been used only to a very limited extent, it does present some very interesting possibilities and appears to be a very worth-while subject for further investigation.

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The Manufacture of Metallurgical Smelting Carbon from Non-Coking Coals

KURT BAUM

Until the present time, the manufacture of metallurgical coke has been limited to certain types of bituminous coals possessing a so-called caking or coking power during their dry distillation, through which the total mass of the original coal particles is transferred by coalescence into a solid porous residue called "coke".

This caking or coking quality of certain bituminous coals has more or less come to be considered the most important characteristic for the valorization of coals in general, whereas formerly the main consideration was the heating value of the coal. This is principally due to the increased demand for coke by the constantly growing iron and steel industries and various others, such as chemical processes, the production of water gas and other applications where a more or less pure carbon in solid form must be applied.

In addition to the fact that such coal deposits are not available in many countries, it should also be considered that those existing are being rapidly diminished, especially in the great steel-producing countries. Thus, the pig-iron industry is now searching for new processes which do not necessitate the utilization of lump size coke for iron ore reduction. This tendency is inspired also by the fact that the creation of new coking facilities requires ever heavier capital investment without assurance of a sufficient profit. The coke problem is of extraordinary importance for the South American countries where, in spite of the presence of huge ore deposits, iron and steel production is still in an inceptive stage.

Many processes have been developed in recent years which do not require the use of lump size metallurgical carbon. Examples are the Krupp Renn, the Stuerzelberg, and the Basset processes, which use reducing and smelting carbon in the form of coke breeze or even low volatile coals. There are also the processes using CO and H_2 for direct gaseous reduction, such as the Wiberg, Brassert and others.

Even the electric pig-iron furnace, where two-thirds of the coke normally used in blast furnace practice are replaced by electric heating, must be considered as being among such processes, since the coke which is used as a reducing agent may be of smaller size and of less strength.

The low shaft furnace (operated with or without oxygen) using briquetted carbon and ore fines is another solution, particularly where fine ores must be used for pig-iron production.

There is no doubt, however, that the standard blast furnace still represents the most economic form of large-scale pig-iron production, especially since the blast furnace gases have found ample application in "integrated" steel works.

The writer has for fifteen years given extensive study to problems connected with pig-iron production and has concentrated on the use of non-coking high-volatile coals, or black or brown lignites as possible raw materials, especially since such fuels may very often be produced at much lower prices than ordinary deep shaft mined bituminous coal.

The only method available in this case for getting a lump size product was briquetting or extrusion of pre-fabricated pieces. For coals still having some, although weak, coking power, a one-step coking process might possibly be applied, as in the National Fuel Process using low- or high-volatile bituminous coals and direct coking of such pitch-bound briquettes.

For lower-grade coals, excellent results have been obtained in Germany by previous cleaning of the fines by the so-called "Convertol" process, followed by briquetting and coking them. With this cleaning process, a certain amount of oil is introduced into the fines. An additional percentage of pitch, recovered from the distillation of a previous batch, may be applied as an added binder and coking agent. In the case of pitch-bound high-volatile coal briquettes, where the distillation and coking processes are combined with smelting and coalescence, the temperature rise has to be kept within the usual limits because either spalling or sticking of the charge during heating will otherwise occur.

Coking of lignite briquettes, for instance, as produced by the high-pressure type of briquetting presses without binder, if the drying and devolatilization steps are passed by careful temperature control, may also result in a lump size coke of relatively good crushing strength; however, as soon as such coke is treated by shatter or drum test, very disappointing results have been observed with regard to hardness and abrasion due to the fact that no smelting and coke formation does actually occur. The form is kept by adhesion only.

The method applied to high-volatile non-coking coals is a three-step process by devolatilizing the original fines at medium temperatures, so that most volatile matter is already extracted, followed by briquetting of the fines with pitch or some percentage of coking coal, and a rapid heating, during which coking occurs, of the briquettes, since at this stage there are no limitations with regard to coking speed.

The original method applied for such a coking process was the use of indirect heated retorts as generally applied in coking practice for both steps of the thermal treatment. In spite of the fact that by gas re-cycling it was possible to obtain an increased capacity when using grain size coal or prefabricated briquettes, a new method had been developed using direct heat transfer by pre-heated re-cycle gases. This process, known as the Baum-Panindoo Process, makes use of the fact that flash heating, i.e. the quickest possible heat transfer, should be applied for the coking step in order to have a most complete cracking and coking effect. The still hot gases enter into the second stage of primary devolatilization, since sufficient potential heat is left at about 700° C. to be used for the devolatilization of the original coal fines. When using lignites or brown coals the so-called "Buettner Devolatilizer", which has operated successfully for oil-shales or oil-chalk and lignites, is one possible solution, within the lower range of temperatures up to 500–550° C. For black lignites or non-coking bituminous coals where higher temperatures have to be applied, the so-called "fluidized devolatilization system" currently being developed in the United States and in France, certainly represents the most promising method. It was, furthermore, most interesting to see that with this method, in spite of the two-step processing, the coking time is much shorter, including both steps of 1–3 hours, depending on the size of the briquettes, which varies from 200 to 900 gm. This is certainly a considerable advantage compared with even most modern coke oven practice, where 16 to 18 hours' coking time is still required.

The physical data of smelting carbon manufactured by this process are about the following:

Lump weight	200–900 gm.
Lump size	230–1,000 cm^3
Bulk weight	35–38 lb./cu. ft.
Porosity	45–40%
Shatter test	83–88

Large-scale application tests were performed with such so-called "formed coke" made, for instance, from Yugoslav lignite in a blast furnace test run as early as 1939. A special cupola smelting coke has also been manufactured from high-volatile non-coking coals and tests were successfully run in 1943–44 in Upper Silesia. Several hundred tons of such fuel were applied in each case.

One of the adverse characteristics of such prefabricated smelting carbon has been its relatively high apparent specific weight and the resulting bulk weight of 35 to 38 lb. per cu. ft. This certainly is at the upper limit of the usual blast furnace coke and blast furnace operators very often prefer to use a more porous coke with a lower bulk weight, even when such cokes have not the same strength.

Extensive research work has been devoted to this problem and it is now possible to identify and compare cokes of various structure by the following characteristics:

$$\text{Total density, D} \quad \frac{\text{pore volume in \%}}{\text{volume of cell-walls in \%}}$$

The pore-size factor, P, determining the pore-size distribution of 100–1,000 microns and above

The cell-wall factor, Z, which demonstrates the per cent of cell-walls 50–500 microns and above

Besides the physical structure, this qualification is also responsible for the strength of the coke and there are indeed means of influencing these physical properties according to the requirements of any blast furnace operation or ore to be smelted. Influence of such properties should not, however, be overestimated.

For instance, charcoal, the genuine smelting carbon for pig iron, shows no visible pores even under the microscope. Nevertheless, charcoal has the lowest apparent specific gravity of all metallurgical carbon materials. The pore-size, as observed in younger coals or lignites, is sub-microscopic, 40–90 milli-microns. This specific property is mainly responsible for the excellent reactivity and the low bulk weight of 10–14 lb. per cubic foot. Certainly, this most important factor, responsible for the excellent and smooth blast furnace operation observed, is due to the large void volume available in the blast furnace charge and the corresponding ample height of the reducing zones in the blast furnace.

This fact has led the writer to the opinion that it is not the physical structure and/or reactivity which is the main problem for the suitability of a smelting carbon, but the bulk weight or the specific volume of one ton of carbon charged with the ore to the furnace. Comparative data for the Fontana blast furnace, where coke with 25 lb. cu. ft. is applied, have, for instance, proved the lowest carbon consumption ever observed in blast furnace practice, overcoming the disadvantage of its macro-porosity (large pores, thin cell-walls) and the resulting fragility and propensity to abrasion. Formed coke, however, has been manufactured, showing 23–28 lb. cu. ft. and a shatter or drum test of 88–92, which is certainly a most ideal smelting carbon.

The purpose of these few examples is to show the possibility of manufacturing smelting carbons with any physical property, once the step has been taken of giving up the gifts which nature has prepared in natural caking coals and of manufacturing metallurgical smelting carbon for its own merits.

Summary of Discussion

Utilization of High-Volatile French and Saar Coals for the Production of Metallurgical Coke, *presented by the author*

Mr. CHERADAME, after summarizing the salient points in his paper, stated that he did not wish to go into more detail, but would make a suggestion for the benefit of his Latin-American colleagues.

As to the minimum quality of coke necessary for the blast furnace, it was quite probable that the figure varied with each country. Another point would be whether the minimum established would be economically feasible. Again, that problem would vary in different countries and each Latin-American country would have to seek solutions to its own particular problems.

Laboratory experiments were invaluable, not only to countries with little experience in coal production, but indeed to those countries with a history in that field behind them. Small ovens were quite suitable for laboratory experiments, but those should be followed by full-size blast-furnace tests.

The most appropriate methods of cutting the coal into various specific-gravity fractions should also be studied.

In conclusion, he again invited Latin-American participants to send samples of their coals to France and the CERCHAR laboratories would undertake studies of those coals. He pointed out, however, the impracticability of transporting the samples for such long distances due to the danger of oxidization. A solution to that problem, of course, would be the sending of Latin-American observers to study the methods and techniques adopted in France.

Mr. LEUSCHNER here intervened to remind the assembly that the United Nations Technical Assistance Administration granted scholarships and fellowships under certain conditions for purposes of study abroad and that those of the Latin-American participants present, interested in training young engineers abroad, might explore such possibilities in connexion with Mr. Cheradame's suggestion.

Mr. ALBALA said that he wished to make a few comments on Mr. Cheradame's paper in respect of the best methods of ascertaining the coking properties of coals. That, of course, was a moot point. In Chile the Gieseler plastometer was used with success and the experimental oven was found to give good results. The usual amount placed in the experimental oven was 250 kg., and it was thus possible to predict the coking properties of the coal.

As to Mr. Cheradame's statement to the effect that French practice called for the crushing of coal to a rather fine size in order to increase the homogeneity of the blend, Mr. Albala stated that experience in his own country had indicated that when the coal was ground too fine, the results were not satisfactory.

With regard to the petrographic constituents of coal, he was of the opinion that it was almost impossible to determine the properties of coal from petrographic studies and he did not believe that coal technology had yet reached the point were petrographic studies were all-conclusive.

Mr. PRIETO agreed entirely with Mr. Albala, stating that the only practical method of determining the carbonizing properties of coal was by oven tests. For the past fifteen years much progress had been made in coal technology, not by the use of one single method but by the correlation of different tests. Furthermore, he believed that small-size oven tests could give a fair correlation of tests with commercial results. His country (Colombia), was primarily interested in commercial results, not in the purely academic point of ascertaining to a degree of precision which was the most accurate method of determining the properties of coal.

Mr. CHERADAME noted that the afternoon meeting of that day would discuss a paper presented by Mr. Powell on the subject under debate. He thought it might be interesting to continue the discussion of correlation of tests at that time.

While he agreed, in general, with the views of the two previous speakers, he would reserve further comment until Mr. Powell had presented his paper.

Mr. BAUM intervened to remark that during a conference in Pittsburgh in 1931, a paper by Lehman-Hoffmann was read, dealing with the separation of the petrographic constituents of bituminous coal. The theory expounded was based on separation by a pneumatic stroke. Mr. Baum made the observation that the petrographic and chemical experts participating in that meeting seemed to have discussed the matter from their respective points of view and consequently they did not reach an agreement. In later years, the hopes originally raised by the aforementioned paper had vanished, and it was surprising that after so many years it should come up again in Mr. Vogel's paper scheduled for the afternoon.

In his opinion, most of the younger and weakly coking coals could be advantageously subjected to petrographic studies in view of the advanced systems of separating the petrographic constituents that had been developed.

Mr. LEUSCHNER observed that it was rather unfortunate that Mr. Chapman of the United Kingdom could not attend the meeting, for he was sure that Mr. Chapman would have contributed substantially to the debate by presenting the United Kingdom view, which might have represented a third possible solution.

He added that the four original petrographic constituents could now be divided into ten or twelve groups, and that the subject could not be debated, except in an international conference devoted entirely to that subject. He thought that Mr. Cheradame had provoked a very challenging discussion.

Mr. LOPEZ stated that he had gathered from Mr. Cheradame's paper that there were basic differences in the methods used to determine the physical properties of coal, and he wondered whether the various methods could not be made uniform so that more precise comparisons and more definite conclusions might be drawn.

Mr. LEUSCHNER informed the assembly that the Economic Commission for Europe (ECE) had a special coal research committee which for the past two years had been engaged in the standardization and correlation of measurements and indices applicable to the United States and to the European countries.

In any event, Mr. Waring of the ECE would join the meeting at a later date and he urged the Latin-American participants who were interested in the subject to consult Mr. Waring regarding any questions they might wish to raise.

Mr. CHERADAME observed that the ECE had conducted many experiments and had sought the collaboration of laboratories in several countries with the hope of comparing the results of tests with similar types of coal undertaken in the respective countries. Certain French laboratories had also conducted studies and had compared their results with those of other countries.

The Blending of Western Coals for the Production of Metallurgical Coke, *presented by the author*

Mr. CHERADAME opened the discussion of the paper by observing that in the experiments conducted at Marienau, it was found that the fineness of the powdered coke was of great importance and that sizes below 0.5 mm. were never used.

Low-temperature Char as a Substitute for Low-Volatile Coal in the Production of Metallurgical Coke, *presented by the author*

Mr. POWELL, in opening the discussion of the paper, emphasized the importance of the operations carried out in the western part of the United States, as described by Mr. Price in the two papers he had just presented. He wished to stress the point that not all United States coals were good coking coals. Mr. Price happened to be a resident of the western United States and was, by virtue of his experience in that part of the country, an authority on the treatment of coals found in that area.

Mr. CHERADAME expressed great interest in the papers presented by Mr. Price, who, he said, had specifically discussed the different processes that had been used in his country to avoid the agglutination of char, and had indicated his preference among the four processes presented. He pointed out that in one of the mines in the north of France, the pre-oxiding process had been used with excellent results.

The previous speaker had also stated that the coal selected for char production should meet certain requirements. In his own experience, he had discovered that char could be made from non-coking, high-volatile coal. In Lorraine, for instance, it had been found that the type of coal chosen for the production of char did not represent too serious a problem. Mr. Palmé, one of the experts from Marienau, was of the opinion that in order to produce the best coke it was necessary, when the blend was heated, that the semi-coke should have the same degree of contraction as the other parts of the blend.

He entirely agreed with Mr. Price on his discussion of the fineness of char. In Marienau, experience had proved that the fineness of the char was not important and that it was not essential to have finely ground char.

It had been mentioned in the paper under review that the most desirable volatile content of the char was between 15 and 20%. He also held that view. As to Mr. Price's remarks on the importance of the physical properties of the char and on the theory that the formation of cenospheres yielded a better coke, he wished to reserve judgment. There seemed to be a contradiction in terms in the author's views, and he wondered how Mr. Price was able to reconcile the two conflicting theories presented.

Mr. PRICE remarked that it seemed that Mr. Cheradame had assumed from the fact that two types of retort were listed that those represented the preference of the industry in the western United States. He explained that the Hayes had been installed in 1940 and, because of the low cost involved, it would have been unwise to turn it down. He added that his statements on pre-oxidization had been based on experience.

He then went on to express a personal opinion which might be subject to correction. He thought that the reason that high-volatile coal by itself did not yield a good coke was because the contraction of that coal caused the coke to break after production. High-volatile coal needed the addition of an element to counteract contraction. A non-coking coal usually produced neutral char. In the United States, a contracting char was not desirable. An expanding or neutral char suited their purposes, hence their preference in that regard.

In his own experience, the char produced at 750° C. was far superior to that produced at 500° C., although coal of the same volatile content was used in both cases. Microscopic examination revealed a difference in the physical aspects of the char produced and therefore it was concluded that the difference was due to the variation in the temperature. With the proper type of coke, it was quite possible that operation at a lower temperature would be successful.

The Manufacture of Metallurgical Smelting Carbon from Non-coking Coals, *presented by the author*

Mr. ALLARD commenced the general discussion of the paper by stating that he also held the view that reactivity was not the prime characteristic in respect of suitability of smelting carbon. He wondered whether the height of the blast furnace was considered important in Mr. Baum's experiments. The desire to obtain a hard coke was a universal one. However, it would perhaps be difficult to compromise between the two methods used, one utilizing a low furnace and the other involving the smelting of carbon with the hardness of good and reactivity of bad coke. He ventured the opinion that it was not as important to obtain a very good coke for the low-shaft furnace but a coke good enough to be utilized in a high furnace.

Mr. BAUM replied that, as he had already mentioned, large quantities of coke were necessary in order to conduct blast furnace experiments. Lignite char could be used instead of coke in small ovens of 80- to 85-ton capacity. However, for the cupola type furnace, the coke had to be extremely hard. As to whether such coke could be used in large furnaces, he did not believe that any problem would arise in that regard. In his own experience with the blast furnace operations, no change was effected in the crushing strength of the coke while it was progressing down the shaft. The only effect seemed to be a difference in the amount of abrasion.

FUELS

Production of coke from poorly coking coals and substitute fuels (*continued*)

16 October 1952—Afternoon

Chairman:

Joaquin PRIETO, Assistant Manager, Empresa Siderúrgica Nacional de Paz de Río, S.A., Bogotá, Colombia

Later:

Argenis GAMBOA, Ministerio de Minas e Hidrocarburos, Caracas, Venezuela

Contributed Papers:

Improving Coking Qualities of Coal by Addition of Various Materials
Alfred R. POWELL, Associate Manager, Research Department, Koppers Company, Inc., Pittsburgh, Pa., U.S.A.
Correlation of Small-Scale Carbonizing Tests with Commercial Coke-Oven Results
Alfred R. POWELL, Associate Manager, Research Department, Koppers Company, Inc., Pittsburgh, Pa., U.S.A.
Argentine Asphaltites as Blending Material for Poorly Coking Coals
Alfred R. POWELL, Associate Manager, Research Department, Koppers Company, Inc., Pittsburgh, Pa., U.S.A.
The Utilization of Petroleum Pitches and Asphalts for the Production of Metallurgical Coke
M. D. CURRAN, President, Curran Carbonizing and Engineering Company, St. Louis, Missouri, U.S.A.

Summary of Discussion:

Participants: Messrs. POWELL, CHERADAME, BAUM, ALBALA, SCHERESCHEWSKY, PRICE, STURGEON, LEUSCHNER

Improving Coking Qualities of Coal by Addition of Various Materials

ALFRED R. POWELL

METHODS FOR IMPROVING COKING QUALITIES

A number of methods are available for improving the physical quality of coke made from poorly coking coals. Some of these have been in large-scale usage for many years, while others are now in various stages of development. The more important methods will be reviewed here, some of them very briefly, but those involving the blending of materials such as coke breeze, anthracite fines, low-temperature char, pitch, and other non-coal materials with coking coals will be reviewed somewhat more in detail, since these were the methods requested as the subject matter of the present paper.

The blending of two or more different coking coals to obtain improved coke is, of course, a most common procedure in nearly every part of the world and is almost universal practice in the United States. Of considerable interest in connexion with the improvement of coke from high-volatile B coal was the practice in one of the former plants of our company of blending not only low-volatile coal but also some high-volatile A coal, the latter being added to increase the fluidity, since both of the other component coals had very low fluidities, as illustrated in Figure 1. The resulting coke was much stronger and had a greatly improved structure as compared to the coke made from the B coal and low-volatile alone (1).

Proper pulverization of coal is sometimes an important factor. Large pieces of ash or other non-coking constituents in the coal often lead to cracks or cleavage planes in the finished coke, and fine pulverization prevents this. When blending is practised, intimate

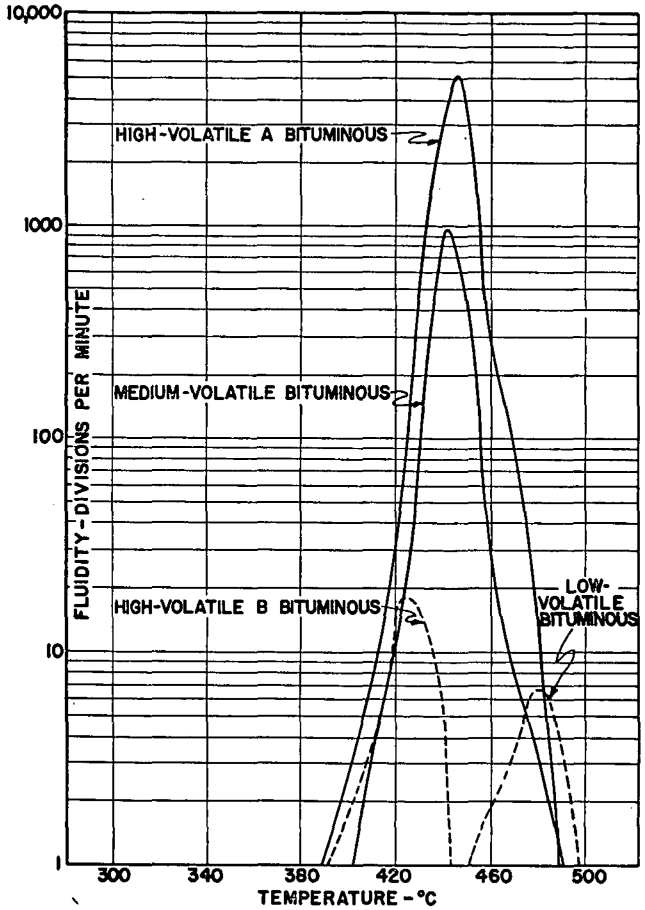


FIG. I
GIESELER FLUIDITY OF FOUR TYPICAL COALS.

contact between the various components is essential to secure maximum benefit in coke quality, and sufficient pulverization, preferably after the components are mixed, assists in securing this intimate contact. Interesting and important developments in improvement of coke by processes involving not only special pulverization but also separation and re-blending of petrographic constituents have been made in France and are described in another paper presented at this conference.

Because of the low fluidity of weakly coking coals while in the plastic state, some European coke plants have improved coke quality by increasing the bulk density of the coal before it enters the oven. This is done by "stamping" the coal into a large cake which has the same shape and almost the same size as the coke-oven chamber and this stamped cake is then slid into the oven for carbonization. The coal particles are pressed together by this procedure, which probably compensates to some extent for lack of fluidity, since somewhat improved coke structure is often obtained.

It is a well-established fact that rapid rates of heating have a marked effect on improving the quality of coke from low-rank, weakly coking coals. Many years ago our company designed and constructed coke ovens for the carbonization of the poorly coking Utah coal mentioned previously, securing rapid heating by having the oven width somewhat less than normal and the wall temperatures somewhat higher than normal. Without these special provisions for rapid heat input at relatively high temperatures, it would have been impossible to manufacture coke at all from this coal.

BLENDING WITH COKE BREEZE

For many years it has been common practice for some coke plants in the United States to mix small percentages of fine coke breeze with the coal. In all cases the coals have had strongly coking characteristics and the usual objective has been to make extra large size coke for use in foundries. Extensive and systematic plant-scale tests made several years ago (2) showed that fine sized coke breeze (0.4 mm.) added as 5% of the oven charge substantially increased the coke size, improved the shatter test index, but had little effect on the tumbler stability factor. In contrast, coarse coke breeze (6 mm.) decreased the coke size and the shatter index and decreased the stability factor so decidedly that, in this respect, the coke was similar to coke made from poorly coking coal.

As shown later in tests made by our company, coke breeze by itself is probably of little interest in connexion with the carbonization of weakly coking coals. It has the effect of actually decreasing the coherence of the coal during the critical plastic period. With the highly fluid high-volatile A coals this is of minor importance, but with the very slightly fluid B coals, it can easily destroy whatever small coking power the coal has. The beneficial effect of the fine coke breeze appears to lie in its ability to counteract shrinkage during the latter stage of carbonization, thereby making the pieces of coke larger. As shown later, blending of fine coke breeze with poorly coking coals can be of considerable benefit provided another material such as pitch is also added to increase fluidity of the mixture during the critical plastic stage.

BLENDING WITH ANTHRACITE FINES

Considerable quantities (about 250,000 tons annually) of fine anthracite are blended with strongly coking coals in the United States but, like coke breeze, its use is confided to a relatively few plants, mostly for production of foundry coke (3). The possibilities of blending this material with poorly coking coals are about the same as coke breeze, as just described.

BLENDING WITH LOW-TEMPERATURE CHAR

The effects of low-temperature char, when properly prepared, are quite different from the effects of coke breeze or anthracite fines. This type of blend has been investigated very thoroughly in the United States and is now being seriously considered as a means of improving the physical qualities of coke made from western coals from Colorado and Utah. As mentioned before, weakly coking Utah coal, unblended with other materials, has produced coke that has given very good results in moderate sized blast furnaces, but the larger furnaces of 1,000 tons or more capacity require coke having somewhat better physical qualities.

Since the present paper attempts to explain the quality of coke as obtained from various kinds of coals and blends by means of their coking behaviour during the two critical stages of carbonization, it will be pertinent to theorize on the effect of low-temperature char on the coking behaviour of poorly coking coals. When about 15% of this char, properly prepared, is blended with the coal, the resulting coke is substantially larger and stronger. Since the char does not become plastic or liquid when heated, it could not increase the fluidity of the coal in the plastic zone of the charge. One possible explanation is that the char serves as a "connector" between the slightly plastic particles of coal which by themselves do not liquefy sufficiently to run together and cohere as would a strongly coking coal. If this explanation is correct, it is probable that the surface characteristics of the char permit it to be "wetted" by the plastic coal, thereby securing adhesion, whereas other materials, such as coke breeze, would not have this essential surface characteristic. In addition to this possible cohesion property of the char, it is probable that it also benefits the coke by decreasing shrinkage during the latter stage of carbonization, thus resembling coke breeze in this respect. Further investigations of the fundamentals of coking behaviour in coal-char blends would give a better foundation for explanations of this interesting and important procedure for improving the coke from weakly coking coals.

BLENDING WITH PITCH

The mixing of pitch with coals of the high-volatile B type, either alone or along with other blending materials, is of much interest, since pitch improves coke quality by providing greater fluidity in the plastic zone, and (as explained earlier) lack of fluidity is the major deficiency of these low-rank, high-oxygen coals. Additions of small percentages of coal-tar pitch, along with some low-volatile coal, to poorly coking Utah coals has been successfully practised for several years in some coke plants of the western United States (4).

Thompson has described some systematic tests of the effect of pitch on coke quality made in a test oven, using Utah high-volatile B coal as the chief constituent of the blend. The following table summarizes the results.

Composition of blend		*Properties of coke*		
Utah coal, per cent	*Pitch, per cent*	*Size, per cent on 2 in.*	*Shatter index*	*Tumbler stability*
100	0	63.0	13.0	11.0
97½	2½	63.8	16.9	15.1
95	5	69.0	22.8	18.8
90	10	71.2	34.0	23.8
85	15	75.0	39.1	23.0
80	20	63.3	33.5	19.3

With increasing amounts of pitch, the coke size and strength increased up to about 15% pitch content, beyond which there appeared to be a drop. Unfortunately, the amount of pitch available from this type of coal is less than 2% of the coal charged, so that improvement in coke quality would not be great unless larger quantities of pitch were brought into the plant from outside.

In our own investigations for improving the coking quality of weakly coking coals from various parts of the world, we have used coal-tar pitch as a blending component, either alone or along with other blending materials, and the results of some of these tests are given later in this paper.

OTHER BLENDING MATERIALS

In several American coke plants the blending of small quantities of oil with coal has been carried on either to increase the yields of gas and other products or to increase the bulk density of the coal charge. The amount added has varied from about 0.25% up to about 3.5% by weight. In all cases the coal had strong coking properties and no effect on coking properties of the coal was noted, nor was any effect expected by the addition of these small quantities of petroleum oils.

SOME APPLICATIONS OF BLENDING TECHNIQUE TO POORLY COKING COALS

To avoid any misunderstanding it should be pointed out that coke ovens that have been designed to give the maximum coke quality from low-rank, weakly coking coals, also represent the proper type of oven to coke blends of these coals with minor percentages of the various blending materials, since the coal is still the major component and rapid carbonization still brings out the best coking power in this component.

It should also be pointed out that the coking behaviour of a blend and the quality of coke to be expected can quite definitely be predicted in those cases where actual plant operating results with 100% charging of the weakly coking coal are available. This can be accomplished by using properly designed and properly operated small-scale test carbonizing equipment and then coking in this equipment under identical conditions (1) 100% coal, and (2) the blend under consideration. The differences in coke quality between these two tests will indicate to a fair degree of accuracy the differences to be expected in commercial operation between the known large-scale operation with 100% coal and the unknown operation with the blend.

The economics of carbonizing blends in a coke plant as compared with carbonization of the weakly coking coal by itself must be worked out separately for each specific plant. The chief item entering into the differences in cost between the two kinds of operations is the cost of the blending material as produced or delivered at each plant and this may, of course, vary considerably from one location to another. The only other item of importance is the difference in yields of coke, gas, and coal chemicals, but this is usually of rather minor importance since the coal is still by far the major component of the oven charge. The coke plant investment costs and operating costs and credits, other than the two just mentioned, are not appreciably affected by use of the blends that comprise the subject matter of this paper.

TESTS OF A FRENCH COAL

Large samples of what was known to be a weakly coking coal were sent to our research laboratories in sealed containers from France. Prior to larger scale experimentation with this coal, a series of experiments was made in coking test boxes to determine approximately the effects of various factors on the quality of coke produced. As explained in another paper at this conference by the present author, the usual box coking test is made in a steel box containing about 60 lb. (27 kg.) of the test coal or blend which is then coked while buried in a regular charge of coal in a commercial oven. The results of these preliminary tests showed:

(1) Stamping the coal in the box so as to obtain high bulk density gave practically no improvement in coke quality;

(2) Blending the coal with 20% low-volatile coal showed very considerable improvement in coke quality, the pieces of coke being noticeably larger and stronger;

(3) Addition of 4% of coal-tar pitch gave the coke a better appearing structure, but did not reduce the fracturing of the coke appreciably;

(4) Blending the coal with 15% of low-temperature char gave much better coke than that made from the straight French coal, but, judged by appearance, not quite so good as the coke from the blend with low-volatile coal;

(5) The addition of 4% of pitch to the French coal-and-char blend gave a slight improvement in coke quality.

As a result of these observations of the preliminary tests, further experimentation was carried out on a scale large enough to permit standard physical tests to be made of the resulting coke.

The blending components used in these larger-scale tests were (1) the French coal, (2) a typical American low-volatile coal, (3) low-temperature char made by the Disco process, (4) a low-temperature char made in France, and (5) coal-tar pitch.

The proximate analyses of the various materials are shown in the table below:

	Per cent			
	Volatile matter	*Fixed carbon*	*Ash*	*Sulfur*
French coal......	38.8	56.3	4.9	0.82
Low-volatile coal..	17.6	76.7	5.7	0.78
Disco char.......	19.6	69.4	11.0	2.10
French char......	15.8	70.8	13.4	0.58

The plastic properties of the two coals, as determined by the Gieseler plastometer (3) were:

	Softening temp., °C	*Maximum fluidity, divisions per min.*	*Temp. of maximum fluidity, °C*	*Solidification temp., °C*
French coal......	330	12.0	429	454
Low-volatile coal..	390	38.0	465	500

The coal-tar pitch used in the test blends was the grade known as "fuel pitch" which has a melting point of 150°C.

In order to secure uniform blending of the various components the coal and other materials were pulverized to such an extent that approximately 90% was less than 2 mm. in size.

When the plastic properties of these two coals are compared with those shown for various ranks of coal in Figure 1, it will be noted that the French coal is similar to the high-volatile B bituminous coal shown there, but that the low-volatile coal becomes somewhat more liquid than the typical low-volatile coal shown in the figure. In this connexion it should be mentioned that low-volatile coals showing higher fluidity, such as this, give better blending results with low-rank coals than do the more viscous low-volatile coals, due to the relative lack of fluidity of coals such as this one from France.

In order to obtain definite and more accurate information on the properties of the cokes resulting from different blending procedures, the following six tests were made:

Test No.	*Blend*	*Test apparatus*
1.	100% French coal......................	Movable-wall oven
2.	80% French coal—20% low-volatile coal.......	200 lb. box test
3.	85% French coal—15% Disco char..........	" " "
4.	80% French coal—16% Disco char—4% pitch.	" " "
5.	85% French coal—15% French char.........	" " "
6.	80% French coal—16% French char—4% pitch	" " "

For purposes of comparison it is unfortunate that the same test apparatus was not used in all cases, but nevertheless it is believed that the results on coke quality from the different tests will indicate the trends to be expected from the blending of the above components with the low-rank coal. Both the movable-wall oven and the 200 lb. (90 kg.) box test are referred to by the author in another paper at this conference.

In test No. 1 listed above, the coal was coked for about 7 hours, 45 minutes in the movable-wall oven and the average flue temperature of the oven was about 1,350°C. Bulk density of the coal was 48.1 lb. per cubic foot (770 kg. per cu.m.), and the maximum pressure developed was 0.97 lb. per square inch (0.068 kg. per cm.^{2}), which indicates that the coal is perfectly safe from the standpoint of carbonization pressure. In the box tests, the exact time of coking is not known, since the box is considerably narrower than the commercial oven in which it is placed. Average flue temperatures of the oven were about 1,325°C., and the bulk density of the coal in the boxes was about 50 lb. per cubic foot (800 kg. per cu.m.). Carbonization conditions were therefore not too far different in the two test procedures, except that the coal in the test box was under considerably more pressure from the weight of the coal charge in the oven above it.

The following table lists the more important properties of the cokes from each test charge:

Test No.	*Composition*	*Size, per cent on 2 in.*	*Shatter index*	*Stability factor*
1.	100% French coal..................	59.6	26.0	12.5
2.	80% French coal—20% low-volatile coal............................	86.2	55.8	42.6
3.	85% French coal—15% Disco char...	76.8	63.0	32.7
4.	80% French coal—16% Disco char—5% pitch.......................	75.0	57.0	35.7
5.	85% French coal—15% French char..	81.2	63.6	28.4
6.	80% French coal—16% French char—4% pitch.......................	77.7	57.0	31.1

As pointed out in another paper by the author at this conference, the physical properties of coke from the movable-wall oven are inclined to be slightly lower than those of the corresponding coke from a commercial coke oven. Even if this is compensated for by adding a few per cent to each of the properties of coke from Test No. 1 in the above table, the results from the other five tests show definitely that all of the blends have decidedly better coking properties than the French coal by itself.

Many coke technologists in the United States consider the stability factor from the tumbler tests as the most reliable single means of measuring the physical value of coke for blast-furnace use. On this basis, the blend with low-volatile coal stands out ahead of all others, with the Disco char-pitch blend second, and the plain Disco char blend third. On the other hand, if the shatter test is considered the measurement of physical coke quality, the two blends of coal with low-temperature chars stand out as the best. The general conclusion from this series of tests is that all of the blends showed great improvement over the French coal by itself, with the blend with low-volatile coal probably slightly in the lead. Addition of pitch did not appear to be especially beneficial in this case.

Emphasis should be put on the opinion that probably the best type of char was not used in this series of tests. As judged by volatile matter content, which was 19.6% for the Disco and 15.8% for the French, these two low-temperature chars were quite different, since it is usually considered to be important to maintain volatile matter content between very narrow limits. It is quite possible that another char, possibly with a volatile content somewhere between that of the Disco char and that of the French char, would have given still better results than were obtained in these tests, but lack of sufficient coal prevented continuation of experimentation in this case. With development of a better type of char, it is quite conceivable that even better coke could result from char blending than from low-volatile coal blending with this French coal.

TESTS OF AN ITALIAN COAL

A large sample of low-rank coal from Italy was shipped to our research laboratories for the purpose of determining whether acceptable coke could be made from it. Prior to larger-scale testing, some preliminary 60 lb. (27 kg.) box tests were made, in order to study the coking behaviour of this coal under a variety of conditions:

(1) The coal as received was only weakly coking and was not suitable for coke manufacture as such;
(2) Stamping of the coal to high bulk density improved the coke only slightly;
(3) Addition of 5% pitch to the coal, in its condition as received, resulted in slight improvement;
(4) Since relatively large pieces of non-coking material in the coke appeared to be partially accountable for poor structure, the coal was pulverized, reducing the percentage that was more than 3.4 mm. in size from 50% down to 5%. As a result, the coke quality was materially improved.

As the result of these preliminary observations, it was decided to make two large-scale coking tests in the movable-wall oven. One test was to be made with 100% of the Italian coal, finely pulverized as indicated above. The other test was to be with a blend of 95% Italian coal, finely pulverized, and 5% of coal-tar pitch, similar in quality to that used in the tests with the French coal.

The Italian coal used in this series of tests had the following proximate analysis:

Volatile matter	*Fixed carbon*	*Ash*	*Sulfur*
	Per cent		
45.7	45.1	9.2	8.35

The extremely high sulfur content of this coal makes it rather exceptional, especially in combination with the moderate content of ash.

The plasticity characteristics of the coal, as determined by the Gieseler plastometer, were as follows:

Initial softening temperature, °C.	330
Temperature of maximum fluidity, °C.	420
Solidification temperature, °C.	454
Maximum fluidity, divisions per minute	2.2

This illustrates very clearly the chief reason for poor coking power. This coal has very slight fluidity while in the plastic condition, considerably less even than the French coal previously described.

In each of the carbonizing tests in the movable-wall oven, the bulk density of the charge was about 50 lb. per cubic foot (800 kg. per cu.m.), the average temperature in the heating flues about 1,350°C., and the coking time about 7 hours 45 minutes.

The more important of the physical properties of the two cokes from these tests were:

Test No.	*Composition*	*Size, per cent on 2 in.*	*Shatter index*	*Stability factor*
1.	100% Italian coal	56.5	29.0	10.6
2.	95% Italian coal—5% pitch	68.5	38.0	15.5

Although the addition of 5% of coal-tar pitch to this weakly coking coal led to an appreciable improvement in the quality of coke, comparisons with the previously described experiments with the French coal would indicate that addition of a small percentage of pitch alone is, in general, not as effective as blending the coal with low-volatile coal or low-temperature char.

It should be pointed out that the above results compare quite well with those of Thompson referred to earlier in this paper. In those tests Thompson used a Utah coal, the coking properties of which were similar in many respects to the Italian coal.

TESTS OF A UTAH COAL BLENDED WITH BREEZE AND PITCH

As explained earlier in this paper, fine coke breeze blended with good coking coals makes the resulting coke larger and somewhat more resistant to shatter. With weakly coking coals, however, lack of sufficient fluidity during the plastic stage of carbonization makes this procedure ineffective. Since addition of pitch has the effect of increasing the fluidity of the plastic coal somewhat, it was decided to make a few small-scale tests to determine whether Utah coal, which is weakly coking, could be made to produce larger, improved coke by the addition of both fine coke breeze and pitch.

This series of tests was made on quite a small scale and did not include actual determination of the comparative physical properties of the resulting cokes. However, visual examination definitely established the following conclusions:

(1) Addition of 10% fine breeze (0.8 mm.) to the coal (75% through 3 mm.) almost completely destroyed whatever small coking power the coal had;
(2) The coal was then finely pulverized (through 0.8 mm.) and the breeze further pulverized (through 0.25 mm.). When 90% of this coal and 10% of the breeze were carbonized, a fair coke resulted;
(3) Since fine pulverization of the coal and very fine pulverization of the coke breeze made such an improvement in coking power, a blend of 90% coal (0.8 mm.), 5% coke breeze (0.25 mm.), and 5% pitch (0.8 mm.) was carbonized. This made a good coke, comparatively large in size and with little visible fracture;
(4) Since coke breeze as fine as 0.25 mm. is probably not economically practicable because of the high cost of pulverizing this very abrasive material, further tests on a somewhat larger scale were made of blends containing coke breeze sized to pass a 0.8 mm. screen. In a 60 lb. (27 kg.) box tests and a small oven test, a blend of 90% coal (90% through 3 mm.), 5% breeze (0.8 mm.), and 5% pitch (0.8 mm.), as well as another blend of 85% coal, 7½% breeze, and 7½% pitch, all sized to pass a 0.8 mm. screen, were carbonized. In each case, considerable improvement in coke quality, as compared to that made from 100% coal, was observed. This improvement was not quite so good as that observed when very fine 60-mesh breeze was used.

In conclusion, this series of tests illustrated very clearly the importance of fine pulverization for blends involving weakly coking coals and also the necessity of adding pitch in addition to fine coke breeze when this method of making larger coke from a weakly coking coal is used.

c_8	$d_8 = bxc_8$
otal:	Total:
=	$f_8 =$
$= \frac{e_8}{20}$	$T_8 = \frac{f_8}{e_8}$

D

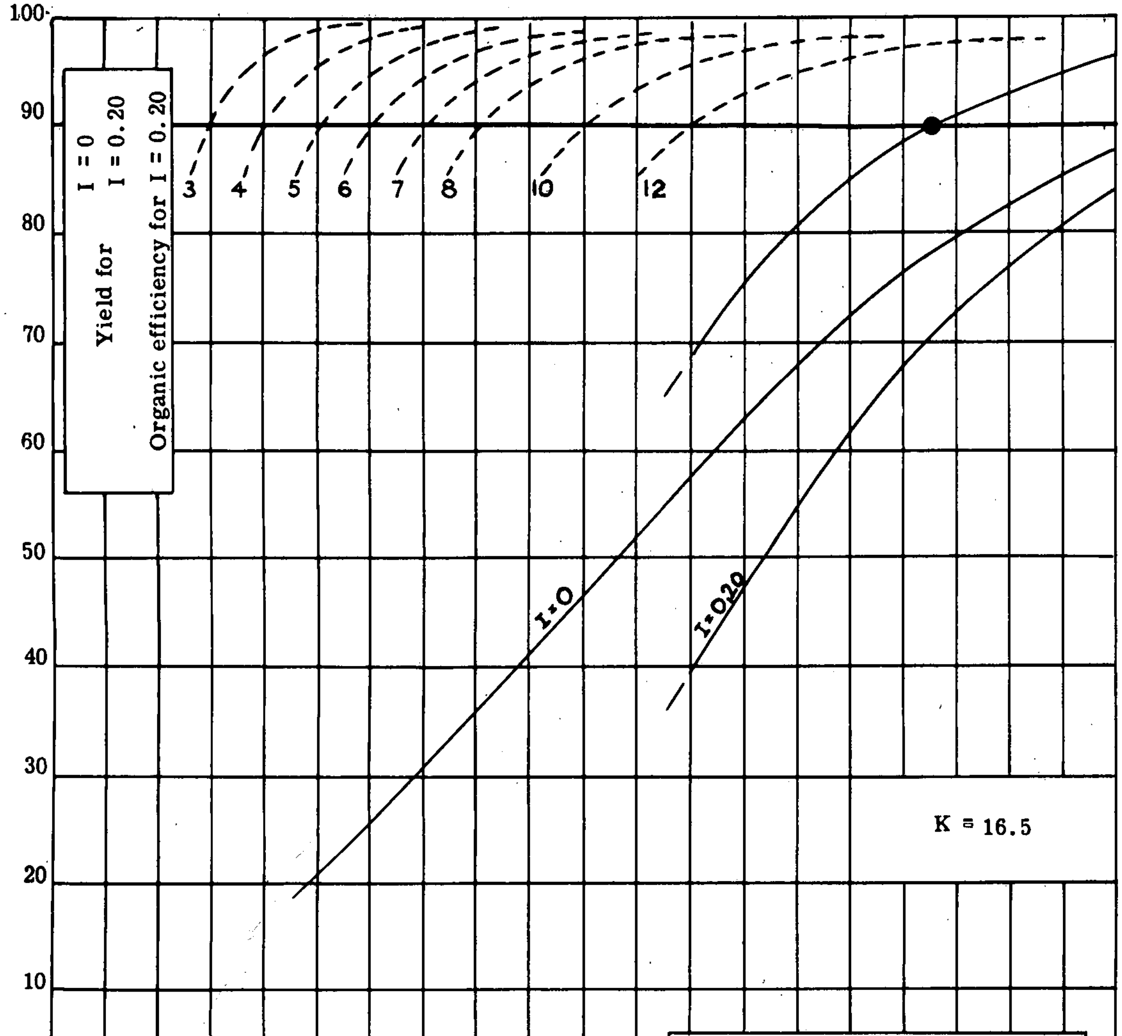
CURVES GIVING ORGANIC YIELD AGAINST ASH CONTENT OF CLEAN COAL

-- Yield (in weight) for I = 0

-- Yield (in weight) for I = 0.20

-- Organic efficiency of I = 0.20

Determination of K (washability number)



THEORETICAL WASHAI LITY FOR I = 0.20

Value of k, Washability N ber

Coal investigated: Brazil J ot Group, 0.5-8 mm.

Sample 206

HOW TO USE THE FORM

Plot on page A the washal ty curves of raw coal against ash content.

Plot on page B the washal ty curve of raw coal against specific gravity (it is suggest to use liquors with a rather high specific gravity, namely, 2–2.4 and even 2.6).

Plot on page B the relatio hip between ash content and specific gravity.

We find on page B six d-l normal partition curves for I = 0.20. For each curve, v shall calculate the yield (in weight) and the ash content f clean coal.

For that purpose, raw coa split into 20 fractions, equal in weight. The relationship sh content-density gives the average ash content for eac fraction.

Write on page C, column ash content for each fraction.

1. CALCULATION OF YIE) (IN WEIGHT)

Draw the verticals by the ints of the washability curve against specific gravity hav as ordinates 2.5–7.5–12.5, etc. (For low specific graviti several verticals might merge together).

Write in column c for t investigated partition curve, the ordinate of the intersect point of those verticals with that partition curve. Add th e ordinates: yield R is equal to $\frac{1}{20}$ of that total.

2. CALCULATION OF AS CONTENT

Write in column d the v e of product b.c, which gives the weight of ash in the cle coal obtained from the considered fraction.

Add all figures obtained hat gives 20 times the value Σbc of ash contained in cle coal.

Ash content of clean coa therefore $\frac{\Sigma bc}{20\,R}$.

Note: With the form, it possible to make calculations for 2 additional partition urves, differing from those printed, either by the par on density or by the shape (e.g., normal curve in d l normal or any other shape found experimentally).

3. VALUE OF WASHABI TY NUMBER k

On page D, K is defined the ash content of clean coal obtainable with an organic fficiency of 90% in a device having d-l log normal parti n curves giving I = 0.20.

That number is obtaine directly on the curve giving organic yield against ash co ent. If this curve has a normal position within the printe etwork which corresponds to usual coal qualities, that nu er is somewhat reliable. If, on

CONCLUSIONS

From studies made by the Research Department of Koppers Company and by other investigators, and confirmed by commercial or reliable small-scale carbonization tests, improvements in coke quality by blending various materials with weakly coking coals, such as occur in South America, may be summarized as follows:

(1) The poor physical properties, and especially the small size and lack of desired strength, of coke from low-rank, weakly coking coals are due to (a) low fluidity of the coal while in the plastic stage of carbonization leading to poor coherence of the particles, and (b) excessive shrinkage during the latter stage of carbonization leading to breakage of the coke mass into small pieces;

(2) In the western United States poorly coking coals have been used for many years to make blast-furnace coke that has given good results. In this case, the coke ovens were designed for use of such coals, the blast furnace was of moderate size, and the iron ore was sintered to compensate for the small coke size;

(3) Procedures to improve the coke quality from weakly coking coals by blending with various materials require that the coal be pulverized to a reasonably fine size;

(4) Rapid rates of heating for carbonization of poorly coking coals are necessary, and the same oven design and method of operation to secure this effect are also desirable for blends of those same coals with other materials;

(5) Blending poorly coking coals with fine coke breeze is not beneficial unless pitch is also added to increase fluidity during the plastic stage;

(6) Low-temperature char has a decidedly different blending effect than does coke breeze. The char appears to increase coherence in the plastic stage without increasing the fluidity, and, in addition, tends to counteract shrinkage during the latter stages of carbonization. Blending of a low-rank coal with about 15% of properly prepared low-temperature char leads to very material improvement in the size and strength of the coke. The degree of this improvement is about the same as secured by the addition of certain low-volatile coals. Where these low-volatile coals are not economically available, blending with low-temperature char appears to be one of the best procedures now available for improving the quality of coke from low-rank, weakly coking coals;

(7) Coal-tar pitch improves the coke quality somewhat but, because of limited availability, cannot usually be added to the coal in amounts exceeding 5% at the most. It is usually used in combination with other blending materials, such as low-volatile coal, low-temperature char, etc.;

(8) The economics of blending must be determined separately for each plant. The chief cost item is the cost of the blending material as delivered or produced at the coke plant. The only other item that noticeably affects coke-plant costs is the difference in yields of the various carbonization products.

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Correlation of Small-Scale Carbonizing Tests with Commercial Coke-Oven Results

ALFRED R. POWELL

Since carbonization tests are preferably made in full-size coke ovens, our company has made a practice of carbonizing test coals or blends as a single charge in any one oven of a coke plant, provided this is possible. The conditions necessary for such a procedure are that the required large amount of test coal be available in non-weathered condition, that the plant be equipped for such single-oven tests, and that preliminary examination has indicated the coal can safely be coked without damage to the oven structure. In this testing procedure the coke from the single oven can be separately collected, weighed, and samples taken. Thus, the yield of coke, the size distribution of the coke, and all pertinent physical and chemical tests of the coke as made in a full-size commercial oven can be obtained.

Normally, the single-oven test gives no information on the yields of gas, tar, light oil, or ammonia, since these products flow into the collecting main along with the volatile products from the other ovens of the plant. In one plant in the United States, that of the Bethlehem Steel Company at Johnstown, Pa., there was provided experimental equipment to prepare and coke a standard commercial-oven charge of coal, and this unit was equipped also to collect and measure all of the products from the single-test oven (1). At the present time this equipment is no longer available for tests.

As a complement to the single-oven test or other tests designed to determine primarily coke quality, there is available a laboratory scale method for determining the approximate yields of coke, gas, tar, ammonia, and light oil to be expected from the usual high-temperature carbonization of a coal or blend. This is the so-called "tube test" of the Chemists Committee of U.S. Steel Corporation (2), and a modified form of the equipment and test procedure has been successfully used by our company for many years. It must be emphasized that this laboratory test, which is very useful for yield determinations, gives no indication of the coke quality to be expected in commercial operation.

Since full-scale oven tests of coals or coal blends are often impossible or too expensive to carry out, especially in those cases where preliminary evaluation only is desired or where a broad survey of many coals and blends is necessary, many attempts have been made throughout the world to devise small-scale carbonizing tests. These have varied from laboratory tests requiring only a few grammes of coal up to pilot-plant procedures requiring several hundred pounds for a single charge. A summarized description of some of these small-scale carbonizing tests has been given by Russell (3).

Some of the laboratory tests, such as determinations of the "free-swelling index" (4), the "agglutinating value" (4), and the Gieseler plastometric method (5), which require samples of only a few grammes of coal, are useful as indicators of coking qualities, provided complete coking behaviour on a large scale of similar coals is known.

Box coking test

The box coking test is made by placing a sheet-steel box containing about 60 lb. of coal in a regular coke oven. Sometimes a larger box containing 250 lb. of coal is used. After the box has been placed, the oven is filled with coal in the usual manner so that the test coal is carbonized under substantially normal oven conditions. Our company has used this box coking test for many years and considers it an approximate indicator of the coking properties of a coal or blend, although not of equal reliability, of course, to the coking of a full charge of the test coal in a regular coke oven.

Carbonizing tests in small ovens

Since 1929 the United States Bureau of Mines has made a comprehensive series of tests of the carbonizing properties of American coals (6). Many coals and blends of coals have been carbonized at various temperatures in cylindrical metal retorts, with the result that physical properties of the coke, yields of products, etc., are available for many coking coals. While it is true that the coke made in these small, cylindrical retorts is not strictly comparable with that made in coke ovens, the relative characteristics of this test are of significance and value.

The State Geological Survey of Illinois also has made studies of coals and blends of coals in a small carbonizing apparatus of its own design (7). Physical properties of the coke and the yields of the various products have been determined, most of the studies involving the use of the high-volatile B coal of Illinois. Interesting comparisons were made of the physical properties of the cokes produced in this test oven with cokes produced in commercial coke ovens using the same coals.

In addition to comparative tests by the Illinois Geological Survey, a large steel company in the United States has also correlated the physical and chemical properties of its regular plant coke with coke made from the same blends in the Illinois Geological Survey test oven (8). Although this involved only the comparatively narrow range of coals actually used by this company, the test oven results compared very well with plant results.

Koppers' movable-wall test oven

This small oven was designed primarily to determine the carbonization pressure of coals or coal blends, so that use of coals or blends showing dangerously high pressure against the walls could be avoided. The oven consists of a slot-type chamber heated by gas burning in flues in the two walls. This chamber is 12 in. wide, 30 in. long and 42 in. high and holds more than 400 lb. of coal. One wall is mounted on a carriage that is free to move and to which a pressure-indicating system is attached, so that at all times during the carbonization period the actual pressures developed by the coal can be recorded (9).

In addition to measuring the pressure developed by an unknown coal, this movable-wall oven, being rectangular and of several cubic feet capacity, also produces sufficient coke for determination of its physical characteristics. Because of the oven size and also the height, the coke produced is not precisely the same as coke made in commercial ovens, but a fairly reliable correlation can be made. Later in this paper detailed comparisons are given between coke made in the movable-wall oven and coke made from the same coal in commercial ovens.

This movable-wall oven of Koppers Company, Inc. has been in use for more than twelve years, and over one thousand American and foreign coals have been tested in this equipment by our company alone. Over twenty of these test ovens have been built, the majority of which are in the United States, with a few in other countries in South America, Europe and Asia.

Comparison of coking quality in test ovens and commercial ovens

On certain occasions the same coal or blend of coals has been carbonized in both the movable-wall test oven and a commercial coke oven, and the physical properties of the coke from each have been determined. Actual comparisons are therefore possible and the reliability of the test oven as an indicator of the coking qualities of a coal can be judged.

The methods for testing the physical properties of the coke are those of the American Society for Testing Materials (4).

The coke ovens were of the Koppers-Becker type averaging 17¼ in. in width and with a capacity of about 18 tons of coal. Coking time was 15½ hours. In contrast the movable-wall oven is 12 in. wide with a capacity of about 0.2 ton and a coking time of about 7½ hours.

Some comparisons of the physical properties of the respective cokes are shown in the table.

The average results for ten different coal blends in the United States are quite typical of normal commercial coking coal mixtures. All of these coals were from the eastern fields of the United States. In general, the cokes from the movable-wall oven were smaller, weaker, and lighter than the corresponding cokes from

COMPARISON OF PHYSICAL PROPERTIES OF COKES FROM TEST OVENS AND COMMERCIAL OVENS

	Screen test			Shatter test	Tumbler test		Apparent specific gravity
	+4 in.	+2 in.	−½ in.	+2 in.	Stability factor	Hardness factor	
Average of 10 north American coal blends							
Movable-wall oven	4.6	67.1	3.7	52.6	36.3	63.2	0.854
Coke oven	8.6	73.1	3.7	58.6	41.2	66.0	0.959
Average difference	−4.0	−6.0	0.0	−6.0	−4.9	−2.8	−0.105
South American coal No. 1							
Movable-wall oven	12.3	83.6	5.1	70.4	47.1	57.1	0.864
Coke oven	19.6	80.7	4.6	68.5	52.4	67.8	1.160
South American coal No. 2							
Movable-wall oven	7.9	63.8	3.5	34.2	15.4	61.0	0.891
Coke oven	15.0	52.5	7.4	41.3	6.7	56.0	0.974

the commercial oven. For normal commercial coal blends of the eastern United States these differences would be expected when the differences in oven dimensions are considered. Such coals tend to produce larger and stronger coke with greater width of the oven and with slower coking time. Lower apparent specific gravity of the coke from the smaller oven would also be expected since the coal is under much less pressure than in the relatively deep commercial oven. When such coals are tested in the movable-wall oven, allowance can be made for these rather consistent differences with the result that the quality of coke to be obtained in a commercial oven can be predicted with a fair degree of accuracy.

South American coal No. 1 was a very unusual type of coal, quite different from any with which our company had previously had any experience. As a result the comparisons of coke properties from test and actual practice were somewhat different from our experience with the better known coals of the eastern United States. In this case the smaller size of the test oven made no very significant difference in the size of the coke nor in the shatter test. However, this coal was very markedly affected by the pressure of the mass of coal in the commercial oven, leading to the very high increase of 0.3 in apparent specific gravity over the coke from the small test oven. Possibly related to this decided decrease in porosity of the commercial-oven coke as compared to the test-oven coke was some improvement in both factors of the tumbler test. It should be emphasized that the differences between test and actual results for this coal are not great. The test results indicated that this coal would produce an exceptionally good grade of coke, as far as physical properties were concerned, and this was later confirmed in a large oven test.

South American coal No. 2 provides another example of deviation from the usual commercial coking blend of the eastern United States. This coal would be considered rather a poor coking coal, somewhat similar in this respect to some coals of the western United States. The results from the movable-wall test oven indicate this clearly and later this was confirmed by the commercial-oven test. The differences between test and actual results are not quite the same as for normal good coking coals, however. The test coke is weaker according to the shatter test but in general gave a somewhat better size, possibly due to the greater amount of handling to which the commercial-oven coke is subjected. The stability factor from the tumbler test was somewhat higher for the test coke but in neither case is the numerical figure of any great significance, since both are far lower than is normally considered acceptable for good metallurgical coke. Here again, some allowance can be made in figures obtained from the test oven and, knowing the characteristics of the coal, the quality of coke to be obtained in a commercial oven can be predicted fairly satisfactorily.

CONCLUSIONS

The quality of coke to be made in a commercial coke-oven plant can be predicted with a fair degree of accuracy by carbonization of the coal or blend of coals in a small test oven, such as the Koppers' movable-wall oven. The physical properties of the test-oven coke will not be exactly the same as the commercial-oven coke, but with knowledge of the type of coal, proper allowances can be made that will give fairly accurate predictions, with at least no widely misleading indications. Of course, actual tests of the coal in a commercial coke oven are necessary to give a completely accurate determination of physical coke quality.

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Argentine Asphaltites as Blending Material for Poorly Coking Coals

ALFRED R. POWELL

During 1947 and 1948 the Research Department of Koppers Company made a series of studies of Argentine asphaltites, with particular reference to the possible use of this material as a component in blends with coal for manufacture of coke.

The asphaltite (known in Argentina as "asfaltita") was shipped to Koppers' research laboratories by the Dirección General de Combustibles Sólidos Minerales of Argentina. Three different samples of several hundred pounds each and designated as No. 1, No. 2, and No. 3 were received but no further information was available except that the samples of asphaltite came from the province of Neuquen. Abraham (1) describes an unusual variety of grahamite that occurs on the eastern slope of the Andes mountains in the province of Neuquen, and it is presumed that the material used in these investigations was the same or at least closely related to the grahamite described by Abraham.

PROPERTIES OF ASPHALTITES

Proximate analyses (A.S.T.M. methods) of the three asphaltites gave the following results:

PROXIMATE ANALYSES

	Asphaltite		
	No. 1	*No. 2*	*No. 3*
Volatile matter, %	32.2	46.2	44.0
Fixed carbon, %	63.9	49.2	53.0
Ash, %	3.9	4.6	3.0
Sulfur, %	1.96	2.14	2.02

It should be noted that the sulfur content of these asphaltites, although lower than that of many other asphaltites found in other localities throughout the world, is relatively high for a raw material entering into the manufacture of metallurgical coke.

Since the plastic properties of coal or any other material that is carbonized are of great importance in determining the physical qualities of the resulting coke, the asphaltites were tested in the Gieseler plastometer (2). The Gieseler tests of Asphaltite No. 1 showed a softening temperature of 380°C. and a solidification temperature of 475°C., with a maximum fluidity of 1.8 divisions per minute. This temperature range of plasticity is similar to some strongly coking high-volatile coals, but the maximum fluidity attained is very low, much lower even than most low-volatile coking coals. In decided contrast to Asphaltite No. 1, the other two asphaltites softened below 300°C., and became so fluid that it was impossible to measure their fluidity in the apparatus.

Following the volatile matter determinations, the "buttons" of coke from the laboratory crucibles were examined in order to get an approximate idea of the coking properties of the three asphaltites. No. 1 had fused into a solid button that was fairly strong and occupied about the same volume as the original material, while the other two samples yielded buttons that were of a fluffy nature, at least twice the original size, and very brittle.

ADDITION OF ASPHALTITE TO CHILEAN LOTA COAL

The Lota coal of Chile is generally known as a poorly coking coal. It had been determined (by the Gieseler plastometer) that this coal became only slightly fluid when heated, while two of the Argentine asphaltites became very fluid, as indicated above. Thus it was believed that by increasing the fluidity of the Chilean coal with the addition of asphaltite it would be possible to improve the coke quality.

It was decided to test this out on a fairly large scale by carbonizing a mixture of 70% Pique Nuevo coal from the Lota property in Chile and 30% Asphaltite No. 2 in the Koppers' movable-wall oven, which is a test oven carbonizing a charge of over 400 lb. of coal (3). This test oven not only indicates the carbonization pressure of the coal against the oven walls throughout the coking period but also produces approximately 300 lb. of coke on which tests of physical quality can be made. In order to obtain a comparison of coke quality, a charge of 100% Pique Nuevo coal was also coked in the movable-wall oven under identical conditions.

Proximate analyses of the Chilean coal, of the Argentine asphaltite, and of the 70-30 blend of the two materials are shown:

PROXIMATE ANALYSES

	Pique Nuevo	*Asphaltite No. 2*	*70% Pique Nuevo 30% Asphaltite No. 2*
Volatile matter, %	41.3	46.2	42.1
Fixed carbon, %	51.8	49.2	51.8
Ash, %	6.9	4.6	6.1
Sulfur, %	0.62	2.14	1.10

Gieseler plastometer tests showed a very marked increase in fluidity as a result of the addition of asphaltite to the Chilean coal:

GIESELER DATA

	100% Pique Nuevo	*70% Pique Nuevo 30% Asphaltite No. 2*
Initial softening temperatures, °C.	321	Less than 300
Temperature of maximum fluidity, °C.	418	415
Solidification temperature, °C.	444	450
Maximum fluidity, divisions per minute	2.3	1,500

Although it was not expected that carbonization pressures would be high, these were obtained in both tests. The 70-30 blend showed a maximum pressure of 0.6 lb. per square inch as compared with 0.49 for the straight Pique Nuevo coal. These pressures are far under those that might cause damage to coke-oven walls.

The following table summarizes physical tests made on coke from the straight coal and coke from the 70-30 blend, so they may be compared:

CARBONIZATION AND COKE DATA

	100% Pique Nuevo	*70% Pique Nuevo 30% Asphaltite No. 2*
Coking data		
Bulk density of coal, lb./cu.ft...	48.4	47.8
Average flue temperature, °F...	2506	2436
Maximum pressure, lb./sq.in...	0.49	0.6
Screen test of coke		
Per cent on 4 inch...........	0.9	11.8
Per cent on 3 inch...........	8.7	18.8
Per cent on 2 inch...........	25.9	32.9
TOTAL % ON 2 IN.	35.5	63.5
Per cent on 1½ inch.........	27.5	15.2
Per cent on 1 inch...........	17.2	7.4
Per cent on ½ inch..........	12.0	6.7
Per cent through ½ inch......	7.8	7.2
Shatter test of coke		
Per cent on 3 inch...........	0.0	6.6
Per cent on 2 inch...........	10.7	22.0
TOTAL % ON 2 IN.	10.7	28.6
Per cent on 1½ inch.........	26.0	28.0
Per cent on 1 inch...........	26.0	20.0
Per cent on ½ inch..........	24.5	14.4
Per cent through ½ inch......	12.8	9.0
Tumbler test of coke		
Per cent on 2 inch...........	0.0	0.0
Per cent on 1½ inch.........	0.0	0.0
Per cent on 1 inch...........	0.8	6.7
STABILITY FACTOR	0.8	6.7
Per cent on ½ inch..........	29.8	31.5
Per cent on ¼ inch..........	30.7	20.5
Hardness factor..............	61.3	58.7
Per cent through ¼ inch......	38.7	41.3
Apparent specific gravity.......	0.773	0.826
Per cent coke yield (dry basis)...	66.2	67.7

The coke made from the asphaltite blend was not as "pebbly" as that from the straight Pique Nuevo coal and in general had a firmer structure. Most of the above physical tests show that the addition of the asphaltite caused a marked improvement in the physical qualities of the coke. Thus, by increasing the fluidity of this Chilean coal by this method the coking property is distinctly improved. It is quite possible that better results could be obtained by changing the proportions of the two components of the blend, but such a study was not possible with the limited amounts of Chilean coal and asphaltite available.

ADDITION OF ASPHALTITE TO STRONGLY COKING COALS

A series of coking tests was made to determine what results would be obtained when 40 or 50% of Argentine asphaltite was blended with strongly coking coal, either high- or low-volatile. Most of these tests were made in steel boxes holding 60 lb. of coal, with the boxes buried in a regular charge of coal in a commercial coke oven. One larger scale test was made in the movable-wall oven.

In general, it was found that all three of the asphaltites blended satisfactorily with both the high-volatile and the low-volatile coals, both of these being strongly coking American coals. Large blocky coke was obtained in all cases, and all cokes had a fairly hard structure and appeared to be well fused.

One interesting fact gained from these tests was that 50% of Asphaltite No. 3 added to low-volatile coking coal reduced the carbonization pressure to a point where the blend could be carbonized with little danger of damage to the coke ovens, provided certain precautions were observed. It is, of course, impossible to carbonize 100% of a strongly coking low-volatile coal under any conditions without immediately rupturing the coke-oven structure.

CONCLUSIONS

Although the investigations of the blending of Argentine asphaltites with poorly coking coals, such as occur in South America, are quite incomplete and of a preliminary nature, the evidence is strong that material improvement in the physical quality of coke can be so obtained. No information has been secured regarding the economics of mining and transporting these asphaltites, but assuming that the economics are favourable, it would appear that more extensive investigations should be made with equipment on at least as large a scale as the movable-wall oven used in some of the tests just described.

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The Utilization of Petroleum Pitches and Asphalts for the Production of Metallurgical Coke

M. D. CURRAN

High temperature distillation of petroleum residuals has been carried on for more than twenty years in the processing equipment designed and installed by our predecessors and ourselves. The objective during the early years was to provide the oil refiner with a means by which he could convert his heavy residuals into lighter and more saleable oils at times when heavy oils could be sold only at depressed prices, and at the same time provide him with an oil-free coke of high quality which could command a top price in the metallurgical coke market. While this general plan had a certain appeal, the oil refiners generally did not adopt it, preferring to dump their heavy oil at times when necessary or to continue the use of vapour phase cokers with which they were familiar. However, several plants were installed during the intervening years where processing methods and equipment design were developed and which provide the basis for our present-day equipment and its operation. The coke produced has found ready acceptance in various metallurgical industries and for electrode carbon manufacture.

In recent years industrial development has created a demand for high carbon, high density coke particularly for electro-thermal processing industries for use in the manufacture of carbon electrodes. One of the largest of such consumers is the growing aluminium industry, where many hundreds of tons are now used daily. Accordingly, the objective has changed, and emphasis is now placed on the need for high carbon, high density coke which can best be obtained from substantially ash-free petroleum base raw material of which there is an abundance in various forms of oil refinery residuals. So at this time our thinking and processing methods are directed toward coke quality and maximum coke production, even though the largest values may and generally do lie in the oils recovered from operations. Other more recent developments, which will be discussed in more detail later are (1) aromatic conversion of all the vapours distilled off in the coking operation to produce benzols, tar acid oils and aromatic pitches, and (2) the blending of pitches with non-coking coals to produce high grade metallurgical coke.

1. DESCRIPTION OF PROCESS

The individual coking chamber is an oven with a refractory floor approximately 33 ft. long by 10 ft. wide. The floor is heated from below by industrial or oven gas and is maintained at a temperature of the order of 1,800°F to 2,000°F. Refractory material is sprung in the form of an arch over the oven floor and all brick is encased in a steel shell. This construction eliminates the possibility of air leakage through the refractory material. The ends of the oven are sealed by removable refractory lined doors, which are raised while the coke is being removed, then lowered and sealed with a prepared luting clay prior to beginning the next run. The vapours formed during carbonization leave the oven through a specially designed gas off-take where they are cooled to avoid overcracking. After the coke has formed on the refractory floor the doors are raised and the coke removed by a motor-driven ram or pusher. The ram travels the entire length of the oven and discharges the incandescent coke into a coke conveyor, which delivers it to a quenching station where it is sprayed with water and then discharged on to a wharf where it is allowed to steam quench until thoroughly dry.

After the coke has been removed from the oven, the pusher is withdrawn and the doors lowered and luted. The heavy residue of about 0 to 2 A.P.I. gravity, and at a temperature of 750°F., is charged into the oven, and the rate of feeding is so regulated that the temperature in the vapour space does not fall below 850°F. When sufficient oil has been charged so that the thickness of the coke bed will be between 6 and 8 in., the charging is stopped and coking continued to completion.

The charging time will vary between 2½ and 4 hours, and the subsequent drying time will be between 2½ and 4 hours. At the end of the coking cycle the temperature in the vapour space immediately above the coke bed will rise to 1,200°F. or higher. This temperature assures substantial removal of all volatile matter from the coke. At this point an atmospheric vent valve is opened and the remaining vapours allowed to escape. The doors are then raised and the pusher removes the coke. The complete oven cycle varies from 6 to 8 hours according to the character of the charging stock and the kind of coke desired. In the event coke with less than 1% volatile matter is desired, low pressure air is admitted through the charging ports immediately after the oven has been vented, to the atmosphere in the amount required to consume the volatiles in the top of the coke layer, before pushing the coke from the oven.

During the early portion of the coking cycle, considerable volumes of gas oil vapours are formed, and as the coking reaction proceeds the total volume of vapours decrease until at the end of each individual cycle, only small amounts of fixed gas are formed. In view of the fact that the quantity and quality of the oven overhead varies considerably during the coking cycle, the vapour standpipes of a battery of ovens are connected to a common vapour main, which in turn discharges the vapours to a fractionating tower. The individual oven cycles are uniformly spaced so that the total vapour overhead to the tower is maintained substantially uniform, both in volume and composition.

The coking plant as outlined above, when operating in conjunction with a viscosity breaker on cracked residium from mid-continent crude, is capable of producing 60 to 65% liquid overhead products, 20 to 25% of hard coke with less than 2% of volatile matter, and between 10 and 15% dry gas.

2. PROCESS FLOW DIAGRAM

The coking unit shown in the accompanying flow diagram—Figure 1—is designed to operate on cracked residium, heavy topped fuel oil or heavy asphaltic crude.

In the arrangement shown the feed stock to the plant is delivered to the accumulator from whence it is pumped through the heating furnace where it is raised to the highest practical temperature, which usually runs from 700°F. to 750°F. The hot oil discharges through a pressure control valve into the evaporator, in which the lighter oils are flashed off and the heavier material remains in the liquid state.

The overhead vapours from the evaporator pass to the overhead fractionating tower where gas and crude gasolene are removed overhead, the gas oil removed as a liquid side cut and the re-cycle material drops to the bottom. The gasolene is condensed in the gas cooler and drains to the distillate receiver from which it is pumped to reflux and storage. The gas is drawn through the fractionating tower and the gas cooler by the exhauster and delivered through an absorber to the plant fuel gas lines or to a gasometer. The re-cycle material is drawn from the tower bottom by a flushing pump and re-circulated over the collector main and pressure regulating valves, the surplus being delivered to the charge accumulator where it re-enters the circuit as a mixture with the plant feed stock.

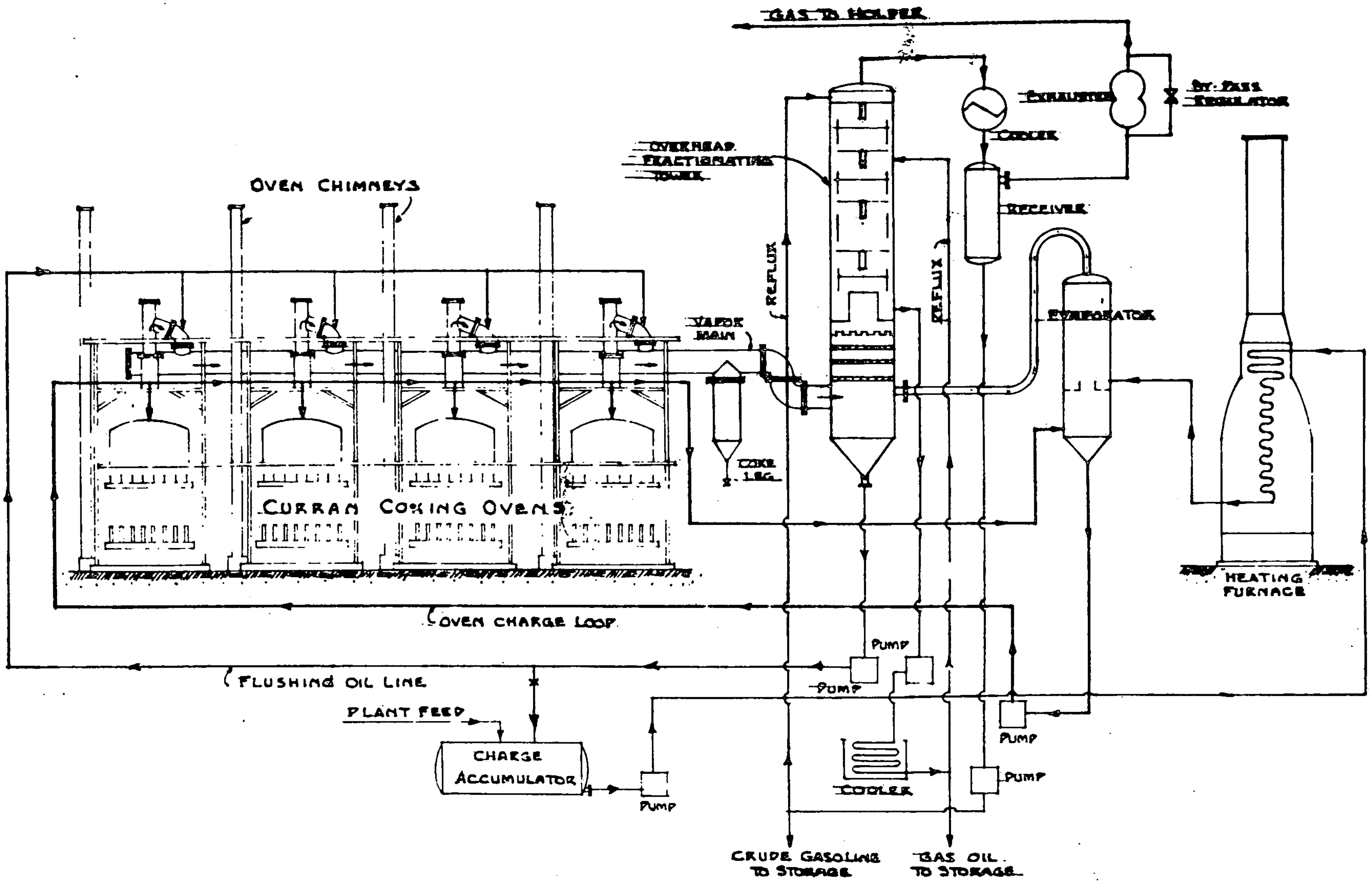

Figure 1
Flow diagram of coking plant with viscosity breaker.

The heavy residue produced in the evaporator is pumped to the oven charging loop where continuous circulation is maintained with the surplus returning to the evaporator bottom. Oven charges are drawn from the charging loop as required. The vapours from the ovens pass into the collector main through hydraulic pressure regulating valves, which maintain substantially atmospheric pressure in the oven chambers. The collector main delivers the vapours through a pressure regulating valve to the bottom of the overhead fractionating tower where they unite with the vapours from the evaporator and pass upward through the tower.

The flush oil which is used to wash the regulating valves and collector main free of carbon deposits returns to the bottom of the overhead tower passing first through a carbon trap where solid particles drop out and are removed as required and wasted. A constant level regulator in the bottom of the tower actuates a flow valve to permit the surplus heavy oil to run to the oven charge accumulator.

A most important feature of the process is the method of heat application to the floor of the coking chambers. Accurate temperature control is necessary for regulating the quality of the overhead products and the coke and minimizing the deposition of graphitic carbon on the oven floors. Our heating system employs the regenerative heat recovery principle in a manner which enables the regenerators to serve as a heat reservoir and stabilizer. This system of heating prevents excessive cooling of the oven floor during the early part of the charging period, and provides exceedingly uniform distribution of heat to the floor and to the charge. Experience with this heating system in coke ovens and industrial furnaces over a period of twenty years has proved that the floor temperature can be maintained uniform within plus or minus 50°F. at 2,000°F. Excessive production of graphitic carbon is readily avoided with such uniformity of heating since its formation usually occurs at 2,300°F., which is well above the floor temperature required for good operating practice. Accordingly, the charges of coke are easily removed by the pusher ram, and the floor carbon is generally completely removed by the decarbonizer attached to the pusher ram where high pressure air is applied on the backstroke of the ram.

Coking plants incorporating a viscosity breaker as shown on Figure 1 are designed to operate as a separate unit, or they may be connected with an existing viscosity breaker or asphalt unit, where oven charge stock of 0 to 2 A.P.I. gravity or heavier can be supplied for coking. The process is capable of considerable flexibility as to charge stock and will produce in addition to industrial, metallurgical or electrode coke, a good yield of gas oil or zero cold test fuel oil and a substantial quantity of gasolene of good octane and end point.

The following data are submitted to indicate yields and character of products obtained from a plant operating on mid-continent crudes.

	A.P.I. gravity	
Plant feed stock	10	Cracked residuum
Feed to ovens	2.1	Residue from evaporator
Gas oil produced	28.6	Side cut from fractionating tower
Re-cycle oil	8.7	From bottom of fractionating tower
Condensed gasolene	54.2	From fractionating tower overhead
Absorption gasolene	74.2	From non-condensable gases

Dry gas 1390 B.T.U. and 0.765 S.G.

YIELDS

	Percent/Wt.
Gasolene	7.22
Absorption gasolene	4.48
Fuel oil	53.10
Coke	25.60
Dry gas	8.80
Loss	.70
TOTAL	100.00

COKE ANALYSIS

	Per cent
Volatile matter	1.15
Fixed carbon	97.85
Ash	1.00
TOTAL	100.00
Sulfur	1.09
B.T.U. per lb	14,500
Moisture	1% to 2%

3. NATURAL ASPHALTS

Natural asphalts such as asphaltite, grahamite, gilsonite and other similar materials can be coked to produce oil suitable for refinery charging stock and good strong metallurgical coke. The chemical characteristics of the coke in any case are governed by the material used. Natural asphalts containing large amounts of ash and sulfur are not suitable for processing, but there are many deposits of natural asphalts which are quite free of impurities and capable of beneficiation. The economics of such processing will depend upon the market value of the coke and the oil recovered.

Solid materials such as gilsonite are readily processed by first melting and thereafter handled exactly the same as refinery pitches. Gilsonite containing ¼% of ash and ½% sulfur has yielded 35% coke of excellent quality and 3¼ barrels of oil per ton which ran 28 to 30 A.P.I. gravity.

Natural asphalts deposits such as that forming the Bermudez Lake in Venezuela can readily be processed to yield a good refinery charging stock and metallurgical coke of no higher ash and sulfur content than some cokes used in the steel industry. A small sample of this material which was submitted to us a few years ago assayed 3% ash and 2½% sulfur, showed an indicated coke yield of 25% which assayed 11½% ash and 2% organic sulfur. It is to be noted that carbon-sulfur compounds do not contaminate the metal in the iron blast furnace or cupola to the same extent as iron sulphide, the most common source of inorganic sulfur found in the raw materials forming the blast furnace charge.

The oven floor of the liquid charge oven is formed of silicon carbide tile but all other brick work consists of fire brick of various grades as required to provide the service necessary, at the various places in the structure. The firebrick is of course backed up with suitable insulating refractories next to the oven steel shell. The fireclay refractories employed have been developed through considerable research and experience during many years and are capable of resisting oil and carbon penetration very satisfactorily. The same is true of the silicon carbide floor tile.

Figure 2 shows a photograph of the pusher side of a liquid coking plant. The pusher machine shown at the right travels on rails and will serve many ovens. The ram is driven by a rack and pinion and is powered by either a D.C. electric motor or gas engine. The overhead fractionating tower is seen at the right. The four-pipe structure seen in the lower foreground is a high pressure (100 p.s.i.) air receiver which supplies decarbonizing air to jets located at the rear of the pusher ram head. The vapour collector main is shown near the centre with diaphram expansion joints between the ovens. The atmospheric vent valves and hydraulic pressure regulating valves are seen projecting above the upper operator's platform. The oven doors are raised vertically by hoisting cables actuated by an electric door hoist machine operated by controls located at each oven.

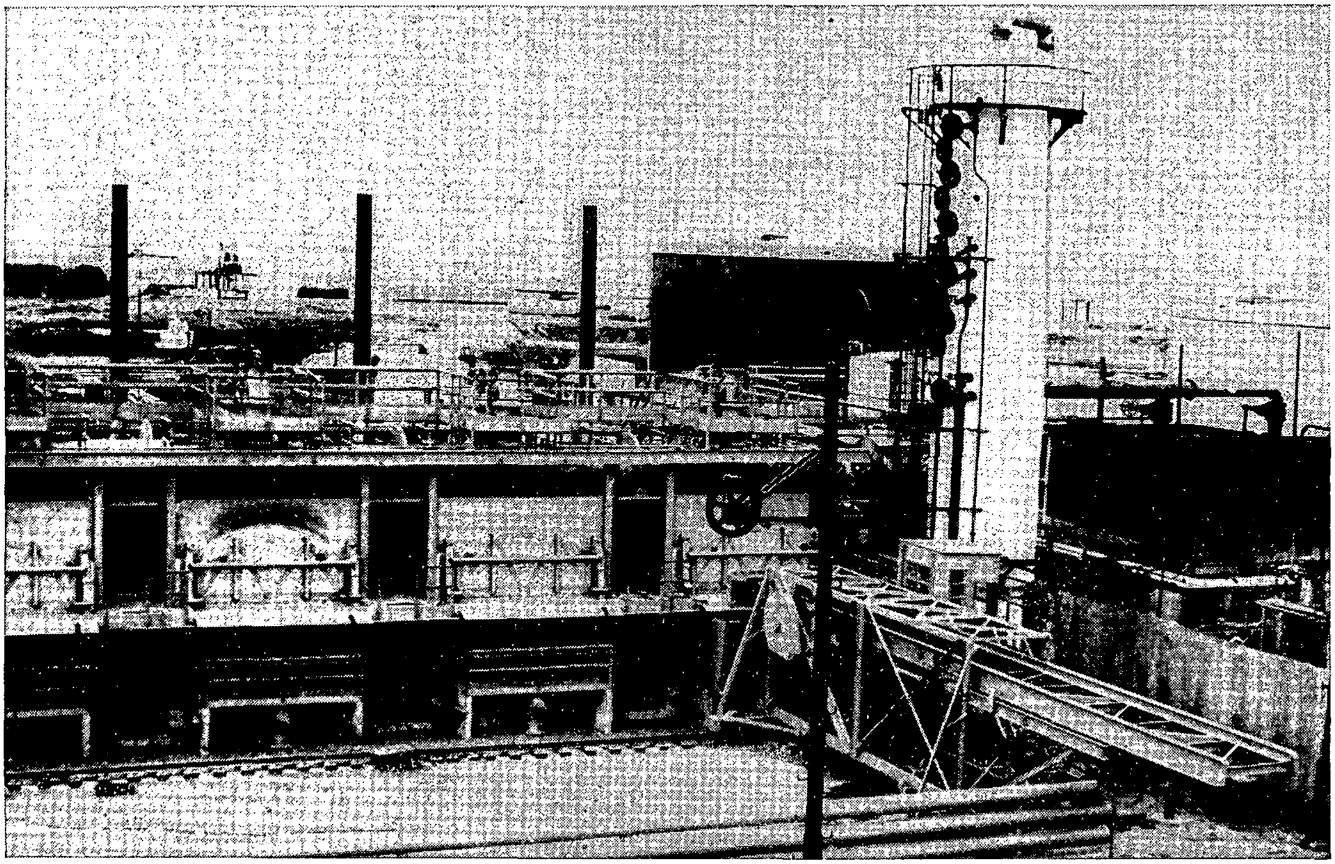

Figure 2
Photograph of the pusher side of a liquid coking plant.

The coke conveyor used for handling the hot coke from our ovens, was developed many years ago for our coal coking ovens. This installation is a residual coking plant where the discharge end of the conveyor was extended to provide for steam quenching and cooling of the coke, rather than install a coke wharf. A depressed ramp was in process of installation below the end of the conveyor to provide for discharging the coke into trucks. The quenching station is seen mounted above the conveyor and the quench water supply tank supported by a structural steel tower is seen at the left opposite the quenching station. This system of quenching the hot coke permits very accurate control of the coke moisture since a regulated amount of water can be applied to the uniform layer of coke in the conveyor in a definite time.

4. Aromatic conversion

Petroleum residuals and asphalts can be carbonized to produce aromatic products equivalent in quality to those produced from coal. During the last world war and since we have witnessed the large scale production of benzol and toluol from petroleum distillates which met all requirements for nitration. The aviation industry has created growing need for high octane gasolene which has been produced by various reforming processes. With the object of up-grading petroleum residuals we have developed plant equipment to reform or crack our oven overhead vapours to yield benzol, toluol, xylol, phenols, cresols, creosote and aromatic pitches. The amounts of these various compounds produced in any given case are governed by the character of the oven charge stock, the coking rate and temperature, and the reforming temperature.

Figure 3 presents a flow diagram of a coking plant equipped for aromatic conversion of the oven overhead vapours. The plant feed stock is vis-broken for recovery of fuel oil and the residual delivered to the ovens through a charging loop as it is in regular coking operations. The vapours overhead, however, instead of going to a fractionating tower are conducted through wall insulated mains to cracking furnaces where they are raised to the desired cracking temperature. Upon leaving the cracking furnaces the

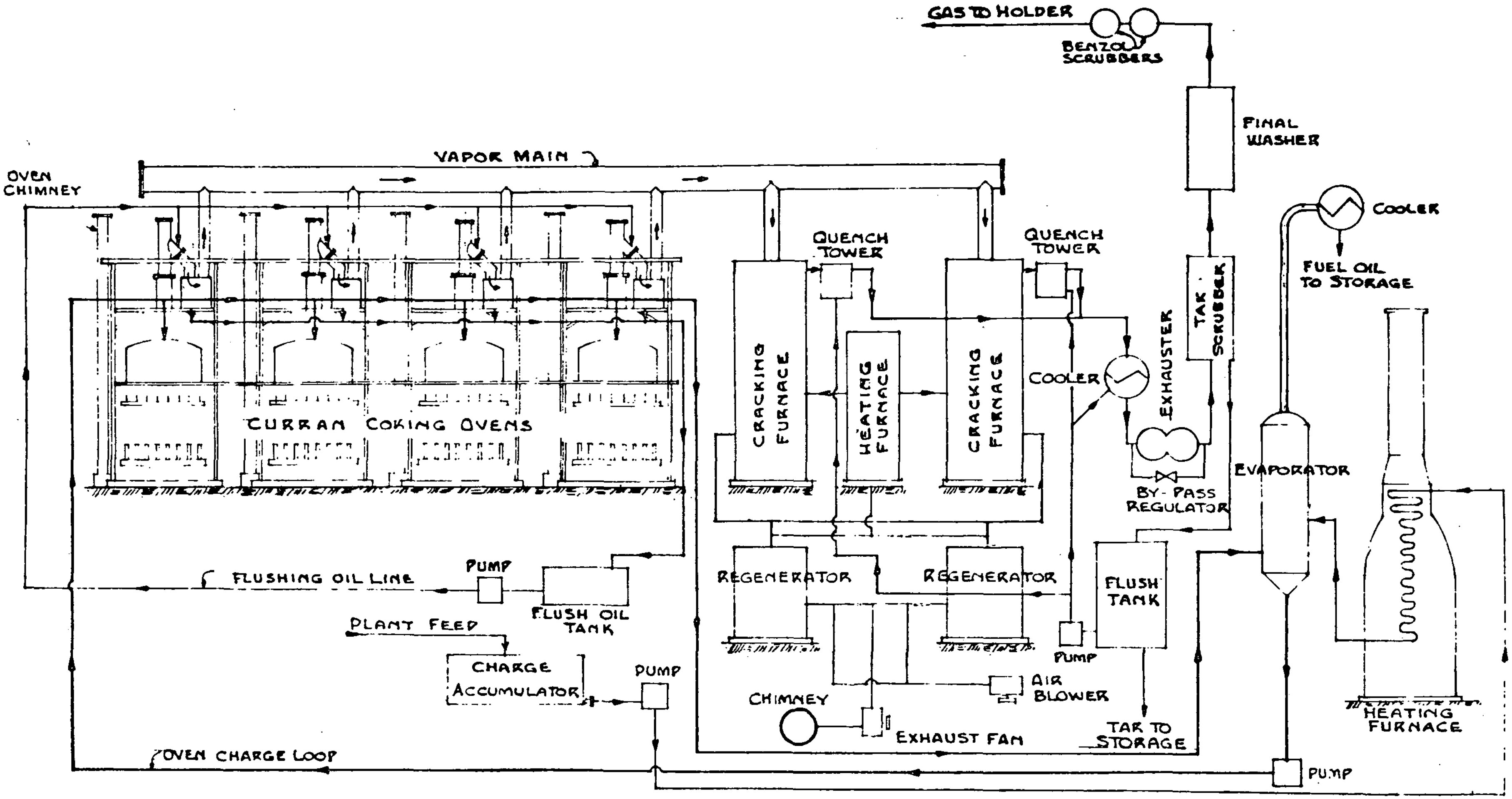

Figure 3
Flow diagram of coking plant equipped for aromatic conversion.

vapours are water quenched, cooled and scrubbed for removal of tar and benzols and the remaining non-condensable gas delivered to a gasometer and the plant fuel gas lines. The scrubbed clean gas has a heating value of 1,000 to 1,100 B.T.U. per cubic foot. The cracking furnaces operate alternately on a one-hour reversing cycle, and during the off period are automatically decarbonized by the admission of steam and air. This operation is completely handled by a reversing machine independent of all other operations.

The cracking furnaces employ silicon carbide tubes in a vertical assembly which are heated by hot combustion products from a gas fired heating furnace situated between the cracking furnaces. The heating gases, are maintained at the desired temperature usually in the range of 1,200°F to 1,600°F, by thermostatic control of an air inlet valve which regulates the flow of air into the combustion chamber of the furnace. The spent heating gases are drawn from the cracking furnaces through regenerators by an exhaust fan which delivers them to the chimney at a temperature of 400°F to 500°F. The regenerators operate alternately on a thirty-minute reversing cycle to store up heat and the succeeding cycle preheats the air used for combustion in the heating furnace. The reversal of the regenerators and the waste gas and air are handled by an automatic reversing machine which can be set to operate on any desired cycle from twenty minutes to one hour duration. The air and waste gas fans are adjusted to maintain substantially atmospheric pressure in the heating section of the cracking furnaces.

Thermal cracking of petroleum base vapours as herein carried on is accompanied by the production of lamp black carbon and increased quantities of fixed gases. The silicon carbide cracking tubes are installed in a two-pass arrangement with a large expansion chamber to facilitate precipitation of the carbon dust and to assure complete contacting of the vapours with the tubes. Steam and air for decarbonizing are admitted through the floor of the expansion chamber as required to consume all the carbon produced during the previous one-hour cycle. The interval of air admission may vary from ten minutes to thirty minutes, depending upon the cracking temperature used and other variables. By thus maintaining the tubes free of carbon deposits it is possible to maintain positive control of the cracking operation and minimize the fuel gas consumption on the heating furnace. Our experience shows that about twice as much fuel gas is required for cracking as is used for underfiring the ovens.

To illustrate the results of cracking as compared with a normal coking operation the following example is cited. An asphalt pitch of 130°F. melting point when coked yielded 25% coke, 62% gas oil and gasolene and 13% gas and loss. The gas metered 8,000 cu. ft. per ton at 1,300 B.T.U. per cu. ft. When the oven overhead vapours from this material were cracked the yields were 25% coke, 48% tar of 1.10 S.G. and 27% gas and loss. The gas metered 14,000 cu. ft. per ton at 1,100 B.T.U. per cu. ft. The cracking temperature was in the range of 1,200°F to 1,300°F. While the yield and character of the tar and gas varies somewhat with operating conditions, this example does show the substantial increase in gas make which necessarily occurs from vapour cracking.

A simple distillation of the tar is all that is required to develop attractive values from light oil, wood preservative, and soft and hard pitches. Redistillation of the light oil for recovery of benzol, toluol and xylol may or may not be feasible. Phenols, cresols and animal dips, if desired, can be extracted from the wood preservative fraction without unduly impairing its toxic value.

5. Pitch blending with non-coking coals for reduction of metallurgical coke

We believe that the large reserves of petroleum and asphalts provide the key which can unlock the door to iron and steel manufacture on a sustaining basis in Latin America. High grade iron ores in substantial quantities are well distributed among the countries. The lack of good coking coal has been a deterring influence which has obstructed progress in establishing iron and steel manufacture. Abundant reserves of non-coking or poorly coking coals are available in some areas which have suitable chemical properties but which lack only the bonding material required for coking. Since pitch produced from petroleum base materials contains suitable bonding substance, it is only necessary to blend such pitch with the coal, and coke it in such a way that the blend does not separate during carbonization. Aromatic pitch such as that recovered from the distillation of the tar is more effective and therefore smaller quantities are needed, usually on the order of 5% to 15% of the blend. Generally only one-fourth to one-half of this amount is recoverable from the coal tar so the shortage must be supplied with petroleum pitch. Since the volatiles distilled off the petroleum pitch are subject to aromatic conversion, the tar produced from a blend acquires increased amounts of aromatic pitch above the normal yield from the coal with the result that pitch production from the distillation of the tar is substantially increased. It is therefore only necessary to add such quantities of petroleum pitch as may be required to replace the bonding elements consumed in carbonization, which will vary with the coal used, but generally may be of the order of 10% to 15% of the coal-pitch blend. All pitch used must be sufficiently hard to be pulverized in a hammer mill and otherwise handled in the same way as coal. Such pitch usually has a melting point of about 100°C.

As previously stated the utilization of pitch binder with non-coking coals in a simple coking operation is dependent upon conditions of carbonization which prevent separation of the pitch from the coal. It is well known that if a coal-pitch blend is charged into a conventional slot type by-product coke oven, the pitch melts and runs to the bottom of the oven chamber. In order to overcome the problem, European operators have resorted to briquetting non-coking coal with pitch and then coking the briquettes. However, it has generally been necessary to remove the moisture from the coal before briquetting, and in order to provide the necessary quantity of pitch for briquetting they have usually found it necessary first to coke the coal under low-temperature conditions for maximum tar production, then briquette the low-temperature char with pitch and finally carbonize the briquettes at high temperature. In other words the final product, a coke briquette, is the result of a four-stage operation. We doubt very much if coke manufacturers of Western Europe would ever have resorted to such a costly method if they had been blessed with the abundant supplies of petroleum base materials that exist in Latin America. They would surely have developed a simpler way by incorporating the use of petroleum pitch to reduce their plant investments and operating costs.

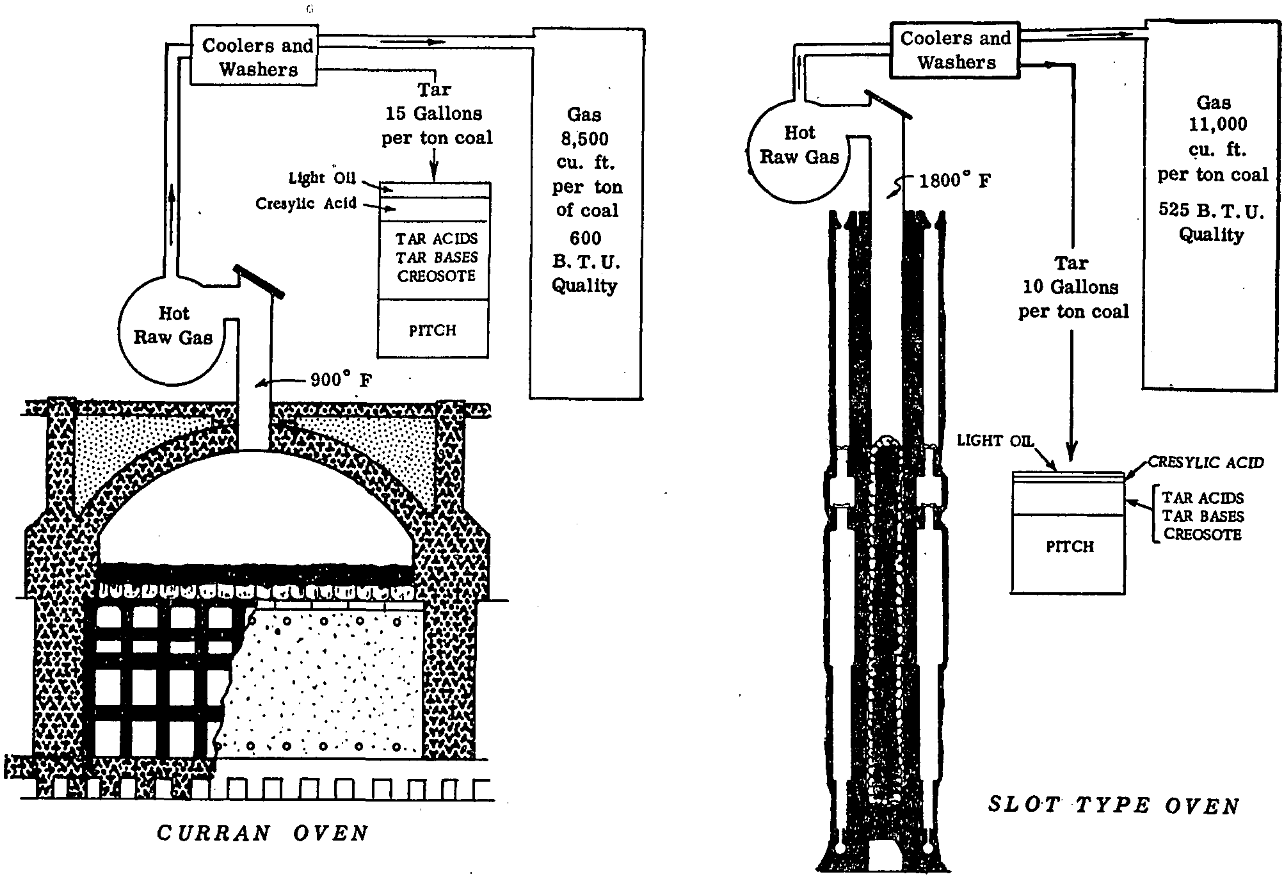

Figure 4
Sectional comparison between a Curran and a slot type oven.

Our coal coking oven which we use for coal-pitch blends was developed to make good coke from feebly coking coal, in other words, to use the bonding elements of the coal to the greatest advantage. Figure 4 gives a sectional comparison with the conventional slot type oven, and shows to scale the yields and characteristics of the gas and tar from both types of oven. It further indicates the difference in coking conditions which cause the difference in the by-products. The volatiles from the coal in a slot type oven travel to the gas off-take along the hot refractory walls and hot coke surfaces adjacent thereto, distances varying up to 50 ft., during which travel they have been heated to a temperature of approximately 1,800°F., and thermally cracked to give increased amounts of hydrogen and other fixed gases, together with carbon dust which accumulates in the tar. By comparison the volatiles enter the gas off-take of our sole heated oven at approximately 900°F, after having travelled a maximum distance of five feet along the hot refractory floor. Accordingly, there is only a limited decomposition of the tar acid oils, and the tar produced contains only small amounts of free carbon, naphthalene and pitch. The coking reaction which takes place in the sole heated oven is quite different from that which occurs in the slot type oven, where a coking speed of one inch of coal per hour is developed as compared with approximately one-half inch per hour from each side wall of the slot type oven. This is due to the fact that the gases travel in the same direction as the heat flow in the sole heated oven. The moisture in the coal passes upward through the coal layer as steam without contacting the oven floor, thereby eliminating a substantial heat loss common to the slot type oven. This feature is most important in coking coal-pitch blends where the steam serves as a counterflow force in preventing the flow of pitch downward through the coke layer. Accordingly, we do not need to remove the coal moisture before blending and coking but on the other hand sometimes find it necessary to add some water to the coal to get best results when coking coal-pitch blends.

As heretofore mentioned in carbonizing feebly coking coals we develop maximum use of the bonding elements in the coal because of the partial condensation and re-distillation of the tarry vapours as carbonization proceeds upward through the coal layer. Thus the bonding elements of the coal are filtered out and retained in the plastic conversion layer where the coke bond is developed. It is this feature that enables the successful use of petroleum base pitch as a bonding material for non-coking coals.

The general arrangement of a coal pitch coking plant is diagrammatically shown on Figure 5. For this work we use our coal coking ovens shown on

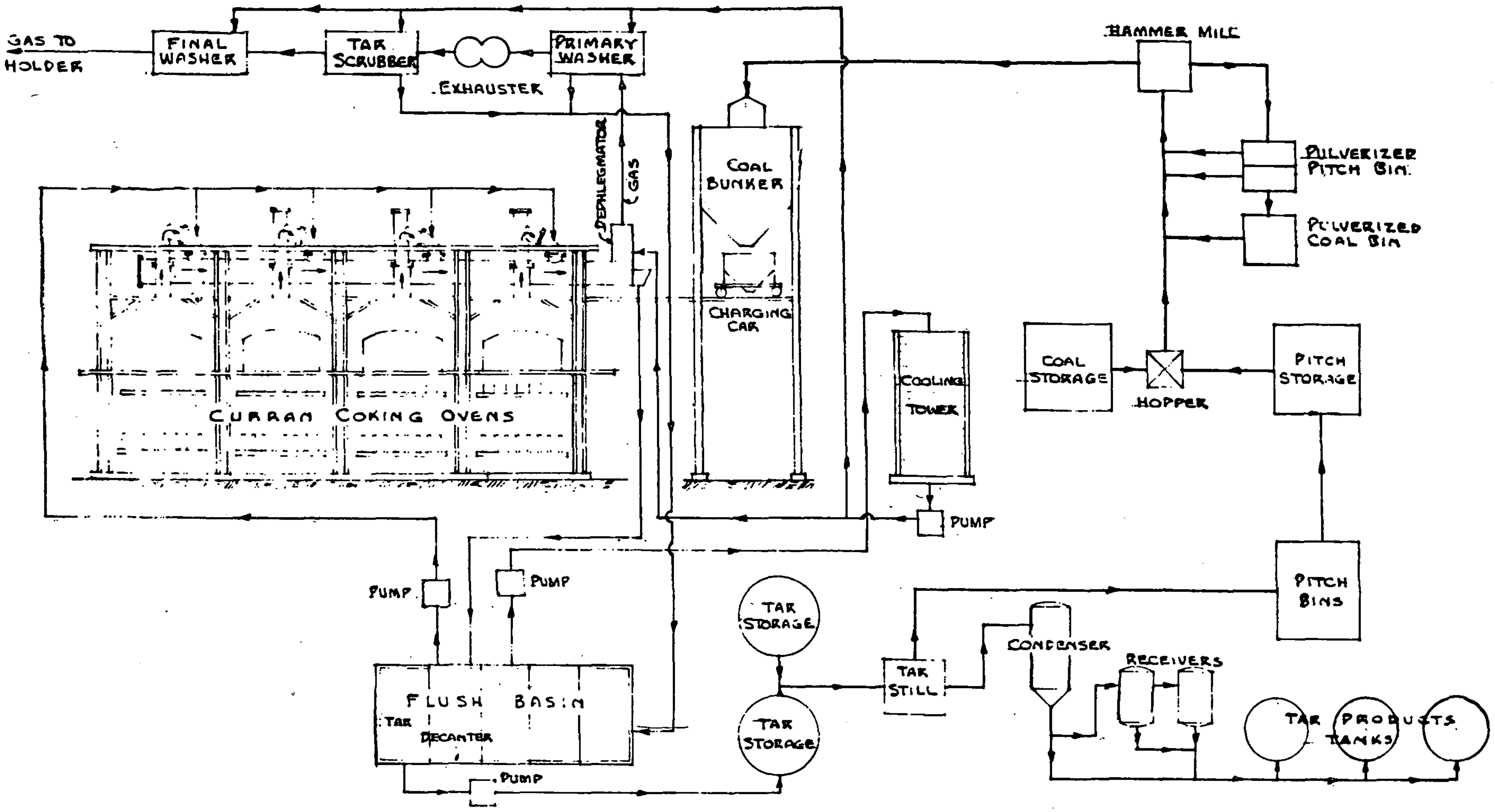

Figure 5
Flow diagram of coal coking plant for production of metallurgical coke from non-coking coals.

Figure 6. The diagram on Figure 5 shows essentially a coal coking plant with mixing bins for blending pulverized pitch with pulverized coal. The pitch bin has two compartments, one for purchased petroleum pitch and the other for the aromatic pitch recovered from distillation of the tar. The coal and pitches are first run through the hammer mill and accumulated in the mixing bins. Proportioning feeders deliver the materials from the mixing bins on to the hammer mill feed conveyor, which runs them through the mill again for intimate mixing. The coal-pitch blend is run to the coal bunker, from which it is withdrawn into the charging car for charging to the ovens.

The equipment indicated on Figure 5 is very similar to that used in many by-product coke plants where two or more coals are blended together with the exception of the tar distillation plant. Coke manufacturers in the United States have generally sold their tar to tar distillers or have burned it for fuel because choice coking coals have been available to them in great abundance.

Figure 6 shows the type of construction used in our coal coking ovens. It will be noted that no steel casings are used, and that the ovens are erected in blocks or batteries on a common foundation. The installation here shown comprises a battery of 16 ovens, which are producing metallurgical coke from an expanding coal. This battery is carbonizing 250 tons per day. It may be observed that the pusher machine used here is of different design than the one shown in Figure 2, which is used for liquid charge ovens. The machine shown here performs two operations, levelling and pushing and has a detachable pushing ram which is automatically attached to the levelling ram only when the coke is being pushed from the ovens. This machine is operated by a single 25 h.p. gas engine with clutch and gear arrangement for driving the ram and propelling the machine along the tracks in front of the ovens. The ram is operated by a steel cable instead of the rack and pinion drive employed on the pitch coking plant pusher. Below the ram carriage and supported on the truck frame are two gas storage tanks which when filled with coke oven gas at 100 p.s.i. will operate the machine for 12 hours. It will be noted that the oven doors are raised and lowered by a single cable attached to the top centre of the door frame. The gas collector main is seen on top of the battery with its off-take dephlegmator where the hot gas is shocked with cold flush water.

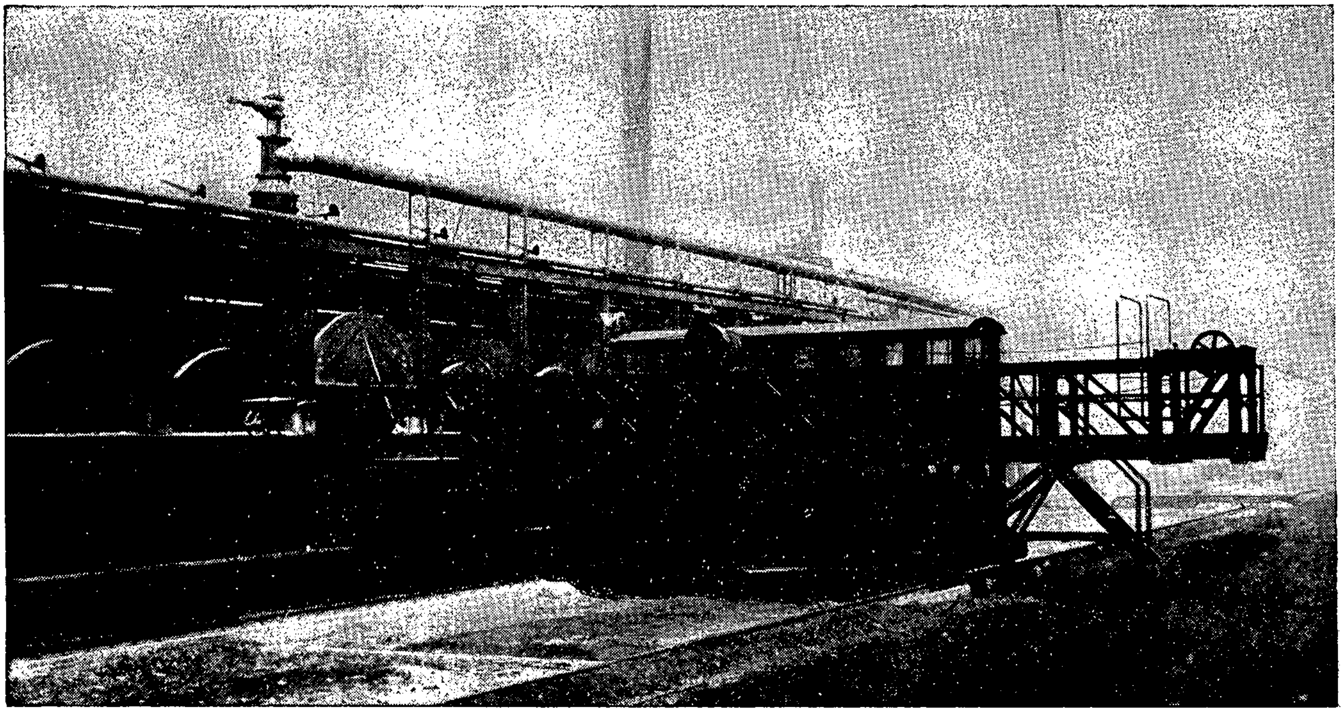

Figure 6
Photograph of type of construction used in Curran coal coking ovens.

A plant of the type shown on Figures 5 and 6 including the tar distilling plant is no more costly per ton of installed capacity than conventional slot type by-product coking plants, which require coals of good coking quality.

The following results were obtained from blending a Latin American sub-bituminous coal with 5% coal tar pitch and 15% asphaltic pitch containing 3½% sulfur.

	Per cent *Coal analysis*	*Coke analysis*
Moisture	7.00	1.6
Volatile matter	43.3	1.8
Fixed carbon	40.3	84.9
Ash	9.4	11.7
Sulfur	1.05	0.51

The coke was physically strong with standard coke structure and reactivity, and classified as a good blast furnace fuel. The extent to which sulfur is retained in the coke is largely a function of its composition in the charge, and its reaction to this carbonizing method. As a rule the moisture in the coal notably assists in decreasing the amount of sulfur retained by the coke. In this particular case the sulfur reduction was somewhat greater than usual.

6. INVESTMENT AND OPERATING COSTS

The installation cost of a liquid charge plant as shown on Figure 1 with capacity of 1,000 barrels of 10 A.P.I. residium per day is on the order of $3 to $3.50 per bl. of annual throughput for such a plant located in the midwestern area of the United States. Operating labour and supplies will run on the order of 30 cents to 40 cents per bl. based on operating labour at $2.00 per hour.

The addition of cracking equipment to a plant of the same capacity and as indicated on Figure 3 increases the installation cost approximately 50% and the operating cost 20%.

A coal-pitch coking plant as indicated on Figure 5 and illustrated on Figure 6 with a capacity of 500 tons per day of coal-pitch blend will cost approximately $12 per ton of annual capacity. Operating labour and supplies will run on the order of $1.25 per ton of charge based upon operating labour at $2 per hour.

Experience has shown that our high rates of pay to construction workers, practically offset the costs for transportation, insurance and erection costs, so that the installation cost figures given above will approximate plant installation costs at coastal locations in most Latin-American countries.

In all cases the gas produced from operations is adequate to supply all the plant fuel gas requirements. Generally there will be substantial amounts of surplus gas which, if marketable, may provide a credit against the operating costs named above.

CONCLUSION

It should be obvious that the processes described herein provide for considerable flexibility in combinations of plant equipment. We are confident that they can be used to advantage in utilizing the raw materials which are now known to be available and by which the best economy can be achieved. In the years ahead, when the world is at peace, and competitive markets return, it may not be possible to afford the use of foreign coal, such as is now obtained from the United States, for blending with local coals for the manufacture of metallurgical coke.

This paper is presented to direct attention to the advantages which may result from unorthodox consideration of all phases of any proposed iron and steel manufacturing installations and to suggest that processes and equipment may be employed which are capable of handling the available raw materials, rather than installing conventional equipment and then be obliged to purchase other and more costly materials in world markets to make the plant operable. We are not familiar with all the Latin-American coal and iron ore deposits, but we are acquainted with a few of them, from which we conclude that if such a basic industry is to be placed on a self-sustaining basis and one which will encourage the installation of iron and steel consuming industries and allied developments, the use and importation of high grade coking coals will have to be abandoned, ways and means developed for utilizing local raw materials. To accomplish this objective, orthodox procedures can only be followed where raw materials and markets fit into the common pattern.

Summary of Discussion

Improving Coking Qualities of Coal by Addition of Various Materials, *presented by the author*

Mr. CHERADAME opened the discussion on the paper, stating that the method of blending high-volatile coal with anthracite was also used in France, but not in the Lorraine field, and hence had not been described in his paper. Good results had been obtained. He would also like to know the size of the special ovens, because in the experimental plant at Marienau, there were large-scale ovens with four different widths, as it was felt that it would be interesting to compare the difference of width. One of the effects of different sizes was the faster or slower heating of the blend, but to date there had been no substantial variations in the cokes obtained from the different widths.

Mr. POWELL replied that it was common practice in the United States to have an average width of 16 in., but where the ovens had been built for the carbonization of western coals, the width was 14 in., slightly narrower, and the rate of heating was not proportionate to the width of the ovens. The rate of heating increased much more rapidly as the ovens were narrowed.

Mr. BAUM pointed out, in connexion with heating practice, that when considering coke ovens between. 500 mm. (20 in.) and 300 mm. (14 in.), the difference in coking speed in millimetres per hour was rather small, being between 20 and 30 mm. per hour, but it had been observed that with that difference certain unexpected results had sometimes been obtained. Were the heat progress to reach 80 mm., some coal would give coke which had not previously done so, but that was impossible because of the poor heat conductivity of the coal itself. With a briquette that could be heated on all sides, 60 to 80 mm. per hour could be reached. In the process he himself had described that morning, there was a heat progress of 250 to 280 mm. per hour, which could only be explained by the heat developing inside the layer.

Mr. POWELL said that his firm had conducted experiments on small ovens and had made a study of commercial ovens, some being as wide as 24 in. or more. He could not give the mathematics at that moment, but as the oven became narrower, there was a very decided pick-up in the rate at which the heat travelled, so that the coking time was not proportionate to the width of the oven, but was considerably shorter.

Mr. ALBALA said that it was possible to obtain good coke by blending Chilean coal with medium-volatile coal like Pocahontas No. 7 or 31. In Huachipato, Chile, a blend containing 20% of American medium-volatile coal had been used with success. Pitch production in the plant was not more than 2% of the coal used. An extensive series of tests had been made on the subject, but at present pitch was not being used for the purpose, as a 2% addition did not result in any noticeable changes.

Mr. POWELL replied that it was an interesting point, as he had mentioned that high-volatile coals did not give the desired type of coke.

Mr. SCHERESCHEWSKY referred to the four experimental cells operated at Marienau and mentioned by Mr. Cheradame in connexion with remarks made during the morning by Messrs. Prieto and Albala. They had insisted on the necessity of doing experimental work in normal-sized test ovens to ensure that various types of blend were acceptable. The University of Illinois had designed a comparatively small but very satisfactory test oven. The four cells at Marienau were rather unique; they had been designed to be as identical as possible to full-sized ovens, the only difference being that they were shorter. Their widths were 25-30—35-45 cm. approximately. It had proved very useful to use those ovens before going into full operation with a battery of 50 or 75 ovens, especially in the research programme on metallurgical coke made from high-oxygen and high-volatile coals.

Mr. POWELL said that the point raised was covered in another paper of his.

Correlation of Small-Scale Carbonizing Tests with Commercial Coke-Oven Results, *presented by the author*

Mr. CHERADAME opened the general discussion on the paper by agreeing that the use of small ovens might save time and money when one had to ascertain rapidly the quality of cokes to be obtained from new blends, which was an important problem in France. It was, however, important to gain an accurate knowledge of the correlation between results in the experimental oven and in the industrial one and that appeared to be difficult. For instance, such a small oven was being built at the Verneuil laboratories of CERCHAR, but that correlation had not yet been determined. The oven in question was heated with fuel oil, which did not give the same characteristics of flame and heating as gas; hence the problem was complicated. That was one of the reasons why France had decided to build large-scale experimental ovens.

Mr. POWELL replied that in his view the comparative results between the small and the test oven were considerably more significant than the absolute results. In many cases where a series of tests had to be made, the comparative results were of considerable value even though corrections might have to be made in the absolute results, and thus good carbonizing conditions could be selected before proceeding with large-scale commercial oven tests.

Therefore, for preliminary sifting out and as a general indicator of the coke that could be made, the small ovens were still being used by Koppers Company.

Mr. PRICE described the arrangements made at his plant in order to conduct a box test using a small amount of coal as part of the normal charge of a commercial oven. He agreed that the full-scale oven was the final test which should be made, but the box test described had been devised so as to approach as nearly as possible the coking conditions as encountered in the standard oven. In his opinion, the customary tests did not yield completely satisfactory results; in the case of the square steel box, the greater rate of heat conductivity of steel created a non-standard condition of coking in the sample, whereas in the carbide can or other round container, the shape of the pieces was distorted from that normally obtained.

In order to eliminate such features and to obtain results more nearly approaching standard oven results, he had been using a wooden box surrounded by a wire net cage. When the coking of the regular charge in the oven had been completed, the oven was pushed in the regular way, and the wire cage remaining after the wood burned away could be recovered from the quenching car or from the wharf.

The coke so produced should amount to about 100 lb., which was sufficient for making complete laboratory chemical and physical tests. Each test should consist of three boxes charged in different ovens. Occasionally a test might be lost through burning or breaking the wires, in which case it should be repeated so that all tests were based on duplicate results. It might therefore be wise to prepare sufficient sample coal for at least five boxes.

Laboratory tests which should accompany each box test were the following:

On coal: Proximate analysis, sulfur, phosphorus, bulk density, free-swelling index, screen test and, in special cases, expansion tests.

On coke: Proximate analysis, sulfur, phosphorus, screen test, shatter test, tumbler test, porosity and, in special cases, analysis of coke ash.

Mr. CORTÉS OBREGÓN stated that, using the technique just described by Mr. Price, several tests had been made in Mexico using wooden boxes, with very good results. Mexico had been trying to obtain some financing for the construction of a plant, and while the experiments were being conducted, the financing group had discovered that the coal swelled too badly in the experimental ovens, and they decided that full-scale experimental work should be done before the money was granted. That coal had been used in normal operations for over thirty years, showing that even though the information had some real value, too many factors which could not be considered in a small oven had to be taken into account and no decision should be made solely on the basis of small-scale experiments.

The CHAIRMAN thought that Mr. Powell had put the issue very clearly, to the effect that where millions were involved, the immediate step should be the commercial oven test. He would, however, be giving a practical example of the value of the several small-scale tests in a paper to be presented on the following day. There were no commercial ovens in Colombia for testing purposes, and the despatch of many tons of samples to the United States in sealed cans was very expensive. Furthermore, there were many varieties of coals in the several mines that had to be investigated near Paz de Río, as participants would see for themselves when they visited the plant.

It was therefore no exaggeration to place Colombia's coal reserves at 40,000 million tons, which meant that one country's reserves alone exceeded the figure of 37,000 million tons for all Latin America given in one of Mr. Price's papers.

Argentine Asphaltites as Blending Material for Poorly Coking Coals, *presented by the author*

Mr. STURGEON opened the discussion by asking, first, whether it would be possible to use the coke obtained from the asphaltite mixture in an ordinary high-shaft blast furnace; secondly, what degree of co-operation could be encouraged between the oil refineries and the coke producers.

Mr. POWELL replied that with proper development work, the use of Argentine asphaltites with a low-rank coal probably would produce coke suitable for the ordinary blast furnace. However, no positive assertion could be made, because the tests were very preliminary and no blending with additional coals or materials nor tests with different proportions of blends had taken place. Until promising results had been obtained, only small-scale experimental oven tests would be run; once the results were encouraging, the tests could be made in full-scale ovens, when it would be decided whether the coke could be successfully used in the blast furnace.

The second question, as he understood it, concerned the possible use of petroleum pitch. That had not been mentioned in his paper, but some work had been done on it, with two or three American petroleum companies which were currently active in that field. The results were, however, too indefinite to mention. Petroleum pitches as received from the companies had the disadvantage over coal-tar pitch, in that the coke yield was only about 30%, as compared with 60% using coal-tar pitch. In his opinion, coal-tar pitch was probably preferable, but on the other hand it was scarce, whereas petroleum pitch was a product with insufficient uses to cover production.

Mr. ALBALA mentioned the stability factor, saying that an increase of that factor to 6.7% for a blend containing 30% asphaltite was too low, as the value for the same percentage of Pocahontas coal with 70% Chilean coal should be around 35%. He wondered whether it would be possible to find some optimum mixture with asphaltite.

Mr. POWELL replied that a stability factor of 6.7% was not very good; however, in practice, it might be proved that 30% was too much, or else that something would have to be added to decrease the shrinkage of the coke. He was certain, however, that a higher stability factor than the one indicated could be achieved.

The CHAIRMAN requested Mr. GAMBOA to take over the chair.

The Utilization of Petroleum Pitches and Asphalts for the Production of Metallurgical Coke, *presented by Mr. Sturgeon in the absence of the author*

Mr. LEUSCHNER opened the general discussion by explaining that considerable trouble had been taken in order to obtain the paper, with a view to giving the Venezuelans an opportunity of investigating the prospects for their iron and steel industry. Venezuela had a considerable market for steel which could justify a rather large industry based on standard blast furnaces. There were large mines of an excellent ore and many possibilities of finding the other reducing or heating agents for the iron ore. In his view, the greatest difficulty in Venezuela was the fact that wages were extremely high, and that factor pointed towards the use of the standard blast furnace. Another possibility would be to take advantage of the vast asphalt lakes. That would require careful investigation, as the asphalt was very sticky and difficult to handle. The value of Mr. Curran's paper lay in the proof it provided of the possibility of making hard coke suitable for blast furnaces from asphalt.

Mr. STURGEON replied that the cost of construction of a typical Curran plant was given as $3 to $3.50 per annual barrel of material processed. Taking those figures, and estimating a charge stock of $1.50, adding to it 30 to 40 cents for labour and 60 to 70 cents a barrel for any additional cost for pay-out over five years, say a total cost of US$2.40 to US$2.60 would result. Those were his own, not the author's figures.

The cost figures were the following:

Gasoline	11.7% at US $4.20	US $ .49
Fuel oil	53.1%	.80
Coke	25.6% (90) at US $15 per ton	.60
Dry gas	8.8%	.13
Loss	0.7%	
TOTAL	100	US $2.02

Given a charge value of $1, a five-year payout and a slight profit could be achieved. With material at less than $1, a paying plant would be possible.

The Venezuelan asphalt region looked somewhat like Trinidad, except that it had pools of sticky oil in it, so that the engineering problem involved was quite difficult and would have to be solved with a minimum of labour. He had not shown the effects of Venezuelan construction costs, which would run about 150% of Stateside Gulf costs. Labour would run about 120% of Stateside, the labour being excellent on repetitive work, but on entirely new operations such as plant construction, only 85% efficiency would be obtained.

Mr. BAUM said that the same problem regarding the use of petroleum refinery residual products had arisen in Peru, but there the problem was to use the minimum of such products. Where there was an abundant quantity of oil residues available, furnaces like the Curran one described certainly provided one solution, but the coke yield varied between 25 and 30%. The next step would be the mixture of petroleum residues and coal using blends of 30% pitch and 70% coal, which could be done in any standard coking ovens, as in the case of a German plant producing electrode coke. Beginning from the opposite end, it was possible to use mixtures of 70% petroleum products and 30% finely ground coal and for such blends ovens such as the Curran one were indicated. Finally in some cases it would be necessary to use a maximum of coal and a minimum of residues and binders, as in the case of the new plant under construction in Peru where 30% pitch was used. There was about 50% recovery in the form of oils and aromatic hydrocarbons which again yielded pitch that could be re-cycled.

Mr. STURGEON replied that the Curran oven was particularly adapted to mixtures of pitch and coal because the oven was heated from the bottom and the difficulties such as flowing out which might arise with some mixtures of coal and pitch in a conventional type of oven were avoided.

Mr. POWELL, referring to the last statement, quoted an extract from the paper: "It is well known that if a coal-pitch blend is charged into a conventional slot-type by-product coke oven, the pitch melts and runs to the bottom of the oven chamber," and added that in spite of that statement, pitch coal blends had been used for many years in the western plants to make metallurgical coke, and the pitch did not run to the bottom of the oven, because if it had, its use would have been discarded.

Mr. STURGEON replied that he was not in a position to defend Mr. Curran's statement, unless the reference was to mixtures of pitch and coal, which could not be considered in a slot-type oven due to an excess of fluid pitch, but which could be used without difficulty in the horizontal Curran ovens.

Mr. BAUM said that he had mentioned the figure of 30% which actually had been proved on a large scale, as there was the same unjustified fear in Germany that the pitch would run out of the oven, but with 70% coal and 30% pitch, very strong coke was obtained. He considered that the Curran ovens started with mixtures far above 30% and could attain up to 100% pitch alone.

Mr. LEUSCHNER asked whether Mr. Sturgeon could not give an idea regarding some of the valuable aromatic products which, according to current literature, could be obtained.

Mr. STURGEON replied that he wished to speak for himself in answering that question. There had been considerable progress in the petroleum industry over the last few years concerning valuable aromatics, which included toluene, zuolene and benzolene; he believed, therefore, that their prices were bound to decrease in the world markets. Mr. Curran's calculations, based on the calorific values of the liquid by-products, seemed to him to be conservative but justified.

FUELS

Production of coke from poorly coking coals and substitute fuels (*concluded*)

17 October 1952—Morning

Chairman:

Eduardo Pyles Lozano, Companhia Mineração Geral do Brasil Ltda, São Paulo, Brazil

Contributed Papers:

Coking Properties of the Coal for the Steel Industry in Colombia

Joaquín Prieto, Assistant Manager, Empresa Siderúrgica Nacional de Paz de Río, S.A., Bogotá, Colombia

Jaime A. López T., Chief, Belencito Chemical Laboratory, Empresa Siderúrgica Nacional de Paz de Río, S.A., Colombia

Benjamín Alvarado, Assistant Manager, Empresa Siderúrgica Nacional de Paz de Río, S.A., Bogotá, Colombia

Vicente Suárez Hoyos, Chief, Cleveland Office, Empresa Siderúrgica Nacional de Paz de Río, S.A., Colombia

Production of Metallurgical Coke in Mexico

Salvador Cortés Obregón, Oficina de Investigaciones Industriales, Banco de México, S.A., Mexico, D.F.

Metallurgical Coke from Chilean Coals

Américo Albala, Assistant Superintendent, Coke Plant, Cia. de Acero del Pacífico, S.A., Huachipato, Chile

Manufacture of Metallurgical Coke from Peruvian Anthracite

Kurt Baum, Consulting Engineer, Paneuropéenne d'installation d'equipements industriels, Essen, Germany.

Summary of Discussion:

Participants: Messrs. Prieto, Albala, Cortés Obregón, Leuschner, Cheradame, López, Mercier, Baum, Saniter

Coking Properties of the Coal for the Steel Industry in Colombia

JOAQUÍN A. PRIETO ISAZA, JAIME A. LÓPEZ T., BENJAMÍN ALVARADO AND VICENTE SUÁREZ HOYOS

1. Experimental method adopted

In the particular case of the coals found in Paz de Río, Colombia, where there are no industrial facilities available for making metallurgical coke, the sending abroad of samples of adequate size and in sufficient quantities to undertake industrial tests was considered expensive and complex, because of the extent of the coal-bearing area, the number of coalfields, and the wide diversity of coals.

In these circumstances, it was decided to establish a research process for determining accurately the carbonization characteristics of the coals selected in the Paz de Río-Sogamoso area for supplying the steel plant, without excluding the possibility of later essays on an industrial scale, as soon as the best coals available were determined.

The selected method simply consisted in the following order of experimentation:

(1) Proximate analysis of coals and their classification;

(2) Plasticity tests to determine the fluidity of the coals;

(3) Expansion tests, according to the results of plastometer measurements in order to determine the

pressures developed during the carbonization process;

(4) Physical tests, such as the determination of the shatter, tumbler and density indices by means of the coke fabricated in the preceding test;

(5) Test on the blending of coals, in accordance with the results obtained from any particular coal;

(6) New coking tests, with the best coals or blends of coals available, after fixing the movable-wall oven and with a wet sample crushed to a predetermined size, in order to duplicate the best carbonizing conditions in the industrial oven.

In such conditions, it is only necessary to submit the coal or the selected samples to a series of tests on an industrial scale.

There is nothing original in this method, but it can be offered as a concrete and practical example of the utilization of modern analytical methods in a systematic research, when there are, as there were in our case, an extensive carboniferous area, numerous coalfields and a wide variety of coals; these conditions are quite common in various regions of our country, and in several South American nations. This research method offers an additional advantage; it permits undesirable coals to be eliminated, and it quickly leads to an accurate evaluation of coals offering the most favourable conditions.

2. LOCATION OF COALFIELDS AND RESERVES

The country's most important carboniferous areas were shown in another paper presented by the authors[1] while the location of the coalfields in the Paz de Río-Sogamoso region, in relation to those areas and to the Belencito steel plant site, is shown in Figure 1.

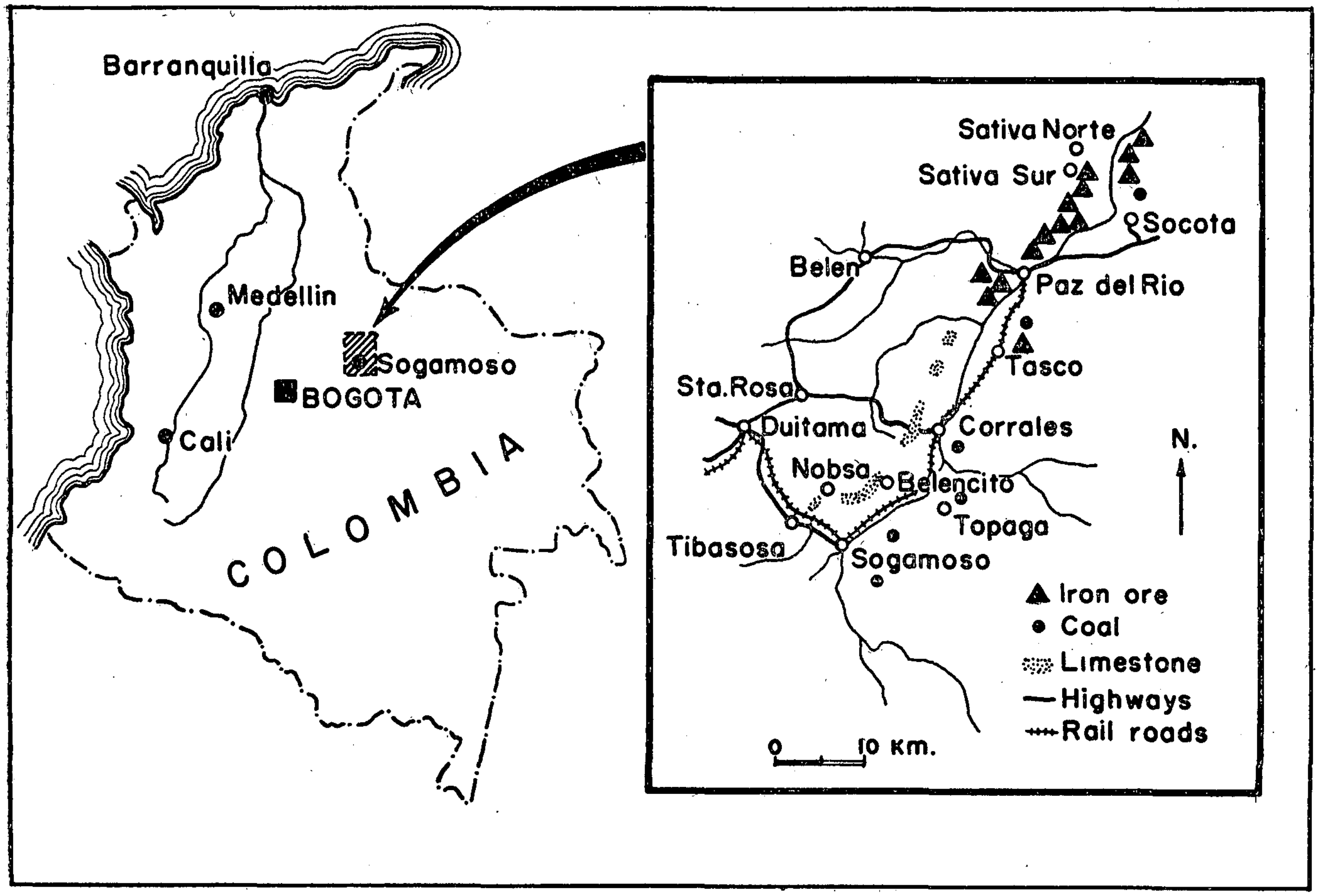

Figure 1
Location of coal, iron ore and limestone deposits in the Paz de Río-Sogamoso area.

The total reserves in the coalfields studied amount to 1,920 million tons, all within a 40-km. radius from the plant site. (See Table 5).

3. SAMPLES

In the various coalfields, there were some rudimentary mining pits, several of which were in operation while others were abandoned. Samples were taken from such pits in order to make a preliminary classification of the coals, and subsequent proximate analysis. A selective survey of the coalfields was later undertaken and for this end tunnels and trenches were driven in every field, substituting the diamond drills, which were not considered adequate for the peculiar topography of the area. Samples were carefully taken in order that the results obtained from the tests would be accurately indicative of the various types of coal. Moreover the samples were always taken from fresh working faces, gutters being driven

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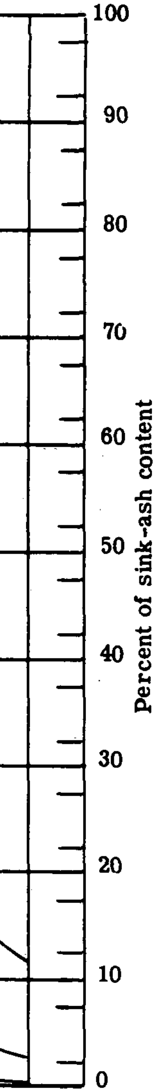
[1] "Problems Related to Colombian Coal used for Steelmaking", Figure 1.

CALCULATIONS FOR EACH PARTITION CURVE

avity



a	Ash content of the fraction	I dp = 1.4		II dp = 1.5		III dp = 1.6		IV dp = 1.8		V dp = 2.0		VI dp = 2.2			
		c_1	$d_1 = bxc_1$	c_2	$d_2 = bxc_2$	c_3	$d_3 = bxc_3$	c_4	$d_4 = bxc_4$	c_5	$d_5 = bxc_5$	c_6	$d_6 = bxc_6$	c_7	$d_7 = bxc_7$
97.5	2	97	194	100	200	100	200	100	200	100	200	100	200		
92.5	5	91	455	97.5	488	100	500	100	500	100	500	100	500		
87.5	7	84	588	95	665	99	693	100	700	100	700	100	700		
82.5	9	77.5	698	92	828	98	882	100	900	100	900	100	900		
77.5	11	71	781	89	979	97	1067	100	1100	100	1100	100	1100		
72.5	13	63	819	85	1112	95	1235	100	1300	100	1300	100	1300		
67.5	15	56	840	82	1230	92	1380	99.5	1493	100	1500	100	1500		
62.5	16	48	768	79	1264	91	1456	99	1585	100	1600	100	1600		
57.5	17	42	714	76	1292	89	1513	98.5	1675	100	1700	100	1700		
52.5	19	35	675	71	1349	86	1634	98	1862	100	1900	100	1900		
47.5	20.5	30.5	625	65.5	1343	82.5	1692	97	1989	100	2050	100	2050		
42.5	22.5	26	585	58	1305	79	1777	96	2160	99.5	2240	100	2250		
37.5	25	21	525	49	1225	74	1850	95	2375	99	2475	100	2500		
32.5	27	16.5	445	40	1080	68	1836	92.5	2500	98.5	2660	100	2700		
27.5	32	9.5	304	27.5	880	54	1728	87	2783	96.5	3090	99.5	3185		
22.5	41	3.5	144	11.5	471	29	1189	68	2788	89.5	3670	96.5	3960		
17.5	52	0.5	26	2.5	130	10.5	546	39	2028	73	3796	87.5	4550		
12.5	64			0.5	32	2.5	160	16.5	1056	41	2655	69	4416		
7.5	76					0.5	38	5	380	18	1368	38	2928		
2.5	72							1	72	6	432	18.5	1332		
		Total: $e_1 = 772$ $R_1 = \frac{e_1}{20}$ = 38.6	Total: $f_1 = 9186$ $T_1 = \frac{f_1}{e_1}$ = 11.9	Total: $e_2 = 1121$ $R_2 = \frac{e_2}{20}$ = 56	Total: $f_2 = 15873$ $T_2 = \frac{f_2}{e_2}$ = 14.2	Total: $e_3 = 1347$ $R_3 = \frac{e_3}{20}$ = 67.4	Total: $f_3 = 21376$ $T_3 = \frac{f_3}{e_3}$ = 15.9	Total: $e_4 = 1592$ $R_4 = \frac{e_4}{20}$ = 79.6	Total: $f_4 = 29446$ $T_4 = \frac{f_4}{e_4}$ = 18.5	Total: $e_5 = 1721$ $R_5 = \frac{e_5}{20}$ = 86	Total: $f_5 = 35836$ $T_5 = \frac{f_5}{e_5}$ = 20.8	Total: $e_6 = 1809$ $R_6 = \frac{e_6}{20}$ = 90.4	Total: $f_6 = 41271$ $T_6 = \frac{f_6}{e_6}$ = 22.8	Total: $e_7 =$ $R_7 = \frac{e_7}{20}$ =	Total: $f_7 =$ $T_7 = \frac{f_7}{e_7}$ =

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from the roof towards the bottom of the stratum to ensure that the different bands of components were intersected in every sample. Each of those samples was reduced to the weight of 2 kg., through the habitual quartering process, and the standard analytical methods were employed.

Once a better idea had been obtained concerning the quality of the coals, and other economic factors pertaining to the development and exploitation of the coalfields from the mining, locational and transport standpoints had been duly considered, larger samples were taken outside of the meteorization zone, in each of the seams of the various fields. For this purpose tunnels were driven to the appropriate depths, operating always over fresh working faces, where mining had not been stopped for more than three days. Driving shafts were also useful in investigating reserves, and in fact one was driven approximately every 100 m., on each seam. Tests were made at various depths until it was clearly established that the meteorization zone had been passed over. After that, still larger samples were taken of about 500 kg. for the coking tests, and were shipped to the laboratory, packed in hermetically sealed drums. Once at the laboratory, samples were crushed by hammer mill to sizes varying from ⅛ in. up to 1 in., according to testing necessities; finally, samples were put through the common crushing processes and reduced to 200 kg., which is the amount required for the charge of the Koppers oven. Moreover, in every case a small sample for proximate and granulometric analysis was taken, for the purpose of securing such data for each of the carbonization tests.

4. CLASSIFICATION OF THE COALS

The chemical composition of the coals, as demonstrated by proximate analysis, is shown in Tables 1, 2, 3, 4 and 5. Data concerning only five of the selected samples are presented, but such data may be considered as representative of each one of the coalfields. However, it should be noted that more than 1,500 chemical analyses were made in the course of the investigation.

As a result of such tests it appears that coals from Fields I, II, III and IV (Table 1) may be classified in accordance with A.S.T.M. standards as high volatile coals, Group B, because the content of volatile matters varies between 38% and 42%. Such coals, therefore, belong to the Group occasionally used for the production of coke. Nearly all these coals are high in oxygen and demand costlier heating and higher coking temperatures than the superior grade coals. They may, therefore, be compared with the Utah coals, employed by some steel plants in the United States, as for instance the Fontana Steel Plant, with about 10% in oxygen and more than 40% in volatile matters, over dry basis. (1)

Coals sampled from the Fields V, VI and VII (Tables 2 and 3) have been classified from medium- to high-volatile coals, Group A, with volatile matter content varying from 30% to 38%. A considerable part of the analyses shows a volatile content of over 30%, and therefore most of the tested samples are of high-volatile coals, Group A, especially in view of the fact that the ash content is generally over 5 to 6%. For that reason those coals belong to the group which, alone or blended with high-volatile coals, is commonly used for making coke.

Finally, the coals sampled from Field VIII (Table 4) are classified as low- to medium-volatile coals, seldom used alone for the production of coke, due to their relatively high pressures. Such coals are, however, commonly employed in admixtures with high-volatile types.

Thus the Paz de Río-Sogamoso area has coals of different categories, which present variations from high- to low-volatile contents, within a distance of only 40 km. Most of these coals are high in volatile content, but there is a sufficient gradation which includes low-volatile coals; under such conditions suitable blends can be prepared for making a satisfactory metallurgical coke.

Tables 5 and 6 summarize the calculated reserves of each type of coal, and the average variation in the analysis.

5. PLASTICITY TESTS

Once the proximate analyses gave the required information about the average quality of the coals, plasticity tests were undertaken in order to ascertain the behaviour of the coals under the action of heat and possible carbonization characteristics.

Table 1

PROXIMATE ANALYSIS OF HIGH-VOLATILE CLASS B COALS— FIELDS NOS. I, II, III AND IV

Field No:	*% H_2O*	*% V.M.*	*% Fixed carbon*	*% Ash*	*% Sulfur*	*Average heating value*
I	1.00	38.60	53.02	7.38	0.74	13,700 B.T.U.
	1.02	41.56	51.88	5.54	1.27	
	1.98	41.06	51.28	5.68	1.04	
	1.60	41.20	51.30	5.90	1.00	
	1.22	39.08	52.80	6.90	0.77	
II	3.24	39.62	52.88	4.26	0.77	13,000 B.T.U.
	2.03	44.44	50.49	3.04	0.74	
	1.17	44.08	50.85	3.90	0.84	
	1.90	38.58	56.55	2.79	0.84	
	3.76	39.90	50.10	6.24	0.83	
III	1.02	43.21	50.86	4.91	0.90	13,500 B.T.U.
	1.02	39.26	51.69	8.03	1.84	
	0.49	41.82	53.36	4.33	1.27	
	1.86	40.54	52.99	4.61	1.22	
	0.51	42.52	52.25	4.72	1.06	
IV	1.42	42.16	50.86	5.56	0.93	12,500 B.T.U.
	3.79	39.97	47.98	5.22	1.47	
	4.80	38.88	45.88	3.16	0.64	
	3.36	39.72	49.76	7.16	0.90	
	2.44	41.09	51.49	4.98	0.92	

Table 2

PROXIMATE ANALYSIS OF HIGH-VOLATILE CLASS A COALS—FIELD NO. V

Sample	% H_2O	% V.M.	% Fixed carbon	% Ash	% S	% H_2	% C	% N	% O_2	B.T.U.	Ash			Expansion index [d]
											Td [a]	Ta [b]	Tf [c]	
I														
Coal dried in the air........	1.2	35.3	58.0	5.5	1.1	5.6	79.1	1.8	6.9	14,250	1,354	1,438	1,499	8
Run-of-mine coal...........	2.0	35.0	57.6	5.4	1.1	5.6	78.5	1.8	7.6	14,150				
Moisture-free coal..........		35.7	58.7	5.6	1.1	5.5	80.1	1.9	5.8	14,430				
Moisture- and ash-free coal..		37.8	62.2		1.2	5.8	84.8	2.0	6.2	15,280				
II														
Coal dried in the air........	1.3	37.1	56.0	5.6	0.8	5.5	79.3	1.7	7.1	14,280	1,449	1,488	1,599	7½
Run-of-mine coal...........	2.6	36.6	55.3	5.5	0.8	5.6	78.3	1.7	8.1	14,090				
Moisture-free coal..........		37.6	56.7	5.7	0.8	5.5	80.4	1.7	5.9	14,470				
Moisture- and ash-free coal..		39.8	60.2		0.8	5.8	85.2	1.8	6.4	15,330				
III														
Coal dried in the air........	1.1	35.2	50.6	13.1	2.1	5.2	71.9	1.5	6.2	13,110	1,438	1,499	1,560+	4
Run-of-mine coal...........	2.1	34.9	50.1	12.9	2.1	5.3	71.2	1.4	7.1	12,990				
Moisture-free coal..........		35.6	51.2	13.2	2.1	5.1	72.7	1.5	5.4	13,260				
Moisture: and ash-free coal..		41.0	59.0		2.4	5.9	83.8	1.7	6.2	15,280				
IV														
Coal dried in the air........	1.4	35.8	51.2	11.6	1.0	5.2	72.9	1.4	7.9	13,130	1,438	1,527	1,599+	5½
Run-of-mine coal...........	2.2	35.4	50.9	11.5	1.0	5.2	72.2	1.4	8.7	13,020				
Moisture-free coal..........		36.3	51.9	11.8	1.0	5.1	73.9	1.4	6.8	13,310				
Moisture- and ash-free coal..		41.4	58.9		1.1	5.8	83.7	1.6	7.8	15,090				

[a] Td—Initial deformation temperature.
[b] Ta—Softening temperature.
[c] Tf—Fluidity temperature.
[d] Free swelling index.

Table 3

ANALYSIS OF THE ASH IN COAL FROM FIELD NO. V

Sample	% SiO_2	% Al_2O_3	% Fe_2O_3	% TiO_2	% CaO	% MgO	% SO_3	% K_2O+Na_2O	% Losses by calcining
I..................	53.5	25.4	15.9	1.3	1.2	0.7	0.9	0.8	0.3
II...................	58.7	25.9	11.1	1.2	0.9	0.5	0.5	0.9	0.3
III..................	65.2	25.2	6.2	1.4	0.3	0.5	0.3	0.7	0.2
IV	67.1	23.9	4.0	1.3	2.1	0.4	0.4	0.6	0.2

Table 4

PROXIMATE ANALYSES OF TYPES OF COAL, FIELDS VI, VII AND VIII

Type of coal Field No.	% H_2O	% V.M.	% Fixed carbon	% Ash	% Sulfur	Average heating value
High-volatile Class A.......VI	0.82	33.72	57.12	8.34	0.75	14,600 B.T.U.
	0.42	32.00	56.38	11.00	0.76	
	0.61	35.99	52.46	10.94	0.76	
	0.49	32.91	57.86	9.74	1.06	
	0.77	34.76	53.65	10.82	1.03	
High-volatile Class A.......VII	0.87	32.71	61.47	4.95	0.78	
	0.73	33.05	59.06	7.16	0.73	
	1.63	31.93	60.28	6.16	0.66	
Low-volatile.............VIII	0.33	18.81	77.39	3.37	0.58	15,000 B.T.U.
	0.60	18.52	76.14	4.74	0.60	
	0.67	21.47	74.59	3.27	0.71	
	0.90	21.06	72.02	6.02	0.78	
	0.91	19.44	74.93	4.72	0.42	

The Gieseler Plastometer was used for this work, with the modification made by the Bureau of Mines of the United States, which seems to be the simplest device of this type (2). Express reference is made to the corresponding specification of the A.S.T.M., which describes this device (3) and which explains in detail the instructions required to carry out the measurements.

The degree of fluidity of typical samples of each one of the fields studied, is demonstrated by Figures 2 to 16. The same data are included in Table 7.

Fig. 2: La Chapa, Seam I, JC 422

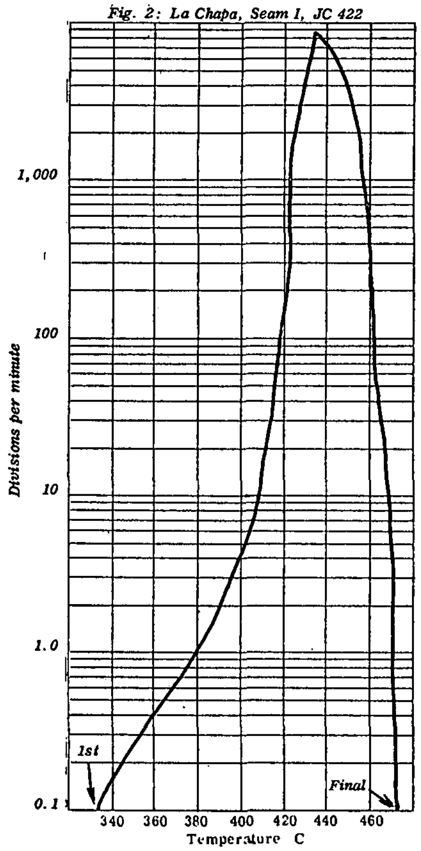


Fig. 3: La Chapa, Seam II, JC 425

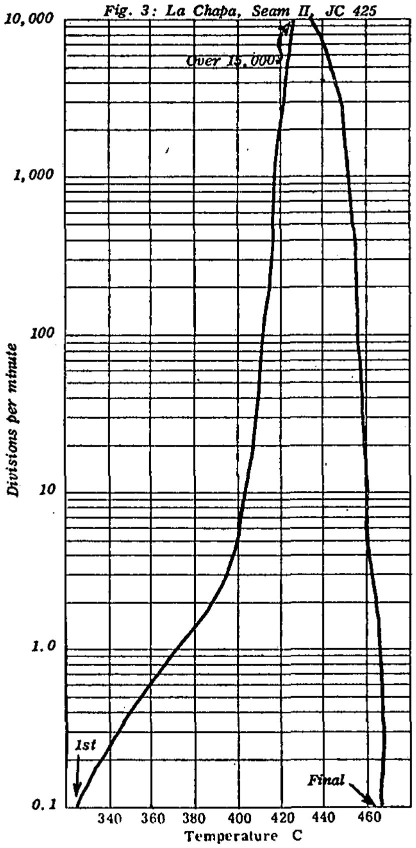


Fig. 4: La Chapa, Seam III, JC 426

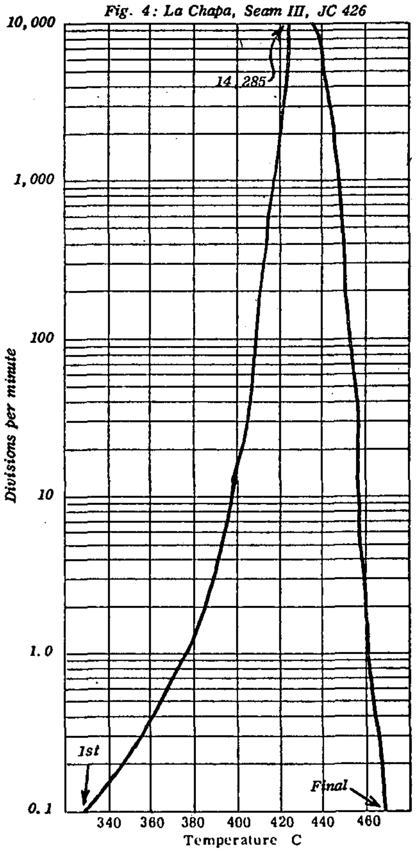


Fig. 5: La Chapa, Seam IV, JC 437

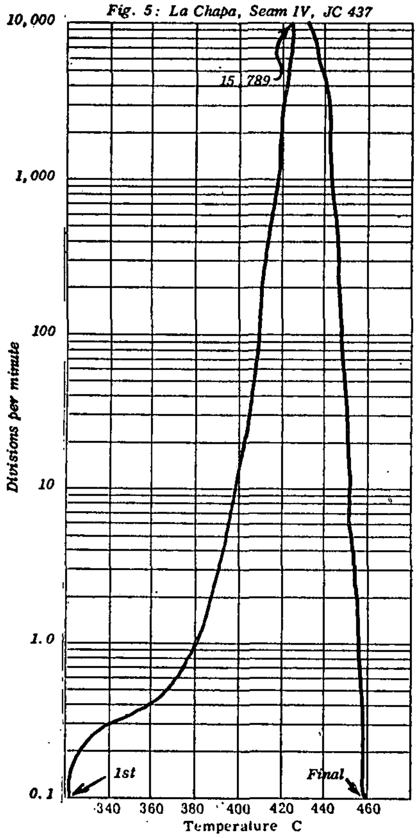


Fig. 6: La Chapa, Seam V, JC 441

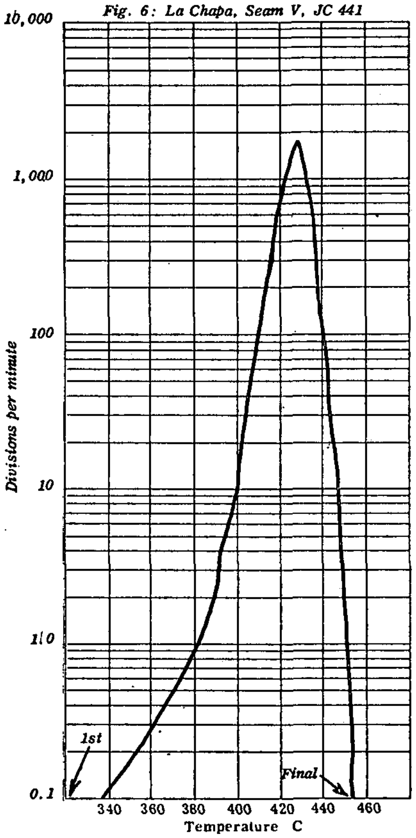


Fig. 7: La Chapa, Seam VI, JC 449

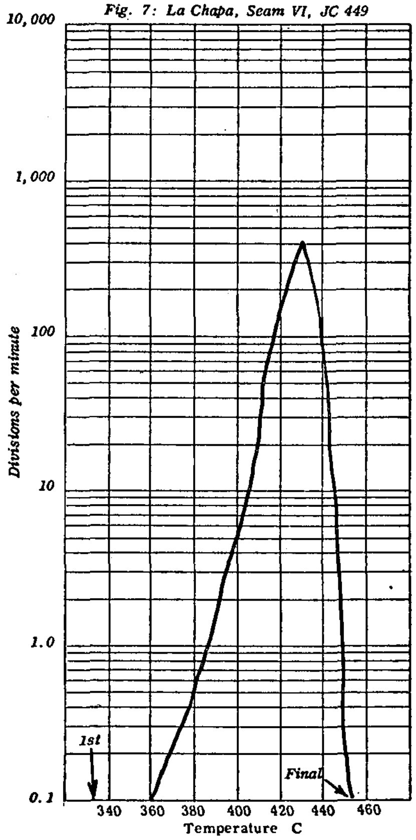


Fig. 8: La Chapa, Seam VII, JC 447

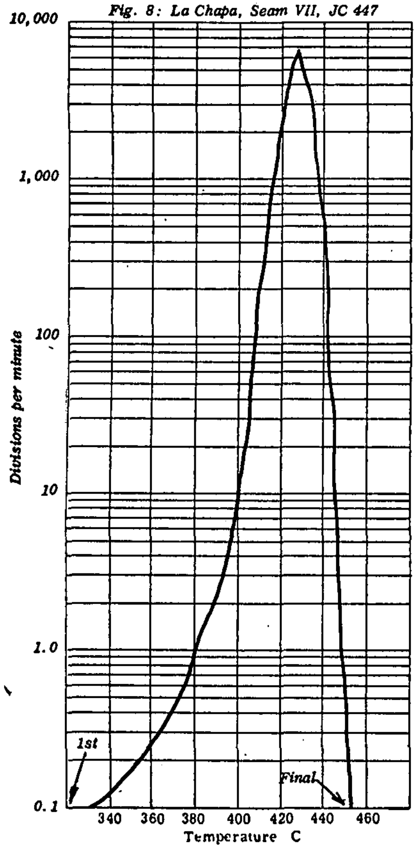


Fig. 9: Socha Viejo, Blend No. 273

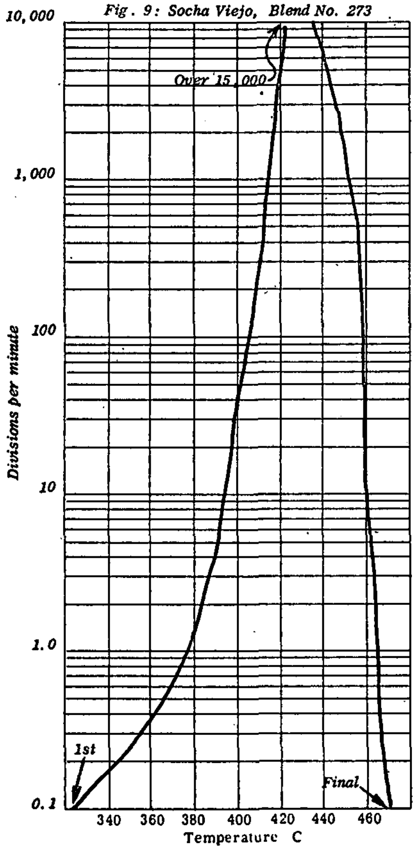


Fig. 10: Topaga Seam A, JT 230

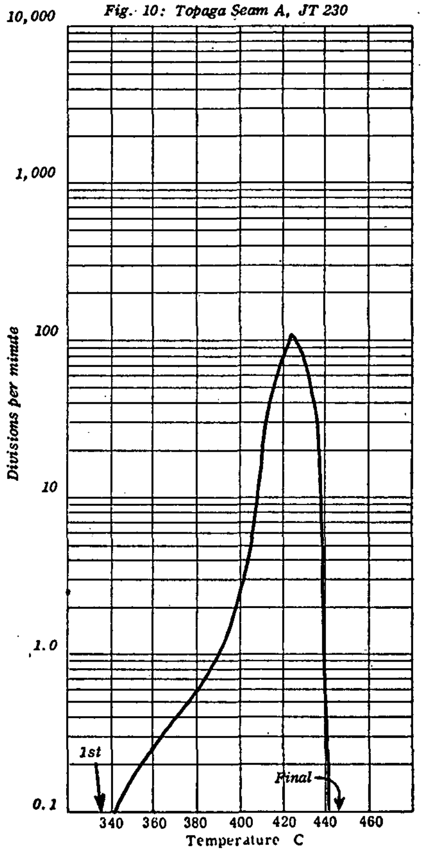


Fig. 11: Tópaga, Seam E, JT 231

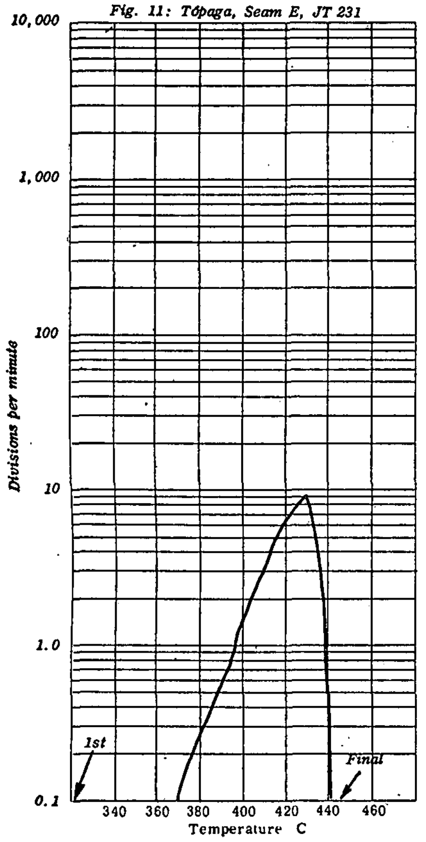


Fig. 12: Blend No. 306

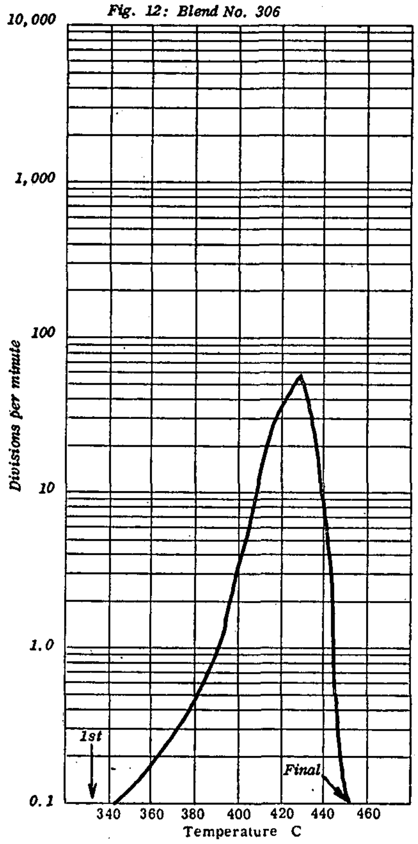


Fig. 13: Socota

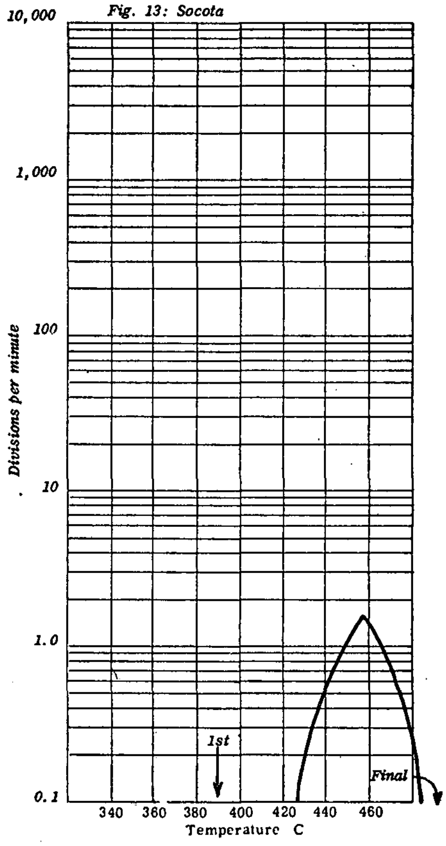
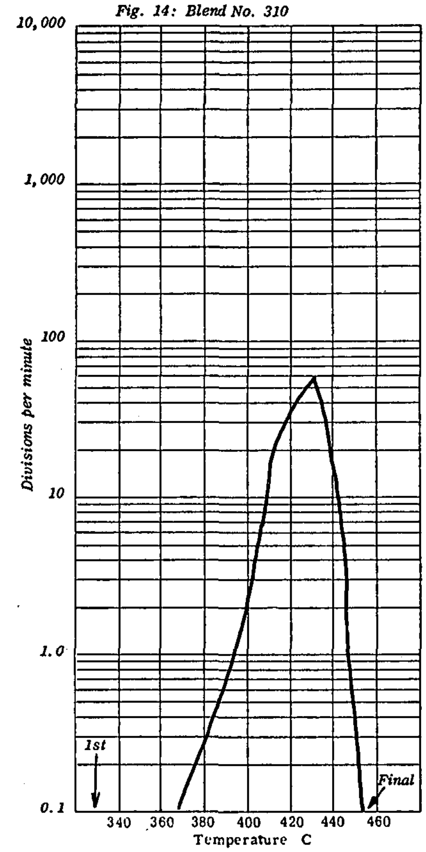


Fig. 14: Blend No. 310



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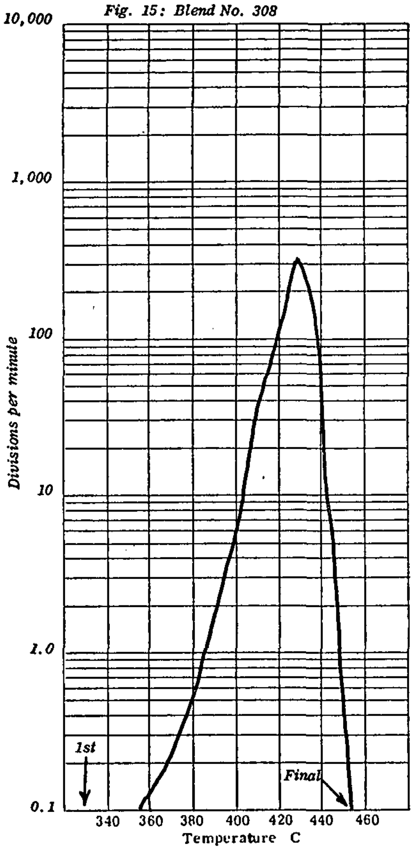
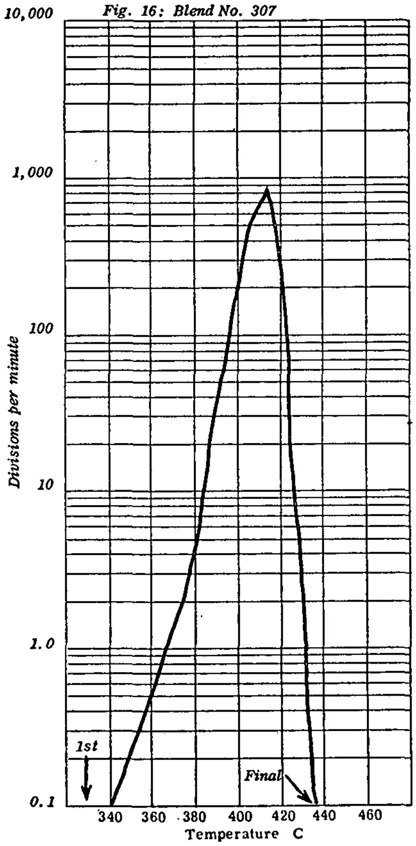


Fig. 16: Blend No. 307



The plasticity characteristics of the tested coals behave well in accordance with the generalizations made by Soth and Russell. (4) That is to say that the maximum fluidity increases in direct correlation with the percentage of volatile matter, until the high-volatile coals of Group A are reached; these coals attain the maximum degree of fluidity. Thus the samples from Field VIII (Figure 13), appear to have a much lower fluidity than the coal samples from Fields V and VI (Figures 2 to 9). Within this latter group the opposite occurs; as the volatile matter increases, the fluidity decreases (Figures 10, 11, 12). Figures 14, 15 and 16 demonstrate the measurements of maximum fluidity of blends of high-volatile Group B coals with those of Class A, and such Figures are illustrative of the progressive increase registered in the maximum fluidity as greater portions of high-volatile coals of Class A are added to those of Group B. Thus, these Figures represent blends of Group B coals with 10%, 30% and 50% respectively of high-volatile coals, Group A; and the maximum fluidity varies from 35.9 divisions per minute in the 10% mixture, to 857 divisions per minute in the mixture made with 30% high-volatile Group A coals.

The data also shows the variations in the plasticity temperatures to be well in accordance with the indications made by Russell. (4)

Consequently, in a final analysis, it may be concluded that these coals behave very similarly to the American coals on which Russell made his investigations and experiments. It is on the basis of the Russell experiments that the foregoing generalities have been stated.

A Koppers expansion movable-wall oven was used in this research work to determine the expansive characteristics of the coals. The Belencito Laboratory oven chamber measures 12 in. x 28 in. x 42 in., having a capacity of 8 cu. ft., that is to say, a load capacity of 400 lb. of coal with a charge density of 50 lb. per cubic foot.

Table 5

COAL RESERVES IN THE AREA

Reserves	*High-volatile, Group A*	*High-volatile, Group B*	*Low-volatile*	*Total*
Proven	8,200,000	8,900,000	700,000	17,800,000
Probable	64,300,000	72,500,000	1,000,000	137,800,000
Possible	127,500,000	1,608,600,000	28,300,000	1,764,400,000
TOTAL	200,000,000	1,690,000,000	30,000,000	1,920,000,000

Table 6

AVERAGE PROXIMATE ANALYSIS OF COALS, BY CLASSIFICATION

Analysis	*High-volatile, Group A*	*High-volatile, Group B*	*Low-volatile*
Moisture	0.5–3.0	1.0–4.0	0.5–2.0
Volatile matter	30–38	38–42	18–25
Fixed carbon	50–63	44–55	66–76
Sulfur	0.7–1.8	0.7–1.6	0.4–1.0
Ash	6.0–15	6.0–10	3.0–8.0
Heating value	13,500–14,500 B.T.U.	12,500–13,500 B.T.U.	14,000–15,000 B.T.U.

Table 7

DATA ON PLASTICITY TESTS OF VARIOUS TYPES OF COALS

Figure No.	*Type of coal*	*Temperature of the first movement*	*Maximum fluidity*	*Temperature of the final movement*	*Maximum fluidity*
2.	High-volatile, Class A	333° C.	433° C.	472° C.	833 Div/Min.
3.		323° C.	430° C.	465° C.	×15,000 "
4.		327° C.	429° C.	465° C.	14,285 "
5.		320° C.	430° C.	457° C.	×15,000 "
6.		320° C.	428° C.	452° C.	1,775 "
7.		352° C.	430° C.	453° C.	427 "
8.		322° C.	427° C.	451° C.	6,977 "
9.		326° C.	422° C.	470° C.	×15,000 "
10.	High-volatile, Class B	338° C.	423° C.	442° C.	109.1 "
11.		322° C.	430° C.	445° C.	9.0 "
12.		333° C.	430° C.	452° C.	55.9 "
13.	Low-volatile	390° C.	457° C.	498° C.	1.5 "
14.	Blends of high-volatile, A and B coals	333° C.	430° C.	452° C.	55.9 "
15.		331° C.	430° C.	453° C.	292 "
16.		333° C.	432° C.	455° C.	857 "

To carry out the experiments, the pertinent instructions contained in the corresponding specifications of the A.S.T.M. were carefully observed. The coal was air-dried until less than 1% of humidity was attained, and during the whole coking process a temperature of 1,370° C. was maintained. The operation ended when the centre of the charge was at 900° C. All the precautions relating to the density of the charge, degree of pulverization and heating velocity were taken into account.

Typical average curves of the pressures developed in the furnace during the carbonization period, are shown in Figures 17, 18, 19 and 20, and Table 8 points out the maximum pressures obtained, the coking temperatures and the densities of charge.

The preceding results indicate that in no case did the pressures developed exceed the limit of 1.50 lb. per square inch, which is considered as dangerous for the walls of industrial ovens. The developed pressures generally go from 0.40 and 1.00 lb. per square inch for the high-volatile Group A coals, and pressures of 0.60 to 1.10 lb. per square inch belong to the coals of Group B.

In the case of the low-volatile coals, the pressures developed are naturally a little stronger.

6. COKING TESTS

Characteristics of the cokes obtained

It is generally known that the cokes fabricated in the movable-wall oven are not identical but only similar to those produced in ordinary industrial ovens. (5) Nevertheless, there is no doubt that the physical tests accomplished with such movable-wall oven cokes

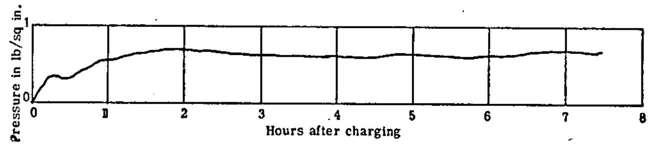


Figure 17

Curve of the pressure developed during coking
Coal from Mine No. 1.

Coking data:

Density of the charge................47.6 lb./cubic foot
Moisture................................1.0 per cent
Temperature of the smoke conduit........2462°F (1349.8°C)

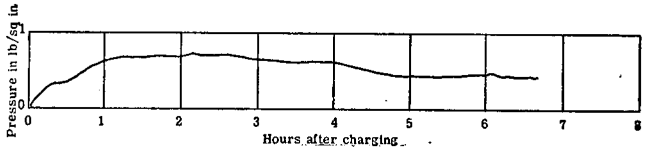


Figure 18

Curve of the pressure developed during coking
Coal from Mine N° 3

Coking data:

Density of the charge..................50.2 lb./cubic foot
Moisture................................0.4 per cent
Temperature of the smoke conduit......2449°F (1342.6°C)

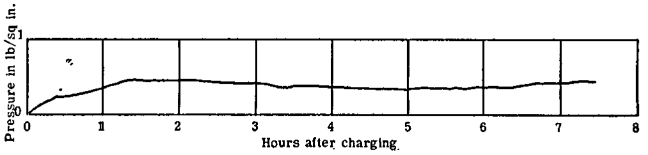


Figure 19

Curve of the pressure developed during coking

Coking data:

Density of the charge..................47.4 lb./cubic foot
Moisture................................0.2 per cent
Temperature of the smoke conduit......2470°F (1354.3°C)

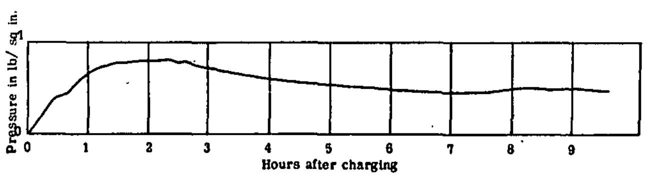


Figure 20

Curve of the pressure developed during coking
Coal from Mine No. 6

Coking data:

Density of the charge.................50.3 lb./cubic foot
Moisture.............................0.3 per cent
Temperature of the smoke conduit......2470°F (1354.3°C)

do indicate the quality of the coke which can be produced with the particular coal on a large scale. The screening, shatter, tumbler and apparent density tests, generally produce lower indices for cokes fabricated in movable-wall ovens, than for cokes fabricated in industrial ovens. The magnitude of the eventual variations of such characteristics do not substantially alter the anticipated results and, moreover, are subject to modifications by means of the numerous resources which are a matter of current practice.

On the other hand, by reproducing, in the movable-wall ovens, conditions as similar as possible to those prevailing in industrial ovens, analogous results are probable. For this purpose, new carbonization tests were undertaken with coals considered as of a major interest. The movable-wall of the oven was fixed, the coal crushed to ⅛ in. or 1 in. sizes, according to requirements, and the samples damped until a 5% humidity index was reached, so as to reproduce as nearly as possible the prevalent charging conditions of the industrial ovens.

Table 9 demonstrates the physical characteristics of the cokes fabricated with high-volatile Classes A and B coals, during the expansion tests made without fixing the movable wall of the oven. As was to be expected, the results showed clearly that the cokes made from high-volatile Class A coals have good physical characteristics and are apt for metallurgical use, while the cokes fabricated with Class B coals are of poor physical characteristics and cannot be employed for the metallurgical ends considered here.

The physical properties of the cokes fabricated with mixtures of high-volatile coals, Class A, from Fields V and VI, with coals of the group B, are shown in Table 10.

As the percentage of the former coals in the mixtures is increased, the chemical characteristics of size, shatter and tumbler indices are progressively improved, and eventually cokes useable for metallurgical ends are obtained.

The proximate analyses of such cokes are presented in Tables 11 and 12. It will be observed that when high-volatile Class A coals are employed, the ash content is high and variable. Considering the origin and the characteristics of those fields of "drift coals", the ash content variations are easily understood, and by the same token the washing of such coals is recommended as a condition for obtaining a uniform product. The washing does not present any difficulty whatsoever, and as described elsewhere, very satisfactory yields, superior to 85% and ash contents under 8%, are obtained when the coals are washed at densities of the order of 1.45 to 1.55.

Some results of coking tests with mixtures of washed coal samples from the various seams in Field V, providing an example of similar tests with washed coals of different fields, are shown in Table 13. For such experiments the coal fractions of under ¼ in. were eliminated and the washing was done at a density of 1.45. The clean coal was divided in two fractions which were crushed to sizes of ⅛ in. and 1 in., respectively. Coking tests were undertaken with each fraction at temperatures of 1,100° C. for the purpose of determining the most suitable size and temperature for the fabrication of a good metallurgical coke. Likewise, tests were made with mixtures of washed coal and unwashed fractions of under ¼ in. All these tests, as the preceding ones, were accomplished after fixing the movable wall of the oven and wetting the coal up to a humidity index of 5%, as previously explained.

It is a well-known fact that the evaluation of a good coke is always somewhat arbitrary, because it depends largely upon the economic factors which influence the operation of the blast furnace in every particular case. Thus, cokes having a high ash content may give better economic results when the washing of the coals determines low yields after a certain ash limit, further washing being a costly operation, apart from the considerable volume of refuse. The iron ore composition, its physical conditions, prior sintering, the slag volume, etc., are interdependent factors which combine with those related to the yield in the coal-washing, the uniformity and physical properties of the coke, making the economic analysis a very complex matter, to which it is quite difficult to apply general rules.

Table 8

EXPANSION PRESSURES OF THE COAL

Field No.	*Coking temperature*	*Density of the charge lb./cu. ft.*	*Maximum pressure lb./sq. inch*
I................	1,329° C.	51.2	1.06
II................	1,332° C.	50.3	0.75
I................	1,336° C.	51.1	0.68
V................	1,339° C.	52.6	0.49
V................	1,375° C.	53.1	0.45
V................	1,300° C.	51.5	0.87
VI................	1,357° C.	51.70	0.61
VI................	1,370° C.	51.59	0.69
VI................	1,351° C.	51.79	0.64

Table 9

PHYSICAL PROPERTIES OF COKES OBTAINED FROM HIGH-VOLATILE A AND B COALS

Class A						Class B					
Classification by sizes		Rupture Index	Abrasion			Classification by sizes		Rupture Index	Abrasion		
% over 2 in.	% below ¼ in.	% over 2 in.	Stability % over 1 in.	Hardness % over ¼ in.	Apparent Density	% over 2 in.	% below ¼ in.	% over 2 in.	Stability % over 1 in.	Hardness % over ¼ in.	Apparent Density
80.4	1.9	43.5	30.2	69.6	1.148	54.2	5.0	32.9	8.7	72.0	0.911
83.99	1.8	55.0	48.0	72.7	0.983	64.3	3.8	25.8	11.4	57.9	0.835
80.41	2.5	48.6	33.6	68.3	1.000	70.2	3.7	29.9	8.4	58.9	0.809
81.10	2.4	47.6	41.8	71.5	1.083	69.7	12.7	47.8	25.9	35.1	0.915
80.90	2.9	58.5	29.4	63.2	1.022	66.9	2.54	37.9	38.07	69.4	1.203

Table 10

BLENDS OF HIGH-VOLATILE A AND B COALS

Classification by sizes		Rupture index	Abrasion		
% over 2 in.	% below ¼ in.	% over 2 in.	Stability % over 1 in.	Hardness % over ¼ in.	Apparent density
83.2	1.7	50.0	26.2	61.4	0.930
83.2	2.71	54.7	26.7	57.6	0.952
82.6	2.56	45.7	40.1	66.6	0.915
80.9	2.59	49.9	35.7	65.7	0.758
81.9	2.24	50.91	26.53	61.85	0.948

Table 11

PROXIMATE ANALYSIS OF COKES OBTAINED FROM HIGH-VOLATILE A AND B COALS

Class A

% H_2O	% V.M.	% Fixed carbon	% Ash	% Sulfur
0.48	1.00	86.18	12.34	0.48
0.12	1.16	83.60	15.12	0.76
0.34	1.46	81.74	16.46	1.31
0.99	1.93	82.42	14.66	0.74
0.36	1.22	88.78	9.64	0.53

Class B

% H_2O	% V.M.	% Fixed carbon	% Ash	% Sulfur
1.77	2.41	85.28	10.54	1.08
1.19	2.39	84.86	11.56	0.89
1.79	2.13	84.56	11.52	0.99
1.54	2.50	84.94	11.02	1.05
0.85	2.65	83.84	12.66	0.84

Table 12

PROXIMATE ANALYSIS OF COKES OBTAINED WITH BLENDS OF HIGH-VOLATILE A AND B COAL

% H_2O	% Volume of matter	% Fixed carbon	% Ash	% Sulfur
1.83	2.65	81.42	14.10	1.06
1.67	1.43	85.50	11.40	0.97
1.58	1.44	85.94	11.04	1.12
1.54	2.50	84.94	11.02	1.05
0.74	2.16	83.96	13.14	0.82

It is generally considered, on the basis of various tests, that a good quality coke must have more than 70–80% over 2 in.; shatter indices of more than 60–70%; stability factors of more than 40–50%; hardness coefficients of nearly 60–70%. Inferior results of such tests would be sizes of under 50%; shatter indices under 40% and stability indices of under 15%. But there are no precise limits for the variation of those coefficients, and the above-mentioned percentages are only approximate.

It follows, therefore, that as shown in Table 13, the variations observed in the indices of the cokes produced during the tests remain within limits which are considered satisfactory. For instance, the size is superior to 80%; the shatter indices vary from 55 to 75% with an average of 60%; the stability factor oscillates between 30 and 40% with an average of 35%, and the hardness indices vary between 60 and 70%.

From the above-mentioned results it follows that the stability factor is the only index probably under the desirable level. Nevertheless, as previously explained, the coke physical tests made in a movable-wall oven ordinarily show results inferior to those produced by industrial ovens. In view of this fact, and considering that there are in practice numerous resources for improving certain characteristics of the coke, the general results obtained from the test may be accepted as satisfactory. Furthermore, it should be observed that the sulfur and ash contents average less than 1% and 12%, respectively, which percentages remain within the limit of adequate specifications.

Table 13

COKING TESTS—BLENDS FROM THE SEAMS OF "LA CHAPA" MINE

Tests				Coal						Coke								
				Analysis			Densities		Yield	Analysis			Physical tests					Aspect
No.	Temperature	Origin and size of the coal		H₂O	Ash	Sulfur	Cu. ft.	Charge		H₂O	Ash	Sulfur	Density	Sifting	Shatter	Tumbler		
		Seams I, IV and VI																
117	1,091		1″	5.88	7.72	1.24	20.515	47.59	70.95	1.75	10.80	0.86	0.926	91.72	63.82	31.80	61.35	Very large, shaly
118	1,095		⅛″	5.79	8.20	1.14	16.99	47.78	65.48	2.78	11.68	0.99	0.913	87.48	73.06	36.13	65.38	Very good, homogeneous spongy at the centre
125	1,371		1″	6.27	8.02	1.21	20.69	47.65	70.50	16.45	10.96	1.07	0.740	79.61	57.75	29.73	57.20	Rather good, broken shaly
126	1,321		⅛″	6.20	7.98	0.99	17.45	43.69	66.47	19.75	11.60	0.90	0.900	79.72	52.27	33.18	62.45	Good, homogeneous, somewhat broken
		Seams I, IV and VI																
127	1,351	With the fines	1″	6.06	7.94	1.04	20.165	48.22	69.45	4.53	11.30	0.97	0.840	80.75	60.56	31.97	67.95	Rather good, irregular, broken, shaly
128	1,348		⅛″	6.58	8.42	0.84	17.710	43.32	66.31	13.15	11.86	0.78	0.824	78.72	58.78	32.83	60.13	Good, fine grain
M.W. 131 [a]	1,373		⅛″	6.51	8.50	1.09	17.50	42.33	68.78	16.45	11.62	0.89	0.894	77.00	49.86	31.15	54.70	Good, fine grain, broken
136	1,143		⅛″	5.43	9.26	1.10	21.545	42.27	68.93	17.92	12.40	1.01	0.830	85.48	58.70	32.68	62.98	Very good, average grain, compact homogeneous
		Seams I, IV and V																
130	1,357	With the fines	⅛″	3.91	8.84	1.52	19.365	43.25	68.14	3.27	12.07	1.09	0.891	78.50	59.11	36.60	63.78	Good, rather fine grain, broken
M.W. 132 [a]	1,376		⅛″	4.70	9.18	1.28	18.105	47.57	70.98	17.36	12.54	1.18	0.865	72.70	39.53	36.03	63.68	Good, rather fine grain, slightly broken
137	1,176		⅛″	5.80	9.68	1.32	20.180	42.94	69.34	3.27	12.84	1.16	0.909	84.13	64.03	41.80	71.55	Very good, fine grain, homogeneous

[a] M.W. = Tests made with moving wall.

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Production of Metallurgical Coke in Mexico

SALVADOR CORTÉS OBREGÓN

The Rosita coking plant produces about 32,000 tons per month of coke (at 900° C) in 65 Willputte underjet ovens. The temperature of the gas ducts is 1,250° C, and carbonization time is 17 hours. Forty-five of the ovens are 40 ft. 7⅜ in. long by 11 ft. 10 in. high and 18 ft. ⅜ in. wide; the other twenty are 40 ft. 2 in. long by 11 ft. 10 in. high and 16 ft. ¾ in. wide.

YIELDS PER METRIC TON OF DRY COAL

Standard coke	75%
Coke breeze	6.6%
Gas	10,150 cu. ft.
Ammonium sulphate	7.00 kg.
Pitch	8.00 gal.
Light oil	2.70 gal.

No by-products are processed in the plant.

Laboratory data are given below on the coal carbonized in the Rosita coking plant, with the chemical analysis and results of physical resistance tests of the coke obtained.

WASHED COAL INTENDED FOR COKING

Proximate analysis	*Per cent*
Volatile matter	25.8
Fixed carbon	59.2
Ash	15.0
Ultimate analysis	
Carbon	73.2
Hydrogen	4.0
Oxygen	5.8
Nitrogen	1.2
Sulfur	1.2

PLASTICITY (GIESELER PLASTOMETER)

Initial softening temperature	412°C
Temperature of maximum fluidity	445°C
Temperature of solidification	529°C

TESTS IN MOVABLE-WALL OVENS

Sole-heated oven	(*U.S. Bureau of Mines*)
Apparent density	0.902
Contraction %	13.0

VERTICAL-SLOT OVEN (*U.S. Bureau of Mines*)

Apparent density	0.824
Expansion pressure	0.086 kg./sq. cm.
Coking time	7.8 hours

ANALYSIS OF THE COKE

(Production August 1952)

Volatile matter %	0.75
Fixed carbon %	79.86
Ash %	19.39
TOTAL	100.00%
Sulfur %	0.9
Phosphorus %	0.03

PHYSICAL RESISTANCE OF THE COKE

Tumbler test A.S.T.M	*Differential %*	*Cumulative %*
Over 2.000″ in screen	3.0	3.0
″ 1.500″ ″ ″	23.6	26.6
″ 1.060″ ″ ″	33.1	59.7
″ 0.530″ ″ ″	14.3	74.0
″ 0.265″ ″ ″	2.2	76.2
Through .265″ ″ ″	23.8	100.0

Stability factor	59.7%
Hardness factor	76.2%

PHYSICAL RESISTANCE OF THE COKE

Shatter test A.S.T.M	*Differential %*	*Cumulative %*
Over 2.00″ in screen	62.8	62.8
″ 1.50″ ″ ″	17.5	80.3
″ 1.00″ ″ ″	10.4	90.7
″ .50″ ″ ″	3.9	94.6
Through .50″ ″ ″	5.4	100.0

Shatter loss	37.2

Note: The earlier tests and analyses correspond to an average sample taken over a month of production.

Some laboratory experiments indicate that the coal from the Sabinas coalfield has a strong expansion pressure (over 4 lb. per sq. in. in the vertical slot, Bureau of Mines, oven, with 2% moisture content and 53.1 lb. per cu. ft. of apparent density); others indicate that the same Sabinas coals can safely be coked for the by-product ovens always providing that

there is a careful check on the apparent density (in the same vertical slot, Bureau of Mines, oven, pressures of 1.3 and 1.7 lb. per sq. in. were obtained, considered safe for the ovens, with charges of 6.3% and 5.9% of moisture and 47.46 and 48.66 lb. per cu. ft. of apparent density, respectively). It should be pointed out, however, that the Rosita coking plant has worked for nearly thirty years with coals from the Sabinas coalfield (it has in fact tried using coal from all the mines in operation, including coal from the Saltillito deposit) without appreciable damage to the oven walls.

Palau coal (Sabinas coalfield), washed (to 14% ash) in an experimental plant and submitted to a plasticity test in a Gieseler plastometer, melted at 433° C, reached a maximum fluidity of 540 divisions at 469° C and solidified at 506° C.

Approximately 55% of coke production at Rosita is for non-ferrous metal foundries and sundry consumers, the remaining 45% being consumed by an iron and steel plant. The rest of the coke used by the Mexican steel industry comes from beehive ovens, as may be seen from the following table:

COKE PRODUCTION IN BEEHIVE OVENS

	Agujita Sabinas coal	*Agujita AHMSA*	*Cloete*	*Palau*	*Esperanzas*	*Total*
Number of ovens	170	290	240	59	49	808
Carbonization time, hours	72	72	48	48	48	—
Charging capacity	4.65	5.0	3.0	4.0	4.5	—
Monthly production of coke (May 1950) tons	3,600	5,000	4,600	2,300	2,200	17,700

The production shown in this table is not the maximum with this time for carbonization as all the ovens were not operating at that date. At present about 700 ovens are working with an annual production approximating 200,000 tons. Furthermore, since 1951 two other batteries of beehive ovens have been working, one in Barroterán and the other in El Hondo with an (estimated) joint monthly production of 4,000 tons, so that total annual production of coke in beehive ovens may be estimated at 250,000 tons.

Coke production in beehive ovens, coming from several sources, with various firms using different techniques, naturally lacks the uniformity and quality to be desired. In order to replace and increase the production now obtained from beehive ovens, a carbonization plant with by-product recovery is being built, at a total investment of $10 million, in the city of Monclova, adjacent to the iron and steel plant of Altos Hornos de México S.A. The new coke plant is being built by Koppers in Germany and has the general features given below:

Capacity: 2,200 tons a day, using washed dry coal (15% ash).

Production: 1,600 tons of coke a day, 48,000 tons a month. Two batteries of 57 ovens in groups of 28 and 29, that is, 114 ovens in all.

Dimensions: 14.50 m. long by 4,000 mm. high by 420 and 480 mm. wide. Five charging doors in each oven, 16 tons being charged per oven with a carbonization period lasting from 18 to 20 hours.

Type of oven: Koppers Compound Circulation.

The temperature in the gas ducts will be, for carbonizing in:

16 hours	1,400° C
18 "	1,350° C
20 "	1,300 °C
22 "	1,250 °C
24 "	1,200 °C

By-products

Benzolics, 90% to 79.5°C	Amount not given
Crude benzol, 1% toluene	" " "
Crude toluene	" " "
Solvent naphtha of 120 to 190°C	" " "
Solvent naphthaline oil, 91% from 190 to 235°C	" " "
Pitch	" " "

Sulfuric acid, 20 tons a day (for making ammonium sulphate)

Plants:

Pitch, 100 tons of crude pitch a day to obtain: light oil, medium oil, washing oil, anthracene oil, ordinary tar.

Gas, 700,000 normal cubic metres per day.

Benzol, 25 gr. per normal cubic metre of gas.

About 48% of the gas will be used for heating the ovens.

It is expected that this new plant will be in operation by the year 1954. About 45% of present coke production is consumed in the iron and steel industry, 43% is used in non-ferrous metal foundries and the remaining 12% by smithies and sundry consumers. In future, the share taken by the iron and steel industry will be much higher, as it is developing much more rapidly than the other consumers.

Metallurgical Coke from Chilean Coals

AMÉRICO ALBALA

INTRODUCTION

Before the installation of an integrated steel mill in Chile could be contemplated, one of the first problems to be considered was the making of metallurgical coke from Chilean coals for use in blast furnace operation.

In 1943, when it was decided to make a complete study of this problem, the possibilities were extremely precarious. Up to then, it had been considered in various quarters, in view of certain studies made years ago, that the chances of making metallurgical coke from Chilean coals were very poor. Furthermore, this drawback was one of the reasons which had, up to that time, proved a setback to the idea of making steel in Chile in a plant with sufficient production capacity to meet Chilean requirements. Meanwhile, all the metallurgical coke used in foundries came from foreign sources and the only existing blast furnace in Corral, province of Valdivia, was operating on charcoal.

A brief outline will be given below of the work done so as to record the various phases of the investigation and the process carried to its successful conclusion.

A preliminary study of the coals available in Chile indicated that research had to be directed towards a detailed study of the coking properties of coals from the Lota and Schwager mines.[1] The production of these mines represents more than 80% of Chile's entire coal production, which in 1950 was close to two million metric tons. In view of the magnitude of the steel plant envisaged, only these mines were able to supply the coal in the required amounts. Furthermore, these coals were classified as bituminous "High-Volatile Coals A", according to the American Society for Testing Materials Classification (1). These coals accordingly offered possibilities of favourable results, which was not the case with the lower rank coal, of the sub-bituminous type, which in carbonizing left char as residue.

PRELIMINARY INVESTIGATIONS MADE IN THE UNITED STATES ON PROPERTIES OF CHILEAN COALS

Research involved the study of the following points:

(1) Chemical properties, particularly ash and sulfur content;

(2) Plastic properties;

(3) Washability characteristics;

(4) Physical properties of the coke;

(5) Improvement of the physical characteristics of the coke by blending the Chilean coal with other coals of high coking power.

The first phase of these studies includes tests on the coal from Lota and Schwager mines, without considering its classification by seams. These tests were made in 1943 and 1944. Later on, during the latter part of 1944, the work was directed towards the study of the various seams separately.

I. *Tests with Chilean coals without classification by seams*

1. Samples of Schwager and Lota coals tested by Koppers Co., Seaboard, N. J., in 1943, in the movable-wall experimental oven. This oven has been described by Russell (2).
2. Schwager and Lota coal samples tested on a commercial scale in the Coke Plant of Kaiser Co., Fontana, California. The coke ovens are Koppers-Becker "underjet type".
3. Schwager and Lota coal samples tested in the U.S. Bureau of Mines Experimental Station, Pittsburgh, Pa. in the BM-AGA Pilot Plant.

This equipment has been described in other publications (3) and the results have been published (4, 5). This research was undertaken with the co-operation of Pennsylvania State College.

The purpose of these tests was to obtain comparative data on the coking properties of the Chilean coals compared with the great majority of United States coals, which, in accordance with a complete co-operative programme between the U.S. Bureau of Mines and the American Gas Association had been tested in the same equipment. Data were obtained relative to the properties and yields of coke, gas, light oils, tar, ammonia and products of the fractional distillation of tar and light oils.

In each of the studies shown in Group 1, individual tests of Schwager and Lota coals were made, as well as of blends with North American coals of high coking power (Pocahontas, etc.). All these preliminary studies led to the conclusion that it was perfectly possible to obtain a metallurgical coke of good quality in blends with certain United States coals such as Pocahontas No. 3, with a proportion of these not below 20%.

Furthermore, it was established that the Lota and Schwager coals have coking properties, and so forth, very similar to those of Columbia mine coal, lower Sunnyside-bed, of Utah, which was already being successfully used in the Kaiser Company's plant at Fontana, California (6) and in Provo, Utah.

II. *Tests of Chilean coals by seams*

After these preliminary conclusions it was considered necessary to make more detailed studies of the properties of Lota and Schwager coals in their different seams.

In 1944, the Corporación de Fomento de la Producción of Chile, reached an agreement with the United States Government, under which a mission was sent to Chile to report on Chilean coal mines and coal properties. This mission consisted of Albert L. Toenges and Leon W. Kelly of the U.S. Bureau of Mines, who prepared a study of the Schwager and Lota mines as well as of smaller mines.

For this purpose, they obtained channel samples, without roof and floor impurities, from each of the seams worked in such mines, to be used in coking and washing tests. These samples were tested in the U.S. Bureau of Mines Experimental Station, Pittsburgh, Pa. (7). The results of these tests led to a more exact definition of the research work on Chilean coals for subsequent use in the coke plant of the Compañía de Acero del Pacífico.

Among others, samples were obtained from Alta, Chica and Arriba seams of Lota mine and Nos. 3 and 5 seams of Schwager mine.

Table 1 shows the results of the proximate and ultimate analysis of the coal samples indicated before. This table shows at a glance, that Nos. 5 Schwager seam and Alta and Chica Lota seams have sulfur contents equal to or below 1%, whereas the sulfur content of No. 3 Schwager seam and Arriba Lota seam is 3.3 and 4.5%, respectively, that is, too high for metallur-

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[1] See "Description of Chilean Coals Used in the Steel Industry", by Alfredo González.

gical purposes. Alta and No. 5 seams also have a lower ash content.

The carbonizing tests were made in the BM-AGA equipment mentioned earlier (3, 8). Two steel drums were used for retorts, one 33 cm. (13 in.) and the other 46 cm. (18 in.) in diameter, both being 66 cm. (26 in.) high. The smaller drum had a capacity of 38 to 48 kg. of coal and the other 80 to 90 kg. Samples of 100% coal were used for each of the seams indicated above and in blends with 10% Pocahontas No. 3 coals. Complete results were also obtained on the various by-products, taking advantage of the recovery train which was part of the equipment.

Table 1

PROXIMATE AND ULTIMATE ANALYSIS OF COAL FROM SEAMS 3 AND 5, SCHWAGER MINE, AND THE ALTA, CHICA AND ARRIBA SEAMS FROM LOTA MINE

Channel samples tested by the U. S. Bureau of Mines

Coal	*Proximate analysis, % Dry basis*				*Ultimate analysis, % Dry basis*					*Heating value Cal/Kg.*	*Fusion temperature of ashes °C*
	Moisture	*Vol. mat.*	*Fixed carbon*	*Ash*	*Hydrogen*	*Carbon*	*Nitrogen*	*Oxygen*	*Sulfur*		
Schwager, seam No. 3	2.5	43.9	47.2	8.9	5.5	74.9	1.2	6.2	3.3	7,580	1,040
Schwager, seam No. 5	4.0	42.0	53.2	4.8	5.6	78.6	1.4	8.6	1.0	7,880	1,155
Lota, Alta seam (Pique Nuevo)	4.0	42.9	54.0	3.1	5.6	80.2	1.4	9.0	0.7	8,020	1,090
Lota, Alta seam (Pique Grande)	4.0	40.8	51.5	7.7	5.5	75.3	1.3	9.7	0.5	7,530	1,460
Lota, Chica seam	3.9	42.6	48.8	8.6	5.3	74.4	1.3	9.4	1.0	7,490	1,130
Lota, Arriba seams	2.9	44.9	46.4	8.7	5.4	72.8	1.1	7.5	4.5	7,460	1,230

Table 2 shows the results obtained with the coke from the coal samples of each seam separately. The method used for the shatter and tumbler tests corresponds to the one developed by the Columbia Steel Co. (3, p. 19), which is used for cokes made from poorly coking coals. It is a test which is less severe than the one adopted by the American Society for Testing Materials and, therefore, permits the recording of small variations in the quality of weak cokes.

Table 2

PHYSICAL PROPERTIES OF COKE DERIVED FROM SEAMS 3 AND 5 OF SCHWAGER MINE, AND THE ALTA, CHICA AND ARRIBA SEAMS OF LOTA MINE

Channel samples tested on BM-LAGA equipment of the U. S. Bureau of Mines
(Carbonization temperature = 900°C)

Coal	*Retort diameter in cm.*	*Specific gravity*		*Porosity %*	*Shatter test Cumulative percentages over*				*Tumbler test Cumulative percentages over*			
		Real	*Apparent*		*1½ in.*	*1 in.*	*¾ in.*	*½ in.*	*1½ in.*	*1 in.*	*¾ in.*	*½ in.*
Schwager, seam No. 3	33	1.98	0.74	62.6	28	55	65	81	2	20	30	50
Schwager, seam No. 3	46	1.96	0.79	59.7	25	49	61	80	3	16	28	50
Schwager, seam No. 5	33	1.91	0.73	61.8	27	63	75	88	5	36	52	67
Schwager, seam No. 5	46	1.91	0.75	60.7	29	60	72	86	5	32	47	62
Lota, Alta seam (Pique Nuevo)	33	1.89	0.72	61.9	35	72	82	90	5	40	56	70
Lota, Alta seam (Pique Nuevo)	46	1.88	0.73	61.2	37	69	84	88	4	38	52	64
Lota, Alta seam (Pique Grande)	33	1.93	0.73	62.2	22	53	67	84	3	23	41	61
Lota, Alta seam (Pique Grande)	46	1.91	0.74	61.3	35	60	70	85	5	29	41	58
Lota, Chica seam	33	1.94	0.73	62.4	38	69	74	86	6	35	47	56
Lota, Chica seam	46	1.94	0.76	60.8	48	72	78	87	9	35	43	53
Lota, Arriba seam	33	1.97	0.75	61.9	33	64	73	87	5	27	41	56
Lota, Arriba seam	46	1.97	0.70	64.5	28	54	65	81	3	19	30	50

It should be mentioned that the results indicated are only comparable among themselves or with those from other tests made under the same conditions and with the same equipment.

Here again, this table shows that coke from No. 5 Schwager and Alta Lota seams—and to a lesser degree, from Chica Lota seam—is better than that from other seams, with regard to the +½ in. sizes of the tumbler test. Table 2 does not include the results for blends with Pocahontas No. 3 coal, which is a low-volatile coal. However, the addition of this coal considerably improved the quality of the coke.

In addition to the coking test already mentioned, the samples were also used for making float-and-sink tests. Washability characteristics of Lota and Schwager coals will be described in a different section.

INVESTIGATIONS MADE BY THE COMPAÑÍA DE ACERO DEL PACÍFICO ON PROPERTIES OF CHILEAN COALS

The samples taken by the U.S. Bureau of Mines were, as stated before, obtained from certain parts of the various seams. As a result of the Bureau of Mines' report, it was considered necessary to continue more extensive studies of each seam to verify whether the results would be the same for representative samples covering the whole extension of the seam.

In view of the project's size, the Compañía de Acero del Pacífico decided to erect a fully equipped, modern coal laboratory in Chile. This laboratory for coal testing speeded up the work, by the avoidance of periodical shipments of samples to the United States and, furthermore, the good condition of those samples was ensured, because of the proximity of the laboratory to the coal mines. On the other hand, its equipment has also been used for the routine control of the coke plant products and by-products.

The laboratory is equipped with an electrically heated experimental oven of the same type (9) as is used by the Illinois Geological Survey, Illinois, in the United States, with slight modifications.

The volume of the oven is 0.3 m^3 and the width is 35 cm., with a capacity of approximately 225 kg. of coal.

The laboratory has all the equipment necessary for chemical analysis, the determination of the physical properties of the coke, and the study of the by-products.

I. *Systematic study of the ashes and sulfur of the various seams at Lota and Schwager. Washability studies* (float-and-sink tests)

An extensive sampling programme was conducted in seams Nos. 3 and 5 from Schwager mine, and Alta, Chica and Arriba, in order to establish the ash and sulfur contents in detail and to be able to compare or revise the results obtained from the samples taken by the U.S. Bureau of Mines.

In the case of seams No. 3, of Schwager, and Arriba of Lota, when the tests made by the U.S. Bureau of Mines had shown unfavourable sulfur contents, channel samples were taken every 25 m. at the different work faces, excluding roof and floor impurities. Thus, 31 and 49 samples, respectively, were taken from No. 3 and Arriba seams. In general, this study confirmed that the high sulfur content is characteristic of these seams. In fact, in all but a few isolated sections, the values exceeded 2% and, in a majority of cases, were over 3%.

The float-and-sink tests showed that the sulfur in these coals cannot be appreciably reduced by washing because it is an inherent part of the coal itself. Other studies established that this sulfur is mainly organic or is found in finely divided pyrites embedded in the coal.

Composite samples were prepared for each different section of the more favourable Alta and Chica seams (Lota) and Schwager No. 5, including usual mining impurities, in order to establish distribution by size fractions and to study, by the float-and-sink method, the general washability characteristics.

Samples were also sent to Koppers Company for coking tests in their movable-wall oven (2) and to the U.S. Bureau of Mines, for complete float-and-sink tests.

Previous results indicating a sulfur content usually below 1% were confirmed. It was seen that the size of the coal had no effect on the values, that is to say, there was no higher sulfur concentration in the smaller sizes where impurities foreign to coal were finely mixed with it.

The results show the following:

(1) There is an evident increase in ash content as the size of the particles decreases. The composite samples had 10.8% for Schwager coal and 7.5% for Lota coal. According to the analysis for ash content in regard to size, it increases progressively from 6.7% for the +4 in. fraction, to 25.4% for the minus 48-mesh fraction in the case of No. 5 seam, and from 2.8 to 45%, respectively, in the case of coal from the Alta seam. This must be attributed to a concentration of foreign impurities in the smaller sizes;

(2) No such concentration is recorded with sulfur. For a content of 1.42 and 0.79% in the composite samples, the extreme values were from 1.39 and 1.62%, in the case of Schwager coal, and almost without variation in the case of coal from Lota;

(3) The float-and-sink tests show that, from the point of view of ash reduction, both coals are easy to wash owing to the definite separation of the coal from the impurities. The intermediate specific gravity fractions between the coal proper (specific gravity 1.30 to 1.35) and the impurities (specific gravity over 1.70) are in all cases found in very small percentages, showing that there is no bone coal in these coals;

(4) It is not so easy to reduce the sulfur by washing, but, as these coals originally had an acceptable sulfur content, the problem represents no difficulty from the point of view of metallurgical uses.

II. *Chemical analysis of ashes from Schwager No. 5 seam and Lota Alta seam*

In order to control the slag composition during blast furnace operation, it is desirable to know the chemical composition of the coke ashes.

Studies were therefore made on coal samples from Schwager No. 5 and Lota Alta seams.

III. *Tests of Schwager No. 5 seam and Lota Alta seam coals, and in blends with Pocahontas, Wyoming and Jewel Ridge coals*

The Huachipato coal laboratory made numerous comparative studies of the coking properties of the Schwager and Lota coals, and of these coals in blends with various highly coking United States coals.

The Compañía de Acero del Pacífico coke plant has normally used Pocahontas coal for blending on a commercial scale. Lately, Wyoming coal has also been used. In the study that follows, the samples of Schwager No. 5 seam, Lota Alta seam and Pocahontas Nos. 7 and 31 coals were obtained from the regular shipments arriving at the plant.

The Wyoming coal comes from the Sewell seam, Wyoming County, West Virginia. The Jewel Ridge coal is obtained from the Jewel seam, Tagwell County, Virginia. These samples arrived from the United States in sealed drums. These are all medium-volatile coals.

Table 3 indicates the proximate analysis of the coals and blends used in this series of tests, as well as of the cokes obtained. The sulfur content of the samples of Chilean coals confirms the values previously obtained. Table 4 indicates the results of plasticity tests made with the modified Gieseler plastometer (10, 11). This table includes separately the values for each coal and for the blends.

Table 3

PROXIMATE ANALYSIS OF COALS FROM SCHWAGER SEAM NO. 5, ALTA SEAM, AT LOTA, POCAHONTAS (MINES NOS. 7 AND 31); WYOMING AND JEWEL RIDGE

			Coal				*Coke*			
			Dry basis, %				*Dry basis, %*			
No.	*Description*	*% H_2O*	*Vol. mat.*	*Fixed carbon*	*Ash*	*Sulfur*	*Vol. mat.*	*Fixed carbon*	*Ash*	*Sulfur*
1.	100% Schwager seam No. 5	4.6	41.2	51.8	7.0	1.16	1.6	87.0	11.4	1.14
2.	100% Lota Alta seam	4.0	43.1	53.3	3.6	0.81	1.1	92.8	6.1	0.76
3.	100% Pocahontas mines Nos. 7 and 31	3.2	24.0	67.3	8.7	0.61	—	—	—	—
4.	100% Wyoming	2.2	22.7	72.1	5.2	0.56	—	—	—	—
5.	100% Jewel Ridge	3.0	23.5	71.1	5.4	0.61	—	—	—	—
6.	80% Schwager seam, No. 5+20% Pocahontas	3.9	36.5	55.4	8.1	1.00	1.3	87.9	11.0	0.99
7.	80% Schwager seam No. 5+20% Wyoming	3.2	36.7	57.1	6.3	1.10	0.7	90.9	8.4	0.97
8.	80% Schwager seam No. 5+20% Jewel Ridge	4.3	36.5	56.3	7.2	1.16	1.1	89.2	9.7	1.04
9.	80% Lota Alta seam + 20% Pocahontas	3.3	38.0	56.0	6.0	0.76	1.0	91.3	7.7	0.73
10.	80% Lota Alta seam + 20% Wyoming	3.2	38.2	57.7	4.1	0.76	1.6	92.6	5.8	0.73
11.	80% Lota Alta seam + 20% Jewel Ridge	3.6	38.5	57.8	3.7	0.79	1.2	94.0	4.8	0.75
12.	65% Schwager seam No. 5 + 35% Pocahontas	3.5	34.9	56.6	8.5	0.94	1.2	88.1	10.7	0.94
13.	65% Schwager seam No. 5 + 35% Wyoming	3.4	34.7	59.1	6.2	1.06	1.0	89.6	9.4	0.82
14.	65% Schwager seam No. 5 + 35% Jewel Ridge	3.1	35.0	58.5	6.5	1.00	1.1	89.7	9.2	0.79
15.	65% Lota Alta seam + 35% Pocahontas	3.0	34.5	60.0	5.5	0.68	1.5	91.3	7.2	0.69
16.	65% Lota Alta seam + 35% Wyoming	3.0	35.4	60.1	4.5	0.74	1.0	93.1	5.9	0.70
17.	65% Lota Alta seam + 35% Jewel Ridge	3.2	34.3	61.9	3.8	0.70	1.2	92.3	6.5	0.70
18.	50% Schwager seam No. 5 + 50% Pocahontas	3.7	31.5	60.3	8.2	0.86	1.3	87.6	11.1	0.79
19.	50% Schwager seam No. 5 + 50% Wyoming	3.0	31.0	62.7	6.3	0.86	1.4	89.8	8.8	0.80
20.	50% Schwager seam No. 5 + 50% Jewel Ridge	3.1	32.1	67.7	6.2	0.84	0.6	90.7	8.7	0.74
21.	50% Lota Alta seam + 50% Pocahontas	3.6	33.2	60.7	6.1	0.68	1.4	90.8	7.8	0.71
22.	50% Lota Alta seam + 50% Wyoming	2.7	32.2	63.3	4.5	0.95	1.0	93.3	5.7	0.65
23.	50% Lota Alta seam + 50% Jewel Ridge	3.1	32.5	63.0	4.5	0.68	1.6	92.5	5.9	0.73

Table 4

RESULTS OF GIESELER PLASTOMETER TESTS ON COALS FROM SCHWAGER SEAM NO. 5; ALTA SEAM AT LOTA; POCAHONTAS (MINES NOS. 7 AND 31); WYOMING JEWEL RIDGE AND BLENDS

		Temperature °C				
No.	*Coal*	*Initial softening*	*0.1 Div. per min.*	*Maximum fluidity*	*Solidification*	*Maximum fluidity Div./Min.*
1.	100% Schwager seam No. 5	334	380	433	459	7.2
2.	100% Lota Alta seam	350	389	441	465	3.6
3.	100% Pocahontas mines Nos. 7 and 31	375	393	471	508	1,005
4.	100% Wyoming	377	389	466	507	780
5.	100% Jewel Ridge	386	399	468	509	1,150
6.	80% Schwager seam No. 5 + 20% Pocahontas	332	385	435	463	9.9
7.	80% " " " " + 20% Wyoming	332	393	435	463	9.7
8.	80% " " " " + 20% Jewel Ridge	336	392	434	470	9.8
9.	80% Lota Alta seam + 20% Pocahontas	345	394	438	463	4.8
10.	80% " " " + 20% Wyoming	333	392	438	461	5.4
11.	80% " " " + 20% Jewel Ridge	332	392	438	461	5.3
12.	65% Schwager seam No. 5 + 35% Pocahontas	348	393	438	471	11.3
13.	65% " " " " + 35% Wyoming	336	393	434	469	11.1
14.	65% " " " " + 35% Jewel Ridge	332	390	444	471	13.0
15.	65% Lota Alta seam + 35% Pocahontas	345	395	445	470	6.7
16.	65% " " " + 35% Wyoming	345	395	441	466	6.5
17.	65% " " " + 35% Jewel Ridge	341	392	442	469	8.0
18.	50% Schwager seam No. 5 + 50% Pocahontas	359	392	450	480	31
19.	50% " " " " + 50% Wyoming	354	389	448	479	32
20.	50% " " " " + 50% Jewel Ridge	341	393	444	477	34
21.	50% Lota Alta seam + 50% Pocahontas	354	396	455	482	22
22.	50% " " " + 50% Wyoming	368	393	446	476	9.6
23.	50% " " " + 50% Jewel Ridge	363	393	454	482	23

The maximum fluidity values of the Chilean coals are very low, but a certain progressive increase is noticed as the proportion of imported coal increases. It is evident from this table that the plastic properties of the coal are not of the additive type, that is, there is no direct relation between the maximum fluidity of the blends and the values corresponding to the individual coals.

The results of the screen analysis, the shatter and tumbler tests,[1] provided values corresponding to experimental oven tests, and are not comparable to values obtained on a commercial scale in much larger ovens; they must, therefore, be interpreted relatively. The average testing conditions were as follows:

Coking time (net)................	10 hrs. 52 mins.
Final flue temperature, approx......	1,110° C
Final coke temperature............	1,075° C

The results are shown in Table 5.

These tests showed that, generally, there is no notable difference in the quality of the coke from 100% Schwager No. 5 seam coal and 100% Lota Alta seam coal. From a point of view of a practical application, they may be considered as very similar.

The progressive addition of North American coals of a higher coking power appreciably improves the physical properties of the coke. A most convenient index of comparison is the "stability factor", which in the case of the A.S.T.M. method corresponds to the product over 1.06 in. of the tumbler test. From the analysis of these values it can be noted that the effect of any of the three North American coals is similar, although, in some cases, the Wyoming and Jewel Ridge coals show slightly superior values.

A normal coking time of approximately 11 hours has been standardized by the Huachipato laboratory coking tests tending to obtain properties that can be relatively compared to those of blast furnace coke.

Tests were also made with blends of 50% Chilean coal and 50% North American coal. Here the average coking time was 19 hours 32 minutes, in order to obtain data for foundry coke, for its use in cupolas, etc.

The longer coking time represented an evident increase in the values of the screen analysis. For instance, the product of +2 in. increased from 70–80% to more than 90%. Also, an appreciable increase in the stability factor was recorded (+1.06 in. in the tumbler test, A.S.T.M. method). Present practice for the production of foundry coke is to employ a blend of 50% Chilean coal, 50% imported coal, at 24-hour coking time and screening the coke on a 2 in. screen.

IV. *Coking tests of coal from Schwager No. 3 and Lota Arriba seams*

With a view to completing the study of these high sulfur coals as regards their coking properties, the necessary tests were carried out. In general, the previous conclusions were confirmed, that is, that resulting coke has properties which are definitely inferior to those of the Schwager No. 5 seam and Lota Alta seam. This fact added to their high sulfur content, made their use at the Huachipato plant inadvisable.

V. *Effect of atmospheric oxidation in coals from Schwager No. 5 and Lota Alta seams, on the quality of coke*

The Compañía de Acero del Pacífico has been concerned, from the start, with the problem of probable deterioration of the coking properties of Chilean coals by atmospheric oxidation during storage.

For the purpose of obtaining the relevant data, various coking tests were made in the experimental oven, with samples of lump, run-of-mine and slack coal from the Schwager No. 5 and Lota Alta seams. Plasticity tests were also made with the modified Gieseler plastometer (10, 11).

For this study, each one of the above coals was stored outdoors for a long period and samples were tested at given intervals.

Briefly, the conclusions obtained indicated that, in general, these coals can tolerate an atmospheric exposure of at least 90 days, and even longer, without any deterioration of their coking properties. In the case of lump coal, this held good for an even longer period.

Nevertheless, as a safety measure, the plant receives freshly mined coal from Lota and Schwager mines. This coal is loaded directly into cars without passing through the coal storage system at the mines. Furthermore, a system of rotating the coal piles has been instituted to ensure a maximum storage period of two months for a given shipment of Chilean coal.

VI. *Effect of the degree of pulverization of the coal on the quality of the coke*

Several comparative studies were made using various degrees of pulverization ranging from 50 to 90% below ⅛ in. mesh. Crushing took place in a pilot scale hammer mill for the purpose of trying to duplicate the size distribution condition at the plant, where a ring-mill crusher is used for the individual coal and a hammer mill for the blended coals.

No great variation in the coke properties was noted in the experimental oven coking tests for the various degrees of coal pulverization, but it was believed from visual inspection that coarse pulverization affected the quality of the coke.

It is believed that an increase in size of the particles, which, to a certain extent, brings about an increase of the bulk density, thus improving the quality of the coke, does tend to offset the negative effect of a lesser homogenization in the case of blends produced by a coarser pulverization.

BRIEF DESCRIPTION OF INDUSTRIAL PRACTICE AT THE HUACHIPATO BY-PRODUCT COKE PLANT

The Huachipato by-product coke plant has 57 Koppers-Becker, underjet-type ovens (14). Another 13 ovens are being built and will start operating in 1953. The coke ovens are normally heated with blast furnace gas, but have been designed for using domestic gas as well (mixture of coke oven gas and blast furnace gas to 4,000 cal/m^3).

The coke ovens are of the narrow type, 13 in. wide on the pusher side and 15½ in. wide on the coke side, with an average width of 14¼ in. and having a capacity of 13 metric tons of coal.

The normal coking time is determined by the coke requirements of the blast furnace. The current production schedule is 79 ovens a day (15.8 hours gross coking time) for the production of blast furnace coke and five ovens a day (24 hours gross coking time) for the production of foundry coke. The present coal consumption at the ovens is 1,100 metric tons a day.

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[1] The shatter and tumbler tests were made following the methods of the Columbia Steel Company (3, p. 19) and of the A.S.T.M. (12, 13).

Table 5

PHYSICAL PROPERTIES OF COKE PRODUCED FROM COALS OF SCHWAGER NO. 5 SEAM, LOTA ALTA SEAM AND BLENDS OF THESE WITH POCAHONTAS NOS. 7 AND 31, WYOMING AND JEWEL RIDGE COALS

Tests in an experimental pilot oven. Average coking time: 10 hours 53 minutes

		Screening Analysis									*Shatter Test*										
											A. S. T. M.						*Columbia*				
No.	*Coal*	*+4″*	*+3″*	*+2″*	*+1½″*	*+1″*	*+¾″*	*+½″*	*−½″*	*−¼″*	*+3″*	*+2″*	*+1½″*	*+1″*	*+½″*	*−½″*	*+1½″*	*+1″*	*+¾″*	*+½″*	*−½″*
1.	100% Schwager seam No. 5	7.8	22.2	50.7	69.7	81.4	87.4	92.1	7.9	4.6	0.0	11.4	32.3	55.0	84.0	16.0	21.2	54.0	69.5	85.2	14.8
2.	100% Lota Alta seam	2.5	13.0	47.0	67.0	83.5	89.1	93.0	7.0	4.6	0.0	9.9	28.9	61.4	88.1	11.9	16.9	53.0	72.8	87.4	12.6
6.	80% Schwager +20% Pocahontas	6.9	28.8	72.2	85.9	93.9	95.8	96.9	3.1	1.8	2.6	29.0	59.4	84.1	95.5	4.5	37.8	79.9	89.5	94.2	5.8
7.	80% " +20% Wyoming	3.7	23.5	67.6	86.8	94.3	96.2	97.5	2.5	1.5	0.0	26.9	59.5	85.3	92.5	7.5	50.4	84.4	91.5	95.5	4.5
8.	80% " +20% Jewel Ridge	8.8	26.7	75.3	88.3	94.3	96.3	97.5	2.5	1.6	4.2	37.9	56.7	83.3	95.4	4.6	42.1	65.6	82.1	96.3	3.7
9.	80% Lota +20% Pocahontas	5.1	21.8	68.5	86.8	93.8	96.0	97.1	2.9	2.0	1.3	33.9	60.8	85.4	94.8	5.2	51.4	86.0	93.5	96.0	4.0
10.	80% " +20% Wyoming	6.4	25.4	66.5	84.9	93.7	96.2	97.3	2.7	1.9	0.0	24.5	59.7	84.6	95.7	4.3	41.1	81.8	92.0	96.4	3.6
11.	80% " +20% Jewel Ridge	6.1	23.7	61.7	84.0	94.1	96.4	97.4	2.6	1.9	0.0	33.9	63.1	87.0	95.9	4.1	42.1	83.9	94.1	97.4	2.6
12.	65% Schwager +35% Pocahontas	7.8	38.6	79.9	90.3	95.4	96.8	97.6	2.4	1.6	2.0	40.9	65.9	88.9	96.3	3.7	44.9	85.2	93.5	96.6	3.4
13.	65% " +35% Wyoming	4.2	29.7	76.5	92.0	95.0	96.9	97.7	2.3	1.5	1.9	36.9	68.1	89.7	96.6	3.4	62.9	88.6	94.8	97.7	2.3
14.	65% " +35% Jewel Ridge	7.4	34.5	77.0	89.0	95.4	97.0	97.8	2.2	1.5	3.6	33.4	69.5	89.2	97.1	2.9	46.0	85.8	93.4	96.7	3.3
15.	65% Lota +35% Pocahontas	10.1	38.5	74.8	88.3	95.8	97.7	98.6	1.4	—	6.8	48.6	75.8	91.5	96.4	3.6	41.3	81.6	93.2	96.6	3.4
16.	65% " +35% Wyoming	4.0	24.2	66.4	83.3	94.4	96.8	97.7	2.3	1.6	2.4	31.6	66.7	89.8	97.1	2.9	48.2	82.6	94.5	97.6	2.4
17.	65% " +35% Jewel Ridge	—	7.1	60.6	80.5	94.0	96.8	97.7	2.3	1.7	0.0	25.8	58.8	87.8	97.4	2.6	38.2	79.7	93.3	96.2	3.8
18.	50% Schwager +50% Pocahontas	13.7	—	79.9	—	—	97.4	—	2.0	—	—	47.4	—	—	—	2.9	—	—	—	—	—
19.	50% " +50% Wyoming	12.2	42.8	79.4	89.9	95.9	97.4	98.1	1.9	1.2	0.0	48.0	69.1	90.2	96.4	3.6	—	—	—	—	—
20.	50% " +50% Jewel Ridge	13.0	39.9	76.2	88.1	95.8	97.4	98.1	1.9	1.2	2.9	39.2	65.7	90.7	97.4	2.6	—	—	—	—	—
21.	50% Lota +50% Pocahontas	3.2	28.5	70.8	84.6	94.9	96.9	97.8	2.2	1.5	0.0	36.9	65.0	88.3	97.1	2.9	—	—	—	—	—
22.	50% " +50% Wyoming	2.2	29.2	71.9	86.9	96.2	97.6	98.2	1.8	1.2	0.0	36.6	63.9	90.8	97.0	3.0	—	—	—	—	—
23.	50% " +50% Jewel Ridge	3.3	26.5	66.8	85.9	95.6	97.2	97.8	2.2	1.6	0.0	50.7	73.0	92.4	97.6	2.4	—	—	—	—	—

		Tumbler Test													
		A. S. T. M.						*Columbia*							
No.	*Coal*	*+2″*	*+1.5″*	*+1.06″*	*+0.53″*	*+0.265″*	*−0.265″*	*+1½″*	*+1″*	*+½″*	*+¼″*	*−¼″*	*Apparent Specific Gravity*	*Real Specific Gravity*	*Porosity*
1.	100% Schwager seam No. 5	0.0	0.0	2.3	30.0	61.2	38.8	10.6	44.7	70.3	79.4	20.6	0.76	1.81	59
2.	100% Lota Alta seam	0.0	0.0	1.7	36.7	63.5	36.5	10.4	52.3	73.6	80.5	19.5	0.76	1.78	62
6.	80% Schwager +20% Pocahontas	0.0	3.3	23.9	55.5	62.0	38.0	31.5	72.4	82.8	84.5	15.5	0.82	1.90	57
7.	80% " +20% Wyoming	0.0	6.0	32.2	60.7	66.1	33.9	34.5	70.8	84.5	86.2	13.8	0.82	1.83	55
8.	80% " +20% Jewel Ridge	0.0	4.2	28.6	58.2	64.7	35.3	37.2	74.1	84.4	86.1	13.9	0.83	1.88	56
9.	80% Lota +20% Pocahontas	0.0	4.7	28.9	58.4	64.3	35.7	29.1	70.6	84.9	87.2	12.8	0.80	1.82	56
10.	80% " +20% Wyoming	0.0	6.4	33.3	66.0	69.8	30.2	33.9	75.2	88.8	89.9	10.1	0.79	1.86	58
11.	80% " +20% Jewel Ridge	0.0	8.6	38.4	64.7	67.8	32.2	31.0	74.8	87.5	88.5	11.5	0.79	1.81	56
12.	65% Schwager +35% Pocahontas	0.6	8.6	38.9	59.1	62.6	37.4	35.0	76.6	84.3	85.3	14.7	0.83	1.92	57
13.	65% " +35% Wyoming	0.9	12.4	38.0	64.8	68.0	32.1	43.8	78.8	86.0	87.3	12.7	0.82	1.89	57
14.	65% " +35% Jewel Ridge	0.0	11.9	38.9	63.9	66.4	33.6	30.1	76.2	85.8	86.9	13.1	0.85	1.82	53
15.	65% Lota +35% Pocahontas	1.8	11.9	41.6	63.4	65.6	34.4	31.1	74.8	85.7	86.7	13.3	0.82	1.84	55
16.	65% " +35% Wyoming	0.8	10.9	43.0	66.6	69.1	30.9	28.8	74.5	88.1	89.0	11.0	0.81	1.86	56
17.	65% " +35% Jewel Ridge	0.4	10.6	44.9	69.1	70.9	29.1	32.3	74.3	88.7	89.2	10.8	0.81	1.86	56
18.	50% Schwager +50% Pocahontas	—	—	37.0	63.0	65.9	—	—	—	—	—	—	0.87	1.90	54
19.	50% " +50% Wyoming	2.7	14.2	42.3	66.5	69.2	30.8	—	—	—	—	—	0.86	1.88	54
20.	50% " +50% Jewel Ridge	1.4	16.2	46.2	65.2	67.1	32.9	—	—	—	—	—	0.86	1.91	55
21.	50% Lota +50% Pocahontas	4.7	15.8	43.0	66.0	68.4	31.6	—	—	—	—	—	0.84	1.86	55
22.	50% " +50% Wyoming	0.5	11.5	45.8	72.8	74.7	25.3	—	—	—	—	—	0.83	1.85	55
23.	50% " +50% Jewel Ridge	5.6	23.9	51.5	68.6	69.6	30.5	—	—	—	—	—	0.81	1.85	56

Varying coal blends have been used for producing blast furnace coke, ranging from 50 to 80% Chilean coal (Schwager No. 5 and Lota Alta seams) and 50 to 20% North American coals (Pocahontas Nos. 7 and 31, and, recently, Wyoming coal). The Chilean coal used is lump (+1 in. hand-picked Lota mine and +¾ in. cleaned in a Berrisford separator at Schwager mine), and slack coal washed at the mines. The North American fuel is slack coal.

The mixed coals are ground in a reversible hammer mill, to a pulverization of about 80% under ¾ in. mesh.

The coke produced is sent on conveyors to two bins located at the blast furnace where it is screened in rectangular screens having ½ in. in the smaller dimension.

The undersize is the coke breeze, which, depending on the market conditions, is further re-screened on ⅛ in. × 3 in. screens for the production of pea coke (+⅛ in.) and fine breeze (−⅛ in.).

Owing to insufficient production of Chilean coal, the Huacipato plant uses at present a blend of 60% Chilean and 40% North American coals, respectively, but blends having up to 80% Chilean coal have been used successfully. Some increase in Chilean coal production may be expected over the next few years due to the opening up of new faces and mechanization of the pits. Furthermore the Compañía de Acero del Pacífico is developing a boring programme in the Arauco zone, to the south of those belonging to the Compañía Carbonífera e Industrial de Lota. The results so far are promising and this may lead to an important increase in Chilean coal production for the future.

As an illustration, a detail is given of some of the values indicative of the physical properties of coke obtained at Huachipato from the blend of 60% Chilean (No. 5 and Alta seams) and 40% North American coals (Pocahontas Nos. 7 and 31).

Operating conditions

Coking time (gross)	15 hrs. 48 mins.
Coking time (net)	15 hrs. 30 mins.
Average temperature at the flues	1,275 °C
Average temperature of the coke	1,080 °C

Screening analysis of coke charged in the blast furnace

Plus 2 in.	75–85%
Plus 1¼ in.	95–97%
Shatter test (A.S.T.M.), + 2 in.	60–70%

Tumbler test (A.S.T.M.)

Stability factor, + 1.06 in.	40–45%
Hardness factor, + 0.265 in.	66–68%

The plant also supplies Chile's foundry coke requirements, thereby eliminating imports previously necessary.

The gas from the coke ovens is diluted with blast furnace gas to 4,000 cal/m^3, to take care of the domestic consumption of the zone, which prior to the installation of the Huachipato plant was using retort gas of an equal heating value. The plant uses part of this gas for internal purposes, while the balance supplies the region's entire domestic consumption requirements, as well as those of some industries.

Other by-products are tar, now being consumed at the open hearths and, eventually, the production of creosote to be shipped to a wood-impregnating plant at Valdivia. Motor fuel is obtained from the light oils recovered from the gas by absorption in washing oils. This is either mixed with gasoline, or is exported for refining into various products, such as benzol, toluol and xylol, and intermediate light oils, which are used in the open hearths together with tar.

The average yield in coke and by-products corresponding to a blend of 60% Chilean coal (No. 5 Schwager and Alta Lota seams) and 40% Pocahontas coal, is given below:

Yields per metric ton of coal

Blast furnace coke	600 kg.
Coke breeze (−½ in.)	68 kg.
Coke oven gas	300 m.^{3}
Tar	38 litres
Light oils	11 litres

Yield of the light oils

Motor fuel (benzol, toluol, xylol)	91.5%
Intermediate light oils	3.8%

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Manufacture of Metallurgical Coke from Peruvian Anthracite

KURT BAUM

In a document,[1] presented to another section of this meeting, by the author, some of the difficulties have been described which face washing of Peruvian Santa region anthracite fines entails. A brief description is also given regarding the "Convertol" phase washing process, as developed by the Bergwerksverband, Essen, West Germany.

The resulting product is fine coal blended with the oil used in the preparation process, this oil ranging between 3 and 7% of the raw coal by weight. The special advantage of this, furthermore, is the fact that such concentrates can be used for making briquettes, which in turn can be coked in order to manufacture smelting carbon.

Under prevailing local conditions an artificial binder has to be introduced anyway, due to the lack of any bituminous matter in such anthracites. Petroleum pitch or heavy residues of the Talara refinery were the most likely ones and were readily available in Peru. The large percentage of Talara hard pitch necessary (20–25%) and its high price, however, have so far prevented industrial scale production of coke using such blends, in, for instance, Beehive or Curran-Knowles ovens.

Using pre-fabricated briquettes a much smaller amount of pitch (10–12%) would be required.

The Talara pitch with a high melting point of 115–120° C, however, is not very suitable for standard briquetting practice. The introduction of 4–6% fuel oil on the coal surface by the Convertol process, however, is a very favourable factor facilitating the following briquetting procedure. By adding 7–8% of hard pitch in a usual pugmill and heating the mass, a pitch-oil mixture is formed, resulting in a soft pitch with 70–75° C smelting point representing a very favourable briquetting binder under standard conditions.

The fine grain size of 0–1 mm. has to be applied with regard to a suitable coke structure, since coking of such briquettes is the final aim, and an elevated amount of binder had to be introduced anyway. Standard briquetting of 0–3 mm. grain size coal would not require such an elevated amount of binder.

Another great advantage of the coking process as applied is that 4–5%[2] of the original binder added is recovered in the form of pitch and oil during the coking process itself. The method used for coking such briquettes is a combined heating system applying normal indirect heating of the continuous or semi-continuous working vertical chambers and re-cycle gas inside. Due to the fact that pre-fabricated briquettes are charged cold, gas can be introduced as purge gas at the bottom of the retort, recovering hereby the sensible heat of the hot coke before its discharge and re-introducing such heat into the coking. Heating velocity and throughput are nearly doubled by this method as compared with the data known for standard coke ovens (20–22 kg/m^2–h. against 10–12 kg/m^2–h). The result therefore is a dual one:

(1) Dry cooling of the coke before its discharge and improved heat economy of the coking process;

(2) Increased capacity of the coking chamber resulting in lower capital investment. In fact, two chambers are foreseen for a capacity of 30 t/h. each.

The pitch and oil vapours partly redistilled and reformed are entrained in the re-cycle gas and recuperated by fractional condensation, the pitch and oil fraction separately, so that the pitch fraction can be re-introduced to the briquetting plant, the heavy oil fraction to the Convertol processing. The actual consumption of binder therefore is not much higher than in ordinary briquetting (7–8%),[2] the remainder being transformed into coke and gas as well as valuable aromatic light oils; 4.5%[2] correspondingly are transformed into coke acting as the coking agent by smelting and coking; 3–3.5%[2] are cracked into gas and aromatic light oils of which the latter represent a most valuable by-product. This is due to the fact that besides the fuel oil even the refinery hard pitch in spite of its high melting point contains a considerable percentage of paraffinic fraction due to the fact that the melting point of such pitch does not result from previous high temperature treatment, but from distillation or cracking processes where much lower temperatures have been applied. If such heavy fractions are treated at temperatures of 900–1,000° C, a secondary cracking action takes place, forming in particular low and high boiling aromatic or cyclic hydrocarbons.

Basing on the most favourable results so far obtained, a 60 tons/day demonstration unit is currently being erected by Carbonera Pallasca S.A. in the Chimbote region. The coke so far produced has about the following qualifications:

Uniform lumpsize	85×60×50 mm.
Shatter test	83–86
Porosity	38–45%
Carbon	86–87%
Ash	10–14%
Sulfur	0.6–0.8%
Volatile matter	1–2%
Bulk weight	40–35 lb./cu. ft.

Thus it is obviously an excellent metallurgical smelting carbon. Some primary smelting tests in a cupola furnace have already been performed with satisfactory results. Larger blast furnace tests will follow in the near future. The final aim is the erection of an iron industry in this district, applying ordinary blast furnace technique, and based on Marcona iron ores and coke prepared from a formerly practically useless waste fuel. Final results will be available in about a year's time from now.

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[1] The Washing of Peruvian Anthracite Fines.

[2] These percentages refer to the total weight of the blend.

Summary of Discussion

Coking Properties of the Coal for the Steel Industry in Colombia, *presented by Mr. Prieto, co-author*

Mr. ALBALA opened the general discussion of the paper under review by associating himself with the views expressed by Mr. Prieto to the effect that extensive research was necessary in order to determine the coking properties of coal from a new region for an enterprise of considerable magnitude. The paper under review had led him to believe that Colombia would be able to use its own coals for the production of coke.

In Chile also it had been preferred to use the small-scale oven for testing and he hoped to discuss that subject in greater detail at a later date. In general, the methods used by Colombia and Chile appeared to be fairly similar and he suggested that other Latin-American countries might benefit from the experience gained in those two countries, as well as in Brazil.

Mr. PRIETO thanked Mr. Albala for his kind comments on the paper. In Colombia, it had been found that by mixing high-volatile B coal and medium-volatile coal a satisfactory grade of coke could be obtained. Both types of coal existed in the neighbourhood of the new steel plant.

Production of Metallurgical Coke in Mexico, *presented by the author*

Mr. LEUSCHNER requested some information of the author on the development of the mining enterprises in the Sabinas Valley. He had been given to understand that small entrepreneurs were wont to work from the outcrops towards the centre. That method seemed a rather costly one, and he wondered whether any steps had been taken towards the consolidation of small mining enterprises in an effort to reduce over-all costs and to increase production.

Mr. CORTÉS OBREGÓN replied that although it might seem incredible, it was much more profitable at the outset to mine at the surface in what might be described as a primitive fashion than to undertake deep mining. However, many small operators who originally devoted themselves to outcrop mining invariably turned to deep mining because of the expense involved in moving from one outcrop to another. Several deep-mining concerns were conducting simultaneously the exploitation of outcrops in order to reduce costs.

Mr. LEUSCHNER intervened to observe that the papers contributed by various participants concentrated on successful exploits rather than on failures. He urged those present to elaborate upon the setbacks and difficulties they had encountered, so that others might not only profit from their experience but might be encouraged to continue their own struggles. The purpose of the meeting was to uncover failures as well as successes.

Mr. CHERADAME commented on the fact that both Mr. Cortés Obregón and Mr. Prieto had spoken of their experience with the Gieseler test. He wished to say a few words on the merits of the Gieseler plastometer, based on his own experience. In France, the Gieseler plastometer was found to be too sensitive to allow comparison, with a reasonable degree of accuracy, of data obtained from several plastometers in different locations. It should be mentioned also that many people had executed variations on the Gieseler plastometer, and that the different apparatus would then give different results.

Another disadvantage of the Gieseler plastometer was that pronounced oxidization of high-volatile coals had too much influence on the result.

For the foregoing reasons, the Gieseler plastometer was seldom considered in France—and the same was true in Belgium—as a normal control apparatus for coking plants.

Mr. PRIETO stated that, in general, he was inclined to agree with Mr. Cheradame's criticism. Colombian experience with the Gieseler plastometer had found it valuable only for statistical purposes. Several tests with the same apparatus were necessary before any definite conclusion could be reached.

He thought that Mr. Cheradame's comments had demonstrated that a standardization of plastometer tests had not yet been achieved. However, despite its many shortcomings, in his own opinion, it was the simplest type of plastometer for the purposes of his country.

Mr. ALBALA contributed his own views on the merits of the Gieseler plastometer. He realized that there were many modifications practised on it, but in Chile the Gieseler plastometer designed by Koppers had been used quite successfully. Tests utilizing that plastometer could be duplicated within a fair value. Perhaps a different type of plastometer was used in France, hence the variation in the results obtained. Chile had chosen the Gieseler plastometer for its simplicity and the results were satisfactory for preliminary tests.

To prove his point on the possibility of fair duplication of tests, he was willing to provide Mr. Cheradame with statistical data.

Mr. CHERADAME stated that he would be most interested to review the statistical data offered and would reciprocate by providing Mr. Albala with two papers that were presented in Paris during a recent conference by staff members of CERCHAR laboratories.

Mr. LÓPEZ remarked that the Gieseler plastometer had been used in Colombia with fairly satisfactory results.

INTRODUCTION

This volume contains the working papers presented by the experts who participated in the ECLA/TAA Expert Working Group on Iron and Steel Industry in Latin America; it also includes the discussions on those papers. The volume, then, consists of the proceedings of nineteen working meetings: three deal with the reduction of coal impurities; three with the production of coke from poorly coking coal and substitute fuels; one with the economics of the blast furnace; one with the charcoal blast furnace; three with methods of reducing ores otherwise than in blast furnaces; four with the comparison of economics of different processes of steelmaking; one with alternative methods to the rolling mill for finishing steel; two with the range of application of steels made by different processes; and one with general subjects connected with steelmaking in Latin America.

The proceedings of the first two meetings of the working group have been omitted, since they were not concerned with substantive matters. Similarly, the discussions of the papers presented by the Secretariat, which occupied the four final meetings, are excluded, since these papers and the findings resulting from their discussion have been incorporated in the report of the meeting (Volume I).

The agenda for the meeting, list of participants and the list of papers are contained in an appendix to Volume I, and are therefore not repeated here. An index of authors, in alphabetical order, may be found at the end of this volume, together with the titles of their contributions. In the findings of the meeting (Volume I), all the papers are referred to simply by the final number of the original symbol, and therefore the above list of papers in the original numerical order has to be consulted in order to verify the title and thus be able to find the corresponding paper in Volume II.

To avoid any unwitting duplication, many of the original papers have been revised and abridged, in consultation with the authors. Their valuable collaboration at all stages of the meeting and the preparation of this report is hereby most gratefully acknowledged by the United Nations Technical Assistance Administration and the Economic Commission for Latin America. It should be borne in mind, however, that there may have been modifications in facts and figures since the papers were originally prepared in 1952. They therefore describe conditions prevailing at that time; in some cases experimental or new processes described in the papers have since become well-established, or have changed from a pilot to an industrial scale.

Metallurgical Coke from Chilean Coals, ***presented by the author***

Mr. Prieto opened the discussion of the paper by commending the author on the excellent delivery of his presentation. Certain conclusions could be drawn from the paper under review, e.g., the importance of coal research in all of the Latin-American countries, as well as the establishment of pilot plants and laboratories to continue such research.

Current literature indicated that Latin America contained only 1% of the coal deposits of the world. He thought that the meeting was proving the inaccuracy of that figure.

Mr. Mercier remarked that much had been said about laboratory tests and industrial tests in the blast furnace. In his own experience with Saar coals, attempts had been made to use blends for laboratory tests containing the same properties as blends used in large-scale industrial tests. Coals liable to expand were, for security reasons, tested in small boxes and the results obtained had been very favourable.

Manufacture of Metallurgical Coke from Peruvian Anthracite, ***presented by the author***

Mr. Price observed, at the conclusion of Mr. Baum's presentation, that one very important characteristic of coke had not been included in any of the standard tests. That characteristic was three-fold: (1) the temperature of ignition; (2) the rate of combustion; and (3) the reactivity of the coke. Speaking of briquettes as a means of agglomerating the coal fines, he had noticed that the edges of the coke were much lower in porosity. He wondered whether the porosity of so-called coke briquettes had been taken into consideration in Mr. Baum's investigations on the use of coke briquettes.

Mr. Baum replied that in the case of anthracite, with its poor ignition properties, combustion was of course slow and relatively high temperatures were needed. The reactivity tendencies of either powdered or fine coals were not comparable with the combustion observed in large-size coals. He had come to the conclusion that combustion could be influenced by the size of the coal used.

Mr. Saniter addressed two questions to Mr. Baum: firstly, was it necessary to take special precautions to prevent briquettes from sticking together in the coking process; secondly, had Mr. Baum considered the feasibility of artificially increasing the voidage so as to produce a briquette of the size and shape of a doughnut.

Mr. Baum replied that, provided the pitch and oil were not excessive, there was no danger of sticking. However, that phenomenon was more apparent with small-size than with large-size coals. In any event, the operations in the oven were not too greatly hampered by a slight degree of sticking.

As to the second question, that problem had been studied intensively in the United States. To produce form coke, an extrusion process was used in lieu of briquetting, so that the coke assumed a somewhat cylindrical shape. By that method, the bulk weight was considerably reduced—by about 9 to 10%. That aspect was most important because the density of the coke was very high. He did have some experience with the type of briquetting described by Mr. Saniter. However, due to the more pressing problems that had occupied the attention of his firm, he had not explored the further improvement of bulk weight.

IRON ORE REDUCTION

Economics of the blast furnace

17 October 1952—Afternoon

Chairman:

P. L. SCHERESCHEWSKY, Ingénieur au Corps des Mines, Chargé de Mission de la Chambre Syndicale de la Sidérurgie Française, Paris, France

Contributed Papers:

Coal Washery Performance as Related to Blast Furnace Costs
John D. PRICE, Superintendent, Coke Plant, The Colorado Fuel and Iron Corporation, Pueblo, Colorado, U.S.A.

The Influence of the Ash Content on the Hardness of Cokes Made from High-Volatile Coals
Walter VOGEL, Consulting Engineer, Santiago, Chile

Development of the Cauca Valley Coals
Alberto VARGAS MARTÍNEZ, Instituto de Fomento Industrial, Bogotá, Colombia
Thomas FRASER, Supervising Engineer, Coal Preparation Section, Bureau of Mines, U.S. Department of the Interior, Washington, D. C.

The Cerrejón Coal Mining Project
Sidney A. MEWHIRTER, Mining Engineer, Instituto de Fomento Industrial, Bogotá, Colombia

Summary of Discussion:

Participants: Messrs. PRICE, CHERADAME, LEUSCHNER, BAUM, ALBALA, CORTÉS OBREGÓN, MERCIER, GONZALEZ BALLESTEROS, VARGAS MARTÍNEZ.

Coal Washery Performance as Related to Blast Furnace Costs

JOHN D. PRICE

INTRODUCTION

Many of the coals available for the production of blast furnace coke contain so much ash or sulfur that the washing of these coals before carbonizing is highly desirable. Regardless of the quality of the iron ore, a coke approaching 35% ash is almost worthless as a blast furnace fuel. In the United States we have been very careless in the use of our high quality coking coals and are now being forced into using the higher ash coals which we once refused to consider. In the Latin-American countries, particularly in Brazil and Mexico, the only coking coals available are of high ash or sulfur content. Washing of these coals before carbonizing therefore becomes an absolute necessity. But because a coke of 25% ash content made from a washed coal may perform satisfactorily in a blast furnace where a coke of 35% ash content will not, it does not necessarily follow that further washing of the coal so as to produce a coke of 15% ash content will result in more desirable over-all results at the blast furnace.

Considering the over-all effect of coal washing, coke production and blast furnace operation, there is an optimum point of ash reduction in the washery beyond which it will be found more economical to remove the remaining ash as slag in the blast furnace. This may be stated in another and broader way as follows: "There is, for any specified coal, any specified washing plant and washing practice, and any specified end use of the coal, a definite point beyond which it is not economical to further reduce the ash in the coal washing plant" (1–a). This paper describes a method by which this optimum point of impurity removal may be determined; it applies the methods based on United States coals and operating practices, using as examples Latin-American coals and blast furnace practice.

ELIMINATION OF VARIABLES

There are a great number of variable conditions of practice which will be encountered in the combination of coal washery–coke oven–blast furnace operations.

Some of these are in no way affected by variations in washed coal ash. If the effect of changed ash is to be successfully evaluated, such variables must be maintained constant. Since it will obviously be impossible to select a number of operating periods of different levels of washed coal ash during which all such factors remained constant, one base period of a certain ash level must be selected and the effect of varying the level of ash content must be calculated.

Periods of different ash levels might, of course, be selected and the data from such periods corrected for the variations found which are not truly resultant from the ash. But such a method would be cumbersome, although it will be of value in comparing efficiencies and costs which may actually be found at different ash levels. Therefore, in making the calculations, it must be assumed that the factors of raw coal cost, analysis and washability, coal washery methods, through-put and operating cost, oven practice and operating procedure, blast furnace ore and limestone cost and analysis and operating practices, are not changed.

Conditions affected by variations in ash

A change in ash content of washed coal and coke will, however, be accompanied by variations in certain features of practice in each of the three departments. In order to wash the coal to a lower ash content it must be washed at a lower specific gravity and, as this is done, the difficulty of separation will become greater. This will be accompanied by a decrease in the efficiency of separation and an increasingly lower recovery of washed coal from that indicated by the float-ash curve, and the cost of the raw coal used to make one ton[1] of washed coal will increase, as will the total cost of washed coal. At the ovens, the increased volatile content of coal which will accompany reduced ash will result in a reduced total oven yield of coke and breeze, and an increased yield of by-products. It is needless to say that while the ash content of the coke is reduced the cost of the coke will be increased. The reduced coke ash will result in an increase in pig iron production at the blast furnaces as well as a decrease in coke used per ton of iron. The amount of limestone needed per ton will decrease and the overhead and fixed costs, assumed to remain constant per day, will be reduced per ton of iron. All of these factors are important in considering the effect of varying coke ash on blast furnace costs and each will be given consideration.

Coal washability

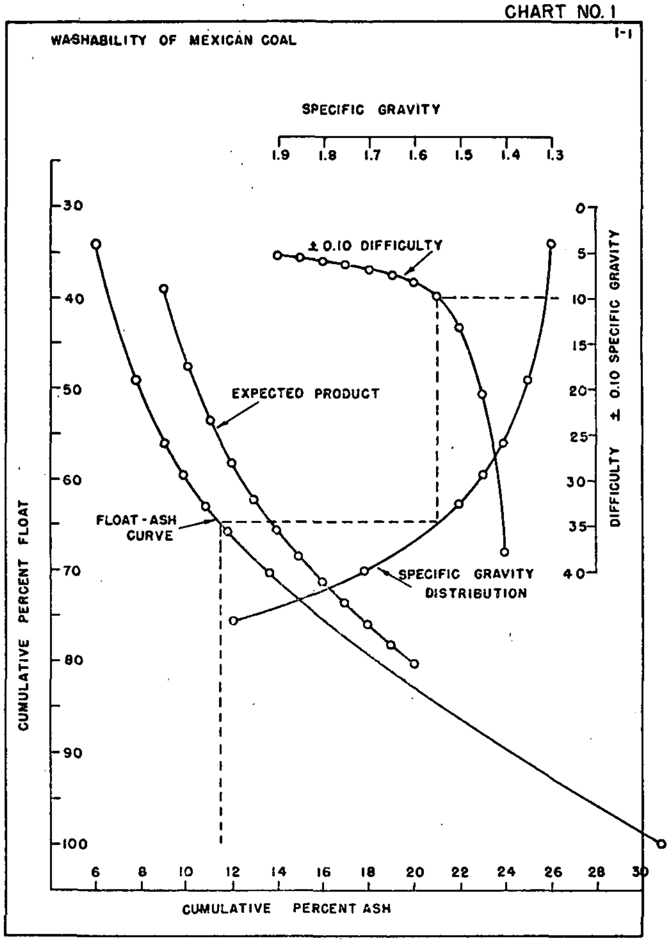
As a base coal for our calculations, I have selected a coal from the Sabinas region of Mexico which is currently being washed and carbonized for the production of metallurgical coke (2-a). Basic washability curves derived from float-and-sink tests are shown on Chart No. 1, the following curves being shown:

1. *The Float-Ash Curve,* obtained by plotting the cumulative per cent float against the cumulative ash. This curve provides a knowledge of the amounts of washed coal and refuse which are present in a coal at any particular ash content.

2. *The Specific Gravity Distribution Curve* is plotted from the cumulative per cent of float material at each point of specific gravity test. This curve is of use in determining the specific gravity at which a separation must be made in order to secure a washed coal of desired ash content.

3. *The Difficulty Curve* is determined by a calculation of the amount of material present in the coal within a range of plus and minus 0.10 S.G. of selected gravities on the Specific Gravity Distribution Curve. This curve is the one which tells us the most about the washability of our selected coal because, as the amount of "near gravity" material increases, so also does the difficulty of making a sharp separation and of close approach in practice to our theoretically perfect Float Ash Curve (5). A number of authors have given us data regarding the effect of difficulty on quantitative (1–b) efficiency (3) and on sharpness of separation (4). Such information serves to calculate a curve showing the relationship between difficulty and efficiency, as shown on Chart No. 2, Figure 1. Since each point of difficulty is based on a point of specific gravity, and since this in turn determines the ash to which it is theoretically possible to wash the coal, each point of difficulty, as shown on Chart No. 1, can be related to a point of theoretical ash in washed coal. From this relationship of difficulty to theoretical ash and from the curve of difficulty against quantitative efficiency, it is possible to calculate efficiencies for various ash contents. Such values are shown as Figure 2 on Chart No. 2. Applying the expected efficiencies as shown at each per cent of washed coal ash between 9 and 20 to the float-ash curve on Chart No. 1, a value is secured for expected yield at each per cent of ash. These are shown as the "Expected Product" Curve on Chart No. 1.

CHART NO. 1



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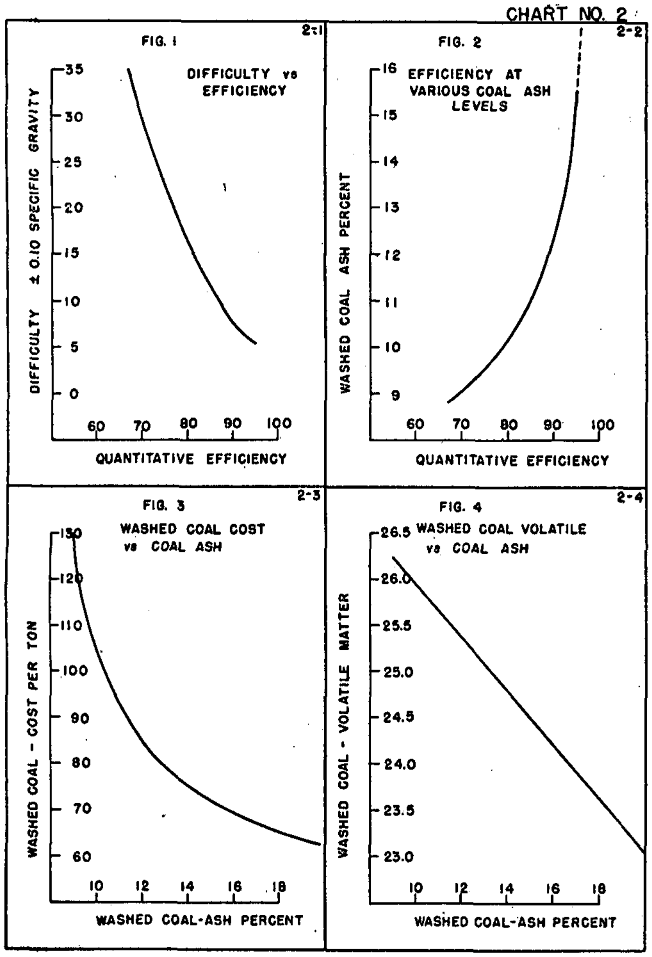
[1] The term "ton" refers to metric tons, approximately equal to 2,204 pounds.

CHART NO. 2



Cost of washed coal

It will be noted from the Expected Product Curve on Chart No. 1 that, as the coal is progressively washed to lower ash content, the yield of washed coal drops off rapidly, and, conversely, the loss in rejected material increases. This requires progressively greater quantities of raw coal for each decrease in ash and results in progressively greater increases in the cost of washed coal. Let us assume, in order to set up an example, that the raw coal described above can be delivered to the washing plant at a cost of 40 Mexican pesos per ton, and that the cost of washing is 10 Mexican pesos per ton of raw coal. Since we have already assumed that the amount of raw coal to the washery will be held constant per unit of time, the sum of the above costs divided by the actual percentage recovered at each per cent of washed coal ash will represent the cost of washed coal at that ash. The costs of washed coal so secured are shown as Figure 3 of Chart No. 2.

Yield and ash of coke

As coal is washed to a lower ash content, the volatile content of the washed coal increases. This variation for each per cent of ash, with our selected coal showing 24.5% volatile content at an ash of 15%, is calculated by applying the formula $(100-\%\ \text{Ash})\,\frac{(\text{VM at 15 ash})}{85}$ and the results so secured are shown as Figure 4 on Chart No. 2. The yield of coke in a by-product coke oven is determined from the volatile content of the coke and the carbon deposited on the coke from cracking of hydrocarbons in the gases. This deposited carbon will amount to approximately one-sixth of the volatile content of the coal when coked in by-product ovens at temperatures approaching 1,100°C. In the bee-hive oven a loss is encountered due to the combustion of a portion of the coal. Thus in a by-product oven the total oven yield will approximate $(100-\text{volatile matter})+(0.16\times\text{volatile matter})$ while in the bee-hive oven the formula will be $(100-\text{volatile matter})\times 0.875$. Curves showing the total oven yield in each type of oven at the various washed coal ashes are given as Figure 1 in Chart No. 3. Loss in breeze and ash in the by-product oven will be about 5% and in the bee-hive oven 6%; the deduction of these amounts from the total oven yield will give the net yield of blast furnace coke. From these curves it will be seen that at 15.0 ash in washed coal (18.8 ash in by-product coke) made from our base coal we indicate a total by-product oven yield of 79.58% and a total bee-hive oven yield of 66.06%. Deducting the breeze and ash losses we show blast furnace coke yields of 74.58 and 60.06%, respectively, for the two types of ovens. While a substantial portion of the blast furnace coke of Mexico is at present produced in bee-hive ovens, it is anticipated that practically all will be made in the by-product oven within a relatively short time. For this reason all further considerations are based on by-product coke. Ash in coke produced in a by-product oven may be calculated by dividing the washed coal ash by the total oven yield. Resultant coke ashes from washing the base coal to different ash levels are shown as Figure 2 on Chart No. 3. Because of obvious difficulties, no attempt is made to forecast the coke ash of coke from bee-hive ovens.

Cost of coke

As the oven yield decreases, the amount of washed coal needed to make one ton of coke increases and this factor of increase tends to further raise the cost of coke at the lower limits of ash content over the effect of the increased cost of the washed coal per ton. Dividing the cost of washed coal per ton at each per cent of washed coal ash by the blast furnace coke yield will give the cost of washed coal per ton of coke. There are several factors other than oven yield which are dependent upon the volatile matter of the washed coal. Increased volatile and the corresponding reduced oven yield will cause increased fuel consumption at the ovens per ton of coke and decreased production of coke per oven pushed. However, it has been demonstrated that these increased oven costs will be compensated by increased credits from recovered by-products. Therefore and since our purpose is to show the effect of variations in washed coal and coke ash and not to determine exact coke costs, it is not in error to state that this cost of washed coal per ton of coke approximates closely the cost of coke. This figure then, of cost of coke per ton for each per cent variation in washed coal ash, is shown as Figure 3 on Chart No. 3.

Blast furnace factors

It has been shown above that the production of washed coal and coke of progressively lower ash content results in progressive increases in the cost of coke. It is well known that certain costs at the blast

CHART NO. 3

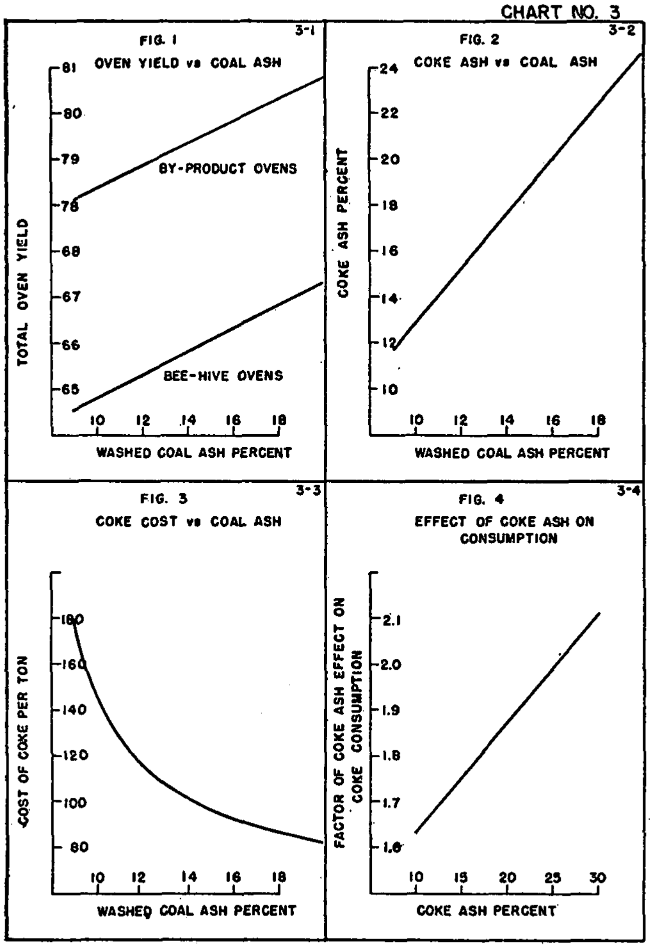
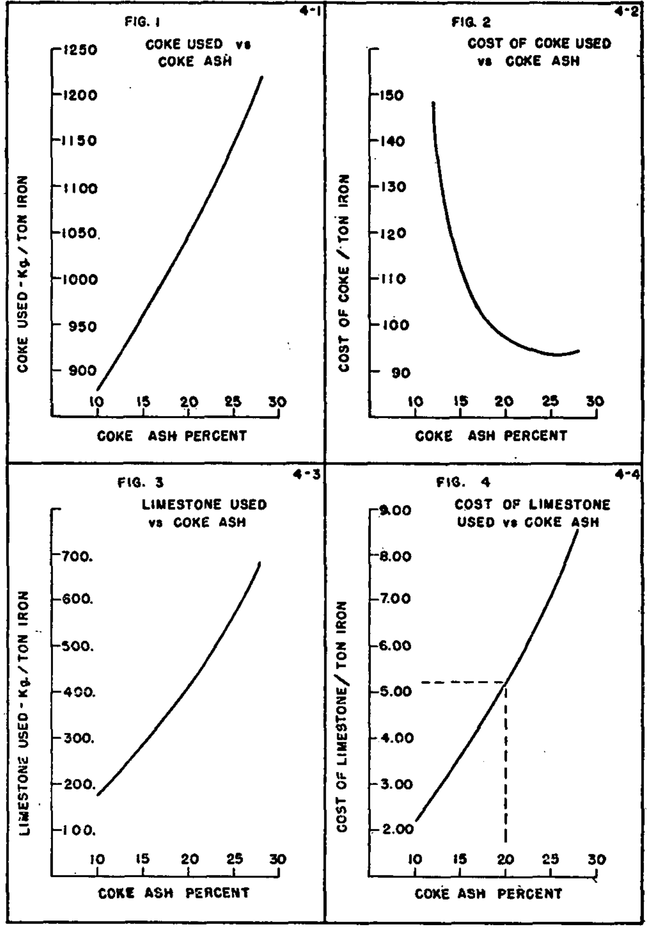


CHART NO. 4



furnace will progressively decrease as the ash in coke decreases. This will be found to be due to two factors. The first of these is the reduced coke consumption per ton of iron produced. At 20% ash in coke, a consumption of 1,050 kg. of coke for each ton of iron produced is not out of line. As the ash level in the coke varies above or below that amount, the coke consumption will also vary. Here it will be found that the variation in consumption for 1% ash is greater in the higher ash levels, 1% ash change at 10% ash level affecting the consumption 1.63% while, at 20% ash, a 1% change in ash will affect the consumption by 1.87% (6). This factor of coke ash effect on coke consumption is shown as Figure 4 on Chart No. 3. This factor takes into account the fact that slag volume increases with increased coke ash and that a portion of the available carbon is used in the melting of this slag. Actual coke consumption as affected by coke ash is shown as Figure 1 on Chart No. 4. This figure of coke used at each level of coke ash, multiplied by the cost of the coke at that ash level, gives us the cost of the coke per ton of iron. This information is shown as Figure 2 on Chart No. 4.

The second factor is a combination of the effect of changed limestone requirements brought about by variations in coke ash and the effect on furnace operating costs of variations in iron production. Assuming coke ash to be all acids, it will require two parts by weight of stone to flux each part by weight of ash (2–c). With a normal usage of 420 kg. of stone per ton of iron at a coke ash level of 20%, the variations in limestone requirement will be as shown as Figure 3 on Chart No. 4. At a cost of 12.50 Mexican pesos per ton, the cost of the limestone will be as in Figure 4 of Chart No. 4.

The effect of blast furnace costs due to variations in coke ash is caused by the increase in the daily production of iron, which will result from decreased coke ash. It is generally accepted that a blast furnace will handle only a given amount of coke per day, and this regardless of the ash content of the coke. As the ash in the coke decreases the total carbon to the furnace increases, and thus makes possible the addition of increased amounts of ore. With a normal production of iron of 300 tons per day with a coke containing 20% ash, expected variations in production with changing coke ash is given in Figure 1 of Chart No. 5. Accepting 50 Mexican pesos per ton of iron with 20% ash coke as a normal blast furnace operating cost, the effect of variations on blast furnace operating costs is shown as Figure 2 on Chart No. 5.

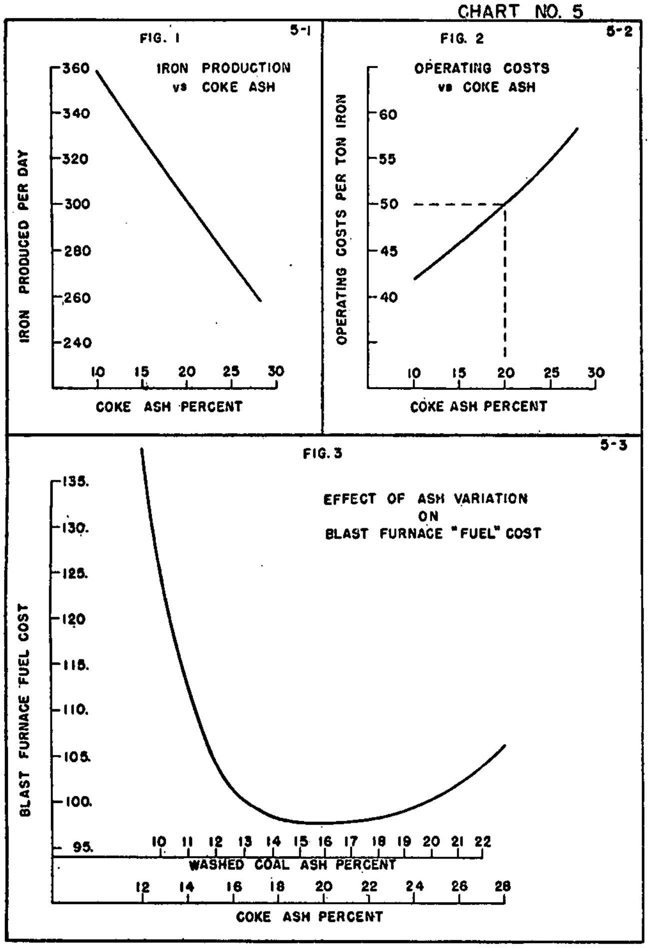
Blast furnace fuel costs

We have now accumulated the necessary data from which to calculate the combined effects of the various factors which have been discussed. These are as follows:

(1) The cost of coke per ton of iron at each level of coke ash, shown as Figure 2 on Chart No. 4.

(2) The gain or loss resulting from variations in cost of limestone from the normal cost of 5.25 Mexican pesos, shown as Figure 4 on Chart No. 4.

CHART NO. 5



(3) The gain or loss resulting from variations in blast furnace operating costs from the normal of 50 Mexican pesos and shown as Figure 2 on Chart No. 5.

These various effects on cost may be taken from these charts at the various levels of coke ash, and if cumulated will provide a monetary indication of the effect of varying ash levels on blast furnace fuel costs. In this case the term "fuel" is somewhat a misnomer but is used for want of a better name. Rather, this cumulated cost figure represents, for each level of coke ash, the monetary effect of such variation on all factors of blast furnace cost subject to such effect. This total fuel cost is shown as Figure 3 on Chart No. 5. It will be noted from the curve that the lowest cost of fuel to the furnace is at a coke ash level of 20%, corresponding to a washed coal ash of 16%, and that operating the coal washery to produce washed coal at this ash should give, in the case of the example here illustrated, best over-all economic efficiency.

Conclusion

It should again be pointed out that the above data is shown for purpose of illustration only. While it closely approaches the actual practice and costs of a Latin-American blast furnace plant, the data as shown is not exact and the conclusion as reached in the last curve must not be accepted as final for any plant. However, any operator, from his own practice, may insert specific and exact data in place of the approximations here used and by so doing secure correct final information as to the over-all efficiency of such combined operations.

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The Influence of the Ash Content on the Hardness of Cokes Made from High-Volatile Coals

WALTER VOGEL

General aspects

Those countries which do not possess bituminous coals with good coking properties, but which do have all the other basic factors for a steel industry, such as iron ores, fluxes, cheap electric power and a sufficiently large market, have had to study ways and means of compensating for the lack of such coals.

Two different approaches have been used in this question:

(1) On the metallurgical side, employing reduction methods which permit the use of other non-coking fuels, or else

(2) On the side of the available coals, trying to obtain a coke which can be used in the blast furnace, by blending the coals with a certain proportion of good coking coal brought from other sources.

While in the first there is the inconvenience of lesser-known and usually more costly metallurgical processes than the blast furnace, it is necessary in choosing the second course to meet the higher costs of the blending coal brought at high freight charges and the uncertainty involved in obtaining supplies from great distances. It must be remembered that this is a raw material which cannot be stored in large emergency stocks, because of the loss of coking properties a certain time after extraction from the mine.

This paper refers to observations related to this latter solution, when using younger high-volatile coals. It is based on studies of the Chilean coals in the Bay of Arauco, which constitute the basic fuel for the new Chilean steel industry. Its purpose is to clarify the possibilities for improving coking properties at the coal preparation stage, eliminating or reducing the unfavourable elements in order to reduce to a minimum the necessity of bituminous coal additions.

THE INFLUENCE OF THE PETROGRAPHIC COMPONENTS OF THE COAL

As usually the younger coals in themselves have less coking power, and the coke irremediably has a fingery structure because of the heavy contraction due to the greater volatile content, it is logical that they should be very susceptible to any factor which reduces the adherence of the particles among themselves or produces fractures which are transversal to the axis of the core fingers.

In the Chilean case it was necessary to make a detailed selection not only of the different seams but even of the workings within each seam in order to achieve satisfactory results. This alone shows that there are petrographic factors to be considered.

The influence of the petrographic constituents has been considerably studied in countries where coking has been more developed, and it is well known that the favourable conditions of clarite and vitrite have been shown, as well as the unfavourable ones of durite, fusinite and, in particular, coal-bearing schists.

In bituminous coals the components of durite and semi-fusinite are harder than the components of vitrite and clarite, so that logically there is a differentiation in size, the smaller coal due to the greater concentration of vitrinite giving a better coke than the nuts or pieces of the same coal. A differentiation of this nature is not observed in the Chilean coals, the smaller coal[1] of which, other conditions being equal, gives a slightly more fragile coke. This is due to the fact that the vitrite mass of these coals shows almost entirely a cellular structure, and under the microscope does not show the typical shrinkage fissures which characterize the vitrinite in older bituminous coals and cause their greater fragility. The sole granulometric differentiation to be noted is the concentration of fusinite and schist in the fine dust of that coal. This explains the different behaviour of the nuts and small coal of the younger coals such as those of Chile, in relation to the bituminous ones of the older geological formations.

On the other hand there is another feature which is equally present in the younger coals, and this is the greater percentage of ashes in the components which are adverse to coking. As an illustration some values of ash contents of the petrographic components are given below:[2]

	Seam 14 Wilhelmina Victoria	*East Kirkley*	*Hampstead*	*Johann Deimelsberg*	*Bruckenberg*	*Winterslag*
Vitrite . . . %	0.5	0.9	1.11	1.5	2.3	3.1
Clarite . . . %	0.6	1.3	1.22	4.0		
Durite . . . %	3.4	7.8	6.26	12.6	51.0	4.0
Fusite . . . . %	5.0	13.8	15.59	8.0	30.6	11.4

These are analyses of ashes in the layers of coal, hand picked for their macroscopic aspect. This explains why fusite comprises both the fusinite component as well as part of the semi-fusinite, the former being less charged with mineral substances than the second. On the other hand the durite also includes the carbonaceous schist scattered in the combustible matter. If the petrographic components are considered separately, the following rising scale in the ash content would be found:

Vitrinite
Fusinite
Resinite
Exinite
Micrinite } Main components of durite
Semi-fusinite
Carbonaceous schist

It should also be mentioned that the observations of many researchers agree that fusinite, duritic grains, semi-fusinite and carbonaceous schist do not modify their outward shape during coking. Fusinite, on account of its vegetable fossil carbon, is of course absolutely inert, but where it is found clean, the coking bitumins still adhere to its surface. With durite, on the other hand, it may be observed that the superficial tension of the coking bitumen, during its liquid or pasty state, is very weak in relation to the organic matter of the durite; this gives rise to a lack of resistance along the contours of the durite grains and the production of transversal cracks. A mere glance is often sufficient to see how these transversal cracks separate from the larger durite grains in cokes made from high-volatile coals. Much greater still is this influence in the particles of carbonaceous schist which do not show any adherence by the surrounding coke. Its contours indicate that the mass has contracted. Of course these trends are all the more noticeable when the grains of these components are larger.

These observations gave rise to the practice of crushing small coal down to less than 3 mm. and blending it well. In coals with better coking properties this practice itself eliminates the inconveniences of these constituents of the coal.

In bituminous, high-volatile coals, this crushing and blending is not sufficient. Coals of this type undergo a sharp contraction during coking, which is directed towards the centres of the particles with greater superficial surface tension to the bitumen, which in turn retracts from those to which it does not adhere, and in particular from the carbonaceous schist.

Years ago the author had an opportunity of studying these effects in detail as a result of difficulties which arose in the Borbeck blast furnace belonging to Krupp, when the Hanover mine which supplied the coking coal for this furnace started to work a new seam under circumstances whereby it was forced simultaneously to make substantial changes in its preparation plant. The study carried out showed that on the one hand the coal of the new seam was characterized by a relatively high percentage of carbonaceous schist and that the changes made in the washing plant had resulted in a poorer elimination of precisely the smaller particles of this carbonaceous schist. This was due to the fact that the dust extractors had been temporarily replaced by vibrating screens with rectangular mesh. In the dust extractors, the schist passes with the bulk to the wash, whereas in rectangular screens, due to its flat structure, it passes directly to the dust which, in the Ruhr, is added without further cleaning to the

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[1] Below 20 mm.

[2] Winter, *Gluckauf Review*, 1936, No. 50.

small coal. It was also seen that the coal from the new seam did not wash well because the fraction below 10 mm. retained a quantity of grains of intermediate specific gravity, charged with unfavourable petrographic components, and in particular with carbonaceous schist. The resulting coke had a large proportion of small sizes. The air pressure in the blast furnace rose considerably and serious difficulties arose in spite of the fact that all the other factors remained constant. After a series of changes in the washing system and a petrographic control of the percentage of carbonaceous schist in the small coal added for coking, a coke was obtained which, although fingery, had less transversal cracks and resisted blast furnace work perfectly. The interesting point is that after having achieved this result and readjusted the operation of the blast furnace, it became apparent that the coke rate could be reduced. This was also reflected in the composition of the blast furnace gas. The explanation lay in the greater reaction power of the cokes obtained from high-volatile coals.

Similar observations have been made in Huachipato, and have resulted in the blast furnace having a greater capacity than was originally foreseen.

It was precisely the earlier experiences in the case of the Hanover mine which led the author to conduct a study on the influence of ash content on the quality of the coke based on the fact, already explained, that the unfavourable components have a higher ash content. The routine reports of the coal testing laboratory at Huachipato were used in this study.

THE INFLUENCE OF ASH CONTENT ON THE HARDNESS OF COKE FROM THE CHILEAN COALS OF ARAUCO BAY

The Chilean coals of Arauco Bay belong geologically speaking to the Eocene phase of the Tertiary Age. They are characterized by the following average analysis:

Table 1

Seams	*Volatile matter*	*Fixed carbon*	*Ash*	*Sulfur total*
	(in % of pure carbon)		*(in % of dry carbon)*	
Schwager mine, No. 5				
Pieces 20 mm. selected.....	42.5	57.5	5.1	1.25
Small coal 20 mm. unwashed...	44.9	55.1	18.4	1.26
Small coal 20 mm. washed.....	44.3	55.7	3.5	0.97
Lota mine, Alta				
Pieces 20 mm. selected.....	43.9	56.1	2.9	0.75
Small coal 20 mm. unwashed...	44.5	55.5	10.5	0.75
Schwager mine, No. 3				
Pieces 20 mm. selected.....	43.3	56.7	3.1	0.71
Small coal 20 mm. unwashed...	43.0	57.0	11.5	0.77
Small coal 20 mm. washed.....	42.7	57.3	1.7	0.69
Lota mine, Alta				
Pieces 20 mm. selected.....	43.8	56.2	3.4	0.96
Small coal 20 mm. unwashed...	45.5	54.5	15.3	0.86
Small coal 20 mm. washed.....				
Schwager mine, No. 2				
Pieces 20 mm. selected.....	48.5	51.5	7.1	2.37
Small coal 20 mm. unwashed...	45.9	54.1	15.2	1.38
Small coal 20 mm. washed.....	48.3	51.7	4.0	1.83

In the crucible the coals give compact but slightly swollen coke buttons. They already show a progressive change of coking power, according to the sequence of the preceding table, the Seam No. 5 of the Schwager mine being the most favourable, and No. 2 of the same mine the least advantageous. Incidentally, the Lota Alta Seam corresponds to Schwager No. 5, and the Arriba to No. 3.

To compare coking results, I have preferred to refer exclusively to the results of the drum test and selected the criterion used in Germany to appreciate the results of this test, known as the Ilsede figure. It corresponds to the difference between the stability coefficient (remainder over 26 mm. [1 in.]) and that of hardness (per cent which passes through the 6.5 mm. screen [¼ in.]). My reasons for preferring the Ilsede index lie in an observation made during the aforementioned experience. In the Krupp blast furnace at Borbeck, the inverse ratio of the changes in blast pressure required for a given charge was much better correlated with the Ilsede figure than with any other index of tests or combinations of figures which have been proposed for assessing the characteristics of cokes. In this, moreover, the author coincides with Yancey and his collaborators.

It was nevertheless necessary to base the study of Chilean coals not on the drum test according to the "Micum" or "ASTM" rules, but on that variation derived from the second known as the Columbia drum test, which was used particularly for studying Utah coals similar to the Chilean ones. This test varies the conditions of the drum test to a certain extent, making them less severe. For high-volatile coals, I consider that it has decided advantages.

The reports of the Huachipato coal-testing laboratory referred to a group of coking tests conducted under the same conditions in an experimental oven with a narrow chamber, the same as those installed in the coking plant, that is, 35 cm., and a final temperature of 1,050°C. for a coking time of 13 hours.

Of a large number of tests conducted, only those were selected which would serve for the purposes of the research described in this paper.

Table 2 summarizes the most important figures for No. 5 Seam or Alta seam.

This table shows that the index figure of the Columbia drum test usually remains below the figure of 60 shown for those cokes suitable for blast furnaces (Fontana and Huachipato). In order to reach this figure, however, up to 20% of bituminous coals have to be added to Chilean coals. The problem posed to Chilean industry is how to lower this percentage.

The table shows an obvious dependence between the index figure and the ash content. This is particularly clear in comparing the tests of unwashed coal below 20 mm. with the washed product from the same sample. This washing took place in organic heavy media liquids at a specific gravity of 1.35 and is therefore theoretically a perfect separation at that density. As may be seen from Figure 1, which summarizes the data of the table, the coking power of Seam No. 5/Alta varies from one section of the mine to another, but within the same section the influence of the ash and with it of the those petrographic components containing most ash is decisive, and there is a marked parallel between the lines of the Lota sections and those of San Pedro-Schwager and between the San José and Arauco sections, there being only a slight difference between both groups. The index figure diminishes by an average of 2.3 points for each 1% of increase in the ash content.

Table 2

Section of the mine	*% coke 20 mm.*	*Ilsede index Columbia drum*	*% of ash*	*% of sulfur*
Envidia-Schwager				
Selected screened	90.8	39.0	4.2	0.97
Washed small coal	—	—	—	—
Unwashed small coal	86.1	20.6	11.4	1.03
San Pedro-Schwager				
Screened	91.5	37.5	3.7	1.11
Washed small coal	92	37.5	1.9	0.91
Unwashed small coal	85.8	15.8	11.1	1.21
San José-Schwager				
Screened	91.4	32.2	6.0	5.1
Washed small coal	89.7	29.0	3.8	1.09
Unwashed small coal	78.1	−17.9	26.5	1.64
Arauco-Schwager				
Screened	89.6	26.7	6.7	1.62
75% washed small coal + 25% screened	87.8	26.9	3.6	0.99
Unwashed small coal	73.3	−33.7	30.1	0.95
Pique Nuevo-Lota				
Screened	90.0	32.3	2.5	0.69
Washed small coal	90.0	31.4	2.2	0.75
Unwashed small coal	71.5	−10.6	19.6	0.75
Fortuna (Pique Grande) Lota				
Screened	89.1	27.4	2.7	0.68
Unwashed small coal	74.8	9.1	10.4	
San Juan-(Pique Grande) Lota				
Screened	89.3	31.4	2.9	0.83
Unwashed small coal	73.9	+4.8	14.0	0.87
Pique Alberto-Lota				
Screened	86.1	25.7	2.9	0.64
Unwashed small coal	64.8	11.6	10.4	0.75
90% Schwager No. 5 screened 10% Pocahontas No. 7	94.3	51.1	3.0	0.98
80% Schwager No. 5 screened 20% Pocahontas No. 7	96.4	64.7	3.3	0.96
90% Lota Alta screened 10% Pocahontas No. 7	93.9	57.6	3.7	0.71
80% Lota Alta screened 20% Pocahontas No. 7	96.3	69.2	4.1	0.69
100% Sunnyside (Fontana)	87.8	45.45		
87.5% Sunnyside 12.5% Oklahoma Fontana	91.2	62.60		

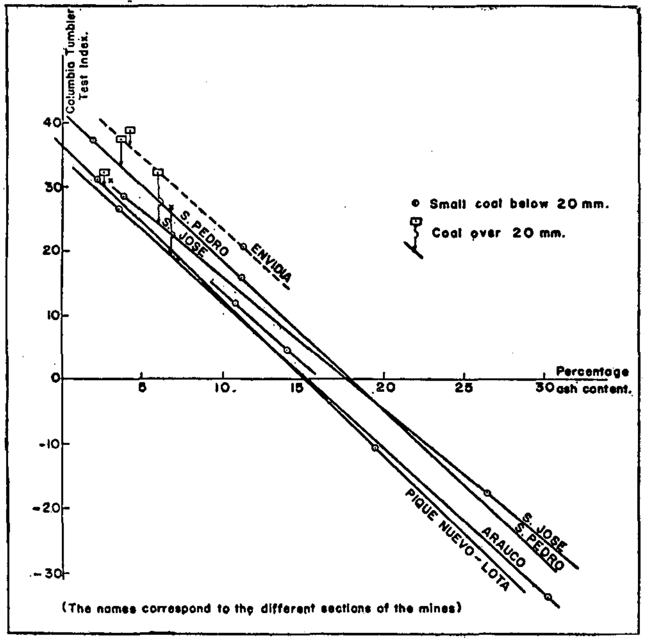
Now if the figures for lump coal are compared with the corresponding lines for the small coals from the same sections of the mines, it may be seen that for Seam No. 5/Alta the lump coal has an index figure from 4 to 7 points above that of the small coal with the same percentage of ashes for the Schwager sections, the Lota ones remaining partly above and partly below those for the small coal. It is very probable that the greater proportion of fusinite in the small coal is the reason for these differences.

A similar study for Seam No. 3/Arriba may be seen from the following table:

Table 3

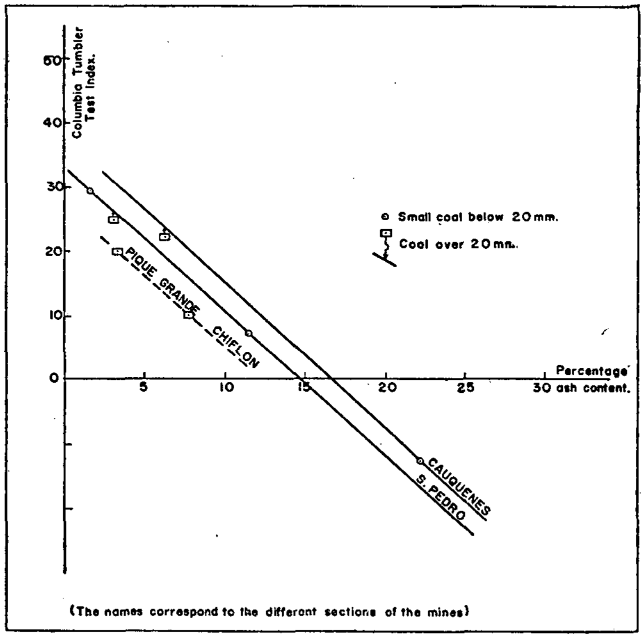
Section of the mine	*% coke 20 mm.*	*Ilsede index Columbia drum*	*% of ash*	*% of sulfur*
San Pedro-Schwager				
Lumps	86.9	25.0	3.1	0.71
Washed small coal	88.1	29.6	1.7	0.69
Unwashed small coal	84.5	7.2	11.5	0.77
Cauquenes-Schwager				
Lumps	89.4	22.4	6.3	4.05
Unwashed small coal	89	−12.9	22.2	3.48
Chiflon Carlos-Lota				
Lumps	82.9	20.0	3.4	0.56
Unwashed small coal	57.9	−10.2	15.3	0.86
Pique Grande-Lota				
Lumps	83.9	10.2	7.9	4.60
Blends				
90% San Pedro-Schwager 10% Pocahontas No. 7	92.2	43.8	3.2	0.69
80% San Pedro-Schwager 20% Pocahontas No. 7	95.6	61.3	4.0	0.67

Figure 2 permits a better study of the joint ratio of the different values of the drum index. It shows clearly that this index falls by 2.3 points for each 1% of increase in ashes, that is, in the same average proportion seen in Seam No. 5.



(The names correspond to the different sections of the mines)

Figure 1
Variations in the Columbia Tumbler Test index figure for cokes from Seam 5-Alta in relation to the ash content of the coal.



(The names correspond to the different sections of the mines)

Figure 2
Variations in the Columbia Tumbler Test index figure for cokes from Seam 3-Arriba in relation to the ash content of the coal.

Seam No. 3 gives average drum figures some 9 points below those for Seam No. 5 for equal ash conditions.

Whereas in No. 5 the lump coal showed better conditions than the small coal, in Seam No. 3 there is not much difference between the two figures.

The foregoing tables include the results obtained by the addition to the screened coals of a good coking coal with low-volatile matter from the respective seams.

It is seen that the improvement for each 1% of addition reflected by the index figure is as follows:

	Average first 10%	*Average between 10 and 20%*
Screened Seam No. 5, Schwager	1.73	1.36
Screened Seam Alta, Lota	1.84	1.16
Screened Seam No. 3, Schwager	1.88	1.75

Although no experiments have been made on the influence of ashes on the quality of the coke from the blend, it may be deduced that in the worst of cases a 1% reduction of the ash content would give the same result as an addition of 2.30 : 1.84 = 1.25% of Pocahontas No. 7 coal. This is important because foreign coal costs practically twice as much as national coal.

The economic factor of a reduction in the ash content of coking coal can now be analysed; but first the sulfur problem must be considered.

THE SULFUR CONTENT OF CHILEAN COALS

The first objection of a metallurgical nature against any considerable decline in the ash content of coke would be that the corresponding reduction of the slag in the blast furnace would hamper the control of the sulfur content in the pig iron. It may be shown, however, that the opposite is the case.

In fact if the washability curve of the coal is compared with that of the sulfur content of the different fractions of sink and float of the Chilean coals, it will be found, as shown in the typical case of the 1.5 to 25 mm. from Seam No. 5 given in Figure 3, that the fractions with specific gravities exceeding 1.3 have a sulfur content higher than that corresponding to the carbonaceous matter which they contain. Thus for example the fraction between the specific gravities 1.30 and 1.35 already have 2% of sulfur instead of the 1% contained in the carbonaceous matter and the maximum of sulfur is shown in the fraction between 1.50 and 1.70, which reaches 4.5%. This increase in the percentage of sulfur in the fractions of intermediate density is very frequent.

THE ECONOMIC VALUE OF THE FRACTIONS OF INTERMEDIATE DENSITY OF CHILEAN COALS

In order to arrive at the economic position of a stricter washing, the economic values of the different fractions of density of the small coal from Seam No. 5 will be calculated, following Bansen.[1]

He bases his calculations on a self-sufficient coke, that is, a coal having an ash content which results in the remaining pure coke being barely sufficient for the operation of slagging and melting the slag of its own ash, and fixing of its own sulfur. This coke has the following theoretical composition:

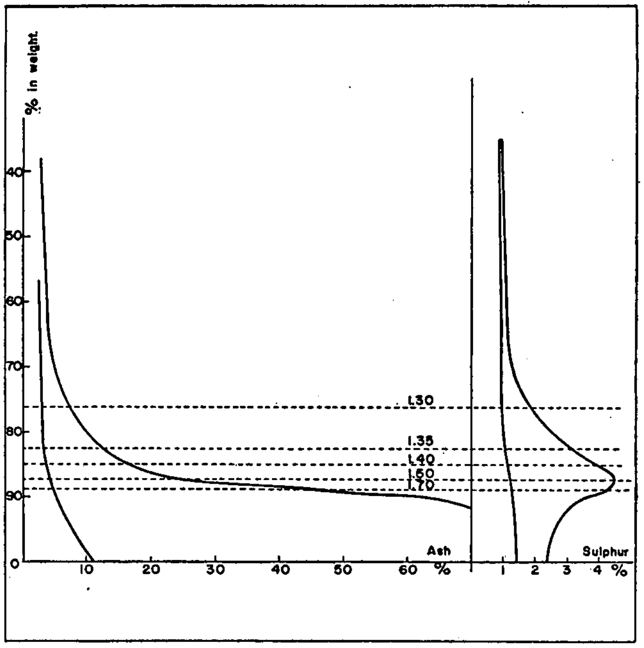


Figure 3
Curves for washability and sulfur content.
Fines below 25 mm. Seam 5–Schwager.

Ashes	58.2%
Sulfur	1.8%
Pure coke	40.0%

It requires 110% of its weight in limestone and produces 7.9% of pig iron. The value of this self-sufficient coke corresponds to the value of the pig iron, slags and usable gases less the costs of operation, maintenance, amortization and interest of the blast furnace and its adjoining installations. These last costs may be estimated at U.S.$15 per ton of pure coke burned in the blast furnace. The following balance would result for this self-sufficient coke:

	U.S.$
Value of 79 kg. of pig iron at $0.06/kg	47.4
Value of the blast furnace gases	0.16
Value of 1.1 ton of slag at U.S.$4/ton	4.40
	9.30
Less	
Operating costs, etc., of the blast furnace, U.S.$15 per ton of pure coke×0.4	6.00
Cost of 1.1 ton of limestone at U.S.$7	7.70
	13.70
Adverse balance per ton of self-sufficient coke	4.40

As only 80% of the coke produced passes to the blast furnace, the rest being used as fuel, and for these cases the self-sufficient coke would have approximately the value 0, then for every ton of this coke produced in the coking plant, the blast furnace would have a loss of U.S.$3.52. Added to this, there is the loss from the coking plant itself as all the sterile matter, which gives no by-products, passes through it and all the costs involved have to be covered. This loss of by-products represents per ton of self-sufficient

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[1] Bansen and Krebs, *Archiv für das eisenhüttenwesen*, 1940/41, p. 91.

coke a loss of $\frac{55}{100} \times 7.0 = \$3.85$. Altogether, solely because of the higher ash content, the self-sufficient coke means a loss per ton of $7.37, that is a lower value in relation to normal coke of U.S.$37.37.

Let us see now the surcharge produced on costs per kg. of sulfur which cannot be absorbed by the normal slag of the charge. If it is not desired to resort to desulfurization through the soda procedure, which has a series of drawbacks, the slag content of the charge will have to be increased artificially at the rate of 52 kg. of slag per kg. of sulfur, plus the addition of limestone required by this kg. to produce calcium sulphide. Even assuming that there is suitable material available in the plant at no cost, it would have to count on an addition of some 30 kg. of limestone and an additional consumption of 30 kg. of coke. The costs which originate therefore from the elimination of 1 kg. of sulfur would be:

		U.S.$
Cost of limestone	30 kg.×0.007	=0.21
Cost of coke	30 kg.×0.030	=0.90
Cost of operating the blast furnace per ton of coke	30 kg.×0.015	=0.45
		1.56

As these costs only originate for that portion of the coke which passes through the blast furnace, that is, 80% of that produced, there is an additional loss of U.S.$1.25 per kg. of sulfur of the coke which is not eliminated in the slag of the normal charge.

In order to evaluate the different fractions of density of the coals, we may consider the resulting coke from each of them as being composed of a part of normal coke (from 7.1% of ashes and 1% sulfur), self-sufficient coke (58.2% of ashes and 1.8% of sulfur) and the surplus of sulfur. This would give the following table for the different fractions of Seam No. 5 of Schwager:

Table 4

	Composition coal			*Composition coke*		*Ideal composition of coke per ton*			*Lower value in relation to normal coke*			
Density fraction	*Ash %*	*Sulfur %*	*Yield in coke %*	*Ash %*	*Sulfur %*	*Kg. normal coke*	*Kg. self-sufficient coke*	*Kg. excess of sulfur*	*Self-sufficient coke $*	*Sulfur $*		*Total U.S.$*
1.30–1.35	9.1	2.3	63.6	14.3	2.2	844	145	11.0	5.42	+13.75	=	19.17
1.35–1.40	13.8	3.5	65.5	21.0	3.3	704	275	20.8	10.28	+26.00	=	36.28
1.40–1.50	19.9	4.4	68.0	29.4	4.2	534	437	28.5	16.33	+35.63	=	51.96
1.50–1.70	32.2	4.5	72.8	44.3	4.3	245	728	27.2	27.20	+34.00	=	61.20

This lower value of the coke in the different fractions corresponds entirely to the lower value of the original coal, to which must still be added the lower value resulting from the greater percentage of blending with foreign coal which is set at $0.125 for each 1% of addition. The following value then results for the fractions of Seam No. 5-Schwager:

Table 5

	Composition of the coal					
	Ash %	*Sulfur %*	*Greater % of ash over the normal*	*Cost of the higher addition of foreign coal $*	*Value of the fraction $*	*Lower total value U.S.$*
	4.3	1.05	—	—	—	—
Fraction 1.30–1.35	9.1	2.30	4.8	0.60	12.15	12.75
" 1.35–1.40	13.8	3.5	9.5	1.88	23.78	26.66
" 1.40–1.50	19.9	4.4	15.6	1.95	35.35	37.30
" 1.50–1.70	32.2	4.5	27.9	3.50	44.50	48.00

In the preceding tables, the fractions exceeding 1.70 were not considered because there is no discussion as to whether these ought to be eliminated in the wash.

From the above table it can be clearly seen that these fractions with a specific gravity exceeding 1.35 should be eliminated because they have a negative value in the blast furnace of 15.66 which exceeds the value of $10 for the coal which the mine needs to meet its economic requirements. It is therefore cheaper to pay these $10 to the mines, rather than to use this fraction.

Figure 4 shows that the separation, where the washing equipment is at the plant, should be done at the specific gravity of 1.35, considering that the fractions eliminated would have no other use. As in actual fact they always have some value as fuel, the cheapest separation would really be at the specific gravity of 1.30.

It should be mentioned that the new processes of phase separation should also be considered, as they may be particularly favourable for treating these intermediate fractions. Pitch, which in any case improves the hardness of the coke, may be used entirely as a

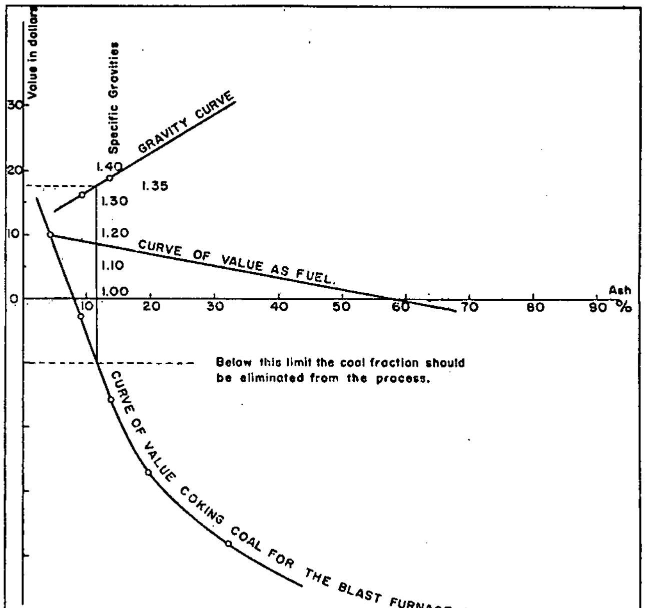

Figure 4
Curve of relative value of coking coal for the Huachipato blast furnace extracted from Seam 5–Schwager.

substitute for, or a complement to, the heavy oils required by these processes for purifying the intermediate fractions between 1.30 and 1.70 and the treatment of fines below 1.5 mm. As the explanation of these procedures will be made in a separate paper, no further detail will be given here. It is sufficient to indicate that when applied to the intermediate fractions between 1.30 and 1.70, which in the case of the sample from Seam No. 5 represents 15% of the small coal with an average of 13.5% ashes, these processes would result in an 80% yield. This is 12% of the total of the original small coal, with only 2 to 3% ashes and 1% of sulfur added and well blended with 8% of pitch. As this process costs less than $2 per ton, these fractions acquire a value of $\frac{80}{100}$x 10 − 2 = $6 instead of the average negative value of −$15 which they would have in passing through the coking plant and the blast furnace. To this balance should be added the higher coking value which they would attain through the addition of pitch.

From all the preceding discussion it may be seen that in the case of Chilean coals, a greater elimination of ashes through a more rigorous wash at low densities, far from being onerous, would benefit the general economy of the plant.

Should the washing plant be installed at and operated by the mines, it would only be a matter of agreeing upon a price scale in relation to the ash and sulfur content. This scale should be prepared to meet the economic requirements of the mine, together with envisaging the qualitative requirements of the steel plant.

This study based on the case of Chilean coals will certainly find application to many analogous cases.

IDEAS ON THE DESIGN OF WASHING PLANTS FOR HIGH-VOLATILE COKING COALS

From the foregoing, it may be gathered that in the mechanical preparation of high-volatile coking coals it would be useful, if not essential, to consider the following points:

(1) The accumulation of ashes and fusinite in the grains below 1.5 mm. in this type of coal requires their prior separation, to be specially treated or used for other purposes.

(2) In washing the grains exceeding 1.5 mm., the fractions with intermediate specific gravities should be eliminated as much as possible, because of their higher content of petrographic components adverse to coking, their higher ash and sulfur contents.

(3) The above makes it essential to use washing methods which ensure a separation with the minimum imperfection even for low densities (1.35 for example) and which prevent as far as possible the formation of clayey agglomerations which contaminate the washing water, and consequently the washed products.

(4) Where the market has no economic utilization for the intermediate products it would be worth while to consider for them, as well as for the fines below 1.5 mm., the application of the new phase separation processes.

In order to meet these requirements it will be necessary, of course, to discard the procedure of pneumatic concentrating tables because of their great imperfections precisely in separating at low densities and with schisty grain coals. Special attention should be drawn to this fact because there may be a tendency to exaggerate the advantage to be obtained from the greater density of the charge in the chambers of the coke batteries, by the use of fines with less than 4% moisture. The importance of this factor has, however, been exaggerated as a result of the tests made in experimental ovens; these do not reflect the conditions of bulk weight of charge which are in fact obtained in modern coking plants with very high chambers and the resultant compression of the charge. The figures given here indicate that the advantages in coke hardness, if in fact it is improved by higher bulk weights, may be very much lower than the drop in the index figure through the higher content of undesirable petrographic elements and the higher percentages of ash and sulfur, which are inevitable with the use of the pneumatic process.

It would be difficult to obtain the desired results with ordinary jigs, owing to bad separation at low densities, as a narrower margin has to be demanded in these fractions between the maximum and minimum size of grain. It therefore follows that in these cases, the normal flow sheet of Baum plants, washing without previous screening all the coal comprised between 1 and 100 mm. and treating the middlings in a second stage, is not adequate. Plants with this distribution are known to give, apart from a lower total yield, a worse separation of the intermediate fractions in the smaller grains, which tend to pass into the washed coal. That is to say, there is neglect of those fractions which most require a good separation with a view to bringing the coking properties of coal below 20 mm. up to those of the lump coal. Because of its simplicity, this process can only be recommended in those cases where there is no need for exact separations at low densities. In the case of Chile, and certainly also in that of various other Latin-American countries, these exact separations will be essential if it is desired to reduce dependence on imported coal.

There is no doubt that the ideal solution for these cases is washing by heavy media, particularly after these processes have been developed to the extent where they are competitive both as regards investment and operating costs, if due credit is given to their better yields. In any case separation into three products will have to be considered, using the intermediate products for other purposes or subjecting them after fine grading to a second treatment by a phase separation process.

Development of the Cauca Valley Coals

ALBERTO VARGAS MARTÍNEZ AND THOMAS FRASER

INTRODUCTION

The rapid economic and industrial development of the Cauca Valley has made it necessary to modernize the collieries in the region in order to meet the growing demand for fuel of the quality required by industry and the railways. Furthermore, interesting export prospects are opened up due to the strategic geographic location of these deposits in relation to the Pacific Coast markets of the American hemisphere. These coalfields, together with those in the region of the Gulf of Arauco (Chile), are the only ones being worked on the west coast of the hemisphere that can supply coking coal to a Pacific port and they may therefore be an important factor in developing the Latin-American steel industry.

THE DEPOSITS

A. *Geology*

The Cauca Valley coal deposits have been studied by prominent geologists, the most complete survey having been made by Messrs. Benjamín Alvarado and Enrique Hubach in 1945 (1).

All the coals in the region are tertiary, but they have been subject to various tectonic actions, the formations extending south-west almost without a break, resulting in a great variety as to quality and types. From the area in the immediate vicinity of Cali, southwards to the Ecuadorean frontier, there are found five to 25 workable seams, running approximately parallel, with a sharp inclination towards the Valley.

B. *Reserves*

The most complete estimate of the reserves was that made by Alvarado and Hubach, who estimated a recoverable total of 406,500,000 tons, distributed by districts as indicated in Table 1.

Although it is not yet possible to evaluate completely the coal deposits which may exist in the deepest parts of the formation, exploration to date indicates that there are sufficient reserves to cover the increasing requirements of industry in the Valley and to provide for an important export programme over a long period.

Table 1

ESTIMATE OF RECOVERABLE COAL RESERVES IN THE CAUCA VALLEY

Region	*Metric tons*
Region of Cali	106,000,000
River Lile-River Ponce sector	84,000,000
River Ponce-River Claro sector	56,000,000
River Claro-River Guachinte sector	45,000,000
River Guachinte-River Tima sector	40,000,000
River Tima-River Caleto sector	43,000,000
Inguito-River Dinde sector	32,500,000
TOTAL	406,500,000

C. *Quality of the coals*

All the coals are bituminous, but they may be divided into three groups, thus:

(*a*) Those of the southern section, with a low-volatile content, which have been found satisfactory for producing metallurgical coke. These coals are similar to those currently being exported from the southern region of the Appalachian Mountains in the United States, and can supply the low-volatile component generally required by Latin-American steel industry for making metallurgical coke.

(*b*) Those of the northern section, close to the city of Cali, usually with a high-volatile content, suitable for industrial use and for steam locomotives. Properly cleaned, these coals are comparable to Pittsburgh steam-coals, which they resemble as regards ash and moisture contents, heating value and storage properties.

(*c*) Those of the central section, immediately south of Cali, with a medium-volatile content. These coals are similar to the famous Welsh steam-coals so popular on the South American market.

A complete series of representative samples has been taken methodically from fifteen of the principal mines in the region, in order to study the properties of the coals. These samples were analysed and tested at the Central Experiment Station of the U.S. Bureau of Mines at Pittsburgh, Pa. Table 2 indicates the average analysis of the run-of-mine coal, for each of the regions mentioned.

These same samples were subjected to washability and coking tests at the laboratory indicated, and it was concluded that after preparation in a modern washing plant, these coals could be delivered in uniform condition, comparable to European and North American coals which have traditionally supplied South American markets and those on the Pacific coast, where the Valley coals have a decided advantage from the standpoint of transport.

GENERAL SITUATION OF THE CAUCA VALLEY COAL INDUSTRY

The Cauca Valley coal industry is currently producing coal at the rate of 25,000 tons a month, all of which is consumed in the department itself.

As a result of geographic conditions, coal requirements for the industry and railways of this area must necessarily be met by the mines in the region, as it would not be economically feasible to carry this fuel from other coal-bearing regions such as the plateaux of Cundinamarca and Boyacá, or from the department of Antioquía.

At the same time, the current demand from industries which are being installed or expanded introduces a need for improving working methods and the quality of the product so that it can compete with liquid fuels.

Table 2

ANALYSIS OF RUN-OF-MINE COAL

	A *Southern Section*	*B* *Northern Section*	*C* *Central Section*
Ash . . . . %	11.10	16.65	12.92
Sulfur . . . . %	1.10	0.95	2.86
Volatile matter . . . . %	20.38	42.38	38.54
Fixed carbon	69.53	40.97	48.54
Hydrogen . . . . %	4.70	5.32	5.20
Carbon . . . . %	80.70	67.12	70.90
Nitrogen . . . . %	1.65	1.24	1.32
Oxygen . . . . %	3.95	8.78	6.80
B.T.U.	13,585	12,235	13,002
Ash fusion point (softening temperature —°F)	2,568	2,594	2,268

TRANSPORT

The Pacific railway links the city of Cali with the port of Buenaventura. Branches of this railway extend to the south of Cali throughout the Valley coal-bearing region, and to the north to connect with the railway to Caldas and Antioquía. Together with this railway system, there is also a road network, so that there is a complete transport system.

The Pacific railway is a single track with a 1-yard gauge; it uses steam locomotives burning coal and a few of which have been converted to use fuel oil. Five diesel-electric locomotives have been ordered which are beginning to reach the country; these are expected to bring about a considerable increase in the railway's carrying capacity.

Most of the freight on the Pacific railway consists of imports for the interior entering through the port of Buenaventura.

The movement of coal in the opposite direction, from Cali to Buenaventura, is therefore desirable, as it will be largely a return freight, and will contribute considerably to the economic running of the railway.

The railway is in itself one of the main coal consumers, but its position as a future purchaser will largely depend on any improvements in the methods of mining and preparing the coals in the region, since in operating a line with gradients of 4½% between Cali and Buenaventura, the locomotives can only work with uniform high grade coal.

If a uniform coal with 10% ash and 13,500 B.T.U. per lb. were available, there is no doubt that the steam locomotives would use coal fuel exclusively with a corresponding reduction in their operating costs as compared with costs using fuel oil. According to figures prepared by the Pacific railway for 1949, the average cost per train-kilometre was 0.6206 pesos in trains using coal and 1.0762 pesos in trains using petroleum.

EXPORT PROSPECTS

The geographic position of the Valley coal deposits and their accessibility to the port of Buenaventura immediately open up export prospects.

The success of any export programme will depend mainly on the following factors:

(1) Quality of the coal;
(2) Facility for transporting it to Buenaventura;
(3) Loading on to ocean-going ships;
(4) Markets for Colombian coals.

1. *Quality of the coal*

The quality of the coal has been demonstrated by means of the research described above, which indicates that by preparation in a washing plant the Valley coals are comparable in quality with those European and North American coals currently supplying South America and the Pacific coast.

2. *Facility for transporting it to Buenaventura*

This is the limiting factor to the volume of exports. The Administration of the National Railways has given careful study to this point and has reached the conclusion that the Pacific railway can at present transport only 10,000 metric tons of coal a month to Buenaventura, and that by carrying out some improvements on the line and with the arrival of the diesel-electric locomotives on order, that capacity could be raised to 20,000 tons a month.

Although this amount is relatively small, it is important as it would mean practically doubling present output from the mines.

3. *Loading on to ocean-going ships*

The port of Buenaventura, terminus of the Pacific railway, has recently been modernized and provides adequate anchorage for fairly large steamers, throughout the year. When the recent modernization and expansion of the port facilities was carried out, a special site at the end of the mole was reserved for the installation of equipment for coal handling.

The Ministry of Public Works is expediting designs for the installation of the corresponding facilities. The capacity of the port equipment will be sufficient to handle loading in a normal time-lapse.

4. *Markets*

Of the coals found in the Cauca Valley, the low-volatile, coking coals of the southern section are those most likely at present to encounter the greatest demand abroad, although this does not mean that good prospects do not also exist for the export of other types of coal.

Since the volume of exports is restricted by the railway-carrying capacity, it is logical to try and export that coal which commands the best price on the international market and which also has a low-volatile content.

Taking into account the total transport distance from Buenaventura, as well as the availability of return freight for the ships, the study on the subject concluded that the potential export markets offering the best advantages to the Cauca Valley coals from the transport angle, were the following:

(*a*) The south of South America;
(*b*) Central America and the Caribbean area;
(*c*) Uruguay and Brazil as far as Rio de Janeiro;
(*d*) Japan.

It should be borne in mind that on these potential markets, coal exported from Buenaventura would have the advantage not only of shorter hauls, but also in every case, except the Caribbean area, there would be no need to pass through the Panama Canal, which currently costs Col. pesos 1.38 per metric ton of coal.

In view of these specific advantages and the small volume of exports envisaged (20,000 tons a month), there will obviously be no difficulty in placing this volume abroad, even if there were any contraction in the present world market for coal.

DEVELOPMENT PROGRAMME

There are two main objectives in the programme for developing the Cauca Valley coal industry:

(*a*) To meet adequately the fuel requirements of the Cauca Valley's growing industry; and

(*b*) To make Cauca Valley coals available for export through the port of Buenaventura.

The essential elements for achieving this development programme are the following:

(1) The installation of a central washing plant, having a capacity for 100 tons per hour and so designed as to be able to prepare adequately the various types of coal obtained from the different coal-bearing regions in the Cauca Valley;

(2) Provision of the Pacific railway with adequate wagons for hauling the coal;

(3) Installations at the port of Buenaventura for loading ships; and

(4) Expansion and modernization of the collieries in the region. In order to carry out this programme, the work has been distributed as follows:

    (i) The Instituto de Fomento Industrial will undertake to assemble the central washing and coal preparation plants;

    (ii) The National railways will undertake to purchase the necessary rolling stock to move the coal on the Pacific railway;

    (iii) The Ministry of Public Works, in association with the National railways, will undertake to assemble the port equipment for handling the coal; and

    (iv) The Co-operativa Hullera de Occidente or the Corporación Carbonera Colombiana, in Cali, with the advice of North American experts loaned under the Point IV Technical Assistance Programme, are studying the means for introducing modern working methods in the region's collieries.

It is therefore to be anticipated that this development programme for the Cauca Valley coal industry and the export of Colombian coals may become a reality within a short period.

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The Cerrejon Coal Mining Project

SIDNEY A. MEWHIRTER

GENERAL

The Cerrejón coalfields lie in the department of Magdalena near the north-eastern boundary with Guajira territory. They are 135 km. by highway east of the town of Valledupar and 106 km. by highway south of the port of Riohacha on the Atlantic coast. Cerrejón is 30 km. west of the boundary between Colombia and Venezuela. Cerrejón takes its name from a prominent limestone mountain used as a local landmark.

The coalfields lie on the right (eastern) bank of the Río Ranchería, which rises in the Sierra Nevada de Santa Marta and flows eastward through a broad open valley between mountain ridges to a point 25 km. east of Cerrejón. There it turns northward around the eastern extremity of the western ridge and empties into the ocean at Riohacha, forming in this part of its course the boundary between Magdalena and Guajira.

The narrow mountain ridge forming the western side of the Ranchería Valley has a height of 300 m. in front of Cerrejón, falling abruptly to the coastal plain, which is some 40 km. wide.

The eastern side of the valley is formed by the foothills of the Montes de Oca range, which here forms the frontier between Colombia and Venezuela.

The valley is 8–10 km. wide at Cerrejón but is much wider after it turns northward.

The coal measures outcrop over a width of 2,000m. and have been studied in detail over a length of 10 km. Coal outcrops are known 20 km. to the north-east and 70 km. to the south-west. It is not known if the formation is continuous.

The elevation at Cerrejón varies from 100 m. above sea level along the Río Ranchería to 150 m. at the base of the foothills.

The climate of the area is arid, 75% of the total rainfall occurring during the months of May and November. The rest of the year is hot and dry. Daytime temperatures are generally above 30° C., night temperatures are 25° C. or below. The north-east trade winds blow from November till May.

The area, except along the rivers, is covered by sparse, student hardwoods and cactus, averaging 500 trees per hectare, of which 40% are usable as mine props. The woods along the rivers grow to 20 m. or more in height, much of which is merchant timber.

The population is sparse, mostly of Indian or Negro origin, and is concentrated in occasional small villages along the roads. Their occupation is the raising of a few head of cattle and goats. Thirty kilometres to the west of Cerrejón some incipient agricultural efforts begin using irrigation, and increasing in activity towards Valledupar.

The one highway passing through the area is a two-way, all-weather gravelled road, well maintained. The bridges and culverts are generally constructed for a 10-ton load.

GEOLOGY

These coal measures were first reported upon, technically, by a French engineer over a hundred years ago, and have since been described in a reconnaissance report by Drs. Enrique Hubach and Benjamín Alvarado of the Servicio Geológico de Colombia during 1943–44 and in detail by Dr. Diego Henao, also of the Servicio Geológico, during 1951. During 1945–46 a group from the Servicio Geológico under the direction of Drs. Pava and Arango did a small amount of trenching and drove six slopes some 25–30 m. long on different veins at points of major outcroppings. The sample results of this work are contained in a report of the Servicio Geológico. Unfortunately these slopes did not leave the zone of meteorization so that oxygen and moisture results are high.

In 1945, the Instituto de Fomento Industrial, under the direction of its manager, Ing. Juan de Dios Ceballos, obtained a concession on a 10 km. length of the coal measures. Since that time an extensive programme of investigation and exploration has been carried out, under the direction of Dr. Edouard Reymond, resident geologist.

The coal veins of the district occur in shales and sandstones of Eocene age.

At Cerrejón the Eocene formation is exposed between two parallel thrust faults, with older formations outcropping beyond the faults due to uplift.

The Eocene formations extend many kilometres to the north and south of Cerrejón, forming a narrow strip at the base of the foothills of the Montes de Oca range.

Within the area studied by the Instituto de Fomento Industrial, the coal veins so far explored outcrop within a width of less than 2 km. Two profiles, drilled across the formation, and separated by some 4 km., show nine veins over 2 m. thick and at least six minor but exploitable veins.

In the southern half of the area studied, the veins strike N. 30°–50° E. but in the northern half bend to strike N. 80° E. The average dip is 35°–40°, but in some areas are as flat as 20°. Near the Ranchería fault, the lower beds are repeated by folding and faulting but have an almost vertical dip.

To the north and south of Cerrejón the Eocene formations, where exposed, strike parallel to the Cordillera or N. 30° E.

COAL

The deposition of the coal evidently was in a shallow lake or bay near a land mass, the coalbearing formation showing a repeated cycle, from poorly cemented sandstone, through finer material to claystone and coal, and back to sandstone. This gives a topography of a series of low parallel ridges 20 to 100 m. apart.

Often the coal occurs immediately above a white clay. Generally the roof is a clayey shale of little strength. No sandstone has been found forming a wall of the coal bed.

The major veins generally give little surface expression other than a faint topographic depression, while minor veins are strongly marked by outcrops or coal dust brought to the surface by ants or rabbits.

The major veins maintain a uniform thickness over considerable distances; several have been traced over 5 km. by drilling with little change in characteristics. The minor veins appear to be lenticular, but this may be due to insufficient exploration.

Veins are generally altered to coal dust to a vertical depth of 15 m., sometimes giving no indication of their presence at 3 m. depth but showing 3 m. more of oxidized coal at 10 m. depth.

Where fresh coal is exposed in development, it is clean, hard and bright, free usually from partings and with a weak cleat. Less than 25% of the exposures in trenches, slopes or drillholes show partings, although some of the very thick beds show very dirty coal in the bottom metre or so.

The major veins average 3 m. in thickness, although one bed has been drilled in several spots showing 1 to 19 m. thickness, probably due to the joining of two closely parallel beds.

Samples taken from five major veins indicate the following dry analysis (made by the U.S. Bureau of Mines).

Ash	1.7– 5.2%
Sulfur	0.5– 1.0%
Volatile	38.3–43.2%
Fixed carbon	52.6–59.2
Hydrogen	5.3– 5.4%
Carbon	77.3–78.7%
Nitrogen	1.8%
Oxygen	11.2–11.9%
B.T.U.	13,500–13,890
Fusibility ash, softening temperature	°F. 2,230–2,570

The coal is a lignite metamorphosed dynamically to a high-volatile bituminous coal, but retaining some of the physical characteristics of lignite. It is noncoking.

EXPLORATION

In 1945–46 the Servicio Geológico de Colombia studied the area, sinking a number of test-pits and slopes. Six of these slopes on five different veins were re-opened in 1950 and the above-mentioned samples taken from the faces. All have a slope depth of about 30 m. The veins vary in thickness from 2.20 m. to 4.00 m.

Starting from these slopes, attempts were made to follow the outcrops by trenching. This was abandoned when it was discovered that the maximum economic depth of a trench (3–4 m.) would expose only a few centimetres of coal dust or none at all, while a test pit 10 m. deep at the same point would show several metres of oxidized coal. Veins were then followed by vertical shafts, but this was slow and expensive, as it often required several shafts to intersect the vein, especially in areas where the outcrops are covered by a thin quaternary formation. A rotary drill was obtained from another governmental department by the intervention of President Laureano Gómez, and all surface exploration is now done by drilling.

One hundred and five holes with a total depth of 5,000 m. have been drilled to date. At the beginning, very short holes were drilled, but as the crew obtained practice longer holes were drilled until now the standard hole is 100 m. deep. A 100 m. hold can be drilled in two or three days. Deeper holes have been drilled, but for geologic mapping are too expensive.

The method used is to drill a profile across the coal formation, spacing the holes according to the dip of the strata to give a 50% overlap. This gives detailed information as to coal thickness, quality and dip and records all veins regardless of thickness.

The programme has been to trace veins and determine the detailed geology, leaving exploration in depth for the future.

Coring equipment has been obtained and occasional holes will be cored for complete analysis.

The slopes on vein 6, previously unsampled, were sunk 35 m. One showed clean coal while the other revealed several sandy partings. This is the uppermost known vein and does not seem to be as strong as the lower veins. It cannot be reached by drilling due to the topography.

Two slopes are now being sunk on separate veins of the five series (a group of five or more closely spaced veins). At present they are 80 m. deep. It is planned to sink them to 200 m. depth in order to conduct experiments on extraction and roof conditions and to furnish coal for the local market.

Neither gas nor water in quantity has been found in the development work so far.

In 1946, the Tropical Oil Co. drilled ten exploration holes slightly to the north of Cerrejón, over a formation length of 2,500 m. The deepest hole reached 2,000 ft. and cut the lowest coal bed at 1,500 ft. depth. Only two of the ten holes failed to cut coal due to insufficient depth. The logs of these holes indicate that some of the veins are lenticular showing considerable variation in vein thickness between adjacent holes.

Proved reserves as of this date are 11 million tons while probable reserves will exceed 35 million tons.

To exploit this potential reserve of national wealth will require export of the coal as the local market is very limited. This in turn will require the construction of a standard gauge railway to the coast and a coal loading port.

Summary of Discussion

Coal Washery Performance as Related to Blast Furnace Costs, *presented by the author*

Mr. Price first referred to a statement made at a preceding meeting to the effect that Latin-American coals should be washed in such a way as to possess the lowest practicable ash content. He did not agree with that rule because it could be proved that an optimum degree of washing existed at which the cumulative costs of coal washing, production of coke from the washed coal and blast-furnace operation with the resulting coke—if considered as a function of the ash content in the washed coal—presented a minimum. It was not only useless but uneconomic to try to obtain an ash content lower than that minimum.

He then proceeded to prove his point, which was developed *in extenso* in his paper in the case of one particular Mexican coal, actual operation figures having been distorted in order to avoid the disclosure of costs. He explained the various stages of calculation, which—other factors being constant—permitted the determination of the influence of the ash content of the coal on that portion of blast furnace costs directly affected by the ash content, that portion consisting of the cost of coke, the cost of limestone, and the operating costs per ton of pig iron. He emphasized that the quantitative data given should only be considered as examples, but that the calculation method had a general application and that the determination of the optimum ash content could always be made on the basis of the specific conditions in a given plant.

Mr. Cheradame stressed the interest of the method developed by Mr. Price and the importance, in studying coal washing problems and costs, of taking into account the interrelation between the different phases of operation and the cost advantages to be gained by the coal consumer from increases in the cleanness of coke. In that connexion he felt that the actual results of similar calculations would be different in the case of France, where middlings were used as steam-raising materials in electric plants, thus modifying the cost structures.

Mr. Leuschner noted that the next paper to be presented discussed the same problem as that in Mr. Price's paper, but in addition took into account the fact that the various petrographic components of coal had varying influences on coking qualities.

Mr. Price pointed out, further, that expansion during coking increased considerably in the case of Mexican coal if the ash content were very low, indicating therefore that the coal should not be too clean.

The Influence of the Ash Content on the Hardness of Cokes Made from High-Volatile Coals, *presented by Mr. Baum in the absence of the author*

Mr. Baum stressed the salient points in the paper, and pointed out that selective preparation of coal based on the principles outlined therein could be beneficial both from the standpoint of the economic functioning of the blast furnace and from that of the reduction in the amount of highly-coking coal which had to be added to Chilean coals before carbonization.

The method used by the author in his economic calculations was similar to that described by Mr. Price and also to that developed in Germany by Messrs. Bansen and Krebs. The conclusions reached, however, were different from those in Mr. Price's paper. In that connexion the factor recommended by Mr. Cheradame was that the utilization of middlings should be borne in mind. He himself suggested that middlings might also, after grinding and a "Convertol" separation, be reincorporated in the mixture fed into the coke oven.

Mr. ALBALA expressed disagreement with some statements in Mr. Vogel's paper concerning the analysis of No. 5 Schwager coal, used for coking by the Compañía de Acero del Pacífico. First, he noted that there were almost no middlings to recover in that coal, the fraction between 1.35 and 1.75 being negligible. Secondly, the experience of the Compañía de Acero del Pacífico was that the coal's sulfur content was not reduced by washing.

Mr. LEUSCHNER stressed the relative nature of the term "middlings", showing that the definition used by Mr. Vogel was stricter than that considered by Mr. Albala.

Mr. CORTÉS OBREGÓN agreed with Mr. Price that Mexican coals presented excessive expansion properties when washed to a very low ash content. He also indicated that middlings were not used for steam raising in Mexico inasmuch as it was considered better to recirculate them, increased coke production being deemed preferable to the production of electric power.

Mr. BAUM indicated that the problem of the influence of ash content was further complicated by the question of the composition of the ashes. For instance, coal with 18% ash could lead to lower blast-furnace costs than 12% ash coal if the difference in ash content were represented by lime. Also ashes might contain considerable quantities—up to 40%—of iron.

Mr. MERCIER asked whether he was correct in assuming that the limey ashes mentioned by Mr. Baum were to be found only in lignite coke, in which case the situation described by him was to be considered an exceptional one, to which Mr. Baum gave an affirmative answer.

Mr. LEUSCHNER mentioned that in certain cases in the Monclova steel plant in Mexico the ash content was so low that slag-forming material had to be added in the blast furnace. He felt that 17 to 20% was an adequate washing limit for Mexican coals.

Mr. GONZALEZ BALLESTEROS stated that in Mexico coal was washed up to a carbon content of 80% if it was to be carbonized in by-product ovens and to 76% in bee-hive ovens. Costs with bee-hive ovens increased by 10% when an 80% carbon content was sought.

Mr. CORTÉS OBREGÓN remarked that the particular bee-hive coke referred to was not as clean as others because the washing plant was only provisional.

Development of the Cauca Valley Coals, ***presented by Mr. Vargas Martínez, co-author***

The Cerrejón Coal Mining Project, ***presented by Mr. Vargas Martínez in the absence of the author***

Mr. CORTÉS OBREGÓN opened the general discussion on the two papers presented by Mr. Vargas Martínez by stressing the importance of many facts brought to light during the meeting. For instance, Mexico could as a result thereof consider utilizing its rich iron ore deposits in the Pacific area, taking advantage of Colombian coal for smelting.

Mr. LEUSCHNER drew the attention of those present to another possibility which was linked with the Cerrejón deposit: Venezuela could use such coals, blended with its own asphalt, to make coke. On that basis, Venezuela could establish its steel industry.

Mr. VARGAS MARTÍNEZ agreed with Mr. Leuschner's remark, adding that he felt, furthermore, that the Cerrejón coals could be blended, for coking purposes, with Santa Catarina (Brazil) coal.

IRON ORE REDUCTION

The charcoal blast furnace

21 October 1952—Morning

Chairman:

Alfredo Gonzalez BALLESTEROS, Assistant Superintendent, Departamento de Altos Hornos, Compañía Fundidora de Fierro y Acero de Monterrey, S.A., Monterrey, Nueva León, Mexico

Contributed Papers:

Pig Iron Production in Blast Furnaces Using Charcoal
Louis ENSCH, Managing Director, Companhia Siderúrgica Belgo Mineira, Belo Horizonte, Brazil

Operation of the Charcoal Blast Furnace at Corral Using Mixtures of Metallurgical Coke and Charcoal
Danilo VUCETICH, Administrator, Altos Hornos de Corral, Compañía de Acero del Pacífico, S.A., Corral, Chile

The Sintering Plant at Monlevade
Francisco J. PINTO DE SOUZA, Chefe do Departamento Materias Primas, Companhia Siderúrgica Belgo Mineira, Belo Horizonte, Brazil

Summary of Discussion:

Participants: Messrs. LEUSCHNER, LANARI, MARTIJENA, ALLARD, KALLING, SEM, CANGUILHEM, MARIN GONZALEZ, VUCETICH, PRADO UCHOA.

Pig Iron Production in Blast Furnaces Using Charcoal

LOUIS ENSCH [1]

INTRODUCTION

The importance of the proper preparation of the blast furnace burden is generally recognized. Adequate permeability and distribution ensure the maximum productivity and economy of the apparatus for each type of ore.

The results obtained at the Companhia Siderúrgica Belgo Mineira's Monlevade plant, from the use of a high percentage of sinters (60–70%) in the burden together with a calibrated and screened ore offer such excellent prospects that we feel justified in basing our study of the possibilities for an iron and steel industry using charcoal on the information recorded for the four blast furnaces installed at that plant.

The value of the data shown below is emphasized by the fact that the four identical furnaces are working under exactly the same conditions, which is apparently unique in Latin America.

In order to stress the importance of the improvement, it should be pointed out that net production of the furnaces rose from 0.8 ton per 24 hours (in cubic metres of net volume for a burden of 100% ore), to 1.17 tons per 24 hours in cubic metres of net volume with a burden made up of 70% sinter and 30% ore. This also permitted an appreciable saving in charcoal consumption whilst at the same time improving the rate of production.

DESCRIPTION OF THE BLAST FURNACES

The Monlevade plant of the Companhia Siderúrgica Belgo Mineira has installed four identical blast furnaces, with the characteristics and measurements shown in Table 1.

The special features of the equipment are a wide hearth and the sharp slope of the walls.

The furnaces are charged by buckets and vertical cranes. The walls are not cooled.

The theoretical burden of the blast furnaces, that is 100 tons every 24 hours, has been substantially exceeded by using sinters therein.

The highest monthly average of the furnaces was 140.1 tons per 24 hours, and the highest average for the decade 156 tons in 24 hours, whilst the highest daily production recorded was 185 tons in 24 hours.

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[1] Due to the untimely decease of the author, this paper is published in its original version, unabridged and unrevised.

The annual production of the furnace exceeds 45,000 tons.

The average ratio in tonnage of raw material to the final product (yield) is 2.9 m^3/t. of pig iron, equivalent to 548 kg. C/t of pig iron. The best result obtained, from the economic standpoint, was 510 kg.C/t.

Table 1

CHARACTERISTICS OF THE BLAST FURNACES AND THEIR OPERATING DATA

Total height	H =14.900 m.
Net height	Hu=13.100 m.
Height of the hearth	Hc = 2.600 m.
" " " walls	Hr = 3.000 m.
" " " lower bosh line	Hv= 2.200 m.
" " " shaft	Hc= 7.100 m.
" " " tuyeres	Ha= 1.900 m.
" " " slag tap holes	He = 1.300 m.
Diameter of the hearth	Dc = 3.100 m.
" " " lower bosh line	Dv= 3.836 m.
Top diameter	Dg= 2.784 m.
Angle of the walls	83° 4^1
" " " shaft	85° 45^1
Total volume	V = 135.070 m^3
Net volume	Vu = 120.700 m^3
Number of tuyeres	N = 6
Diameter of the tuyeres	Dv= 120 m.
Theoretical burden	100 tons/24 hrs.
Air pressure	0.3 atm.
" temperature	750°C.
Amount of air injected	2,600 kg./t. pig iron
Coal consumption	2.9 m^3/t. pig iron
Charcoal consumption	548 kg.C/t. pig iron
Volume of slag	230–250 kg./t. pig-iron
Proportion of CO/CO_2 in the gas around the top	1.8–1.9
Labour	15 workmen/oven shift
Composition of the pig iron produced:	
C	4.0%
Si	0.4%
Mn	0.8%
P	0.25%
S	0.01%

The volume of air injected per ton of pig iron is relatively small in proportion to that required for furnaces using coke. This is due to the small amount of slag and the presence of oxygen in the charcoal.

The ore used for the production of sinters is a very pure, friable hematite (Itabirito), which is charged directly into the furnaces.

The following is a typical analysis of this ore:

Fe =66.2%, entirely in the form of Fe_2O_3
SiO_2 =1.7%
Al_2O_3 =1.5%
P =0.08%
S =0.01%
Mn =0.07%
Te =traces

The crude ore from the mine is calibrated in a jaw crusher with an opening of 40 mm. and the fines are separated by a 20 mm. screen. These fines, sinterized by the Greenawalt process, are combined with granulated slag from the blast furnaces to produce a sinter of which the following is a typical analysis:

Fe = 61.3%
SiO_2 = 6.5%
Al_2O_3 = 3.2%
CaO = 2.6%
P = 0.10%
S = 0.02%
Mn = 0.25%
Degree of oxidation = 94 – 96%

DESCRIPTION OF THE FOREST RESOURCES

All the forest resources worked for the Monlevade plant are located in the Basin of the Rio Doce, a part of the coastal forest area (Fitogeographic system of A. J. Sampaio), which extends into the interior, mainly in the form of forests bordering the Rio Doce and its tributaries, the Casca and Piracicaba rivers.

These resources, originally estimated to cover 30,000 sq. km., are of two types: original formations, or virgin forests, and secondary formations (either second, third or even fourth regenerations) or small forests and underbrush.

The original formations follow the usual pattern of tropical forests, that is to say, a great wealth of broad-leaved species and the absence of coniferous species, in a closed stand or alternatively spread out in layers. Finally, there is an abundance of cipós and parasitic types (orchids).

The secondary formations, the small forests and underbrush, mature in the course of 20 to 30 years in areas which are not burned out, in which case their composition is noticeably different from that of the virgin forests. They show a predominance of light softwoods, the stand is irregular and the undergrowth is not so thick. Moreover, a remarkable abundance of cipós types is found, but scarcely any parasitic species.

Little systematic knowledge is available regarding these virgin forests. The few surveys carried out were only concerned with the more valuable species from the economic standpoint.

Among the latter, the following should be noted:

Angico, angico vermelho	Piptadenia peregrina, Bent.
Bicuiba	Myristica Gardnerii, A.D.C.
Cedro	Cedrela app.
Ipe	
Jatoba	Hymenaea app.
Jequitibá	Cariniana excelsa
Peroba, ipe-peroba or peroba de Camposr	Paratecoma peroba (Record) Kuhl
Sapucaia	Lecythis app.
Sucupira	Bowdichis virgilicides, H.B.K.
Vinhático	Platymenia foliosa, Benth.

YIELD OF THE FORESTS

Data regarding the growth and volumetric yield of the forests should be divided into two parts, namely:

(*a*) Natural, original and secondary formations; and

(*b*) Native and regenerated planted formations.

(*a*) The volumetric rate of growth of the natural formations is not known owing to the fact that the age of the virgin forests has never been accurately ascertained.

The yield of these natural original formations varies considerably within the region. However, the average yield may be assumed to be 1,000 cu. m. of lumber per geometric "alqueire", i.e. 210 cu. m. of lumber per hectare (though this does not include saw logs and such wood as cannot be used for the manufacture of charcoal).

For secondary formations, and especially the small forests, a volumetric rate of growth of from 4 to 6 cu. m. per hectare per annum has been estimated, that is, a yield of 120 cu. m. for stands aged from 20 to 30 years.

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(*b*) In the case of the planted formations (for productive purposes) exact data are available in connexion with the yield of certain species of eucalyptus.

The Forestry Service of the Companhia Paulista de Estradas de Ferro (São Paulo Railway Company) has compiled a vast series of data on this topic.

However, it should suffice to quote the following information with respect to the working of stands of a single species of eucalyptus planted over a period of 19 years, spaced out in rows, with 2 m. between each tree and 2 m. between the rows, and worked on the felling régime.

Table 2

Formation		*Age of each group*	*Rate of growth cu. m. per hectare per annum*	*Yield cu. m. per hectare*
Type	*Age (years)*			
Native trees..........	0 to 7	7	48	335
Regenerated trees (shoots thinned)....	7 to 12	5	37	185
Regenerated trees (shoots not thinned in the 12th year)....	12 to 19	7	78	550

The above data refer to forests cut for lumber.

For the production of charcoal for the iron and steel industry, no estimate has yet been made as to the optimum economic age for working the different species of eucalyptus. The Forestry Service of the Companhia Siderúrgica Belgo Mineira is about to begin studying this subject.

The production of charcoal

Almost all of the coal consumed by the Monlevade plant is still produced by the primitive method of "balones" (balloons), also known as "caieiras" (lime kilns) or "medas" (ricks). Iron and other semimetallic retorts have recently been successfully introduced, as have others made of brick as well as the so-called "Japanese ovens".

The forest resources are subjected to total fellings. With the exception of the sawlogs and the species of eucalyptus which cannot be used to make charcoal, all the lumber cut down is consumed in the production of charcoal.

The product is evidently of a heterogeneous nature owing to the many different species used.

Carbonization conditions (temperature, time required, amount of air let in) also vary in accordance with local practice, the methods adopted by the charcoal workers or the system and earth used in the construction of the pile. The average yield is 1 cu. m. of charcoal per 2 cu. m. of lumber.

Little is known of the physical, chemical and mechanical characteristics of the charcoal thus produced other than the results recently published by Engineer Jaime Araujo of the National Department of Mineral Production. With the permission of the above-mentioned research worker, some of the results obtained are shown below:

Table 3

	Eucalyptus[a]	*White Angico*[b]	*Red Angico*[b]	*Jacaré*[b]	*Miscellaneous*[b]	*Miscellaneous*[c]
Weight—kg. per m^3:						
With natural humidity..........................	224	232	229	216	—	238
Dry..	210	222	220	204	200	220
Friability:						
Loss in the drum:						
ASTM..	66.8	68.8	68.8	61.3	59.1	65.7
LPM-Tb..	57.7	51.0	49.0	46.8	—	48.8
Resistance to radial compression—kg. per cm^2.......	30	45	55	42	32	43
Percentual composition of dry coal:						
Volatile matter..................................	18.9	27.6	14.8	18.7	16.1	14.4
Fixed carbon..................................	79.6	70.9	84.3	79.5	80.3	82.7
Ash..	1.5	1.5	0.9	1.8	3.6	2.9
P..	0.065	0.016	0.013	0.030	0.036	0.028

[a] The results represent the average of the information collected from ten different species of eucalyptus: carbonization was effected in ricks; the data were obtained and supplied by the Forestry Service of the Companhia Paulista de Estradas de Ferro.

[b] Charcoal produced in ovens.

[c] Charcoal of the type used at the Monlevade plant, produced with lumber obtained from the felling of native species.

Discussion of the possibilities of an iron and steel industry based on charcoal

The following material is entirely subjective; it represents the author's personal views and opinions, and is published in the absence of any other proved experimental data.

The first question to be considered is an estimate of the maximum capacity of blast furnaces using charcoal.

Working with a properly prepared burden, that is a rich, calibrated ore that has been partially agglomerated by sintering or any similar process, together with sifted charcoal of less than 10% humidity, air heated to 800°C., a well designed furnace should smelt 1.3 tons every 24 hours, per cubic metre of net volume. The average for the decade reached this level at the Monlevade plant, and, in fact, daily averages have exceeded it.

The consumption of coal per ton of pig iron, produced as shown above, would amount to approximately 500 kg., equivalent in other words to a tonnage ratio of raw material to a final product of 2.7 cu. m.

of sifted coal or 3.0 cu. m. of ordinary coal. In our opinion, it would be perfectly feasible to have a blast furnace with a net volume of 230 cu. m., of which the total height would be 18 m., and the diameter of the hearth 3.8 to 4m. It is of interest to note that the Companhia de Aços Especiais Itabira S/A (The Itabira Special Steel Company Limited) is operating a blast furnace with a total height of 19.54 m., and a hearth diameter of 3.66m. Thus, the blast furnace has a daily capacity of 300 tons of pig iron or approximately 100,000 tons annually.

COMPARISON OF BLAST FURNACES USING CHARCOAL WITH THOSE USING COKE

It may well be worth drawing a comparison between the respective merits of an iron and steel industry using charcoal and one employing coke.

The good mechanical resistance of coke enables blast furnaces using this fuel to be built to a very large scale—1,500 tons or more of pig iron per 24 hours, with the following advantages:

(1) Smaller investment per ton of pig iron;

(2) Lower operating and repair costs per ton;

(3) Less loss of heat per unit of pig iron.

The bell of a furnace using coke is usually larger than that of one using charcoal. However, in South America the supply of coking coal (or coke) has to be partly if not entirely imported from more highly industrialized countries, such as the United States, Germany or Canada. This gives rise to a serious economic problem for the country involved which, on the other hand, has no alternative but to export its raw materials.

The blast furnaces using charcoal probably should not exceed the 300 tons in 24 hours mark, as was shown above. On the other hand, they offer several advantages in relation to those using coke, namely:

(1) The pig iron is of a better quality, and has a low sulfur content;

(2) Lower consumption of fuel, despite greater losses of heat stemming from irradiation, convection, etc., per ton of pig iron. This is due to the possibility of producing pig iron at a lower temperature and obtaining more adequate slag, since the problem of eliminating the sulfur (high temperature in the hearth and high basicity of the slag) need not be considered. At the present stage of development in the methods of steel manufacture, pig iron composed of 0.3% Si, 0.6% Mn and 0.2% P can be successfully produced in basic Thomas converters as well as in Siemens Martin electric furnaces by adding oxygen to the air injected or alternatively by using pure oxygen;

(3) The production of pig iron using domestic raw materials exclusively. The plant, by using the charcoal prepared from replanted eucalyptus, could raise the country's forest resources, promote rapid industrialization and raise the standard of living throughout a vast cultivated area. A chemical industry, based on the by-products obtained in distilling the lumber, would develop simultaneously with the growth of the iron and steel industry.

COMPARISON OF BLAST FURNACES USING CHARCOAL WITH ELECTRIC FURNACES

According to data supplied by the Elektrokemisk Plant of Norway, recent developments in the production of Söderberg electrodes and in the construction of the furnaces themselves make it possible to raise the capacity of electric furnaces to 400 tons per 24 hours for six electrode furnaces, and to 600 tons per 24 hours for nine electrode furnaces. Formerly, the limit was considered to be 200 tons owing to the excessive loss of electric power. Three alternatives should be considered in comparing an electric furnace of the above size with a blast furnace using eucalyptus charcoal:

A. *Electric furnace using charcoal*

(1) The low mechanical resistance of the charcoal does not constitute a problem nor does it limit the size of the furnace as it would in the case of a blast furnace;

(2) Charcoal consumption amounts to some 350 kg. C per ton of pig iron;

(3) The electric furnace offers greater flexibility in the regulation of the composition of the pig iron;

(4) An electric furnace calls for a larger investment whilst labour consumption in both cases is fairly equal;

(5) The size of an integrated plant, based on electric furnaces using charcoal made from eucalyptus, might be proportionately larger than that of a plant with blast furnaces, say a proportion of 500:350.

However, an electric furnace of this type could be built only on a site in which forest reserves are found alongside a source of hydro-electric power. The most serious disadvantage is the high consumption of electric power—2,500 kWh per ton of pig iron. To compete with a blast furnace, the price of this amount of power should equal the price of 150 kg. of C or 0.8 cu. m. of charcoal. In other words, the cost of 1 kWh should be about 0.06 cruzeiros, which probably could not be obtained in Brazil, at all events. In this general discussion the problem created by the gas emanating from the furnace has thus far been overlooked. This gas in an electric furnace has a higher heating power but its volume is smaller than that issuing from a blast furnace, even taking into account the gas consumed by the Cowpers.

B. *Electric furnace using coke*

The advantages of an electric furnace as enumerated under A. also apply in this instance, with the exception of point (5) which would not exist since there is no size limit for an integrated factory. It should be noted that the elimination of sulfur is considerably simpler in this case than in that of a blast furnace using coke, owing to the smaller amount of reducing agent required and the higher temperatures. The quality of the pig iron is slightly inferior to that produced with charcoal but, on the other hand, a poor quality coke, having a high ash and sulfur content, can be used. In addition to the problem of electric power, and taking into account the disadvantages of the process, there is also a question of price and location of the plant, and finally the necessity of importing coke.

C. *Electric furnace using a mixture of coke and charcoal*

The greatest advantage of this system is its great flexibility. The percentage of coke or charcoal in the burden can be regulated in accordance with the policy or availabilities of the plant at any time. If it is mixed with charcoal, the coke can be of a very poor quality. Used in limited amounts, it could be produced by domestic deposits, or at least a fair proportion of it could be supplied from within the country.

CONCLUSIONS

With regard to the possibilities of developing an iron and steel industry in South America, it is our opinion that:

(1) The vast potential mineral resources and the energy supplied by the sun, aided by a favourable climate and intense vegetation, may be used more advantageously by employing charcoal in an iron and steel industry. A blast furnace using this fuel appears to be the most economic means of reducing the ores and it can be readily installed on a large scale;

(2) Electric furnaces offer a more rapid solution. They can begin by operating with coke and progressively, in a measure with their development, switch over to the use of charcoal. The question of the supply and price of electric power is, however, a very serious disadvantage;

(3) The most simple solution and the most rapid for the problem of developing an iron and steel industry is a blast furnace using coke.

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Operation of the Charcoal Blast Furnace at Corral Using Mixtures of Metallurgical Coke and Charcoal

DANILO VUCETICH

In 1908 the French firm Hauts Fourneaux Forges et Aciereries du Chili started building the first blast furnace on the site now occupied by the Corral plant. This blast furnace was designed for the use of El Tofo iron ore; the iron ore was reduced by charging logs of wood directly into the blast furnace, which had been designed by Prudhomme. It commenced operations in 1914 and operated occasionally, mostly in an experimental way, until 1919, without achieving any large commercial production. The site had been chosen because of the thick natural forests which covered the hills in the region. Corral is located in the bay of the same name, at the mouth of the Valdivia River, which is one of the few navigable rivers in Chile; it has a safe harbour and ships of up to 10,000 tons can tie up at the only mole. Lack of space and the geographical difficulties for establishing good land communications, either rail or road, the distance from the iron mine, as well as from the main consuming centres, constitute the factors which limit the development of the steel plant in Corral. The limitations caused by fuel supplies will be described elsewhere in this paper.

In 1933, a private company was set up, with financial support from the government, for the purpose of working the Corral installations; it was called the Compañía Electro Siderúrgica e Industrial de Valdivia (ESVAL), and it began operating the same blast furnace, using charcoal instead of wood. Many technical difficulties arose, but as there were also financial and marketing difficulties, which often caused stoppages, no definite conclusions can be drawn from these operations. Over a period of 12 years, this blast furnace produced 83,000 tons of pig iron, and in general it can be said that it worked satisfactorily, in view of the fact that its design was not of the conventional type for the use of charcoal (see Figure 1).

In November 1944 the old blast furnace was abandoned, and a new one came into operation which was designed to operate on charcoal. The main features of this furnace, which has been called blast furnace No. 2, are those shown in Figure 2, and it was designed for a production of 60 tons a day. Henceforth in this paper reference will be exclusively to the experience gathered in operating this unit.

This blast furnace No. 2 was built with a suitable height for making use of the existing elevator equipment and the loading platform. It is estimated that it should have been some two metres less.

OPERATING CONDITIONS OF BLAST FURNACE No. 2 UNTIL DECEMBER 1950

Until December 1950 the Corral blast furnace was operated by ESVAL, and used charcoal to which a small percentage of gas coke from the gas works in Concepción was added. Chart 1 shows the average annual consumption of the main raw materials per ton produced.

During this period output totalled 87,000 tons, giving an annual average of 14,000 tons. The highest

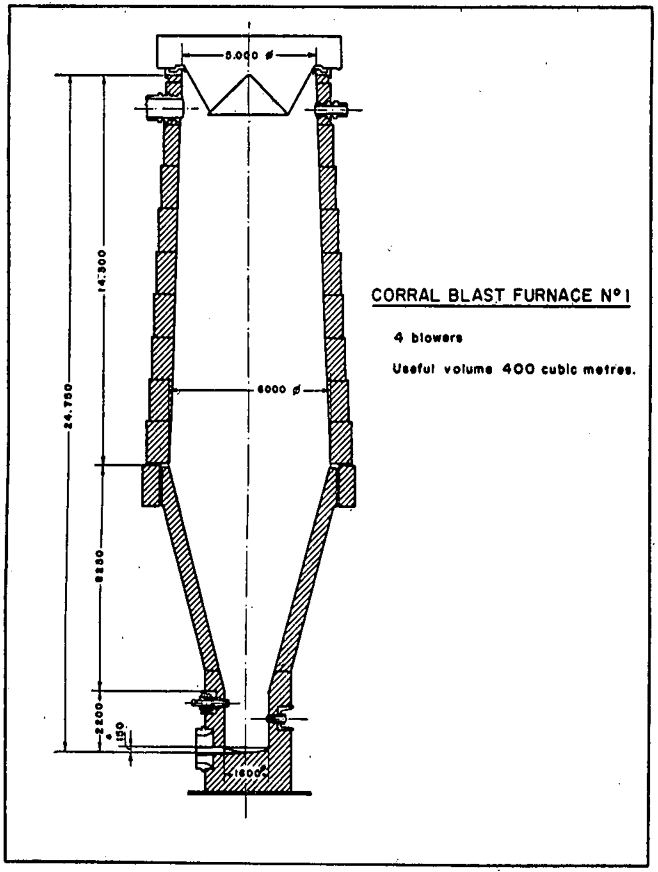


Figure 1
Corral blast furnace No. 1.

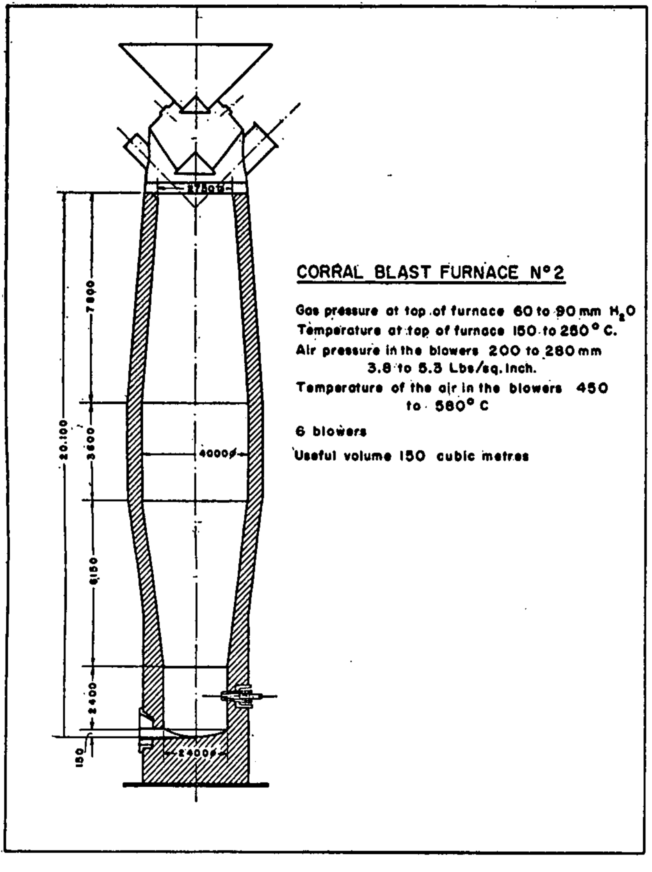


Figure 2
Corral blast furnace No. 2.

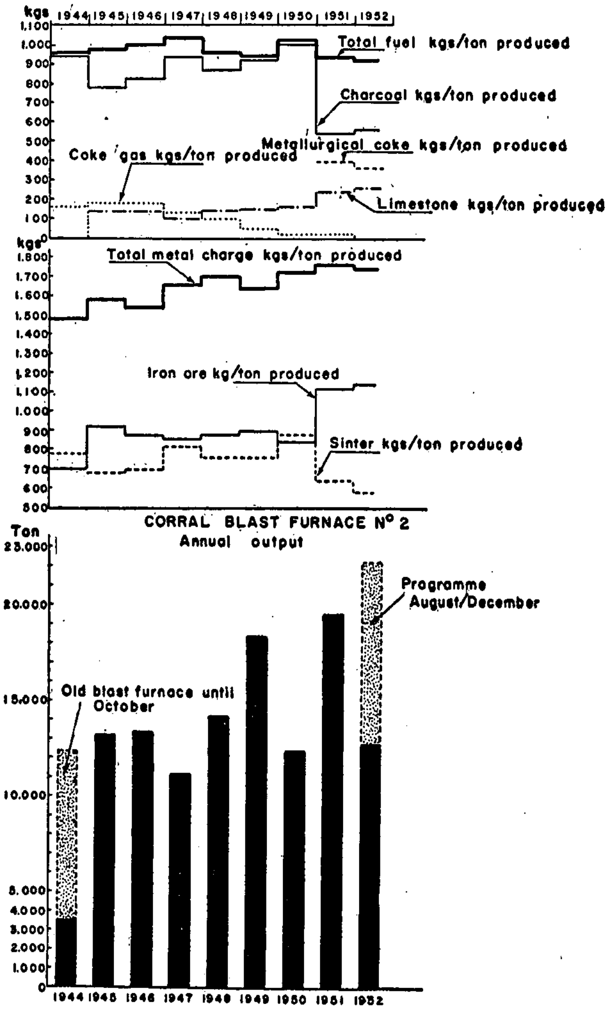


Chart 1
Corral blast furnace No. 2.
Annual output.

annual production was 18,600 tons, in 1949, while the highest monthly total was 2,084 tons, in January 1949. The highest total for one day was 89 tons, on 16 January 1949. All these figures include from 8 to 10% of scrap.

The analysis of the different types of pig produced was:

Type	*C*	*Si*	*Mn*	*P*	*S*
0.55	3.7/4.0%	0.75%	0.3/0.5%	0.10/0.15%	Traces to 0.01%
105	"	0.76%/1.25%	"	"	"
155	"	1.26%/1.75%	"	"	"
205	"	1.76%/2.25%	"	"	"
255	"	2.26%/2.75%	"	"	"

Both in order to permit a better comparison of the operation of the blast furnace between this and the subsequent period, when a higher percentage of metallurgical coke has been used, and to facilitate the drawing of conclusions, a summary is given below of the main conditions under which the blast furnace operated during this period.

Air pressure and temperature

This was operated by the compressor originally installed for blast furnace No. 1. It is a reciprocal machine, which has an approximate capacity of 180 cu. m. per minute with a pressure of 200 to 250 mm. of mercury.

For pre-heating the air there were seven Cowpers or stoves, built for blast furnace No. 1, with an area of 3,800 sq. m. Only three of these were used, changing them every three or four hours, with an average consumption of 40% of the total blast furnace gas produced; it should be noted that two of them had their original brick work. The temperature of the air was 500 to 600° C.

The compressor did not operate at its normal capacity, because of leakages due to wastage of its valves and through maladjustment of the steam distribution system.

Iron ore

Iron ore from El Tofo mine was used, which has an average grade of 55 to 57% of Fe. A representative analysis is:

SiO_2	Al_2O_3	Fe_2O_3	*Fe*	*P*
8% to 17.1%	2.1%	79.2%	55.4%	0.042%

Sinter

There is a Greenawald plant with an area of 9 sq. m., with a gas exhauster of 150 cu. m. capacity and a vacuum of 500 to 600 mm. of water. The oxidation obtained was around 92%; 7% of ground charcoal was used in the mixture; daily production was about 55 tons.

Charcoal

The Chilean Government had granted the company a concession of 21,400 hectares for the utilization of the natural forest for making charcoal. Some 30% of the charcoal consumed was obtained from this source, the rest being purchased from producers in the neighbourhood. All the charcoal is produced on the mountain itself; in order to carbonize it, wooden logs are conveniently stacked and covered with mud to control the combustion of the wood. In this way the combustion of part of the wood and the by-products provides the necessary heat for the carbonization of the rest. This coal is packed in sacks of ⅛ cu. m., with an average weight of 27.5 kg., and transported to the plant, where it is stored in covered sheds. Most of the coal derived from the concessions is brought from the mountain on an aerial tramway which tips it directly into the storage shed with a capacity of 25,000 cu. m.

Carbonization on the mountain takes place exclusively during the less rainy months of the year, which vary from four to six. During the rest of the year work is confined to cutting the wood and preparation for carbonization.

Many of the stoppages of the blast furnace during winter months were due to lack of charcoal. This was mainly due to:

(*a*) Lack of storage space in the sheds;
(*b*) Lack of equipment, particularly sacks;
(*c*) Financial difficulties, as a sufficient quantity had to be accumulated for operation during the rainy months;
(*d*) Small productive capacity;
(*e*) The climatic conditions are a decisive factor in producing charcoal with the methods employed at present. A dry year had an immediate effect on production, as was the case in 1949, within the period analysed. Moreover, this was the only year of this period when there was no stoppage during winter months.

ANALYSIS OF THE CHARCOAL

Calories	*Weight m^3*	*Volatile matter*	*Fixed carbon*	*Ash*	*Moisture*
6,800 to 7,000	220 kg.	10/11%	73/76%	2.7%	5 to 12%

Gas coke

In order to offset the lack of charcoal to some extent, coke from the city gas plant of Concepción was added during this period. The percentages used are indicated in the corresponding Chart 1.

ANALYSIS

Volatile matter	*Fixed carbon*	*Total sulfur*	*Ash*	*Moisture*
8%	70.60%	1.89%	17%	4.40%

Limestone

Limestone came from a deposit in the province of Coquimbo, in the north of Chile. On an average 150 kg. were used per ton of pig produced. This figure, which may seem relatively low, is due to the addition of open hearth slag.

ANALYSIS OF THE LIMESTONE

SiO_2	Al_2O_3 *plus* Fe_2O_3	*CaO*	$CaCO_3$	*MgO*
3%	1.90%	52.7%	94%	0.43%

OPERATING CONDITIONS OF BLAST FURNACE NO. 2 AS FROM JANUARY 1951

As from January 1951, the Compañía de Acero del Pacífico S.A. (CAP) took charge of the operation of the Corral blast furnace. Blast furnace coke from Huachipato has been used since, on a sufficient scale to permit continuous working of the blast furnace throughout the year. See Chart 1 for the average annual consumption of the principal raw materials per ton of pig iron produced, as well as those for the preceding period. Chart 2 presents the same figures, but expressed as monthly averages since January 1949.

In 1951 production amounted to 19,770 tons. By July 1952, 12,950 tons had been produced, and maintenance of that rate would imply an annual rate of 22,200 tons. This figure may be exceeded if the new turbo-compressor should be put into operation in good time. During the period under review the month showing the highest production is May 1952, with a total of 1,960 tons, and the record day is 4 May 1952, with 76 tons, 37% of coke having been used. All these figures refer to clean pig iron, excluding scrap, and when comparing them with those for the preceding period, some 8 to 10% should be added.

The analyses of the different types of pig iron produced are similar to those indicated for the earlier period, except that sulfur rose from 0.02 to 0.04%.

As in the preceding case, present operating conditions are given below, to facilitate comparisons:

Air pressure and temperature

The same compressor has been in use; in August 1951 a general adjustment was made of the speed

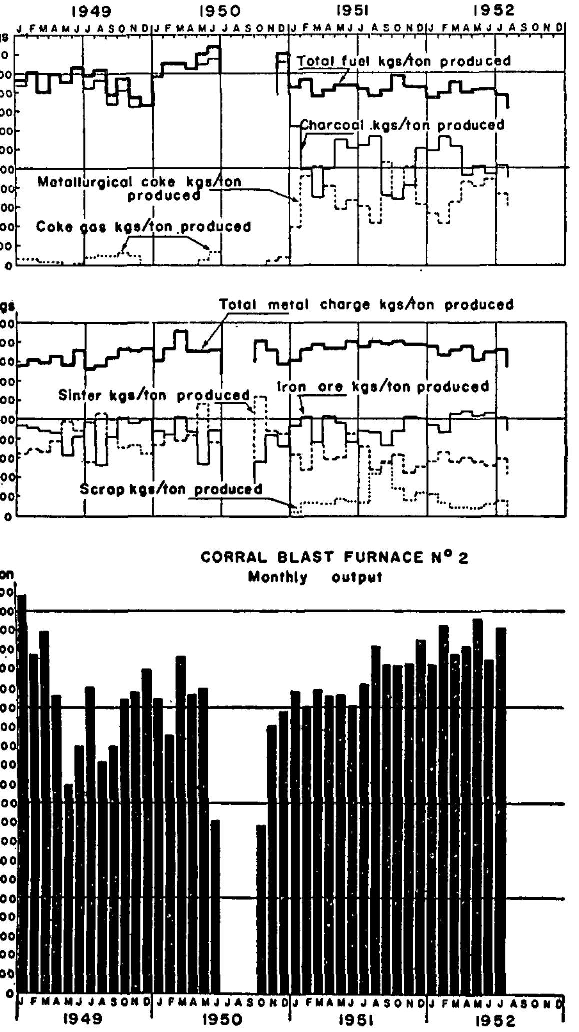


Chart 2
Corral blast furnace No. 2.
Monthly output.

regulator and of the distributon system of the steam engine, which immediately resulted in higher production, as may be appreciated from Chart 2, showing the monthly data.

The same Cowpers were used to pre-heat the air, from 550 to 650°C.

Iron ore

The same iron ore is used, and its average analysis has remained practically the same.

Sinter

The same sinter is used; in recent months its quality has deteriorated a little due to wastage in the exhauster.

Charcoal

The same method of carbonization on the mountain is used. Various steps have been taken to ensure a greater supply, i.e.:

(*a*) A new storage shed was built and it has been used since the autumn of 1952;

(*b*) A sufficient quantity of sacks was bought;

(*c*) New contract systems were established in order to ensure a better supply of charcoal bought from third parties;

(*d*) A stricter control was set up to obtain a better quality.

Weather conditions were not very favourable, particularly during the latter months of 1951, so that stocks of charcoal decreased, and were practically exhausted in November, thus affecting the stocks planned for the winter period of this year.

The analysis of charcoal is practically the same as that given for the earlier period.

Calories	*Weight m³*	*Volatile matter*	*Fixed carbon*	*Ash*	*Moisture*
6,800 to 7,000	220 kg.	10/11%	73/76%	2.7%	5 to 12%

Limestone

Limestone from the Guarello deposit was used. This is located on one of the islands in southern Chile:

SiO_2	Al_2O_3 plus Fe_2O_3	CaO	MgO
0.63%	1%	54.1%	Traces

Coke

Metallurgical coke from the Huachipato plant was used. This coke has been made using a mixture of Pocahontas and Chilean coal. The percentages have varied, and so has the time of coking. The representative analysis is:

Volatile matter	*Fixed carbon*	*Total sulfur*	*Moisture*	*Ash*
0.10%	88%	0.91%	0.16%	10.83%

PLAN TO IMPROVE THE OPERATING CONDITIONS OF THE BLAST FURNACE IN CORRAL

Air temperature and pressure

A new turbo-compressor is being installed, with a capacity of 400 cu. m. per minute and 600 mm. mercury column; it is expected that this will be in operation by the end of 1952.

One Cowper has new refractories, and those of another one are being changed. Thus there are three Cowpers with new refractories and a fourth in reserve with the original ones. It is expected to raise the air temperature to 700 or 740°C., without any fear of the charcoal beginning to burn at the top.

It is anticipated that production can be increased up to 130 or 140 tons a day, using the same type of charcoal as at present and some 25% of coke.

Sinter

A new exhauster with a capacity equal to that of the existing one is being installed, and tests are to be made with the two exhausters in series, after reconditioning the present one, or alternatively, to install an exhauster with a greater capacity (200 to 250 cu. m. per minute and 700 to 750 mm. of water) with which it is expected to improve the sinter, increasing the degree of oxidation from 92 to 96 and also to increase production from the present plant.

Charcoal

A total survey has been made of the reserves on the mountains within the Company's concessions, classifying the different species. The analysis of this information, and the investigation of the nature of the soils, leads to the forecast that within a period of eight to ten years a rotation can be obtained which will permit an annual production of 26,000 tons of charcoal. This will be achieved only providing that the wood is carbonized in suitable ovens, that the branches and parts of the tree trunks at present lost be utilized, that during the next season an intensive reafforestation plan is commenced, and that the existing mountain is exploited, with forestry control so as to permit a faster growth of the most suitable natural species. At present the company is studying which carbonization oven would be the most appropriate.

Briquetting plant

At present some 30% of the coal tonnage received at the plant consists of fines unsuitable for the blast furnace. A small percentage of this is used in the sintering plant, or sold to the nitrate companies for the manufacture of gunpowder, but there is always a considerable surplus which at present is lost. Briquetting experiments have been made in the laboratory, without very satisfactory results, with first tar and then pitch being tried as binding agents. Starch, molasses, powdered vulcanized rubber (by-products from the manufacture of tyres), etc., have also been used, none of them with very promising results, mostly because they impart no resistance to compression and erosion when in combustion. New experiments have recently been started with an organic resin which it is hoped will give better results.

The best results to date have been obtained from tests using 25% (in weight) of limestone, calcined and then quenched in water.

Other improvements

A mechanical screening system for the charcoal has been installed, with its mechanized storage and transport system.

A mud gun was installed for the automatic tapping of the blast furnace. A pig casting machine is planned to eliminate making sand moulds for the billets.

A foundry has been developed using the hot metal from the blast furnace. This foundry mainly casts ingot moulds and their bases, both for use in Huachipato. In July last, production amounted to 450 tons of ingot moulds with fairly satisfactory results. The analysis of the iron used corresponds to the type 155 mentioned before.

COMPARISON BETWEEN THE OPERATION OF THE BLAST FURNACE WITH CHARCOAL AND WITH THE MIXTURE OF CHARCOAL AND METALLURGICAL COKE, AND CONCLUSIONS

The operating conditions which have just been described show that it was necessary to make large repairs and introduce many improvements. It has, therefore, not been possible to operate the blast furnace for any length of time with varying coke-charcoal mixtures while keeping other factors constant. This would have been the only way of establishing valid comparisons regarding the results of such mixtures. Nevertheless, some interesting observations may be made:

(*a*) The fuel consumption per ton produced has fallen from the 980 to 1,000 kg. when charcoal was used, to 900 or 920 kg. using the mixture with coke, but part of this gain may be due to better quality of the charcoal;

(*b*) The average coke used in the mixture has been around 40% (due above all to supplies of charcoal) but it is estimated that a mixture of 25% coke would give better results. This is with the currently available type of charcoal made on the mountain, but it is probable that through carbonization in distilling ovens the percentage of coke would be reduced even further and might even be completely eliminated without affecting production;

(*c*) The quality of the charcoal has a considerable effect on production, even when an appreciable quantity of coke is used in the mixture. Inversely, a charcoal which is carbonized at a suitable temperature can have a favourable influence on production, even when a coke mixture is not used (in the case of the production of 89 tons per day achieved on 16 January 1949);

(*d*) There are other factors which may have a more decisive influence on the production and consumption of fuel, such as the suitable volume of air and the air pressure required by the blast furnace at a given moment, the air temperature and quality of the sinter.

The Sintering Plant at Monlevade

FRANCISCO JOSÉ PINTO DE SOUZA

REASONS WHICH JUSTIFIED THE INSTALLATION OF THE SINTERING PLANT

There were three main reasons which motivated the installation to which this paper refers:

A. *Ore*

The Monlevade plant utilizes ore obtained from the Andrade deposit, 10 km. from the plant, and served by a railway owned by the Company. This deposit, one of the best known in Brazil, has approximately the following reserves:

(*a*) Compact hematite and itabirite: 30 million tons, 67% Fe, 0.015 to 0.04% P;

(*b*) Pulverous pure jacutinga: 40 to 50 million tons; grade similar to (*a*);

(*c*) Canga: 4 million tons, 61 to 63% Fe, 0.08 to 0.3% P;

(*d*) Poor ores (mainly pulverous): over 20 million tons, 40 to 60% Fe, and about 40% SiO_2.

Many years experience in Monlevade had shown that the blast furnaces, working with 50% canga, could admit an additional 50% of hard itabirite and hematites, without a noticeable drop in production, or an increase in the coal rate. If the percentage of canga dropped, the same happened to the throughput of the blast furnace, while the coal rate increased.

As the canga described under (*c*) in any case contained 50% itabirite, there was only a practical reserve of 4 million tons of ore. In addition, the deposit is a hill with an average slope of 40 degrees, and the canga is distributed all over the surface, with a thickness which only occasionally exceeds 2 m. This fact complicated extraction and made mechanization impossible.

The sinter plant, by allowing the use of any ore, increased the reserves of Monlevade from 4 to 100 million tons and, with its crushing and screening plant, made mechanization possible.

Moreover, in order to produce 600 tons per day, it had formerly been necessary to employ 300 workers plus 12 to 15 five-ton trucks; at present, with one steam shovel and five 15-ton trucks, 45 men are sufficient to handle a daily production of 1,000 tons.

B. *Coal*

Charcoal constitutes without doubt the primary problem of the Minas Gerais steel industry. This is not due to the exhaustion of the forests; it has been proven that the Vale do Rio Doce forests can maintain indefinitely a one-million-ton-per-year steel industry, depending either on natural growth or on artificial forests.

The critical point for the steel industry is the cost of the charcoal and the possibility of its intensive production.

A forest can be compared with a coalfield, with the following advantages: it renews itself and produces an extremely pure coal. The principal disadvantages are: less heating value and lower resistance (which limits the size of the furnaces), while, from the point of view of its exploitation, it would be equivalent to a coal seam of only a few centimetres in thickness.

This last disadvantage, the most serious one, imposes a dispersion of work and the continuous moving of the working fronts, long hauling distances, and so on.

In addition to a reafforestation plan, Belgo-Mineira tried to improve the utilization of fuel in its plant. It was for this reason, among others, that the sinter plant was installed.

The hauling of the charcoal to the plant produces a considerable deterioration, the fines (under 12 mm.) being inadequate as a blast furnace fuel. It is precisely this dust, previously discarded, which is used in the sinter plant.

C. *Increase of production of the blast furnaces*

This was the third reason for installing the sinter plant, and as will be shown elsewhere in this paper it has been satisfactorily attained.

DESCRIPTION OF THE PLANT

The plant consists of a sintering screen with a capacity of 300 tons of sinter in 24 hours.

The ore comes from the mine with a maximum size of 50 cm. and passes over a vibrating bar screen with 38 mm. between bars; the larger sizes are fed into a jaw crusher which reduces them also to a maximum of 38 mm., and this crusher has an hourly capacity of 50 to 60 tons.

All the ore below 38 mm. is passed over a vibrating screen with 25 mm. mesh. The 25 to 38 mm. fraction is sent to the blast furnaces, having a more uniform size than any ore received previously. The −25 mm. fraction is used for the sinter blend.

The charcoal fines are reduced to a maximum size of 2 to 3 mm. in a cylinder mill, the capacity of which is approximately 20 cu. m. per hour.

The various raw materials: sized ore, charcoal, rolling mill scales, refuse under 7 mm. from previous sinter operations, are stocked in bins and transported, in the correct proportions, by belt conveyors to a mixer. The box shaped mixer is provided with two shafts which carry the necessary agitating blades. A variable percentage of moisture is added within the mixer to the blend.

The mixture is then carried to the sintering grate, or panel, the grates of which are first covered with a thin layer of sized ore, in order to prevent the fine material from dropping through the grate. Covering the mixture, another layer richer in coal is spread out, in order to facilitate ignition. The total thickness of the bed may, of course, be regulated by adjusting the speed of the feed distributing car.

Diesel oil is atomized over the sinter bed by the ignition car. Under the sintering grate, two exhausters, working in series, produce a vacuum of up to 1.5 m. water column, and exhaust the combustion gases through a chimney.

The capacity of the sinter plant is about 40 tons of sinter per batch. After combustion has been completed, the sinter drops, passing through a crusher into a silo where it is cooled, and then, after screening, taken to the blast furnaces.

SOME OPERATIONAL DETAILS

The finished sinter is screened on a 7 mm. mesh and the undersize is returned to be added, as mentioned before, to the sinter mixture.

About 180 to 200 tons of rolling mill scales are returned monthly to the sinter plant. They contain about 73% iron and 1 to 2% SiO_2, and constitute, therefore, an excellent sintering raw material.

Sawdust has unfortunately not proved satisfactory; a maximum of 2% could be added, after lengthy experiments, without affecting the resulting sinter either way.

Calcinated lime has also proved unsatisfactory; the calcinated material of the bottom layers, receiving moisture from the upper layers when ignition began, was reduced to dust, and clogged the grates. Good results have been obtained with open hearth slag and water granulated blast furnace slag, especially the latter, which really constituted a revolutionary change. Working with charcoal and extremely rich ores (67%), in order to ensure the minimum indispensable slag forming material in the blast furnace, it was decided to include in the sinter a certain amount of silica and lime. Other Brazilian blast furnaces return blast furnace slag to their blast furnaces. In the case of Monlevade, on the other hand, the difficulty encountered in sintering jacutinga was largely due to its very small size, and its extreme lack of slag forming material. Now, the granulated blast furnace slag, with its coarser grain, would provide the missing slag forming material and increase the permeability of the sinter.

As a result of the research work done in Monlevade, and of these modifications, the rated capacity of the sinter plant, originally 300 tons per day, was raised to 450 tons; the hardness factor of the final product improved, the porosity increased to such an extent that the weight of a cubic metre is now 1.5 tons. The degree of oxidation is now 94 to 96%, whereas originally it amounted to only 90%.

The amount of blast furnace slag which permitted the attainment of the above-mentioned improvements represents about 10% of the ore employed.

The charcoal dust mixed into the sinter has dropped from something between 450 to 500 litres, to the present figure of some 300 litres. Investigations have shown that, consequently, the carbon added to the mixture, on a dry basis, ranges between 2.8 and 3.2%.

Diesel oil consumption has been reduced to less than 1.5 litres per ton of sinter.

In addition, some changes have also been made in the mechanical equipment, as for instance: the original simple cyclones have been replaced by water injection cooled cyclones, thereby reducing the temperature of the gases; this improves both the suction and dust precipitation.

Originally, when employing a considerable amount of jacutinga ore, an operation cycle in the sinter plant took from 22 to 30 minutes. At present, it has dropped to 15 minutes and less, and is occasionally only 12 minutes.

As regards labour consumption, the sintering plant requires ten men per shift, plus two who work only one shift on the crusher and sieves. Adding the men necessary for maintenance and minor repairs, the total amounts to 49 men per day. The installation of a second sintering unit of the same capacity, already decided upon, would require only one man more. Supervision is done by two operational foremen, one mechanic, one engineer and one clerk.

RESULTS OBTAINED

The sinter plant, even after the improvements described, is unable to supply more than 50% of the ferrous material for the blast furnace. If the latter is operated with sintered material alone, a charcoal consumption of 650 kg. per metric ton of pig iron can be obtained, which represents an improvement of more than 20% against operation with 50% canga and 50% hard compact ores. The duplication of the present sintering capacity has been envisaged for the very near future, and it has been decided that any enlargement of the blast furnace capacity should be based entirely on sintered material.

As stated at the beginning, after almost three years of operation of the sintering plant, research and experiments for its improvement are still continuously conducted and it is felt that the above figures do not represent the limits of the possible gains which can be obtained through sintering.

GENERAL CONCLUSIONS

Finally, a few observations and conclusions are given which might be useful for the planning of similar operations:

(1) The Central Minas Gerais ores are extremely abrasive, and the same is true for the ground charcoal, this disadvantage being further increased by the indispensable moisture. Every precaution should be taken to overcome this difficulty and, wherever possible, rubber belt conveyors should be employed instead of steel elevators or conveyors;

(2) When planning the silos and bins for stocking the raw materials, it should be borne in mind that it is better, in order to ensure a more homogeneous mixture, to provide for a larger number of smaller silos;

(3) Care should be taken with the cooling of the sinter, so as to ensure the return of the sinter fines to the mixture for re-use, after as perfect a cooling as possible.

Summary of Discussion

At the invitation of the CHAIRMAN, Mr. LEUSCHNER commenced the proceedings with a brief introductory address. The meeting would be devoted exclusively to iron ore reduction in the charcoal blast furnace. That subject was perhaps of interest only to Latin America and the most important aspect of the problem was to determine whether the charcoal blast furnace constituted just another instrument of deforestation or whether it could be of permanent value to the steel industry. For that reason, the views of the Brazilian experts were particularly welcome and timely, since Brazil had had much experience with the charcoal blast furnace.

Pig Iron Production in Blast Furnaces Using Charcoal, *presented by Mr. Lanari in the absence of the author*

Mr. MARTIJENA opened the general discussion of the paper by expressing his support of the opinion that the charcoal blast furnace was ideally suited to Latin America, especially to Brazil and Argentina. Indeed, Argentina had no other metallurgical fuel but charcoal at its disposal. He was, therefore, keenly interested in the subject under discussion. Argentina had followed the example set by Brazil rather closely and had decided that the best solution to the blast-furnace problem lay in the use of artificially cultivated forests. The development of such a venture in the sub-tropical zones of the country guaranteed a considerable supply of wood and the only attendant problem was the cost of transport. Although the quality of the charcoal produced was usually good, operation costs were very high. It was believed that there was some possibility of making charcoal in vertical retorts with internal heating. Such a process had been tested in Sweden with good results. However, the experiments undertaken by Argentina were still inconclusive and he wondered whether Professor Kalling would care to express an opinion on the process in question and forecast its possibilities.

Mr. ALLARD asked Mr. Lanari whether, in his view, the combination of the use of charcoal with sintering would present any problem, and whether the total yield of charcoal would be increased by the use of charcoal fines.

He was also puzzled by the reportedly small amount of air needed in the charcoal blast furnace and requested clarification on that point by Mr. Lanari.

Mr. LANARI replied that the application of sintering had yielded good results. Quite a large percentage of fines were used and, in that respect, discarded or otherwise unusable minerals were utilized to advantage. Mr. Allard's first question could certainly be answered in the affirmative.

Referring to a point made by Mr. Martijena, he said that Brazil's main problem in connexion with charcoal production was the transport of the wood to the ovens. The current practice was to make the charcoal at the locations where the wood was obtained.

Mr. KALLING had noted the data contained in Mr. Ensch's paper with special interest and had found that the results obtained by Brazil were in close agreement with those observed in Sweden. The Brazilian output seemed to be somewhat higher than that registered in Sweden, where the maximum weekly output was not over 1,050 tons. In Sweden the heavily increased price of charcoal had forced an extensive change-over from charcoal to coke in the blast furnaces during the past few years, even in the case of pig iron intended for high-grade steels. A difficulty had been the higher sulfur content in the coke pig iron, but that problem had been solved by using effective desulfurization methods after the blast furnace. In addition, the usually somewhat higher phosphorus content in the coke pig iron had caused some trouble, especially when the iron was converted into steel in the acid bessemer or open hearth processes, but it had been found possible to select coke with a satisfactory phosphorus content for that purpose too. No difference in the quality of the steel made from charcoal or coke had been observed as yet.

Mr. LEUSCHNER remarked that the phosphorus content of charcoal depended on the type of wood used and the soil of the location of the forest. In Brazil, for example, experiments had been made with four or five different varieties of eucalyptus with the result that certain varieties had been found to be practically free of phosphorus.

Mr. KALLING stated that, in his view, the points expressed in the paper under review in respect of sintering and agglomeration were quite sound. In Sweden it had been found that the advantageous influence on the charcoal (or coke) consumption by using a high percentage of sinter in the burden was particularly marked when only sintered ore was used. The best results were obtained when the limestone was also sintered together with the ore.

Mr. LEUSCHNER observed that the largest percentage of charcoal fines was produced from the bark of the tree, where the phosphorus content was highest. That fact, in the opinion of some Brazilian operators, might be important.

Mr. SEM expressed interest in the comparison made by Mr. Ensch between the charcoal blast furnace and the electric furnace, to the effect that the installation cost of an electric furnace was higher than that of the charcoal blast furnace. From comparative data that he had examined, he had understood the reverse to be the case.

Regarding the capacity of the electric smelting furnace, it should be noted that an increase in the number of electrodes had the tendency of creating disturbances in the furnace. The electrical resistance of charcoal was probably higher than that of coke, and that factor should be considered in connexion with the use of electric furnaces in a country like Brazil.

Mr. LANARI, replying to Mr. Sem's remarks on the cost of installing electric furnaces, said he could not presume to speak for Mr. Ensch, but a possible reason for the higher cost of an electric furnace might be the cost of the electric generating plant which, if added to the investment in the furnace, exceeded that of producing charcoal.

Mr. CANGUILHEM observed that Mr. Ensch had drawn the conclusion that Latin America would have to depend on the charcoal blast furnace to support the iron and steel industry. He was of the opinion that such a conclusion might be somewhat exaggerated, inasmuch as there were several Latin-American countries which operated the coke blast furnace and which produced a substantial amount of iron and steel. He could not, therefore, support the view expounded by Mr. Ensch as applicable to Latin America as a whole.

Mr. LANARI replied that, in his own personal opinion, the charcoal blast furnace was well suited to Latin America. He did not wish to imply that it was the only type of furnace that could be used. Certainly, coke blast furnaces and electric furnaces could also

be used in the region. Mr. Ensch's comments on the charcoal blast furnace were prompted by a desire to use the raw material represented by suitable woods which existed in most of the Latin-American countries.

Mr. MARIN GONZALEZ remarked that during the previous century the charcoal furnace had been used in Mexico for the production of pig iron, but at the beginning of the twentieth century large plants had been built which did not use the process. Smaller plants, however, continued to produce a good grade of charcoal pig iron.

Operation of the Charcoal Blast Furnace at Corral Using Mixtures of Metallurgical Coke and Charcoal, *presented by the author*

Mr. VUCETICH began his presentation by stating that he intended to dwell mainly on statistical data on Chilean pig iron omitted from the paper.

The day of maximum output based on charcoal operation exclusively showed the following figures:

	Tons
16 January 1949 (day of maximum output)	89
Three previous days' output	64.5 66.5 74.9
Three following days' output	70 65 68

The figures for charcoal and coke operation were the following:

	Tons
4 May 1952 (day of maximum output)	76
Three previous days' output	71.8 66.8 73
Three following days' output	72 68 67

It was possible that the production figure given for 16 January 1949 might be too high compared with the figures for the three previous and three following days. The figure of 89 tons might well have been the result of statistical differences, so that a reliable maximum figure would be the average for that 7-day period.

He also gave the following data on the woods found on the company property, which covered an area of 21,000 hectares:

	Per cent
Tineo, ulmo, tepa	49
Coigue, mamo, olivillo	35
Canelo luma	14
Others	2

The foregoing were all soft woods averaging a yield of 220 kg. per cu. m. They were not representative of the woods found in the country as a whole.

Mr. MARTIJENA stressed that Argentina had great faith in the use of charcoal. There were three varieties of wood used: hard, medium and soft, with an average yield of 250 kg. per cu. m. As was the case in Chile, the soft woods yielded a considerable amount of fines. Experiments with the eucalyptus had indicated its suitability for the blast furnace and resulted in higher production and lower fuel consumption. Hard woods were also found to be highly satisfactory for the blast furnace. He invited the comments of the experts present on the practicability of increasing the pressure of the blast, taking into account the fact that woods with a yield of 200–250 kg. per cu. m. were used. The temperature of the blast was not modified.

Mr. PRADO UCHOA commented on Mr. Vucetich's statement to the effect that, owing to the frequent loading and unloading of charcoal in Chile, a considerable loss was sustained due to the resultant increase in the amount of fines. He wondered whether it would not be advisable to transport the wood in its natural state to the plant site and carbonize it on the spot.

Mr. VUCETICH replied that the main problem was the cost of transport. It was much more expensive to transport natural wood than charcoal and, for that reason, the charcoal was produced where the woods were found and later transported to the plant.

Mr. LANARI stated that Brazil was fortunate in being able to work all year round in producing charcoal. Chile was obliged, because of climatic conditions, to cease operations during certain months of the year. Brazil's only difficulty during the rainy season was due to transport hazards.

With regard to charcoal consumption and the blast furnace burden, he had observed that there was a great variation in the latter, which should perhaps be attributed to the utilization of fines and residues. There was no such indication in the paper under review, according to which it appeared that fuel consumption increased in ratio to any increase in the sinter charge. In his view, the reverse should be the case, but it was quite possible that fuel consumption was affected by other circumstances.

He also noted that there was a difference in the dimensions of the furnaces used in Chile and those used in Brazil. The height of the bosh of the Chilean furnace would be too great for Brazilian purposes, and that factor might well be the cause of the difference in fuel consumption.

Mr. VUCETICH referred to the apparent discrepancy between fuel and sinter consumption. If a poor sinter were used it would damage the furnace. As he had pointed out before, the equipment in the sintering plant at Corral did not produce good sinter. The temperature of the air blast was also low and it was hoped that an increase in temperature would lead to greater output.

At the invitation of the CHAIRMAN, Mr. Ballón took over the chair.

The Sinstering Plant at Monlevade, *presented by Mr. Leuschner in the absence of the author*

In view of the earlier discussion on sintering, there were no questions on this paper.

IRON ORE REDUCTION

Methods for reducing ores otherwise than in blast furnaces

21 October 1952—Afternoon

Chairman:

Alfredo GONZALES BALLESTEROS, Assistant Superintendent, Departamento de Altos Hornos, Compañía Fundidora de Fierro y Acero de Monterrey, S.A., Monterrey, Nueva León, Mexico.

Contributed Papers:

Progress in the Manufacture of Pig Iron and Ferro-alloys with the Low-Shaft Electric Furnace
Hermann WALDE, Direktor der Demag-Elektro Metallurgie, A.G.m.b.h., Karlsruhe, Germany
The Electric Reduction Furnace
Alfonso BALLÓN, Director del Departamento de Siderurgia, Corporación Peruana del Santa, Lima, Peru
Electric Smelting of Pig Iron
M. Ø. SEM, Technical Director, Elektrokemisk A/S, Oslo, Norway
Making of Steel in Electric Furnaces
H. S. NEWHALL, Manager, Electrothermic Division, Pittsburgh Lectromelt Furnace Corporation, Pittsburgh, Pa., U.S.A.

Summary of Discussion:

Participants: Messrs. NEWHALL, WALDE, BALLÓN, SEM, WARING, SANITER, PYLES LOZANO, CANGUILHEM, LOBO, YDSTIE, KALLING, PRIETO, DURRER, SHEROVER

Progress in the Manufacture of Pig Iron and Ferro-alloys with the Low-Shaft Electric Furnace

HERMANN WALDE

The arc furnace has always been used in the ferro-alloy industry built up to satisfy the large-scale demand for alloy steels and depending on high reaction temperatures. The use of this furnace was based on the types of furnace employed in the carbide industry and the experience afforded therein. The electric furnace is ideal for developing reduction processes with a concentrated energy consumption.

More recently, consideration was given to the use of electric heating for iron ore reduction to obtain pig iron, particularly in countries poor in coal but rich in hydro-electric power. In this respect the following characteristics of electric heating are of decisive importance:

(*a*) By electric energy, heat can be concentrated in the material to be melted. In principle, this can be done either by using the electric arc or by induction heating. Inductive heating has the advantage that the heat is developed entirely and exclusively in the material to be heated, but it is usually so costly that preference is given in practice to arc heating;

(*b*) Impurities are introduced on a negligible scale or not at all. With arc heating, unlike induction heating, a certain carburation is inevitable, as is the passage of traces of ash from the electrodes into the material to be melted. When delicate alloy steels are produced in the arc furnace, this may be an important factor, but it is insignificant in reduction processes as in most cases the raw materials are not pure. They are minerals, that is to say, mixtures of the most diverse combinations and degrees of oxidation, containing foreign matters in any event;

(*c*) Electric heating replaces coal heating when saving in coal is a decisive factor from the metallurgical viewpoint and when acceptable carbon-bearing substances are expensive while electric power is cheaply available;

(*d*) Electric heating is very convenient and the temperature can be rapidly adjusted to suit the work in hand.

Problems of pig iron production by the electrical method will be dealt with first, because they are more important from the economic and technical standpoint and because much more progress is being made in that field than in the manufacture of ferro-alloys, since in the latter case the technique of the processes is more advanced.

At a conference in 1947 Durrer (1) mentioned that in the manufacture of pig iron, the costs of coal and electric power are decisive factors in the purely economic comparison between the electric furnace and the thermic blast furnace, equality being attained when one kilo of coal costs eight times more than a kilowatt hour; this is calculated on the basis of a theoretical equivalence between a kilowatt hour (860 k./cal) and 1 kg. of coal (7,000 k./cal).

Durrer further pointed out, however, that in practice the ratio is lower, that is, 1:6, because apart from the purely economic equation, there are other factors to be considered. The points most decisively in favour of the electric furnace are the following:

(1) The blast furnace requires solid and resistant metallurgical coke, able to support the burden which also has to be permeable to gases in order to allow them to perform the all-important indirect reduction. On the other hand, the low-shaft electric pig iron furnace can use such carbon-bearing materials as anthracite, charcoal, peat and even lignite.

(2) In contrast to the blast furnace, good economic results can be obtained using small electric furnaces, as, for instance, furnaces of some 3,000 kVA and upwards in size, producing 25 to 30 tons of pig iron per day. In addition, installation costs for an electric furnace are much lower because costly equipment for blast heating is eliminated once and for all.

(3) The electric furnace gives a gas free from nitrogen ballast, containing nothing but CO, H_2 and CO_2.

(4) The electric furnace is adjustable to the supply of power and is thus easily adapted to prevailing power supply conditions. The halts caused by current interruptions can be met with relative simplicity, and when stoppages do occur, the low cost of the installation affects amortization costs favourably.

The drawbacks to the blast furnace described in points (1) and (2) have led to consideration being given to the introduction of the low-shaft furnace which could, in principle, be developed either as a thermic furnace or as an electric furnace. However, under the influence of the progressive electrification in industry, the first electric low-shaft furnaces were built and operated over thirty years ago. Moreover, so much practical experience has been obtained that the low-shaft electric furnace may be regarded as being entirely developed, whereas the thermic low-shaft furnace is still in the development stage, even though such work is currently being undertaken in different places.

As the thermic furnace also has to use blast which, besides using up energy, gives rise to the undesirable ballast of nitrogen in the gas output, the electric low-shaft furnace will be preferable to the thermic furnace when hydro-electric energy is relatively cheaper than coal and when a nitrogen-free gas is required.

It follows that in the selection of types of furnaces, the prices of power and coal or coke must be studied first.

In recent years, there has been considerable progress in the study of the carbon-bearing materials that may be used in the electric pig iron furnace. In this connexion, the work of Sjögren (2) merits special mention. In these tests described elsewhere, it was proved that coke or anthracite may be replaced entirely by dried lignite.

Whilst normally the gas obtained in the pig iron electric furnace using coke or anthracite contains about 76% of CO, 8% H_2 and 16% CO_2, the values change, for instance, to approximately 55% CO, 22% H_2 and 13% CO_2 when lignite of this kind is used. That is to say, the moisture in the coal partly occasions the water gas reaction which increases the hydrogen content at the expense of carbon monoxide.

Development will doubtless proceed along lines of prior distillation of the high gas-containing coal at low temperature, thus obtaining a low-temperature coke (char), the high reaction capacity of which acts favourably on the reduction of iron ores. At the same time the idea is put forward of thoroughly mixing the iron ores with lignites and submitting it (briquettes) to previous low-temperature distillation to eliminate part of the carbon, tars, etc., while at the same time effecting a pre-reduction of iron ores. The possibility of using this process to make low quality coal and inferior ores apt for use will, together with the obtaining of good gas, undoubtedly be very important for the future of iron manufacture.

The average composition of pig iron obtained in the electric furnace is as follows:

Carbon	4–4.5%
Silicon	0.5–4%
Manganese	0.2–1.5%
Sulfur	0.01–0.004%
Phosphorus	0.05–0.3%

The carbon content corresponds to saturation with dissolved carbon. The silicon content depends on the temperature of the melt in the furnace as it increases with working temperature. It may be kept within given limits through action on the ratio of amperage to voltage in the electric furnace.

Sulfur slags more easily in the electric furnace than in the blast furnace, because the temperature in the fusion zone is higher.

Phosphorus passes into the iron in the usual way. If the pig iron is to be used for foundry purposes, the desired composition can generally be cast without difficulty. If the pig iron is destined for steel, however, then, in the first place, it has to be blown in the converter by the basic or acid process or, alternatively, the phosphorus may be slagged off in the arc furnace in which, under certain conditions, working with oxygen is recommendable.

Power consumption for manufacturing pig iron in electric furnaces naturally depends primarily on the iron content and the purity of the ores. The lowest figure obtained in practice is about 2,000 kWh/t, using excellent Swedish ores, while, for example, with ores containing about 53% Fe and about 11% of silica, energy consumption figures of up to 3,000 kWh/t have been observed, because large quantities of lime for slagging had to be added and melted. Power consumption is estimated at 550 to 650 kWh/t per ton of slag. In the first case mentioned, the quantity of slag amounts to about 300 kg. per ton of pig iron, in the second it is about 600 or 700 kg.

The slag ratio:

$$\frac{CaO+MgO}{SiO_2-Al_2O_3}$$

should approximate 1.3.

For uniform operation, the electric furnace should have a fusion charge with relatively high electrical resistance, so that the arc may be spread over the largest possible area to produce uniform heating. It is therefore desirable to distribute as finely as possible the lime required for slagging, the acid components and the sulfur. The best way of doing this is to sinter the ore with lime. Nevertheless, for every ore there must be a careful study of the extent to which ore preparation is justified. It may also be necessary to decide whether the ore is to be agglomerated or sintered, and what process is best for this.

Table 1 shows an average analysis of an ore compared with that of the pig iron and slag produced. As will be seen, the iron content of the ore appears almost integrally in the pig iron. The latter contains also the amounts of silicon which are desirable in the process, that is, in practice, between 0.3 and 6%. The rest of SiO_2, Al_2O_3, CaO and MgO pass into the slag.

Table 1

ANALYSIS OF ORE, PIG IRON AND SLAG PRODUCED BY PARTIALLY SINTERED MAGNETITE ORE

	Ore %	*Pig iron %*	*Slag %*	*Remarks*
C	—	4.5	—	
Fe	53.5	93.2	1	1% loss
SiO_2	11	1.4	35	
Al_2O_3	1.4	—	5.3	Total content derived from the ore, limestone and slagged coal ash
CaO	1.3	—	22	
MgO	9	—	32	
Mn	0.5	0.8	1	
S	0.02	0.01	0.04	Predominantly slagged
P	0.04	0.08	—	Practically totally in the pig iron

The resulting gas is a valuable product which can be used for heating furnaces, or else in gas motors, but in addition, because of its lack of nitrogen and its concentration of carbon monoxide, it can be very useful as a synthesis gas for the chemical industry. For comparison purposes between the electric furnace and the thermic blast furnace, it may be mentioned that in the electric furnace 76% of the coal introduced into the furnace escapes in the form of CO gas whereas in the high-shaft thermic furnace this value is approximately 49%. This means that, due to its high reaction column, the thermic furnace effects the indirect reduction of the ores more integrally, and therefore makes better use of the carbon content in the furnace, that is it produces a larger quantity of carbonic anhydride. On the other hand, however, it shows that the electric pig iron furnace is not only an apparatus for the manufacture of iron, but is also a large-scale producer of gas. While the ratio between CO and CO_2 in the thermic blast furnace is approximately 2:1, in the electric furnace it is 5:1. If at the same time the nitrogen content of the gas is considered, which is about 55% in the thermic blast furnace, and is practically nonexistent in the electric pig iron furnace, it must be deduced that, in installing an electric pig iron blast furnace, very special attention should be paid to the problem of utilizing the gas.

There has been considerable progress in recent years in the development of synthesis processes for the production of hydrocarbons such as gasoline, heavy oils and paraffin. Thanks to the works of Kölbel (3) and Engelhardt, it has been possible to produce this kind of hydrocarbons on the basis of carbon monoxide. The gas obtained from the electric furnace is, in this respect, considered specially useful for similar synthesis. Some figures are given below.

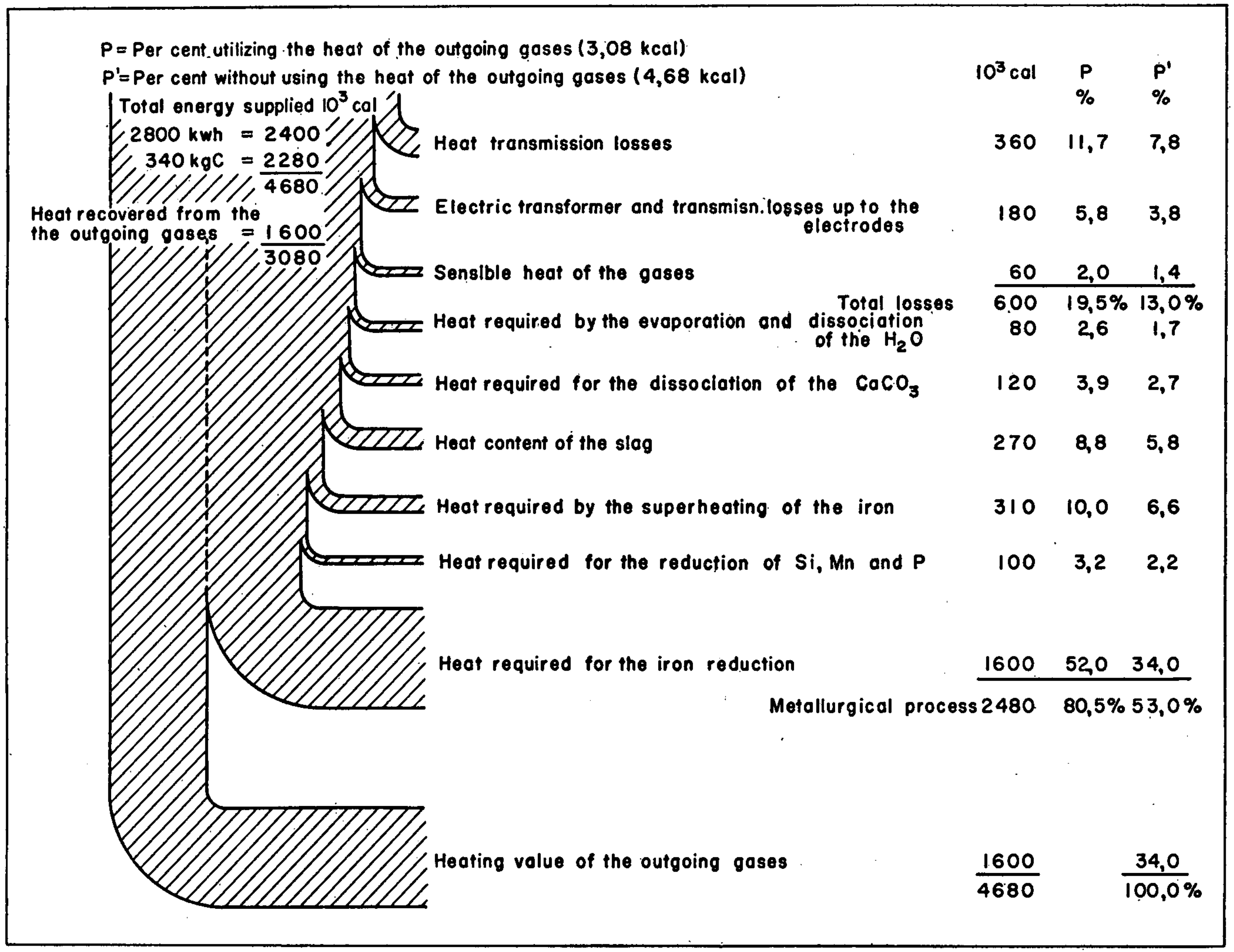
The amount of gas obtained from three closed pig iron furnaces, of 12,000 kVA each, amount to 8,000 Nm^3/h, with an average heating value of 2,500 $kcal/Nm^3$. About 1 ton/h of hydrocarbons can be produced with this amount of gas, always providing that it is a closed furnace. The economic yield of this type of installation naturally depends on the size thereof, as is generally the case in the synthesis plants. The hydrocarbons obtained from furnace gas by the Kölbel process, together with their use as fuels, are very adaptable for subsequent chemical transformation for different applications.

The thermic balance of electric smelting is given below in the case of an ore similar to that described in the above Table 1 (4). The charge estimated per ton of pig iron obtained is composed as follows:

1,900 kg. magnetite ore, partly prepared and sintered;
40 kg. manganese ore;
280 kg. limestone with about 50% CO, 6% SiO_2 and 3% MgO;
510 kg. anthracite and coke with a total C content of 340 kg.

The slag amounts to some 700 kg./t Fe. Energy consumption is 2,800 kgw/ton of pig iron. The outlet temperature of gas is 300°C.

Figure 1 shows an estimate of the thermic balance during electric smelting of this material. After establishing the amount of heat used for the different reactions and the electric losses, a remainder of approximately 7.8% of total heat supplied is obtained. It is mainly composed of losses through conduction and radiation, including heat lost by cooling of the roof and furnace shell. This diagram also shows the quantitative importance of the iron content in the fusion charge, gives indications for calculating energy consumption with other raw materials which can be used: for instance, if the ore does not contain 53% Fe, but for example, 68% Fe, and the amount of slag is simultaneously reduced from 700 kg. to some 250-300 kg; or if calcined lime is added to the fusion charge instead of limestone, and the ore and carbon-bearing material is dehydrated by means of a prior calcination or drying process, which permits considerable economies in energy consumption.



THERMAL BALANCE OF THE ELECTRIC LOW SHAFT PIG IRON FURNACE

Figure 1

As may be gathered from the thermic balance, the electric losses are relatively low. Modern transformers are so perfect as regards yield that their losses (about 1%) are insignificant. Losses from secondary electric conduction between the transformer and electrodes are more important, as they amount to approximately 5% of the energy consumption. Certain constructive progress may still be expected by applying electrophysic knowledge, which will result in a further reduction of these losses. The values corresponding to the electric resistance of the electrodes are also concealed among the electric losses, an amount of energy which in any case is largely utilized when the self-baking electrodes bake, in so far as this baking process occurs through the I^2R effect of the current.

Various ways of recovering the sensible heat from the working gases have been considered. These include using the gases for indirect reduction and ensuring that they circulate through the feeding tubes connecting the charge-hoppers to the furnace, at the same time releasing their energy to the burden and resulting in simultaneous filtration of the gases from the dust of the furnace. However, this method would be considerably complicated, and in order to ensure perfect purification of the gases with a view to their subsequent use, the most suitable technical solution is to moisten them with water, at the same time arranging a hermetic water seal next to the furnace. The latter has the added advantage of providing reasonable precautions against possible explosions. Moreover, since the contribution to the thermic balance is relatively low, recovery of sensible heat from the gases would have to be abandoned.

It is unnecessary here to enter into any detail regarding modern electric low-shaft furnace construction. Only one fundamental question should be discussed, and that is the most advantageous construction of the furnace as regards electrotechnical arrangement. In this connexion, the basic idea is that the furnace should not only have a high electric yield as regards ohmic losses, but that there should be the least possible inductive losses in current conduction so that a favourable power factor may be expected from the installation of the furnace. Evidently, the electric power companies desire to have, as their fundamental load, large consumers of current with a favourable power factor (cos phi) who are in a position to compensate the reactive current existing in the receiving networks. It is not therefore surprising that, in establishing new projects, the aim is often so to build the electric furnace from an electrotechnical point of view that it will work with a very high power factor. By means of the special arrangement mentioned above of the secondary conductors, power factors of up to 0.92 may be achieved for furnace units up to 12,000 kVA; this value is somewhat lower for larger units but, nevertheless, there are special means for raising it even, more. Since metallurgical plants, particularly where combined with steel works and rolling mills, have very poor power factors because of the numerous driving motors in partly intermittent service, such as, for instance, large motors for driving rolling mills, the importance of this power factor in the low-shaft electric furnace can readily be understood.

In this connexion, there has been considerable controversy as to the best arrangement for the electrodes, the merits of the method currently used for work with three-phase current in which the electrodes are arranged at the tips of an isosceles triangle being set against those of an in-line arrangement. From the point of view of obtaining a good power factor, it is naturally easier to equip the furnace with six electrodes in line. There is no doubt, however, that the energy concentration in a furnace where the electrodes are arranged in a triangle is considerably better and that, for achieving metallurgical reactions, a round and uniform fusion hearth between the electrodes offers more essentially advantageous conditions, above all when smelting ores of difficult reduction which are very demanding as regards slag work.

Purely geometric reasoning proves that the triangular arrangement of the electrodes has the advantage of offering a concentrated area of fusion which, due to the small radiation surfaces, permits a better maintenance of temperatures; this is a great advantage, above all with a variable delivery of energy.

Furnaces with electrodes in triangle may be regulated downwards as the energy decreases and may even be disconnected during pauses, without any considerable loss of heat.

As far as the suppliers of electric current are concerned, the question frequently arises of the curve of current consumption in a low-shaft electric furnace. Sometimes there is a tendency to draw conclusions from the uneven curves shown for current consumption of arc furnaces for steelmaking. In this case, however, it must be remembered that electric steel furnaces, the electrodes of which move freely above the bath or the scrap to be melted, produce shocks in the networks, particularly during the period of melting, and at times even cause disconnexions which cannot be avoided with the more sensitive electric regulation apparatus.

By contrast, the electric reduction furnace operates with the electrodes buried in the charge. There is in no sense an arc furnace with isolated arcs which burn beneath the electrodes but the formation of many super-heated points (through the high transfer resistances) throughout the region of the electrodes and between these and the respective parts of the charge. The more uniform the charge from a physical and chemical point of view, and the greater its electric resistance, the more even is the work of the furnace as regards its energy consumption from the network.

Figure 2 shows a diagram of a charge of this kind from the network.

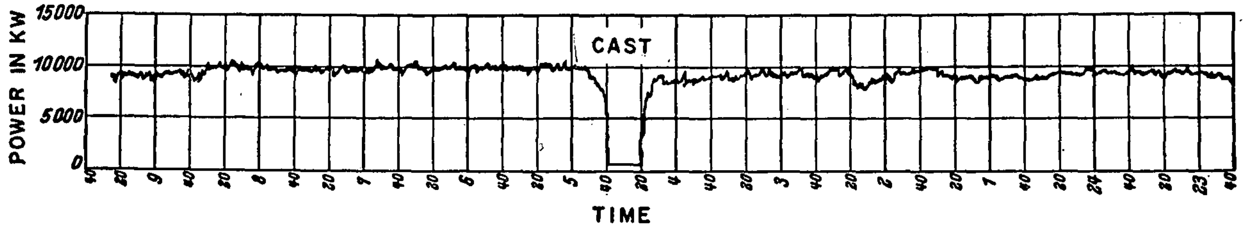


Figure 2

Operating power-time schedule for a 12,000-kVA pig iron furnace.

While the foregoing has dealt with the low-shaft electric furnace for obtaining pig iron from iron ores, a brief reference must also be made to some problems in the production of ferro-alloys. Since the electric furnace, as explained at the beginning, has the advantage over the thermic furnace from the metallurgical standpoint of the high concentration of energy in the melt, its utilization in the manufacture of ferrosilicon, ferromanganese and ferrochrome is all the more indicated as the content of silicon, manganese and chrome in these alloys increases. The electric furnace should always be used for obtaining ferro-alloys such as ferrotungsten, ferrovanadium and ferronickel.

The technique of the carbide furnace has always been used for obtaining ferrosilicon because here, above all, higher reaction temperatures are usually required than, say, in obtaining pig iron. Usually, FeSi with 75% or even 90% Si is required for the manufacture of silicon steel sheets with a low wattage loss. For this purpose a type of 7,500 kVA furnace was developed in Germany which shows very low figures of energy consumption. At the same time modern secondary conductors were provided, so that from the electrical viewpoint also the furnace was particularly well constructed. During and after the Second World War, this type of furnace was constructed with a rotating hearth in order to test the favourable effect of rotation, which had been observed for some time in other electric furnaces, on power consumption and on uniform wear of the lining. As two tons of CO are obtained per ton of silicon when reducing SiO_2 with carbon, and as this gas has a good market, the building of FeSi furnaces of the sealed type would also appear logical. However, this presents special difficulties because, in the manufacture of FeSi, apart from the amounts of dust through abrasion of the mixture, which also occurs in other furnaces, there are, in addition, considerable quantities of dissociated silicon escaping with the gases, which tend to sublimate in the exhaust conduit and therefore to obstruct it. Special precautionary measures have therefore to be adopted if it is desired to obtain the CO gases.

The manufacture of ferrochrome, due to high demand thereof for making stainless steels, has increased considerably in recent years. At the same time a material is requested with the lowest possible carbon content (super refined). The three-stage manufacturing process is mainly used, wherein the chrome ore is first smelted jointly with an intermediate blend of silicon-chrome (as this tends less than ferrochrome to absorb carbon), then the resulting slag is prepared, and finally, in a third operation, low carbon silicon-chrome is obtained.

The execution of this process and its derivations has required the design of new furnaces in which the hearth may be tilted up to 55° and, at the same time, may be run clear from the roof. This is rather one of the so-called electrode mast types which are more similar to an electric arc furnace for steelmaking, in which the electrodes are not suspended from devices separated from the furnace, but are attached to supporting arms linked through the electrode mast with the shell of the furnace.

The same type of furnace is also used to obtain ferrotungsten, ferrovanadium, ferronickel and similar alloys.

These explanations naturally give no more than a slight idea of the present state of development of the electric manufacture of pig iron and ferro-alloys. Their main pupose is to stress that the manufacture of this kind of installation requires, for a maximum technical yield, a combination of knowledge and experience in the fields of metallurgy and electrotechnics. Furthermore, when constructing the furnace, the builders should work in close harmony with metallurgists and electrotechnicians, in order to produce installations which meet all practical requirements from both the technical and economic points of view.

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The Electric Reduction Furnace

ALFONSO BALLÓN

The electric reduction furnace is a flexible metallurgical unit for producing pig iron of varying qualities, particularly in so far as silicon content is concerned. Its use is clearly indicated for those countries having abundant high grade iron ore resources and plenty of cheap hydro-electric power, but lacking in good deposits of coking coal so located as to facilitate the establishment of an iron and steel industry.

The electric reduction furnace is currently used in Norway, Sweden, Finland, Switzerland, Italy and Japan, among other countries. Its real development took place in Scandinavia, where there are installations which are proof of its economic and efficient operation.

One of the better known electric reduction furnaces is the Tysland-Hole furnace, similar in structure to the electric steel melting furnace, but with three self-baking electrodes, arranged in a triangle, as patented by Söderberg. It is provided with openings, distributed symmetrically next to the walls of the furnace. The furnace also comprises two twin units for washing the gases, each with capacity to operate singly, when the other is being cleaned or repaired.

THEORETICAL ESTIMATE OF THE ELECTRIC POWER CONSUMPTION IN THE REDUCTION FURNACE

It is assumed that the following raw materials are available since the author considers them to be typical of conditions prevailing in Latin America:

	Iron ore %	*Manganese ore* %
Fe	60.00	12.00
SiO_2	5.00	8.00
Al_2O_3	2.50	3.00
Mn	0.50	42.00
CaO+MgO	0.70	1.00
P	0.35	0.01
S	0.30	0.10

	Anthracite or coke %		*Limestone* %
C	81.00	CO_3Ca	91.00
Volatile matter	3.00	CO_3Mg	5.00
Ash	10.00	SiO_2	1.00
S	0.80	$Al_2O_3+Fe_2O_3$	1.00
H_2O	5.20	H_2O	2.00

In order first to estimate the weights of the iron (x) and manganese ore (y), it is admitted that:

(*a*) 50% of the Mn contained in the ores goes to the pig iron;

(*b*) Pig iron of 94% Fe and 1.00% Mn should be produced;

(*c*) Loss by the charging opening amounts to 6%, so that 1,000 kg. of Fe has to be supplied for every ton of pig iron produced;

(*d*) The loss of Fe in the slag is offset by that supplied by the anthracite ashes.

The equations are:

$$0.60x + 0.12y = 1{,}000 \text{ (kg. of Fe)}$$

$$0.005x + 0.42y = \frac{2 \times 10}{0.94} = 21.3 \text{ (kg. of Mn)}$$

$$x = 1{,}660 \text{ kg.} \quad (98\%) \qquad y = 31 \text{ kg.} \quad (2\%)$$

For the 6% loss noted under (*c*), one has: $(x + y) = 0.94 = 1{,}590$ kg. of mixture which contains:

	Kg.
Fe	940.00
SiO_2	80.38
Al_2O_3	39.85
Mn	20.40
CaO+MgO	11.19
P	5.46
S	4.71

Assuming that P of the ashes represents a negligible value, an example of pig iron production with a low silicon content may be used:

$$C = 3.8\%;\ Si = 0.5\%;\ Mn = 1.0\%;$$
$$P = 0.54\%;\ S = 0.02\%$$

The slag produced by the minerals will be:

	Kg.
$SiO_2 = 80.38 - 5 \times 60$	$= 69.67$
Al_2O_3	$= 39.85$
CaO+MgO	$= 11.19$
$Mn = 50\%$ of $20.40 \times \frac{71}{55}$	$= 13.15$
S (admitting total desulfurization)	$= 4.71$
TOTAL	138.57

Taking an index of 1.2 for the ratio $\frac{CaO + MgO}{SiO_2}$ and calling k the amount of lime (CaO) required for slagging the gangue of the ores, the following equation would result, within the percentages contained in the limestone type indicated earlier:

$$\frac{0.96k + 11.19}{0.017k + 69.67} = 1.2$$

from which k = 77 kg.; so that the total amount of slag by ores is:

$$139 + 77 = 216 \text{ kg.}$$

Assuming the total Fe content as Fe_2O_3 and admitting that 1/6 of the C required for the reduction has a dual effect (according to Professor Durrer), the following amounts of carbon may be calculated for the different elements:

		Kg.
Fe	$= 940 \times \frac{6}{7} \times \frac{36}{112} =$	260
Si	$= 5 \times \frac{24}{28} =$	4.30
Mn	$= 12 \times \frac{12}{55} =$	2.62
P	$= 5.46 \times \frac{60}{12} =$	5.29
S	$= 4.71 \times \frac{12}{32} =$	1.75
	TOTAL	273.97 kg. of carbon

Adding the C corresponding to the carburation of the pig iron, and subtracting that delivered by the electrodes,

$$274 + 38 - 10 = 302 \text{ kg. of C}$$

which is equivalent to:

$$\frac{302}{0.31} = 373 \text{ kg. of anthracite}$$

The slag produced by the anthracite ash will be:

$373 \times 0.1 = 37.3$ kg., the weight of the ash, + 18.7 kg., weight of the lime necessary, = 56.0 kg. of slag.

Then, the lime for desulfurizing should comprise the 4.71 kg. of S contained in the ores and 2.981 of the same element which are present in the anthracite. Therefore, desulfurization requires:

$$7.694 \times \frac{56}{32} = 13.45 \text{ kg. CaO (14 kg. of lime)}$$

producing an additional slag of

$$7.694 + (14 - 13.45) + \frac{40}{36} = 23.19 \text{ kg.}$$

Consequently, the charge would be made up of:

	Kg.
Iron ore	1,660
Manganese ore	31
	1,691
Lime (CaO): 77+19+14 =	110
Anthracite	373
Slag produced: 216+56+23:	295

Once the charge is known, the analysis of the elements for the thermic balance may be started. First, the volume must be calculated and the composition of the typical resultant gases must be assumed.

The anthracite contains:

$$373 \times 0.81 = 302 \text{ kg. of C}$$

Adding to this weight of carbon the 10 kg. contributed by the electrodes and subtracting the 38 kg., for the carburation of the pig iron, the C which can be used is: 274 kg. of C.

As it had been calculated above that the C required for the reduction of Fe_2O_3 is 260 kg., and once it is admitted that 1/6 of this weight of C has a dual reaction, producing CO_2, the volume of the gases becomes:

$$274 - \frac{260}{6} = 231 \text{ kg. of C}$$

which produce:

$$231 \times \frac{22.4}{12} = 431 \text{ m}^3 \text{ of CO}$$

$$260 \times \frac{22.4}{12} \times \frac{1}{6} = 81 \text{ m}^3 \text{ of } CO_2$$

As the anthracite contains 3% of volatile matter, according to the supposed composition, it may be assumed that it contributes the following volumes of gases:

$$\begin{aligned} H_2 &= 23\ m^3 \\ N_2 &= 8\ m^3 \\ C_mH_n &= 4\ m^3 \\ \hline &\quad 35\ m^3 \end{aligned}$$

The total volume of the gases will be 547 m^3 with the following percentages of contents:

$$\begin{aligned} CO &= 79\% \\ CO_2 &= 15\% \\ H_2 &= 4\% \\ N_2 &= 1.5\% \\ C_mH_n &= 0.5\% \end{aligned}$$

Considering heating values of:

$$\begin{aligned} CO &= 3{,}000\ \text{calories}/m^3 \\ H_2 &= 2{,}500\ \text{"}\ \text{"} \\ C_mH_n &= 8{,}500\ \text{"}\ \text{"} \end{aligned}$$

the total volume of gases, per ton of iron produced, amounts to:

$$547\ (0.79 \times 3{,}000 + 0.04 \times 2{,}500 + 0.005 \times 8{,}500) = 1{,}372{,}000\ \text{calories.}$$

To determine the real heat of the gases upon escaping from the furnace, let us estimate a temperature of 400°C with 0.33 calories per cu. m. per degree centigrade. This gives:

$$547 \times 400 \times 0.33 = 72{,}000\ \text{calories}$$

The heat consumed in the different reductions is given by:

$$\begin{aligned} 940\ \text{kg. of Fe} &= 940 \times 1{,}740 = 1{,}635{,}600 \\ 12\ \text{" " Mn} &= 12 \times 1{,}654 = 20{,}000 \\ 5\ \text{" " Si} &= 5 \times 7{,}000 = 35{,}000 \\ 5.46\ \text{" " P} &= 5.46 \times 5{,}887 = 32{,}000 \\ \hline \text{Total heat reductions} &\ \ 1{,}722{,}800\ \text{calories} \end{aligned}$$

The heat consumed in the desulfurization of the total S contained in the ores and in the anthracite (7,691 kg.) is given by:

$$\frac{7{,}691}{32} \times 61{,}000 = 14{,}650\ \text{calories}$$

The drying of the moisture content in the ores and anthracite comprises:

	Kg.
Ores	0.414
Anthracite	19.396
TOTAL	19.810 H_2O

In order to vaporize and superheat a kilogram of H_2O at 400°C, the following is required:

$$637 + 0.48 \times 300 = 781\ \text{calories.}$$

Then the moisture in the charge absorbs:

$$781 \times 19.81 = 15{,}470\ \text{calories.}$$

Summing up:

	Calories
Heat consumed in reduction and desulfurization	1,737,450
Sensible heat of the gases	72,000
Drying of the charge	15,470
Melting of iron (300×1,000 kg.)	3,000,000
Slag melting (500×295)	147,500
15% loss through radiation and cooling	340,863
TOTAL CALORIES	2,613,283

The heat contribution due to the formation of CO and CO_2 in the thermic balance should be deducted from the total amount of calories required.

431 cu. m. of CO develop:

$$\frac{431}{22.4} \times 29{,}000 = 558{,}000\ \text{calories}$$

81 cu. m. of CO_2 develop:

$$\frac{81}{22.4} \times 97{,}000 = 350{,}000\ \text{calories}$$

Total heat of formation = 908,000 calories

The difference, which should be contributed by electric energy, is:

$$2{,}613{,}283 - 908{,}000 = 1{,}705{,}283\ \text{calories}$$

equivalent to:

$$\frac{1{,}705{,}283}{860} = 2{,}000\ \text{kWh}$$

Power consumption in actual practice

Although these calculations are theoretical, nevertheless power consumption is primarily adjusted to compensate the thermic balance analysed above. In actual practice, other factors must also be taken into account, such as the resistivity of the charge. This does not depend solely on the size ratios of the charge materials, but rather each material has its own crystallographic characteristics, giving rise to variations in its conductivity.

There are also considerations of a physical-chemical nature, particularly in so far as the reactivity of the reducing agent is concerned, as it requires a greater or lesser power consumption, according to its capacity to release the carbon for the reduction reactions. Thus, anthracite does not, usually, have the high reactivity of coke or charcoal. It has been observed experimentally that in blends of anthracite with coke breeze, the greater the proportion of anthracite, the greater is the consumption of power.

Under normal conditions the consumption of power per ton of pig iron produced ranges from 2,400 to 2,600 kWh, often falling to only 2,200 kWh/ton. Nevertheless, it is advisable to take the figure of 2,700 kWh/ton, when estimating production costs, mainly when poorly reactive anthracite has to be partially used. The above consumption figures refer to the production of pig iron with up to 1% silicon. For each additional 1% silicon, 100 kWh should be added.

Economic considerations

Assuming an operation using 2,700 kWh per 1,000 kg. of pig iron, the cost of this power consumption in the electric furnace should be compared with that of the coke used in the blast furnace for the same purpose. Formerly it was calculated that 373 kg. of anthracite, or even coke, were required for the reduction of a charge producing a ton of pig iron. In round figures, therefore, the ratio between the blast furnace and the electric unit is 1:0.4 in the consumption of coke. Thus, the 2,700 kWh should be compared economically with the value of 600 kg. of coke, which is equivalent to assuming:

1 kg. of coke = 4.5 kWh (cost-wise)

Nevertheless, this is only a reference and, when applied, it should be considered that the coke breeze or anthracite used in the electric furnace is cheaper than the fuel required by the blast furnace. Also the flux requirements are lower in the electric furnace, which results in fewer impurities contained in the charge, and permits the production of a better quality iron.

The high temperature within the fusion zone permits the electric furnace to operate with more basic slags and to achieve excellent desulfurization.

The electric furnace, typical of the larger plants currently in operation, produces up to 100 tons in 24 hours. These units are of 13,200 kVA and they operate with a power factor of 0.85. Other furnaces are designed to produce 200 tons/day, with 20,000 kW.

It is evident that electric reduction furnaces can be used successfully from an economic point of view, when the production capacity is limited, and when other factors are present to make possible the production of pig iron which is otherwise metallurgically or economically unfeasible.

An abundant supply of cheap hydro-electric power is then essential. The ceiling price for power payable by an electric furnace installation depends on its whole economic operating factors, but usually should not exceed US$0.005/kWh.

The electric solution should basically be related with the coal problem. That is, where the available coal is non-coking, or when it does not yield a good metallurgical coke, the electric unit constitutes the feasible furnace for producing pig iron.

The electric solution should be adopted when, with cheap power available, a limited installation at a low investment cost is desired, in order to produce iron with special and varying properties. Hence the importance, for some steelmaking regions in Latin America, of using electric furnaces in their installations.

With the immense hydro-electric power potentialities existing particularly in South America, the use of the low-shaft electric furnace can be visualized as the solution in some territories, or at least the pilot plant solution, permitting economic production of pig iron and steel to be commenced, and possibly to expand it as necessity dictates.

Electric Smelting of Pig Iron

M. Ø. SEM

INTRODUCTION

The electric smelting of iron ore for the production of pig iron is an industry long since established in Scandinavian countries. The old Elektrometall furnace dates back to 1910 in Sweden. It uses charcoal as a reducing agent and gives good service provided that the ore is fit for the particular operation. In the 1920's the Tysland-Hole furnace was developed in Norway for the use of Söderberg electrodes and with coke as a reducing agent. It soon spread to other Scandinavian countries and took the place of most Elektrometall furnaces. At the present time some 42 electric pig iron furnaces of various types using coke have been built in a large number of countries. A total of about 360,000 kW are developed in these furnaces, which means that more than one million tons of pig iron yearly may be produced in electric furnaces. The table below lists these furnaces and is based on the information available to the writer at the time of preparing this paper.

FURNACE

Country	*Type*	*No.*	*Kw. rating*	*Reductant*	*Remarks*
Sweden	Elektrometall	2	5,000	Charcoal	
"	Tysland-Hole	4	10,000	Coke	
"	" "	2	6,000	"	
Norway	" "	1	8,000	"	
"	?	1	10,000	"	Originally by Siemens changed as much as possible to T-H type.
"	" "	3	16–20,000	"	Under erection.
Finland	" "	1	10,000	"	
Italy	" "	7	6,000	"	
"	" "	4	10,000	"	
"	" "	2	6,000	"	
"	Siemens	3	10,000	Anthracite	
Switzerland	Tysland-Hole	1	8,000	"	
Yugoslavia	Siemens Demag	1	10,000	Coke	Under erection.
"	Tysland-Hole	3	10,000	"	" "
Japan	" "	2	6,000	"	
India	" "	2	10,000		Now starting.
Peru	" "	2	10,000		Under erection.
	TOTAL	42	360,000		

The furnaces, as will be seen from the table, are mostly of the 6,000 to 10,000 kW type, producing some 50-100 t/day. Three larger furnaces of 16,000 to 20,000 kW each are now being installed in a governmental steel plant at Mo in Norway. In this plant a total of about 180,000 tons of pig iron per year is expected to

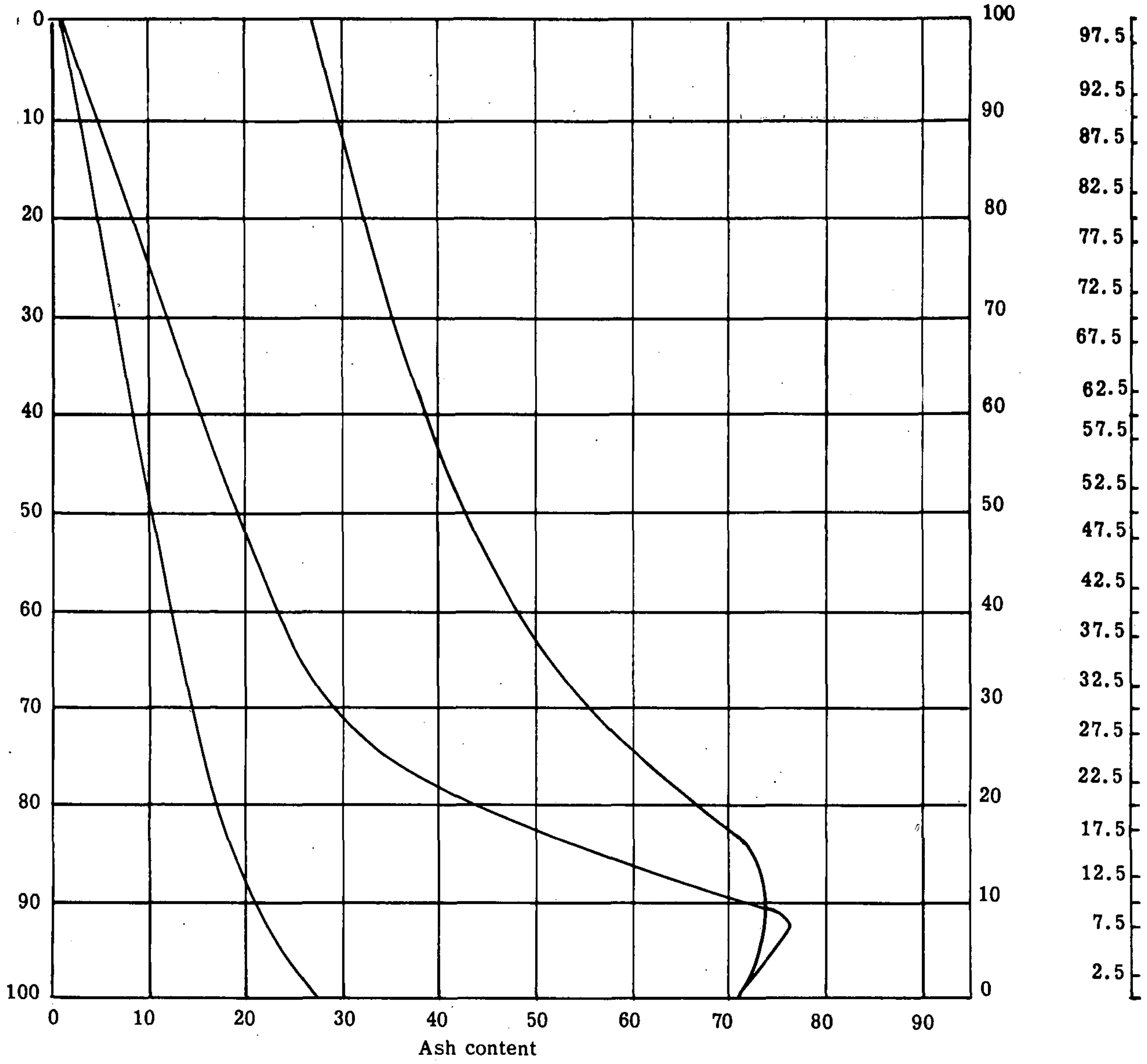
A

WASHABILITY CURVE AGAINST ASH CONTENT

-- *Elementary characteristic*

-- *Characteristic of light products*

-- *Characteristic of heavy products*



be produced, solely dependent on the electric smelting process (1). South American readers may also be interested in an electric pig iron plant consisting of two 10,000 kW Tysland-Hole furnaces now being installed in India by the Mysore Iron and Steel Works. Also, the Corporación Peruana del Santa in Peru is being provided with these furnaces. In both cases steel plants and rolling mills will depend on the pig iron furnaces for a major part of their raw materials.

RESULTS OBTAINED

There are several publications which describe the results obtained in electric pig iron furnaces (2, 3). Under average conditions the results will vary with the quality of the raw materials and local conditions approximately as follows:

	Per metric ton pig iron
Slag produced, Kg.	350–700, or more
Gas produced, Nm^3	650–700 (23–25,000 cu. ft.)
Ore, kg.	1,500–1,800 (3,300–3,740 lb.)
Limestone, kg.	200–400 (660–880 lb.)
Reducing agent, kg. (coke, coke-breeze, anthracite, charcoal, etc., 10–15% ash)	375–450 (825–992 lb.)
Electric power, kWh	2,300–2,700
Electrodes, kg.	8–25 (17–33 lb.)

These results apply to pig iron containing ½% to 1½% Si. If foundry pig iron with 2-3% Si is produced, the kWh consumption will increase to about 2,600 to 3,000 kWh per metric ton.

POWER CONSUMPTION REDUCED

The electric pig iron furnaces so far have been operating on a charge which has not been dried nor is it of uniform physical composition. On the contrary, the coke often contains 8-15% of moisture and the grain size of ore and coke has varied a lot. Fines (<¼ in.) have often amounted to 30-40% or more. In some cases the electric furnace has been charged to whatever the blast furnace cannot accept (4). Professor Kalling recently published a report about the improvements obtained in the Tysland-Hole furnaces at Domnarfvet. Domnarfvet is the best known Swedish pig iron and steel plant. The company has been using a sintered charge with most of the lime to be added as a flux having been sintered into the ore. Furthermore, the raw materials are selected so that the screen analysis of the charge is nearly constant. The power consumption in the Tysland-Hole furnaces has thereby been reduced to about 2,000 kWh per metric ton on a yearly basis. These are the results reported by Mr. Kalling as being average results:

PER METRIC TON OF PIG IRON (1% Si, 0.02% S)

Electric power		2,050 kwh
Coke breeze	115 kg.	
Pearl coke	165 kg.	
Blast furnace coke	105 kg.	385 kg.
Electrode consumption paste		15.5 kg.
Quantity of gas of 2,200 Cal./cu.m.		450 cu.m.

This is important news to all producers of electric pig iron. However, it seems particularly important to those using concentrates making it indispensable that the iron ore be sintered or agglomerated in some way. It is also important to ore-exporting countries where ore fines are available but do not seem fit for export. These fines, now screened off but not sufficiently utilized, may be used to advantage in electric smelting as explained above.

The above is an example showing that it is possible to reduce the power consumption. In general the following steps may be adopted for this purpose:

(1) *Using dried coke*

The most important drawback to the content of moisture in the charge is that it always varies considerably. The composition of the charge has, therefore, to be changed accordingly. Furthermore, it may cause occasional eruptions which disturb the operation. Even though a good deal of the moisture is dried out in the upper part of the furnace without causing trouble it seems fair to add approximately 100 kWh/t to the power consumption for each 100 kg. of moisture.

(2) *Use of burned lime*

Most iron ore needs an addition of limestone or similar basic material (dolomite) as a flux. If the CO_2 content must be driven off in the smelting furnace it will consume approximately 80-100 kWh for each 100 kg. added per ton of pig iron. An addition of 200 kg. of limestone will consume some 160-200 kWh/t pig iron. In one case where 500 kg. were added the saving would amount to 400 kWh/t.

This may be saved by adding burnt lime (dolomite) to the charge. It has the disadvantage, however, that the lime will easily be slacked by any moisture content in the coke or ore. In many cases it is, therefore, preferable to add the basic flux as a ferrite. This may be accomplished by sintering part of the ore (the fines or part of them) with limestone in a relation of $CaO:Fe_2O_3 = 2:1$.

A very good sinter which can stand much handling and storing can be produced that way. If the iron ore should be sintered anyhow (iron ore concentrates) it may be preferable to mix the limestone with all the ore. In this way it is possible under favourable conditions to sinter as much as about 8% or more CaO into the charge. Also this sinter will stand handling and storage.

On the basis of the experience obtained in the many electric pig iron furnaces now operating under very varying conditions, it is easy to calculate the production costs of electric pig iron under the various conditions prevailing in the different countries.

On this information a comparison can be made with the blast furnace process. It is more difficult to make a comparison with the new processes considered for direct reduction (sponge iron) or with the low-shaft blast furnace operating with or without oxygen-enriched air, now being tried out in Europe. These new processes are still in an experimental stage as far as production of commercial iron and steel is concerned.

The basis of the electric pig iron furnace is the availability of relatively cheap electric power. Dr. Durrer has dealt with this subject in various papers, arriving at the conclusion that the electric smelting furnace would be feasible in cases when 1 kg. of coke costs more than about 6 kWh of electricity (5). This is based on the assumption that the electricity used in the electric furnace makes up for about 350-400 kg. of coke in the blast furnace. This may be useful as an indication. However, Dr. Durrer is fully aware of the fact that many other factors enter into the picture, such as the cost of installation, etc. More important, in most cases, is the quality of the coke or coal available in the country in question. The blast furnace must have a strong coke which can carry the heavy load from the high column of charge existing in blast furnaces. Furthermore, the charge must be sufficiently porous to allow the blast to pass through the furnace without too high pressure. In comparison, the electric furnace can use nearly any kind of coke of reasonable grain size. In this furnace (perhaps particularly so in the Tysland-Hole furnace) the gas is collected around the electrodes without passing through the shaft at all. Consequently both coke and ore may contain considerable quantities of fines which could not be allowed in a blast furnace. It is a commercial practice to add about one-third to one-half of the coke as coke breeze. Other reducing agents such as anthracite and charcoal are also used to advantage. Lately, an interesting test has been carried out on a commercial scale in one of the 10,000 kWh Tysland-Hole furnaces at Fiat, Turin (Italy), dealing with the use of lignite as a reducing agent. A detailed report on this test has been published by Mr. E. Sjögren (6). It will be noted from this publication that Austrian lignite was used to advantage, producing approximately 1,000 tons of commercial pig iron. This fact may prove interesting to some South American countries where coke is scarce but where lignite is found in quantity. This seems to be the case in Venezuela where they have no good coal but large quantities of lignite.

FUTURE POSSIBILITIES

A typical gas produced in the electric pig iron furnace is as follows:

CO_2	15% by volume
CO	72% by volume
C_mH_n	2% by volume
H_2	8% by volume
N_2	2% by volume

About 650 cu. m. (23,000 cu. ft.) of this gas is produced for each metric ton of pig iron (0°C, 760 mm. Hg dry). The gas averages 2,500 Cal./metric cube (290 BTU per cubic foot). In comparison the caloric value of the ordinary blast furnace gas is only about 900 calories per metric cube (100 BTU per cu. ft.).

One electric pig iron furnace of about 10,000 kW actually produces a gas which corresponds approximately, in volume and heat value, to a gas works of a fairly large town. So far, this gas has been put to different uses for heat purposes in various electric pig iron plants. The value of this should be credited to the pig iron operation. Various users have not fully realized that the electric pig iron furnace may be considered as a good gas producer. If there is sufficient need for a good gas in the plant, the credit to the electric smelting process should be sufficient to pay for an important part of the reducing material consumed in the furnace. Actually, the heating value of the gas corresponds theoretically to some 65% (about 1,700 kWh) of the electric heat developed in the electric pig iron furnace. It does not seem to be too optimistic to expect a gas from these furnaces to be used to considerable advantage for chemical synthetics, production of hydrogen, for sintering of ore fines, production of sponge iron or similar purposes.

PREHEATING AND PRE-REDUCTION OF THE CHARGE

If the plant cannot use the gas to advantage, for other purposes, this may, of course, be used to preheat the charge before filling it into the electric furnace whereby considerable saving may be obtained in regard to the consumption of electric power. Such preheating is already in use in certain electric smelting furnaces on a commercial scale. Tests are also being carried out at the present time in connexion with electric pig iron furnaces. A calculation shows that it should be possible by proper pre-heating (to about 600°C) to effect a saving in power consumption equal to approximately 250 kWh per metric ton.

Because of the high content of CO in the gas, it can be used for reduction purposes before being finally burned. Professor M. Wiberg is well known for his contribution to the process named after him for the production of sponge iron. As this process will be dealt with in other papers at this meeting, no details will be given here. Wiberg, however, in co-operation with Elektrokemisk, Oslo, patented a process dealing with the use of electric furnace gas for pre-reduction of the iron ore to be smelted in the electric pig iron furnace. According to Wiberg's calculations, a reduction in power consumption corresponding to approximately 300–400 kWh per metric ton of pig iron should be obtained by using this process. Accordingly, it seems possible when utilizing the gas to a full extent under favourable conditions to arrive at a figure of something like 1,500 kWh per metric ton of pig iron produced, depending upon the amount of slag formed and the nature of the ore.

The pre-reduction process is subject to tests taking place at Norbottens Jernverk in Sweden. Some 4,000 tons of electric pig iron have been produced in this way and it seems that Mr. Wiberg's calculations have been fully sustained by the results obtained. For various reasons the tests have unfortunately not yet been brought to a conclusion as the pre-reduction so far has not been carried beyond the FeO stage.

THE ROTATING HEARTH FURNACE

The Elkem rotating hearth furnace (7) has awakened much interest in the ferro-alloy and carbide industry. Tennessee Valley Authority (T.V.A.) has tried this furnace to advantage for the production of phosphorus (8). A total of some 22 furnaces ranging up to 39,000 kVA each are already in operation or in course of erection.

These furnaces have proved to be less sensitive to fines than a stationary furnace. The installation costs are approximately 8% higher than for a stationary furnace but this may easily be saved through cheaper operation, less wear on the furnace bottom as well as side walls, etc. An 18,000 kVA rotating hearth furnace for smelting of pig iron is just being developed. Figure 1 shows a model of this furnace.

These figures are given to show that the cost of electric power per ton of pig iron may not be so all-important to the economy of electric pig iron smelting as is normally considered. On the contrary, it may be said today that other charges such as depreciation of installation costs, etc., may in the long run prove to be of considerably higher importance. So far, the capacity of electric smelting furnaces has been small compared with that of the blast furnace. Even this is now rapidly changing. The big furnaces at Mo in Norway will produce some 160-200 metric tons daily. The furnaces are equipped with Söderberg electrodes of about 5′ Ø. These electrodes can now be made so reliable that a 6-electrode furnace could be made for a total production of approximately 300-400 tons/day.

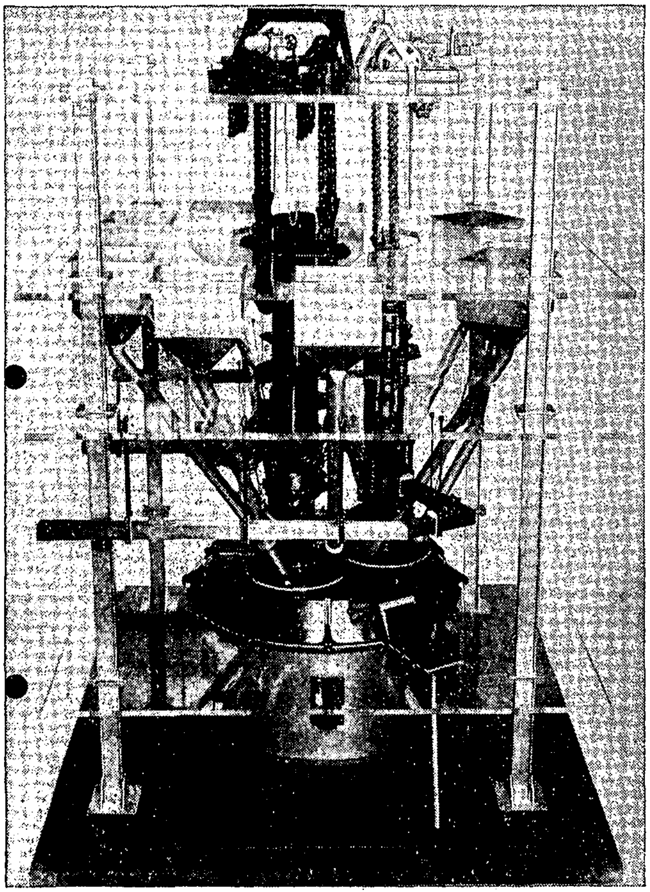


Figure 1
The Elkem rotating hearth furnace.

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Making of Steel in Electric Furnaces

H. S. NEWHALL

Looking at Latin America's industrial development—particularly in the field of the iron and steel industry—from an engineering standpoint, it is difficult to make statements applicable to conditions prevailing in all the Latin-American countries. It is in fact necessary to make a complete study of all the local factors involved in any area where it is planned to erect an iron and steel mill.

What is the best process to be used under any specific conditions? In countries having abundant coking coals, the first thought is the blast furnace. In those places where inexpensive power is available and where there is a lack of coking coal, the electric furnace comes to mind, and in still other localities it is quite likely that one of the gaseous reduction or sponge iron operations might be more applicable.

The blast furnace uses around 1,800 lb. of coke per ton of pig iron, while the electric furnace (operating with coke) uses 900 lb. per ton. For a quick and approximate analysis, the 900 lb. used (for heating) in the blast furnace in excess over the coke consumed in the electric furnace may be compared with the latter's consumption of power and electrodes. Assuming the cost of power to be US$.005/kWh and consumption to be 2,000 kWh/net ton, the corresponding cost per ton will be US$10. Assuming that electrodes cost 15 cents a lb., and that 25 lb. are used per ton of pig iron, the result is US$3.75, making a total of US$13.75 on coke and electrodes per ton of pig iron.

The paramount question then raised is whether it is possible in a particular locality to obtain good grade metallurgical coke for this price? In any event the electric furnace does not require metallurgical coke. A number of the coals throughout South America can be charred and their high ash content fluxed suitably for use in an electric furnace.

Assuming that coke costs US$30 per net ton, 1,800 lb. for the blast furnace would amount to US$27. Assuming that, in the electric furnace, instead of 900 lb. of coke, 1,200 lb. of coal at US$12 a ton are used, the cost would be US$7.20. Taking credit for the tars and adding about US$1 for the pre-treatment, this would give the cost of the reducer. Adding power at 0.5 cents and electrodes at 15 cents, the result is US$21.95, as compared with US$27 for the blast furnace. Other factors to consider are the higher investment and operating costs of the blast furnace (in the 200 ton a day capacity range) as compared with the costs of the electric furnace.

With the larger daily output now under consideration, the most efficient electric furnace is the six-electrode furnace.

The Lectromelt Furnace Corporation has felt that the manufacture of pig iron can be carried out more economically in a six-electrode-in-line furnace, when considering units from 10,000 kVA upwards. An in-line furnace, where a slag bath is used, will give more uniform distribution of heat throughout the furnace. This results in more efficient power consumption and lower temperature of the fumes. Roof refractories have a longer life and a 5 to 10% reduction in kWh consumption is anticipated in the manufacture of pig iron.

In the design of electric furnaces there is one very important item, and that is the power factor. While it is possible to put in capacitors to overcome a low power factor, there is no reason to go to the added expense of capacitors, unless it is necessary. With a six-electrode-in-line furnace, three transformers are connected to the primary 3-phase circuit in delta; each transformer in turn is connected to two electrodes. The plus and minus of the transformers are interleaved to a point close to the electrode where the cables are connected in a parallel formation in order to minimize losses. In other words, there are really three single-phase furnaces in one furnace body with the three single-phase furnaces connected into a 3-phase circuit. The result of this design is a minimum length of bus bar, and maximum interleaving, resulting in a high power factor.

The design of the six-in-line furnace can very easily embody changes of spacing. When operating on various types of ores it is sometimes advantageous to change the spacing between the electrodes in order to obtain the most effective operation. One of the many things to control this is the particle size, and the minimum spacing applicable is, generally speaking, the most efficient. With a furnace thus designed, it is very simple to slide the electrode longitudinally so as to obtain the best operating conditions at the time. Designs are embodied today in such a way that spacing can be changed without shutting down the furnace.

When designing the furnace, it is not, of course, possible to know what type of ore will be used in that furnace ten to twenty years from the date of construction. Crushed lump ore or sinter may be available which allows the furnace to be filled completely with sufficient porosity in that charge to allow for the evolution of the developed gases. However, in the world today there is increasing concern regarding the use of fine ores. As the better ores in the world are used up fine ores must be used. Though they can be sintered it would be a great advantage to be able to smelt them directly. Under these circumstances the six-electrode-in-line furnace can be used by employing a "V" type charging instead of the "M" type, which is the conventional type. In the "M" type charging, the charge is inserted somewhere between the electrodes and the wall and owing to the angle of incline of the ore will form pyramids at the charge ports which result in the resemblance to the letter "M". When using fine ore it is, of course, necessary to reduce the overburden to a point where the gases can escape without causing crusting and blowing. In the oblong furnace this can be carried out by charging down the side walls so that the bottom of the "V" meets at the electrodes. This banks the walls to protect them and yet allows the centre of the reaction zone to have a low overburden so that gas can freely escape from the reaction. A furnace can be designed with charge ports in such a way that both of these types of operation can be carried out in the furnace from time to time merely by changing from one set of charge ports to another. If a very fine charge is to be used of the order of 100-mesh, then it is possible to charge down the banks, raise the voltage and operate with an actual arc condition along the side, allowing the flash of the arc to smelt the charge on the banks. It is admitted, under these conditions, that the power consumption will increase due to the high temperature gases. However, the power consumption increases only 3 to 4%. It is anticipated that this would be advantageous economically as against normal procedure.

In designing a mill where a battery of furnaces is to be installed, a six-electrode-in-line furnace occupies considerably less space. It is very simple to line the furnaces up side by side using the common slag bays and common transformer room so that as much as one-third of the space can be eliminated, thus reducing capital investment.

Concerning the charging of this furnace, a very simple mechanical arrangement can be installed—either two monorails, or bucket charging, or two scrapers down the top of the furnace. This arrangement is very simple and easy to operate and is considerably less costly than some of the complicated designs necessary to cover completely a circular furnace. In the design of electrode clamps, for example, the six electrode furnace might have 30 in. electrodes and the three-electrode furnace 40 in. electrodes. In general, going over the cost on this type of equipment, a 30 in. clamp is approximately one-half the cost of a 40 in. clamp and, therefore, one can supply six-electrode superstructures just as inexpensively as three-electrode superstructures for higher amperage with the same kW conditions.

The reaction areas around the six electrodes are smaller than those of the three electrodes in triangle having the same power input and therefore the furnace body is more expensive than that of the standard circular furnace.

With individual tap changers on each transformer it is possible to change the local conditions under any pair of electrodes. For example, when making a lower carbon iron which, of course, requires higher temperatures for tapping, it is possible to change the voltage conditions under one pair of electrodes which is close to the metal tap hole and put the heat from the electrodes deep in the furnace in order to heat the metal more than the slag. The same idea can also be applied to the slag tapping. Voltage at the electrodes at the slag tap hole can be adjusted for the local conditions required at that tap hole. In other words, each pair of electrodes can have its own local conditions depending upon the conditions desired in that area of the furnace.

Great flexibility is available with the six-electrode furnace, even when damage occurs to the clamps, which does not necessarily force a shutdown of the furnace. By arrangement with the power company, four of the electrodes can be kept in operation while the other two are repaired.

The next stage is the making of steel using the pig iron made in the electric reduction furnace. A few years ago the open hearth operators stated that good rimming steel could not be made by electric refining; furthermore that hot metal could not be taken to the electric furnace and that the open hearth should necessarily be used. Today, all those impossibilities are being performed and more plants are under construction with this process in South America. Some of the Latin-American countries are considering a charge of hot metal as high as 60 to 80%, the only scrap being used consisting of circulating scrap found around the plant or coming from the rolling mills. In some of these countries there is not much scrap available.

An important part has been played by the use of oxygen and ore as agents for reducing the metal's carbon content. The use of iron oxide as a decarburizer for the higher carbon range and the completion of the operation with ore or oxygen has made it possible to use the electrically powered furnaces in this manner.

Further development is continuing to determine the time schedules and sequences of ore additions as well as those of the oxygen required. One of the important items is that there are no iron losses as in the case of the converter; there is in fact an actual gain in iron content, so that there is finally more steel than pig iron, whereas, with the converter, allowance has always to be made for about 10 to 15% losses.

Some years are required for the development of this type of research in the field of iron and steel electric manufacture and the future will undoubtedly show a different operation from that of today since, by its very nature, progress demands changes.

Summary of Discussion

At the invitation of the Chairman, Mr. Waring, Chief of the Industry Divisions of the Economic Commission for Europe, took a place at the speakers' table.

Progress in the Manufacture of Pig Iron and Ferro-alloys with the Low-Shaft Electric Furnace, *presented by the author*

Mr. NEWHALL opened the general discussion of the paper with a reference to the author's comment on the circular 3-electrode furnace as being a more uniform heat furnace than the 6-electrode-in-line furnace, adding that he was in disagreement with that view.

Mr. WALDE replied that he thought the point could be taken up later on.

The CHAIRMAN asked Mr. Walde to inform him as to how much iron was produced in Germany by the method described, and how many such furnaces were in operation throughout the world.

Mr. WALDE, in answer to the first question, said that no iron was produced in Germany by that method, as coal was too cheap. With regard to the second question, there were many such furnaces throughout the world, and there was a review of the situation in another paper to be presented that afternoon.

The Electric Reduction Furnace, *presented by the author*

Mr. BALLÓN prefaced his presentation by explaining that, although he had been one of the most sceptical Latin-American engineers as far as the use of the electric pig iron furnace was concerned, he had, after visiting several plants, and after studying some of the factors involved, become one of the defenders of the process, because he believed that under certain circumstances it was the proper solution, particularly for the production of pig iron in countries such as his own (Peru).

Electric Smelting of Pig Iron, *presented by the author*

The CHAIRMAN, in declaring open the discussion on the two papers, observed that it seemed as if the electric low-shaft furnace had a very limited application, being feasible only for those countries having cheap hydro-electric power and no coke.

Mr. BALLÓN confirmed that that appeared to be the case, but that there were very few Latin-American countries without potential hydro-electric development.

Mr. WARING pointed out that whereas in Mr. Ballón's paper there was a reference to a kilogramme of coke being the equivalent of 4.5 kWh, Mr Sem had referred to a kilogramme of coke as being equivalent to 6 kWh. In view of the considerable difference between the two figures, he would like clarification thereof.

Mr. SEM replied that the figure 1:6 was a formula worked out by Mr. Durrer, who would, he thought, agree that it was only an average figure, and that the exact one would have to be worked out in each particular case. A great many factors were subject to change, particularly the question of the price of raw materials, and he could think of conditions under which electric smelting would have to be used notwithstanding the cost of electric power; in a great many cases, however, there was no doubt that the figure 1:6 would require considerable modification.

Mr. BALLÓN said that his figure of 4.5 kWh as compared with 1 kg. of coke as a reducing agent was based on the carefully calculated figure of 373 kg. of reducing agent per ton of pig iron produced, which had been rounded to 0.4 tons. It was normal to calculate one ton of coke per ton of pig iron in the blast furnace, so that the ratio of 1:4 was, in round figures, quite correct.

Mr. Saniter said that Mr. Sem seemed to be quite embarrassed by the amount of carbon monoxide produced, but that some years ago he himself had worked out a scheme for a half-million ton steel plant based on the electric smelting process. The ore, while pure, contained some phosphorus, and therefore the basic open hearth had been considered to be the best conversion process. The amount of gas derived from the smelters almost exactly met the requirements of the open hearth furnace, so that a well-balanced integrated unit could be built under such conditions.

Mr. Pyles Lozano requested the following information on the efficiency of the electric furnace:

(1) Mr. Walde had spoken of 92% efficiency for that furnace and some years ago it was believed that the larger the furnace, the lower would be its efficiency and the frequency of the current. Mr. Sem had spoken of 18,000 kVA furnaces, but had not mentioned the frequency of the current. He wondered therefore whether a lower frequency should be used to improve the efficiency. Brazilian current was 50 to 60 cycles per second;

(2) With regard to pre-heating and pre-reducing the charge, he inquired whether or not the tests had been discontinued;

(3) He would like to know what Mr. Sem's experience had been in using charcoal or a mixture of charcoal and coke as the reducing agent, from the standpoint of electricity consumption;

(4) He wondered what Mr. Sem's experience had been with coke having a very high ash content. Perhaps Messrs. Kalling or Durrer might also contribute their comments concerning high ash coke in electric furnaces. He understood that in Italy the furnaces operated with very high ash contents in the coke.

Mr. Sem gave the following replies:

(1) With regard to electrical efficiency, 50-cycle power was being used in the electric pig iron furnace. In Norway, when the furnaces were under construction, there had been considerable discussion as to the desirability of using 50 or 25 cycles. However, the electrical technicians had discovered that by using capacitors it was possible to build furnaces of 18,000 kVA without detriment to the power stations or to other consumers;

(2) There was no objection to using 60-cycle power for the same purposes as 50-cycle power had been used, but before making any definite statement, his firm preferred to wait until the big furnaces in Norway were in operation;

(3) On the question of operation with charcoal, there were one or two charcoal furnaces in operation in Sweden, and a mixture of charcoal and coke had been used to a considerable extent there, without any difficulty whatsoever;

(4) On the question of ash content, naturally there should be no more slag-building materials in the charge than necessary, but the influence of high ash content was difficult to analyse. He knew that Mr. Durrer had had interesting experiences in his operations with high ash anthracite;

(5) Pre-reduction tests were being carried out in Sweden at Norbottens Jernverk, about 4,000 tons of pig iron having been produced in that way.

Mr. Canguilhem, referring to the relationship between coke and electric power, suggested that there should be an alignment test, taking every independent variation into consideration; the figures mentioned were misleading as they did not explain clearly the relationship between the price of coke and that of electric power, and so forth.

Mr. Sem replied that there was a useful formula for preliminary investigations but that detailed calculations would have to be made in each case.

Mr. Ballón confirmed that the figure given was merely a general indication. Many charts could be made taking into account such factors as the price of coke and other materials, in order to compare electric pig iron furnaces with other methods, but no general calculation could be made because of the many other factors involved.

Peru had anthracite coal but no coking coal, so that, even if all the charts were in favour of the blast furnace, the lack of coking coal would still lead to the use of the electric pig iron furnace.

Mr. Lobo said that he had gathered from Messrs. Walde and Sem's remarks that most operations had been conducted at frequencies below 50 cycles, and he would like therefore to know the power factor reduction at 60 cycles.

Mr. Sem replied that there would be exactly double the index voltage at 60 cycles along the path of the current as there would be at 50 cycles.

Mr. Lobo remarked that, if the electric furnace method were selected for Venezuela, operations would evidently have to be effected at 60 cycles.

Mr. Ydstie stressed that the power factor was exactly proportionate to the cycle.

Making of Steel in Electric Furnaces, *presented by the author*

Mr. Sem opened the general discussion of the paper by pointing out that his firm, Elektrokemisk, had designed the arrangement for connecting the three monophase transformers with six electrodes in the case of the Petsamo furnace, as was borne out by an article in the *Engineering and Mining Journal* of August 1947. The furnaces were of the 12,000 kVA type and developed 9-10,000 kW. They used 50-cycle power, not 25-cycle.

Elektrokemisk had proposed that six electrodes be used, but arranged in two sets of triangles, although the first Westly furnace (the Petsamo furnaces being slag resistant or Westly furnaces) had six electrodes in one line.

There was an interesting reason for arranging the electrodes in line: the nickel companies had been accustomed to smelting their concentrates in reverberating furnaces. Those were long, rectangular furnaces where the metals were tapped at one end and the slag at the other. They wanted, in principle, to continue the same method of operation, and therefore Elektrokemisk arranged the electrodes in line in a reverberating furnace.

The furnace operated at a high power factor because only 1,660 kW was developed per electrode and the slag resistance was high. If the same power were developed per electrode then the arrangement would be no better, and in fact slightly worse, than the arrangement of the electrodes in triangle.

When the big pig iron furnaces currently being built in Norway were designed, six electrodes were considered. The company decided to use three electrodes in triangle, thus reducing electrode trouble to half. There was, however, a limit to the electrode diameter, although he was not prepared at that time to say just where the limit was. Six electrodes should be used in order to arrive at larger furnaces producing perhaps some 300 to 400 tons a day.

He disagreed with Mr. Newhall on the question of a superheated zone between the electrodes. The highest temperature was at the electrodes, and it would be a great advance if more heat could be developed between the electrodes. The charge did, however, penetrate between the electrodes, which was an advantage. In his opinion, the triangular arrangement was better both from the electrical and thermal standpoints, although the advantage of the in-line arrangement was that the charging mechanism and roof structure were simple.

The way to take advantage of high electrical resistance in the reducing agent was to increase the power input in the furnace so as not to force the electrodes down. The position of the electrodes could be regulated in other ways. Mr. Durrer's furnace had been increased from 6,000 kW to 8,500 kW by using an anthracite with a high ash content (30-35%).

The furnaces in Canada to which Mr. Newhall had referred were operated as arc furnaces. He did not believe that any successful pig iron furnace could be operated in that way.

Mr. NEWHALL replied that, notwithstanding Mr. Sem's remarks, he was still convinced that the in-line arrangement was best.

Mr. KALLING summed up the discussion by stating that the furnaces used in Domnarfvet were 3-in-line electrode furnaces, and he thought those had given the best results obtained to date. The kWh consumption averaged about 2,000 kWh per ton. There were other furnaces in Sweden, however, such as that at the Hofors steel plant belonging to SKF, which used electrodes in triangle, and where the latest results obtained appeared to be not very far from those achieved at Domnarfvet.

Mr. NEWHALL, referring to Mr. Kalling's last statement, said that the reports received by his company appeared to indicate the reverse.

Mr. KALLING replied that undoubtedly very good results were being obtained with both types of furnace, but that Domnarfvet, having become accustomed to the three electrodes in-line, had some predilection for that type of furnace, which gave good possibilities, especially in regulating the charging of the burden.

Mr. PRIETO asked for information as to the limiting factors thus far preventing a larger capacity for the electric furnace, that being a very important point when making comparisons with the blast furnace. He inquired whether the triangle, as compared with the in-line furnace, was more or less susceptible to increasing the capacity.

Mr. NEWHALL replied that, from the technical standpoint, 12 electrodes might be used if desired, as there were currently three single furnaces in one body.

The CHAIRMAN requested details on the size of the largest electric furnace in use.

Mr. NEWHALL replied that there was one of 30,000 kVA on phosphorus, and another of 35,000 kVA on steel, and various other furnaces ranging from 50,000 kVA upwards.

Mr. PRIETO repeated his question concerning the limiting factors.

Mr. NEWHALL replied that there were several, due to the fact that the industry was still learning.

Mr. PRIETO put a further question concerning the temperature of the electrodes and the existing limitations to the building of larger electrodes.

Mr. NEWHALL asked Mr. Sem to answer Mr. Prieto.

Mr. SEM thought that there was no practical limit to the size of the electrodes, as furnaces had been made with electrodes 2.5 m. in diameter, but so far five feet had been the limit for 3-phase furnaces. There was a limit which should not be overstepped, relating to electrical and thermal factors and to the design of the furnace.

There followed an interchange between Messrs. SEM and NEWHALL concerning the definition of the power factor.

Mr. DURRER said the information on electric furnaces had been very enlightening and he had learned a great deal and added to his experience in electric smelting. The fact was that electric smelting had become just as safe as smelting in the blast furnace. The question as to whether to use the blast furnace or the electric furnace was, in the first place, an economic one, but that afternoon a 1:6 ratio had been discussed. It had been based on the average commercial result of all electric smelting plants in existence at the time, and of course could never be a real figure for special locations. It was, as Mr. Sem had said, just an indication, and he thought that if calculations were made on the basis of those made many years ago, the average would be in favour of electric smelting. The ratio 1:4.5 might be correct for one place, but in making such calculations it was not merely a question of comparing electric energy and coke energy, but of taking into consideration all the different factors, as they would vary.

He pointed out that the blast furnace was some 600 years old as against the electric furnace's fifty years and there was, therefore, a great wealth of knowledge concerning the former. The great disadvantage of the electric furnace as against the ordinary blast furnace was one of capacity; furthermore, electric power was usually more expensive than the coal calory and that was why the electric furnace was not used more frequently.

Apart from the economic disadvantages, he thought the electric low-shaft furnace was almost ideal as the inherent purity of electric energy permitted greater margins for the selection of the other raw materials.

Mr. SHEROVER put a question to Messrs. Newhall, Sem and Durrer in relation to the situation in Venezuela, asking whether it would be better to use a proved method or to experiment with a relatively new one, especially from the economic point of view.

The CHAIRMAN thought that the problem involved a great deal of paper work and discussion and would, in the last resort, have to be solved by the country itself. Mr. Waring had, however, indicated that the choice was entirely dependent on the price of coke.

Mr. NEWHALL said that the calculations could be based on the cost of imported coke compared with the cheap power available, although, even under those conditions, a great deal of experimental work would probably have to be done.

IRON ORE REDUCTION

Methods for reducing ores otherwise than in blast furnaces (*continued*)

22 October 1952—Morning

Chairman:

Alfonso BALLÓN, Director del Departamento de Siderurgia, Corporación Peruana del Santa, Lima, Peru

Contributed Papers:

Considerations on the Development of the Production of Iron
Robert DURRER, Professor of Metallurgy, Federal Institute for Technology, Zurich; General Manager, The Louis de Roll Iron Works Ltd., Gerlafingen, Switzerland
Notes on the Production of Pig Iron
Georg BULLE, Gutehoffnungshütte Oberhausen A.G., Werk Sterkrade, Oberhausen-Sterkrade, Germany
International Research on the Low-Shaft Blast Furnace
Marc ALLARD, Director, Institut de Recherches de la Sidérurgie, Saint Germain-en-Laye, France
The Krupp Renn Process
Friedrich JOHANNSEN, Professor at the Mining Academy at Clausthal, Germany

Summary of Discussion:

Participants: Messrs. DURRER, MARIN GONZALEZ, WARING, BULLE, GONZALEZ VARGAS, KALLING, ALLARD, SEM, BUEHL, SANITER, LANARI, CAVANAGH, CORTÉS OBREGÓN, COHEUR, BAUM, JOHANNSEN, PRADO UCHOA, SHEROVER

Considerations on the Development of the Production of Iron

R. DURRER [1]

The production of iron is based on the elimination of oxygen from the ferrous matter. This operation requires a reducing agent as well as a source of heat. Carbon is the only practical reducing agent, whereas heat is obtained mainly through the combustion of an additional amount of carbon and, to a smaller extent, from hydro-electrical energy.

A convenient means of studying the basic principles of these methods of iron production in which carbon is used both as a reducing agent and as a source of heat—that is, of the methods which are still almost exclusively in use today—is the analysis of the functioning of the old Catalan furnace: a hole is made in the ground and used as a charcoal fire in which successive layers of ore and charcoal are loaded. A part of the ore is reduced to pasty iron (which assumes a lenticular form), another part is melted to a slag. The elements, iron, carbon and oxygen, tend towards a state of chemical equilibrium, which is, however, permanently disturbed by the addition of oxygen from the surrounding air. A stationary condition develops which is far removed from the theoretical equilibrium. As a result of the strong oxidizing action of the air, the reduction is very incomplete, i.e., the slag is rich in iron. A substantial enrichment of the iron in carbon is impossible under these conditions and it therefore assumes a chemical composition which falls within the limits corresponding to steel.

When under similar conditions pig iron is used instead of ore as a raw material, the additional elements are burnt out with the exception of phosphorus (only a small part of which is transformed into $FeO.P_2O_5$), and the pig iron is transformed into steel. The oxidizing action of the furnace can be increased intentionally. This complete change in the character of the reaction—

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[1] Lecture given by Professor R. Durrer on the occasion of the main meeting of the Austrian Iron Institute at Leoben on 15 June 1952. Published in *Berg- und Hüttenmännische, Monatshefte*, September 1952.

from a prevailingly reducing operation to a prevailingly oxidizing one—became necessary when, as a result of a modification in the conditions and temperature of reduction, pig iron replaced forgeable iron as the product of the primary smelting operation and had to be further transformed into steel. This happened in Europe at the beginning of the fourteenth century. It should be noted that basically the same kind of furnace as the one described above was used for the transformation of pig iron into steel, but the oxidizing influence had to be enhanced. In fact, reduction and re-oxidation into steel are relative notions; in the Catalan furnace the iron oxides of the ore are partially reduced; in the steel-making furnace a pig iron which is rich in additional elements and impurities is oxidized.

In the Catalan furnace the air penetrates freely into the reduction zone. The reducing action can be reinforced by excluding air or admitting only certain reduced quantities of it. The air can be excluded if, for instance, ore and reducing carbon are loaded into saggers which are heated from outside, so that the only carbon with which the air has contact is that furnishing heat. Under these conditions the reduction may be almost complete. When, as in the case in the Höganäs process, the production of sponge iron is aimed at, the reducing action is not as pronounced as in operations taking place at a higher temperature, and the resulting sponge has therefore the characteristics of a steel. If, on the other hand, just enough air is blown as is required for the combustion of the heating carbon, so that carbon is always in excess, the reduction can be almost complete at the temperature which reigns in the furnace; the iron can therefore saturate itself with carbon and the resulting product is pig iron. This is the case of the blast furnace, which has developed from the Catalan furnace.

Direct reduction of ore into steel is therefore only possible either if reduction and production of heat occur in different places, or if the reduction is made to be incomplete. Apparently this direct road from ore to steel seems to be the shorter one, but so far it has not proved to be economical for mass production until now, although it is not impossible that it may become so in the future. According to present experience, direct production of steel can be considered only in locally favourable circumstances and for the obtaining of specially valuable products such as high-grade scrap. For mass production of iron and steel, the indirect method involving intermediate production of pig iron is the cheaper one. The first part of this indirect method is almost exclusively the high-shaft blast furnace, which in the course of centuries has been so perfected technically that only insignificant improvements can still be expected. Although it is thermically very efficient and allows for large-scale production, it also involves such considerable basic drawbacks that it will be replaced in the foreseeable future by more adequate smelting furnaces. These drawbacks of the blast furnace have always existed; they were not, however, so apparent in the past as they have become since the production of iron assumed such a vast scale. Above all, it is only in this century that basically improved methods for iron ore reduction have been discovered. These methods must now be developed.

The main defect of the traditional blast furnace from which all its disadvantages stem, is the utilization of ordinary air for the combustion of that part of the carbon which generates heat. Every volume of oxygen blown into the furnace carries with it four equal volumes of dead weight in the form of nitrogen. This massive amount of gas absorbs a considerable quantity of heat and the temperature of the furnace is therefore relatively low. The rising gas column has the dual function of transforming as much as possible of the heat it carries to the burden which is travelling in the opposite direction, and also of reducing the iron oxides of this burden by the action of carbon monoxide (indirect reduction). In order to perform these functions, the blast furnace must have a considerable height: some 30 m. This in turn results not only in high construction costs, but in the necessity of utilizing raw materials of very great mechanical strength, in view of the height and weight of the burden itself. The only reason why we are not struck by these drawbacks is that we are so accustomed to the thermically very efficient blast furnace that we cannot imagine smelting operations taking place on a large scale without it.

This characteristic drawback of the high-shaft blast furnaces disappears whenever it is possible to perform smelting operations adequately whilst keeping the burden low. A much greater freedom in the choice of raw materials appears in low-burden reduction, as opposed to the high-burden reduction performed in the blast furnace. As a result, the cost of raw materials is reduced inasmuch as the mechanical demands to be made of them are considerably diminished. In addition, certain raw materials, which cannot be considered at all for use in the traditional blast furnace, can be used for low-burden reduction, so that in certain regions the production of iron can only be undertaken economically by means of low-burden smelting processes.

The transition from high-burden reduction in a blast furnace to low-burden reduction is made possible by retaining the layer-furnace principle but reducing the specific gas volume so that a high burden is unnecessary. The extreme case arises when no blast is used at all, that is if the furnace is heated from outside or through electric energy. Reduction with external heating is used in the sagger method, which has been used for centuries in China and for the last forty years in Sweden. These processes, however, are only economical under special local conditions and for the manufacture of special grades of iron (sponge iron, high-grade scrap) and cannot be considered for mass production purposes.

The methods involving electric heating—reduction in an electric furnace with a low shaft—are of practical interest. Only a small quantity of reduction gas is produced—approximately one-seventh of the quantity produced in the blast furnace. The shaft is only a few metres high and can therefore be adapted to local raw material conditions much more easily than the high-shaft blast furnace. However, the great drawback connected with electric reduction is the shortage of hydroelectric energy and the consequent high price. If this difficulty were eliminated, the use of the electric low-shaft furnace would become much more widespread. As things stand now, however, its application is limited to a reduced number of areas, but is steadily, if slowly, gaining ground.

The transition from high-shaft to low-shaft furnaces can take place also if oxygen is blown, without ballast gas. The elimination of nitrogen drastically reduces

the volume of gas which rises in the furnace, so that the burden of this oxygen low-shaft furnace need only be a few metres high. It should be noted that whereas the electrical low-shaft furnace has been used continuously for many years, reduction in the oxygen low-shaft furnace is still only at the experimental stage.

The utilization of an oxygen-enriched blast in the high-shaft furnace has been studied in more detail, but this method does not involve the considerable and revolutionary possibilities inherent in reduction performed with an oxygen-enriched blast in the low-shaft furnace. A limited improvement in high-shaft blast furnace operation can of course be obtained, but this method is to be considered, if at all, only as far as existing high-shaft blast furnaces are to be used. It would be unreasonable to build a new high-shaft furnace to be used with an oxygen-enriched blast.

Although the low-shaft furnace imposes considerably less stringent conditions on the mechanical properties of the raw materials than the traditional blast furnace, it still requires a certain lump-structure of the materials to allow the gas to flow through the burden. This requirement involves a drawback because large pieces of raw materials naturally react much slower than fines, which have to be agglomerated before being used in the furnace. If this agglomeration is practised on a mixture of ore and coal (and possibly additional elements), the conditions for a more rapid reaction are provided as a result of the intimate mixture of the raw materials and notwithstanding the fact that the burden appears in a lump form. In view of the more rapid indirect reduction and transmission of the heat from the gas to the solid burden, the shaft need not be so high as in the case of the blast furnace. The difference, however, would not be very substantial. In view of these factors, this process, in order to allow for the full exploitation of the advantages connected with it, should be combined with the reduction of ballast in the blast, that is with the utilization of oxygen or of electric energy. The practical possibilities of using this process depend on local conditions. They are favourable where ore and coal are available in fine form.

A still lower burden is used in those smelting methods which involve the use of horizontal furnaces instead of vertical ones—such as, for instance, rotary kilns—and in which the shaft completely disappears. In these extreme low-burden processes the heating gases flow through the burden, which is formed of thin layers of fine material. The outside heating process is thermically less favourable than that of the shaft furnace, in which the heat is generated within the materials to which it is to be applied. On the other hand, the necessary heat can be obtained in these cases from low-value solid fuels and also from gaseous and liquid fuels, and is therefore cheaper and more easily adaptable. Above all, this reduction method permits the utilization of fine ores and coals and is therefore not only advantageous from the standpoint of the reduction mechanism, but also from that of the raw materials which can be used. The limitation formerly envisaged—that the heating gases should not be oxidizing, because otherwise they would not allow for a sufficiently complete reduction—does not hold true as long as the solid burden includes enough reducing agents. If this is the case, the oxidizing gases (oxygen, carbon dioxide, steam) are reduced by the carbon; the excess of reducing carbon acts indirectly as a heating agent

Several rotary kiln processes (rotary tube kilns and drum kilns) are already in commercial use but they present considerable weaknesses. A few years ago they could not be used industrially, principally because the lining was destroyed too rapidly. This drawback has now been largely eliminated, as can be seen in the few works which operate continuously. The specific volume of gas is as large in the rotary kiln as in the high-shaft blast furnace, so that the kiln must be extended longitudinally in the same way as the high-shaft furnace is extended vertically, or it must be provided with a device for the utilization of residual heat. It may be possible that the utilization of an oxygen blast would allow of a better solution here.

A method which appears promising for the reduction of ores through "any kind of coal" with a low burden, would be a modification of the present ore-pig iron-steelmaking method. This method performs the transformation of pig iron into steel in a furnace through the addition of fresh ore which is reduced in the process. There is no reason why it could not be so modified as to be transformed from a steelmaking to a reduction method, i.e., that the reduction of the ore which at present is only a means for the attainment of the ultimate goal, becomes the goal in itself. Here again the utilization of oxygen could be of considerable benefit.

The heart of these considerations on a modification in reduction methods lies in recognition of the fact that the high shaft of the smelting implement which at present is practically the only one used—i.e., of the blast furnace—is in itself a basic drawback. This drawback is in turn due to the utilization of an atmospheric blast. An evolution is taking place towards low-shaft methods, but it will of course require several years to develop. Even after it has been carried to its successful end, the operation of existing plants will not simply be discontinued, so that reduction with the utilization of an oxygen blast will presumably take place first in newly built plants.

As in the case of iron ore reduction, the ballast included in the blast is also the reason for the inadequate functioning of large-scale methods of steelmaking: that is of the hearth furnace and of the converter. Under present working conditions, the hearth furnace has certain advantages over the converter, which explains the fact that it is used notwithstanding its higher operating costs. The hearth furnace is the scrap smelter *par excellence:* in the converter only a few per cent of the scrap can be added to the charge and thus regenerated into steel, while the open hearth allows for the utilization of 75% of scrap in the charge, in the case of the most current method (scrap-pig iron process), against 25% of pig iron. However, in the case of the scrap-coal process all the ferrous material appears in the form of scrap. Moreover, the hearth furnace is qualitatively superior to the converter from many points of view (however, this disadvantage of the converter has recently been reduced to a large extent, even with the utilization of a normal blast). Finally, the hearth furnace is much more elastic, as far as the pig iron used is concerned, than the converter. This explains why in the United States, for instance, the hearth furnace is almost exclusively used for mass production of steel.

If it becomes possible to blow pig iron industrially with a high-oxygen blast instead of air, the excess heat can be used to regenerate approximately half a ton of scrap per ton of pig iron. In addition, oxygen-blown steel is at least equal in quality to open hearth steel—although this improvement in quality can be obtained largely without the utilization of oxygen. Finally, almost any kind of pig iron can be converted if an oxygen-enriched blast is used. In view of these considerations, which will undoubtedly be proved correct in the short or in the long run, mass transformation of pig iron into steel will mainly take place in the future through oxygen-blowing, and this production process will therefore replace the hearth furnace and the air-blown converter. This production method has already been utilized industrially for the converting of low-phosphorus pig iron; in the case of high-phosphorus pig iron certain difficulties still exist, but they are not of a basic character.

The foregoing considerations do not show the future picture of the production of iron in its details, but allow the main trends to be seen. It may be considered as certain that mass production of iron from ore will shift from high-shaft to low-shaft processes with a concomitant changeover from air blast to oxygen blast. The metal thus obtained, apart from the relatively small quantities of iron used for foundry purposes, will be an intermediate product of high silicon content and of considerably variable composition, which will be transformed into steel according to a continuous range of processes. The main part of steel-making will involve the utilization of oxygen. The electric furnace will play an important role as producer of high-grade products and may also become a mass production tool involving the blowing of pig iron with an oxygen-enriched blast, and the refining of the resulting steel through utilization of electric energy. The heat which will be produced as a result of the oxidation of the accompanying elements will be used to smelt a corresponding amount of scrap. Other combinations can, of course, be considered and put into practice.

The modification in iron production methods on the lines indicated above, will take place at the latest when the raw material conditions will make it essential. It can also take place earlier if responsible circles recognize its necessity in time. When this takes place, a method of production of iron—this all-important metal for our future—better geared to the characteristics of the raw materials available, will replace reduction in the blast furnace and steelmaking in the hearth furnace and in the converter, and lead humanity upon new paths.

Notes on the Production of Pig Iron

G. BULLE

In order to expand steel production in Latin America it is essential, first of all, to develop the production of pig iron, for, owing to the shortage of scrap, steel production will have to be based on pig iron. As a rule, pig iron production will probably move along conventional paths, that is to say, the coke blast furnace will be used. As Latin America possesses little good coal and hardly any coke, but good quality ores, it will be less interested in the blast furnaces of the regions with rich coal resources—Pennsylvania, England, Germany—which consume 900–1,000 kg. of coke per ton of pig iron in the blast furnace, and will prefer to follow the example of the regions poor in coal—California, Sweden, Austria, and so forth, where a good preparation of the burden and efficient practice have reduced carbon consumption figures to 600–700 kg. per ton of pig iron. One plant in Brazil has already achieved a similar favourable fuel economy in this way.

Some countries, especially Argentina and parts of Brazil, continue to operate charcoal blast furnaces in order to avoid the use of coke, and are accordingly increasing their capacity from a former 10–15 tons to a present 100–200 tons per day. Due to the amount of space they require, however, and to the costs of charcoal production, the employment of these furnaces is limited.

In other localities, as for instance in Peru or in several districts of Brazil which are rich in water power, the blast furnace is now replaced by the electric low-shaft furnace, in which process the coke or charcoal requirement for the production of pig iron can be reduced by nearly one-half. Extension of the works depends, however, on the development of the big and expensive hydro-electric power stations (300,000 tons of pig iron per annum require a power station of 150,000–200,000 kW).

All substitute processes which permit iron to be produced from ores without coke and charcoal (Wiberg, Krupp Renn, Basset, Stürzelberg, Höganäs, Cavanagh) are also being studied with great interest. These processes are described at length elsewhere during this meeting.

Recently, too, the development of the fuel-heated low-shaft furnace is being observed. This new furnace is a small blast furnace with a shaft so shortened that it can be run on breezy, soft fuels or even on raw pit coal or lignite without any risk of clogging. The low-shaft furnace was developed in Germany, and now that Eastern Germany has been cut off from the Ruhr coal by the Russian occupation, 20 low-shaft furnaces are to be erected there for the production of pig iron with lignite coke. Two of these furnaces are said to be in operation already, though still working with inferior pit coal coke.

In the Ruhr district a low-shaft furnace has been working satisfactorily for the past seven months and has produced iron alloys, first ferro-manganese and now ferro-silicon, from fine ores and small coke with oxygen addition. This furnace is the well known experimental blast furnace of the Gutehoffnungshütte in Oberhausen, with which Dr. Lennings has carried out his tests of the blast furnace working with oxygen. This little blast furnace (hearth diameter 2.4 m.) has been shortened to 8 m., and now, as a low-shaft furnace, reduces like a blast furnace, i.e., with normal air pressure, normal air temperature, blast furnace air with oxygen addition, and normal pig iron composition, without any disturbance. As the furnace works economically in the production of iron alloys, it is possible systematically to collect experiences which are still lacking for the operation of the low-shaft furnace in the production of other kinds of iron.

The following questions are involved:

(1) How can the theoretical advantages of the low-shaft furnace be achieved?
  (*a*) The reducing with breezy, soft, small-lumpy coal poor in gas, i.e., experiences must be collected with small coke, lignite coke, etc.;
  (*b*) The reducing with coal rich in gas, tar coal, caking coal, i.e., experiences must be collected for each of these kinds of fuel. How can caking in the furnace be prevented, how are the gases cleaned and utilized?

(2) How can the theoretical disadvantages of the low-shaft furnace be eliminated, which in part occurred already in the tests?
  (*a*) *High consumption of fuel.* If the cut-off blast furnace shaft has hitherto done a lot of metallurgical work, for instance has performed 50% indirect reduction, this means that coke consumption will increase by about 50% when this work is lost to the furnace, and if, instead of gas-poor coke, use is made of gas-rich coke, the gases of which escape wastefully out of the low-shaft furnace, this would mean, in the case of fuels with 50% volatile components, a further doubling of the fuel requirement, i.e., altogether, the requirement will be trebled. Two to threefold increases in the fuel consumption have been observed in practice. Where the blast furnace shaft does little or no metallurgical work, as is the case, for instance, with the iron alloy furnace, the low-shaft furnace can run without any increase in fuel consumption if it simultaneously lowers the high flue-gas temperature of the furnace resulting from the shortening of the furnace by an addition of oxygen. For this reason, the low-shaft furnace with oxygen is of particular interest for iron alloys;
  (*b*) *Low furnace output.* The high fuel consumption of the low-shaft furnace in the production of normal pig iron leads to a low furnace output, and investment and operational costs are accordingly high. These high costs thus frequently more than offset the reduction in investment costs due to the lowering of the blast furnace. The following methods are envisaged to reduce fuel consumption in the low shaft furnace:
    (i) Reduction of the reaction time by reducing the size of the burden and of the fuel granulations;
    (ii) Intensification of the reactions by increased pressure;
    (iii) Intensification of the reactions by addition of oxygen.

It may be hoped that a systematic study of these possibilities will reveal exactly what can be accomplished in the low-shaft furnace. At present our knowledge is still in its infancy.

An interesting suggestion for the intensification of the low-shaft furnace reactions comes from the well known preparation firm of Klöckner-Humboldt-Deutz.[1] Their low-shaft furnace process uses briquettes made from fine ores and coal fines, which are held together with binding agents. Briquettes were made from a whole range of ores and fuel and reduced to pig iron in a little test furnace having a cross section of one square metre. No continuous operation has been possible until now, nor have any results of the tests been published. It is not yet known therefore, how far the improvement in the reactions anticipated from the decreasing of the grain size of ore and coal can be achieved by briquetting; how the difficulties connected with the use of coal can be overcome; how specified kinds of pig iron are made, and so on. The results of further experiments must be awaited.

Recently, too, attention has been given in Germany to the question of the direct utilization of the hot flue gases in a boiler, it being unnecessary in this case to clean the gas (Ko-We process, from Koppenberg-Wenzel).

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[1] Cf. Killing, E. *Inst. Hierro y Acero* 5 (1952), No. 2, pp. 575–584, and *Stahl und Eisen* 72 (1952), nr. 16, pp. 625/28.

International Research on the Low-Shaft Blast Furnace

MARC ALLARD

The production of iron and steel is a problem of world importance while at the same time scrap is becoming more and more scarce. The production of pig iron in the blast furnaces must therefore be increased. Nevertheless, in many countries good coking coal is not available in sufficient quantities, thereby limiting blast-furnace production. Proper sizing of the raw materials of the burden is now a generally accepted means of reducing coke consumption, but the unavoidable crushing of the iron ores produces large quantities of fines which are not directly usable in the blast furnace. Hence, in certain countries an important problem remains to be solved, namely the reduction of fine ores using coal that is not good for coking.

With a view to seeking a solution to this problem, the OEEC invited those countries essentially interested in it to study the question in common. As a consequence, an international organization was set up for the study of the low-shaft blast furnace. Its name is "International Low-Shaft Blast Furnace Research Committee", having its seat in Paris. A pilot plant is under construction in Belgium, near Liège.

This organization's activities are of a non-profit nature and have no connexion with any particular company. The expenses and financial charges of the organization are entirely covered by the participating countries which are at present: Austria, Belgium, France, Great Britain, Greece, the Netherlands, Italy, Luxembourg, USA.

Only one representative per country is admitted on the board of directors and this implies that full agreement exists between the various interested firms of any one country.

The minimum initial contribution for each country has been set at US$20,000. The funds collected so far amount to about US$500,000 and are sufficient to cover the expenses involved in the setting up of the basic apparatus. For further extension of research, annual contributions will have to be made in addition to the initial one. The size of these contributions will obviously depend on the nature of work to be done, each country, for instance, having to pay the major part of the expenses in the event of its desiring to run the furnace with its own raw materials.

The results obtained at the pilot plant might lead to patents; the financial resources accruing from the sale of such patents will be shared among the participating countries proportionately to their contributions. However, only "founding members," i.e., those countries having joined the organization prior to 1 January 1953, will be entitled so to share.

INTRODUCTION

Although the regular blast furnace has a good thermal efficiency, it has the drawback, because of its height, of requiring coke with a high resistance to crushing. Coking coals, which are scarce in many countries, are needed in preparing the coke. The heat value of the coke is nearly 40% more expensive than that obtained by the combustion of raw coal. Furthermore, the raw materials of the burden have to be sized. The amounts of ore and coke fines that can be used are limited. The quantities of these fine raw materials are increasing and will probably continue to increase.

For these reasons, a group of specialists in the field of iron production are seeking a general solution to the problem of utilizing fine ores and poorer grades of fuel in the blast furnace. They are now considering the use of these materials in a low-shaft furnace which will allow the use of fuel with a lower resistance to crushing.

The low-shaft furnace provides good thermal efficiency with oxygen-enriched air, a steep temperature gradient, and a high internal gas pressure, which combine to allow a higher proportion of fines in the burden.

The use of either oxygen-enriched air or of increased top pressure or both also arises from the need to reduce the velocity of the gases in the furnace when the burden contains a high percentage of fine material. An increase in pressure will also improve the physical conditions associated with the reduction of the ore, resulting in a more economical operation. The use of increased pressure can be achieved more easily in a low than in a high furnace.

Possibly greater attention will have to be devoted to the physical preparation of the burden, since drying and reduction processes are somewhat faster in the low-shaft blast furnace.

It is not impossible that a low-shaft furnace blown with oxygen will produce a gas of greater calorific value, appropriate for chemical synthesis. The use of a mixture of oxygen and recycled gas would favour this.

In brief, the following advantages might accrue from a low-shaft furnace blown with oxygen:

(1) A cheaper and more abundant fuel than coke;

(2) A higher percentage of fines could be used;

(3) A richer outgoing gas.

The drawbacks of a low-shaft furnace would be:

(1) The necessity for handling a large supply of oxygen;

(2) The need for physical, and possibly chemical, preparation of the burden.

RESEARCH PROGRAMME

The principal points of interest of the research programme on the low-shaft furnace from the standpoint of European countries are:

(1) The production of Thomas pig iron from fuel of low commercial value, such as coke fines, semi-cokes, fines of low-quality coals, lignite, and so on, and from fine ores coming from the minettes of Lorraine or Luxembourg. The experiments will begin with raw material from 11 to 22 mm. in size and will be aimed at an increasing use of material from 0 to 10 mm. in size which eventually may be in the form of briquettes;

(2) Production can be considered of pig iron and ferro-alloys from ores such as chrome ore from Greece or manganese ores of the Belgian Congo that are difficult to utilize by ordinary reduction methods.

BASIC INFORMATION CONCERNING THE PROJECT

The basic premises on which the project was outlined, as defined by the International Committee, are as follows:

1. *Capacity of the low-shaft furnace and length of each experiment*

For the experiment to be significant, the daily production of iron had to be comparable to that of an industrial scale operation. Each run of experiments must be long enough to eliminate hazardous conclusions or dangerous extrapolations.

It was therefore decided to build a low-shaft furnace with a daily production of 60 to 100 tons of pig iron, the working period for each run being 15 days, which would seem long enough since the burden will remain in the furnace for about two hours.

2. *Importance of the burden*

Using minettes of Lorraine or Luxembourg, with an iron content of the order of 30%, 250/300 tons of ore a day will be required. The use of richer ore will increase the furnace output.

Previous small-scale experiments appear to indicate that the thermal yield will be acceptable, and it can be hoped that the *mise au mille* will be from 1,000 to 1,500.

3. *Amount of air and oxygen.*

To be on the safe side, it was decided to have available more blast than required by calculation, i.e., up to a total of 14,000 m^3 of air per hour. The maximum working pressure in the furnace will be 2.5 kg. per $cm.^2$.

A maximum of 45 tons a day of oxygen, with 50% enrichment of the air, was considered necessary.

4. *Preparation of the burden*

The experiments will be carried out in two ways: firstly, with the usual layer conditioning of the burden, followed by an agglomerated charge, with possible pre-thermal treatment.

The installations include: raw material storing and handling equipment; burden conditioners; low-shaft furnace itself; blowing equipment; gas cleaning units; pouring and casting equipment, and slag disposal equipment.

PRESENT STATUS OF THE PROJECT

The following equipment has been ordered and is now in the process of construction:

1. *Installation for storing and handling of raw materials*

The storage bins will be situated in a covered building 65 x 25 x 16.50 m., provided with a 2.5 ton travelling crane having both a grab bucket and an electromagnet, travelling on 14-metre-high rails.

The storage capacity corresponds to five days' material requirement, i.e., 1,750 tons of ore and 500 tons of fuel, the surplus being stored at the port or at the mine. The storage installation consists of three bins 3 m. deep with a cross-sectional area of 130 sq. m. The building will house, besides the storage installations, the casting and the burden conditioning equipment.

2. *Preparation of the burden*

It has been decided to provide equipment which will permit charging the furnace in layers as in the regular blast furnace, or with a homogeneous mixture having specific proportions of ores and fuel. The raw materials will be provided in sizes necessary for the experiments, i.e., either 11 to 22 mm. or 0 to 10 mm. in size.

From the storage units, the various components will be transferred to three hoppers of 30 cu. m. capacity each. Provisions have been made for adding supplementary hoppers. Withdrawal of material from the hopper and weighing will allow either continuous flow of material with homogeneous mixtures or an intermittent flow if charging is to be in layers.

The equipment has been devised primarily for using minettes and silicious ores, which are the most difficult to handle. It includes two metallic band conveyors, a weighing scale, a helical mixer, and a two-way skip hoist with 800-litre buckets, travelling at a speed of 20 m./min.

3. *Low-shaft furnace*

Because experiments are being conducted with a circular low-shaft furnace in Oberhausen, Germany, it was decided to build in Liège a furnace of elliptical section, since the Committee decided that this would increase the efficiency of the circulation of gases through the burden and thus provide a more uniform combustion zone. It will have a pig iron capacity of 60 to 100 t/day, supplying 60 to 120 t/day of slag, and 18,000 cu. m. of gas per hour (normal temperature and pressure).

The furnace will be fitted with two blowers, one for higher pressure and low flow, with a capacity of 9,000 cu. m. per hour under a pressure of 2.5 kg., and the other for low pressure, 2,000–4,000 mm. H_2O, and high flow—14,000 cu. m. of air per hour. Two Cowper stoves will pre-heat the blast to a maximum temperature of 800°C. The slag will be granulated and loaded directly into railroad cars.

The experimental plant is located on the grounds of the Ougrée-Marihaye steelworks, from which it will be able to obtain laboratory facilities, oxygen, electric power and other supplies. The lay-out of the plant is shown in Figure 1.

PRELIMINARY RESEARCH ON THE REDUCTION OF FINE IRON ORES AND OF AGGLOMERATED SLURRY IN A HOT-BLAST MELTING FURNACE

Before conducting runs in the low-shaft furnace, the IRSID[1] decided to charge briquettes from fine ores and poor grade fuel into a hot blast cupola. This furnace, of regular design, is equipped with hot blast, capable of supplying the tuyeres with 450–550°C air. It has a gas-sealed top.

The top gases, at some 200–400°C, may either go to the atmosphere, or burn in the pre-heaters, after being freed of the heavier dust particles.

Pig iron and slag are separated through syphoning in a ladle and the pig iron runs continuously into a mixer having a horizontal axis, at a temperature of about 1,550°C.

This apparatus is normally used for the production of pig iron from granulated steel. The pig iron is always tapped at a high temperature and its sulfur content is low. The normal yield of such a furnace is, under these conditions, 5 tons per hour.

The briquettes were prepared by agglomerating a mixture containing 56% of Lorraine ore, dry and free from dust, of about 15-mesh size as a whole, with less than 4% fines; 37% slurry of 0–1 mm. size and 7% of a mixture composed of equal parts of tar and pitch. These ingredients were blended in a mixer and briquetted with 4% water. The briquettes were subsequently baked at 700–750°C. Each ton of the mixture, after baking, yielded about 750 kg., furnishing briquettes of 200–250 kg. compressive strength.

The primary objective of this research was only to study the possibility of reducing the prepared briquettes into pig iron. On the other hand, due to the restricted air supply, part of the burden was steel and iron scrap, in order to have a sufficient output of iron. The results using briquettes have been deduced by comparing them with the results obtained when operating with scrap alone.

Table 1 shows the calculations concerning the run:

Table 1

Burden	*Kg. per ton of iron produced*
Briquettes	1,768
Steel shot	530
Ferro-Si	1
Limestone	10
Basic converter slag	6
Shingles	16
TOTAL	2,331
Coke	106
Dust	159
Gross yield: 1,000/2,331 or	42.9%
Efficiency when dust is deducted: 1,000/(2,331 − 159) or	46%

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[2] Institut de recherches de la sidérurgie.

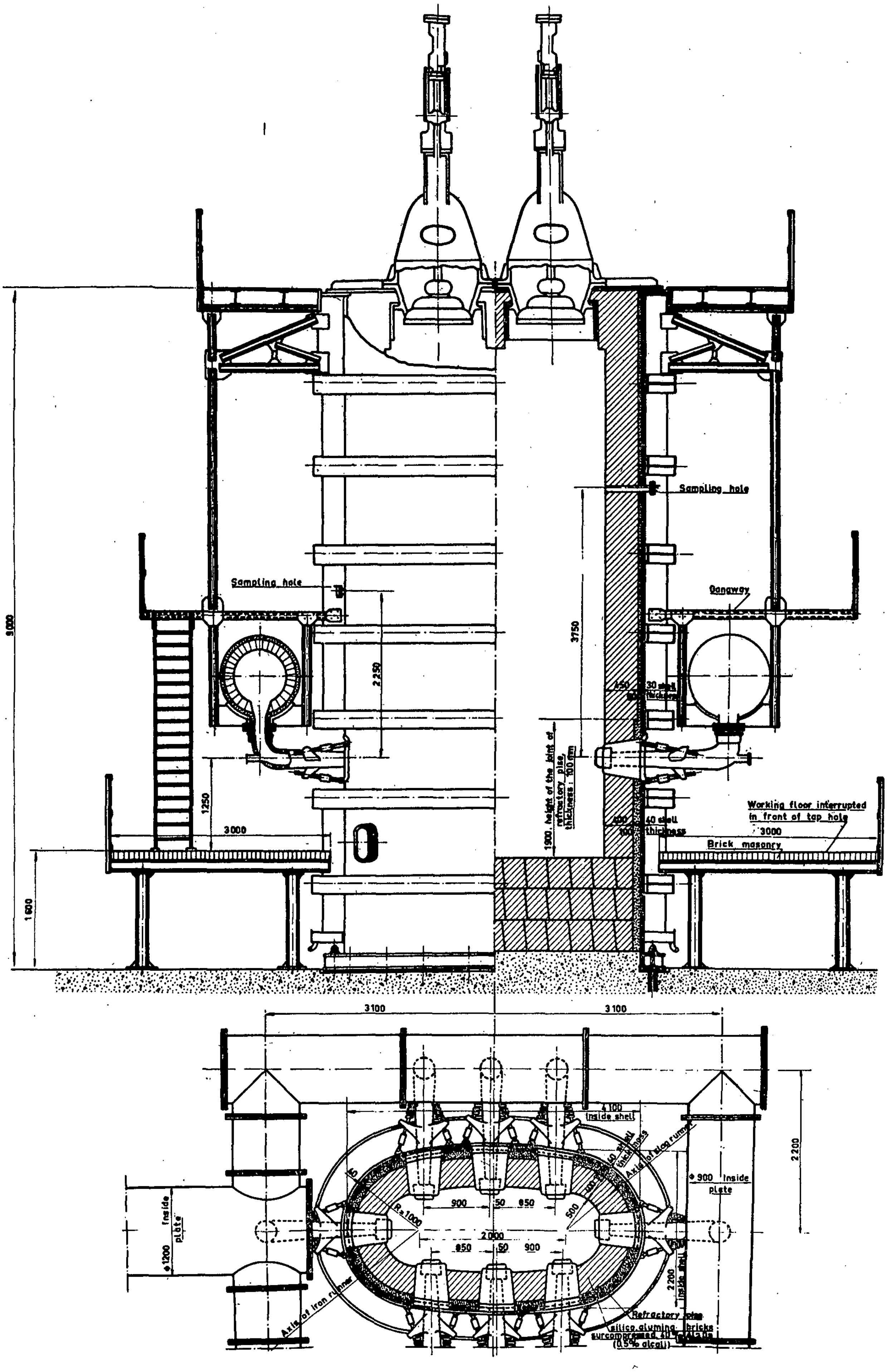


Figure 1
Layout of a low-shaft furnace at Ougrée-Marihaye.

Since 530 kg. of steel shot give 530 × 1.04 = 551 kg. of iron, there are: 1,000 − 551 = 449 kg. of pig iron derived from the briquettes, with a yield of 449/1,768 or 25.4%.

The percentage of briquettes in the burden is: 1,768/2,331 or 76%. The temperature of the blast is 505°C., and the temperature of the gases is 270°C. Blast pressure is 101 cm. water. The temperature of the outgoing iron is 1,490°C., while the slag temperature is 1,500°C.

The production of iron per hour using briquettes was 1.4 tons, whereas, with scrap alone, the capacity was 5.3 tons per hour.

Table 2

	H_2O	SiO_2	Al_2O_3	CaO	MgO	Fe	Mn	P	S	CO_2	C
Briquettes	2	7.3	4	12.1	1.3	24.5	0.2	0.5	0.5	6	34.5
	SiO_2	Al_2O_3	CaO	MgO	Fe	Mn	P	S	CaO/SiO_2		
Slag	29.4	15.6	47.5	1.9	1.1	0.3	0.1	1.7	1.72		
	C	Si	Mn	P	S						
Pig-iron	4.18	1.72	0.46	0.81	0.014						
	CO_2	CO	H_2	N_2	PC	CO_2/CO					
Gas (dry) (volume 5)	4.55	33.1	5.45	56.9	1,141	0.137					

The actual coke consumption of this furnace was 220 kg. (85% C) per ton of iron produced, using only scrap charges; from the above figures for briquette heats, the coke consumption is 1,345 kg. of carbon per ton of iron, or 1,345/0.85, that is, a coke consumption of 1,580 kg. per ton of iron melted. This high figure should be considered in the light of: the low temperature of the blast (505°C.); the high temperature of the top (270°C.); the high thermal losses, low production of the furnace when using briquettes (1.5 tons/hour); the high iron and slag temperature; the high basicity of the slag, 1.72 ratio, and the high calorific power of the flue gas, 1,145 calories.

Admitting that the composition of the gas cannot be changed, there remains the fact that all other items correspond to a waste of fuel and it is hoped that lower coke consumptions can be achieved by: lowering the top temperature, using oxygen-enriched air; increasing the production; decreasing the amount of C in the burden; decreasing the basicity of the slag.

It is hoped to reduce the large amount of dust by improving the condition of the briquettes, which were stored for a long time before their use, and by reducing the speed of the gases.

* * *

This research proved the impossibility of producing iron in a satisfactory way, even with oxygen-enriched air, in such a small furnace, with fine ores and mine refuse.

This was one of the reasons that induced the Committee to try a low-shaft furnace capable of producing at least 3 tons of iron per hour.

It is believed that a marked improvement will be secured in the operation of the low-shaft furnace. It is thought that its production could be markedly increased if fines of the rich South American ores were used. The minimum amount of slag necessary could then be partially introduced in the burden by using unwashed mine refuse as a fuel.

With burdens including the rich fines of South American ores, its further characteristics and economic advantages will only be developed after the furnace has been worked for a certain period.

The Krupp Renn Process

FRIEDRICH JOHANNSEN

The reduction of iron oxides using carbon as a reducing agent takes place at temperatures below the melting point of iron, and every particle of iron oxides in the ore thereby forms a small particle of solid iron, the so-called sponge iron. In the standard blast furnace process, this sponge iron is turned into liquid pig iron, while absorbing carbon, and separated by settling from the liquid slag, which is formed at the same time, on account of its greater specific weight.

The idea suggests itself of abandoning the formation of liquid pig iron by producing iron without resorting to the blast furnace process and to produce only sponge iron, which can immediately be turned into steel.

Unfortunately, the metallic sponge iron produced at low temperatures always contains a certain quantity of iron oxides, which in steelmaking get into the slag and are thereby lost. Moreover, it holds the gangue of the ore in a finely suspended state which cannot be removed to a sufficient degree even by fine grinding and magnetic dressing. For the production of metallic sponge, therefore, only such ores are suitable which contain so little gangue that the reduced material can be fed to the steel furnaces without magnetic dressing, preferably lumpy ores from which the least quantity of metallic sponge particles may subsequently burn away and which show, therefore, the highest recovery of iron from sponge in the steel furnace.

A third method of producing iron in relation to the temperatures applied lies between the metallic sponge method and the blast furnace process, and is represented by the Krupp Renn process. In this process the individual particles of the sponge iron are welded together at temperatures from 1,200–1,300°C., whereby larger iron nodules free of slag are formed, which are called "luppen". This process is carried out in a rotary kiln, so that the semi-liquid slag, formed by the gangue of the ore at these temperatures, is continually kneaded and turned over, facilitating the contact of the individual particles of the sponge iron and their welding together. The luppen are separated from the slag outside the kiln; the kiln discharge has the consistency of a cake dough, in which the luppen are distributed like raisins, and is subjected after a sufficient cooling to a selective crushing, followed by magnetic separation. It is obvious that such a separation into practically slag-free luppen and a slag containing very little iron is achieved very much more simply and completely than the magnetic separation of the individual small iron particles from sponge, in the case of the sponge process. The recovery of iron in the Renn process is therefore at least as high, usually even higher, than in the blast furnace process, in which, as experiences have shown, a certain portion of the iron remains entrapped in the slag in the form of small droplets.

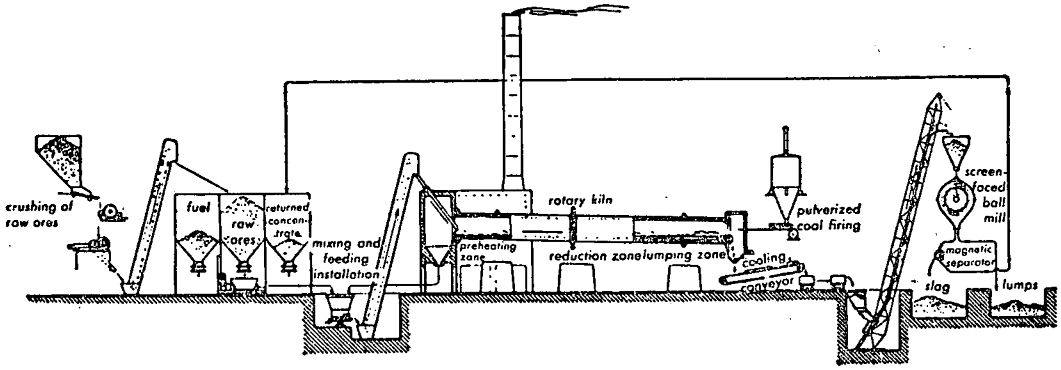


Fig. 1 Flow sheet of a Krupp Renn-plant

The flowsheet of the Renn process is shown in Figure 1. The crude ore, which in the case of limonites must be crushed to < 20 mm. and with magnetites to <6 mm., is mixed with solid fuels ranging from about 0–5 mm. in size. To this mixture a magnetic intermediate product of the process and the flue dust of the Renn kiln are added, and the well-mixed and moistened material is continually charged to the slightly sloping Renn kiln. The rotary kiln, used in this process, is similar in general to the kilns used in the cement industry. The mixture charged at the upper end of the kiln is first dried by the waste gases and preheated at the same time. The gases leaving the kiln with temperatures from 230–400°C. reach the chimney with temperatures of about 200–300°C., after passing through a flue dust chamber.

The greater portion of the rotary kiln—about 60% of the whole kiln length—serves for the reduction of iron oxides to sponge. In the interior of the charge are found, in this zone, gases rich in CO, and therefore reducing gases, whilst the gases in the free kiln area above the charge still contain oxygen, so that the carbon monoxide is burnt at the surface of the charge. The heat thus developed is to a large extent returned into the charge by the revolving kiln-lining and thus directly made use of for the reduction. The Renn process differs from most of the methods of producing pig iron—especially the blast furnace process—in that the waste gases contain no conductive heat in the form of CO. This means a smaller consumption of fuel and a simpler layout, as no auxiliary plants are required for the subsequent utilization of the CO-containing waste gases.

During reduction the total amount of heat, conveyed into the charge, is utilized for accelerating the reduction process. The charge has a temperature of about 1,100°C. and the reduction gases escaping from its interior are to be seen on its surface, burning brightly in the free space of the kiln. The process is regulated in such a manner that reduction is finished within a zone of 15–20 m. from the discharge end of the kiln, and the formation of the luppen may commence. For this purpose, the charge in this last zone of the kiln is raised by means of a firebrick ring at the discharge end and forms here a materially thicker layer than elsewhere in the kiln, remaining in this zone for about 5–6 hours. At the same time, air or oxidizing heating gases are blown on the charge in this zone, so that at the surface a direct combustion of carbon and sponge iron takes place. This results in a sudden rise in temperature to 1,200–1,300°C. A semi-liquid slag is formed, in which the individual particles of sponge iron come into contact with each other and are gradually welded together into luppen in sizes up to 50 mm. The iron oxidized at the surface of the charge is again reduced in its interior, so that the iron content of the total slag amounts only to about 3%. The size of the luppen depends upon the time employed. By adding a sufficient quantity of fuel and by lengthening the time for forming the luppen it is possible to produce them of any size. Usually an average size of about 3 to 8 mm. is sufficient, as these luppen can easily be separated from the slag and this size suffices for the subsequent treatment.

The discharge of the kiln, consisting of a semi-liquid slag with luppen embedded therein, is chilled with water on a cooling conveyor; the slag, solidifying in a glassy and brittle state, is easily crushed. The selective crushing of the slag down to about 2 mm., effected in a ball mill, leaves the luppen untouched, but frees them from the attached slag. Luppen and slag are separated by screening, and concentrates containing very fine luppen disseminated in the slag are subsequently extracted from the finely ground slag by magnetic separation. These concentrates, amounting to about 5 to 10% of the ore and containing about 65 to 70% Fe, are returned to the kiln without additional fuel and cause the formation of new and larger luppen as nuclei of crystallization.

The Krupp Renn process was developed by the author in 1931–33 in the research laboratory of Fried. Krupp Grusonwerk A.G. at Magdeburg. As a result of these tests, in kilns of 2.5 tons and 12 tons daily capacity, which permitted a thorough examination of the process and in which ores and fuels of very different types were tried out in experiments lasting for weeks, the firm of Krupp decided to erect two plants with one kiln each with a daily capacity of 300 tons each. One plant served for the treatment of iron ores of different types, the other for the treatment of a poor and oxidized nickel ore with 0.8–0.9% Ni giving nickel-containing luppen with 8% Ni.

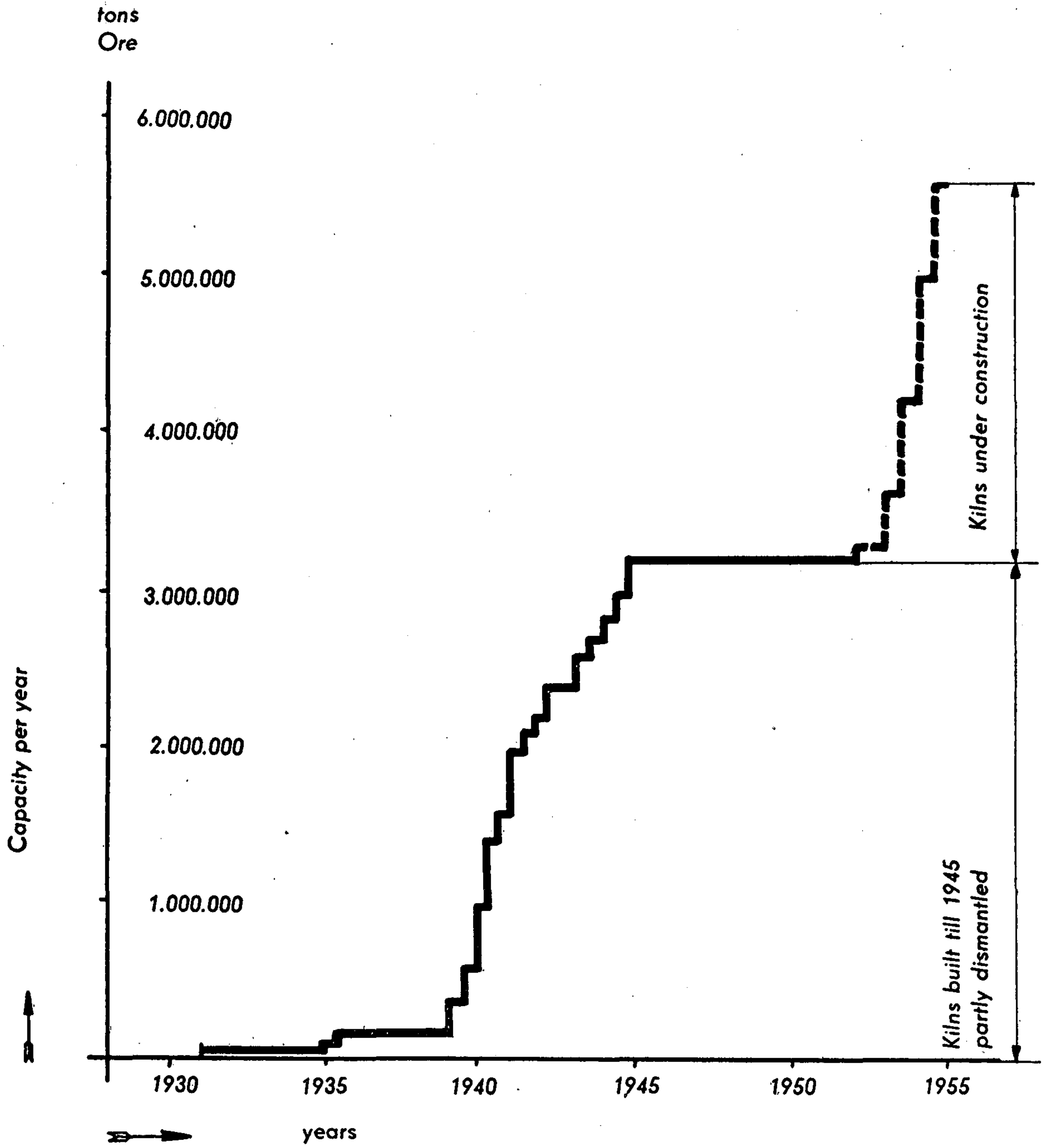


Fig. 2

Development of total capacity of Renn plants

The further development is shown in the graph of Figure 2, based on the annual capacity of ore of the plants built or under construction. Only after Krupp's first two production-size plants had been operating for three consecutive years, and all questions connected with the transfer of the process from a pilot-plant to producing plants had been solved, were further plants projected and built in Germany, Japan, Korea, Manchuria and Czechoslovakia. These plants, which were put into operation from 1939 onwards, had reached, by 1945, an annual capacity of about 3,200,000 tons of ore. At the end of the Second World War some of these plants, among them all the German ones, comprising six kilns with an annual capacity of

750,000 tons of ore, were dismantled. The author does not know how many of these plants have since been re-erected in other places, and are working today. However, the re-building is contemplated, in the near future, of one of the German plants, with three kilns and an annual capacity of 450,000 tons of ore.

Owing to the political conditions prevailing after the Second World War, further development could only be carried out after several years had elapsed. At the time of writing there were—to the author's knowledge—18 Renn kilns under construction in various countries with an additional annual capacity of 3 million tons of ore.

The advantages of the Renn process, which have led to the erection of these plants, compared with the other iron-producing methods, are essentially the following:

(1) Fewer demands on the quality of the ore both with respect to its physical properties—as ore fines can be charged directly to the kiln without previous sintering—and chemical properties, as silicious ores and those containing titanium can be worked with relatively small additions of limestone;

(2) The possibility of using solid fuels of all kinds, including fuels with high ash contents;

(3) A lower fuel consumption than in, say, the blast furnace process, in which the melting of the iron and slag is effected under such conditions that a combustion of the coke to carbon monoxide only is possible, so that fuel consumption is high. In the blast furnace the carbon monoxide, formed in the melting zone, is used partly for the reduction of iron oxides in the stack as well as for the preheating of the air in the cowpers. But there is always a surplus of low-grade blast furnace waste-gas with 900 to 1,000 kcal/Nm^3, which is produced from costly metallurgical coke and for the utilization of which auxiliary plants must be built;

(4) As no plants are needed for the utilization of blast furnace gas, nor coke ovens, plants for sintering fine ores, nor devices for catching up and transporting liquid iron and liquid slag, the initial costs of a Renn plant are considerably lower than those of a blast furnace plant of the same capacity;

(5) The production costs per ton of iron in a Renn plant are lower than in a blast furnace plant because of the simpler operating methods, and the lower consumption of fluxes and fuel, which furthermore may be of inferior quality and are therefore less expensive. Still lower is the consumption of electric energy and very small the consumption of water.

As against these advantages are the somewhat higher consumption of refractories and perhaps certain additional costs for the re-smelting and refining of the luppen. But on the whole there is frequently a rather relevant advantage in favour of the Renn process, especially if new plants are being built or metallurgical coke is not available.

Table 1

RESULTS OBTAINED IN EXPERIMENTS

Experiment No.	*1*	*2*	*3*	*4*	*5*	*6*	*7*	*8*	*9*
Type of ore	*Hematite*	*Magnetite concentrate*	*Magnetite concentrate*	*Hematites (mixed)*	*Limonite*	*Roasted spathic iron ore and limonite*	*Titanic iron sand*	*Red mud*	*Limonites (mixed)*
Analysis (dry substance)									
Fe . . . . %	64.7	50.6	55.9	55.5	47.1	43.0	35.9	32.2	38.5
SiO_2 . . . . %	0.6	24.1	8.0	18.1	10.0	11.0	27.7	4.3	13.4
Al_2O_3 . . . . %	0.8	2.2	1.5	1.0	5.5	3.1	8.3	13.3	5.4
CaO . . . . %	—	1.2	0.5	0.2	1.2	2.3	0.4	13.6	4.0
MgO . . . . %	—	0.9	9.0	—	0.7	2.7	0.6	0.8	0.9
S . . . . %	0.05	0.05	0.04	0.09	0.2	0.9	0.1	0.3	0.2
%						5.1 Mn	5.8 TiO_2	5.7 TiO_2	
Moisture . . . . %	2.0	0.4	1.5	1.5	11.5	8.3	20.7	56.4	8.0
Reducing fuel	*Charcoal*	*Low temperature coke*	*Anthracite*	*Coke breeze*	*Coke breeze*	*Low temperature coke*	*Low temperature coke*	*Raw brown coal*	*Low temperature coke*
Analysis (dry substance)									
Solid carbon . . . . %	68.1	69.2	63.8	83.8	80.8	57.8	80.1	. . .	65.8
Volatile constituents . . . . %	26.8	9.3	6.1	3.9	3.6	14.5	6.4	. . .	14.6
Ash . . . . %	2.8	19.2	29.2	11.7	12.7	26.5	12.9	18.1	16.8
Sulphur . . . . %	0.06	0.26	0.40	1.15	1.1	2.5	0.2	1.1	0.7
Moisture . . . . %	18.7	5.7	5.7	16.4	14.3	18.0	5.7	64.3	26.0
Contents in the lumps of									
C . . . . %	0.9	2.1	1.1	0.8	0.3	1.7	1.6	0.5	1.4
P . . . . %	0.2	0.2	0.05	0.08	1.8	0.2	0.14	0.9	0.5
S . . . . %	0.09	0.06	0.35	0.65	0.7	0.5	0.12	0.3	0.4

The variety of ores and fuels which can be treated by the Renn process can be seen from Table 1, which shows ores and fuels of very different origins processed in the test runs. The table also contains, apart from very rich and gangue poor ores—like the hematite (column 1)—ores with a higher content of silica such as the limonite (column 9). The Renn kiln, like the blast furnace, requires an upper limit of 50% Fe

Problems in the Preparation of Metallurgical Coal in Latin America

THOMAS FRASER

INTRODUCTION

Scarcity of good coking coals has presented a major problem in the promotion of steel industries in Latin America; yet there are several local areas widely scattered in South America and Mexico known to have deposits of coking coal. Furthermore, the great profusion of coal beds in the cretaceous and tertiary measures throughout the Andean region, largely unexplored, justify the expectation that undiscovered reserves of coking coal probably exist.

KNOWN SOURCES OF COKING COAL

The approximate geographical locations of several coalfields, known to be producing or to have workable deposits of coking coals, have been indicated on the map of the Latin-American countries (Fig. 1). Although still other areas probably are locally known, this map will serve to show the wide distribution of potential reserves. Of these, the best known are the medium-volatile Sabinas-Rosita coal of Coahuila, Mexico, long used in ferrous and non-ferrous metal-

Fig. 1. Map of Latin America showing location of known coking coal reserves.

in the charge for normal operation. With richer ores, the blast furnace needs the addition of slag-forming materials whereas, in the Renn kiln, the slag produced in the process can very well be used for the dilution of a too high iron content. Operating in this way, which greatly facilitates the management of the process, only so much slag is taken to the dump as the daily worked quantity of ore calls for, and the loss of iron in the slag is very low in this case, while the recovery of iron in the luppen amounts to 98–99%.

On the other hand, ores with a high percentage of silica can also be treated by the Renn process, and the final slag usually contains 55 to 65% SiO_2, in which case much less limestone is needed than by the blast furnace process, so that the Renn process is correspondingly more economical. As the consumption of fuel and the other operating costs for the Renn process are proportionate to the daily worked quantity of charge and not to the iron content of the same, it is of course more economical to treat rich ores—even with returning final-slag—than poor ores, in which the charge very often carries only 25 to 35% Fe instead of 50% with a rich ore.

Columns 2 and 3 show the treatment of a disseminated magnetite, which was dressed for test No. 2 up to 50.6% Fe and 24.1% SiO_2, and for test No. 3 up to 55.9% Fe and 8% SiO_2 by magnetic separation. These tests prove that in the present case it was technically and economically better to treat the rich concentrate, as shown in column 3, by the Renn process, returning 10% final slag from the ore weight, whereas for the poorer concentrate an addition of 20% limestone was necessary. A large plant built on these principles, with an annual capacity of 400,000 tons of ore, confirmed these results and was later on even charged with a concentrate containing about 58% Fe.

Our table shows also the composition of the luppen attained in the individual tests. The carbon content of these luppen, which have been formed by a process of welding below the melting point of iron, lies in general between 0.5 and 1.0% and can be regulated by operating the kiln correspondingly and may even be brought up to 1.5 to 2.0%.

About 70–90% of the phosphorus content of the ore is brought out in the iron, while the manganese content of the ore goes mainly into the slag. Titanium and silicon are not reduced at all. Because of the low working temperatures and the acid character of the slag, it is impossible to bind the sulfur into the slag to the same extent as in the blast furnace so that, in general, one third of the sulfur present in the ore and in the fuel goes into the luppen. Only by using raw materials very low in sulfur, as in test No. 1, by utilization of charcoal fines, and in tests Nos. 2 and 7, by utilization of a low-temperature coke, poor in sulfur, is it possible to produce luppen with about 0.1% S or less, which can be used without restrictions as synthetic scrap in every type of steel furnace. When using fuels with the usual S content, luppen with sulfur contents from 0.5 to 1.0% S will frequently result. It is indeed possible in such cases to bring about a stronger slagging of the sulfur by a corresponding composition of the slag, and eventually by raising its quantity. Reference is made, for instance, to test No. 6, where a low-temperature coke, even with 2.5% S, was used and where nevertheless, in consequence of the manganese content of the charge and therefore of the slag too, luppen with only 0.5% S were produced; in the case of test No. 8, owing to an alumina and alkali content of the slag, luppen with 0.3% S were produced. It seems obvious that this desulfurizing effect of the slag could in all cases be achieved by corresponding mixtures of the charge and thus luppen could be produced with a maximum sulfur content of 0.1% S, even from raw materials with a high sulfur content. With the low working temperatures of this process and the low carbon content of the luppen this would, however, in most cases result in such an increase in the quantity of the slag that it is more economical to desulfurize the luppen, when treating them for pig iron or steel.

The luppen in their interior are practically free from slag and contain, after the usual crushing and separating, only about 2 to 4% insolubles in the form of adhering slag particles, so that their iron content is about 94 to 95%.

The Renn process is not only suitable when iron works are erected, but it can often profitably supplement existing blast furnaces or electric furnace plants. For these processes, in which the gases are led through the charge, only lumpy ores and fuels can be used, so that the ore fines have to be sintered or pressed into briquettes. Fuels like coke breeze or charcoal fines are unsuitable for these furnaces and may only be used for sintering. In such cases it may be more economical to treat the ore fines, together with the fines from the fuel, by the Renn process instead of sintering them. The resultant luppen can be used as synthetic scrap in a steel plant or can be re-melted additionally to pig iron by the blast furnace or by the electric furnace with a very low consumption of fuel. When running the tests in the table and also in the operation of the production plants, fuels of very different types have been used as reduction agents. It is to be understood that about 80 to 90% of the heat necessary for the process is supplied by the reduction fuel and 10 to 20% by heating from the discharge end of the kiln. The reactivity of the carbon is, with all fuels in question, sufficient, so that for the running off of the process it is immaterial whether anthracite, coke breeze, charcoal, low-temperature coke, crude coal or lignites are utilized for reduction. The reduction fuel and the ore—both of fine size—are mixed together, so that the total charge contains about 12 to 18% fixed carbon, the balance consisting of ore and fuel ash. It is, of course, unimportant whether the proportion of the fuel ash is somewhat greater or smaller, so that for this process low grade fuels with high ash contents can be utilized technically and usually also with an economic advantage. The really effective factor in this process is the content of fixed carbon, as the hydro-carbons out of the coals and lignites are decomposed in the pre-heating zone and the main part of them is driven off without being utilized. The Renn process itself is not influenced by this driving off of the volatile matter as the waste gases contain a sufficient quantity of heat to enforce this endothermic reaction.

A possibility arises of making use of the gas content of a crude coal for the Renn process if about two thirds of the necessary reduction fuel is blown in from the discharge end of the kiln directly into the luppen zone and if only one third of the reduction fuel is mixed with the ore before charging the furnace. Experience shows that only one third of the total reduction matter is required for the initial formation of the sponge iron, while two thirds is consumed in the luppen zone for the direct combustion at the surface of the charge and for the re-reduction of the iron. If particles of coal, 1 to 3 mm. in size, are screened out of the total reduction fuel and blown into the luppen zone together with the necessary quantity of air, then these particles of fuel fall into the luppen zone and are mixed into the charge, so that their gas content is burnt inside the luppen and can thus be utilized for the Renn process.

On the other hand, with higher contents of volatile matter in the coal, with high fuel prices and in larger Renn plants, it may be of advantage to treat the fuel at first by a separate process of low-temperature carbonization, so that low-temperature char is fed to the Renn kiln as reduction fuel and, from the gases of the low-temperature carbonization, liquid products or power may be produced.

In the main, two types of kiln have been used for production plants using the Renn process: the first is a 60–80 m. rotary kiln with a diameter in the shell of 3.6 m. and a capacity of about 250 to 350 tons of ore per day. The other is a rotary kiln with a diameter in the shell of 4.2 m., a length of 70 to 100 m. and a capacity of 500 to 750 tons of ore per day. In special cases it may sometimes prove to be economical to erect smaller kilns with capacities ranging from 100 to 200 tons of ore per day. On the other hand, it is possible today to build kilns with diameters in the shell of 4.6 or 5.0 m. and capacities of 1,000 to 1,500 tons of ore per day. It is advisable to select the kiln sizes for larger plants so that in the end at least two, and preferably three, kiln units are working and production may proceed at the rate of 50 or 66% in case of repairs to one unit.

When transferring the Renn process from a small pilot plant to a producing plant the selecting and testing of refractory materials played an important role. It was found that for the pre-heating and reduction zone—i.e. for about 80% of the kiln length—a normal refractory brick is well suited, having in these zones with comparatively low temperatures, a life of several years if a good job of brick-laying has been done. In the luppen zone—i.e. for about 20% of the kiln length—the attack on the kiln lining with the prevailing higher temperatures is also of a chemical nature and therefore greater. During longer periods of operation in our first two production plants a natural quartzite, magnesite-bricks, colhart-bricks and several other refractories have been tested. Technically and economically the best proved to be quartz-containing bricks, poor in alumina, containing about 70 to 75% SiO_2 and 20 to 22% Al_2O_3, if they have a dense structure. As a slag from the Renn process usually contains 55 to 65% SiO_2, this composition corresponds to that of the fire-bricks and the chemical attack is small. The slag cannot penetrate into these fire-bricks having a dense structure and a small porous volume and therefore the destruction, starting from the pores, will also be negligible and thus the lining of the luppen zone will have a life of six to eight months. Therefore the lining requires only 30–50 days for repairs per year for each kiln. Maintenance costs for the kiln-lining are thus kept within reasonable limits.

A Renn plant always comprises three individual plants:

(1) A plant for the preparation of the raw materials where ores and fuels are crushed when necessary, weighed and mixed with each other;

(2) The Renn kiln with the device for additional heating by pulverized coal, fuel oil or natural gas;

(3) The luppen dressing plant with ball mills, screens and magnetic separators.

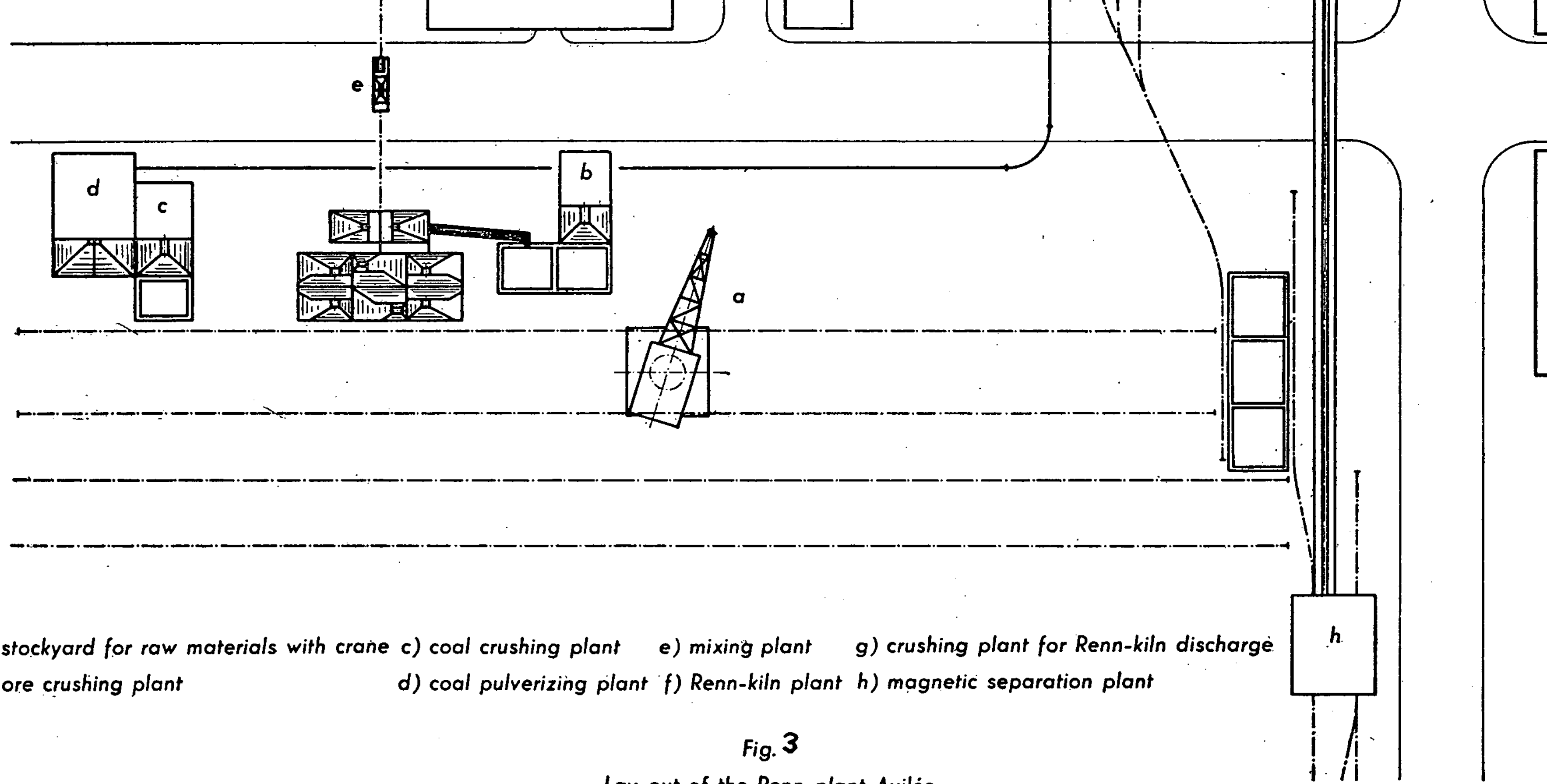
According to the size of the plant and the local labour conditions, the plant may be built quite simply with intermediate transport facilities by cars or completely mechanized, by using belt conveyors or cranes.

Figure 3 shows the lay-out of a plant at present under construction in Spain for the treatment of an ore with 38% Fe and 24% SiO_2, where an anthracite of high ash content will be used for reduction. The plant consists in the first stage of one kiln with a daily capacity of 250–300 tons and can be enlarged later on to four kilns without changing the stockyards for raw materials. The ore will be delivered to the plant by railway from a mine in the neighbourhood and after crushing at (*b*) the ore is dumped by a travelling crane to stockyard (*a*). The anthracite in sizes ranging from 0–15 mm. will also be delivered by railway and is brought immediately to stockyard (*a*). The quantity of anthracite required daily is taken by the travelling crane to the crushing plant (*c*) and is from there, together with the ore, brought by the crane into bins of one day's capacity. The quantities of ore, anthracite and magnetic concentrate are withdrawn from the bins by a weighing car which transports the individual batches to the mixer, whence they are continually charged to the Renn kiln (*f*). The kiln discharge is chilled by spraying with water on a cooling conveyor and drops into tilting cars which transport it to the crushing plant (*g*). Here the slag is crushed down to 2 mm. and the slag and the luppen are re-taken together, by conveyor, to the plant for screening and magnetic separation (*h*), where they are separated by screening and magnetic separation into luppen, magnetic concentrate—which is returned to stockyard (*a*)—and into final slag. At (*d*) is the plant for pulverizing coal, which is required for the additional heating of the kiln.

The initial costs for a Renn plant depend naturally, to a large extent, upon the degree of mechanization, the local conditions and the number and size of kilns. With the present world market prices for mechanical installations and buildings, they amount to about US$20–30 per ton of annual ore capacity. With 50% Fe in the charge and with a rate of 15% for depreciation and interest on capital invested, this results in total capital costs of US$6–9 per ton of iron while for a newly built blast furnace plant, under the present conditions, capital costs of US$10–15 per ton of pig iron have to be taken into consideration.

The production costs of the Krupp Renn process are mainly determined by the iron content of the ore and by the type of fuel and its price. As the quantity of fuel is influenced to only a minor extent by the iron content of the ore, and all other items for wages, power, kiln maintenance and general repairs are directly proportionate to the quantity of ore charged to the kiln, the reduction costs per ton of luppen are naturally lower for rich ores than when treating ores poor in iron.

The addition of fluxes for the formation of a slag suitable for this process is naturally small with rich



a) stockyard for raw materials with crane c) coal crushing plant e) mixing plant g) crushing plant for Renn-kiln discharge
b) ore crushing plant d) coal pulverizing plant f) Renn-kiln plant h) magnetic separation plant

Fig. 3

Lay-out of the Renn-plant Avilés

ores. In general, the gangue of these ores consists mainly of silica and immediately forms a suitable slag, the quantity of which has to be regulated by returning final slag. An ore containing a surplus of, say, 5% SiO_2 requires an addition of limestone of only 3–4%. The carbon content effective in the mixture of ore, return slag and fuel should amount to 12–18%, whereby, as effective carbon, the fixed carbon content should be taken plus about one quarter of the carbon content of the volatiles. An ore with 60% Fe, considering also the quantity of return slag, means a fuel consumption of 500–600 kg. coke breeze, anthracite or charcoal fines per ton of luppen. To this must be added the fuel consumption for the additional heating of the Renn kiln from the discharge end, with 75–100 kg. pulverized coal per ton of luppen or a corresponding quantity of fuel oil or natural gas.

The number of man-hours for the conversion of ore into luppen depends on the size of the kilns and the state of mechanization in the plant, especially that of the transport facilities. They amount to about 1.5–4.0 man-hours per ton of luppen produced with an ore of 60% Fe and include the amount of man-hours in all individual units of the Renn plant.

The power requirement for the stockyards, transport facilities, the crushing plant for ore and fuel, the Renn kiln, the crushing of the kiln discharges and the magnetic separation, amounts to 50–80 kWh per ton of luppen with ores of the aforesaid composition.

According to experiences in various large plants, the maintenance of the total kiln-lining will require about 8–12 kg. of fire-bricks per ton of luppen, which will cost at present about US$1. In addition, there are the costs for auxiliary materials of other kinds and for repairs to the mechanical installation amounting to about US$2.50 per ton of luppen. The total conversion costs, without the costs for procuring ores and fuels, will then amount, subject to certain alterations by local conditions, to about US$6–8 per ton of luppen. Calculating the quantity of ore required for the production of one ton of luppen with US$6 and taking the requisite quantity of fuel, i.e., 600 kg. of fuel per ton of luppen at US$10 per ton of fuel, the production costs will amount to US$18–20 per ton of luppen. By adding US$1 for administration and management and US$7.50 for depreciation and interest, the total production costs will result in US$26.50 to US$28.50 per ton of luppen with about 95% Fe.

This price for luppen calculated in the foregoing example can differ only slightly in an individual case with respect to conversion costs and depreciation. It may be influenced materially by alterations of the supposed prices for ores and fuels and may diminish or rise accordingly. The price thus computed for luppen should then be compared with the price for commercial scrap of good quality, as the luppen may be considered as a synthetic scrap, possessing in comparison to commercial scrap the advantage of a uniform and known composition by the absence of steel alloy elements of every kind. But they have frequently the disadvantage of having a higher sulfur content.

If the Renn kiln is co-operating with blast furnaces or electric furnaces, for instance as a plant for treating the ore and fuel fines, then it may be most expedient to charge the luppen additionally to the burden of these furnaces. Since the capacity of these furnaces depends on the possible conversion of energy in the form of coke or electric power, and as for the remelting, carburization and de-sulfurizing of the luppen only one third of the energy is required, as compared with that necessary for reduction from ore, the production capacity of these furnaces is raised considerably by the addition of luppen to the burden and may reach 150–160% of the normal capacity, according to experiences in German blast furnace plants.

As a rule, however, the luppen will represent a preliminary material for the production of steel. Charged to electric steel furnaces they are to be compared with scrap. Because of their dense accumulation, they melt without major fluctuations of the current and also owing to their somewhat high carbon content they melt more easily and with less electric power consumption. On the other hand, a carbide slag with good desulfurizing properties must be used with sulfur-containing luppen, and therefore, as to costs, the conversion of luppen into steel corresponds to approximately that of scrap, whereby the luppen will produce qualitatively a high-grade electric steel owing to the absence of alloy elements in the charge.

The direct conversion of luppen into commercial steel in the open hearth furnace is equally possible. In this case, the luppen replace commercial scrap either partly or altogether. In test runs, high sulfur-containing luppen have been worked at a 75% proportion of the charge, bringing the total sulfur content of the charge up to and above 0.5% S. The elimination of the sulfur was effected by one or two extra slags high in lime, which had to be removed during the cycle time of the charge. For this kind of work, therefore, tilting open hearth furnaces seem to be best suited. But the aforesaid test runs were not done in tilting furnaces and resulted, even under these conditions, in only a negligible prolongation of the cycle time and a corresponding rise in costs. It was found that the elimination of sulfur in the open hearth furnace was simpler than had been supposed, because it is easier for the steel man to eliminate a known sulfur content—even if it is high—slowly and continually during the whole run of the cycle time of the charge than to remove, by a special subsequent refining, a sulfur content only ascertained at the end of the cycle.

The luppen may also be used in the open hearth furnace partly as a carbon-carrying material, that is instead of pig iron. In such a case they will be re-melted in a hot-blast-cupola or in a rotary kiln, while simultaneously raising their carbon content to 2%, for example, and desulfurizing the luppen in the furnace or subsequently in the ladle. This re-melted iron is then charged in a liquid state into the open hearth furnace, which can be charged besides with cold luppen and/or scrap. On the other hand, this re-melted iron with low carbon content can be converted into steel, without further additions of other materials, by the pig iron-ore process. The re-melting of the luppen in the hot-blast-cupola requires about 120 kg. coke, and in the rotary kiln 180–200 kg. pulverized coal, or a corresponding quantity of fuel oil or natural gas. Naturally, both types of furnaces for the re-melting of luppen allow also of the production of commercial pig iron in plants that have no blast furnaces.

For the further processing of luppen to steel or pig iron—as shown—quite a number of possibilities exist, the most economical being discernible in each case only after a thorough examination of all conditions in question. In some cases—not only in the further treatment of especially low sulfur-containing luppen—no extra costs arise as against processing commercial scrap. When they do arise, possibly by desulfurizing, such extra costs are, according to our experiences in the past, of an order of less than US$2.50 per ton of luppen.

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Summary of Discussion

Considerations on the Development of the Production of Iron, ***presented by the author***

Mr. Durrer introduced his paper by stressing humanity's dependence on iron. World production of iron, including steel, had reached the figure of 250 million tons a year. Apart from iron ore, the resources of manganese, aluminium, copper, zinc and lead were being exploited in an ever-increasing volume. Therefore international co-operation in the conservation and proper utilization of those resources was of paramount importance. When one considered that the industries consuming such resources were not quite a hundred years old, one realized with alarm that better methods of conserving raw materials had to be devised in order to escape the censure of succeeding generations and to preclude the exhaustion of existing deposits.

The implications of the meeting were therefore manifold and far-reaching and the gathering of such a group of international experts in the iron and steel industry constituted a positive step in the right direction.

If past and present methods of iron and steel production were to be continued, the existing deposits of iron ore, or metallic iron, could only be expected to last another six or seven hundred years. He emphasized that the reserves on which the preceding statement had been based represented merely an estimate, and that no one could foretell the pace of iron ore consumption for even the next few hundred years.

The second raw material for the production of iron was carbon which was used for the reduction of the iron ore and for producing heat. The estimate in respect of coal was more encouraging, for if coal consumption were based on the same principles adopted for calculating the estimate regarding iron ore, it seemed likely that existing coal resources would last some three thousand years. However, the estimate should be regarded with caution, in view of the fact that certain deposits might be difficult, or even impossible, to mine.

He strongly advocated the use of coal as a chemical rather than as a fuel. If some other material could be utilized in the blast furnace, not only would coal resources be conserved but the smelting process would not suffer from the effects of the impurities brought into the furnace by the introduction of coal. Electric energy was, of course, one solution, but electric power might be either costly or unobtainable in certain countries. Furthermore, it would not be practical to abandon blast furnaces which had been installed at considerable expense. Nevertheless, hydro-electric energy offered great hope to mankind and the development of electric and other new processes would definitely be a blessing to the iron and steel industry of the world.

Mr. Marin Gonzalez opened the general discussion of the paper by expressing his appreciation of the point so eloquently expounded by Mr. Durrer. Mexico was very conscious of the importance of conserving and properly utilizing raw materials and a movement had been launched with that aim in view. The Banco de México, the organization with which he was associated, had set up a department for the study of the conservation of resources. Although the Mexican iron and steel industry was relatively young, Mexico was as keenly interested as the more experienced countries of the world in co-operating in the vast international plan recommended by Mr. Durrer.

Mr. Waring expressed his support of the views presented by Mr. Durrer. He wished to reiterate some of the comments made by Mr. Durrer in respect of the conservation and utilization of raw materials, and to stress that the problem was a universal one and not limited to Latin America. In 1951, 230.5 million tons of iron and steel had been produced and the estimate for 1952 was 266 million tons. The sharp rise in production demonstrated by those figures was expected to continue year by year and that factor posed a problem for the whole world. That problem was particularly critical for Europe in view of the fact that its own resources were so limited that coking coal had to be imported from the United States. The general shortage of coking coal might register certain repercussions on the current methods of producing iron and steel. One solution of the problem might be the development of a process which would utilize non-coking coals, in which event the tremendous resources of low-grade coals in the Far East and other countries of the world might be used to good advantage.

Although he had no concrete suggestion to offer that would lead to the development of a process not requiring coking coal, he was sure that a solution to the problem would be greatly facilitated by concerted international effort rather than by independent endeavour.

Notes on the Production of Pig Iron, *presented by the author*

Mr. Bulle, after presenting his paper, referred to an idea advanced by Mr. Durrer involving a change in the position of the shaft of the furnace, saying that he did not agree with that theory. Theoretically, the low-shaft furnace was capable of utilizing soft coke, lignite or bituminous coal. That theory was proved by experiments in Germany and as many as 20 furnaces in Eastern Germany were actually being built for, or were operating with, soft coke. That figure was not, of course, applicable to Western Germany, where only one experimental furnace was in operation.

Mr. Gonzalez Vargas opened the general discussion of Mr. Bulle's paper by observing that there seemed to be some controversy on the relative merits of the low-shaft furnace and of the blast furnace. In his opinion, the two types of furnace complemented each other and he was not in agreement with the tenor of the opinions expressed at the meeting advocating the abandonment of one or the other type of furnace. In his view, the main problem seemed to be the development of a process which would utilize material not digestible in the blast furnace. The fundamentals of the process had already been established and what was needed was the perfection of existing processes rather than the introduction of new ones.

Mr. Kalling observed that it might be possible that low-shaft furnaces in the future would prove to be a good substitute for the ordinary blast furnaces in those cases where good coking coals were not available. It was, however, difficult to obtain smooth operation with a soft and fine-grained burden even in furnaces with a very low height. In the electric low-shaft furnace, for instance, where the height of the burden was very low and the gas quantity that had to pass the burden was only about a fourth of that of the blast furnace, it had been found necessary to have a well-prepared, not too fine-grained burden to get satisfactory results, and the same would certainly apply even more in the case of low-shaft furnaces, though the conditions could be made more favourable by using oxygenated air instead of ordinary air.

Another disadvantage with the low shaft was the limited size of the furnace that could be used. To obtain smooth operation with an evenly distributed gas in the shaft it would be found necessary to choose a hearth diameter that was not more than one half to one third of the height of the burden; in other words, a diameter of not more than 1 m. at a shaft height of 2–3 m. When wishing to build a furnace of a more commercial size, say for instance with a hearth diameter of 4 m., the burden height would certainly have to be increased to about 10 m. to get a well-distributed gas flow. But such a furnace could no longer be called a low-shaft furnace, nor could it be expected that such a furnace could be used for a very soft burden. In that connexion, it might be interesting to note that the coke blast furnaces at Domnarfvet in Sweden, having a hearth diameter of 4.5 m., had a burden height of not more than 13 m. According to the foregoing they could be called large, suitably dimensioned low-shaft furnaces. It had been found, however, that all attempts to use a softer coke than ordinary blast-furnace coke in those furnaces had led to unsatisfactory results.

Generally speaking, it was his view that the existing height of the blast furnace could be decreased successfully, but only to a certain degree. The problem lay in making coke suitable for a medium-height blast furnace from coals thus far classified as non-coking.

Mr. Allard observed that experiments on the low-shaft furnace would certainly benefit from the use of oxygen and other new elements. Mr. Saniter had mentioned that considerable research was being conducted on the possibilities of the blast furnace and he hoped that the results would be encouraging.

Mr. Sem remarked that Mr. Bulle had commented on the rotary kiln process, stating that it had not yet been perfected. Perhaps Mr. Bulle had been referring to the fact that the rotary kiln was not used for the production of pig iron, since the rotary kiln process was in fact used in electric and other types of furnaces. The question seemed to be to what extent that process could be utilized in order to obtain a finer and less costly charge in the production of pig iron.

Mr. Buehl stated that his experience with the U.S. Bureau of Mines had not led him to the same conclusions as those expressed in the generalized statements made at the meeting. The Bureau of Mines had for several years been studying the recovery of manganese for blowing into a basic converter to produce slag. For a period of about four months the Bureau had operated an experimental blast furnace which was in many ways actually a low-shaft furnace. The coke rate obtained had been as low as 3,200 lb. per ton spiegel, and 3,000 to 5,000 lb. slag per ton metal. There was a large proportion of heat loss with a very high blast temperature. Since the original experiments, many thermal calculations had been made and it had been concluded that the blast furnace contained several zones with specific heat requirements. In his opinion, if more ore per unit of coke were added in a smoothly operating furnace and the temperature of the blast were increased, the top temperature would fall and the furnace would become thermally more efficient. Consequently, in making a comparison between the high-shaft and the low-shaft furnace, consideration should be given to what had to be kept constant and what had to be varied. High hot blast temperatures would be much cheaper than oxygen enrichment.

With regard to the coke strength requirements, he did not see the necessity of a particularly high furnace. Even a short furnace required a thermally stable fuel. While he was in favour of the low-shaft furnace, he doubted whether it would ever assimilate poor raw materials.

Mr. SANITER commented on the mixing of powdered ore with powdered carbon as proposed by Humboldt. Would it not be possible to mix carbon with pure concentrates and to charge the resultant briquettes into the Talbot furnace, so that the amount of direct reduction from ore to steel might be increased?

International Research on the Low-Shaft Blast Furnace, *presented by the author*

Mr. ALLARD commenced his presentation by informing participants that he represented the International Low-Shaft Blast Furnace Research Committee, which was an international organization formed by the OEEC. His paper dealt with the experiences of that Committee since its inception. The further development of the low-shaft blast furnace was prompted by a desire to utilize the low-grade raw materials available as fuel.

He made special mention of the fact that the United Nations Economic Commission for Europe had been co-operating with his Committee in experiments aimed at improving the low-shaft furnace. He emphasized that the Committee was more interested in concrete results than in the theoretical aspects of the problem.

The CHAIRMAN, at the conclusion of Mr. Allard's presentation, asked him whether any of the Latin-American countries essentially interested in the low-shaft blast furnace had been invited to participate in the Committee organized by the OEEC.

Mr. ALLARD replied that although the OEEC was a European organization, in the sense that it concerned itself with European problems, he understood that certain American countries had been invited to participate in the studies mentioned. Peru, Brazil and the United States were among those so invited.

Mr. WARING thought it opportune to make certain clarifications in respect of the connexion of the OEEC with the United Nations. The OEEC was an independent organization comprising certain Western European countries and was not an organ of the United Nations.

Mr. GONZALEZ VARGAS commented on the Humboldt experiments on the low-shaft furnace, as well as those conducted at Oberhausen. Certain other processes, however, had not been mentioned by any of the previous speakers. One of those was the Gottenberg process which endeavoured to obtain pig iron from the low-shaft furnace not with briquettes but with small-size coal. That process had not yet been published but he understood that publication was in progress.

It had been affirmed at the meeting that the low-shaft blast furnace was not very efficient. He wished to point out, however, that it had been proved that indirect reduction in the low-shaft blast furnace could reach the desired magnitude if conditions were suitable. Experiments with low-grade material in Italy and Germany had been very successful.

Mr. LANARI asked the author, first, what the construction details of a low-shaft furnace would be at pressures of 2–2½ kg. per sq. cm.; second, with reference to the statement that duration of passage was two hours, he wondered what reasoning lay behind the adoption of that type of passage. Obviously, the fuel consumption must be high in the low-shaft furnace, but in Brazil there were small low-shaft furnaces which produced 800 kg. of iron per hour. The fuel consumption seemed to be influenced by the percentage of indirect reduction in the blast furnace.

Mr. ALLARD, in reply to Mr. Lanari's questions, said that the low-shaft furnace in question was of small dimensions. From the point of view of thermal losses, a small furnace was desirable.

As to the duration of passage, the problem had not been approached from as logical a standpoint as Mr. Lanari had taken. The dimensions of the apparatus seemed to govern the time of passage, and the two-hour passage cited was based on experiments.

Mr. CAVANAGH stated that it was indeed regrettable that Canada had not co-operated in the programme sponsored by the OEEC in respect of the low-shaft blast furnace.

Mr. CORTÉS OBREGÓN remarked that he thought that all participants were convinced that the meeting would lead to positive results. He offered the suggestion that a Pan-American Institute should be organized on the lines of the IRSID so that the American continent might also benefit from the interchange of ideas and recommendations that would emanate from such an institute.

The CHAIRMAN expressed his support of the recommendation made by Mr. Cortés Obregón and assured him that it would be considered at the opportune moment.

Mr. COHEUR associated himself with the views expounded by Mr. Allard and hoped that they would serve to answer many questions that had been raised.

The low-shaft furnace being constructed by the Centre national de recherches metallurgiques in Liège was expected to be completed by the end of 1952 and consequently there was still time for those countries that were interested to join the International Committee.

Mr. BAUM remarked that he had drawn the conclusion from the morning's discussion that the main problem seemed to be the scarcity of coking coals, a problem which interested him in view of his concern with coking processes. He thought that a simpler process had to be devised for the manufacture of pig iron.

Mr. WARING observed that the United Nations had three regional commissions devoted to the study of economic and allied questions: ECLA (Economic Commission for Latin America), ECAFE (Economic Commission for Asia and the Far East) and ECE (Economic Commission for Europe). Other organizations with no connexion whatsoever with the United Nations were the OEEC, the Schuman Plan, NATO and so forth.

The Krupp Renn Process, *presented by the author*

Mr. PRADO UCHOA opened the general discussion of the paper under review by stressing Brazil's interest in the Krupp Renn process, because of the poor quality of Brazilian coals.

He wondered whether Mr. Johannsen had any suggestions on the utilization of low-grade coals with rich ore, as there was hardly any literature available on the subject.

He also requested information on the possibility of installing an integrated plant based on the Krupp Renn process, and on a new Japanese process developed for the purpose of eliminating sulfur from luppen.

Mr. JOHANNSEN, in reply to the first question raised by Mr. Prado Uchoa, stated that the charge usually applied under the conditions cited contained a total amount of 12–18% of fixed carbon. A fuel with a 30% ash content could be used with rich ore.

With regard to the second question, he observed that a very simple plant could be installed for the manufacture of electric steel. For an open hearth furnace which was used for carbon steel, it would be advisable to remelt 20% of the luppen before placing it in the open hearth, charging the open hearth with scrap and solid luppen.

As to the Japanese patent mentioned, he had no knowledge of the subject.

Mr. GONZALEZ VARGAS stated that Mexican ores were rich in iron but poor in silica content. He understood that the Krupp Renn process was only suitable for ores rich in silica, and wished to know whether the artificial addition of silica would have an unfavourable effect on the process's thermal efficiency. He had also been informed that the Krupp Renn process had certain refractory difficulties and would welcome Mr. Johannsen's comments on that question.

Mr. JOHANNSEN observed that his paper had dealt with the treatment of an ore with a very low percentage of silica. Normally a certain amount of silica was found in ash; it was not necessary to add any lime. In his experience a slag of 55–60% silica was obtained. With a rich ore, 20–25% of fine slag had to be added, but the returning slag did not affect the heat efficiency if no silica was added.

On the question of refractories, the problem would be solved in the first commercial plant. Many tests had been made with several types of refractories, and with temperatures of 1,100°C., any kind of brick suitable for a rotary kiln could be used. For temperatures of 1,300°C., tests were conducted with normal firebricks and other types which were low in aluminium and high in silica.

Mr. SHEROVER asked whether any experiments had been conducted on the Krupp Renn process with natural gas and what the cost of sponge iron produced by that process had been over the past 20 years.

Mr. JOHANNSEN replied that the Krupp Renn process could not work with natural gas alone, but it was possible and economical to mix carbon powder with oil for the charge.

With regard to the second question, there were some cost data in the paper to answer it.

Mr. ALLARD wondered whether Mr. Johannsen had ever experimented with the reduction of phosphorus as there was no melted iron containing slag in the Krupp Renn process.

Mr. JOHANNSEN replied that the reduction of phosphorus depended entirely upon the reduction of iron. In the Krupp Renn process, a slag of 3–4% iron would be obtained. Of course, it was possible to work with a lower percentage of iron ore reduction. On occasion, it was found that the size of the luppen could be regulated. It was just a question of time and of fuel.

IRON ORE REDUCTION

Methods for reducing ores otherwise than in blast furnaces (*concluded*)

22 October 1952—Afternoon

Chairman:

Alfonso BALLÓN, Director del Departamento de Siderurgia, Corporación Peruana del Santa, Lima, Peru

Contributed Papers:

Production of Sponge Iron According to the Wiberg-Söderfors Method
John STALHED, Söderfors, Sweden

The Basset Process for the Production of Pig Iron in Rotary Kilns
Knud E. JENSEN, F. L. Smidth & Co. A/S., Copenhagen, Denmark

The Rotary Kiln Processes for Sponge Iron, Developed at the Avesta Iron and Steel Works and the Domnarfvet Iron and Steel Works, Sweden
Bo KALLING, Director of Research, Stora Kopparbergs, Bergslags Aktiebolag, Domnarfvet, Sweden

Production of Sponge Iron in a Rotary Kiln at Temperatures Below the Fusion Point of the Material
Russell C. BUEHL, Chief, Ferrous Pyrometallurgy Section, Bureau of Mines, Region VIII, Minerals Technology Division, Pittsburgh, Pa., U.S.A.

Summary of Discussion:

Participants: Messrs. KALLING, CAVANAGH, YDSTIE, CANGUILHEM, MERCIER, ALLARD, GONZALEZ VARGAS, DURRER, BUEHL, PRADO UCHOA, GARZA SADA, JOHANNSEN, SADA

Production of Sponge Iron According to the Wiberg-Söderfors Method

JOHN STALHED[1]

Though a comparison between the various sponge iron processes would be interesting, only the Wiberg-Söderfors method will be dealt with here.

Martin Wiberg began to devote himself to problems of reduction while still a student at the Royal Institute of Technology in Stockholm, and his graduation thesis presented in 1917 dealt with the reduction of iron ores. In 1918 he invented the sponge iron process now designated by his name (Swedish patent No. 46507, 1918). It is noteworthy that the calculations made at that time by the young engineer have been confirmed in conditions of full-scale production today.

The theory of the Wiberg process may be outlined as follows: Figure 1 sketches the process as performed for the present. Reduction gas, consisting of carbon monoxide and hydrogen in ratio 2:1, is generated in an electrically heated carburettor, in which the gas fed back from the reduction furnace shaft is brought to react with the fuel. Principally, this fuel should consist of carbon and hydrogen in one form or another.

On flowing through the final reduction zone at the bottom of the shaft, the gas reduces the wustite to metallic iron. The portion of gas that is not carried off to the carburettor flows through the primary reduction zone, reducing the ore from ferric oxide to ferrous oxide. At a higher level in the shaft air is introduced through nozzles to effect the burning of the remaining combustible gases, thus heating the ore to an adjusted temperature, i.e. usually to 950°C.

During the process more gas is generated than is needed to heat the ore, as can be seen from Figure 1. In order to prevent an excessive temperature rise in the combustion zone a certain amount of gas is shunted before entering it. This gas off-take must be at a sufficiently high level in the shaft in order to

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[1] Translation of an article appearing in *Stahl und Eisen* 72 (1952), pp. 959–966; reproduced by kind permission of the author.

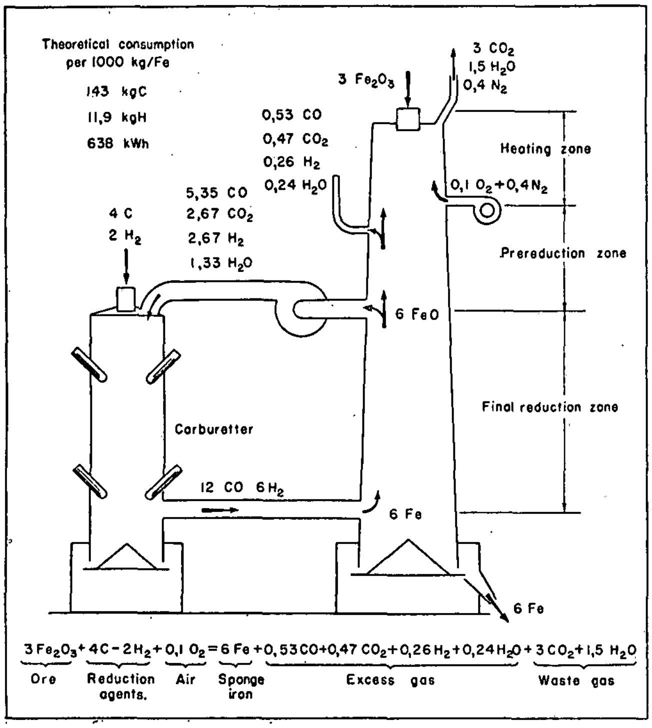

Figure 1
The theoretical principle of the Wiberg process.

utilize the reducing capacity of the gas as long as possible before its exit from the furnace.

During the period 1920–30 trials in pilot scale were carried out at various localities in Sweden, namely Woxna, Sandviken and Domnarfvet. Experiments were also performed in Spain.

The first kiln operating on an industrial scale was erected in 1932 in Söderfors but this plant was not run continuously until 1941. This delay was partly due to the specific requirements on constructional details, primarily on the circulation fan and the carburettor.

The fan is to convey reducing gases of a temperature of 700 to 800° C. The impeller of the present fan is made of a heat-resistant steel of 18% Cr, 20% Ni, and 1% Ti.

Larger fans are under construction and will be equipped with impellers of a gas turbine alloy N 155 with the following constituents:

C = 0.1%
Si = 0.5%
Mn = 1.5%
Cr = 20.0%
Ni = 20.0%
Co = 20.0%
W = 2.5%
Mo = 3.0%
Nb + Ta = 1.0%
N_2 = 0.12%

The carburettor is the heart of the plant. A reduction gas of required composition and uniform temperature is of greatest importance for the performance of the process. In this respect difficulties were encountered with earlier designs, as will be described further on. In the initial trials as well as in the original plant at Söderfors charcoal was used as the carburetting agent, among other reasons because it has a very low sulfur content. Initially a carburettor of the construction shown in Figure 2 was used (Swedish patent No. 121927). This type was equipped with one 600 mm. top electrode and one bottom electrode. The circulation gas to be carburetted entered through openings near the bottom electrode and the carburetted gases were withdrawn at the top of the producer. This design is remarkable in regard to the difficulties originating from it. The electrical conductivity of charcoal increases very markedly with temperature. As the comparatively cold circulation gas is blown towards the inter-electrode zone of the circuit, the conductivity becomes reduced on the inflow side thus dislocating the current flux towards the other side. The consequence is that current flow will take place where a minimum of gas permeation occurs. The temperature there will rise and cause defects in the brick lining, whereas the gas at other points will pass insufficiently carburetted. A further carburation of the gas will take place in the narrow waist section of the carburettor, causing an additional consumption of electrode as it will take part in the carburation.

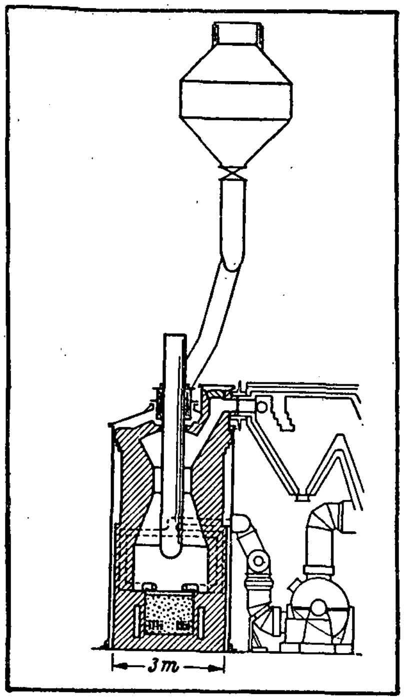

Figure 2
One-phase carburettor.

These difficulties were overcome by the introduction of an entirely new carburettor put into operation in November 1941 (Figure 3). The gas flow in this producer is parallel to that of the charcoal, i.e., directed from above downwards. This arrangement enables the charcoal moisture to pass through the hot reaction zone and become carburetted. The electric current is fed via six electrodes entering through the roof and producing a hot zone at the centre of the shaft. The circulation gas flows into the roof through a hole in the centre, descends to the space between the roof and the charge, and flows essentially along the walls downwards to envelop the central hot zone. The final carburation of the gas takes place in the lower constricted part of the carburettor. This type of construction resulted in a satisfactory performance during very long periods; thus the same piece of equipment was continuously run for over three years in 1942 to 1945.

lurgy; the Barro Branco and Rio Bonito coals of the state of Santa Catarina, Brazil, now being used at the national steel works in Volta Redonda; and the high-volatile coals of the Lota and Schwager properties in the Arauco Bay area in central Chile, being used at Huachipato.

The Goyllarisquizga coal of central Peru, long used for producing coke for non-ferrous metallurgical industries, has not actually been proved by commercial use in steel works but is known to be of coking type. At the many other locations in the Andean area, central Peru, the Oyón region on the western side of the divide in northern Peru, the Cauca Valley of western Colombia, the Lake Maracaibo region of western Venezuela, and many locations in the central Sabana region of Cundinamarca and Boyacá in Colombia, analyses and laboratory carbonization tests have disclosed many potential sources of coking coal of widely different characteristics as to coking power, free-swelling index and general preparation characteristics. None of these have been sufficiently explored to justify specific statistical data on reserves of coking coal, yet all are of substantial importance in view of the relatively small needs of the national steel industries that will be projected in the near future.

The wide variety of coals that may be found in any of these Andean areas is well illustrated by the analysis of Cauca Valley coal samples shown in Table 1.

Table 1

ANALYTICAL DATA OF SOME SAMPLES OF CAUCA VALLEY COALS

Sample	*Moisture*	*Vol. matter*	*Fixed carbon*	*Ash*	*Sulfur*	*Free-swelling index*
1	2.3	43.1	41.7	15.2	1.2	4½
2	1.5	45.8	46.1	8.1	.7	5
3	2.4	43.8	40.0	16.2	1.7	1½
4	3.4	41.2	41.8	17.0	.5	3
5	1.5	42.6	41.4	16.0	.7	4½
6	2.8	37.8	34.8	27.4	.9	3¼
7	.6	28.6	57.6	13.2	3.1	9
8	.4	30.0	62.3	7.3	2.6	9
9	.7	19.2	66.7	13.4	2.9	1
10	.5	20.7	64.3	14.5	4.5	7
11	.5	25.8	53.1	20.6	3.7	9
12	.7	24.4	38.6	36.3	2.9	6
13	4.7	28.6	58.3	13.2	2.7	9
14	4.4	39.2	53.0	7.8	2.8	8½
15	5.3	41.7	44.0	14.3	2.6	6
16	.9	37.4	39.3	22.4	1.0	5
17	.9	24.3	62.6	12.2	1.9	1½
18	.8	43.8	44.8	10.6	1.7	5
19	4.0	41.3	42.7	16.0	2.6	4½
20	4.6	42.0	44.7	13.3	3.6	5
21	5.1	26.6	68.1	9.3	1.8	4½
22	1.6	17.7	42.6	38.1	1.6	1
23	.8	13.3	81.8	4.1	1.4	–
24	5.1	24.7	58.0	12.2	1.3	–
25	.8	10.7	81.7	6.8	1.0	4
26	1.0	38.4	42.4	18.2	2.8	4
27	.7	36.1	40.8	22.4	7.3	6½
28	.7	41.8	47.7	9.8	3.9	6
29	1.4	24.4	66.9	7.3	2.4	9
30	4.9	23.1	70.9	6.0	.8	6½
31	.4	18.2	73.4	8.0	1.2	7½
32	.7	16.9	62.6	19.8	1.1	1½
33	.4	18.4	76.0	5.2	1.4	1
34	.6	17.9	67.5	14.0	1.0	1
35	.5	17.6	67.9	14.0	.9	1½
36	.5	14.3	71.2	14.0	.9	1

These samples represent run-of-mine production at individual small mines distributed geographically over an extent of about 75 km. in the mountain range that skirts the valley on the west, starting with No. 1 in the high-volatile coking area opposite Yumbo, in the northern environs of Cali and running in sequence to the south as far as the gorge near Suarez. The general trend to lower-volatile content toward the south is not to be taken as at all uniform or dependable in any particular property, since the individual beds, which strike generally north-south, tend to vary in volatile—with many exceptions—from low-volatile on the east to high-volatile in the beds farther west.

These rough generalizations regarding the Cauca Valley coals are not precise enough to guide selection of these coals, but they will illustrate the extremely erratic local fluctuations in the physical and chemical properties that characterize the tertiary and cretaceous coals everywhere.

This erratic variability is a major factor in selecting and preparing coking coals for any steel project in Latin America. It necessitates exploration and testing on a much more intensive scale than is generally contemplated in projecting new coal mining operations in the older coalfields of Europe and the Appalachian region of the United States.

The necessity for intensive exploration to prove reserves of suitable coal in the Andean region is, of course, further aggravated by the rough terrain and extensive faulting; but the grand scale of earth movement throughout the Andean uplift appears to have broken the coal measures into relatively large blocks as compared with conditions found in many other parts of the world where the younger coals are being exploited.

BLENDING

The erratic fluctuations in chemical and physical properties of coal that characterize the Andean deposits put major emphasis on blending as a primary process step in preparing these coals for metallurgical use. The prediction that adequate supplies of coking coal will be found in Latin America must be qualified with the reservation that careful selection of local sources and intensive testing ahead of the mining operations will be needed to ensure a continuous supply. Careful blending of two or more types of coal may often be necessary because of the difficulty of establishing an adequate reserve block of one coal that will produce a good metallurgical coke when carbonized alone.

Local transportation facilities will exert a dominant influence on the final selection of coking coal sources. This is particularly pertinent in the Andean region, as typified by the Sabana coalfields where many coking coal deposits, none perfect in every respect, could have been drawn upon to supply the Paz de Río steel industry. In such a situation, comparative transportation costs may be the dominant factor in the over-all economy of raw-material supply. In the compromise most-economical coal supply, scientific blending of coals from the most strategically located sources may greatly improve technical performance.

In blending two or more different coals to prepare a good coking-coal mixture, the simplest situation involves combining low- and high-volatile coals to attain a balance of expansion properties and bulk densities to obtain a coke with a rapidly burning porous struc-

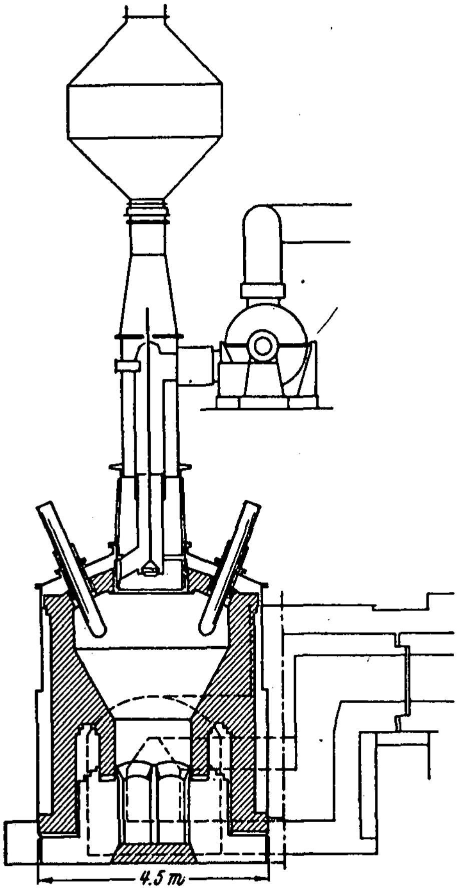

Figure 3
Three-phase charcoal carburettor.

It was nevertheless desired to abandon the expensive charcoal for coke. Trials were initiated in 1949 which led to the carburettor design shown in Figure 4 (Swedish patent No. 134654) and this carburettor has now been in operation for two years.

As can be seen, it differs considerably from previously shown charcoal carburettors. On designing the carburettor the following properties of coke have to be considered: the high ash content, and the electrical properties differing from those of charcoal.

As with charcoal, the electric conductivity of coke increases with temperature, though not at the same rate. The conductivity of coke is consistently higher than that of charcoal in cold as well as in hot condition. Hence a compact furnace vessel cannot be appropriately used as for charcoal, but the carburettor should rather be designed according to the principle of the "conducting burden pillar".

The carburettor consists of a relatively high and narrow shaft through which the circulation gas flows downwards, i.e., in the same direction as does the coke. In the top and bottom parts of the shaft, respectively, three electrodes are introduced. Between each top electrode and its counterpart a current is supplied from a transformer that can be regulated between 110 and 240 volts.

The primary circuits of these transformers are connected in parallel to the same phase of the high-tension line. The carburettor is thus single phase with a current flux in the vertical direction of the shaft. It has been proved in practice that a uniformly distributed temperature can be maintained in the shaft cross section. At the Söderfors plant each transformer is usually run on 150 volts and 2,700 amps which correspond to a total of 1,200 kW. at a cos Phi value of 0.98. Lump coke of 60 to 80 mm. size is a suitable charge. The coke ash is withdrawn by a rotary discharge table. In order to avoid disturbances from the coke ash, 25% of the charged coke weight has to be withdrawn at the bottom of the carburettor. The mixture of ash and coke is screened and the fraction above 30 mm. returned to the carburettor. In this way a coke efficiency of 94 to 96% is achieved.

Figure 4 shows a materials balance of the process with coke as carburetting agent, referring to 1 metric ton of sponge iron and relating to present normal operation at Söderfors. A substantially heavier burden will subsequently be employed so as to increase the Fe content of the sponge iron to 90%.

The sulfur content of the coke is largely gasified and appears mainly as hydrogen sulphide in the gas, which consequently must be cleaned. This takes place in a sulfur filter consisting of a shaft filled with dolomite in suitable lumps. The producer gas enters by an annular orifice in the lower part of the filter shaft and moves upwards through the dolomite layer to the exhaust via another annulus higher up. The dolomite in the top part of the filter shaft is heated and calcined from the latent heat given off by a certain portion of the gas that flows further up the shaft (Swedish patent application No. 8719/1951). This gas is re-fed to the carburettor to constitute a counter-current to the hot coke cinders. This arrangement results in re-heating of the gas and simultaneous carburetting of the carbon dioxide emanating from the dolomite. Consequently the dolomite is heated and burned by means of the latent heat of the residual cinder coke.

The de-sulfurization action of the filter is satisfactory. For a dolomite consumption of 60 kg. raw dolomite per metric ton of sponge iron, the latter absorbs only 0.002 to 0.004% S from the gas.

The used dolomite, containing 5 to 7% S in the form of calcium sulphide and magnesium sulphide, is continuously withdrawn at the base of the filter by means of a discharge table of the same construction as the carburettor discharge. A continuous withdrawal is essential to keep the filter free from dust and to maintain an even gas distribution throughout the shaft cross section. Trials with limestone were also carried out. Dolomite was found to be superior because it has a higher mechanical strength and consequently gives a lesser gas pressure drop in the filter. (Cf. Bull-Simonsson, I, *Stahl und Eisen 52* (1932), pp. 457–461).

Different reducing agents

The composition of the reduction gas is of great importance for the temperature conditions in the shaft in the Wiberg process. In the final reduction zone, ferrous oxide is reduced to metallic iron, according to the following formulae:

$$FeO+CO=Fe+CO_2+3 \text{ kilocals/mol}$$
$$FeO+H_2=Fe+H_2O-6.5 \text{ kilocals/mol}$$

The reduction with carbon monoxide is exothermal whereas with hydrogen it is endothermal. From the reaction heat it can be calculated that a gas mixture

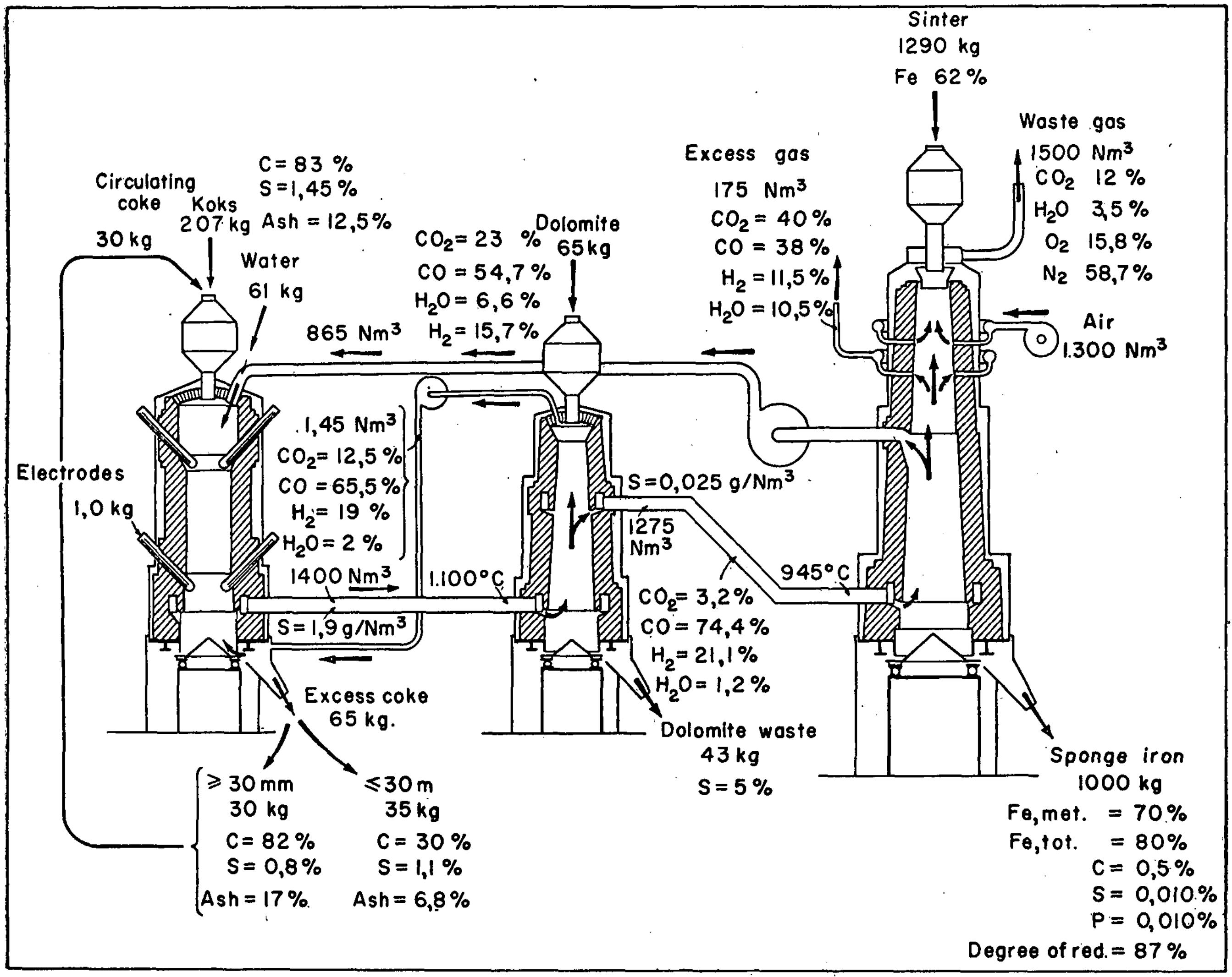

Figure 4
Materials balance per 1,000 kilos of sponge iron.

of the composition 37% H_2 and 63% CO in theory reacts with ferrous oxide without resulting temperature change. In practice equilibrium prevails at less than 37% hydrogen, due to heat losses in the reduction furnace. The sponge iron kiln at Söderfors is run as a rule on about 20% H_2. Operational experiences show that variations in hydrogen content between 15 and 35% exert only minor influence on the production. At a hydrogen content of only 5% the production drops about 20%. Introducing more than 35% hydrogen content has not been tried. Laboratory investigations have shown that high-grade reduction with hydrogen only takes place at a slow rate.

The above indicates that a fuel and a gas producer are required capable of delivering the desired hydrogen content of 20 to 30%. Originally charcoal was used as charge material in the process. At normal hydrogen and moisture contents of the charcoal this automatically yielded the desired hydrogen content of the gas. When using coke a hydrogen content of only about 5% was obtained, which proved to be too low. In order to increase the hydrogen content to the appropriate level, water was added to the circulation gas prior to entering the carburettor, thus making it function partly as an electric water-gas generator. This mode of operation, however, is less economical since a comparatively large amount of excess gas has to be withdrawn from the system as shown by the materials balance.

The ideal fuel would contain 7.70% H_2 and 92.3%C. Coal shows a fair approximation to this composition, but a suitable electric carburettor would have to be developed for its utilization. A solution which is close at hand is the addition to the coke of hydrogenous fuels such as fuel oil, natural gas, or coke oven gas.

Natural gas is out of the question as far as Sweden is concerned, but could advantageously be used in oil-producing countries. Experiments with fuel oil additions are being performed at Söderfors. The possibility of producing a suitable reduction gas entirely from oil or natural gas deserves consideration, and may be performed according to well-known methods for the production of synthetic gas. The importance of the relevant fuel on the economy of the process can be seen in Table 1.

Table 1

COMPARISON BETWEEN DIFFERENT REDUCTION MEDIA

	Consumption per ton sponge iron with 81% met. Fe and 90% tot. Fe	
Reduction media 25% H_2 in gas	*Fuel (kg.)*	*Power (kWh)*
Coke+water	225	1,140
Coke+	111	940
oil	63	
Coke+	151	940
methane	41 m^3	
Coke+	165	940
coke oven gas	75 m^3	

DIFFERENT ORES

For the Wiberg-Söderfors process the following ore requirements apply:

(1) The lump size of the ore must be adequate, for instance within a range of 25–80 mm.

(2) The iron content should preferably exceed 65% to yield a rich sponge iron;

(3) The phosphorus content has to be low, since the entire amount of phosphorus contained in the ore remains in the sponge iron;

(4) The ore should possess sufficient mechanical strength during reduction.

Lump ore as well as sinter can be used. At Söderfors, trials with various ores have been carried out, some of which gave good results, whereas others disintegrated upon being reduced and thereby obstructed the gas circulation.

The mechanical strength of the ore is obviously of the greatest importance for the sponge iron process. Jernkontoret has constituted a research committee to investigate the liability of different ores to disintegrate and the reasons for that. It has been shown that the mechanically weakest stage in the process is the ferrous oxide, or wustite state, which occurs at an intermediate level of the shaft. As the possibility of predicting the behaviour of an ore in the sponge iron kiln is important, the following special testing procedure was devised.

A sample of 500 g. of ore is crushed, sieved between 10 and 16 mm., and subsequently heated to 950°C. in a rotary tube furnace. Reduction gas is supplied from the sponge iron furnace of such a composition as to reduce the ore to wustite. After 3.5 hours the furnace is filled with nitrogen and rotated totally 2,000 revolutions at a speed of 66.6 revolutions per minute. After cooling the sample is taken out and sieved through a 10 mm. sieve. The residue above 10 mm. has proved to give an adequate comparison number valid for the ore strength. This number should amount to a minimum of 70 and preferably 80%.

Rich hematites like Spanish hematite, for example, and Canadian Steep Rock ore are generally suitable, coarse crystalline hematites are frequently disintegrating, as are the El-Pao ore from Venezuela, or some Swedish hematites like the Blötberget ore. Favourable results were obtained with dense magnetites like Kiruna and Tuolluvaara ore.

At Söderfors a sinter of 62% Fe has usually been used. This was made from magnetite concentrates in a Greenawalt sintering plant and gave a sponge iron with 80% Fe. A sponge iron of 90% Fe is aimed at and to obtain this an iron content of 67% Fe in the concentrate is required. If on the other hand an ordinary sinter of higher iron content (above 65%) is used, difficulties like hang-ups in the shaft, etc., will eventually occur.

By contrast, pellets, produced from rich concentrates, appear to behave satisfactorily on reduction. Pelletizing has been widely considered in Sweden in recent years. It renders possible the sintering of high-grade concentrate fines at costs that have been estimated as less than those for ordinary sintering.

Trial runs were performed in the Söderfors sponge iron plant with a variety of Swedish magnetite concentrate pellets holding up to 69.5% Fe, and good results were achieved. Pelletizing plants are being built at Söderfors as well as adjacent to other Swedish ironworks planning the erection of sponge iron plants.

The reducibility of the ore is naturally important. In the present Söderfors furnace more than 30 (metric) tons of sponge iron per day have been produced when using porous sinter. With Kiruna ore, the production decreases to 20 (metric) tons per day.

Important figures for the economy of the process are shown in Table 2, the first column being valid for a long continuous run and the use of coke with water addition in the carburettor. The second column gives the calculated values for a larger sponge iron kiln with rich pelletized charge, always providing that coke and oil can be used as reducing agent, a problem that may soon be solved.

Table 2

PRODUCTION DATA—WIBERG-SÖDERFORS SPONGE IRON PROCESS

	Current results at Söderfors	*Calculated data for larger furnaces*
Analysis of sponge:		
Fe tot	80%	90%
Fe met	70%	81%
Degree of reduction	87%	90%
Daily output	27 tons	60 tons
Consumption per ton of sponge:		
Coke	200 kg.	111 kg.
Oil	–	63 kg.
Electric power	1,100 kWh	940 kWh
Electrodes	1 kg.	1.1 kg.
Raw dolomite	60 kg.	70 kg.
Labour, man/hour, excluding transport	2.4	1.5

THE USE OF SPONGE IRON

In Söderfors, the melting of the sponge iron is performed entirely in an electric arc furnace and in the following way. Heavy scrap is charged on the furnace bottom, the electrodes are lowered to touch the scrap, and the sponge iron is charged via a hole in the roof. As the current is switched on, the arcs will—to start with—burn on the scrap and later on the molten steel and will be surrounded by the sponge iron, which is a relatively poor conductor. The heat efficiency during melting will thus be high.

Sponge iron made from rich pellets possesses a higher electric conductivity and can be treated like ordinary scrap. It has further been stated that sponge iron can be used advantageously in acid open-hearth furnaces. The charge consists of one third liquid pig iron, one third scrap, and one third sponge iron, which gives a higher production than 70% pig iron and 30% scrap. Even in basic open-hearth furnaces up to 25% of sponge iron has been used successfully.

FURTHER PLANNING

The Wiberg-Söderfors process has attracted increasing attention in the Swedish iron industry in recent years. The question thus arose how to construct a larger plant based on the experiences gained in Söderfors. The building of a larger furnace raises the question of how far the kiln dimensions can be enlarged to obtain a correspondingly increased production. When the diameter is increased, the limit is eventually reached where the gas velocity becomes too low in the centre. Another possibility consists in building a higher shaft, whilst keeping the diameter constant. This involves several disadvantages: high pressure fans would be required and the charge material would have to withstand the pressure of a higher burden. It has therefore been decided to keep the height of the final reduction zone unaltered (*circa* 6.5 m.) and to enlarge the shaft diameter from 2 to 3 m. The final reduction zone thus acquires a doubled capacity and the production of one such unit can be expected to amount to 20,000 (metric) tons per annum.

In order to achieve good heat efficiency only one carburettor per furnace should be attached. A larger carburettor for 2,000 to 2,400 kW. having four electrodes at each electrode level has been developed. The layout of such a sponge iron plant is illustrated by Figures 5 and 6. The plant consists of two sponge iron furnaces, each equipped with a desulfurization filter, and three gas producers, one of which is in reserve. The cost of a sponge iron plant of 20,000 tons yearly production amounts to about $800,000. At present, several plants are being built or planned in Sweden.

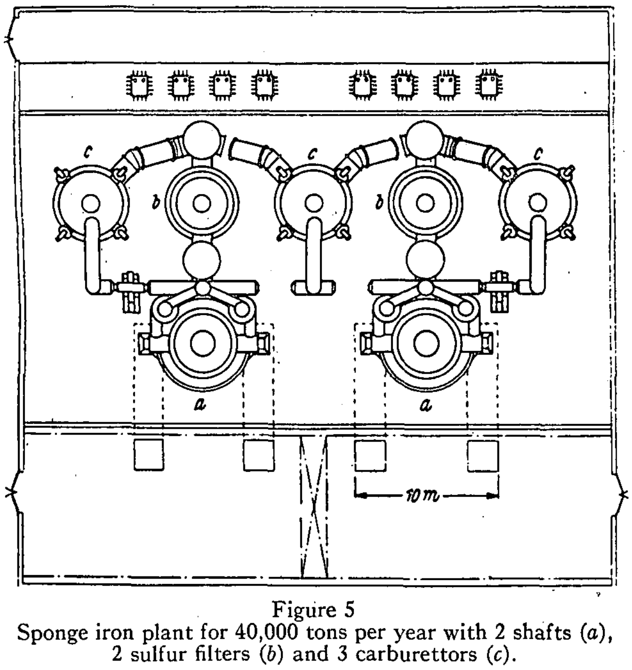

Figure 5
Sponge iron plant for 40,000 tons per year with 2 shafts (*a*), 2 sulfur filters (*b*) and 3 carburettors (*c*).

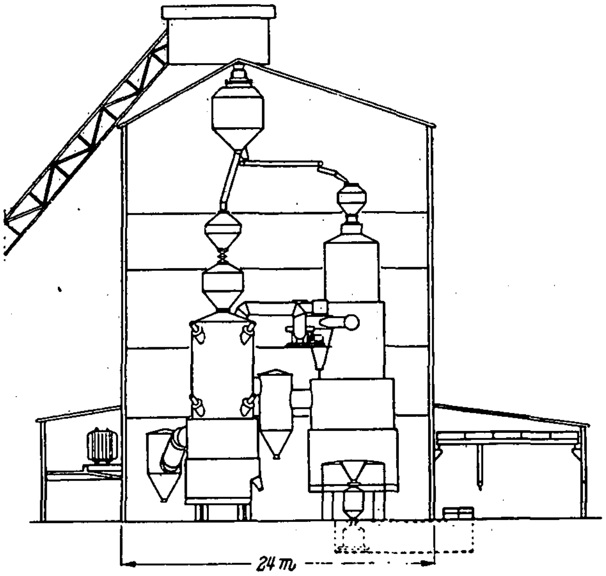

Figure 6
General plan of sponge iron plant.

At the Sandviken Works a sponge iron furnace of an annual capacity of 20,000 tons is completed and has been in operation since January 1952.

Plants of the new design (Figures 5 and 6) are under construction in Hellefors Bruks A.B., Hofors Bruk, and Uddeholms A.B. The first Hellefors furnace was expected to be ready during 1952, and the furnaces of the Hofors and Uddeholm companies in 1953.

The desirability of building larger furnaces producing more than 20,000 tons a year has led to the consideration of reduction shafts of rectangular cross section, as is used in shaft furnaces for lead and copper. This should facilitate the gas penetration in the shaft centre.

The constructional difficulties regarding sponge iron discharge and brickwork construction has so far excluded a development along these lines.

The use of pellets makes feasible a remarkable combination method, namely, direct charging of green pellets into the sponge iron furnace and then sintering in the combustion zone. It should be noted, however, that for burning of pellets a temperature of 1,250°C. is required, whilst the temperature of the sponge iron process is kept at about 950°C.

So far sponge iron has mainly been considered as a raw material for special steel. Whether sponge iron also may be a possible raw material for plain commercial steel depends on its production cost relative to that of coke pig iron. The possibility of producing sponge iron at a competitive price is not excluded.

In oil fields large amounts of gas are often available that are not utilized. That is the case in Venezuela, for example, where rich ore deposits also are found, but domestic coal supplies are small. From natural gas or oil a gas mixture, well suited for the sponge iron process, can be generated. Estimates indicate in this way that sponge iron can be produced at a cost similar to that of coke pig iron.

Provided sufficiently cheap electric power can be supplied, sponge iron produced using coke as the main reducing agent might well compete even with coke pig iron to serve as an economic raw material for commercial steel.

The Basset Process for the Production of Pig Iron in Rotary Kilns

KNUD E. JENSEN

INTRODUCTION

For centuries the only continuously working type of furnace for industrial use was the shaft furnace. At the end of the nineteenth century, however, other types of furnaces, such as the tunnel kiln and the rotary kiln, appeared, and gradually several fields of application were found where these types of furnaces could with advantage replace the shaft furnace.

With regard to pig iron production, the shaft furnace seemed, however, to have reached such a degree of development in the modern big blast furnace that its exclusive position in this field was firmly established forever. Even now, when many new processes have come up, the conventional blast furnace still dominates, and at present none of the new methods for the production of pig iron seems to be able to threaten the position of the blast furnace; the new processes rather complement the traditional smelting process within special fields of application.

The Basset process is one of these new processes. It has got its name from the French inventor Lucien P. Basset. His first patents for a new smelting process were issued as early as in 1921. His original idea was to produce fluid steel directly from ore in a rotary kiln, and, of course, this idea aroused much public interest in metallurgical as well as in financial circles.

His experiments with this process taught him, however, that the direct production of steel was very difficult in practice. At last it was given up, and new experiments with production of pig iron in rotary kilns were started, but here also a great many difficulties had to be overcome.

A pilot plant was built in France, and in the 1930's full-scale plants were built in Spain, Japan and Portugal. All these plants were built by conversion of already existing cement works. None of them is in operation now. The Japanese and the Spanish plants for production of iron worked only for rather short periods. The Portuguese plant was started shortly before the war and worked with interruptions for several years, but was stopped after the war.

In Denmark, experiments with the Basset process were started at nearly the same time as in Portugal. The Danish plant succeeded in developing the process on a commercial scale, and it is still in operation. The results from Denmark will be further accounted for.

THE DANISH IRON WORKS

The Danish company of F. L. Smidth & Co. A/S started to work with the Basset process in the 1930's. The principal sphere of action of this company is the cement industry, but besides they supply rotary kilns for many other purposes, such as lime burning, dead-burning of dolomite and magnesite, calcining of alumina, and nodulizing of ore fines and blast furnace dust; they thus possess an intimate knowledge of rotary kiln operation.

In 1938 they started the first experiments with the Basset process at a cement plant in the Danish town of Aalborg, where an old cement kiln was converted for this purpose.

During the war and the German occupation of Denmark imports of pig iron nearly came to a stop, and the existence of a Danish pig iron plant was therefore very important to the entire engineering industry of Denmark, a fact which, of course, contributed towards increasing the activities in this field, resulting in the conversion of several cement kilns into iron kilns. A few facts will be stated in order to give an idea of the present stage of the work.

Since 1939 there has been a regular output on a commercial scale, although experiments are still carried out periodically in order to improve the technique. Up to now about 300,000 tons of pigs have been produced, and during the last three years each kiln has been working about 340 days a year. It is evident, then, that the process has advanced far beyond the pilot-plant stage. The annual output is now about 35,000 tons of iron turned out by two kilns.

The average length of the periods of kiln operation between the stops for repairs to linings, etc., is about two months, and each stop lasts in general only three of four days, as only a few yards of the kiln lining have to be replaced each time.

Because of the fact that the Aalborg plant is a converted cement works, it is far from being ideal for the production of iron. Of course, existing machines from the old cement plant have been used to the greatest possible extent and the operation has been bound by the original layout. For these reasons the arrangement of the various departments and the sizes and types of machines are not too well suited for the present purpose. The handling of materials is too complicated and expensive, and more power and repair work are called for by the rather old machines of this plant than would be the case in a new and adequate plant.

Denmark is a country of few mineral resources, and practically no ores and fuels are found. During the first years the raw material for the pig iron production was spent pyrites from Danish and Swedish sulfuric acid plants. The reducing carbon was—and still is—coke breeze from Danish gas works, supplemented with imported coke fines, and the kilns were fuelled with imported coal. After the war the kilns have mostly been fired by fuel oil. Chalk is the only material used in the process which is found near the plant.

When ores and fuels have to be imported, it is rather difficult under normal conditions to compete with the British, German and other European blast furnace plants which were the main suppliers of pig iron to Denmark before the war. Transport by sea from these works to the Danish consumers is very cheap.

In Denmark a material which can be used in the Basset process is, however, available in large quantities, namely scrap, mostly in the form of waste from the rather important engineering industry. Most of the scrap is utilized in open-hearth furnaces, but the surplus, consisting merely of cheap scrap, that is, steel turnings and light, galvanized scrap, is used for the production of pig iron. Spent pyrites is now generally used only as a supplement to scrap; only for special purposes is it used alone, for instance if special qualities of iron are wanted, or for experimental work.

The kilns can be run with pure ore, pure scrap, or a mixture of both, but the output will, of course, be much larger if a great proportion of scrap is used. This will also cut production costs considerably.

The Basset kiln

A brief description is given below of the idea and the realization of the Basset process, with a few remarks on the special problems and how they have been tackled.

The complete smelting process is carried out in one simple apparatus, namely, the rotary kiln.

Like other rotary kilns, the Basset kiln is a slightly inclined, slowly rotating tube carried on roller supports. The kiln is fired from the lower end by coal dust, oil or gas, whereas the mixture consisting of ore, reducing agent (for instance coke fines), and limestone is sluiced into the upper end of the kiln and passes down through the kiln in counter-current to the hot gases, whereby it is heated slowly. Free water is evaporated first. Later on combined water is driven out, and at about 500°C. (930°F.) the first reduction of Fe_2O_3 into FeO commences. Calcination of the limestone and reduction of FeO go on from about 750°C. (1,300°F.), and at about 1,100°C. (2,000°F.) the reduction is practically completed.

During the further heating of the solid, metallic iron it absorbs carbon from the residual coke in the charge and melts. Oxides of phosphorus and manganese are also reduced in the kiln, and these substances are absorbed by the fluid iron. Due to the very basic composition of the slag, the sulfur is absorbed by it as calcium sulphide. Silica will not be reduced but will combine with the lime, forming a slag with a high fusing point.

The slag is only sintered, not melted, and it forms small balls like cement clinker in a cement kiln. The lime/silica ratio is about 2.8 to 3.0. At the outlet end of the kiln the diameter is reduced so that a dam is formed in the kiln. This dam will retain the fluid iron, while the clinker, floating on the iron bath and protecting it against oxidizing, will proceed over the dam and leave the kiln continuously.

Iron is tapped periodically, say every 90 minutes, by opening a small tap hole in the cylindrical shell of the kiln. Tapping is carried out while the kiln is rotating, and the fluid iron, having a temperature of about 1,400–1,450°C. (2,500–2,600°F.), pours down into a ladle which follows the movements of the tap hole.

At the outlet end of the kiln (the part of smaller diameter) the clinker substance tends to form a crust on the lining. This crust is easily removed by regular scraping with a water-cooled "boring bar" installed on the burner's platform and operated by hand. Scraping is normally done once in a shift.

The hot gases leave the kiln through the inlet end, which is surrounded by a steel plate smoke chamber. Between the smoke chamber and the kiln a special air seal is necessary in order to prevent the drawing-in of false air. The temperature of the exit gases is about 600°C. (1,100°F.) when the kiln is pressed to its maximum capacity, as will, of course, be the case normally. The kiln can be operated a little more economically with a lower output and a lower gas temperature, but the saving in fuel costs can rarely compensate for the increasing labour and other costs per ton of pig iron, owing to the smaller production.

During the first experiments a lot of trouble was caused by the burning-out of the brick lining in the melting zone and the formation of heavy crusts and rings on the lining in the reduction zone. These problems have been overcome by using special clinker bricks for the lining in the hot zones and by adjusting the composition of the charge in accordance with the special smelting technique developed for this process. Ring formations are now very rare, and they can always be ascribed to serious faults in the operation of the kiln or the preparation of the charge. Of course, mechanical irregularities, for instance kiln stops, are also sometimes the cause of such trouble as ring formation.

The auxiliary installations necessary for the Basset plant consist of machinery which is well known from other industries. These installations will be mentioned briefly together with the raw materials and the end-products for which they are used.

Raw materials, auxiliary installations and products

The use of the rotary kiln and the basic-slag process requires properties of the raw materials different from those called for by the conventional iron production. The ore may be fine-grained, and it may contain sulfur. On the other hand, too much gangue in the ore will considerably increase the amount of slag, increase the fuel consumption, cut the production, and thus increase essentially the cost of production.

If the ore is received in big lumps, a crushing plant will be needed. The particle size of the ore to be used in the kiln will, of course, depend upon the reducibility, but normally a maximum particle size of between 5 and 10 mm. seems to be most appropriate. Less reducible ores can also be used, if they are ground somewhat finer.

Spent pyrites is normally a cheap and excellent raw material. It can be used without any preparation.

Several kinds of reducing agents are usable, and the requirements as to purity and fineness are the same as for the ore. Coke fines, anthracite fines, charcoal and petrol coke can be used. A high percentage of fixed carbon and a relatively low ash content are important in order to ensure a satisfactory economical result.

The reducing agents are usually too coarse and must be ground, for instance in a rod mill.

A pure quality of limestone is also required. In most cases it is easy to procure this, either as a soft chalk or as a hard limestone. The limestone should be ground to below 0.5 mm. (35 mesh), and generally it is necessary to dry it in order to make this grinding possible.

The following can be used for the firing of the kiln: gas coal dust, charcoal dust, fuel oil, natural gas, and any other fuel which can yield a sufficiently hot flame and which has a limited content of ash.

Solid fuels must be ground, and fuel oil must be pre-heated.

It will be realized that many different raw materials and fuels can be used, and further that the properties essential for this purpose are different from those required for the blast furnace process.

In addition to the auxiliary equipment for the preparation of the raw materials, some other installations are necesssary for the treatment of the iron and the by-products, clinker and gas.

The iron from the rotary kiln is of a special quality due to the basic-slag process. The analysis is as follows:

Carbon: 4.2 to 4.5%.

Silicon: below 0.1%.

Manganese: from 0 up to 5% or even more, dependent on the manganese content in the raw materials. The yield of manganese is normally very good, due to a very intensive reduction in the kiln and to the basic slag.

Phosphorus: from 0 up to 2%, dependent on the raw materials. Most of the phosphorus from the raw materials will remain in the iron.

Sulfur: below 0.01%, even with a relatively large amount of sulfur in the raw materials and fuels.

This pig iron is particularly well suited for use in basic open-hearth steel furnaces, for the production of special alloyed steels, for special castings (for instance spheroidal graphite castings) and for the making of iron powder, but it can, of course, also be used for the production of ordinary mild steel and cast iron.

If the iron is to be used for ordinary cast iron, the low content of silicon is a disadvantage. Ferrosilicon must be added, either in a mixer before the pigs are cast, or in the cupola during the remelting.

For a possible treatment of the iron, a foundry crane, mixer, and pig casting installations will be required.

The clinker produced simultaneously with the iron has a composition similar to that of Portland cement clinker and, when it is ground, nearly the same hardening properties. The clinker from the kiln contains 5 to 10% metallic iron, most of which is removed in a combined grinding and separating plant with air separator and magnetic separator. The small iron beads from the separation are returned to the kiln. The clinker fines, practically free from metallic iron, can be utilized in a cement works for mixing with Portland cement and thus represent a certain value.

The gases contain 12–14% CO, 12–14% CO_2, 1–2% H_2, and about 1% CH_4. The calorific value is thus rather low, namely, about 550 cal./cu.m. (60 B.T.U./cu.ft.), and a possible utilization of the gases must therefore be effected directly after the kiln without any preceding dust cleaning, so that the sensible heat in the gases can also be recovered. The most suitable installation for this purpose will normally be a waste heat boiler, and at the Danish plant this system has been used to full satisfaction for about eight years. The steam is used for generating power.

In Denmark this utilization of the gases is of great economic importance. Whether it will pay in other places depends upon the prices of power and fuels. In countries with cheap hydro-electric power or with cheap fuels, as for instance natural gas, it may not pay to install a waste heat boiler and a turbo-generator.

ADVANTAGES AND DISADVANTAGES

It is of interest to compare the Basset process with the coke blast furnace process and with other processes, but unfortunately space is limited and only permits a comparison with the coke blast furnace process.

The principal advantages of the Basset process are the following:

(1) It is possible to use fine-grained fuels as reducing agents, and coal dust, oil, gas or other fuels for firing. Reasonable amounts of sulfur can be allowed in the fuel;

(2) Fine-grained ores, containing reasonable amounts of sulfur and zinc, can be used directly. Less reducible ores, such as magnetites, can advantageously be smelted by this process, where the reduction surface may be increased by grinding the ore;

(3) A special quality of iron is obtained, low in sulfur and silica, which is advantageous for special purposes;

(4) The value of the clinker is higher than the value of blast furnace slag;

(5) The process is technically and economically well suited for commencing small-scale production of pig iron because the small units employed make it possible to keep the total first cost at a relatively low level, which may be important in certain cases;

(6) The installations are simple and sturdy and require comparatively little special training of the operators. Rotary kiln operation is known and realized for the production of cement in many places where few other industries are in existence;

(7) The process is flexible with regard to kiln output, raw materials and fuels as well as in respect of the Mn-content and the P-content of the pig iron.

The disadvantages of the Basset process can briefly be said to be the following:

(1) Considerable increase in fuel consumption and decrease in kiln output, if fuel with a high ash content and/or poor ores with high silica and alumina contents have to be used;

(2) It is necessary to add ferrosilicon, if the iron is to be used for ordinary iron castings;

(3) Relatively high labour costs because of the small production units. This is common to all processes in which the unit capacities have not reached the gigantic capacities of modern blast furnaces;

(4) The waste heat in the gases is of a low grade;

(5) Ordinarily the fuel consumption figured as total input of thermal units per ton of iron is comparatively high, but this may be compensated for by the use of low-priced fuel.

Below are a few comments on the last points. The minimum requirement in thermal units for reducing iron oxides to metallic iron is fixed. To this must be added the requirements for carburizing and melting the iron. The production of molten iron will therefore of necessity require more heat than the production of sponge iron.

A third item on the heat balance is the treatment of the slag, which will in principle call for more heat in a process with basic slag than in a process without slag or with an acid slag. This disadvantage is to a certain extent compensated for by the higher value of the basic slag.

The simultaneous production of gases is normally considered as a loss, a loss which can, however, be recovered partially. The Basset process gives a rather large quantity of gases with a low calorific value; this causes the recovery of the heat in the gases to be less efficient and less economical than in the blast furnace process.

Some losses cannot be recovered, for instance the sensible heat in the iron and in the slag and the radiation losses from the kiln. This applies not only to the Basset process but also to all other processes.

The total consumption of thermal units in the Basset kiln is high, but another fact may be of much more economic importance in the choice of the reducing process, namely the price of the thermal units, and just here the Basset process has its special advantage of being able to use low-grade fuels which are available at low prices per thermal unit, because they are suited only for few other purposes. This fact will very often make up for the high fuel consumption.

THE ECONOMY OF THE PROCESS

The economic possibilities of the Basset process cannot be judged generally but will have to be studied in detail in each individual case. Tables 1 and 2 give, however, a few characteristic examples illustrating how much the cost of production may vary, and moreover they give an idea of the costs of installation and production.

Table 1 shows, for four different raw mixtures, the productions of iron from one kiln unit of the largest size foreseen at the present stage of development of the process. Further, the consumption of raw materials and fuels and the production of clinker and waste heat per 1,000 kg. of iron are given. These figures are based upon the compositions of the raw materials stated in the upper part of the Table.

Table 1

Raw materials	*I*	*II*	*III*	*IV*
Ore:	Spent pyrites	Hematite A	Hematite B	Magnetite
% Fe	66	68	60	70
% SiO_2	3	2	7.5	2
% S	0.5	—	—	—
Reducing agent:	Anthracite	Petrol coke	Charcoal	Coke-fines
% fixed C	90	90	70	88
% Ash	5	2	4	9
% S	1	1	—	1
Limestone:	Quality A	Quality B	Quality A	Quality A
% $CaCO_3$	98	93	98	98
% SiO_2	1	6	1	1
Fuel:	Gas coal	Natural gas	Fuel oil	Fuel oil
H_i, kcal/kilo	6,500	—	9,600	9,600
H_i, kcal/m^3 NPT	—	9,000	—	—
% Ash	10	—	—	—
% S	1	1	1	1
Production of iron per kiln unit: (kiln 3.6 m. dia., 70 m. long)				
Metric tons/year	27,000	29,000	22,000	30,000
Consumption per 1,000 kg. of iron				
Ore	1,580 kg.	1,530 kg.	1,750 kg.	1,490 kg.
Reducing agent	650 "	620 "	930 "	610 "
Limestone	400 "	310 "	810 "	340 "
Fuel	530 "	370 m^3	460 "	330 "
By-products per 1,000 kg. of iron				
Clinker	350 kg.	250 kg.	700 kg.	300 kg.
Total heat in gas, 10^6 B.T.U.	17.5	16.7	21.4	16.3

As to the manpower required, it can be said that for a plant comprising two kilns, of the size mentioned in Table 1, a total of about 35 workers and foremen will be needed for operating the kilns and for the preparation of the raw materials and the treatment of the clinker. To this number should be added the labour which may be wanted for the further handling and treatment of the liquid iron from the kiln and for the attendance of the waste heat boiler installation. Besides, account will have to be taken of the workmen required for repair shop, laboratory and stores.

The power consumption will be normally about 100–120 kWh/ton of iron produced. The amount of cooling water required is negligible. The costs of repairs and maintenance may be estimated at about \$2.50 per ton of iron, whereas overheads and miscellaneous costs (including wages and salaries for the staff, costs of pig casting, etc., will amount roughly to about \$3.75. These figures are just estimates stated as a guidance for a general survey, and they must, of course, be revised in accordance with local conditions in each individual case.

In Table 2 the cost figures for the Basset process are compared with those for the blast furnace process, the electric smelting furnace process (Tysland-Hole), and the Wiberg-Söderfors sponge iron process. The figures and prices for the three last-named processes, which are based on Canadian prices in 1948, have been taken from the United Nations report of 1950 on *World Iron Ore Resources and their Utilization.*[1]

Table 2

OBS.: 1 ton = 1,000 kg.

PRODUCTION COSTS AND INVESTMENT REQUIREMENTS

Item	*Blast furnace*	*Tysland-Hole*	*Wiberg-Söderfors*	*Basset I*	*Basset II*	*Basset III*	*Basset IV*
Plant and furnace capacity:							
Total yearly output (tons)	280,000	110,000	45,000	54,000	58,000	44,000	60,000
Number of furnaces	1	3	2	2	2	2	2
Daily output per furnace	800 tons	100 tons	64 tons	82 tons	88 tons	67 tons	91 tons
Production costs per ton of iron (dollars): (approx. Canadian 1948 price level)							
Fuels: Metallurgical coke, \$16.50 per ton	13.70	2.86	3.60	—	—	—	—
Coke fines, \$8.25 per ton	—	2.26	—	—	—	—	5.03
Gas coal fines, \$12 per ton	—	—	—	6.35	—	—	—
Anthractie fines, \$11 per ton	—	—	—	7.14	—	—	—
Petrol coke, \$10 per ton	—	—	—	—	6.20	—	—
Charcoal fines, \$15 per ton	—	—	—	—	—	13.95	—
Fuel oil, \$25 per ton	—	—	—	—	—	11.50	8.25
Natural gas, \$20 per m³	—	—	—	—	7.40	—	—
Ores: Lump ore (hematite) 50% Fe, \$7.25 per ton	14.50	14.50	—	—	—	—	—
" " (hematite) 56% Fe, \$11.50 per ton	—	—	20.75	—	—	—	—
Spent pyrites, \$5 per ton	—	—	—	7.90	—	—	—
Hematite fines, 68%, \$11.50 per ton	—	—	—	—	17.60	—	—
" " 60%, \$10 per ton	—	—	—	—	—	17.50	—
Magnetite fines, 70%, \$11 per ton	—	—	—	—	—	—	16.38
Limestone: 98% $CaCO_3$, \$4.50 per ton	—	—	—	1.80	—	3.65	1.53
93% $CaCO_3$, \$2.25 per ton	0.97	0.88	0.09	—	0.70	—	—
Electrodes: \$134 per ton	—	1.80	0.40	—	—	—	—
Power: 0.3 cents per kWh	—	7.50	2.70	0.33	0.32	0.36	0.35
Cooling water: 0.003 cents per gallon	0.42	0.33	0.06	—	—	—	—
Labour: \$1.20 per hour	0.94	2.19	2.52	2.27	2.17	2.78	2.16
Repair and maintenance	0.50	0.95	1.65	2.50	2.35	2.75	2.33
Overheads and miscellaneous (including casting)	3.65	3.80	3.25	3.75	3.51	4.53	3.45
Production costs	34.68	37.07	35.02	32.04	40.25	57.02	39.48
Gas credit: 25 cents/million B.T.U.	1.90	1.75	—	—	—	—	—
Gas credit: 5 cents/million B.T.U. (low grade gas)	—	—	—	0.87	0.84	1.07	0.82
Slag credit: \$2 per ton about	1.20	1.00	—	—	—	—	—
Clinker credit: \$4 per ton	—	—	—	1.40	1.00	2.80	1.20
Net production costs (excl. of profits and fixed charges on investments	31.58	34.32	35.02	29.77	38.41	53.15	37.46
Total investment (thousands of dollars) (approx. representative costs in Canada 1948)							
Furnaces and auxiliary equipment	12,000	5,000	1,300	900	900	900	900
Buildings	2,000	1,000	400	500	500	500	550
Ore and material-handling equipment	2,000	1,000	300	400	400	400	450
Clinker grinding plant	—	—	—	70	70	70	70
Working capital (6 months)	4,000	2,000	750	900	950	750	1,000
Engineering and contingencies	2,000	750	250	300	300	300	300
TOTAL	22,000	9,750	3,000	3,070	3,120	2,920	3,270
Investment per ton-year (dollars)							
Total	79	89	67	57	54	66	55
Production costs, inclusive of 10% charge on investment, dollars.	39.48	43.22	41.72	35.47	43.81	59.75	42.96
Production costs, inclusive of 15% charge on investment, dollars.	43.43	47.67	45.07	38.32	46.51	63.05	45.71

The figures given for the Basset process are not exact, as the unit prices in Canada for ores, limestone, fuels, etc., have been estimated; on the other hand, the consumption figures and other technical data are based on the extensive experience gained from the operation of the Danish Basset plant.

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[1] UN document ST/ECA/6, 25 August 1950. Sales No. 1950. II. D.3.

The figures for costs of production show that, under certain circumstances, the Basset process can compete successfully with other processes for the reduction of ore.

Table 2 also gives an estimate of the installation costs. The costs of the machinery for the Basset plant have been taken from a recent quotation worked out by F. L. Smidth & Co. A/S and adapted to the price level ruling in 1948. The other installation costs have been estimated on the basis of figures corresponding to Danish conditions. Today the total cost of installation will probably be 15–25% higher than the figures given in the Table, depending on local prices and conditions. The estimate for the Basset process does not include a waste heat boiler and power station for the utilization of the gases. On the other hand, a rather low value of the gases is indicated. At the same time it should be observed that the figures for the blast furnace process differ from the figures given in the United Nations report mentioned in the foregoing, in that the coke ovens are left out and the price of metallurgical coke increased correspondingly. The figures show that the total installation cost of a Basset plant is relatively low, even for a two-kiln plant, and the initial cost can, of course, be further reduced if only one kiln is installed in the first instance. Also the costs of installation per ton produced compare favourably.

The total investment for a one-kiln plant with waste heat boiler and power station will today amount to around $2.5 to $3 million, depending on local conditions. The cost of a similar two-kiln plant will amount to around $4.5 to $5 million. These prices include iron mixer and pig casting equipment.

COMBINATION WITH OTHER INDUSTRIES

The first cost of a Basset plant will, of course, be lower if it can be built in combination with an existing plant of some kind or other, making it possible to use some of the existing auxiliary installations for the new plant, first and foremost of course the workshops, offices, laboratory, etc., and further part of the transport equipment and, maybe, also of the storage equipment.

If spent pyrites is to be used as raw material for the iron production, it may be advantageous to combine the Basset plant with a sulfuric acid plant.

Another possibility is to combine a new Basset plant with an existing cement plant. In that case one will, in addition to the aforementioned advantages, be able to benefit by the following facts: in the first place, cement plants usually have at their disposal rich limestone quarries already in exploitation. Furthermore, the engineers and the master burners of the cement plant already have experience of rotary kiln operation; and, finally, the Basset clinker can easily be utilized in the manufacture of cement. Installations for the manufacture of clinker lining bricks are also often found at the cement plants.

Still another alternative is to convert an existing cement plant, or part of it, so that it can be used for the production of iron. This idea has in several cases been taken up by cement works which have a surplus of kiln capacity owing to the installation of new, modern and more economical rotary kilns for the burning of cement. At first glance this idea seems very attractive. However, a closer calculation will generally show that the conversion will be more expensive than originally expected and the result less good than imagined. The arrangement of such a converted plant will generally be less suitable and the plant will be expensive in operation, as one will be bound by the layout of the cement plant so that the handling of the materials becomes too complicated and requires too many workmen. Moreover, old cement kilns usually have comparatively small diameters and therefore fairly small outputs, which will further increase the cost of production per ton of pig iron. In general, there will thus be no reason for optimism in regard to the economic possibilities of converting old cement plants into Basset plants. This, however, does not preclude the fact that in special cases it may be a paying proposition to do so.

It will, of course, be a great economical advantage to a Basset plant if the liquid iron produced, or at least part of it, is subjected directly to further treatment, as thereby the costs of pig-casting, handling of pigs, transportation and remelting are saved. In most cases a combination with an open-hearth furnace plant using the molten iron for steel production will therefore be the most suitable arrangement.

FURTHER POSSIBILITIES OF THE BASSET PROCESS

Extensive investigations comprising the working-out of layouts and calculations of initial costs and manufacturing costs for different conditions seem to prove that under special conditions the Basset process will have certain economic possibilities, especially in countries with rich ores, oil or natural gas but without good coking coal and in countries with an engineering industry which is but slightly developed. Examples supporting these views have already been given in Table 2.

Certain raw material problems and tendencies in the development of the iron industry may increase these possibilities, namely:

(1) The increased activities in technically less developed parts of the world, resulting in demands for new local iron and steel industries. A small-scale iron works will be a suitable basis for starting the development of such an industry. Relatively low initial costs for a Basset plant may in such cases prove to be of decisive importance;

(2) The present lack of smelting capacity in the world will probably still for some time keep up the demand for new installations, not only in the present steel producing centres but also in the technically less developed countries;

(3) The lack of good coking coal in many parts of the world where iron ores are available in large quantities and in satisfactory qualities will favour any process using fuels other than metallurgical coke;

(4) In other countries with less valuable resources of ores, special raw materials, such as spent pyrites from the sulfuric acid industry, may open up possibilities of a local iron production on the Basset process;

(5) The shortage of native sulfur will in the future force many sulfuric acid plants to roast pyrites instead of burning sulfur, a change which will

ture combined with adequate strength to support the furnace charge. This will be the common condition in Latin America where high-volatile coals with moderate to weak coking properties are much more widely distributed than the low-volatile expanding coals.

In the broader field of interchange among the Latin-American States to obtain more efficient use of raw materials, movement of small quantities of low-volatile coal to the more widely available high-volatile stocks is indicated.

Looking to the longer-term future, when the Latin-American steel industries may have grown to be large consumers of coking coal, it may be safely predicted that adequacy of Latin-American supplies and their effective utilization will depend upon free interchange of supplies and scientific blending to prepare effective coking-coal mixtures.

In that period it may be expected that general progress in industrialization will have provided adequate and efficient transportation facilities, and more intensive exploration will have developed a more complete knowledge of the available coals. There is ample justification for the belief that the raw material resources exist in adequate volume and variety, requiring only that exploration and research keep pace with developing needs to maintain an adequate coking-coal supply.

Effective blending of widely different coals is a primary preparation problem. There are two major criteria of effectiveness: large-scale proportional blending to even out variations in chemical and physical properties so that the furnace charge may be safely designed on the basis of uniform raw material, and intimate blending of the two or more ingredients to assure most effective combination of the raw-material qualities in the carbonizing process.

Because of the unique importance of the blending step in the preparation of Latin-American coking coals, a separate study of the mechanics of blending should be undertaken in the technical programmes for the steel industries.

OTHER SPECIAL CONDITIONS AFFECTING PREPARATION

Sulfur

In common with the Appalachian region of the United States, the Latin-American steel industries will always be threatened with the baffling problem of sulfur control in preparing their coking coals.

Sulfur conforms only very roughly to the specific-gravity consist of a raw coal mixture. Unlike ash, it may sometimes even be higher in the low gravity than in the high gravity fractions, although this is unusual. Sulfur occurs in several different chemical and physical forms in coal, and usually a combination of forms makes up the total sulfur content. Varying combinations of sulfur forms and sulfur specific-gravity relationships make the washing operation uncertain and unpredictable with respect to its effect on total sulfur, except with a most intensive study of washing characteristics by the float-and-sink method.

Extensive research on the occurrence of sulfur in the important coking coals of the Appalachian region has shown that the problem-sulfur coals have extremely erratic sulfur distribution, and that, the higher the average sulfur, the more erratic its distribution in the bed. United States coals vary greatly in potential sulfur elimination. Some of the most important of these coals show almost no response to washing and the most responsive will rarely show as much as 50% reduction in total sulfur by conventional washing practice, even with 3/8-inch crushing.

In Latin-American coalfields a few local areas that have had a measure of intensive study show a similar pattern in sulfur occurrence. A typical situation is illustrated in the Cauca Valley sample series of Table 1. There appears to be a small field of low-sulfur, high-volatile coking coal in the north, a larger field of high-sulfur, high-volatile in the Golondrinas mountains and the central area to about the Rio Timba region, and another low-sulfur zone in the area between Rio Timba and Suarez, but rising in sulfur erratically to the south. However, in these general areas, exceptional samples differing strikingly from the average are distressingly frequent, and selective blending of the local coals will be necessary to obtain a large output of uniform coking coal from the area.

Of course washed coal sulfur is the final criterion, but the local coals often fluctuate as erratically in float-sulfur content as in total raw sulfur, indicating a wide difference in expected response to washing. The four float-sulfur curves of Figure 2 showing washability data on four Cauca Valley samples illustrate this condition.

The Barro Branco coal of southern Brazil, unique in practically every respect, is truly amazing in sulfur reduction, coming from the pits with 10 to 14% sulfur in the raw state and reducing to 1.50% in the washed product—a phenomenon probably not duplicated anywhere in the coal-producing world. However, many individual location samples show a float sulfur too high for current metallurgical standards, and a uniform blending is necessary to keep the sulfur at an acceptable level.

The sulfur problem can be discussed only in a very general manner because the Latin-American steel industries, being in the pioneer stage of development, can establish raw-material standards, including coke specifications, to suit local conditions. The natural procedure in any case is to prepare as clean a coal as is practicable by washing; the final adjustment to take care of high sulfur can be made in the blast-furnace operation. In each situation, the furnace practice comes to make the most effective use of the raw materials available. With respect to sulfur, the quality of iron ore and limestone enters into the problem of coke-sulfur elimination.

Adjustment to a varying raw material situation is illustrated by the changing pattern of coke specifications in the Pittsburgh steel industry, where early coal-sulfur tolerances were generally at 1.00 to 1.10%, then 1.25% and now nominally at around 1.50% for purchased coal; but substantial tonnages of captive coal are being used at 1.85% or higher sulfur content.

In recent expansion of coking-coal production into the higher sulfur reserves of the Pittsburgh bed, preparation facilities have included facilities for large-scale blending operations to even out erratic fluctuations in sulfur content. It is currently reported that more efficient blast-furnace operation is made possible with a uniform coke and more precise control of gradually increase the production of spent pyrites and probably lower the price of this material. This development will also increase the chances of the Basset process;

(6) Some new methods and productions within iron metallurgy will call for special qualities of pig iron, which can be produced by the Basset process. This applies, for instance, to the manufacture of iron powder and spheroidal graphite castings.

The conclusion of the investigations made and of the said considerations must be that the Basset process—as well as similar new processes—within special fields will gain a certain position within the iron industry in the future, whereas, as far as can be seen today, the steadily improving blast furnace process will retain its position as the most important method of large-scale production of iron in the world centres of the iron industry.

The Rotary Kiln Processes for Sponge Iron, Developed at the Avesta Iron and Steel Works and the Domnarfvet Iron and Steel Works, Sweden

BO KALLING

A short account is given below of the experiments conducted in Sweden for producing sponge iron in a rotary kiln, first at Avesta in 1928–35 and later on at Domnarfvet since 1950.

1. The Avesta method

The reduction principle used at Avesta was the following: finely ground ore mixed with pulverized fuel in excess, for instance coke breeze, was fed continuously into a rotary kiln. The heat was supplied in the final reduction zone by means of an electric current conducted through the charge between two ring-shaped electrodes made of heat-resisting steel. The gas formed through the reaction between the ore and the carbon was burnt before leaving the furnace, and thus its heat content could be used to a great extent, but not entirely, for heating the charge.

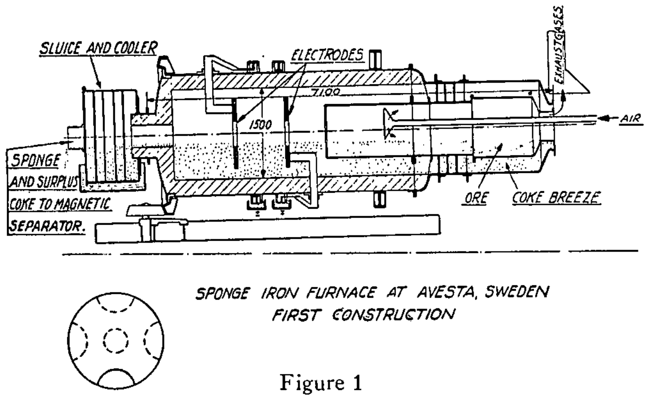


Figure 1

According to the furnace construction shown in Figure 1 the ore was fed into the furnace separately through a central tube that was passed by the reaction gas in the opposite direction. The inner part of the tube thus constituted a pre-reducing zone which was expected to cause a better utilization of the carbon monoxide in the gas. As, however, this arrangement did not function very satisfactorily, the construction was later on simplified to that shown in Figures 2 and 3.

The sponge together with the excess fuel was discharged through a cooler, at the same time serving as a sluice in order to prevent the furnace gas from leaving the furnace this way. The excess fuel was separated from the sponge in a magnetic separator and was then returned to the feeding end.

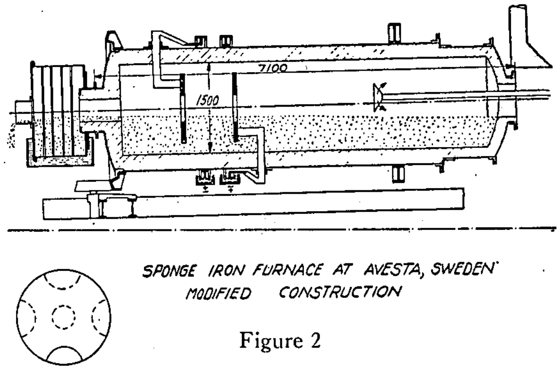


Figure 2

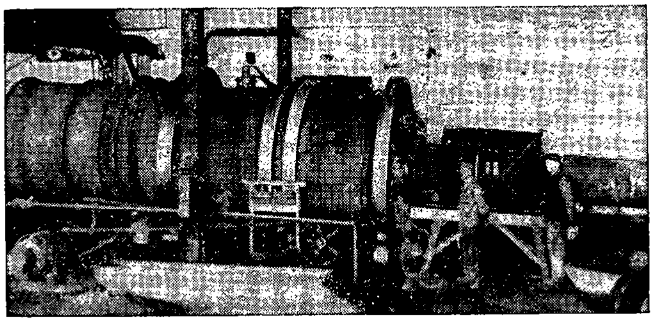


Figure 3
Photograph of the modified furnace at Avesta.

The most important detail of the furnace construction was the way in which the electric power was introduced. It was considered of importance to get the heat for the reduction developed at the same point where it is really consumed. This was accomplished by using the charge itself as an electric resistor. In order to prevent electric current from leaking along the furnace walls the electrode diameter was made smaller than the inside furnace diameter as shown in the figures.

The experimental results indicated a lower furnace capacity than expected owing to the fact that the power supply had to be limited in order to avoid too high a current density on the contact surface between the electrodes and the charge. If the density exceeded a certain value, the ore stuck to the electrodes and caused a formation of balls in the furnace. The production capacity in the furnace shown in Figure 2 could not be increased without difficulties to more than 5–6 tons a day of iron in the sponge, if a satisfactory degree of reduction was to be reached. The consumption of electric power and fuel for an ore containing 60–65% Fe was normally

	Per ton Fe in sponge iron Degree of reduction>90%
Electric power	700–800 kWh
Fuel (usually mixture of coke breeze and coal fines)	500 kg.

The relation between power and fuel consumption could be varied within certain limits by controlling the air supply, but if too much air was introduced, a certain risk of sticking to the walls appeared in the blast zone by using the shown arrangement.

During the first trials charcoal was used as a reducing agent. It proved desirable, however, to replace the charcoal by a percentage of coke breeze, as the dust losses in the exhaust gas were comparatively high when only charcoal was used. By using charcoal there were no difficulties at all in keeping the sulfur content in the sponge iron at a satisfactory value, but when using only coke or coal the sulfur content of the sponge grew high, usually up to 0.2–0.3 per cent. Attempts were made to add lime or dolomite to the charge, but especially when reducing siliceous ores this addition caused an increased tendency to stick and for that reason this method was abandoned. It was found, however, that the sulfur content of the sponge could easily be decreased in a separate refining process after the reduction process. For this purpose the sponge was mixed with about 5% powdered burnt lime and some coke breeze and reheated in a closed furnace chamber to 800–900°C. The sulfur in this way passed over to the lime very rapidly and the process could be carried out without any trouble. At Avesta the same furnace was used as for the reduction process but without any addition of air. The power consumption was about 400 kWh per ton Fe, but subsequent experiments at Domnarfvet have shown that it need not exceed 100–150 kWh, if steps are taken to ensure a better heat economy of the process. The desulfurization must be followed by another magnetic separation. In this way it was possible to decrease the sulfur content to below 0.010% by using the wet concentration method. A high iron content and a high degree of reduction in the finished sponge was achieved at the same time.

The analysis of the finished sponge from a Swedish hematite containing 60–65% Fe, after desulfurization and concentration, was normally

Fe_{tot}	94.5%	Degree of reduction 93%
Fe_{met}	88%	
S	0.010%	

The consumption of power and raw materials was

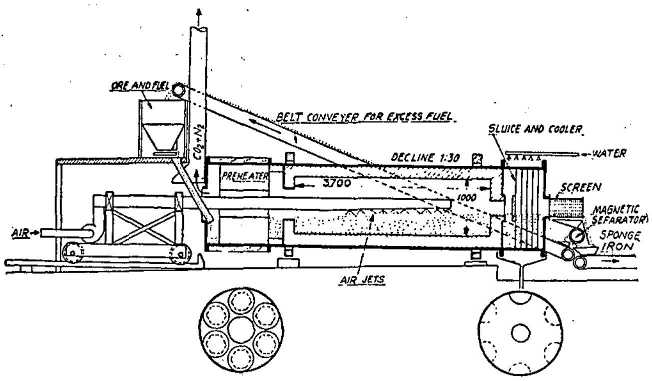
	Per ton Fe of finished sponge
Ore (containing 63% Fe)	1,650 kg.
Electric power (for reduction and desulfurization)	1,000 kWh
Coke breeze and coal fines	530 kg.
Burnt lime	50 kg.

It should be pointed out that power and fuel consumption can be expected to decrease considerably if the process is run in bigger furnaces with slightly altered construction. As the process can be carried out without any preceding agglomeration of the ore even when fine ores are used and with powdered fuel, this method could still be of interest today under favourable conditions.

The furnace at Avesta has been in commercial operation for extended periods producing iron powder but has not been in operation during the last few years. No full scale plant using the Avesta method has been erected anywhere else.

2. The Domnarfvet method

At the end of the 1930's the results of experiments with a method of quite another kind led to a better way for introducing the air supply into the furnace, giving rise to the possibility of increasing the fuel combustion to such an extent that simultaneous supply of electric power was not necessary. The main problem in obtaining a smooth process with sufficient heat supply only by combustion of fuel in the furnace has been the fact that the heat is generated above the charge and not inside it, as when heated by electric resistance. In order to secure a sufficient heat supply to the charge, considerable overheating is required in the furnace above the charge and especially in the combustion zone itself. This overtemperature must be kept higher the larger the production is, and a limit is soon reached where the production cannot be further increased without a risk of sticking, a great and well-known trouble to overcome when running the reduction process in a rotary kiln.



SPONGE IRON FURNACE AT PILOT PLANT DOMNARFVET, SWEDEN'

Figure 4

This problem is very simply solved in the Domnarfvet method. As shown in Figure 4, the air is here introduced into the reduction zone through a central tube from which it is distributed to a series of jets directed towards the surface of the charge during the rotation. It has been found that the oxygen-containing air can in this way be blown against the charge, which at the same time is to be reduced, without any trouble at all. On the contrary, there is no difficulty, by using this method of heating, in reducing the ore to a degree of reduction of 90–95%. The reason is partly that the carbon monoxide generated in the charge during the reduction protects the ore to a certain extent from direct contact with the oxygen. Furthermore it seems as if under certain conditions a concentration of fine carbon material takes place in the surface of the charge which also counteracts reoxidation.

If the air is directed in this way towards the charge, a strong local overheating of the surface is obtained. The stirring action during the rotation of the furnace, however, causes a rapid heat transport from the overheated layer into the charge. Thus the heat transmission becomes very effective, and it is proved possible to run the process with a satisfying production with a wall temperature not exceeding that of the charge. This varies usually between 1,075 and 1,150°C. depending on the nature of the ore. The risk of sticking to the walls is, in turn, considerably diminished. An added advantage in directing the air against the charge is that free oxygen is easily prevented from coming into contact with the wall, a necessary factor if sticking is to be avoided.

Some preliminary experiments had already been run with this combustion principle in 1939 in a small laboratory kiln at Högfors Blast Furnace Plant belonging to the Avesta works. The experiments gave promising results, but were not taken up again until 1950 and at that time at Domnarfvet, where the present method has been developed.

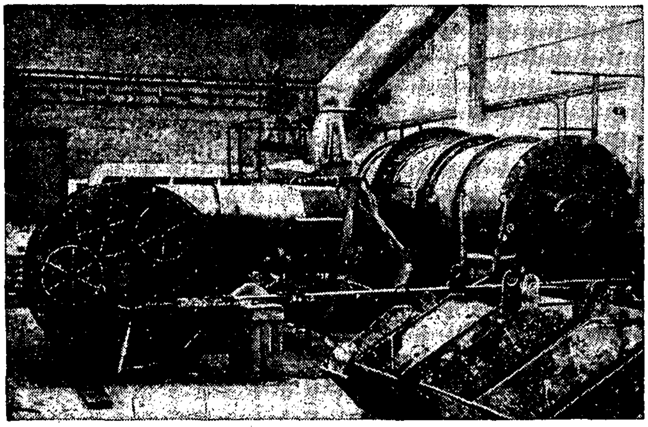


Figure 5
Photograph of the laboratory kiln at Domnarfvet.

Figures 4 and 5 show the laboratory kiln at Domnarfvet, mainly as it was arranged during the last periods of running. A mixture of ore and fuel, usually coke breeze, in excess is fed continuously into the furnace from a stock bin, and the reduced sponge together with the excess coke leaves the furnace continuously from the opposite end through a water-cooled sluice as in the Avesta process. The combustion gases escape in counter current to the ore-coke mixture at the feeding end of the furnace and their useful heat is thereby transferred to the charge in a heat exchanger made of plate, which gives a good contact between the gas, the charge and the plate material that also takes part in the heat transmission. As it has proved possible to combust the carbon monoxide completely within the furnace and at the same time to maintain a low temperature in the waste gas (about 300°C.) the fuel consumption turns out low.

The reduced sponge is separated from the excess coke by magnetic separation in the usual way. In those cases where lump ore or agglomerate is used it is desirable first to take off the coarse sponge in a rotating screen adapted at the exit end of the furnace as shown in Figure 4 and then separate magnetically the fine fraction only. The excess coke is taken back to the charging bin.

The laboratory kiln has an inside diameter of 1 m. only and a length of 3.5 m. In this kiln it has proved possible to produce at least 3 tons Fe per 24 hours in the form of sponge iron with 90–95% degree of reduction when using ore containing about 60% Fe, that is about 3 tons per m^3 of charge. The production per m^3 of furnace volume is about 1 ton Fe per 24 hours against 0.5 in the Avesta kiln. The fuel has mainly been comparatively fine coke breeze and the net consumption has been about 550 kg. per ton Fe in the sponge. Total consumption has been somewhat higher on account of dust losses, as up till now suitable arrangements for dust collection have not been applied. In a full scale furnace with an effective dust separation the total consumption of coke breeze should not exceed about 500 kg. per ton Fe in the sponge.

When using coke breeze or coal fines the sponge becomes high in sulfur as in the Avesta process. A sulfur content of 0.15–0.25% in the raw sponge must be calculated. Experiments with additions of lime or dolomite in grain form to the charge in order to reduce the sulfur content did not give satisfactory results even with the Domnarfvet method, at least not in the few tests when this method was tried. It has, however, been found possible in this case also to desulfurize the sponge by heating it with lime powder and coke breeze in a closed furnace chamber. Laboratory experiments have shown that the desulfurization can be carried out advantageously at about 825°C. The consumption of material and power is calculated to be:

	Per ton Fe in the sponge
Electric power	100 kWh
Burnt lime	30 kgm.
Coke breeze	20 kgm.

The construction of the desulfurizing furnace is very simple and the treating time short. A magnetic separation must be made after the desulfurization. It was found that very effective separation can also be obtained with the material in a dry condition, if advantage is taken of the latest progress in the techniques of ore-concentration.

Another method for desulfurization which seems promising, is also under investigation at present but it is still too early to present this method for discussion.

In this connexion it should perhaps be pointed out that a comparatively high content of sulfur in the sponge iron does not prohibit its use in steel production. The sulfur can easily be removed in connexion with the subsequent melting operation, and it might then be advisable not to complicate the reduction process by desulfurization but postpone that to a later stage in the steelmaking. This is perhaps worth special consideration when new plants are being erected, where one is not bound to methods already in operation. It is for instance possible to melt the sponge in a special electric melting furnace under addition of lime to form a desulfurizing slag, whereafter the molten low sulfur steel is transferred to the steel plant. It is also possible to add carbon during the melting in order to obtain a pig iron that can be refined either by means of lime during the melting or afterwards according to some prevailing desulfurization method.

The reducing agent mostly used for this process has been coke breeze, but coal and anthracite mixed with coke breeze have also been used advantageously. Other fine grained reducing agents can of course also be taken into consideration.

The ore can be charged in powder form, but higher production can be obtained if it is in the form of small sized lumps or agglomerated in some way. Ordinary sinter and pellets are consequently excellent raw materials. Some experiments not yet concluded show that fine ore rolled to small pellets with a suitable binder can be used directly after drying without any previous sintering treatment.

It is not always necessary to concentrate a poor ore before the reduction. It is sometimes more rational to reduce the ore directly after crushing it to suitable size and to grind and concentrate the product first after the reduction. This method has given excellent results, especially for hematites difficult to concentrate.

The size of ore or agglomerates can be varied within rather wide limits without any inconvenience, but the main part of the ore should preferably be smaller than about 25 mm. in a full scale furnace.

Some results obtained by reduction of a silicious Swedish hematite ore (Striberg) in the Domnarfvet furnace will be mentioned. The ore was crushed to below 12 mm. size and was reduced with coke breeze (containing 80% C) ground to below 2 mm. Production amounted to about 2.8 tons Fe a day and the net consumption of coke breeze was about 540 kg. per ton of iron. The temperature in the furnace had to be kept relatively low (max. 1,100° C.) as the gangue in this ore was found to melt rather easily. The degree of reduction of the raw sponge, about 85%, was therefore not quite satisfactory, but after the following dry concentration it increased to above 90.5% and the total Fe-content at the same time rose to 91.8% as shown in the table below. The iron yield from ore to concentrate was 91%, which is a much higher figure than can be obtained by concentrating this ore before reduction.

	Fe tot	*Fe met*	*Degree of red.*	*MnO*	*MgO*	*CaO*	*Al_2O_3*	*SiO_2*	*P_2O_5*	*S*	*C*
Ore.................	55.4	—	—	0.1	0.7	2.1	1.0	18.0	0.04	0.01	—
Raw sponge..........	68.0	57.5	84.5	0.1	0.8	2.5	2.1	23.1	0.04	0.21	1.2
Concentrated sponge..	91.8	83.0	90.3	—	—	—	—	—	—	0.23	0.35

The process has not yet been tried in full scale operation though some licence agreements have already been signed with Scandinavian firms. It is hoped that the erection of a full scale plant will be started before the end of this year. As to the suitable furnace capacity it may be pointed out that a furnace of this kind must not be made too big. At present we consider it advantageous to limit the inside diameter to about 2 m. and the total length to 8–10 m. In a furnace of this size a production of 15–20 tons per 24 hours ought to be reached. It may of course be possible to run larger furnaces as well, but we think the process will be more reliable when the furnaces are not too large. As the process can be made quite automatic from the charging of the raw materials up to the finished product, the number of workers required are few—two to three men per shift, and increases only slightly if more furnaces are installed. Furthermore the equipment for the magnetic separation as well as for most of the transportation can be consolidated to care for all furnaces, so that the capital costs will be very little higher than when furnaces of larger capacity are installed.

No calculations will be given here of the production costs of sponge iron produced by this method. The process has not yet been run commercially, and the costs are so much influenced by the different local conditions that an attempt to make a more general calculation would be of little value. To summarize, I will list some of the advantages of the Domnarfvet method in comparison with other sponge iron methods:

(1) No consumption of electric power for the reduction;

(2) Low fuel consumption;

(3) The fuel can and should be fine grained;

(4) Different kinds of ore, with varying degrees of purity and pre-treated in different ways can be used;

(5) The labour cost is low;

(6) Capital costs are very low and a plant started on a moderate scale can easily be expanded by installing more reduction furnaces.

On the other hand, the method has the disadvantage that a special desulfurizing treatment may in many cases be necessary.

Production of Sponge Iron in a Rotary Kiln at Temperatures Below the Fusion Point of the Material

RUSSELL C. BUEHL

INTRODUCTION

The Bureau of Mines began research on the production of sponge iron in an internally fired rotary kiln in 1920. This early work was concluded in 1924 and is described in a Bureau of Mines bulletin (1). This bulletin contains an extensive review of previous sponge-iron work and theoretical data relating to the reduction of iron ores at temperatures below the melting point with solid and gaseous fuels. The major purpose of the work was to develop a process for producing sponge iron suitable for use as a chemical reagent for the precipitation of copper, lead, silver and numerous other metals from solutions. For this application, the sulfur content of the sponge iron was of little importance, and no extensive work was done on methods for obtaining a low-sulfur product. Only a few analyses for the sulfur content of the sponge iron are given, but these indicate that it contained over 0.2% sulfur even when low-sulfur coals were employed as reductants.

In the early days of the Second World War, it appeared that a serious scrap shortage might develop in the United States, which would limit steel production. The Bureau of Mines received an appropriation for the investigation of gaseous and solid-fuel reduction of iron ores and this work was begun in 1943. During a period of scrap shortage it would be desirable to utilize existing plants, which might be idle at least part of the year, to produce sponge iron. Therefore, in 1943 research was started on those processes that might employ the rotary kilns of the cement industry or the periodic or tunnel kilns used for brick manufacture.

A complete pilot plant was constructed at Laramie, Wyo., to demonstrate that sponge iron could be produced in a rotary kiln on a large scale (2, 3). This plant is in the western part of the United States, which has medium-size deposits of iron ore but the deposits of coking coal are too small to supply many blast furnaces. However, large quantities of high-volatile, non-coking, sub-bituminous coals and lignites are available, which might be used for producing sponge iron. While the Laramie plant was being constructed, additional work was begun at Boulder City (4) and Pittsburgh (5) to provide data that would aid in the successful operation of the Laramie plant.

The Bureau of Mines also produced sponge iron suitable for conversion into steel at several brick plants employing periodic (6) and tunnel (7, 8) kilns, following the general method of the Höganäs process.

Several methods for gaseous reduction of sponge iron were also investigated by the Bureau of Mines. Early experiments with gaseous reduction were performed with a rotary kiln (9) at temperatures around 900° C. Reformed butane was used as the reducing agent. About 1,500 lb. of a good grade of sponge iron was produced per day, but difficulties were encountered with material sticking to the furnace walls. Iron ore was reduced with producer gas in a Skinner multiple-hearth furnace 5 ft. inside diameter and 16 ft. 6 in. high (10). About 1,000 lb. of sponge iron with 85% reduction could be produced per day at a maximum temperature of 900° C. The sponge iron tended to weld to the hearth so that continuous operation was not achieved.

Iron ore was reduced with gas in a shaft furnace at Longview, Texas, and later at Minneapolis, Minnesota. In the tests at Longview (11, 12), mixtures of natural gas and air were reformed catalytically. This gas was forced through batch reactors equipped with a pulsating valve, which varied the pressure from atmospheric to 40 lb. per square inch. The reforming units operated poorly and the gas lost too much temperature before reaching the reactors. Consequently, reduction of the sponge iron proceeded slowly.

The later tests at the Minneapolis station (13) were more successful. Agglomerated iron oxide pellets between 3/8 in. and 3/4 in. diameter were reduced in an externally heated alloy retort. This retort increased from 6 in. diameter at the top to 12 in. diameter at the bottom and was 10 ft. high. Production was about 1 ton a day. Both impure hydrogen and water gas were used as the reducing agent. The tests showed the feasibility of the method but were short, lasting only a few days, so it is not known whether or not the equipment could be operated continuously in commercial production. Tests on the reduction of iron ore fluidized by hydrogen (14) were unsuccessful. Most ores would not remain fluidized when the deoxidation exceeded 50%.

Although considerable research has been done on the production of sponge iron in the United States and some of the work was moderately successful from a technical standpoint, the only plants producing sponge iron on a commercial scale are grinding the product to a powder for use in powder metallurgy.

DESCRIPTION OF PROCESS AND EQUIPMENT

In the process under consideration, a mixture of ore, solid reducing agent, and dolomite, all crushed to 1/8 in., is fed into the upper end of a long, refractory-lined, rotary kiln. The kiln used at the Laramie plant is shown in Figure 1. It is heated by a burner fired axially from the lower or discharge end. The charge is heated gradually in passing through the kiln. Rapid reduction commences when the charge reaches a temperature of around 930°C. The maximum temperature to which the charge is heated is in the range of 1,000° to 1,070°C., depending mainly on the reducibility of the ore and the reactivity of the reducing agent. The sponge iron is then discharged through an air-tight cooler. For some raw materials and methods of operation, the sponge iron may be fed directly to a magnetic separator, whereas in other cases, screening and grinding may be required before magnetic concentration.

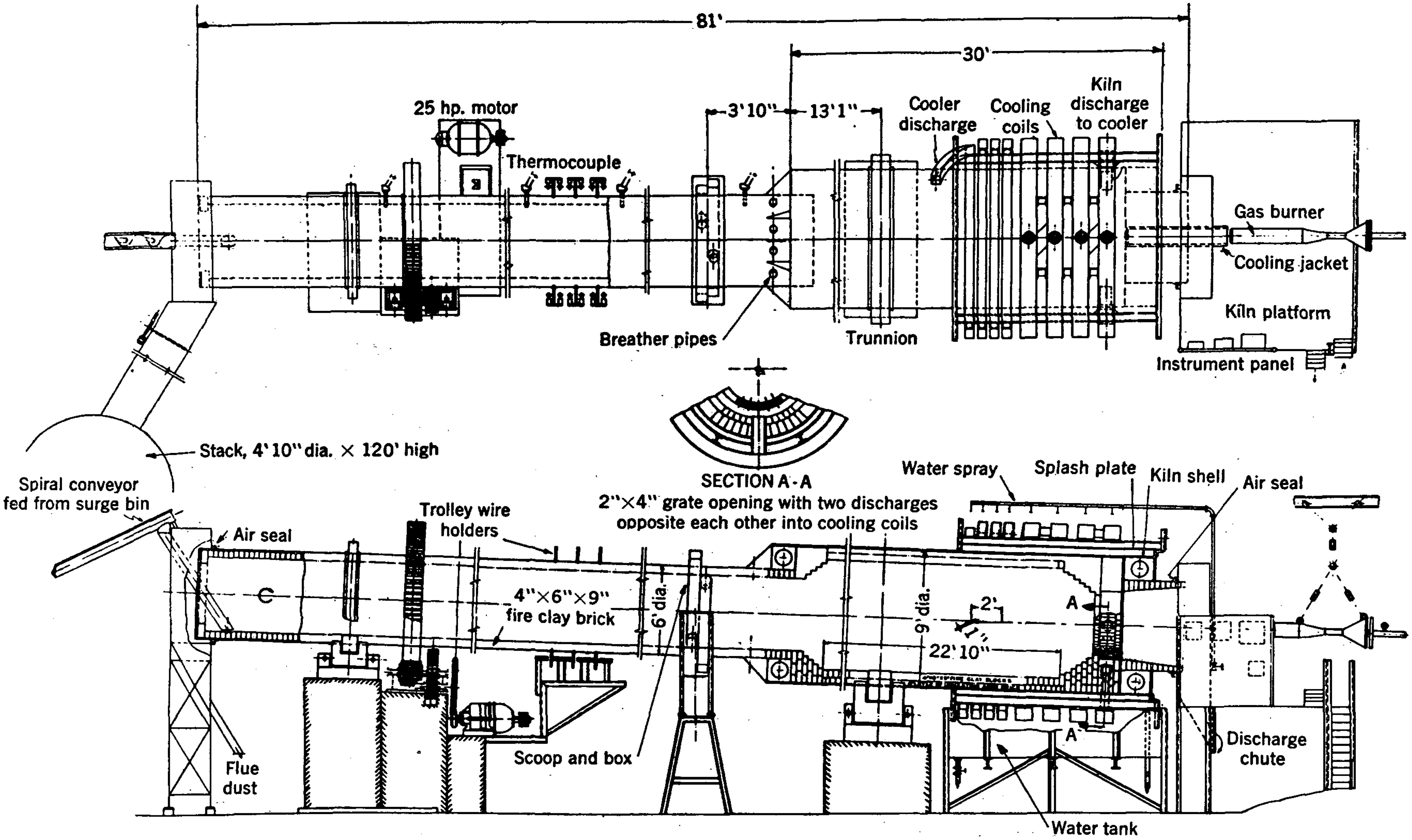


Figure 1
Two-diameter rotary kiln used at the Laramie plant.

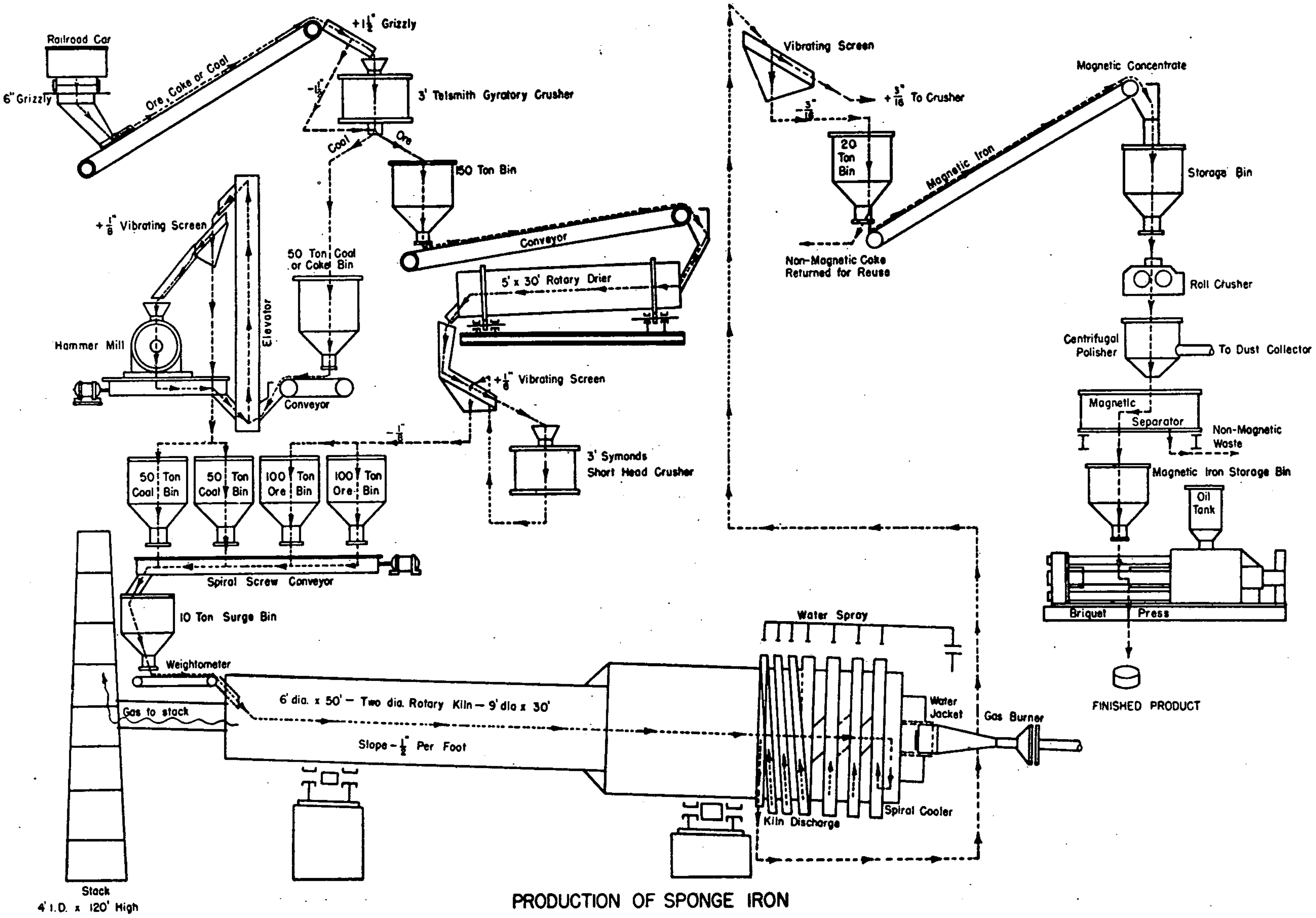


Figure 2
Flow scheme for the Laramie plant.

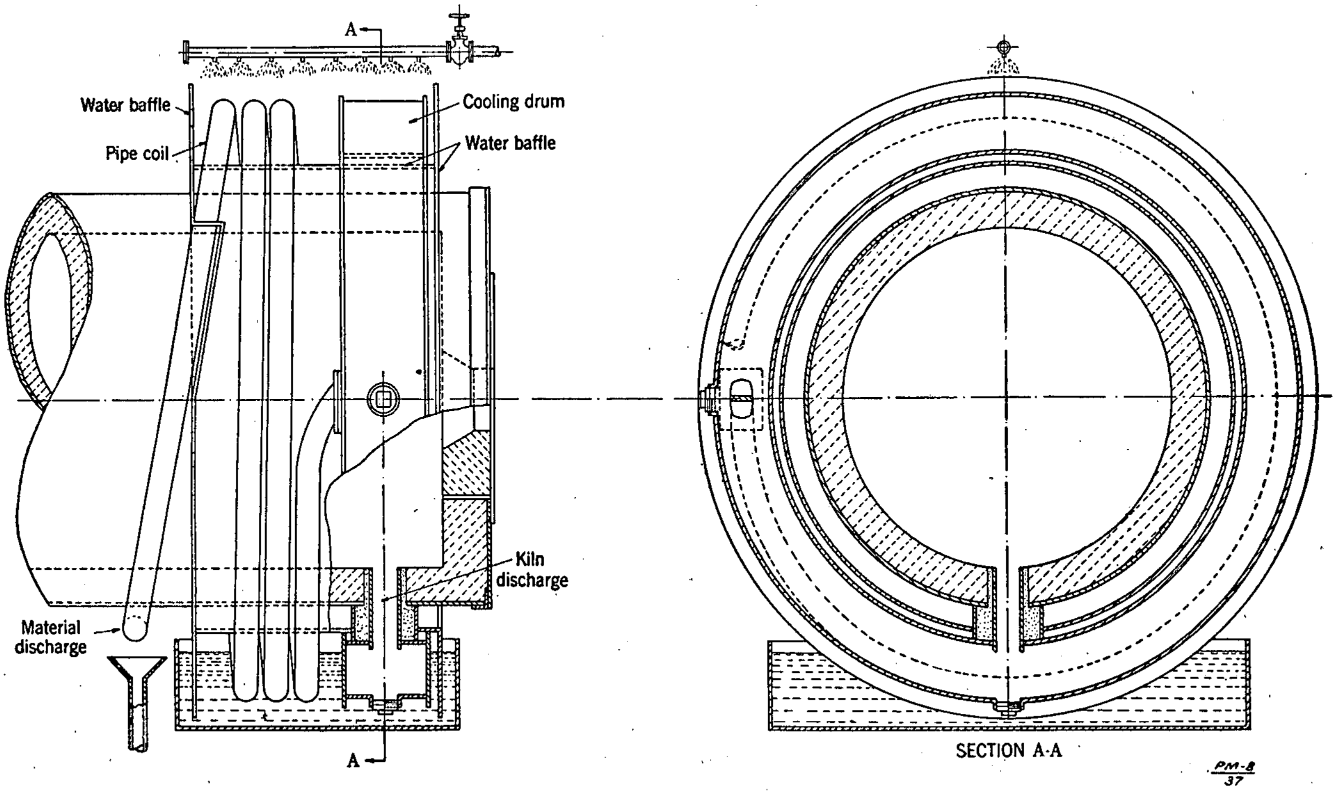


Figure 3
Cooler for sponge iron.

The flow scheme for the Laramie plant is shown in Figure 2. The raw materials were first crushed to 1½ in. size with a gyratory crusher. A hammermill was used to break the coal to ⅛ in., after which it was stored in a bin. The ore often required drying, so it was conveyed to a rotary drier from which it went into a secondary crusher. The ore crushed to ⅛ in. was stored in another bin. Weightometers controlled the mixing of the raw materials and their flow into the kiln. The sponge iron was discharged from the kiln through a cooler of special design, which will be described later. The cooled sponge iron was screened and given a rough magnetic concentration. The magnetic fraction was then passed through a roll crusher and impact pulverizer before the final magnetic concentration. The concentrated sponge iron could be briquetted if desired.

All of the equipment except the sponge-iron cooler was of standard construction. This cooler was developed while the author was engaged in research on the production of iron powder for the Plastic Metals Division of the National Radiator Co., of Johnstown, Pa. (15). The cooler in its simplest form is shown in Figure 3. Sponge iron drops from the kiln into a toroid surrounding the discharge end of the kiln. This toroid is water-cooled with a spray and also rotates partly submerged in water. The sponge iron then discharges through a helix made of pipe. The sponge iron serves as an air seal in the pipe helix. If the drum and coils are water-tight, the discharged sponge iron is light grey, showing that there is virtually no oxidation during cooling.

REVIEW OF RESULTS

Early work (16) on the production of sponge iron in a rotary kiln, which was concluded in 1924, showed that good reduction could be obtained at temperatures around 950°C., but, if the temperature was raised somewhat, higher sticking of the sponge iron to the furnace lining occurred. A kiln with a short reducing section of 5 ft. shell diameter, attached to a 3 ft. 8 in. shell diameter preheating section 22 ft. long, was operated continuously for several months and produced 2½ tons of sponge iron concentrates per day. The sponge iron contained 79.7% total iron, 0.76% sulfur, and 0.2% carbon and was 97% reduced. The equipment for cooling the sponge iron was not satisfactory.

During this work, the idea of a two-diameter kiln consisting of a small-diameter preheating section attached to a large-diameter reducing section was developed, apparently on the basis of theoretical considerations. Using this construction, the charge materials stick to the lining near the junction of the two sections of different diameters if the enlarged-diameter section is too short in respect to the diameter or the preheating section is much smaller than the reducing section.

Commercial production of sponge iron in a rotary kiln was begun about 1941 by the Plastic Metals Division of the National Radiator Co. of Johnstown, Pa. For this work, an old cement kiln was used. The shell consisted of a 6-ft. diameter section 22½ ft. long, a tapered section 7½ ft. long, and a 5 ft. diameter section 30 ft. long, making a total length of 60 ft. A highly reduced sponge iron was made, which was magnetically concentrated and then ground in a ball mill to produce iron powder for powder metallurgical applications. Accretions formed only within a few feet of the discharge end and were easily removed with a bar. Doing this once a shift, operations could be carried on continuously for several months. There was virtually no erosion of the lining.

Research on the production of sponge iron in a rotary kiln was resumed by the Bureau in 1943. The work by the staff of the Pittsburgh Station, which was on a relatively small scale, was directed toward the control of the sulfur content of the sponge iron (17). Additions of pulverized lime or limestone to the charge of a rotary kiln do not result in significant decrease in the sulfur content of the sponge iron, although calculations indicate that the sulfur content of the sponge iron should be below 0.01%. However, if limestone was added to bituminous coal before coking and the resulting coke used as the reducing agent in a rotary kiln, sponge iron with around 0.05% sulfur could be produced. The coal used to make the coke contained 0.9% sulfur.

It was found that, by adding dolomite sized to −6-, +20-mesh to a charge consisting of ore and non-coking coal, low-sulfur sponge iron could be produced. A typical charge consisted of

100 parts of −20-mesh magnetite ore,
80 parts of −¼ in. coke breeze, and
15 parts of −6-, +20 mesh dolomite.

The sponge iron contained 0.02 to 0.06% sulfur, 94% total iron, 89% metallic iron, and was 95% reduced. The coke breeze used as a reducing agent contained over 1% sulfur. When −6-mesh dolomite, which contained all the fine particles resulting from the crushing operations, was used as the sulfur-control agent, the sulfur content of the sponge was considerably higher. Tests have indicated that, when limestone or dolomite are added to the charge, the reduced iron is very low in sulfur—that is, around 0.01%—but, if the desulfurizing agent contains appreciable −100-mesh particles, the sponge iron will be coated with a basic slag layer containing most of the sulfur in the product (18). Limestone of suitable size may be effective for sulfur control if the limestone does not disintegrate as a result of heating and tumbling in the kiln. Additions of magnesia to the charge will not lower the sulfur content of sponge iron, so the magnesia in dolomite plays no direct role in sulfur control. The lime required for sulphur control is about the stoichiometric amount for combining with the sulfur in the charge to give CaS. More limestone or dolomite is required, therefore, when high-sulfur ores or fuels are used. If the charge materials are not too high in sulfur and the temperature is kept so low that no slag is produced, a low-sulfur sponge iron can be made with a small quantity of limestone or dolomite, such as 4½ lb. of dolomite per 100 lb. of ore, even if the ore has a high silica content (32% in some tests).

The large rotary kiln used to produce sponge iron at the Laramie plant is shown in Figure 1. Because of an extreme shortage of facilities for producing industrial equipment at the time the Laramie plant was erected, it was necessary to purchase sections of discarded kilns. Therefore the shape and relative proportions of the kiln were determined by used equipment available. If a new kiln could have been procured, the proportions would have been better and there would have been less sticking of the product to the kiln walls.

About 40 tests were run from February 1944 to September 1945 to study the applicability of different raw materials and the effect of various operating conditions. These tests are described in a Bureau of Mines report (19). Nearly all of the tests lasted less than two weeks. Some were terminated because accretions had formed in the small-diameter section near the junction with the large-diameter section, which prevented the passage of materials through the kiln, or because of other operating difficulties. Other tests were terminated so that a different operating variable could be studied. The first 22 tests were made on hematite ore from Sunrise, Wyoming. Most of the later tests were made on a high-sulfur magnetite ore from Shasta, California. The analyses of both ores are shown below.

	Fe	*Insol.*	*SiO_2*	*Al_2O_3*	*CaO*	*S*	*P*
Sunrise hematite....	52.4	20.3	13.4	5.4	1.1	0.02	0.02
Shasta magnetite....	60.6	8.5	5.5	2.3	3.0	.34	Trace

While the Sunrise ore contained some coarse gangue, which was easy to remove, it cannot be concentrated to as low a silica content as is possible with most ores (20) because of the intimate association of iron and silica. In this respect, the Sunrise ore was not very desirable for producing granular sponge iron below the sintering temperature, but this ore was chosen because it was the cheapest and most readily available ore to the Laramie plant.

A high-volatile sub-bituminous coal from Wyoming was used as the reducing agent, which analysed 44% volatile matter, 46% fixed carbon, 9% ash, 0.4% sulfur, and 11,000 B.T.U. per pound. Although a low-volatile solid fuel would have been more desirable, this coal was readily available at low cost at the Laramie plant, whereas other coals and coke were difficult to obtain in Wyoming, and their shipment for long distances would have involved excessive freight costs. The large amount of volatile matter in the coal often caused trouble because of excessive heat in the stack resulting from burning of volatile matter in the coal. For some of the tests, part of the reducing agent consisted of a char produced by passing the coal through the kiln and burning some of the volatile matter for heat. This char contained 78% fixed carbon, 5.1% volatile matter, and 16.6% ash.

The first 11 tests showed that it was difficult to obtain and maintain proper conditions in the reducing zone of the large-diameter section of the kiln. Optimum conditions in the reducing zone appeared to be reached when the charge consisted of equal parts, by volume, of char and ore at a temperature of 1,000°C. It was evident that the particles of reduced iron tended to weld together as soon as completely reduced rather than remain as individual particles and be discharged as such. At welding temperature, the waste minerals also became plastic and sticky and tended to adhere to the kiln as rings of accretions. At lower temperatures this condition did not prevail, but the percentage of reduction was lower. Whereas the sponge iron produced in the small kiln at Johnstown was essentially of the same size and shape as the ore, the material produced in the large kiln at Laramie contained a high proportion of pelletized iron with the same charge and reducing temperature. Analysis of the partly pelletized sponge iron produced in the large Laramie kiln is shown in the following table.

		Assay, per cent						
	Weight per cent	Total Fe	Met. Fe	Insol.	S	P	CaO_2	SiO_2
+10-mesh metallic	4.04	94.25	89.0	2.8	0.03	0.063	0.1	2.0
+48-mesh metallic	28.76	92.05	88.4	4.4	0.2	.063	.1	3.2
−48-mesh magnetic	22.93	72.75	63.6	18.4	.06	.083	2.3	12.4
−48-mesh nonmag.	44.27	7.20	.0	49.4	.16	.079	9.4	27.0
Calc. head	100.00	50.15	43.6	27.5	.09	.075	4.7	15.8

From these data it can be seen that the +48-mesh pellets represent a product of good grade, but the lower-grade −48-mesh magnetic product contained a large portion of the iron.

In the next nine tests (12 to 20), the sponge iron was intentionally pelletized by raising the temperature to around 1,140° C. The discharge grates became overheated, and it was necessary to move the burner four feet into the kiln and provide it with a water cooled jacket. This overcame the difficulty with clogging of the discharge grates and improved operations.

Two tests were run in which particles of high-volatile coal were blown into the kiln from the discharge end by a high-velocity air stream. The larger particles of coal entered the charge in the large-diameter section near the junction of the small-diameter section. The volatile material from the coal burned with the air and furnished all the required heat after the kiln had been preheated to 800°C. In the first test, which was unsuccessful, the charge contained no fuel and balled up before it reduced. In the second test, the charge consisted of 75% ore and 25% coal, and a granular sponge iron 94% reduced was produced at a temperature of 1,000°C.

The following table gives analyses of a portion of the pelletized material produced in tests 12 through 20 mentioned previously.

	−½ in. pellets	+½ in. pellets
Total iron	91.5	86.6
Metallic iron	90.6	84.4
Reduction	99	97.1
Silica	3.4	5.8
Sulfur	.11	.14
Phosphorus	.07	.08
Total carbon	2.0	1.3
Combined carbon	.86	.58

The tests on Sunrise hematite ore can be summarized as follows: Granular sponge iron could not be magnetically concentrated to a silica content low enough to be considered a good melting stock without excessive loss of iron. Running the kiln at a higher temperature to produce pelletizing of the sponge iron gave a lower silica content material with more sulfur than desirable.

Tests were then performed on the magnetite ore from Shasta, Calif., for which the analysis was given previously. This ore had a high sulfur content, so it was desirable to run it through the preheating section of the kiln under oxidizing conditions to remove sulfur from the ore as SO_2 in the flue gas. For this reason, in nine tests the reducing agent was introduced at the discharge end of the kiln by an air jet, as mentioned before. The magnetite ore reduced to iron with no apparent difficulty, but the tendency of the reduced iron to form into pellets and nodules was distinctly lacking. Attempts at pelletizing resulted in the formation of balls of a mixture of iron and slag cemented together in clusters that did not respond to concentrating by the usual methods of grinding and screening. At lower temperatures (around 950°C.) a granular sponge iron was produced, which was a better product in most respects. The method of blowing coal or coal and char mixtures into the feed end of the kiln gave good results at times but was difficult to control and never was developed enough to warrant adoption as a regular procedure.

A typical analysis of the granular sponge iron produced is:

	Per cent
Total iron	92
Metallic iron	88
Reduction	96
Silica	2.7
Carbon	1.2
Sulfur	.043
Phosphorus	.02

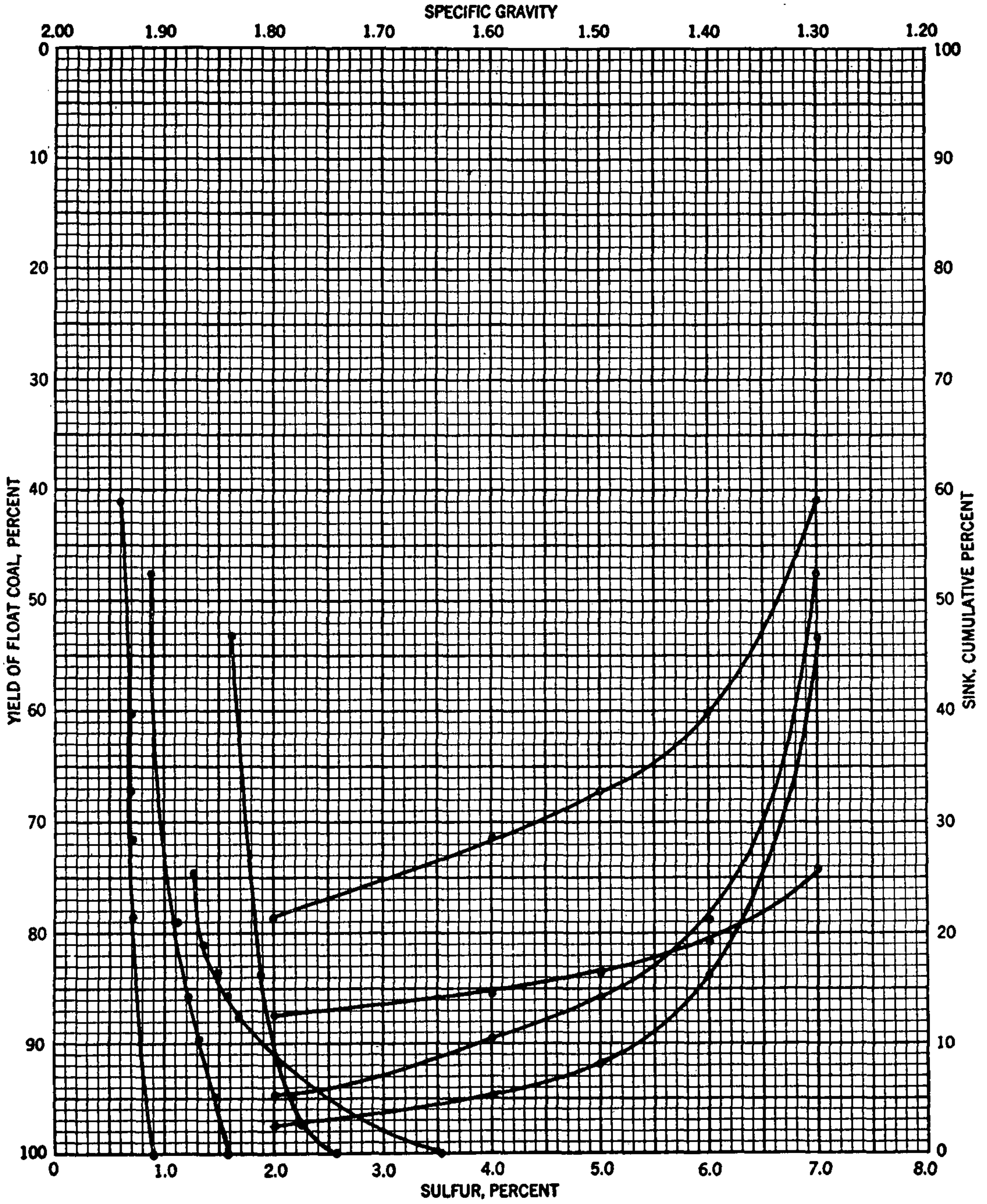
A series of tests was then run in which the kiln was heated with gas, and a mixture of ore, reducing agent (coal, char, or coke breeze), and limestone was charged into the the upper end of the kiln.

When unroasted Shasta ore was used, the sulfur in the sponge iron was 0.06 to 0.07%. However, by first roasting the ore to drop the sulfur from 0.148 to 0.111%, sponge iron with a sulfur content of 0.053% was obtained. In these tests, limestone was added to the charge to the extent of about one fifth the weight of the ore. Near the end of one test, railroad cinders were substituted for char in the charge. These cinders analysed 21% volatile matter, 49.3% fixed carbon, 29.5% ash, 0.67% sulfur, and 9,500 B.T.U. per pound. Comparison of the results given below shows that the grade of product is not appreciably affected by the ash content of the reducing agent, so that high-ash fuels can be used successfully.

	Analysis, per cent				
Material	Met. Fe	Total Fe	Insol.	S	Per cent reduction
Sponge iron made with char	79.4	88.8	4.00	0.06	89.5
Sponge iron made with cinders	80.5	87.2	5.56	.062	92.3

Owing to lack of funds the plant was then shut down but started up several months later. A scoop feeder was installed on the small-diameter section of the kiln six feet from the large-diameter section. Part of the cinder used for reduction was fed through this feeder and appeared to aid operations, but the trial period was too short to establish the definite value of this addition.

In 1948, the Laramie plant was again operated for a short period under a co-operative agreement for producing sponge iron from siliceous French Soumont ore. This ore analysed 34.8% iron, 18.9% SiO_2 7.1%,



Specific-gravity fractions	No. 2		No. 3		No. 7		No. 8	
	Yield	Sulfur	Yield	Sulfur	Yield	Sulfur	Yield	Sulfur
Float - 1.30	47.5	0.88	40.9	0.60	53.3	1.63	74.3	1.28
1.30 - 1.40	78.9	1.13	60.2	.70	83.8	1.88	80.8	1.36
1.40 - 1.50	85.8	1.22	67.2	.70	91.9	2.04	83.4	1.48
1.50 - 1.60	89.7	1.31	71.5	.70	94.9	2.15	85.3	1.59
1.60 - 1.80	94.8	1.44	78.4	.70	97.6	2.25	87.3	1.67
Sink - 1.80	100.0	1.58	100.0	.90	100.0	2.53	100.0	3.51

Fig. 2. Yield-sulfur curves of some Cauca Valley Coals.

Al_2O_3, 3.6% CaO, 15.7% CO_2, 0.07% S, and 6% combined water. The sponge iron produced after grinding and concentration contained 86% total iron, 83% metallic iron, 4.5% silica, and 0.06% sulfur. In the beneficiation treatment, 99% of the metallic iron was recovered in the concentrate. A typical charge consisted of 100 parts ore, 65 parts recycle char from previous tests and 70 parts coal. The results show that it is possible to produce a satisfactory melting stock with an acceptable sulfur content from some siliceous ores.

Formation of rings of accretions in the kiln is an important problem in successful operation of this process. Accretions often formed near the junction of the small- and large-diameter sections. Such accretions are believed due to improper kiln proportions and should be eliminated by making the sections of more nearly the same diameter, using a tapered connexion section, or having a longer section of maximum diameter. However, accretions are also likely to form at any place near the discharge end of the kiln. Such accretions may form if the charge is heated to a high temperature around 1,000° C. before most of the iron is reduced, and this would require a change in operating conditions. If the gangue in the ore has a low fusion temperature, softening of the gangue can likewise cause sticking. However, metallic particles can fuse to each other and to the walls even when the charge contains very little slag-making materials. This imposes a limit on the maximum temperature of operation and on the amount of reduction that can be obtained for some ores. For some raw materials, a water-cooled boring bar may be required for removing such accretions.

In the discharge arrangement shown in the kiln drawing of Figure 2, the product was supposed to discharge over a dam, the fine material going through the cooler and the coarser material rolling out over the end of the kiln extension. Such a dam is very selective, however, in respect to size of material, fine material being discharged in greater proportion than coarse. Agglomerations of sponge iron, which attained a size of about ¾ in. were larger than the thickness of material passing over the dam and would rarely be discharged until the kiln was almost completely full of such larger material. When the charge in the large section of the kiln contained an appreciable fraction of both fine and large material, it appeared that the centre of the charge often rolled in a zone on itself without contact with the hot kiln lining or heating gases above the charge. This centre zone appeared to contain chiefly the fine material that had entered the kiln most recently, whereas the outside was mainly larger agglomerations. The material discharging over the dam would be as much as 100°C. cooler than the material rolling on the surface of the charge. A cooling device that was selective in the other direction, that is, one that would remove all large pieces reaching the discharge end but the fine material at a controlled rate, should improve operating conditions, kiln capacity, and degree of reduction.

SUMMARY OF OPERATING DATA

Because of the wide variation in ores and fuels that have been tested, the charge can best be expressed in terms of the ratio of weight of iron in the ore, the fixed carbon in the reducing agent, and the lime for sulfur control. A typical charge would be: 100 parts iron as ore, 75–100 parts fixed carbon as coal, coke, or char, and 6 parts lime as limestone or dolomite.

Unburned fuel can readily be recovered from the kiln discharge and reused. In this case, the reducing agent in the above charge would consist of: 40–50 parts fixed carbon as new coal, coke, or char, and 35–50 parts fixed carbon as recycled fuel.

On the basis of coke with 87% fixed carbon, the consumption of reducing agent would be 920 to 1,150 lb. of coke equivalent per 2,000 lb. of metal.

The Laramie kiln was operated to produce about 20 tons of sponge-iron concentrate per day. In many of the tests, the production was limited by factors other than the capacity of the kiln to reduce iron, such as feed or discharge mechanisms, burning of volatile matter in the stack creating excessive heat, or heating capacity of the burner. In a test on the Shasta magnetite ore at a production of 20 tons of iron per day, the sponge iron contained 91% total iron, 75% metallic iron, 1.6% silica, and 0.06% sulfur and was 83% reduced. At a capacity of 27 tons of iron per day, the total iron had dropped to 87.5% and metallic to 63.5%, giving a reduction of 72.5%. However, magnetite ore is known to reduce less rapidly than most other ores, so it appears likely that the capacity of the Laramie kiln on a pure, easily reducible hematite would be around 30 tons per day.

Natural-gas consumption was about 7,000 cu. ft. per hour in the tests on magnetite before-mentioned. In other tests, it was as low as 2,500 to 3,000 cu. ft. per hour. For a production of 20 tons of iron per day, the natural gas required would be 8,400 cu. ft. per ton of iron; or, on the basis of 950 B.T.U. per cubic foot, the heat requirement is 8,000,000 B.T.U. per ton of product. The coke equivalent of this heat is 600 lb. per ton of iron, assuming coke with 13,300 B.T.U. per pound. The total fuel requirement would then be 1,500 to 1,750 lb. of coke equivalent per 2,000 lb. of metal.

SUITABLE RAW MATERIALS

A choice between the various smelting methods, that is, use of a blast furnace, electric smelting furnace, or one of the sponge-iron processes, would be dictated largely by available raw materials. The types of ores that are most suitable for producing sponge iron in a rotary kiln at less than the sintering temperatures are listed below:

1. Pure ores with low silica contents, preferably below 3% silica. The use of the fine fraction of such ores, such as −¼ in. material, would be especially advantageous if the fine fraction existed in such a large proportion that it would have to be removed and sintered before the ore could be used in a blast furnace.

2. Ores that can be concentrated to a low gangue content by grinding and magnetic separation either before or after production of sponge iron. Most of the silica can be removed from many low-grade ores by selective grinding after reduction followed by magnetic separation, whereas other ores respond poorly to such treatment.

A wide range of solid fuels can be used for reduction in a rotary kiln. The ash content of the fuel does not seem to be important. As mentioned previously, railroad cinders containing 14% volatile matter, 46%

fixed carbon, and 42% ash were used successfully at the Laramie plant. A fuel with a low- or medium-sulfur content is desirable, as it may be difficult to obtain a sponge iron with satisfactory low sulfur content if the fuel is excessively high in sulfur. Low-volatile, non-coking coals are the simplest to use, as they may be charged into the kiln without preliminary treatment other than crushing. While high-volatile, non-coking fuels could be fed directly into a kiln, this would waste most of the volatile content of the fuel. Such fuels could be used most efficiently by first passing them through an externally heated kiln to remove the volatile matter. The volatile matter from the coal, after the recovery of valuable by-products, could be used as fuel to heat the main kiln.

An initial evaluation of raw materials can be made with small-scale tests, such as described in a Bureau of Mines report (21) in which an alloy retort 8 in. in diameter and 15 in. long was used. If the chosen raw materials will not reduce readily without sticking or balling, production of sponge iron in an internally heated kiln would prove very difficult.

DATA FOR COST CALCULATIONS

The Laramie plant was not operated long enough to provide actual cost-of-operation figures. Consequently, estimates will be prepared on the cost of equipment and man-hours to operate a plant, from which rough cost calculations can be prepared for any location and size of plant.

The total cost of the Laramie plant, including purchase of land and buildings, modifying and expanding existing buildings, and designing, procuring, and erecting machinery, was around $375,000. Taking into account increased prices since construction of this plant, it would seem that a two-kiln plant capable of producing about 100 tons of sponge iron per day could be built for around $700,000.

The rotary-kiln process is basically quite simple and could have low labour cost per ton of product if the plant were properly designed for automatic operation. Assuming the production of granular sponge iron from a pure ore so that the kiln product would require little or no grinding before final magnetic concentration and also assuming low-volatile fuel that could be fed directly into the kiln, the following personnel would be required to operate a two-kiln plant making about 100 tons per day of sponge iron:

3 shifts per day, consisting of	1 kiln operator 2 men for feeding kilns 2 men for concentrating product and storing it
Extra during day shift	1 plant supervisor 2 maintenance and repair men 1 chemist for analytical work 2 labourers for unloading cars and crushing materials

The above works out at about 1.7 man-hours per ton, so the cost of labour would be quite low for this process.

USE OF SPONGE IRON IN STEEL PRODUCTION

Most steel melters would probably consider granular sponge iron a rather poor replacement for scrap in steel-melting furnaces. This may be true for plants designed to handle scrap, but granular sponge iron would have real advantages in a plant designed for its use. While sponge iron can be used to a limited extent in open-hearth furnaces (22, 23, 24) it is much better suited to electric furnaces, since no oxidation of carbon and silicon is necessary. While granular sponge iron is satisfactory as the entire metallic charge of an electric furnace, it normally would be used with considerable scrap from fabricating and processing applications. Such scrap contains voids, which could be filled largely with granular sponge iron to produce a dense charge.

Granular sponge iron is much easier to handle than scrap, since small conveyor systems, such as pneumatic or belt conveyors, can readily be used. Charging granular sponge iron into an electric furnace could be accomplished by a chute feeding material through one of the electrode holes in the roof.

However, other means of using rotary-kiln sponge iron may be even more attractive. The sponge iron could be briquetted into blanks, heated, and drop-forged into finished synthetic wrought-iron parts.

In some applications it might be advantageous to discharge the sponge iron from a rotary kiln directly into a melting furnace and produce a metal low in silicon, phosphorus and sulfur. This has been proposed by Harman (25), and a company has plans to try this method at the Laramie sponge iron plant.

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Summary of Discussion

Production of Sponge Iron According to the Wiberg-Söderfors Method, ***presented by Mr. Kalling in the absence of the author***

Mr. KALLING prefaced his remarks by stating that as the process was well known, he would merely give some points of view concerning the process and the new developments over the last two or three years. It was the only process in large-scale operation in which reduction took place without mixing the ore with solid carbon, which had some great advantages. It gave, for instance, the possibility of using oil or natural gas instead of coke in the process. On the other hand, the process was not suitable for all kinds of ores, such as the hitherto tested Venezuelan ores, without pretreating the ore.

Mr. CAVANAGH observed that the Wiberg-Söderfors method was used as a yardstick to gauge the efficiency of other methods, as it used 7.5 million B.T.U.'s per ton of metallic iron. When deciding whether a new process had any value at all and was worth further investigation, the first step was to see whether it measured up to that standard.

Mr. YDSTIE referred to Mr. Kalling's remark on Venezuelan ore, and said that he knew considerable study had been devoted to the possibility of using the process for Venezuelan ore; he wondered, therefore, whether Mr. Kalling or Mr. Cavanagh could give any information as to whether any experiments had been made to make those ores suitable for the process.

Mr. KALLING replied that he was not prepared to go into the matter just then.

The Basset Process for the Production of Pig Iron in Rotary Kilns, ***presented by Mr. Canguilhem in the absence of the author***

Mr. MERCIER opened the general discussion of the paper, with an expression of regret at the absence of Mr. Jensen, as he had an observation to make in connexion with the comparison made in the paper between the ore-reducing processes and the blast furnace, which produced 800 tons of pig iron per day. He could not believe that either the electric pig iron furnace or the Basset and Wiberg processes were as economical as the blast furnace producing 800 tons a day, since their output was only about 100 tons a day.

The important point to remember was that the pig iron was intended for steelmaking, and its composition was interesting for those plants making special steels. The process was also of interest for use with a rich ore and a fuel with an ash content that was not too high.

He had seen the Basset plant in operation and had received a good impression; production, however, did not exceed 200 tons a day—although that figure might be increased later—as compared with other furnaces making 300, 400 or 500 tons a day.

Mr. DURRER said that the results of a visit he had made to the plant in Denmark led him to coincide with the views expressed by Mr. Mercier.

Mr. ALLARD said that he had had the opportunity of seeing a kiln operating in France; it should be noted that the iron content had to be low enough to get a clinker that was within 6% alumina when making a clinker for cement.

On the basis of the previous speaker's remark that the richer the ore, the higher the production of the kiln, the process could not be adopted for the production of iron derived from very poor ores such as those in France. It might, on the other hand, be a very good method for countries which had rich, fine ores and poor fuel, and there were such countries in Latin America.

Mr. GONZALEZ VARGAS and Mr. MERCIER pointed out that the low silicon content of the resultant pig iron precluded its use in foundry work, without the prior addition of ferrosilicon, which added to costs.

Mr. CANGUILHEM replied that the author had mentioned that drawback, which was one of the limitations of the process.

The Rotary Kiln Processes for Sponge Iron, *presented by the author*

Mr. KALLING opened his presentation by saying that he had been in doubt as to whether to speak about the experiments in making sponge iron in rotary kilns, the reason being that the processes developed were not yet in use for making commercial steel. The results were very promising, however, and he had therefore described them in his paper.

It was interesting that two papers had been presented, one describing a process in which the iron was melted and not the slag, and the other in which the slag was melted but not the iron. The process described in his paper was one in which neither the iron nor the slag were melted, and that was the real sponge iron process. He then briefly summarized the two methods for making such sponge iron, as described in the paper. The main problem was not to make the furnace big, but to obtain a simple operation. There were some essential advantages in the Domnarfvet method, which were indicated in the paper.

Mr. GONZALEZ VARGAS said he wished to make a contribution on the subject, which was a little-known one. About sixteen years previously, another engineer and he himself had taken out patents for reduction of ores in a rotary kiln, similar to the Krupp Renn process, but with the difference that they had not attempted to make either solid or liquid sponge as in the Krupp Renn process, but something in between, the sponge being solid inside but covered externally with a thin coat of residue. A small furnace had been constructed and operated, producing 4 tons a day. It worked well with rich ores and charcoal as well as with coke and bituminous coals. It could be considered as worth developing on an industrial scale, but that was never done and it was abandoned due to lack of opportunity.

Production of Sponge Iron in a Rotary Kiln at Temperatures Below the Fusion Point of the Material, *presented by the author*

Mr. BUEHL, after presenting his paper, said that one of the interesting questions which might apply to the Latin-American countries was that a very high ash fuel, up to 30% ash, could be used in the process without adversely affecting the sponge. Work had also been done by the Bureau of Mines using a method similar to that used by Höganäs in Sweden. The Ontario Research Foundation had continued the work under the direction of Mr. Cavanagh, who was present at the meeting. There was a very fine installation using the process described, at the steel plant of Hojalata y Lamina, in Monterrey, Mexico. Sponge iron was being made very cheaply at that plant and was then melted in an electric furnace. Señor Bernardo Garza Sada, who was responsible for the operation of the plant, was also present and could probably give more detailed information on the results obtained.

Mr. CAVANAGH remarked that approaches had been made to his organization in Canada, asking whether the rotary kiln could be used with a certain Canadian ore or a certain Canadian coal. That could only be verified by tests; moreover, the results obtained from a pilot size plant had to be transferred to a commercial kiln, and the results did not always correspond. He would therefore like to know whether it were possible to proceed direct from the pilot to the commercial scale, with some degree of accuracy.

Referring to tunnel kilns, he explained that the process had been developed as a result of a severe scrap shortage in Canada some years before. Assistance had been received from the Bureau of Mines, but by the time a pilot commercial operation was achieved, there had no longer been a scrap shortage. The information was therefore available, but there was little use for it. The work of the firm in Mexico, however, made it possible to observe facts instead of giving opinions, and he believed that firms in South America using electric furnaces would find it worth their while to investigate the process, as it gave very good and profitable results in Mexico, and could do the same in several South American countries.

Mr. KALLING, in answer to Mr. Cavanagh's first question, said he did not think it was possible from the results in the pilot plant to ensure that those same results would be obtained on a commercial scale, but that there were in fact very few methods where such was the case. Special methods had been developed at Domnarfvet and Söderfors for testing ores and discovering their suitability for the sponge iron processes, and some fairly accurate conclusions had been reached. Of course a commercial size furnace had to be erected and the ores in question tested in it before the suitability of the process could be definitely decided. He agreed with Mr. Cavanagh that the tunnel kiln sponge iron process could be used for many different kinds of fuel and ores, without testing, but it was not any great disadvantage to be compelled to make tests before using the different ores and fuels on a commercial scale, if the process had any possibility of producing a cheaper iron.

Mr. PRADO UCHOA suggested that as Mr. Garza Sada was present, he might give some details of investment costs and all possible data on the Monterrey plant.

Mr. GARZA SADA replied that the only data he could give was that production was 20 tons per day at a price much lower than that for scrap in Mexico; the sponge iron had been melted at a much lower cost than the prevailing one for melting scrap. It was planned to increase production to 5,000 tons per month, which would later be re-melted.

Mr. CAVANAGH pointed out that the bed in the rotary kiln was very different from that used in other processes, and it was very difficult to form an opinion as to the effect of the depth of the bed. He had in mind one specific instance where a rotary kiln had been running for some time and on the basis of the results, another kiln had been built, which was nearly twice as large, yet its output had been only 25% higher than that of the first kiln.

Mr. JOHANNSEN pointed out that it was absolutely necessary, when using rotary kilns, to have correlation between the two furnaces when taking the step from the small furnace to the large-scale operation; in his experience, the results were related to the inside volume of the furnace. He had tried using the resulting figure in several processes, and it was always the same.

Mr. GONZALEZ VARGAS urged the need for knowing figures in order to have a clear picture as to the possibility of developing the Höganäs-Cavanagh process in Latin America. He would like information as to the investment costs per ton/year, the manufacturing costs and those for materials.

Mr. CAVANAGH explained that the information on the Mexican operation was the property of the Mexican company. He could, however, give the following figures on Canadian costs:

$100,000 for the materials;

40,000 man-hours to erect the plant;

$4,000 to equip it with ceramic slaggers;

$60,000 to $70,000 to equip it with metal slaggers;

$40,000 to $50,000 for other equipment, not including the building.

In other words, the rough total was $250,000 for one kiln with a capacity of 30 tons per day. Labour costs for the process were extremely high in Canada, to the extent of making it prohibitive as long as the price of scrap remained low. It was not the cost of the sponge iron that was important, but the cost of the steel made from it. The Monterrey company was a concrete example of a company operating on sponge iron, and he believed that the production of sponge iron was beneficial to the steel operation because the correct melting of sponge iron was cheaper than melting scrap in a furnace.

Mr. GONZALEZ VARGAS said he had not been concerned with the cost of melting sponge iron instead of scrap, as his question had referred only to comparison between several processes making sponge iron, in order to obtain a clearer idea; he had wanted figures to compare the Wiberg process with, for instance, the Höganäs-Cavanagh process, and not to compare scrap iron with sponge iron.

Mr. CAVANAGH replied that he would be giving a comparison between those two processes at a subsequent meeting. Under Canadian conditions the construction cost of the Wiberg process was approximately three times as high as the tunnel kiln method. However, labour costs in Canada were three to four times as high for the tunnel kiln as for the Wiberg process, but that was a favourable factor in Mexico, where the labour cost per man/hour was very much lower than in Canada. He added that there were quite a number of people in Sweden with experience in melting sponge iron and they preferred it to scrap in some cases. The Mexico plant was the first in the world to use sponge iron for making rimmed steel, and he congratulated the firm on its enterprise. It was making achievements which he had believed impossible; for instance, the Mexico firm was using coal, whereas he had never thought anything but coke could be used; labour was being used in a way that could not be imagined in Canada, giving rise to quite different possibilities from those pertaining in Canada. The result was an unusual example of an idea developed in Sweden, investigated further in the United States and then in Canada, and receiving its first practical, profitable use in Mexico.

Mr. PABLO SADA, after explaining that he himself was an open hearth man, belonging to a rival firm in Mexico, congratulated the Monterrey firm. On the question raised by Mr. Gonzales Vargas, he noted that Mr. Buehl's paper stated that a two-kiln plant, producing 100 tons a day, could be built for $700,000, or in other words, almost $19 per ton of annual capacity.

He had two further questions to raise: first, he wished to know what was the largest plant currently in commercial operation, and the number of tons per month produced by such a plant; in the second place, he wondered whether sponge iron was used in connexion with open hearth or only with electric furnaces.

Mr. KALLING replied that he could only give the quantity produced by the Wiberg-Söderfors process and the Höganäs process. It was not thought desirable to guarantee the Wiberg furnace for more than 20,000 tons a year, and more furnaces would have to be installed to achieve a higher output. In Sweden six such furnaces would be in operation within the next few years, producing 120,000 tons altogether, which would be used for open hearth as well as for electric steelmaking.

Concerning the Höganäs process, the plant at Höganäs was currently producing up to 30,000 tons a year, and a plant was being built in the United States with a similar capacity. A third plant was being built in Sweden, also with a capacity of 30,000 tons a year.

Mr. BUEHL referred to Mr. Cavanagh's question, stating that while Mr. Johannsen had said that capacity was proportionate to the internal volume, it was more proportionate to the square of the dimensions than the cube. In the large kilns at Laramie the ratio of the smaller diameter to the larger was too great, and the furnace would overheat at the junction of the two diameters, and interfere with operation. He described the discharge mechanism, explaining that the finer material would tend to ride on the inside of the charge, which limited the capacity of the large kiln. If that factor were eliminated, Mr. Johannsen's statement might possibly hold true. The decision as to which rotary kiln process to adopt depended on the raw materials, the product desired, and so on. Quite possibly, if a rotary kiln were installed for producing sponge iron, the various processes would all be tried, and the one eventually adopted might not be the one originally selected.

STEEL MAKING AND FINISHING

Comparison of economics of different processes of steelmaking

23 October 1952—Morning

Chairman:

Joaquín PRIETO, Assistant Manager, Empresa Siderúrgica Nacional de Paz de Río, S.A., Bogotá, Colombia

Contributed Papers:

The Acid Bessemer Process
W. O. PHILBROOK, Associate Professor of Metallurgical Engineering, Carnegie Institute of Technology, Pittsburgh, Pa., U.S.A.

Basic Open-Hearth Steelmaking Practice in the United States of America
William C. BUELL, Jr., Consultant, Arthur G. McKee & Company, Cleveland, Ohio, U.S.A.

Economics of the Modern Acid Open Hearth Practice
G. R. FITTERER, Dean, Schools of Engineering and Mines, University of Pittsburgh, Pittsburgh, Pa., and Director of Research, Acid Open Hearth Research Association, Inc., Pittsburgh, Pa., U.S.A.

Manufacture of Thomas (Basic Converter) Steel at Paz de Río, Colombia
Edouard DECHERF, Chief Engineer, Empresa Siderúrgica Nacional de Paz de Río, S.A., Bogotá, Colombia

Summary of Discussion:

Participants: Messrs. PHILBROOK, COSTA LINO, SANITER, ALLARD, BUELL, FITTERER, DECHERF

The Acid Bessemer Process

W. O. PHILBROOK

Since the subject of this paper is the acid Bessemer process, the major portion of the discussion will be devoted to the production of Bessemer ingots. It is to be emphasized, however, that the applications for which Bessemer steel is best suited are quite limited. Despite the obvious attractiveness of the Bessemer process[1] in the expansion of the Latin-American industry because of a scarcity of scrap and an abundance of suitable low-phosphorus ores, it does not seem wise to count on the conventional Bessemer to become the major method of steel production. For this reason, some attention will be given to the duplex process and to variations in the converter process which may produce steel having better properties and broader applicability.

POSITION OF THE BESSEMER PROCESS IN THE UNITED STATES AND IMPLICATIONS FOR THE LATIN-AMERICAN INDUSTRY

A brief review of the changing importance of the Bessemer process in the United States is pertinent to planning the development of the steel industry in Latin America. The Bessemer process was the first of the large-scale steelmaking processes to be invented, and it immediately made possible a rapid expansion in steel production and industrial growth in both Europe and North America. The converter process achieved commercial success in the United States in 1864. The open-hearth furnace was introduced to America about 1867, and the basic open hearth was on a commercial basis in 1888. The latter process surpassed its acid counterpart within a few years and grew rapidly to become the leading method of steel production. The growth of the basic open-hearth proc-

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[1] The term "Bessemer process" in this paper will refer always to the "acid" converter process, since the basic converter process is generally called the Thomas process.

ess was a joint result of the increasing availability of steel scrap from manufacturing operations and the obsolescence of machinery, the greater abundance of non-Bessemer ores, and the greater versatility of the basic open hearth with respect to both suitable raw materials and variety and quality of products. The electric furnace process came into importance during the First World War, but its real growth has come during and since the last war. No Thomas steel is produced in the United States because our ores are not high enough in phosphorus to make that process economically feasible.

The Bessemer process now accounts for less than 5% of United States steel production, 4.65% in 1951, and the present trend is downward. The contributions of the other processes in 1951 were: basic open hearth, 87.82%; acid open hearth, 0.74%; and electric furnace, 6.79%. The trend for the past three years has been for electric furnace production to increase more rapidly than basic open hearth output.

The total converter capacity (1) of the United States is about 11 million tons,[2] but the facilities assigned to the production of Bessemer ingots and castings as of 1 January, 1952, were 38 vessels having a rated annual capacity (2) of 4,881,000 tons. Three vessels rated at 123,000 tons capacity have been dismantled during 1952. This is evidence that the Bessemer converter is more important in the United States industry to provide metal for the open hearth than as a primary ingot producer. It is doubtful if any new Bessemer capacity will be built except in conjunction with duplex facilities.

In view of this growth pattern in the States, it seems important in the projection of the Latin-American industry to provide for steel of open hearth quality for the majority of applications, but the problem of limited scrap availability must be met. A duplex shop combining acid converters and basic open hearths does this admirably and provides a highly versatile installation. Bessemer ingots may be made for those applications for which they are suitable, and with proper care an open hearth steel fully equivalent to conventional scrap-practice basic open hearth steel can be produced. The operation can be independent of purchased scrap or may even produce an excess of scrap for sale, or scrap may be melted in the open hearths if a cheap source becomes available. Because of the all-liquid charge, production rate of the open hearths is high. The blast furnace capacity required is little, if any, greater than for the same tonnage of straight Bessemer ingots, and the limitations on phosphorus and sulfur content of the iron are less severe. A possible alternative would be the intensive development of some of the innovations, to be described later, which are designed to produce steel equivalent to open hearth product by converter processes, but their commercial success and range of usefulness are not yet fully established. The situation would, of course, be drastically different with some other method than the blast furnace used to reduce iron ore, but it is not the function of this paper to discuss smelting processes.

Applications and specifications for Bessemer steel

Because of low carbon content and the complete absence of alloying elements (from North American Bessemer-grade ores), Bessemer steel has excellent weldability and is eminently suited to the manufacture of welded pipe. Bessemer steel is superior to ordinary open hearth steel where machinability is the sole consideration and in certain applications where a high degree of stiffness for a given low carbon content is required. Because of its high nitrogen and phosphorus contents, ordinary Bessemer steel is inferior to open hearth steel in shock resistance and is more sensitive to work hardening and to ageing or embrittlement after cold work. The effects of nitrogen can be overcome by full deoxidation to produce a fine-grained steel, and they would be less restrictive to applications of unkilled Bessemer steel if ores of low enough phosphorus content were available to yield steel analysing under 0.05% phosphorus and if precautions were also taken to hold the nitrogen content down (3, 4, 5, 6).

The principal applications of Bessemer steel in the United States are butt-welded pipe and electrical conduit, seamless pipe, free-machining bars, flat-rolled products, and wire. The only structural applications for which Bessemer steel is acceptable under ASTM specifications are plates, shapes and bars not over $\frac{7}{16}$ in. in thickness or diameter which will not be subjected to dynamic loads in service, and concrete reinforcing bars. Permissible railroad applications are low-carbon tie plates and track spikes; rails are no longer made from converter steel in the States.

Bessemer steel is normally made to two plain-carbon and three resulfurized grades as listed in Table 1. Killed Bessemer steel for seamless tubes may be made to carbon contents ranging up to about 0.35%, with about 0.85% manganese. Where the steel is ordered to physical specifications rather than chemical analysis range, the American Iron and Steel Institute standard limitations are 0.11% max. phosphorus and 0.06% max. sulfur. Some companies prefer to hold the sulfur content below 0.035% for use in their own mills, especially for seamless pipe. Silicon content ordinarily is not specified for Bessemer steels and varies according to the deoxidation practice for the ingot structure desired. Alloying elements are neither specified nor analysed for, with the exception of copper if copper-bearing steel is ordered.

Table 1

American Iron and Steel Institute standard acid Bessemer steels(7)

AISI No.	*Ladle analysis limits, per cent*			
	C	*Mn*	*P*	*S*
B 1006	0.08 max.	0.45 max.	0.07/0.12	0.060 max.
B 1010	0.13 max.	0.30/0.60	0.07/0.12	0.060 max.
B 1111	0.13 max.	0.60/0.90	0.07/0.12	0.08/0.15
B 1112	0.13 max.	0.70/1.00	0.07/0.12	0.16/0.23
B 1113	0.13 max.	0.70/1.00	0.07/0.12	0.24/0.33

In order to complete the picture on specifications and composition limits, the permissible variations for check analysis for Bessemer steels are summarized in Table 2. These limits provide for normal variations resulting from segregation. Only if the analysis of a sample from a finished piece (pipe, bar, wire, etc.) departs from the specified range of ladle analysis by more than the limits quoted is it considered that there is reasonable basis for controversy over conformity with the specification.

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[2] All numerical values in this paper are in metric units and the word "ton" will always mean 1,000 kg.

Table 2

SUMMARY OF AMERICAN IRON AND STEEL INSTITUTE CHECK ANALYSIS LIMITS FOR ACID-BESSEMER STEELS (7)

	Standard variations for check analysis, per cent				
	C		*Mn*	*P*	*S*
Type of product	*Under*	*Over*	*Under or over*	*Over max.*	*Over max.*[a]
Welded pipe, plate, structural, sheet, etc.	0.02	0.03	0.03	0.01	0.01
Bars, seamless tube	0.02	0.02	0.03	0.01	0.008
Wire, wire rod	0.01	0.01	0.02	0.01	0.005

[a] No check analysis applicable to resulfurized steels or to S max., in excess of 0.06%.

BESSEMER PLANT

A typical Bessemer shop comprises two to four converters together with their turbo-blowers, control pulpit, mixers, scrap-handling facilities, ladles and transfer cars, pouring pit, bottom house, and necessary crane and railroad services. A common installation is one of three 25-ton vessels, with two normally on blast and one out of service for relining at least part of the time. Such a plant has a monthly capacity of about 90,000 tons of ingots. Most of the newer Bessemer shops, with one notable exception, are built adjacent to open-hearth furnaces to take advantage of duplex operation. The open hearths may be stationary or tilting furnaces, the latter being more common for duplex practice. A typical plant of three 25-ton converters and three 180-ton open hearths will produce 90,000 tons or more of ingots per month, of which 25 to 40% might be Bessemer ingots.

There are two general arrangements of a Bessemer shop. In the newer of these, the vessels are in the open with a minimum of structural work. There is a working platform and charging track along the front of the line of vessels a little below trunnion elevation. The mixers are located at one end of the line of converters and the scrap-loading facilities at the other. Both scrap and molten iron are charged with the vessel in horizontal position by electrically-operated, remote-controlled charging cars. Blanking of tuyeres and other work on the bottoms is done from an electrically operated repair car, and bottom changes are made by means of an electric-hydraulic jack on a bottom car. The converter bay is spanned by a gantry-type crane, so no overhead structural work is required to support or shelter crane runways. This type of plant is the cleanest and safest to operate, but requires more ground area than the older shops.

The second general type of arrangement of Bessemer facilities is usually enclosed except over the mouths of the converters. The molten iron is transported from the mixer scale or transfer car and charged into the vessels by a travelling bridge crane. Between each pair of converters there is a scrap elevator and a covered charging platform high above the vessels. Scrap and other solid materials weighed into buggies are raised to the platform and charged through movable chutes into the vessels while they are in the upright position. A heavy working floor is built between the vessels. Although such a layout is compact and economical of space, the collection of vessel spittings on platforms, elevators, crane runways, and overhead structural work presents a maintenance and safety problem.

Converter

The most common vessel for ingot production is one of 23 to 27 tons capacity, although smaller ones are in use, down to 2 to 6 tons for foundry purposes; and one company is planning to install 36-ton vessels to supply blown metal for duplexing. The shell is fabricated from steel plate by welding or riveting, with cast bottom and nose ring. Steel is preferred to cast iron for the castings because of ease of repair by welding.

The shell of the vessel is lined with natural white sandstone or "firestone," or with a laminated siliceous stone called "mica schist." Although practice varies a little, it is common to use firestone for the shaft and mica schist at the nose and shoulder (where the bottom is joined to the shell), because the latter material is more expensive but has better resistance to erosion in service. The stone is set in a plastic mud of ganister and fireclay. The initial thickness of the lining is usually 40 to 46 cm. A new lining is dried on the stand with a gas flame for 14 to 18 hours before the vessel is put in service.

Lining life varies considerably depending on the type of operation. If the converter is used exclusively to produce "high" blown metal for duplex use, the lining tends to build up with siliceous deposits, especially in the belly. Lining erosion is most rapid when only low-carbon steel is being produced. As an illustration of the variability of lining life and practice, the experience of two large producers may be cited. Both make 25-ton blows in combination ingot and duplex operation. One company obtains about 10,000 tons on a new lining before a patch is required, which may include partial or complete replacement of the nose and shoulder sections. After about 18,000 to 23,000 tons have been produced, a veneer lining is put in to restore the lining to its original thickness. Several veneer linings with intermediate patching may be used before a complete relining is necessary. Another company, using a somewhat thicker lining, obtains 12,000 to 20,000 tons on a new lining before patching and a total life of 25,000 to 40,000 tons with the aid of several patches before a complete relining.

The converter bottom is invariably removable, since it has a relatively short life of 30 to 40 blows. The bottom is fitted with 28 to 30 active tuyeres, the three to five tuyeres which will be at the bottom when the vessel is turned down being blanked when the bottom is made up. The tuyeres are of good quality fireclay, medium burned, about 86 to 91 cm. long and 14.9 to 16.5 cm. in diameter, and usually have either seven cylindrical holes of 15.9 mm. diameter or ten holes of 12.7 mm. diameter. Long, rectangular fireclay tile brick are set among the tuyeres to stiffen the bottom. The bottom is rammed up with a stiff mud of ganister and clay, or a thinner mix is poured in like concrete, depending on preference at the individual plant. The clay is a mixture of about two parts blue clay to one part of white plastic fireclay and constitutes about 28 to 30% of the dry mixture. The clay and ganister are thoroughly ground together and tempered with water in a grinding pan. New bottoms are dried for two or three days at temperatures of 300-400°C. in continuous gas-fired ovens.

During the life of a bottom, tuyeres which are observed to "bore" or fail prematurely are "blanked" by removing the bottom of the wind-box and closing the lower end of the tuyere with clay and a blanking plate. The bottom is changed after 10 to 12 tuyeres have been blanked or when the tuyeres have eroded down to a length of about 20 cm. A bottom change normally takes 25 to 30 minutes.

Air blast

Turbo-blowers supply the blast in all the newer Bessemer installations, although some of the older shops still use steam-driven reciprocal blowing engines. The blower for a 23- to 27-ton converter should have a capacity of about 1,000 m^3/min. (volume at 15°C., 76 cm. of mercury) at a gauge pressure of 2.1 to 2.5 kg/$cm.^2$ Normal blowing rate is about 800-1,000 m^3/min. at 1.8 to 2.1 kg/$cm.^2$ gauge pressure at the pulpit. Ideally, there should be one turbo-blower for each converter, with inter-connections so that any blower may supply wind to any vessel. The blast line is tapped with a 7.8 cm. diameter steam line carrying steam at about 10-12 kg/$cm.^2$ pressure, or perhaps two steam lines if the pressure is lower, for use in cooling the metal during a blow. Control valves for air and steam, together with blast pressure and flow-rate indicators for each vessel, are located in the pulpit. A substantial amount of experience with oxygen-enriched blast has been obtained in the past few years (8, 9).

Mixers for molten iron

A Bessemer shop should be equipped with at least two hot metal (molten pig iron) mixers having total storage capacity for the amount of iron needed for at least eight hours of operation, preferably more. This will provide a reserve of iron to keep the converters in operation if blast furnace output is slackened by tuyere changes or other causes or to store iron if the converter shop is temporarily down. Mixers in use vary in capacity from a few hundred to over 1,200 tons capacity. A reasonable arrangement for a shop producing about 3,000 tons daily of Bessemer ingots and blown metal for duplex is two 750-ton mixers, one of which is used for Bessemer and one for basic iron. A spout for pouring directly from the blast furnace iron ladle to the transfer ladle should be provided for use when a mixer is out of service for relining or other cause. The mixer should be equipped with a "dead man" control to guard against human failure and with a pneumatic piston which will automatically return it to upright position in the event of a power failure while iron is being poured. Mixers vary considerably in design and may be lined with mica schist or with refractory brick. The mixer has been abandoned by at least one plant, which uses mixer-type or "submarine" ladles of 180-ton capacity to transport iron from the blast furnaces and store it until it is needed, when it is poured directly into the transfer car.

Pouring facilities

For ingot production, the finished heat from the converter is emptied into a bottom-pour or teeming ladle equipped with nozzle and stopper. The product of two blows may be combined in one ladle to make a single 50-ton heat if the pouring-pit crane has sufficient capacity. Blown metal for duplex use is transported to the open-hearth furnaces in lip-pour ladles of the same design as the transfer ladles for molten iron.

Ingot moulds are usually of the big-end-down type and are open-top or bottle-top depending on the type of ingot to be produced. The moulds are of cast iron and may be made directly from blast furnace iron or purchased from independent mould foundries. Ingot moulds for billets and blooms are generally about 50 to 65 cm. square, or may have equivalent rectangular cross-sections 5 to 10 cm. off-square, and about 200 cm. high. Ingot weights range from 4,100 to 6,400 kg. Slab moulds for flat-rolled products are commonly 6 to 10% wider than the slab to be rolled, with the ingot thickness about half the width or somewhat less. In a duplex shop, the same moulds and pouring facilities are used for both Bessemer and open-hearth ingots.

Raw materials

The chief limitation on Bessemer raw materials is related to the fact that no phosphorus or sulfur can be eliminated during an acid process, but instead the concentration of these elements increases about 10% during the blow as a result of the elimination of other elements. Additional restrictions are imposed by the heat balance of the process and by chemical limitations on slag analysis to minimize slopping during the blow.

Bessemer ores

The Bessemer ore of North America comes largely from the hematite deposits of the Lake Superior region, primarily from the Mesabi and Gogebic ranges in Minnesota and Wisconsin and from the Michipicoten and Steep Rock deposits in Canada. The recently opened Quebec-Labrador field provides a new reserve of excellent low-phosphorus ore. Some sintered magnetite concentrate from the New York magnetites is also used in Bessemer iron burdens.

In order to classify as Bessemer grade, an iron ore must contain a minimum of 51.5% Fe and not more than 0.045% P on a natural basis. The manganese content of a particular ore is not critical provided a burden can be blended to produce iron to the required manganese range, which will be discussed below. Sulfur, silica and other gangue constituents usually impose economic rather than metallurgical limitations on the use of an ore. Contamination by metallic elements has not been a problem with North American ores. Typical analyses of Bessemer ores are given in Table 3.

Bessemer iron

The composition of molten iron desired for blowing to Bessemer ingots varies somewhat from plant to plant but usually falls within the following ranges:

Silicon	*Manganese*	*Phosphorus*	*Sulfur*
1–1.6%	0.4–0.6%	0.09% max.	0.04% max.

Carbon content is not specified and is seldom determined, but it will range from 4 to 4.5% depending on composition and temperature of the iron. If the blown metal is to be used for duplexing, the phosphorus content may run considerably higher than given above, especially in southern practice.

The silicon content desired by the Bessemer department may run to the upper side of the range given if there is a deliberate intention to use the maximum possible amount of scrap or if there is a large temperature drop in the iron between blast furnace and converters because of poor handling and storage facilities for hot metal. Silicon to the low side of the range is possible with physically hot iron and is favourable to a short blowing time and hence high rate of production and to minimum scrap consumption.

Table 3

TYPICAL ANALYSES OF NORTH AMERICAN BESSEMER-ORES

	Analysis, natural basis, per cent										
Ore field	*Fe*	*P*	*SiO_2*	*Mn*	*Al_2O_3*	*CaO*	*MgO*	*TiO_2*	*S*	*Ign. loss*	*Moisture*
Mesabi	53.08	0.033	9.04	0.34	0.75	0.07	0.06	—	0.006	4.13	9.50
Gogebic	53.48	0.033	10.84	0.09	1.02	—	—	—	0.010	1.67	9.97
Steep Rock	51.38	0.027	5.67	0.23	1.40	0.15	0.09	—	0.027	—	10.34
Michipicoten [a]	50.50	0.016	11.00	2.87	1.57	2.98	7.36	—	—	0.94 [b]	1.13
New York [c]	65.0	0.028	5.73	0.33	2.99	0.38	0.36	0.81	0.039	—	—

[a] Sinter.
[b] Gain on ignition.
[c] Sintered magnetite concentrate, dry basis analysis.

The analysis of the iron is important in relation to the composition and properties of the slag which will be formed in the converter. A stiff or semi-solid slag is necessary to avoid "slopping" or violent ejection of large quantities of slag and metal during the blow. Too high an MnO content in the slag is the most common cause of slopping, the general experience of Bessemer operators being that the silicon to manganese ratio in blowing iron should be no less than 2.5:1 for satisfactory blowing. There is a general opinion among Bessemer operators that titanium in the iron causes slopping, but the investigation reported by Edwards (10) showed that titanium up to 0.2% in the iron, or TiO_2 content of the slag up to 7%, had little or no adverse effect on blowing performance. Even a small increase in the lime and magnesia content of the vessel slag from blast furnace slag carried over with the iron will lead to slopping, so it is important to skim kish from the transfer ladle if necessary to assure clean iron going into the converter.

Physically hot iron is a great aid to fast and efficient operation. For any given blast furnace practice, there is, of course, a direct relation between casting temperature and silicon content. In United States practice, the casting temperature for Bessemer iron is usually within the range from 1,450 to 1,510°C. The temperature drop between the blast furnace runner and the converter depends markedly on the method of handling the iron and on transit time. The temperature loss is most rapid during the first hour and a half and with open-top ladles. With reasonably good handling facilities, the temperature of iron entering the converter runs around 1,370 to 1,430°C.

Scrap

Clean billet crops and other heavy rolling-mill scrap in lengths under 1 m. are generally used in blows for ingot production. Vessel and pit scrap may be used in blown-metal heats for duplex use where control of temperature and composition is not critical. Light scrap is sometimes added to iron ladles at the blast furnace if the iron temperature is high enough to permit this method of extending the supply of hot metal.

Ferroalloys

The alloys used for adjustment of composition and for deoxidation are well-standardized commercial products, which are available in lump or crushed form in a variety of size ranges. The analyses of the ferroalloys commonly used in Bessemer practice are given in Table 4. Aluminium of 85 to 95% purity may be obtained as "shot" of about 5 to 10 mm. size or as notched bars. Scrap copper or roll sulfur of nearly theoretical purity are used when needed for copper-bearing or resulfurized steels.

Table 4

GRADES OF FERROALLOYS

	Composition, per cent						
Alloy	*Mn*	*Si*	*C*	*P max.*	*S max.*	*Ti*	*Al*
Ferromanganese, standard	78–82	1.25 max.	7.50 max.	0.35	0.05	—	—
Ferrosilicon, 50%	—	47–52	0.15 max.	0.05	0.04	—	—
Ferrosilicon, 85%	—	80–90	0.15 max.	0.05	0.04	—	—
Ferrotitanium, high C°	—	2–3	6–8	—	—	15–18	1–2

furnace conditions, and these two factors have more than compensated for the adverse effect of high sulfur.

These developments again focus attention upon the blending operation as a primary process step in preparing of coking coals where the natural raw materials are variable and uncertain.

WASHING PROCESSES

Modern coal-washing plants are usually designed around one of two primary coal-washing units: (*a*) the Baum-jig wash-box (handling at present 75% of all bituminous coal washed), or (*b*) the heavy-medium washing vessel.[1] It may safely be assumed that one or the other of these washing methods, which now handle at least 90% of the coal-washing load in the major coal-producing countries, will be adapted to handle most Latin-American jobs. (1)

The general performance characteristics of the two leading washing methods point to the use of the Baum system in easy and moderately difficult situations, and the heavy-medium process in special situations involving treatment of a very difficult raw coal or the preparation of a special product.

EXAMPLES IN LATIN AMERICA

More specific ideas can be developed by applying these standard processes to some typical Latin-American coal-preparation problems.

Lota and Schwager coals of central Chile

Figures 3 and 4 give the washability characteristics of a typical sample of coal from central Chile. These data, taken from a report of examination of these coals in 1944–45 (2), have been adjusted by adding 10% to the heaviest sink fraction to approximate the washing problem that might be presented after the mining operation has been further mechanized.

An estimate of Baum-jig performance on the coal with the integral washing method applied to the 38 mm.–0 size and adjusted to an effective washing gravity of 1.60 in the 38–18 mm. size, shows a yield of 84.8% of washed coal of 3.3% ash content. More complete data are shown in Table 2.

Table 2

ESTIMATE OF BAUM-WASHERY PERFORMANCE ON THE LOTA COAL

Size range	*Washing gravity*	*Yield*	*Ash*	*Refuse ash*
37 – 18 mm.	1.60	88.5	2.3	78.4
18 – 10 mm.	1.65	88.7	2.4	78.4
10 – 5 mm.	1.80	80.5	3.1	80.8
5 – 2.5 mm.	1.88	81.4	3.4	82.1
2.5 – 1.5 mm.	2.00	82.4	4.3	83.1
1.5 – 1.0 mm.	2.10	84.2	5.3	82.5
1.0 – 0.3 mm.	2.25	84.4	7.6	75.8
Total and average		84.8	3.3	82.0
0.3 – 0 mm.		3.1	11.1	
Total including sludge		87.9	3.6	82.0

Comparison of these estimates with the float-and-sink yield-ash curves indicate that this is virtually complete recovery of the available coal, and no further treatment is justified.

These coals represent almost the extreme case in ease of washing, so far as technical performance is concerned, with all the near-gravity fractions in the raw coal in negligible proportions. The washing errors in the near-gravity range are infinitesimal in importance in affecting the yield- and ash-content of products. Hence, it is an ideal coal for washing by the single-unit integral method without re-treatment of the fines.

Selection of a flow-sheet and plant design in such a case becomes primarily a matter of cost and mechanical operativeness. Major emphasis is placed upon sound over-all plant design for continuity of operation, ease of maintenance, precision of performance control, and efficiency of auxiliary operations such as product moisture control, sludge recovery, and water clarification.

These auxiliary operations are of major importance in the total job of coal preparation, but are in the area of mechanical design problems and outside the central theme of this paper.

Cauca Valley Coals

The Cauca Valley coalfield of Colombia, already discussed at some length in the introductory part of this paper, has a wide variety of bituminous coals ranging from easy to moderately difficult in washing characteristics and from fragile to moderately firm in texture. Such an assortment of coals is typical of the mining fields that have been explored so far throughout the Andean region and, in a general sense, this may be affirmed also with respect to the corresponding Rocky Mountain coalfields of North America. A group of these Cauca Valley coals may be taken as an example of the more difficult washing jobs that may be encountered in the western Cordillera.

Tables 3 and 4 show the elementary float-and-sink data of seven samples of high-volatile coking coal from the central part of the Cauca Valley coalfield. This group of samples may be taken as an example that might represent the component parts of the composite feed to a central washing plant projected to prepare coking coals from these sources. The data of the latter table plus the screen-test data of the same samples shown in Table 5 will outline, in a general way, the variety of raw materials that must be blended into the plant-feed of this typical Cauca Valley project.

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[1] The author's original paper included descriptions of these units, omitted here, with his consent, owing to shortage of space.

BLOWING PRACTICE

The quantitative statements of this section will be based on blows of about 25 tons of steel for ingots, but the qualitative features will apply equally well to smaller units.

Charging

The relative proportions of molten iron and solid scrap (or other cooling agent) required to finish a blow to proper composition and temperature with a minimum of steaming or surface blowing, will depend primarily upon the silicon content and temperature of the iron, but also on the age and condition of the lining and bottom of the vessel and on the temperature of the ladle. All of these factors must be taken into account by the blower and evaluated in the light of his skill and experience before the charge weights are ordered. Any error in judgment is corrected in ordering charges for subsequent blows of iron of the same analysis.

The estimation of charge proportions is strictly a matter of acquired art rather than calculation, and there are no standard formulae. A rough indication of the magnitude of adjustments for moderate fluctuations about average iron composition and temperature (say 1.3% Si, 1,400°C. for a 28-ton charge with 10% scrap) is given by the following values based on experience at one shop: An increase of 0.1% silicon in the iron is approximately compensated by replacement of molten iron by scrap in the amount of 2% of the charge weight; or an increase in iron temperature of 20°C. is approximately compensated by replacement of iron by scrap to the extent of 1.1% of charge weight; or an increase in silicon of 0.1% is approximately equivalent to a 35-degree increase in iron temperature.

Average scrap consumption in the United States is 10 to 12%[3]. In round numbers, this amounts to 2,700 to 3,300 kg. of scrap in a total charge of 27,000 to 28,000 kg. to produce 24 to 25 tons of ingots. The total amount of weighed scrap is charged either before the molten iron while the vessel is turned down or just after the vessel has been turned up, depending on the mechanical equipment used (see page 234). Cold pig iron may be used instead of scrap as a coolant.

Hard ore or mill scale is sometimes used instead of scrap or iron to cool the blow or to shorten the blowing time. (Such additions have also been used in basic converters to minimize nitrogen absorption (6, 11). For equal weights, the cooling effect of ore is theoretically about four times that of scrap because of the endothermic heat of reducing the ore. Cheap iron units are obtained by direct reduction of ore in the converter; but there is danger of an explosive reaction from moisture in the ore, and slopping may occur if the ore is added too late in the blow to be completely reduced by carbon or silicon.

Blowing

After the molten iron has been added, the blast is turned on to prevent iron from running into the tuyeres as the vessel is turned up to blowing position. Turbo-blowers are generally equipped with constant-pressure controls, and no special adjustments of blast pressure or volume are required during the blow. The rate of air flow gradually drops off 10 to 20% during the early part of the blow because of the formation of shells or "stalagmites" around the upper ends of the tuyere holes (12), but the air flow returns to its initial value as these obstructions melt or are fluxed away as the blow progresses.

The blower must control the temperature of the metal during the blow and the carbon content at the point where the vessel is turned down to halt the blow. He does this primarily by visual observation of the flame and sparks, aided sometimes by a photocell device or by a spectroscope as will be mentioned later. Since the art cannot be learned by reading about it, no attempt will be made to describe the indications seen by the blower, but a good description has been given by Loughrey (12).

The charge is usually proportioned to generate a little more heat than needed, so that a final degree of control of temperature during the blow can be accomplished by injecting steam into the blast line to cool the vessel by endothermic reaction. The steam is not metered but is turned on or off by the blower in accordance with his judgment. If too much scrap has been charged and the blow appears cold, the temperature may be raised by side blowing, which is done by turning down the vessel to expose a few tuyeres above the metal. The combustion of carbon monoxide within the vessel by air from the exposed tuyeres releases a substantial amount of additional heat where it may be absorbed by the charge, instead of in the flame above the vessel. Under extraordinary circumstances, ferrosilicon may be added to generate heat. After the first blow or two on a new cast of iron or a new vessel, an experienced blower can adjust the charge so that little steam or side blowing is necessary.

Close control of finishing temperature is important to uniform recovery of ferromanganese, clean pouring, avoidance of excessive segregation, and good surface of the rolled product (13). The correct temperature of the blown metal leaving the converter will depend on the size and condition of the ladle, the amount of cold additions, and other factors. For producing capped or rimmed ingots, it is desired that the heat will just pour clean or leave a small patch of skull in the ladle. Blown metal for duplexing is sometimes finished hotter than steel for ingots in order to speed open hearth production.

Bessemer heats for ingot production are "full" blown to a final end-point of about 0.03 to 0.05% carbon, because this end-point can be judged quite accurately and thus close control of composition is attained. The manganese content is about 0.03% and the silicon below 0.01% at this end-point. Blown metal heats for duplexing are often turned down "young" at a carbon content of 0.1 to 0.2%. In some practices, "high" blown metal containing up to 2% carbon is used for part of the duplex charge, but close control of carbon content of the blown metal is not possible at such high levels. A typical blowing time is about 12 minutes, but the blow might last from 8 to 18 minutes depending on silicon content of the iron, number of tuyeres blanked, and other factors. Normal production for a 3-vessel shop with two converters active is about 40 blows per 8-hour turn.

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[3] These figures are representative of several companies which do not use ores, mill scale, or pig iron as a coolant, and they include some vessel or pit scrap. The industry-wide scrap consumption according to American Iron and Steel Institute statistics is about 4%.

Finishing practice

A large tonnage of Bessemer steel is used to produce rimmed or capped ingots, in which case the only final additions are ferromanganese to meet the specification and sometimes a small amount of aluminium to adjust the dissolved oxygen content for optimum rimming action. Final control of the action of the steel in the moulds is made by the addition of aluminium during the pouring of the ingots. Ferromanganese may be added to the converter after it has been turned down, but more commonly it is shovelled into the stream as the metal pours into the ladle. If the slag has the proper semi-solid consistency, most of it will remain in the converter and will be dumped into a pan below the vessel after the steel ladle has been removed.

Over a million tons of killed Bessemer ingots have been produced for the manufacture of seamless pipe (1, 4). Such steel may contain from 0.1 to 0.4% carbon, 0.35 to 1.25% manganese, and 0.1 to 0.3% silicon. The blown metal is first deoxidized by carbon by the addition to the vessel of molten iron from the mixer in proportion to the carbon content to be made. The reaction between the carbon of the iron and the oxygen dissolved in the blown metal results in the formation of carbon monoxide, which burns at the mouth of the vessel. An advantage claimed for the process is that deoxidation product escapes as a gas rather than forming non-metallic inclusions. After several minutes the boiling reaction subsides, and the ferromanganese and ferrosilicon additions are made to the steel in the vessel by means of a special open-end pan or chute handled by crane. No difficulty is experienced in getting the additions through the slag, and ferroalloy recoveries are as good as those obtained from ladle additions. The aluminium addition, which amounts to about 1.5 kg. per ton for 0.15% carbon steel and 1 kg. per ton for 0.35% carbon product, is added to the ladle. This practice results in fully-killed fine-grained (austenite grain size at 927°C.) steel.

Ferromanganese efficiencies vary, of course, with the temperature, end-point carbon, slag condition at the end of the blow, and type of steel being made, but the following recoveries are indicative of normal experience:

Type of steel	*Low-carbon, open*	*0.15% C, killed*	*0.35% C, killed*
Manganese recovery, per cent...	55–70	77–83	90–95

Teeming practice

Much Bessemer steel is poured as "capped" ingots because this practice results in good surface and maximum yield of billets. The steel is of the "open" or rimming type, is poured up to the shoulder of "bottle-top" moulds, and is treated with aluminium so it will rise into the narrow neck of the mould. Shortly after pouring, a heavy iron or steel cap is used to close the top of the mould and chill the top of the ingot to form a gas-tight seal. Gas evolution is thus automatically controlled by pressure, and the solidified ingot contains widely dispersed blowholes and virtually no primary pipe. A "chemical" cap of 90% ferrosilicon is sometimes used instead of the metal cap to cause rapid freezing of the ingot top. Rimmed ingots are also produced by the same practice as used for open hearth steel.

In recent years, it has been found that satisfactory killed steel for seamless pipe can be produced in big-end-down moulds without hot tops, and this has become general practice. The top surface of the ingot is frozen rapidly by the application of water sprays. The pipe which forms below the solid shell is clean and free of slag or oxide film and welds in rolling, so that higher yields are obtained than if a hot top were used.

Ingot moulds, pouring practices, and billet yields are very much the same for Bessemer steel as for open hearth steel (14).

Ingot yields

The normal yield of acceptable ingots is 88 to 89% of charge weight. Average yield and loss figures for one large producer are as follows:

	Per cent
Good product	87.9
Converter iron and steel scrap	2.0
Ingot butts and condemned ingots	1.5
Iron in cinder, spittings, etc.	0.3
TOTAL INGOTS AND SCRAP	91.7
Loss (including slag to dump)	8.3
TOTAL	100.0

Relatively little difficulty occurs from missed analysis or bad pours. One plant reports about 3.7% of blows diverted to some product other than originally scheduled because of off-analysis, and about 1.8% of heats diverted because of excessive skulls or other adverse pouring conditions.

Photocell and other instrumental controls

A photocell arrangement (15, 16), is in regular use by one large steel company as an aid to the blower for judging temperature and to control the point at which the vessel is turned down. Fig. 1 is a replica of a chart for one blow; because of the design of the recorder, the start of blow is at the right, point A, and the time scale reads from right to left. Portion A-B is the silicon blow, during which time the flame intensity is very low. Oxidation of carbon begins to be rapid at B, and period B-E is the carbon blow. The maximum height of the curve, C-D, is an indication of the temperature of the blow, but it is a relative measure which depends on the blast pressure, the number of tuyeres blanked, and the condition of the nose of the vessel, as well as on the characteristics of the particular installation. The flame intensity falls off rapidly as the carbon content reaches a low value, and an inflection point on the falling curve, which occurs at about 0.06% carbon and is indicated at E on Figure 1, is taken as the "end-point." The blow is continued for a predetermined number of seconds of "afterblow" before the vessel is turned down (point F on the chart). The optimum time of afterblow depends on the weight and silicon content of the iron (slag volume), number of tuyeres blanked, and grade of steel to be made (17). A schedule of afterblow times is set up from a study of records on previous blows.

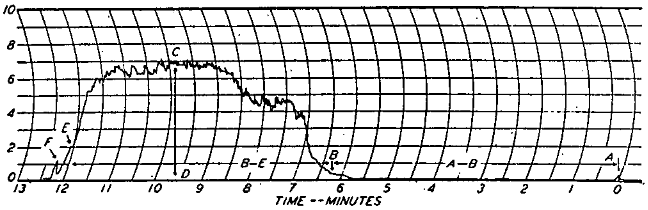

Figure 1
Photocell chart [Work (15)].

The spectroscope was used as early as 1862 in an effort to control the Bessemer blow and has been tried by many people since then, but not with complete success. It is currently in use to some extent in one United States plant to judge the end of the blow by the disappearance of certain iron lines in the spectrum. Efforts have also been made to combine the photocell and spectroscope as control devices, and to measure temperature during the blow by a photocell or radiation receiver located in the windbox and sighting up through a tuyere hole. More extensive discussions of control methods may be found in papers by Wilder (1) and by Work (15).

CHEMISTRY OF THE BLOW

The order of elimination of elements during an acid blow is shown graphically in Figure 2. Silicon and manganese are oxidized simultaneously during the first half of the process, known as the "silicon blow". Carbon removal does not become rapid until after the silicon has been almost eliminated. A reversal may occur, however, and carbon may be oxidized in preference to silicon if the temperature is allowed to become excessively high early in the blow, because at high temperatures silicon becomes a weaker reducing agent than carbon. The fact that some carbon is removed during the silicon blow is confirmed by analyses of converter gases (15, 18), which show oxides of carbon during the first few minutes.

As shown by Figure 2, the nitrogen content of the blown metal rises rapidly during the latter part of the blow. The nitrogen content of bottom-blown Bessemer steel may range from 0.01 to 0.02%, but a normal value is 0.015%. High temperature and prolonged blowing lead to high nitrogen contents. The adverse effects of nitrogen on the properties of steel have been noted earlier, and some methods of minimizing nitrogen absorption will be discussed later.

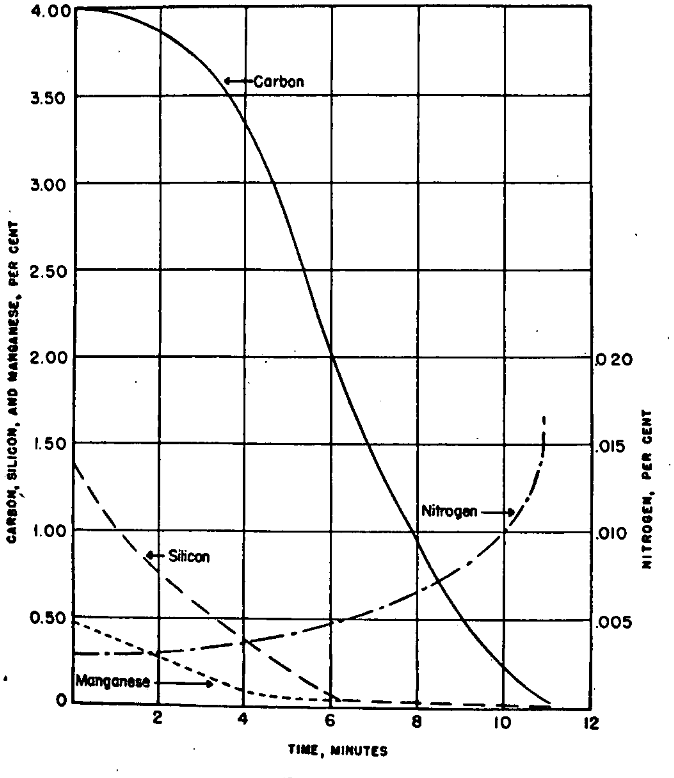

Figure 2
Chemical changes in the metal during a Bessemer blow.

The dissolved oxygen content of blown metal varies inversely with the carbon content and is reported (13) to be a little higher than that of an open hearth bath for the same carbon content. At the low carbon concentrations prevailing at the Bessemer end-point, dissolved oxygen content rises very rapidly if the heat is overblown (17). A normal oxygen content at the low-carbon end-point is about 0.08%. The oxygen content of finished ingots depends on the deoxidation and pouring practice and should not be appreciably different than that of open hearth steel of the same type and analysis.

Typical vessel slags from North American Bessemer practice usually fall within the following composition ranges in weight per cent:

FeO	Fe_2O_3	MnO	SiO_2	Al_2O_3	CaO	MgO
15–20	1–5	8–12	60–65	1–5	1	0.5

Sloppy slags are usually lower in SiO_2 and higher in iron oxides, MnO, or CaO than the ranges given above.

DIRECT COST OF PRODUCING BESSEMER INGOTS

No attempt will be made here to evaluate capital expenditures because the value of existing United States plants would bear little relation to the cost of erecting new facilities in Latin America. Direct costs of producing ingots will be given in physical units in terms of raw materials consumed and human effort employed, because such figures are more easily interpreted for other conditions or economies than dollar costs would be. The consumption of metallic materials will vary somewhat, but not greatly, with changes in practice and types of products. Consumption of refractories will depend appreciably on conditions of operation and on the quality of available materials. Human labour is given in terms of the average working force for an 8-hour turn, rather than as man-hours per ton.

Table 5

MATERIALS AND ENERGY CONSUMPTION FOR BESSEMER INGOTS (UNITED STATES)

Materials	
Charge:	
Molten iron	990–1,040 kg./ton
Steel scrap	138– 110
TOTAL IRON AND STEEL SCRAP	1,128–1,150
Additions:	
Ferromanganese	8.3– 9.7
Ferrosilicon	0 – 0.2
Aluminium, etc.	0.1– 0.2
TOTAL ADDITIONS	8.5–10.0
Total metallic materials	1,138–1,158
Refractories:	
Stone and mica schist	1.8–2.3 kg./ton
Ganister[a]	7.2
Clay[a]	2.3
Tuyeres[a]	0.033 pieces/ton
Tile brick[a]	0.10
Energy	
Steam, including turbo-blowers[a]	140,000 k. cal./ton
Electricity, including all auxiliaries[a]	4.13 kW hr./ton

[a] Based on data from one company only.

Table 6

Labour force for a Bessemer shop, average per eight-hour turn (United States)

Activity or location	*Number of men per turn*
Superintendent or foreman	1
Pulpit (blower, regulator, recorder)	3
Vesselmen	3
Scrap weighing and loading	2 or 3
Hot metal handling	3 or 4
Manganese loading	1 or 2
Vessel cindermen	2 or 3
Steel pouring platform crew	6 or 7
Slag pit	1
Mould yard	2
Bottom house	4 or 5
Ladle and vessel liners and helpers	4 to 6
Cranemen	8 to 10
Locomotive engineers and switchmen	6 to 8
Blowing engineers	1 or 2
Manganese crusher operator[a]	0 or 1
Stopper makers[a]	0 to 2
Skull cracker ground man[a]	0 or 1
Maintenance men	2 to 4
Chemical laboratory and metallurgical inspection	3
General labour[a]	0 to 7
TOTAL MEN PER TURN	60 to 70

[a] May be combined with services to open hearth end of duplex shop.

The reason for this selection is that the working force is likely to be nearly the same for 10-ton as for 25-ton converters and for a two-vessel or four-vessel shop, whereas the man-hours per ton would vary considerably with the size and rate of operation of the facilities. It is obvious that the human effort required will depend greatly on the degree of mechanization of the plant and the efficiency of the labour force. The data given in Tables 5 and 6 are a composite of information supplied on current operations by two large steel companies, one for a straight Bessemer shop and the other for a duplex plant. They may be taken as representative of costs in a well-mechanized plant comprising three 25-ton converters and related facilities, operating under present conditions in the United States.

VARIATIONS ON THE BESSEMER PROCESS

The preceding portions of this paper have emphasized that there is a limited field of application for the conventional Bessemer process based on methods of operation and raw materials similar to those in use in the United States. The limitations arise chiefly from problems of obtaining low phosphorus, sulfur and nitrogen contents in steel produced in bottom-blown acid converters. A brief summary will now be given of ways of circumventing these difficulties.

Desulfurization of Bessemer iron

It is well known that desulfurization of iron can be accomplished far more effectively under reducing than under oxidizing conditions, so any process to remove sulfur from Bessemer steel should operate on the molten iron before it is blown rather than on the finished product. A number of processes are known for desulfurizing iron after it leaves the blast furnace. These usually involve treatment of the iron with a highly alkaline material such as sodium carbonate, sodium hydroxide, calcium carbide or calcium oxide. The treatments are expensive and they have not come into regular use in North America except for the occasional "doctoring" of off-analysis casts of iron or for very special purposes. A notable example is the practice perfected by the A. M. Byers Company (19) for the desulfurization of cupola-melted iron before it is blown in Bessemer converters during the manufacture of wrought iron by the Aston process. Ninety per cent or more of the sulfur may be removed from iron containing as much as 0.2% sulfur by treatment with sodium hydroxide in amounts of 9 to 15 kg. per ton. The process is carried out in a specially designed hooded ladle which facilitates separation of the metal from the reaction slag. Kalling and his collaborators (20, 21) have recently reported excellent results from a process of desulfurizing iron with burnt lime in a rotary furnace.

Dephosphorization of Bessemer steel

Over a million tons of Bessemer steel has been dephosphorized by the Yocom process (22). The treatment consists of adding a dry mixture of 50% impure lime (CaO), 30% roll scale, and 20% fluorspar (CaF_2) to the stream of blown metal as it is poured from the converter into the teeming ladle. The converter slag is held back by means of a wood block placed across the nose of the vessel. Phosphorus is lowered from about 0.1% down to 0.02 to 0.04% in about 30 seconds by the use of about 35 to 40 kg. per ton of the mixture. The process is limited to steels analysing under 0.2% carbon and 0.75% manganese. Scrap consumption is reduced because the blown metal must be finished hotter than for ordinary Bessemer steel. While the steels produced in this way are softer, more ductile, and less susceptible to age-hardening than the normal Bessemer product, the process has not come into widespread use in the United States, and it would be of little interest in those Latin-American regions which have the advantage of remarkably low-phosphorus ores.

Surface-blown acid converters

It has become established through the efforts of many investigators that the nitrogen content of converter steel, both acid and basic, can be reduced by minimizing the contact of the blown metal with nitrogen, especially toward the end of the blow. This can be accomplished in many ways: by decreasing the bath depth, by diluting the blast with oxygen or steam, by the use of iron oxide or limestone, or by surface blowing. Space does not permit a review of the extensive work which has been done, but a recent comprehensive investigation described by Webster and Clark (23) will be cited to indicate what can be accomplished by way of reducing the nitrogen content of acid converter steel.

Development work was begun with a specially designed side-blown, acid-lined converter of 3-ton nominal capacity in which heats of 3 to 10 tons were produced by both surface and subsurface blowing. Results were so successful that a larger converter of 20-ton capacity was built, in which heats as large as 30 tons were blown at rates as fast as 1.8 tons per minute. The nitrogen content of the surface-blown heats ranged from 0.003 to 0.008% and averaged about 0.005%. The steel was applied on a variety of commercial products without difficulty. The cold working and ageing properties of the steel were typical of low nitrogen, high phosphorus (0.1%) steel. It was superior to regular Bessemer steel in cold forming applications.

Several problems remain to be worked out before the process can be said to be entirely practical. The most important of these is the handling of the slag, which tends to be high in iron oxides and is therefore too fluid to be held back in the vessel. The presence of the fluid, oxidizing slag in the ladle leads to low manganese recoveries and would interfere with deoxidation practices. The carbon contents of the surface-blown heats at the end of the blow were about 0.01 to 0.05% higher than for regular Bessemer practice. Using the normal Bessemer blowing iron, blown metal temperatures were higher than for normal operation, with the result that considerably more scrap or ore could be converted into steel. Metallic yields were somewhat erratic and lining wear was faster than for bottom blowing, but these adverse features might be improved with better control of temperature and slag properties. If these technical difficulties can be overcome, the surface-blown acid converter should have very attractive possibilities, particularly for a plant which might have the advantage of iron low in phosphorus and sulfur.

Surface-blown basic converters

Following the work just described, a co-operative investigation was made on a pilot plant scale with a surface-blown, basic-lined converter. The results have been described by Sims and Toy (24). The objective, which was successfully realized, was to produce steel low in phosphorus and nitrogen from iron of the composition ordinarily used in open hearth furnaces in the United States; namely, about 1% silicon, 2% manganese, 0.3% phosphorus. Iron of such a low phosphorus content is not practical for use in the Thomas process because of thermal requirements, but with the added heat of combustion of CO to CO_2 obtained with surface blowing, adequate temperatures could be attained even in a vessel of only 450 kg. capacity. It was found that carbon could be oxidized to below 0.03%, manganese below 0.1%, and silicon below 0.01%. Nitrogen could be maintained below 0.003%. Up to 97% of the phosphorus and up to 50% of the sulfur were eliminated during the regular blow, and an afterblow was neither necessary nor desirable. Burnt lime requirements were from 60 to 90 kg. per ton. A log of an experimental blow is shown in graphic form in Figure 3, which includes temperature, gas analysis, and a photocell record of flame intensity. Results of the small-scale tests were confirmed by a series of heats of about 25 tons which were produced on a basic lining in the same shell that had been used in the study of the side-blown acid converter described above.

The new process was found to be rather versatile with respect to choice of iron composition and blowing practice and was not difficult to control. Steel plates rolled from the products of both the small and the large converters were tested for the usual mechanical properties and for ductility in bending, impact strength, transition temperature and sensitivity, to work hardening and strain ageing. The experimental steels were similar to open hearth steel of the same composition and distinctly different from ordinary Bessemer steel. This new process, which has been christened the "Turbo-hearth", is now in process of further development on a commercial scale.[4]

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[4] The development of the acid and basic surface-blown converters just described is a co-operative project of the Jones and Laughlin Steel Corporation and the United States Steel Company and is protected by patents or patent applications.

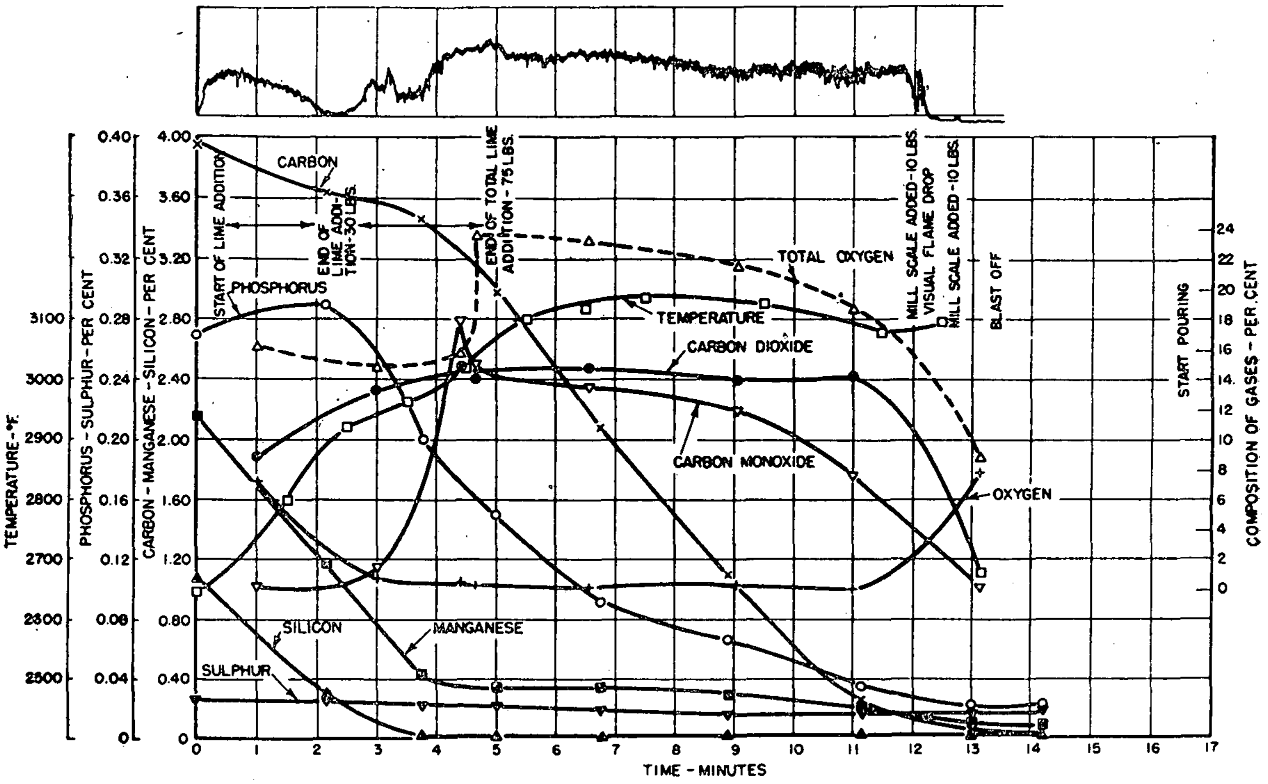

Figure 3

Record of Heat in Surface-blown Converter: Heat 23.

Standard blowing conditions. Mill scale added at flame drop. (Sims & Toy (24)).

Duplex Process

The duplex process as now practised in the United States no longer deserves the bad reputation acquired by its progenitor near the turn of the century, because the requirements for control of the metallurgical properties of steel are now much better understood. With the exception of the Birmingham, Alabama, district where the high phosphorus iron produced from the local ores requires a double slag process, modern duplex practice is a single-heat, single-slag operation. It is rather similar to conventional basic open hearth practice except that the charge is entirely liquid, a combination of Bessemer blown metal and blast furnace iron. If the charge is properly proportioned and the heat is worked to produce a vigorous carbon boil for a sufficient length of time, steel can be produced consistently to meet a 0.005 or 0.006% maximum nitrogen specification. Average time for a 180- to 200-ton heat is 4 to 4.5 hours. The duplex process is perhaps best suited metallurgically and economically to the production of mild steels which are consumed in large tonnages, but it has been used successfully to make practically all grades of open hearth steel including killed alloy steels for critical applications. Further details of the process may be found in bibliographic references 14 and 25 and in other sources cited therein.

Conclusion

This paper has aimed at giving a comprehensive and factual account of the applications, specifications and limitations of Bessemer steel, the raw materials used in its manufacture, and the technology and economics of the process, all with reference to current practice in the United States.

Briefly, it may be said that the Bessemer process, earliest of the large-tonnage methods of steelmaking to be invented, is still by far the fastest and under many circumstances the cheapest way of producing steel. The very speed of the process and relative inaccessibility of the converter have worked against the development of fully successful methods of chemical or instrumental control of it, but this constitutes only a difficulty rather than a total barrier to the general applicability of Bessemer steel. Neither do the circumstances that the method is pneumatic and carried out in acid refractories in themselves give rise to inherent limitations, for competent metallurgists are now agreed that the properties of steel are not determined by any mysterious factors in the method of production but by the composition and constitution of the product (14). Leaving aside economic factors which favour the basic open hearth process in the United States, the chief metallurgical limitations which have restricted the applications of Bessemer steel stem from the relatively high phosphorus and nitrogen contents which result from North American Bessemer ores and bottom blowing with air.

The possible usefulness of the Bessemer process to a Latin-American country would first depend on a potential source of molten iron of suitable analysis. If iron containing, say, under 0.035% phosphorus and sulfur were available, then Bessemer steel might have broader applicability than the higher phosphorus product of North America. The newer developments in surface-blown converters look very promising, but unfortunately these methods are not yet fully developed. For the immediate future, a duplex process combining the acid converter with the basic open hearth seems to offer the greatest latitude in selection of raw materials and application of products while ensuring independence of a source of cheap and abundant scrap.

Acknowledgment

The author is most grateful to friends in the steel industry who have contributed generously to the information contained in this paper, but who prefer to remain anonymous.

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Basic Open-Hearth Steelmaking Practice in the United States of America

WILLIAM C. BUELL, JR.

In August 1952, the steel ingot capacity of the United States was officially stated to be something over 108 million tons per year, and the average production rate was at about 100% of capacity. Before the beginning of 1953, plants having an additional capacity of 10 to 12 million tons, which are now under construction, are expected to be in operable condition. This total tonnage will be available to supply the needs of over 160 million people who will then have three quarters annual ton of ingot capacity per capita.

This capacity of 108 million tons[1] is produced from about 900 basic open-hearth furnace units representing 94 million tons of capacity, 35 acid open-hearth furnaces having about 750,000 tons of capacity, 28 acid-Bessemer vessels with a capacity of over five million tons, and 260 electric melting furnace units with a capacity above eight million tons.

There are in use in the world six principal methods for the production of steel. These are:

1. *The acid-Bessemer process*, which is usually limited in its application to such steel products as will be acceptable with a phosphorus content as high as 0.10 or 0.11%. Only a small percentage of scrap ferrous metal can be absorbed in its manufacture.

2. *The basic Bessemer or Thomas process*, which characteristically employs a molten iron charge containing 1.75 to 2% or even greater phosphorus content. As with the acid-Bessemer process only a small proportion of scrap may be included in the metallic charge.

At this point it might be of interest to note, in passing, that the basic process of making steel from iron of high-phosphorus content was discovered by Sydney Gilchrist Thomas, a clerk of a London police court and a dabbler in chemistry, who, with the help of his cousin Thomas Gilchrist, was endeavouring to secure a source of soluble phosphoric acid for use as a fertilizer. Therefore, steel produced from the basic converter process which bears his name might well be called a by-product of the manufacture of phosphoric acid containing slag, which even today is in no small part responsible for the agricultural productivity of Europe.

3. *The acid open-hearth process*, which is utilized only to a small extent inasmuch as the charged metallics of the process may not contain sulfur and phosphorus in greater percentages than those allowed in the finished steel. These impurities are usually limited to 0.04% or possibly, in a few instances, 0.05% for either of these two elements, neither of which is removed in any acid process. The acid open-hearth process is used to a considerable extent in the British Isles and to a much lesser extent in the United States.

4. *The basic open-hearth process*, through the use of which the largest part of the world's tonnage steel is produced. The basic open-hearth process is capable of converting almost any ordinary type of ferrous metallics to satisfactory steel, provided that the metallics do not include undue quantities of the non-oxidizable elements such as copper, lead, tin, nickel, molybdenum, arsenic, etc. Economically, the prime value to industry of the basic open-hearth process arises from the fact that most of all ferrous scrap originating from all sources may be and is reconverted to specification steel.

5. *The acid electric melting process.* Today the use of this process is mostly confined to small units producing steel for foundry or special specifications such as those of the tool steels. Selected raw materials are required for the process, which is subject to the same metallurgical limitations as the acid open-hearth process.

6. *The basic electric melting process.* This process is capable of paralleling that of the basic open-hearth process in all respects. The stainless and most of the higher alloy grades are produced by this process. However, on account of some of the operating precepts, referring particularly to furnace time and electric current requirements, the metallic charge is usually selected to contain the least possible quantity of the elements that must be oxidized in refining in order to meet stated specifications.

In addition to the six principal processes, there are various operations in use that combine two of the processes in order to provide a better solution for certain local conditions. These include the duplex process of the United States and India, now of diminishing application, which is carried out with acid-Bessemer metal blown to low carbon, manganese and silicon contents. The blown metal is dephosphor-

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[1] Statistics used in this paper are based on net tons of 2,000 lb. each.

ized, desulfurized and finished in tilting basic open-hearth furnaces operated in a modified continuous process, with a dominant pool of refined metal remaining in the furnace during the operating cycle. Steels produced by this method are currently somewhat limited in their application.

A modification of this process is the synthetic scrap process presently finding favour in the United States. This differs from the true duplex process in that the iron is not bessemerized as completely as in the duplex process, but is charged into tilting or stationary basic open-hearth furnaces in analysis similar to that of ordinary steel scrap. The furnace is emptied for each heat. By this method, steels are made from charges that include some solid steel scrap as well as some molten blast-furnace metal. These steels are considered the equal of those produced by the basic open-hearth process.

A duplex process commonly employed in Europe with high phosphorus pig iron combines dephosphorizing the metal in the basic Bessemer and final oxidizing and finishing the steel in stationary basic open-hearth or electric melting furnaces.

An alternate to the above is the English active mixer method. High-phosphorus metal is treated in an active mixer to reduce silicon, principally, to such a low point that when the molten metal is transferred to basic open-hearth furnaces, usually of the tilting type, the basic reagents of the slag will work more actively in removing the contained phosphorus.

In addition to the above, there is a choice of various types of duplexing which are carried on with metal initially processed in a Bessemer or an open-hearth furnace and finished in an electric melting furnace.

The chief advantage claimed for any duplex process is the increased speed of conversion of ferrous metallics to the desired steel specification. A considerable disadvantage is the increased loss of metallic yield which is inherent to any multiple process of conversion.

Under good practice conditions, the Bessemer processes will convert charges of up to perhaps 40 tons, consisting principally of molten iron, to steel in 45 minutes or less, the duplex processes up to 150 tons in seven hours or less, the open-hearth process up to about 300 tons in say, 12 to 15 hours, and the electric furnace up to 100 tons in perhaps five or six hours.

A natural question to follow would be, "With the Bessemer processed metal converted at 60 tons per hour as against the duplex rate of 25 tons per hour, and the open-hearth and electric furnace production rate of 20 tons per hour, why is not steel produced more generally by the fastest of the methods?" While a considerable part of the answer will be found in the economics of the raw material situation, probably the most important factor tending to the use of the less productive processes will be found in the higher reaches of physical metallurgy and will be principally concerned with the nitrogen content of the steel ingots. Steel produced by the Bessemer processes, in which nitrogen is in intimate contact with the metal, may contain this element to the extent of 0.015% or more, while steel converted by the open-hearth processes, in which nitrogen reaches the metal only in the pig iron portion, may be expected to contain 0.005% or less. Steel made by duplex processes will contain nitrogen in intermediate degrees since the nitrogen content of steel probably decreases as a function of longer duration of the process and of the chemical activity.

Nitrogen produces strain ageing or embrittlement more or less in proportion to the nitrogen content. While strain ageing may have little or no effect on the metal during hot working, it may have a marked effect on the metal during cold working or on its behaviour when, in finished form, it is structurally stressed. It is principally on this account that the steels used for construction, when produced by the open-hearth or electric melting processes, are considered superior to those products produced by other methods.

The steel industry of the United States has become naturally centralized in a few sections geographically located in economic proximity to the sources and reserves of good iron ore and of coal of suitable quality for the manufacture of steelmaking iron and to the markets for the finished steel products. On this basis and in a generalized way, these natural sections may be classified as the Great Lakes section, which is served by the ores of the Lake Superior region, the eastern section supplied by ores mostly arriving by sea, the western section utilizing the ores of Wyoming, Utah and California, and the southern section, which is supplied mostly with the so-called red and brown ores of local source. The steel ingot producing capacities of these several sections, in millions of tons per year, are as follows:

STEEL INGOT PRODUCING CAPACITY IN MILLIONS OF TONS PER YEAR

Section	*Capacity in millions of tons*	*Per cent of total capacity*
Great Lakes	82.7	76.0
Eastern	14.2	13.0
Western	7.6	7.0
Southern	4.1	4.0
TOTAL	108.6	100.0

From the foregoing, it will be noted that the two larger sections produce almost 90% of the total ingot tonnage. In view of this, detailed comments will be confined to the practices of the Great Lakes and eastern sections. However, in passing, it might be noted that the practice of the western section is similar in all respects to that of those two sections, while that of the southern section differs from the others of the United States in that their steelmaking iron has an intermediate phosphorus content approaching 1%, for which specialized steel conversion processes have been developed that apply only locally. It is of interest to record that the open-hearth slags from the steel production of the southern section are the only ones of value to agriculture for their phosphoric acid content.

The Great Lakes section, lying south of the Great Lakes system, contains the Pittsburgh, Chicago, Youngstown, Buffalo, Cleveland, Detroit, Wheeling and Ohio River districts, and here the iron to be refined is smelted from the hematite ores of the great Mesabi and the other ranges of Minnesota, Michigan and Wisconsin.

The eastern section, lying generally east of the Susquehanna River and including the Philadelphia and New England districts, is presently served to a considerable extent by imported ores, principally those originating in Brazil, Chile and Venezuela. A few years hence, it is expected that hematite ores from the Labrador deposits, now undergoing development, will be available to this section.

Of United States total steel production, nearly 83 million tons or about 76%, is processed from iron smelted from the Great Lakes ores, and 15 million tons or about 13% is made in the eastern section and, of this, a substantial proportion is made from iron smelted from South American ores.

In 1951, nearly 87 million tons of ore were shipped to the furnaces from the northern ranges. These ores vary considerably in their chemical composition, with the result that usually ores from several mines are combined either before shipping or at the blast furnaces to keep the ores as charged to the blast furnace at a constant, desired average analysis. Thus, some 70 odd million tons of non-Bessemer ore average, on a natural basis, 50.3% iron, 0.075% phosphorus, 9.5% silica and 0.70% manganese, while 10 million tons of Bessemer grade average 53.1% iron, 0.037% phosphorus, 10.2% silica and 0.40% manganese. Currently, most of the imported ores received on the East Coast average above 60% iron with a contingently low percentage of gangue materials.

Whatever the source of the iron ores used to produce basic steelmaking iron in the Great Lakes or eastern sections of the United States, the product of the blast furnace is expected to conform to the rather narrow ranges of chemical specifications as follows:

Composition of blast-furnace iron for basic open-hearth steelmaking

	Per cent
Carbon	3.50 to 4.25
Manganese	0.75 to 2.00
Silicon	0.80 to 1.40
Sulfur	0.05 maximum
Phosphorus	0.20 to 0.40

THE MATERIALS OF THE OPEN-HEARTH CHARGE

The character of make-up of the materials which are used in the manufacture of basic open-hearth steel is subject to considerable variation and is governed principally by the over-all demand for ingot production, the immediate supply of scrap metal and its cost, as well as the method of processing to be employed.

The production of 108 million tons of ingots necessitates the refining of some 125 million tons of ferrous metallics. The current blast-furnace capacity is about 75 million tons of pig iron, of which about 88% or 66 million tons will be used in steel ingot production, leaving nearly 60 million tons of process metallics to be produced as "home" scrap or secured as purchased scrap, not including a small percentage of ferro-alloys, deoxidizers and alloying materials.

"Home" scrap or that resulting from the processing and rolling within the plant itself usually amounts to 28 to 30% of the ingot weight and, in this instance, totals perhaps 31 to 34 million tons, while the ferro alloys and so forth aggregate about 1¼ million additional tons, thus leaving some 25 odd million tons of steel and iron scrap which must be procured through dealer sources. Inasmuch as 20% of the melting furnace charges is represented in the purchased scrap component, it is probable that the average life of steel from ingot to remelting is of rather short duration. Although bridges and building skeletons may have a 40-year life, and railway rails and automobiles a 10-year life, the majority of the dealer scrap originates from the fabricators and manufacturers of finished steel products and is returned in a sufficiently short time to maintain the productive cycle.

Obviously, if there is not sufficient scrap metal to support the desired ingot production, the only method by which a high production rate can be continually sustained is through the installation of sufficient additional pig iron capacity to replace the shortage of scrap. Only a few years ago, the ratio of pig iron to steelmaking capacity was nearly one to one, while today it is about two to three.

In general terms there are four differing methods of producing basic steel in open-hearth furnaces which employ the charged materials in varying form and quantity, but any of these when properly operated will produce the usual carbon and certain alloy steels of high quality. These are:

1. *The cold metal or "semi-integrated" process.* In this process, steel is made from solid materials and without the benefit of a portion of molten blast-furnace iron. Approximately 8 or 9% of United States basic open-hearth and substantially all of our acid open-hearth and electric furnace steel production is made by this method, which, competitively, may be subject to a quite severe economic penalty when compared with the other methods.

2. *The scrap-hot metal process.* This and the two subsequently described methods are carried on only in "integrated" plants, which are producers of blast-furnace iron. In particular and under prevailing conditions referred to productive demand, the employment of the scrap-hot metal process usually indicates a shortage of blast-furnace metal. As defined, less than 50% iron is charged under this method, but little charged ore can be included and all of the process slag remains on the bath of molten metal until the furnace is tapped. Probably this method accounts for about 30% of the basic steel production.

3. *The hot metal-ore-scrap process.* This process, which employs above 50% and generally 60 to 65% hot metal, assimilates and reduces relatively large quantities of charged iron ore and requires little steel scrap beyond that normally produced as home scrap. It is usually operated as "flush" practice, i.e., a large portion of the process slag is removed by overflow during the course of the process. All in all, this method is probably the most economical one, costwise, of the four methods. The use of this process probably accounts for above 50% of the total basic open-hearth ingot production of the United States.

4. *The synthetic scrap process.* This process is described rather than the true duplex process. While the synthetic scrap process is constantly gaining favour with certain producers, steel conversion by the duplex process is decreasing and the tonnage of true duplex steel currently produced is no longer a material percentage of the total steel production, there being only one large producer, who is located in the southern section. Using the synthetic scrap method, specification carbon steels are produced more rapidly than by any other open-hearth process. Bessemer blown metal is introduced into an open-hearth furnace in molten condition. In this metal, the carbon content has been reduced to well below 1%, the silicon content has been almost entirely oxidized and eliminated, and the man-

ganese content is low; the metal therefore resembles ordinary steel scrap in chemical analysis. This material constitutes up to 70% or more of the charge, with the balance consisting mostly of solid steel scrap. Inasmuch as a large part of the thermo-chemistry of refining has previously been accomplished in the Bessemer, the work in the open-hearth furnace consists largely of adjusting the bath to the analysis of the steel specified and thus the process is completed in a minimum of time. The chief disadvantage in this process lies in the rather low yield resulting from the double conversion employed. Possibly 5% of the current steel production is manufactured by the synthetic scrap process.

In accordance with the foregoing, it is evident that the details of the several charges vary so greatly that any worthwhile tabulation of these details must be shown separately for the four categories. This is attempted in the following Table 1. The quantities used in the table represent typical values which, in each case, have been developed from actual operating data and are believed to be representative of normal operating practices.

Table 1
DETAILS OF TYPICAL BASIC OPEN-HEARTH
CHARGES FOR PRODUCING ONE TON OF CARBON-STEEL INGOTS

Type of process	*Cold metal* *lb.*	*%*	*Scrap-hot metal* *lb.*	*%*	*Hot metal-ore-scrap* *lb.*	*%*	*Synthetic scrap* *lb.*	*%*
Charge								
Iron or blown metal								
Hot iron	—	—	960	42.0	1,242	54.0	82	3.8
Blown metal	—	—	—	—	—	—	1,449	67.0
Pig or scrap [a]	678	30.0	114	5.0	92	4.0	60	2.8
Iron in oxides	23	1.0	46	2.0	150	6.5	22	1.0
TOTAL	702	31.0	1,120	49.0	1,484	64.5	1,613 [b]	74.6
Steel scrap								
Home	653	28.9	654	28.6	662	28.8	519	24.0
Purchased	877	38.8	477	20.9	122	5.3	—	—
TOTAL	1,530	67.7	1,131	49.5	784	34.1	519	24.0
Ferro-alloys	29	1.3	35	1.5	32	1.4	30	1.4
TOTAL METALLICS	2,260	100.0	2,286	100.0	2,300	100.0	2,162	100.0
Distribution								
Steel ingots	2,000	88.5	2,000	87.5	2,000	87.0	2,000	92.5
Recovered scrap	79	3.5	80	3.5	81	3.5	76	3.5
Metallurgical loss	181	8.0	206	9.0	219	9.5	86 [b]	4.0
TOTAL METALLICS	2,260	100.0	2,286	100.0	2,300	100.0	2,162	100.0
Fluxes								
Limestone	50		170		130		40	
Lime	65		12		7		60	
Fettling material	70		65		75		30	
Fluorspar	7		4		3		5	
TOTAL FLUXES	192		251		215		135	

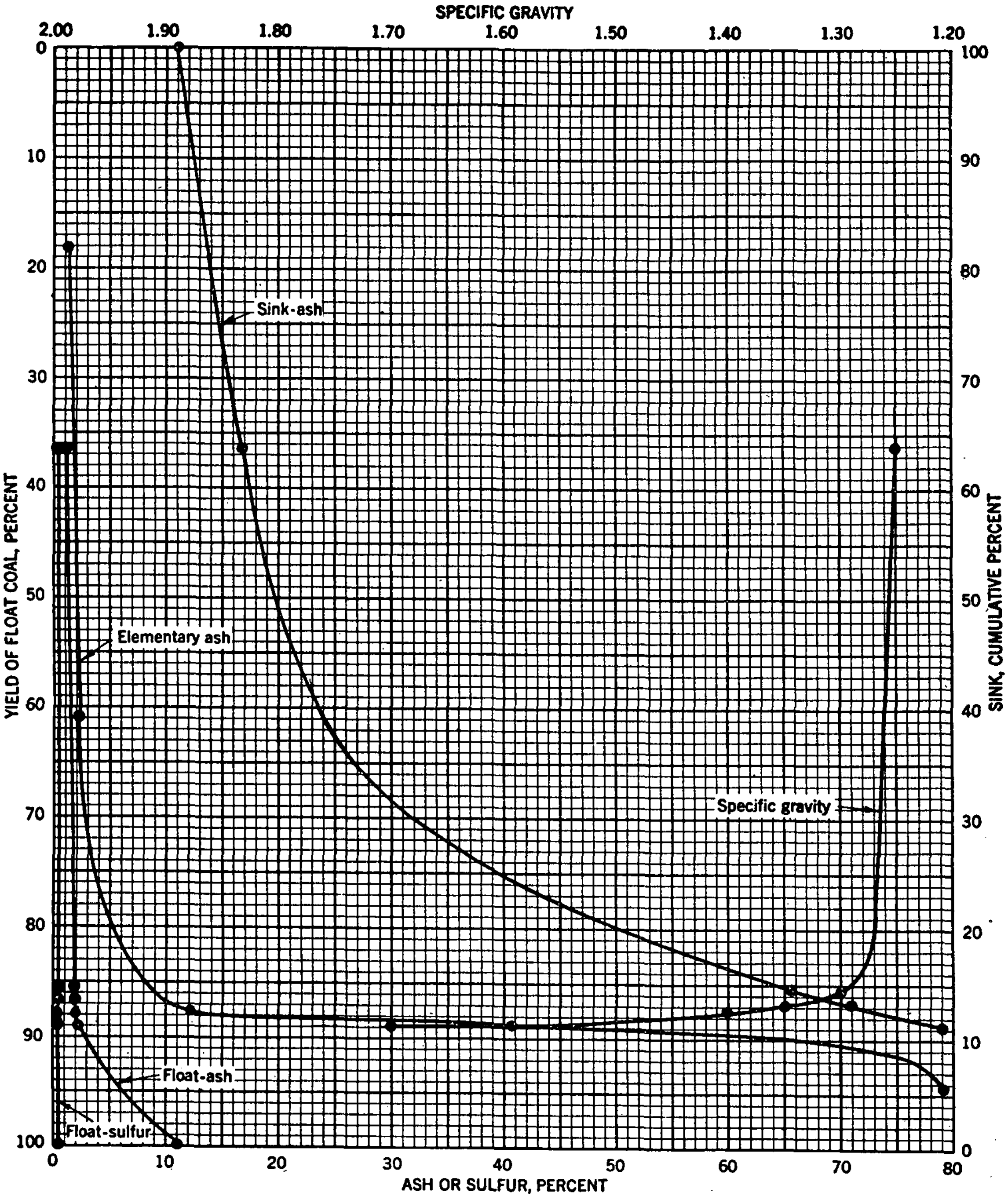
[a] Includes about 60 lb. of condemned moulds and stools.

[b] Does not include a scrap and metallurgical loss of about 160 lb. incurred in the Bessemer conversion of the blown metal.

It may be noted in the four processes described that, in order to produce one ton of ingots, a varying quantity of the charged materials is required. The percentage of the ingot weight to the weight of the charged metallics gives the ingot yield, a relative measure of the metallurgical efficiency of the processes, the efficiencies in these instances being 88.5, 87.5, 87.0 and 92.5%, respectively. However, in the case of the synthetic scrap process, the losses in the conversion of molten iron to liquid scrap are not included, and, if this process loss in bessemerizing, amounting to about 160 lb. per ton of ingots, is added, the ingot yield of the entire conversion process would be 86.2%. The yield factor is important in the economics of any steelmaking process.

THE PROCESSING OF STEEL IN THE OPEN-HEARTH FURNACE

Irrespective of the process by which steel is produced in a basic open-hearth furnace, the charging, melting, working and finishing of a heat proceeds in a substantially similar manner, differing only in detail. Since this is so, a description of the processing of a heat of open-hearth steel can best be offered in generalized



Specific gravity fractions	Elementary data				Computed cumulative data					
	Weight grams	Percent			Float, cumulative percent			Sink, cumulative percent		
		Weight	Ash	Sulfur	Weight	Ash	Sulfur	Weight	Ash	Sulfur
Float - 1.25		36.3	1.17	.62	36.3	1.17	.62	100.0	11.06	.56
1.25 to 1.30		49.1	2.12	.62	85.4	1.72	.62	63.7	16.70	.53
1.30 to 1.35		1.4	12.15	.53	86.8	1.88	.62	14.6	65.71	.22
1.35 to 1.40		.7	18.48	.51	87.5	2.02	.62	13.2	71.40	.19
1.40 to 1.45		.4	23.59	.49	87.9	2.12	.62	12.5	74.36	.17
1.45 to 1.50		.3	28.33	.46	88.2	2.20	.62	12.1	76.04	.16
1.50 to 1.55		.2	32.51	.47	88.4	2.27	.62	11.8	77.25	.15
1.55 to 1.58		.1	36.47	.39	88.5	2.31	.62	11.6	78.02	.14
1.58 to 1.70		.2	40.70	.76	88.7	2.40	.62	11.5	78.39	.14
Sink - 1.70		11.3	79.05	.13	100.0	11.06	.56	11.3	79.05	.13

Fig. 3. Washing characteristics of a sample of Chilean coal. Size 37 – 10 mm.

terms. Besides the oxidation of the major part of the carbon, silicon, manganese and phosphorus which may be included with the charge, the sulfur content is reduced to or below the permissible limit when the basic process is employed, and, in this feature, the basic differs from the acid process in which only the carbon, silicon and manganese may be controlled through oxidation or otherwise.

Fundamentally, the basic process is carried out under a slag in which lime, a chemically basic substance, is the predominant compound, whereas, in the acid process, the slag consists mainly of silica which is acid in its chemical reactions. Thus, in the basic process, the hearth of the furnace holding the metal and slag is, of necessity, constructed and maintained with basic materials, principally magnetite for the original construction and a combination of dolomite and magnesite for maintenance purposes.

Following each heat as tapped, the "bottom" is fettled or repaired to replace such hearth material as has been eroded during the prior heat as a result of chemical and thermal action. When the fettling is completed, the furnace is ready for charging, which generally proceeds as follows:

Through each of the furnace doors, usually five in number, a small amount of light steel scrap is introduced and scattered over the bottom for the purpose of preventing the limestone or lime from sticking to the hearth as it fuses. At the start of charging, the temperature of the empty furnace is probably about 2,600°F. and this will be increased as rapidly as may be possible until the temperature of the molten bath at the beginning of the working period reaches 2,800° to 2,900°F., which will be further increased to about 3,000°F. at tapping time.

After the layer of light scrap has been spread over the bottom of the furnace, the flux, limestone and lime will be introduced evenly over the hearth to be followed by a layer of iron ore or other ferrous oxide if such is to be charged.

Next, the balance of the metal scrap is charged, to be followed by any scrap or solid iron to be included, and this will mark the end of the charging of all solid metallics, with the exception of the small quantity of ferro-alloys which will be introduced towards the end of the processing.

In accordance with the data of Table 1, there will have been charged, exclusive of the ferro-alloys and a small quantity of oxide, 100% of the metallics in the cold metal practice, about 55% in the scrap-hot metal practice, 40% in the hot metal-ore-scrap practice and 27 or 28% in the synthetic scrap process. The time necessary to accomplish the charging to this point ranges from six hours or even more in the case of the cold metal practice to less than one hour with the synthetic scrap process.

With the completion of the charging of the solids, the furnace is fired as hard as possible to expedite the heating and melting of the materials and to initiate the early preliminary development of the slag, which is the medium through which many of the chemical reactions take place in the refining operation.

When the charge is to contain hot liquid iron, this is introduced in one or more increments as soon as conditions in the furnace are proper, that is, when the slag is forming and the scrap charge is melting rapidly. In those cases in which oxides have been charged, and shortly after the liquid metal has been added, the ore boil starts and continues to completion through the release of the oxygen from the oxides which bubbles upward through the bath. This oxygen reacts selectively with the silicon and manganese in the charge and later with some of the carbon and phosphorus, as well as with any miscellaneous oxidizable elements that may be present. The lime boil resulting from the calcination of the limestone follows the ore boil when ore has been charged, or, in the case of cold metal practice, occurs after melting, and the gaseous carbon dioxide released is the medium through which a considerable portion of the oxidation and refining of the bath is caused to take place.

At the completion of the boils and with the composition of the bath known, the working and finishing of the heat proceeds to a point about 30 minutes prior to normal tapping time. At this time, if the metal has been superheated to the required temperature of 2,900° to 2,950°F. and the slag is in satisfactory condition, the furnace additions necessary for deoxidizing the metal are now made. Also, the alloys specified for the attainment of the final chemical analysis of the ingots are added in such quantities as may be charged into the furnace without excessive loss of the alloys to the slag.

The furnace is then tapped and, as the metal flows into the ladle, the remaining, usually small, amounts of the alloys and the necessary carbon are added in the ladle for the final adjustment of the chemical specifications of the steel.

The molten steel in the ladle is then moved to the pouring platform where it is teemed into moulds in which it is solidified into ingots of suitable shape for the rolling operations that will ultimately convert the steel into the usable forms that are so common and familiar to all.

The total time of processing a heat of basic open-hearth steel will vary greatly with furnace capacity and the type of process employed. In ordinary United States usage, the larger cold-metal shops normally employ furnaces of 100 to 150 tons capacity, in which a heat of steel can be made in from 12 to 15 hours. With the other types of practice, the furnaces are generally from 100 tons to 300 tons[1] capacity, with the more modern shops using the furnaces of the larger capacity. Heats of average size are produced in from 10 to 13 hours with scrap-hot metal practice, 9 to 11 hours with the hot metal-ore-scrap practice, and in 4 to 7 hours with the synthetic scrap process. In furnaces of all capacities, when employing any one of the four types of process, all grades of carbon steel can be made and many of the lower alloy grades can also be satisfactorily produced.

AN OUTLINE OF THE CHEMISTRY OF THE BASIC STEELMAKING PROCESS AND ITS PRODUCT

The chemical reactions that take place during the refining of a heat of steel are so many and intricate that it would require a lengthy treatise to describe them. It will be sufficient for the purposes of this paper

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[1] Basic open-hearth furnaces approaching 600 tons capacity are used in a few large shops.

to state that refining in the basic open-hearth furnace is almost entirely carried out through the action of oxidation. Oxygen for the process is supplied through the reduction of iron ore and of the carbon dioxide resulting from the calcination of the limestone as well as from the furnace gases, and in some instances from injection of pure oxygen into the molten metal.

Included with the iron of the charge will be considerable amounts of carbon, silicon, manganese and phosphorus which are to be oxidized, most of the carbon passing off in gaseous form and the oxides of the other elements entering the liquid slag. There will also be some sulfur present, part of which will enter the slag in some form of compound. While most of the metalloids originate in the iron, they are present in varying but lesser amounts in the steel scrap, and small quantities may come from the fluxes.

The silicon oxidizes rapidly and the resulting silica together with liquid iron oxide form the principal constituents of the original slag, which is acid in its reaction and must be rapidly converted to a basic state through the addition of lime, or otherwise the erosion of the basic refractories forming the hearth may be excessive.

Following the oxidation of the silicon, the oxidation of the carbon will proceed at a slower rate and this reaction will be roughly paralleled by the oxidation of the manganese and phosphorus.

While it is important that all of the various metalloids are oxidized, the prime factors of the chemistry in the furnace relate to the control of carbon to develop the desired content of that element and to the formation of a basic slag of proper characteristics before the beginning of the finishing period. If the operator has handled the metal and slag properly, all the metalloids will be at or below the desired maximum percentages by the time the carbon has been adjusted.

The success of the operation of the basic steelmaking process is fundamentally dependent upon the development of a basic (lime) slag of the proper composition. Commonly, the basicity of the slag is measured by the ratio of the lime to silica, CaO/SiO_2. It is the basicity of the slag that permits the reduction of the sulfur and phosphorus contents of the charge. This ratio should fall within the limits of 2.6 to 3.5, the proper figure being determined by the specification and type of finished steel. The iron oxide content of the slag is, to a considerable degree, a function of this ratio, usually being greater as the ratio increases, and is important inasmuch as it has marked control of the oxygen content of the molten metal.

With this very brief description of the process chemistry, the working of a heat is outlined as follows:

About an hour before the scheduled tapping of an open-hearth heat, which marks the start of the finishing period, the carbon, manganese, phosphorus and sulfur contents of the steel, whether processed under one or another of the four classifications of charge previously described, shall have been determined, a slag of satisfactory chemical and physical properties formed and the temperature of the bath raised to the desired point. In addition, should there be alloying elements, such as nickel, molybdenum, chromium or copper, present in sufficient quantities so as to modify the subsequent finishing to the desired specifications, these will also be reported to the operator.

At this point the molten metal might well be termed "raw" steel, because, before its final conversion into a solid steel ingot ready for the rolling operation, its chemical composition will be adjusted to fulfil the requirements of the final analysis specified and its physical structure will be controlled by either the degree of deoxidation practised on the molten metal or the character of the ingot casting practice, or both.

Before describing the finishing operation, it will be well to consider how the initially charged materials were worked down to the desired raw steel analysis.

Shortly after the introduction of the hot metal, or in due time with the cold-metal practice, the entire metallic charge is in a fairly molten condition and a slag consisting predominantly of silica and iron oxide is forming with some lime included. If ore has been charged, its oxygen as released is combining with, first, the silicon and, second, with the manganese and carbon in the bath. As the ore boil proceeds toward completion, the calcination of the limestone and its subsequent fusion progresses as the "lime boil." During both the ore and lime boils, the bath of metal and the slag are in a considerable state of agitation through the action of the freed gases, principally carbon monoxide and carbon dioxide. At about the conclusion of the lime boil, the heat is considered as melted even though small portions of unmelted metal and lime may still be present in the bath.

At this time, the first metal sample will be taken and sent to the laboratory for the determination of carbon, manganese, phosphorus and sulfur, which analyses are necessary to the proper and subsequent working of the heat. Also, nickel, chromium, molybdenum and copper should be determined if these elements are suspected to be present in appreciable amounts.

The fast working of a heat is accomplished by the rapid reduction of the carbon content of the bath. Ideally a heat should "melt-in" with about 0.40 to 0.60% more carbon than is necessary in the raw steel before finishing. The raw steel is usually worked down to about 0.10 or 0.15% below the finishing specifications and later recarburized; thus, if the finished steel is specified as 0.25% carbon, the raw steel is worked down to say 0.15% carbon. For such a heat, the "melt-in" carbon should be at least 0.55 to 0.75%.

To expedite the oxidizing of the carbon content, iron oxide, usually in the form of high-grade iron ore, is fed to the bath, together with some lime, to hold or increase the basicity of the slag to the proper degree. This operation may be repeated several times if the bath carbon is relatively high. Usually with high hot-metal practice, the "melt-in" carbon is much higher than ideal and, consequently, quantities of oxide and lime are added as often as they may be digested by the bath in order to reduce the carbon rapidly. The progress of the reduction of the carbon is closely watched through the medium of frequent samples which are analysed for this element either by means of the appearance of a fracture of a test bar or through the electric resistance method.

During the process of the elimination of the metalloids, the condition of the slag is carefully watched through the examination of samples, and the necessary adjustments made in order to arrive at the desired basicity and iron oxide content. The furnacemen become very expert in reading the slag composition from its colour and the form of the sample.

The foregoing has been a description of the working of a heat that melted with a carbon content sufficiently high to allow the proper refining schedule to be carried on. Occasionally a heat will melt in "low," that is, having an insufficient carbon content to permit the refining to proceed properly. In such a case, molten iron, pig iron or spiegel will be added to the bath to increase the carbon content to a sufficiently high level to permit proper working of the heat. This procedure is time-consuming and therefore costly and is to be avoided if possible.

Assuming that a melt of steel and its slag are both in the chemical and physical shape desired, the finishing of the heat may now proceed. This operation will differ considerably depending on which of the three principal types of steel is specified. These types are as follows:

1. *Killed steel.* When the furnace is tapped, the steel in the ladle will be as completely deoxidized as possible through the addition of silicon-bearing alloys in the furnace and ladle, and aluminium in the ladle. Killed steel will generally, but not always, be teemed in hot-topped moulds in order that the shrinkage cavity or pipe may be confined to the hot-top portion of the ingot. This type of ingot shows less segregation, is cleaner and sounder in structure than the other types and may be made to virtually any chemical specification.

2. *Semi-killed steel.* This is partially deoxidized metal, is teemed into open-type moulds and, qualitywise, lies between killed steel and rimmed steel.

3. *Rimmed or effervescent steel.* The deoxidation of this type of steel, while controlled, is carried out to a much less degree than of the other types, for the reason that when the ingot is solidified a layer of small gas bubbles will exist just under the surface skin of the ingot and these are responsible for the smooth surface of the ingot and of the finished product after rolling. This type of steel is particularly suited for the manufacture of flat-rolled products such as strip and sheets. It is probably the cheapest type of steel to produce but shows the highest incidence of segregation. The use of this type of steel is subject to limitations in that 0.25 or 0.30% carbon and 0.60% manganese are about the upper chemical limits of composition in which it can be manufactured. It is always teemed in open-top moulds. There is a sub-type of rimmed steel termed "capped steel" which is teemed into bottle-top moulds and the rimming action is retarded by capping with a heavy cast-iron cap, thus rapidly cooling the top of the freezing metal.

Given the type of steel to be made and its composition, the first move is to take samples of metal and slag for the final furnace analysis. The usual analyses of the metal are for carbon and manganese in order that the additions may be calculated to secure the final specification, and for FeO in the slag for the knowledge it conveys as to the amount of oxygen in the bath.

With the samples on the way to the laboratory, the heat is "blocked" when killed or semi-killed steel is ordered. The heat is "blocked" by an addition, generally of ferrosilicon, that acts to retard any furthe chemical action in the bath of metal until the silicon in the addition is oxidized, usually a matter of about 20 minutes. A heat of rimmed steel is never "blocked."

The results of the analyses should reach the furnace operator in 15 or 20 minutes and the furnace addition of ferromanganese necessary to give the ladle analysis desired is then made together with any deoxidizer required to produce the ingot type. The carbon in the furnace additions will represent a portion of that necessary to secure the final carbon content desired.

About 15 minutes after the final furnace addition, the steel is tapped, and, as the metal flows into the ladle, the final additions of carbon together with the deoxidizers, mainly ferrosilicon or aluminium, are made. A small addition of fine ferrosilicon or "shot" aluminium may be made at the moulds, in the case of rimmed steel, to control the rimming section in the moulds.

When the steel in the moulds has solidified to the proper degree, the resulting ingots are "stripped" from the moulds by means of a special type of crane and transported to the soaking pits, in which the ingot temperature is equalized for the initial "blooming" or rolling operation from which the steel emerges as semi-finished blooms, slabs or billets, which are subsequently rolled into finished form.

THE COMPOSITION OF THE RESULTING STEEL

The basic open-hearth process, when properly operated, is capable of converting blast-furnace metal and certain scrap iron and steel scrap, that collectively contains amounts of phosphorus and sulfur in excess of those allowable in the finished product, into acceptable metal containing the desired percentages of carbon, manganese and silicon. It is further capable of producing all of the alloy grades which are specified as suitable for manufacture by basic open-hearth methods in section 10 of *Steel Products Manual*, published by the American Iron and Steel Institute. Specifically, the chemical limits are as follows:

1. *Carbon steels* (C10XX Series)—Carbon 0.08 to 1%, manganese 0.25 to 1.50%, silicon a trace only, phosphorus 0.040% maximum and sulfur 0.050% maximum. Some carbon steels are specified to 0.025 or 0.030% sulfur, maximum.

2. *Alloy steels*—Commonly such series as 23XX, 31XX, 33XX, 43XX and 48XX covering steels specified as nickel steel, nickel-chrome steel, nickel-chrome-moly steel, nickel-moly steel, etc. The combination of alloy content may extend to 5% nickel, 1.75% chromium and 0.80% molybdenum. In the manufacture of the alloy steels, the nickel may be added to the bath at any convenient time inasmuch as it does not oxidize under open-hearth conditions; molybdenum, although only slightly oxidizable, is best added later; while a part of the chromium, as ferrochrome, should be added only a few minutes before the tap and the balance as some form of exothermic chrome compound in the ladle.

3. *Ingot iron*—An almost pure iron containing not over 0.15% of the combined elements of carbon, manganese, phosphorus and sulfur, which requires higher furnace temperatures to finish than do the C10XX series.

4. *Silicon steel*—Electrical sheets used in the manufacture of motors and transformers and containing up to 5% silicon with virtually no carbon or manganese and with a minimum of phosphorus and sulfur.

The only types of steel that may not be produced by the basic open-hearth process are the stainless and tool steels and also a few special steels used in the aeronautical industry.

In the foregoing comments on carbon and alloy steels, the ranges of composition of various types of steel have been indicated. For instance, any desired carbon content of from 0.08 to 1% with a manganese content of 0.25 to 1.50% may be specified in the C10XX series. However, this series is divided into about fifty distinct grades, each having a systematic variation of two or three hundredths per cent of carbon and considerable variation in the manganese content as may be needed to meet the physical specifications required in the finished product.

If a steel with, say, 0.20% of carbon and 0.45 or 0.50% of manganese is needed, the purchaser would specify C1020 steel. In accordance with standard specifications, this steel would be supplied and accepted on a ladle analysis within the ranges of 0.18 to 0.23% carbon, 0.30 to 0.60% manganese and with not over 0.04% phosphorus or 0.05% sulfur. In the lower ranges, the carbon may allowably vary two or three hundredths per cent, and in the higher ranges six or seven hundredths on either side of that specified, but with the manganese the variation is fifteen hundredths on either side for all the analyses of the C10XX series. However, it is the aim of any good practical steel maker to meet the specifications as closely as possible. Heats analysing outside of the specification limits are "missed" heats, and while an occasional "missed" heat is inevitable, more than a very few compromise the melter's competence.

CERTAIN COMMENTS CONCERNED WITH THE ESTABLISHMENT OF BASIC OPEN-HEARTH STEEL PRODUCTION

As a preliminary and significant step in their quest for a greater degree of self sufficiency for their country, various governments and quasi-governmental or business groups may seek to establish or extend a steel industry within its borders inasmuch as finished steel in many of its various forms is a necessary prerequisite to increased political, social, agricultural or industrial development above their existing levels.

The feasibility of any such step is dependent on many major factors of which those of markets, raw material supplies and finance are probably all of about equal weight. With the exceptions of the original Latin-American integrated plant of the Compañia Fundidora de Fierro y Acero de Monterrey at Monterrey, Mexico, the recently constructed integrated plant of the Compañía de Acero del Pacifico at Concepción, Chile, and that at Chimbote, Peru, now in the engineering stage, in nearly every case a steel industry of a country has found its beginning in small semi-integrated open-hearth or electric furnace installations, usually producing concrete reinforcing bars or wire rods from local scrap metal and generally existing and continuing economically through the workings of a special protective tariff.

To be economical in a world-wide sense, a steel plant must be completely integrated and have an annual tonnage production upwards of a quarter million tons. Complete integration includes iron smelting and coke converting facilities and the supporting iron ore, flux and coal supplies of proper qualities, reasonably close to the processing plant, together with an ingot producing plant which, depending on many conditions, may have open-hearth or electric melting furnaces or Bessemer converters, and the necessary breakdown (blooming) mill and such finishing mills as are required to produce the steel in its final and desired form.

It can generally be expected that the initial integrated plant in a country undergoing development will be obliged to operate on a high hot-metal charge, using the hot metal-ore-scrap process as detailed in Table 1 inasmuch as only a relatively small and generally insufficient supply of dealer scrap will be available. Such was the situation upon which the original installation at Volta Redonda for the Companhia Siderúrgica Nacional of Brazil was postulated. A similar condition also governed the design of the plant of Paz de Río in Colombia, although the steel conversion there will be by the basic Bessemer rather than the open-hearth method due to certain local conditions. It appears probable that the acid-Bessemer and open-hearth plants of the Compañía de Acero del Pacifico at Concepción, Chile, the open-hearth plant of Altos Hornos at Monclova, Mexico, as well as the electric smelting and melting plant proposed for erection near Chimbote, Peru, by the Corporación Peruana del Santa were premised on the same circumstances.

The foregoing remarks refer to the five larger fully integrated steel producers of Central and South America that have, or will shortly have, substantial pig iron and steel ingot capacities. There is a small charcoal blast-furnace installation in Argentina producing pig iron and also 10 or 12 small blast furnaces in Brazil with two of the producers, Companhia Brasileira de Usinas Metalúrgicas and Companhia Siderúrgica Belgo-Mineira, operating small open-hearth steel and rolling-mill plants.

In the absence of the necessary raw materials within their borders for producing pig iron or the remote possibility of securing these under favourable conditions as imports, the origination of a steel industry which inherently will be of small capacity must be founded on a semi-integrated or cold-metal process using scrap ferrous metals, some of which may be imported should the supply of a district be insufficient to support the tonnage production proposed. Any such operation would employ a charge similar to the cold-metal charge scheduled in Table 1, excepting that, should the electric melting furnace be selected as the conversion medium, the iron component could be almost entirely omitted.

Throughout Central and South America, there are quite a number of small operations of this character of which only a few will be mentioned. Possibly the largest of these is La Consolidada of Mexico City with a basic open-hearth plant at Piedras Negras which is supplied with imported scrap. The ingots are shipped to the Mexico City plant for rolling, and the steel-making operations are there supplemented by an electric furnace melting operation which utilizes the plant scrap and a part of the local scrap supply. There is a sizable semi-integrated open-hearth operation at São Paulo, Brazil, the Mineração Geral do Brasil Ltda.

(Brazilian Mining and Metals Company), and there are various small semi-integrated operations in a number of the Latin-American countries.

A measure of comment has been included herein on the steelmaking methods and facilities of all of the steel producing countries comprising the Americas with the exception of those of our close neighbour to the north, Canada. Canada has the substantial ingot producing capacity of nearly four million tons of ingots per year with considerable new capacity now under construction. Of the steel production, more than 80% is produced by the basic open-hearth process. There are four large integrated producers and a number of smaller non-integrated plants employing either the open-hearth or electric-furnace method.

Three of the four integrated companies produce steelmaking iron from the ores of the Lake Superior region, and consequently have a basic open-hearth practice similar in all respects to that of the United States Great Lakes section. The fourth integrated producer located at Sydney, Nova Scotia, produces iron from the ores of Wabana, Newfoundland, which are of high phosphorus content, the phosphorus in the iron exceeding 1.5%. At this plant, steel is produced in tilting open-hearth furnaces from high iron charges and without the benefit of pre-processing to expedite the elimination of phosphorus.

THE STEEL INGOT CAPACITY OF THE AMERICAS

As a closure, it may be of interest to scan certain statistics relating to the steel-producing capacity of our Western Hemisphere. These are listed in Table 2. Excepting as to Canada and the United States, the data relating to the steel-producing capacity has been estimated from sources believed to be reliable. There may be some small steelmaking plants that have been unavoidably omitted from the tabulation, but, if so, their inclusion would not substantially affect the data as presented.

While the data of Table 2 are quite self-explanatory, it would seem that those countries of Central and South America having a population of five million or more have made or are making substantial progress in establishing a steel industry that must surely have a definite value in heightening the economic standard of their entire population, and, as the ways and means are uncovered for the expansion of these industries, the economic betterment to all of their population will be found in rapidly increasing ratio.

Table 2
THE STEEL INGOT CAPACITY OF THE AMERICAS

Country	Area in square miles (000 omitted)	Population (000 omitted)	Steel ingot-producing capacity		
			Tons per year (000 omitted)	Pounds per capita	Per cent basic O-H
South America					
Argentina	1,079	17,100	n.a.	—	—
Brazil	3,286	52,600	1,000 [a]	38	95.0
Chile	286	5,800	275	94	53.5
Colombia	440	11,300	200	35	none
Ecuador	176	3,100	none	—	—
Paraguay	151	1,400	none	—	—
Peru	514	8,400	75 [a]	18	none
Uruguay	72	2,300	none	—	—
Venezuela	352	5,000	20	8	none
TOTAL	6,356	107,000	1,570	29	67.0
Central America					
Guatemala	42	2,800	none	—	—
Honduras	44	1,500	none	—	—
Mexico	760	25,600	550	43	60.0
Nicaragua	57	1,100	none	—	—
Panama	29	800	none	—	—
Salvador	13	1,900	none	—	—
TOTAL	945	33,700	550	33	60.0
North America					
Canada	3,845	13,800	4,000 [a]	580	80.0
United States	3,022	155,000	120,000 [a]	1,550	88.5
TOTAL	6,867	168,800	124,000	1,470	88.2
GRAND TOTALS	14,168	309,500	126,120	820	87.7

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[a] At completion of present construction programmes.

Economics of the Modern Acid Open Hearth Practice

G. R. FITTERER

1. RECENT PROGRESS IN ACID OPEN HEARTH TECHNOLOGY

The technology of acid open hearth operation has been advanced considerably throughout the past eight or ten years by the Acid Open Hearth Research Association. A list of the member companies of this group is given in another paper regarding products.

It is now possible to refine low carbon, plain carbon steels in the acid open hearth so that the refining operation can be reduced to as little as one half hour. Meanwhile, the heat will be raised to a desirable tapping temperature and simultaneously be ready for tapping from the standpoint of chemical analysis. This indicates that the acid open hearth can be highly active chemically under certain controlled conditions to the extent that the elimination of carbon can take place at a rate which is equal to that of any other steel making furnace except for the Bessemer. The development in technology to which this can be attributed is to be found partially in a chemically balanced charge, and secondly, in the control of temperature during the refining process. Proper manipulation of fuel will produce the desired results listed above.

It will be recalled that the heat data which were published many years ago indicated that the acid open hearth was a very slow steel refining method. Many technologists have repeated this mis-statement without the realization that with the new control methods it may now be caused to refine rapidly with a very active carbon boil. The older mis-impression possibly originated from the fact that most of the older heat data (logs) described furnaces which were heated with producer gas. The calorific value of this gas is so low that insufficient heat for making it an efficient process was supplied to the acid open hearth. Only recently have acid heats made with more modern fuels been described in the literature. The acid slag is now recognized as a good heat insulator if the heat is inactive and there is little agitation. Once the carbon boil has begun, the transfer of heat to the metal bath is quite rapid. In recent years due to the use of natural gas (or fuel oil with special atomization), the acid open hearth has become a rapid steel producing process to the point where it can compete economically in certain instances with the basic process.

2. LOG SHEET OF A TYPICAL ACID OPEN HEARTH HEAT

Figure 1 (1) illustrates the trends which occur in the refining of steel in the acid open hearth according to modern practice in the United States. This 22-ton heat required two hours for charging in two steps and melted down within two hours after the charge was completed. Oil and air constituted the fuel. The charge consisted of 37,840 lb. of scrap, 5,800 lb. of low phosphorus pig iron and 500 lb. of silicon pig. The scrap consisted of 30% foundry scrap and 40% blooms, wheels and turnings, whereas the remainder was miscellaneous scrap. The pig iron was charged at the ends of the furnace and at the bottom to prevent excessive refractory (SiO_2); erosion by the iron oxide formed during melting. The estimated charge analysis as shown was 0.80% carbon, 0.75% manganese, and 0.70% silicon.

It may be seen from the figure that the heat melted with about 0.60% carbon, 0.08 each of silicon and manganese. A decrease in carbon content had already occurred at melt down showing that the carbon boil started prior to the addition of ore (i.e., 300 lb.). In this case, limestone is occasionally added to a heat to hasten the carbon boil. The CaO tends to replace the iron oxide in the slag and free it for reaction with carbon.

It may be seen that the refining time for this heat was about one and one half hours. With a slightly lower amount of manganese and silicon in the charge, the time could have been reduced considerably. At the end of the heat, ferro-silicon, ferro-manganese, and some silico-manganese were added to the furnace in order to deoxidize and complete the heat. In this instance, the alloys were added to the bath in the furnace and some aluminium was added to the ladle.

It may be seen from the figure that during refining, the temperature of the liquid metal increased in proportion to the decrease in carbon content. Just prior to the greatest rate of carbon elimination, the temperature increase was most rapid. Also, the increase in temperature and decrease in carbon were accompanied by a proportionate change in the slag fluidity. A change in slag fluidity is indicative of a change in the density of the slag. The fluid slags are more dense and contain a greater percentage of iron oxide. A less fluid slag indicates a high SiO_2 content.

In most heats made according to modern acid open hearth practice in the United States, the manganese oxide content of the slag is low as a result of the low manganese content of the charge. In this particular heat, the manganese oxide content of the slag was approximately 12 to 14% at which level it remained throughout the heat.

From this log sheet it is rather obvious that by controlling the rate of temperature rise, (i.e., the fuel input) a heat can be refined at any desired rate. Further, by decreasing the fuel input and holding the temperature constant towards the end of a heat, the carbon elimination may be retarded to a point where it will essentially cease. This is an advantage peculiar to the acid process.

3. THE ACID OPEN HEARTH CHARGE

None of the acid open hearth furnaces in the United States are charged with hot metal and only one or two plants have blast furnace facilities. Because of this, the acid open hearth can be compared only with the "cold metal" basic shops where cold pig iron is used as a portion of the charge.

Pig iron

When high grade pig iron is available at an economical price, the acid shops prefer to use it. However, the quantity used is a matter of plant preference and varies from 4 to 25% of the charge. Chateaugay pig is typical of the high grade desired. It contains 0.035% maximum phosphorus and/or sulfur. The silicon content varies from 0.50 to 3% and the manganese from 0.75 to 1.25%.

Some intermediate Bessemer grades are also sometimes available, and in these the sulfur is 0.05% maximum and the phosphorus is 0.075% maximum. Some ordinary Bessemer pig is also used and mixed with the

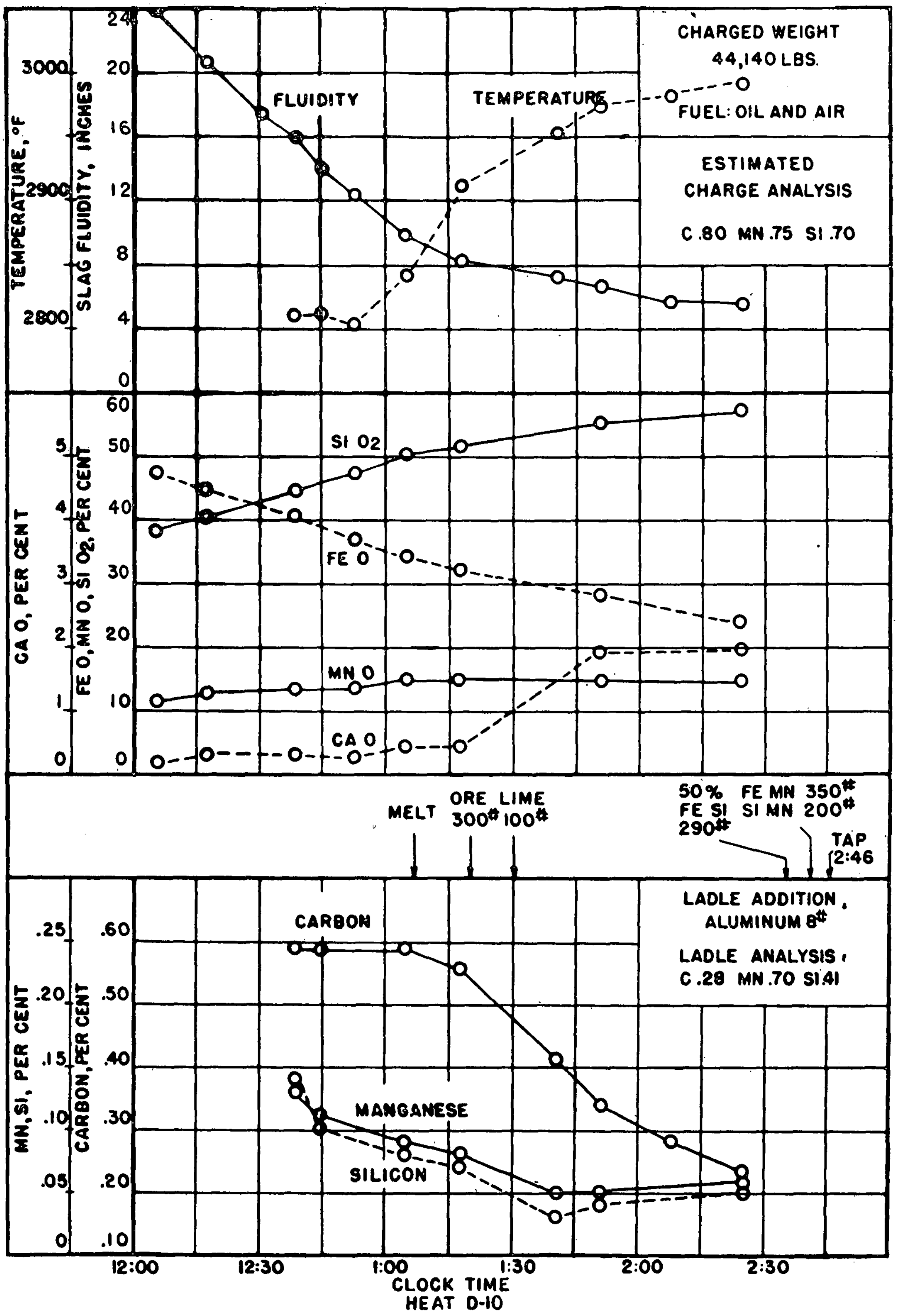


Figure 1
Log Sheet of a Modern Acid Open Hearth Heat (U.S.A.).

charge. This seldom is acceptable if the sulfur is above 0.050% and the phosphorus above 0.100%.

During recent years, the acid open hearth plants in the United States have had such difficulties in obtaining good pig iron that many of them are not using any pig iron. Instead, so-called "carbon-substitutes" such as graphite, carbrite, charkets, etc., are used. The carbon content of the charge is thus raised and small amounts of ferro-silicon and ferro-manganese are also charged, as needed, to increase the metalloid content. Otherwise, the charge is entirely scrap. When the local economics permit, however, a scrap-pig charge is preferred.

Type of scrap

A. R. Altman and Edward Walkowski (2) described many interesting relationships regarding the modern acid open hearth charge. It was shown particularly that if scrap is purchased on a selected basis so that such very light materials as purchased bundles or turnings are kept to a minimum, the efficiency of the acid open hearth is increased materially. They divided scrap into two categories, "good scrap" and the "balance of the charge". The "good scrap" was defined as "home" heavy, purchased crops, and purchased pipe. The "balance of the charge" was listed as "rusty", light turnings and bundled scrap. With an increase in percentage of the "good" scrap, the production rate in tons per hour increased materially. For example, when the good scrap varied from 15 to 35%, the production rate in tons per operating hour increased from 3.4 to 4.0.

It was also shown by these authors that a better production rate was obtained with low manganese in the charge (i.e., from 0.20 to 0.30%) as compared with a higher manganese charge (i.e., from 0.70 to 0.80%). The effect of the so-called poor or light scrap on the operation of the acid open hearth is probably due to its low density and bulkiness. Such scrap has a much lower heat conductivity.

Effect of furnace efficiency

Studies made by the Acid Open Hearth Research Association (3) regarding the effect of charging on the efficiency of operation have been very interesting. It has been found, for example, that a given furnace has an ability to melt scrap at a certain rate depending upon its fuel efficiency at a given stage in the furnace campaign.

Charging rate

With a given efficiency, a furnace should melt scrap at a certain rate. However, some mechanics are involved in the charging of a furnace which may prevent a furnace from operating at its maximum efficiency. This is particularly true where more than one charging cycle is used in the operation of a single heat. It has been found (3) that numerous charging cycles are satisfactory and can assist the furnace in operating at its highest efficiency, provided that each cycle is properly scheduled. In other words, after the first charge has been made to the furnace, there is a certain optimum time for the addition of the second charge and similarly for a third and fourth. If the charge is not delivered to the furnace at this optimum time, then the melting will be less efficient. Figures 2 and 3 show two types of heats, No. 1 Controlled Heat operated close to the optimum efficiency of the furnace. In other words, charges were delivered at the best time, whereas the other (No. 2 Controlled Heat) shows a charging delay and its effect upon the efficiency of the furnace in normal operation.

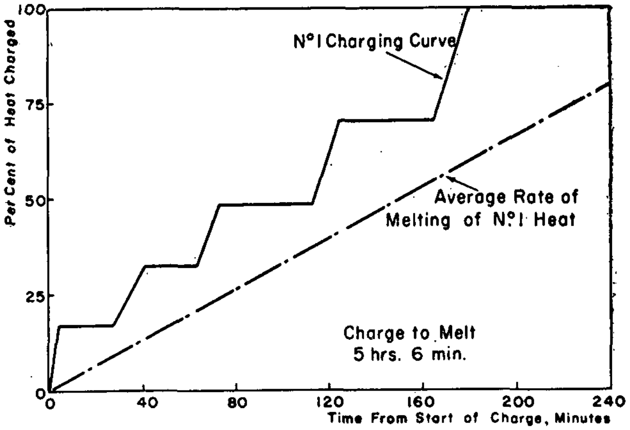


Figure 2. Rate of Charging for No. 1 Controlled Heat (Plant 2-O).

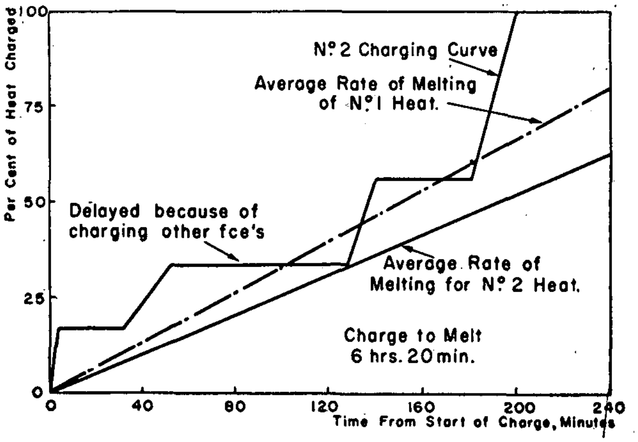


Figure 3. Rate of Charging for No. 2 Controlled Heat (Plant 2-O).

The chemical balance of the charge

Table 1 (3) indicates a survey of a number of acid open hearth heats relative to the conditions of the heats at melt down as compared to the amount of metalloids charged. This table also includes some other interesting information such as the size of the furnace, the type of product, the fuel used and the average amount of metalloids, carbon, manganese and silicon which are lost during the melt down period. It is rather obvious from this chart that although appreciable amounts of manganese and silicon are charged in the furnace, there are only small amounts of these materials remaining at "melt down". In most instances, this is less than 0.10% of either of these two constituents.

The amount of metalloids charged is one of the most important factors of modern acid open hearth control. If the silicon and manganese are too low, the heat will probably melt-in on a carbon boil. Under such circumstances, the carbon might be reduced below the desired finishing analysis before the tapping temperature has been reached. If the amount of silicon and manganese charged is too high, the heat will melt-in "dead" and considerable time will be lost before the carbon boil starts.

The need for such a chemical balance of the charge is now common knowledge. Thus, only sufficient manganese and silicon are charged in modern American acid open hearth practice to protect the carbon suffi-

Table 1

COMPARISON OF MELT DOWN PRACTICES OF TWENTY-EIGHT ACID OPEN HEARTH HEATS

					Charge analysis			*Melt down analysis*			*Metalloid losses*		
Heat	*Type*	*Fuel*	*Charge weight (tons)*	*Tons/hour Charge/melt*	*C %*	*Mn %*	*Si %*	*C %*	*Mn %*	*Si %*	*C %*	*Mn %*	*Si %*
A-1	Castings	Natural gas	26.0	6.23	0.89	0.71	0.64	0.65	0.08	0.06	0.24	0.63	0.58
A-2	Castings	Natural gas	41.4	5.40	1.32	0.75	0.56	1.08	0.08	0.05	0.24	0.67	0.51
M-1	Castings	Natural gas	40.3	5.57	0.99	0.66	0.40	0.41	0.06	0.03	0.58	0.60	0.37
M-2	Castings	Natural gas	40.1	6.68	0.95	0.65	0.40	0.35	0.05	0.05	0.60	0.60	0.35
M-3	Castings	Natural gas	39.4	4.85	0.90	0.65	0.40	0.29	0.05	0.04	0.61	0.60	0.39
X-1	Castings	Natural gas	26.0	4.81	0.75	1.20	0.61	0.40	0.07	0.05	0.35	1.13	0.56
C-1	Ingots	Oil and steam	37.8	6.99	1.24	1.28	1.07	0.99	0.31	0.30	0.25	0.97	0.77
C-7	Ingots	Oil and steam	38.3	7.66	1.02	1.27	1.12	0.84	0.35	0.34	0.18	0.92	0.77
G-1	Ingots	Oil and steam	62.5	10.10	1.18	0.72	0.64	0.76	0.08	0.07	0.42	0.64	0.57
N-1	Ingots	Oil and steam	49.4	9.88	1.42	1.32	0.82	0.97	0.10	0.04	0.45	1.22	0.78
N-2	Ingots	Oil and steam	49.5	9.90	1.44	1.28	0.87	0.91	0.14	0.07	0.53	1.14	0.80
S-1	Ingots	Oil and steam	68.4	10.25	0.84	0.60	0.35	0.21	0.04	0.02	0.63	0.56	0.33
T-1	Ingots	Oil and steam	105.0	10.22	2.03	0.80	0.43	0.96	0.12	0.04	1.07	0.68	0.39
F-1	Castings	Oil and steam	20.5	3.97	1.01	0.68	0.70	0.52	0.04	0.05	0.49	0.64	0.65
V-1	Castings	Oil and steam	20.8	5.12	1.06	0.86	0.43	0.54	0.05	0.05	0.52	0.81	0.38
Z-1	Castings	Oil and steam	45.0	6.43	1.06	0.75	0.41	0.45	0.03	0.05	0.61	0.72	0.36
E-1	Ingots	Oil and air	42.5	8.37	0.78	1.00	0.60	0.58	0.13	0.08	0.20	0.87	0.52
L-1	Ingots	Oil and air	46.6	7.86	1.72	0.68	0.51	1.50	0.10	0.10	0.22	0.58	0.41
D-1	Castings	Oil and air	18.0	4.59	1.00	0.80	0.70	0.64	0.09	0.06	0.36	0.71	0.64
H-1	Castings	Oil and air	23.0	4.45	0.92	1.00	0.52	0.45	0.04	0.06	0.47	0.96	0.46
I-2	Castings	Oil and air	18.2	5.75	0.72	0.93	0.59	0.37	0.08	0.10	0.35	0.85	0.49
K-1	Castings	Oil and air	11.2	3.46	0.81	0.61	0.40	0.15	0.03	0.03	0.66	0.58	0.37
O-1	Castings	Oil and air	20.0	5.46	1.15	0.67	0.44	0.54	0.07	0.05	0.61	0.60	0.39
P-1	Castings	Oil and air	32.2	4.68	0.79	1.03	0.45	0.44	0.07	0.04	0.35	0.96	0.41
Q-1	Castings	Oil and air	13.0	3.19	0.52	0.90	0.41	0.23	0.09	0.05	0.29	0.81	0.36
U-1	Castings	Oil and air	26.0	5.21	0.71	0.71	0.46	0.24	0.08	0.05	0.47	0.63	0.41
W-1	Castings	Oil and air	42.6	7.51	0.82	1.21	1.02	0.56	0.12	0.12	0.26	1.09	0.90
Y-1	Castings	Oil and air	25.9	——	0.92	1.62	0.62	0.52	0.08	0.05	0.40	1.54	0.57

ciently so that at melt down the metal has the desired carbon content. The practice, however, varies from both the standpoints of plants and products.

In manufacturing low carbon steel castings, only small amounts of manganese and silicon are normally charged. Often in such heats no ore is needed for refining. The slag and furnace atmosphere supply all the necessary oxygen for refining.

For higher carbon grades of steel (i.e., particularly forgings and alloy grades), the manganese and silicon contents are usually charged sufficiently high that the heat does not melt-in on a carbon boil. Instead, there is usually sufficient manganese and silicon present at the melt down point that an ore addition is required to remove the manganese and silicon and to start the carbon boil. Once the carbon boil has started, little ore is needed for the refining.

The effect of fuel on the chemical balance

The type of fuel has a considerable effect on the amount of manganese and silicon charged in acid heats. It has been found by the Acid Open Hearth Research Association (3) that the fuels used are oxidizing in the increasing order as follows: 1. natural gas, 2. oil atomized with air, and 3. oil atomized with steam. The latter is extremely oxidizing and larger quantities of manganese and silicon must be charged in order to protect the carbon. Similarly, a furnace using oil and air as fuel requires at least the same and possibly a little more manganese and silicon in the charge than does a natural gas heat.

4. THE REFINING PERIOD AND ITS CONTROL

Influence of the charge on refining

It may be seen from the above that if the proper balancing of the charge is considered with respect to carbon, manganese and silicon, other factors such as the type of scrap, the efficiency of the furnace and the type of fuel, etc., an acid open hearth can be charged intelligently to a point where most heats melt-in with the desired analysis. As shown above, this depends largely upon the product and the desires of the operator. A heat may be caused to melt-in on a carbon boil or "dead" as desired. During the carbon boil, heat is transferred quite readily to the metal from the slag blanket and furnace refractories because of the agitation of the boil. However, when the heat melts-in with sufficient silicon and manganese to be called a "dead" melt, the slag acts as a strong heat insulator. Heat is accumulated in the refractories of the furnace to the extent that once the carbon boil does start, the temperature pick-up is extremely rapid through radiation from the furnace walls and roof and convection from the flame. Once started, the carbon elimination is very rapid as will be shown.

Rate of refining more dependent upon rate of temperature increase than the amount of ore added

It has been found in investigations by the Acid Open Hearth Research Association (3) that the rate of temperature increase in the acid open hearth very strongly influences the rate of carbon elimination. The greater the rate of temperature increase, the faster will the carbon be eliminated from the steel. This is caused by an upset of slag phase equilibrium and the greater availability of FeO from the slag (4). Contrary to general opinion, the refining rate in the acid open hearth can be as great as 0.50 to 0.60% carbon per hour. This will be recognized as being appreciable.

If the charge has been properly balanced with respect to metalloids, it may be possible to make heat after heat in the acid open hearth without any oxidizing addition such as ore for the elimination of these metalloids. In other words, during the increase of temperature as mentioned above, iron oxide is made available from the slag (i.e., freed from other chemical association). Sufficient iron oxide is present in the slag at melt down for the elimination of all the carbon desired in a heat under these controlled conditions. Therefore, it is only necessary for one to balance the charge properly, control the rate of the bath temperature increase and usually no ore need be added. Thus, the amount of ore added for carbon elimination may vary from zero to about 50 lb. per ton of the heat. In the latter case, some iron is gained. Of course, in practice this is not always true because of unexpected variations in the scrap analysis. It is difficult for one with experience only with the basic hearth to remember that the FeO content of the slag decreases with decreased carbon content in the acid furnace (see Figure 1). A higher iron oxide content of the slag is required for lower carbons in the basic process.

Modern control requires knowledge of bath temperature, slag fluidity, and carbon analyses

As stated above, it is very important to know the temperature trends of a heat of acid open hearth steel. In recent years, the Acid Open Hearth Research Association (5) has developed practical control methods. Three special devices are used in connexion with this control procedure, that is, a platinum, rhodium thermocouple, a slag viscosimeter and a carbanalyzer (or carbometer). Through the use of these devices, the operator may follow the trends in the temperature and the carbon content of the metal as well as the analysis of the slags.

It has been shown in previous papers (4, 5) that the analysis of the acid open hearth slag may be revealed directly by the slag viscosimeter test (Figure 4-A). Furthermore, the relationship between the slag analysis and its fluidity is a direct temperature relationship (Figure 4-B) so that the analysis of the slag may be quickly estimated. Changes in the analysis of the slag strongly affect the oxidizing conditions of the heat.

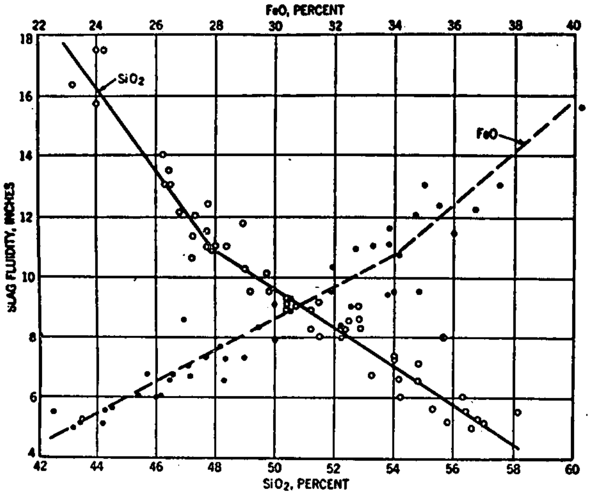


Figure 4-A. Relation of Slag Fluidity to the SiO_2 and FeO Contents of the Slag.

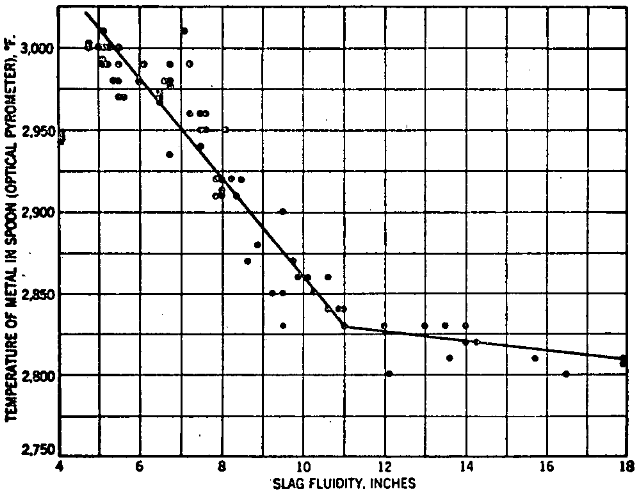


Figure 4-B. Relation of Metal Temperature to Slag Fluidity.

Through the use of these control methods, the heat may be adjusted so that the oxidizing conditions may be reduced towards the end of the heat. In other words, if the carbon boil is too rapid, the condition can be corrected by reducing the fuel input. This will cause the acid slag to approach equilibrium so that very little oxidation of the metal will take place unless the temperature continues to rise. In other words, the carbon boil in the acid open hearth furnace may be stopped at will at any temperature level by simply reducing the input of heat to the furnace. Thus, as a given heat approaches the desired tapping carbon content and tapping temperature simultaneously, a reduction in the fuel input will cause a marked decrease in the rate of carbon elimination just prior to tapping.

This is an unusual feature which has no parallel in basic open hearth technology. Reducing the fuel input does not mean in this instance that the fuel is shut off entirely, but rather that it is merely reduced as needed. Sufficient fuel is used to maintain temperature, but unless the temperature is increased, there will be no further oxidation of the carbon and this is a major advantage of the acid process.

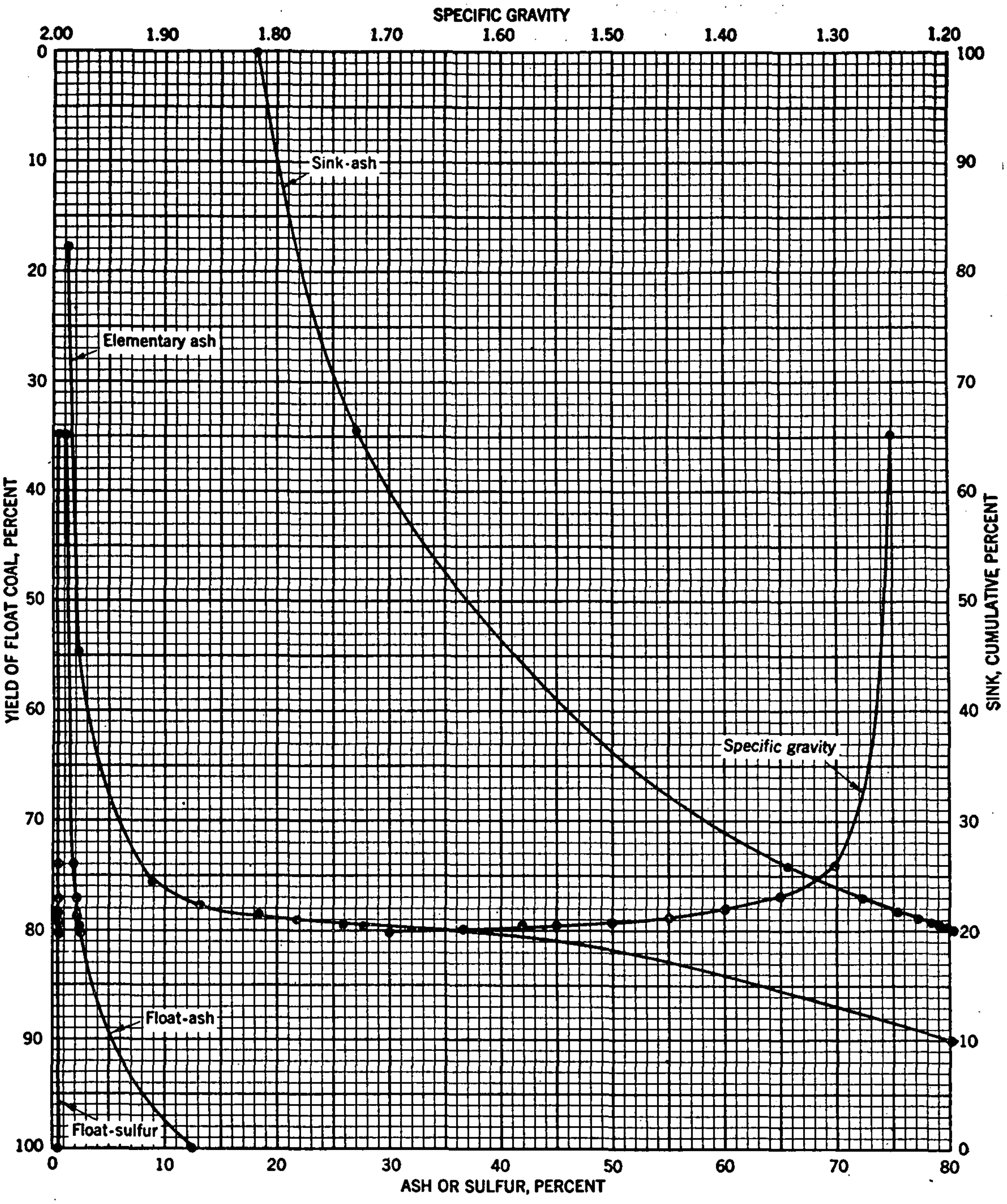
Because of the fact that the heat may be brought under control with respect to carbon elimination and temperature, the operator soon develops a certain degree of confidence in his ability to operate the heat rapidly and still keep the analysis under control. This feature of modern acid open hearth practice in the United States has considerably reduced the average refining time and additions. In fact, as stated previously, many heats are made today without any ore additions whatsoever.

5. COMPARISON OF MODERN WITH OLDER ACID OPEN HEARTH PRACTICE

The knowledge of these control methods has greatly increased the usefulness of the acid open hearth process as well as its economic feasibility.

Fuel differences

Older European practice involved the use of producer gas as the fuel and its low B.T.U. content was responsible for the slow heating rate of the metal. Consequently, there was a slow supply of iron oxide from the slag to the metal, and thus, a very slow rate of carbon elimination. Many of the older heat logs indicated that considerable ore additions were necessary in acid open hearth heats. Also, a considerable degree of erosion of the furnace banks was encountered.



Specific gravity fractions	Elementary data				Computed cumulative data					
	Weight grams	Percent			Float, cumulative percent			Sink, cumulative percent		
		Weight	Ash	Sulfur	Weight	Ash	Sulfur	Weight	Ash	Sulfur
Float - 1.25		34.8	1.21	.61	34.8	1.21	.61	100.0	18.23	.50
1.25 to 1.30		39.4	2.34	.62	74.2	1.81	.62	65.2	27.31	.44
1.30 to 1.35		2.8	8.86	.55	77.0	2.07	.61	25.8	65.44	.17
1.35 to 1.40		1.3	13.23	.54	78.3	2.25	.61	23.0	72.33	.13
1.40 to 1.45		.6	18.31	.50	78.9	2.37	.61	21.7	75.87	.11
1.45 to 1.50		.4	21.72	.50	79.3	2.47	.61	21.1	77.50	.09
1.50 to 1.55		.3	25.76	.55	79.6	2.60	.61	20.7	78.58	.09
1.55 to 1.58		.2	27.34	.42	79.8	2.62	.61	20.4	79.36	.08
1.58 to 1.70		.4	36.90	.41	80.2	2.79	.61	20.2	79.87	.08
Sink - 1.70		19.8	80.74	.07	100.0	12.23	.50	19.8	80.74	.07

Fig. 4. Washing characteristics of a sample of Chilean coal. Size 10 – 0.3 mm.

Difference in the charge

A second item which exemplified older European practice in contrast to modern American practice is that in most cases the manganese and silicon contents of the charge were considerably higher in the older heats due partially to higher pig iron charges. This resulted in "dead" melting and a high MnO slag. As stated, the average MnO content of acid open hearth slags in North America today is probably 12 to 13%, whereas in the older European practice, the manganese oxide content varied normally from 20 to 35%. Such slags which are now known to be much less oxidizing, reduce the refining rate and require many more ore additions.

Silicon reduction practice

The high MnO slags also originated the so-called "silicon-reduction-practice" in which the heat was held at a high temperature for long periods of time so that considerable quantities of silicon were reduced from the slag into the metal. As much as 0.40 to 0.50% silicon was picked up by the metal toward the tapping stage of such heats. Silicon may be reduced from the slag into the bath on lower manganese oxide slags, but this requires considerably higher temperatures and much more time at those temperatures.

Certain claims have been made in the literature regarding the high quality of steels made by silicon reduction practice. However, this has not been borne out in recent years. No company in the United States is following such a procedure, and acid steel is currently required to meet much more rigid specifications than ever before. In fact, there is considerable information available today to indicate that a lower manganese charge, a more vigorous carbon boil and a controlled temperature programme at the end of the heat, produces steel of very high quality and probably is as good if not better than any other procedure which has been developed for the acid open hearth to date. The vigorous boil which can now be obtained in the acid hearth undoubtedly prevents the pick-up of such gases as hydrogen which are known to be the chief source of difficulty in other processes such as the basic electric. True flakes which owe their existence to hydrogen are seldom found in acid open hearth steel. This is a real accomplishment particularly in view of some of the very large sections which are being made by this process for dynamic applications. Further, the control methods described above eliminate the need for any late ore additions and this is also in line with good steel making practice.

6. COMPARISON OF THE ACID OPEN HEARTH WITH THE BASIC

The comparison of the acid open hearth process with the basic requires many considerations. In the first place, there have been so many repeated statements regarding the disadvantages of the acid open hearth process that many misconceptions exist regarding its usefulness. Because of the fact that there have been many recent developments in the technology of the acid open hearth process, an attempt will be made to make a comparison of the processes with these older concepts in mind. This must be done, however, on two entirely different bases; first, their economics and, second, the type of products. Because of the fact that another paper is being presented at this meeting regarding the products of the acid open hearth, this phase of the problem will not be discussed in any great detail here. However, some data are available herein to permit a comparison on the basis of economics.

Relative economics of the acid and basic open hearth processes

No comparison should be made unless they are on the production of the same type of steel. For example, there is very little low carbon rimming steel being made in the acid open hearth as a regular procedure. Also, some of the high carbon-medium alloy grades of steel which are being made in the acid open hearth are not made in the basic. Probably the best steels for comparison would be the medium and forging grades, such as 0.30 and the 0.50% carbon non-alloy steels. This raises the question, of course, as to whether the acid open hearth with its premium payment for low phosphorus and sulfur scrap and pig iron is able to compete economically in the production of this grade of steel with the basic open hearth.

Relative production rates

In March of 1948, the American Iron and Steel Institute published a report (6) in which a comparison of production rates of basic open hearth furnaces on the basis of the cycle from tap to tap was made. This is important because it includes the bottom repair time.

The relationship between the charge weight for the heats against the respective production rate in tons per hour from tap to tap for the basic furnaces is shown as circled points in Figure 5. The line was drawn between these points according to the procedure of least squares. It may be seen from this line that the average production rate for basic open hearth furnaces increases with the charge weight from about four tons per hour with a 30-ton furnace to 10 tons per hour with a 125-ton furnace. The larger furnaces probably have a greater thermal efficiency, and also their charge is covered earlier by liquid metal than in the smaller furnaces, thus protecting the scrap from excessive

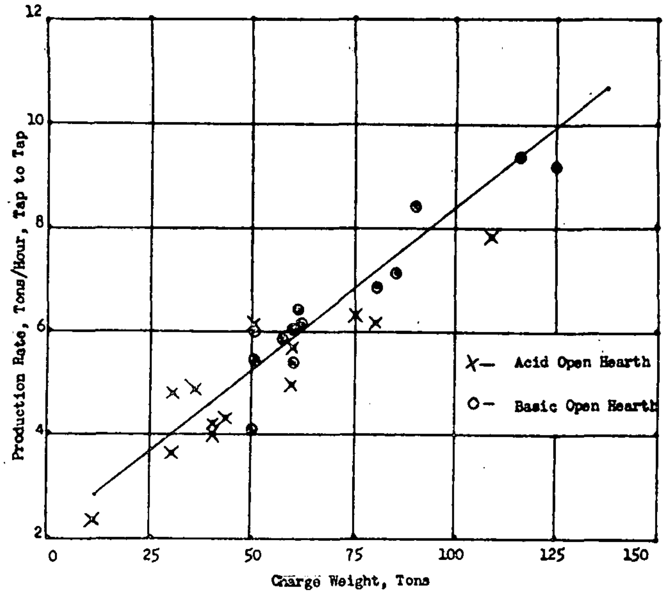


Figure 5. Comparison of Production Rates of Acid with Basic Open Hearths of Various Sizes.

oxidation. The larger furnaces also usually have a greater bath depth and this also coincides with the scrap oxidation theory. Because none of the acid open hearth plants in the United States are hot-metal shops, only those basic furnaces using a cold pig charge were plotted in this figure.

A similar survey was recently made by the Acid Open Hearth Research Association (3) regarding the production rates of acid open hearth furnaces and these data are shown in Figure 5 as the x-points. It may be seen from the figure that the acid open hearth furnaces have almost identical production rates when compared on a charge weight basis.

From the above it might be concluded that when acid open hearth furnaces are compared with cold-metal basic open hearths, they show an almost identical production rate in tons per hour from tap to tap with a given charge weight. If properly constructed, an acid open hearth has essentially the same fuel efficiency and ability to melt and refine steel on the basis of tons per hour from tap to tap as the basic open hearth. This would indicate that there is no peculiarity in either modern acid or basic open hearth practices which interferes with the fuel efficiency.

The situation might be stated in still another manner. If two furnaces were constructed with the identical regenerator design and differed only in the bottom lining material, there is no reason to believe that their ability to melt the charge would be different. The only difference which could occur would be found in the refining technique or time lost in repairing the bottom or side walls between heats. Even with respect to these items, the acid furnace also compares favourably with the basic.

It might be stated that without the previously described developments in acid open hearth technology, the acid open hearth production rate would be far inferior to that of the basic open hearth. The older silicon-reduction process was slow and inefficient. If, on the other hand, the basic operators had not found in the 1920's that considerable time may be saved by reducing the limestone charge, then the acid open hearth might show a greater efficiency in production rate than the basic in Figure 5.

However, both of these technological changes have taken place in the two types of furnaces and the production rate is largely a function of fuel efficiency. As stated previously, the combustion problems involved in the open hearth furnace are essentially the same in the acid process as they are in the basic.

Comparison on the basis of bath depth and/or hearth area

It is well known that the depth of the bath greatly affects the rate of refining in the open hearth. This is no less true of the acid lined furnace than it is of the basic. Consequently, it is agreed by many operators that the best method for comparison of the production of one open hearth with the other is on the basis of "tons per square foot of hearth area" per operating hour. This information is not too easy to obtain, and at best, the hearth area must be estimated. Obviously, if the charge is varied not only does the bath depth change, but also the area of the hearth because of the sloping banks. The hearth area can be estimated, however, knowing the originally designed hearth area from brick to brick and then estimating the bank refractory thickness. A normal charge must be assumed.

In the American Iron and Steel Institute report (6) mentioned previously, data were made available for the "tons per square foot of hearth area" together with the heat time from tap to tap. These two important factors have been plotted for convenience in Figure 6. An average line drawn through these points varies from 0.150 tons per square foot of hearth area and 7½ hours total time to 0.250 tons per square foot of hearth area and 15 hours. This may be expressed by the relation:

$$\text{Tons per square foot of hearth area} = 0.0133\ (\text{hours}) + 0.048$$

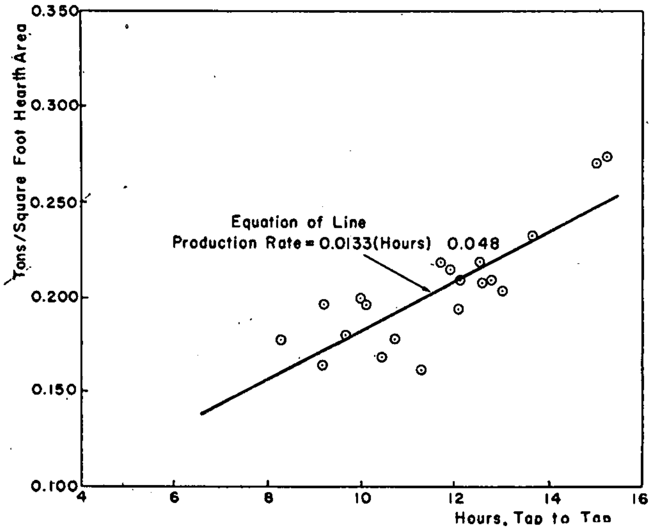


Figure 6. Production Rates of Basic Open Hearth Furnaces as Effected by Size of Furnace and Area of Hearth (Cold Iron Shops).

Similar data have been made available from reports of the Acid Open Hearth Research Association, but here it is essential that the heats be separated on the basis of whether the products are castings or ingots. These data are plotted in Figure 3 where it is obvious that the acid and basic open hearths again compare favourably on ingot heats. However, the casting heats made in the acid open hearth require markedly less time than do the ingot heats as shown. For example, at 0.150 tons per square foot of hearth area, the acid open hearth can produce a casting heat in six hours as compared with about ten hours from tap to tap for an ingot heat. At 0.250 tons per square foot of hearth area, a casting heat may be made in 9½ hours as compared with 13 or 14 hours for an ingot heat. Here then is a marked advantage in favour of the acid open hearth. Casting heats may be made much faster in the acid furnace. This combined with the greater apparent fluidity of acid open hearth steel, the lack of flakes and the present control features make it ideal for the manufacture of medium and large castings. Although ingot heats are normally made with greater refining time than casting heats in the acid furnace, there is no information to indicate that an ingot heat which is processed as rapidly as the casting heat would necessarily suffer in quality. The greater time cycle is merely a precautionary measure.

If acid heats were all normally made like the casting heats, the acid furnace might far excel the basic furnace in production rate. One reason for this may be the fact that in the manufacture of acid steel, no flux such as lime is charged. There is no need to wait for the lime boil and the solution of CaO in the slag as there is in basic practice. Consequently, the refining rate in the acid furnace is dependent upon the amount of the metalloids in the charge and the ability to supply heat to the metal. With these two factors well under control, the production rate for the acid furnace apparently can exceed that of the basic as shown in Figure 7. This is true when casting heats are compared. Acid forging heats are usually refined more slowly, but these compare favourably with the basic.

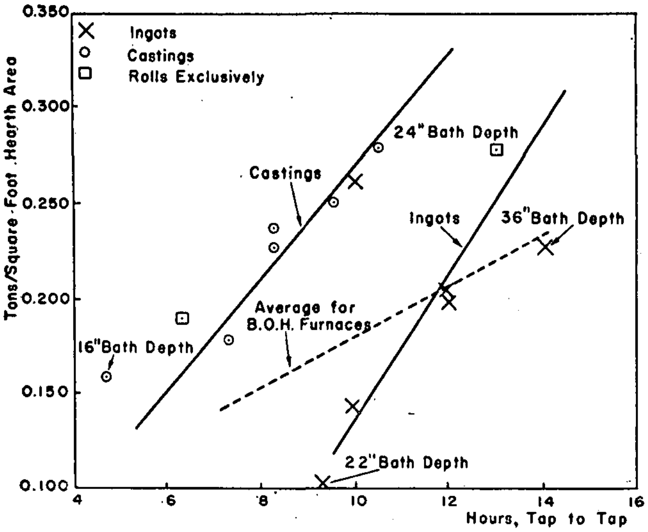


Figure 7. Production Rates of Acid Open Hearth Furnaces as Effected by Size of Furnace, Area of Hearth and Type of Product.

The economic discussion of the two furnaces, however, must be returned to the fact that the acid furnace will not remove phosphorus or sulfur. If sufficient high grade materials were made available to the acid furnace, it would be more economical to use this process. If means could be devised to remove these objectionable elements prior to charging in the furnace, the acid process would exhibit great advantages. Further, if the acid furnace were used with a hot metal charge, it would greatly assist the process.

It might be mentioned in passing also that the basic refractory costs are increasing so that this is becoming a major item of costs in the basic process.

The manufacture of alloy steels in the acid furnace represents another advantage of the process. Most of the oxides of the alloying elements are acidic in nature and because of the high SiO_2 content of the acid open hearth slag, alloy steels may be made with very little alloy losses. Consequently, analyses of heats are relatively easy to meet. Such a heat may be held in the furnace without change.

CONCLUSIONS

The data furnished in this paper have led to some conclusions with the objective of explaining why the acid process is still in use. Certain fields of application were described in which the acid process exhibits some decided advantages. Attention must again be returned to the fact that to date no technique has been developed to remove phosphorus and/or sulfur from acid steel. Without this feature, the acid open hearth must make use of its advantages in the manufacture of certain products.

Recent advances in technology have added to the list of advantages of the acid open hearth process which are summarized as follows:

1. The furnace may be charged so that a heat melts-in on a carbon boil or with a "dead" melt as desired.

2. Through fuel and other control methods, the heat may be refined with little or no ore addition, but with a vigorous carbon boil. This reduces the gas (hydrogen) content and "flakes".

3. The acid hearth is as efficient as the basic from the standpoint of over-all production rate.

4. The production rate of the acid furnace is greater than that of the basic when acid casting heats and hearth area are considered.

5. Heats may be held in the furnace without change in analysis.

6. Refractory costs are lower than the basic.

7. Alloy heats may be made without losses of alloys and with assured analyses.

ACKNOWLEDGMENTS

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Manufacture of Thomas (Basic Converter) Steel at Paz de Río, Colombia

EDOUARD DECHERF

INTRODUCTION

The purpose of this paper is to present the main departures from current European Thomas steelmaking practice imposed by local conditions, such as: quality of raw materials, altitude above sea level, size of plant and production programme of the rolling mills, in the design of the Paz de Río plant in Colombia. To ensure a better understanding, the paper contains a very summarized description of European—especially French—practice, and describes the departures therefrom in the order in which they appear.

MAIN CONDITIONS FOR OBTAINING GOOD QUALITY THOMAS STEEL

These conditions are:

1. Quality of the Thomas pig iron;
2. Quality of the lining;
3. The blow;
4. Precautions in teeming.

1. *Quality of Thomas pig iron*

(a) *Chemical analysis:*

Silicon: must be less than 0.50%.

Phosphorus: must be greater than 1.60/1.65% in order to obtain a good teeming temperature, especially if the Si content is around 0.25–0.35%.

Manganese: its requirements are a controversial point. If manganese is scarce, 0.30–0.40% is acceptable, but normally 0.45–0.60% should be used.

A good Thomas pig iron should show the following analysis:

Si—0.30/0.40% Mn—0.45/0.60% P—1.70/1.75%.

A consistent pig iron is essential. A mixer of adequate size should be used to minimize the variations in the pig iron from one heat to another. In general, the blast furnace operation is regulated to the basic ratio:

$$\frac{CaO}{SiO_2} = 1.40$$

The raw materials available in Paz de Río show the following analysis:

Table 1

ANALYSIS OF THE RAW MATERIALS TO BE USED IN PAZ DE RIO (percentages)

Components	*Iron ore*	*Coke ashes*	*Limestone*
Moisture	3.30	—	—
Weight lost in fire	11.12	—	39.52
Iron	48.00	—	—
Manganese	0.26	—	—
Phosphorus	0.99	traces to 0.4	—
Ferrous oxide (Fe_2O_3)	—	9.0 to 14.0	1.80
Silicium oxide (SiO_2)	10.53	58.0 to 62.0	1.57
Alumina (Al_2O_3)	5.88	22.0 to 28.0	0.51
Calcium oxide (CaO)	0.47	0.5 to 1.5	53.20
Magnesium oxide (MgO)	1.10	0.1 to 0.4	0.47
Sulfur	0.07	—	0.03
Carbon monoxide (CaO)	—	—	40.64
Carbonate of lime (CO_3Ca)	—	—	0.95
Others	—	—	—

It follows that the Paz de Río Thomas pig iron will be high in phosphorus, namely about 2%. This could be reduced through addition of scrap into the blast furnace charge, but the skips will make the handling of scrap difficult and this practice will, therefore, be omitted. Instead, scrap will be added into the converters and adequate equipment has been planned.

Total scrap production from the rolling mills will represent about 130 kg. per ton of pig iron used in the converters.

The low grade of the limestone will lead to an important proportion of limestone in the charge in order to keep the relation: lime to silica at about 1.40. On the other hand, the high phosphorus content of the ore will require a careful operation in order to keep the silicon content of the pig iron between the boundaries of 0.30 to 0.40%. In order to reduce these operating difficulties, 14% limestone will have to be added to the converters' charge.

The raw materials analysis shows also that the manganese content will be low, unless manganese ore is added. For the production of most of the more common use steels, the manganese content in the pig iron will be limited to 0.4%.

(b) *Pig iron temperature*

Outlet temperatures of 1,370°C. may be registered. It is essential to reduce to a minimum the temperature drop between the furnace and the mixer. From this, an outlet temperature of 1,250°C. is desirable for smooth operation of the converters.

2. *Quality of the lining*

This is a point of major importance, to which sufficient attention is not always given. A typical analysis of a good French dolomite is as follows:

	Fire losses	*CaO %*	*MgO %*	*SiO_2 %*	*Al_2O_3 %*	*Fe_2O_3 %*	*Without dosage*	*Remarks*
French dolomite								
Crude	45.85	31.50	21.00	0.350	0.800	0.400	0.100	Calcined with an addition of iron oxide
Calcined		55.7	37.10	0.60	1.45	5.0		

The SiO_2 content of the calcined product must not be greater than 2%, and the sum of the ferric oxide plus the alumina must not exceed 5 to 6%.

Colombian dolomite from Muzo has an excellent analysis; its silica content is very low and its iron oxide is adequate for a proper calcination. Its small grain and considerable friability exclude its calcination in a cupola furnace, the use of which has, in addition, the disadvantage of adding impurities from the coke to the dolomite. In order to use all the grain sizes of the Muzo dolomite, the particles under 0.8 mm. will be nodulized before being charged into the rotary furnaces.

These particular characteristics of Colombian dolomite impose some special precautions in the manufacture of dolomite bricks; for instance, a higher proportion of tar and the use of higher pressures are indispensable. If these precautions are taken, the resulting bricks are equal to good quality French or United States material.

3. *The blow*

There are three distinct periods in the refining of Thomas pig iron:

(*a*) The elimination of Si and Mn;

(*b*) The elimination of carbon;

(*c*) The elimination of phosphorus.

The amount of air required per ton of pig iron is always around 300 cu. m. The risk of metal projections commences with the beginning of the decarbonization process, and ceases at the end, during which period blowing pressure is limited. The classical blow commences with a relatively low pressure, increasing gradually after the risk of projection has passed up to the maximum capacity of the blower. This method has three defects:

(i) The pressure is increased manually, and is therefore seldom the same under identical charge conditions;

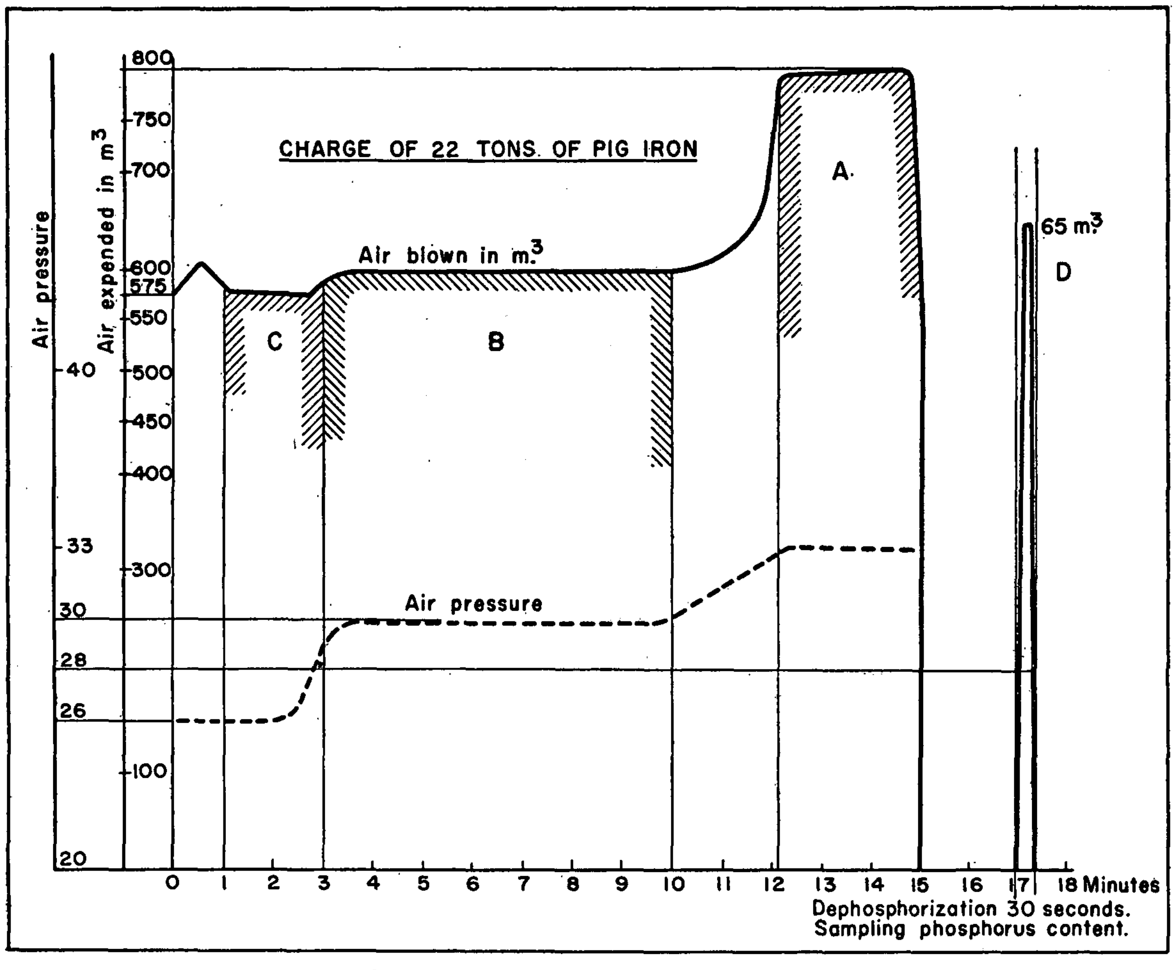


Figure 1. Air Pressure and Quantity Graph.

(ii) If the blow is commenced at low pressure, advantage is not taken of the possibility of blowing at high pressure during the elimination of the Si and Mn, with a lowering of pressure during the later projection period;

(iii) The quality of the refined metal improves in direct proportion to the speed of the blow. The manual operation, through over-caution or inertia on the part of the operator, is often too slow.

A complete study of a blow should preferably be made on the basis of the variation of quantities and not those of pressures.

4. *Precautions in teeming*

(a) *Ladle linings.* There was formerly a prejudice against sand linings on the grounds that they "dirtied" the steel. This, of course, is not the case, and sand linings are in extensive use in Europe today. They have the advantage of ease of repair, and of replacement of the ladle bottoms, which are normally reinforced with two courses of aluminous bricks set at right angles.

The most important points in the lining of ladles are:

(i) The natural sand must be kept in a dry place for at least three months before use;

(ii) The aged sand must be mixed in a crusher with 15% of clean sand from used linings;

(iii) It is usual to bind the lining with pattern-makers' nails;

(iv) The ramming must be done in thin layers, roughing after ramming to ensure penetration by the succeeding layer.

(b) *Plugs.* The porosity of the plug can be reduced by pressure impregnation with dolomite tar.

(c) *Centering of the steel stream.* It is essential to teem in the centre of the ingot mould. The life of the moulds can be prolonged by covering with a steel plate having a 200 mm. hole in the centre.

(d) *Speed of the steel flow from the ladle.* During the filling of the upper third of the ingot mould, pouring speed must be reduced.

(e) *Protection against sprinkling:*

(i) A layer of wood shavings to prevent metal projections on the walls of the ingot mould at the beginning of the pour;

(ii) Corrugated cardboard sheathing, with or without wood shavings. This is better than a metal tube, which may cause segregation.

(f) *Increase of efficiency in the rolling of unkilled Thomas steel:*

The use of heating powders to maintain the surface of the ingot in liquid form, permitting the easy emission of the gases.

The Moselle metallurgical laboratory has designed a chimney for use in the ingot moulds which (1) permits easy withdrawal of the gases; (2) produces ingot heads which can be easily rolled in the blooming mill; (3) increases the yield per ingot.

(h) *Care of ingot moulds.* Cleaning and blowing with compressed air after each pouring. Polishing with lacquers which leave a fine film of varnish on the interior of the moulds. Finally and most important, the observation of a heating and cooling cycle of the moulds.

Examples of Thomas steel products

1. *Thomas steel for rails*

(a) *Chemical properties of the pig iron*

Sulfur should be 0.3%. If it cannot be obtained in the blast furnace, the soda treatment applied at the outlet of the mixer will solve the problem.

Manganese should be 0.5 to 0.6%. At the end of the blow it should reach 0.2%, the balance being added during teeming.

Silicon should be 0.3 to 0.4% (maximum).

(b) *Physical properties of the pig iron*

Good fluidity, determined by testing a sample from the mixer spout.

(c) *Precautions during blowing*

Temperature indication can be obtained from visual observation of the fumes, and if the slag is too liquid, lime can be added. After completing the blow, deoxidize with silico-manganese in large lumps and two minutes later make the final addition of ferro-manganese. Scrape the slag carefully and do not allow it to pass into the ladle until the last moment. A period of five to seven minutes should be allowed to elapse between the addition of ferro-manganese and pouring. During pouring, ferro-silicon 75%, in small weighted packages, should be added at regular intervals to give a final silicon content of 0.09 to 0.12%. To meet French railway specifications, ferro-titanium is added in the ladle. Small pieces of aluminium are added during the filling of the moulds, which should be used at a temperature of 60 to 80°C. Moulds should be removed 25 minutes after pouring the last ingot.

2. *Thomas steel with high sulfur content for high speed cutting*

The technique is as follows: reduce the sulfur to 0.04%. Blow at normal speed to obtain P = 0.05%. Add ferro-manganese for deoxidation and wait two minutes. Add more to obtain final content of 0.1%. Sulfur in lumps or synthetic iron sulfide is added in the ladle. The most important point is pouring, where great care is necessary. It can be done either using the aforementioned chimney or reducing pouring speed.

3. *Extra mild effervescent Thomas steel for shaping and stamping*

The most important point is the distribution of the gas bubbles with relation to the wall of the ingot, and very careful attention must be paid to the manganese content.

4. *Hematite pig iron and malleable pig iron from Thomas pig iron*

The procedure is as follows:

(*a*) After careful removal of the sulfur content, the Thomas pig iron is transformed into low carbon steel (P = 0.040);

(*b*) In a second converter, which must never before have been used for the making of Thomas steel, so that the lining is quite free from Thomas slag, coke is brought to a white heat;

(*c*) The low carbon steel mentioned under (*a*) is poured into a ladle and the slag is carefully removed;

(*d*) This steel is then run into the second converter, where it is carburized in contact with the whitehot coke. The carburization is checked by sampling;

(*e*) Add 75% silicon in the ladle. Cast into sand moulds or pour into the mixer.

Use of oxygen

In recent years oxygen-enriched air, or mixtures of oxygen, steam and air have been used in Thomas steel making. Use of oxygen reduces the nitrogen content of the steel, permits shorter blowing times, reduces the minimum silicium content or, for even silicium contents, permits lower pig iron temperatures. It also allows larger percentages of scrap additions to the converter.

Experience so far shows that the optimum conditions are attained when air enriched to 30% oxygen is injected either in the converter during the blow, or at the surface of the metal in a ladle. Consumption of oxygen may be as much as 25 cu. m. per ton of pig iron.

As stated earlier, the basic lime/silica rate of 1.40 in the blast furnace charge results, with French ores, in the obtaining of a pig iron with 1.75% phosphorus, a relatively low silicon content (0.5%) and a high blast furnace temperature (1,370 to 1,420°C. at the tap). Lately, manganese additions to the charge have either been substantially reduced or have been suppressed altogether; furthermore, the possibility of desulfurizing the pig iron with the aid of soda ash provides the operator with the following alternatives:

(*a*) To continue with the same lime/silica ratio, and produce a new type of pig iron with a lower manganese content;

(*b*) To operate the blast furnace with an acid reaction, taking full advantage of the possibility of extracting the sulfur from the molten metal. This alternative presents the following advantages: (i) in France, where acid and basic iron ores exist, the proportion of the latter may be reduced; (ii) where no basic ores are available, as is the case in Colombia, the amount of lime in the charge can be reduced; (iii) consumption of coke is less, as the melting point of acid slags is lower than that of basic slags.

1. *Difficulties encountered in blowing acid Thomas acid pig iron*

In practice, it has been found that the acid operation of the blast furnace presents the following disadvantages: (i) with even temperature, the silicon content is higher than in basic operation; (ii) pig iron temperature is considerably lower; (iii) silicon and sulfur contents of the pig iron are much more difficult to regulate during operation.

These changes in the pig iron temperature and composition pose the following difficulties for the steel maker: (i) the high silicium content and the low temperature of the pig iron both make blowing difficult. There is a greater tendency for projections, and consequently pig iron consumption per ton of steel increases; (ii) lime consumption in the converter increases with the silicium content. Furthermore, dosage of lime is regulated by the colour of the fumes, and there is a tendency to underestimate the requirements of it. Handling of additional quantities of lime also increases the blowing time; (iii) there is greater wear on converter linings and bottoms; (iv) there is a tendency towards the formation of slag pockets, with additional loss of time for their removal.

In order to illustrate the difficulties, three examples taken from actual practice are here discussed:

(a) *Silicon* 1%. Although the soda treatment reduces the silicon content to 0.8%, blowing pressure must be reduced to the minimum before plugging the tuyeres. Even so, considerable projections are unavoidable during the first half of the decarburization period, and the weight of ferrous material per ton of steel is as high as 1,200 kg.

(b) *Silicon 0.4 to 0.6%, after soda treatment.* In these cases, the temperature usually remains high enough for easy blowing.

(c) *Silicon content 0.4% at the blast furnace taphole.* The sulfur is high and the temperature very low. Normally a double desulfurization would have to be done in the ladle, thus further reducing temperature and silicon content. The remedies are either to add ferro-silicon to the converter, or to resort to oxygen blowing, as will be seen later.

2. *Correction of faulty acid Thomas pig iron*

Within certain limits, the following steps can be taken to improve defective acid Thomas pig iron:

(*a*) The temperature of the pig iron can be increased before pouring into the converter. This can be accomplished by heating the mixer, in addition to taking all steps conducive to saving the heat contained in the pig iron upon leaving the blast furnace.

(*b*) The silicon content can be reduced. If the mixer can be heated, it can be used for a preliminary refining of the pig iron through the addition of various materials such as: iron ore, scale, lime and limestone, which would not affect the phosphorus content for the blow. An alternative is to operate the blower with two slags. The first blow, without addition of lime, should be stopped when the silicon content is sufficiently low. The disadvantage of the latter procedure is: (i) the increased wear on the lining during the first blow, and (ii) the loss of time involved.

(*c*) Blowing without prior reduction of silicon content or raise in temperature may take place. This can be achieved in two ways: (i) two blows are again used, the full amount of lime and one half of the pig iron is charged in the first and the blow proceeds until decarburization is almost finished. This procedure, like the previous one, results in longer operations. (ii) by blowing with mixtures of oxygen and steam, allowing easy operation of pig irons with up to 0.75% silicon. Its use depends on the cost of oxygen and the desirability of obtaining low nitrogen steel.

(*d*) There may be a combined reduction of silicon content and raise of temperature before pouring into the converter. Acid Thomas pig iron is usually chemically "hot" and its low tapping temperature makes it physically "cold". The first disadvantage can be used to compensate for the second by blowing, in the ladle, oxidizing gas on the metal, immediately under the slag, by means of injectors. Pure oxygen, enriched air or mixtures of oxygen and steam have been used as oxidizing agents.

Addition in the ladle has the following advantages: good control of the amount of oxygen, easier operation and considerable gain in temperature owing to the injection of nitrogen-free oxygen. Before applying the procedure in practice, several points had to be investigated, such as:

(i) Wear on the injectors. They are now made of good stainless steel tubes of 27 mm. external diameter and 3 mm. thickness. On an average, they wear off at a rate of 0.15 m. per cubic metre of oxygen injected.

(ii) Pressure of the oxidizing mixture. This usually ranges from 3 to 4 kg. per sq. cm.

(iii) The amount of oxygen to be used. This is normally 135 cu. m. for a 30-ton ladle.

(iv) Duration of the oxygen injection; this is usually 12 minutes.

(v) The influence of the treatment on the pig iron analysis. The analysis is affected as follows: (1) the silicon is reduced to 0.30%; (2) the manganese is reduced to 0.20%; (3) phosphorus and carbon are unaffected; (4) sulfur does not increase if the pig iron is covered with blast furnace slag but rises considerably if soda ash treatment slag is used; (5) the rise in temperature is generally about 20°C.

The benefits obtained from this process are: a reduction in the weight of ferrous material per ton of steel due to easier blowing, reduction in the amount of lime used in the converter, shorter blowing time and the possibility of increasing the scrap addition to the converter by 1%. Its main disadvantage is a reduction in the ratio of soluble to total phosphorus in the slag, the percentage usually dropping from 17 to 16.

3. *Preliminary reduction of silicon in basic Thomas pig iron*

Silicon content in the pig iron grows as the blast furnace is operated at higher temperature. It has been shown that basic blast furnace operation requires higher temperatures and, as the allowable limit for silicon content is small, injection of oxygen into the pig iron in a ladle, prior to its pouring into the converter may often be advantageous, even with basic slag operation. This may also be the case in Paz de Río. The relative advantages of the process, and its operation, are the same as in the case of acid Thomas pig iron.

The exposition given above leads to the conclusion that oxygen treatment is imperative when producing Thomas pig iron by the acid process.

4. *Oxygen blowing to reduce nitrogen content in the steel*

Thomas steel is often unfavourably compared with Martin steel because of its high nitrogen content—a good Thomas low carbon steel contains some 0.01%. To reduce this, oxygen must be added to the air when blowing, and this may be supplemented by the addition of iron ore or scale to the converter charge. Examination of results indicates that 30% oxygen, with a suitable amount of scale, will produce a Thomas steel with a nitrogen content similar to that of Martin steel. The phosphorus content after blowing with oxygen appears to bear no relation to the nitrogen content, and the residual manganese and iron contents are unaffected. The final temperature remains the same.

An air enrichment to 40% oxygen has been found to be excessive for a variety of reasons, mainly connected with the gases produced.

FUTURE OF THOMAS STEEL

It has been seen that the use of oxygen brings improvements not only in the quality of the metal, but also in the output of the mill.

The possibility of a change in the shape of the converter at some time in the future has often been discussed. The bottom certainly presents problems, and many methods of blowing a heat with a bottom of different design have been unsuccessfully attempted. However, it is hoped that improvements in the oxygen injection equipment and reduction in the cost of oxygen will one day allow the elimination of the classic bottom of the basic converter.

FURTHER REFINING OF THOMAS STEEL

1. *Thomas steel refined in an electric furnace*

(a) *Making of plain carbon steel*

After blowing, the following objectives are sought: a phosphorus content of 0.05 to 0.07%; partial deoxidization in the converter, using ferro-manganese; separation of the slag.

Before pouring into the electric furnace, lime is added to thicken the slag. The steel flows beneath the cold lime and the liquid slag flows only when the ladle is nearly full. The Thomas slag must not be allowed to enter the electric furnace. Anthracite is added in the runner to increase the carbon content to 0.15–0.25%. Current is applied to the electric furnace when it is about half full, in order to avoid solidification of the Thomas steel. Phosphorus is eliminated by a good boil through addition of lime and ore. The whole operation takes approximately two hours.

Before pouring the molten Thomas steel into an electric furnace of 26 tons capacity, 1,350 kg. of cold material are charged into it as follows: 54% lime lumps, 35% hematitic iron ore lumps, 8% carbonaceous iron ore powder, 3% dry silica sand. When the molten metal reaches this mixture at the bottom of the furnace, a considerable boil begins and care must be taken, by repeated stirring, to avoid adherence to the furnace walls. During this oxidizing period, part of the remaining phosphorus is removed from the steel.

(b) *High carbon steel for use in the automobile industry*

Final analysis: C—0.700; Mn—0.450/0.500;
P—0.035; S—0.010; SI—0.200

Complete withdrawal of slags, beginning 10 minutes and ending 30 minutes after connecting the electric power. Total time: 1 hour 35 minutes. Output: 16.5 tons per hour; power consumption: 180 kWh per ton.

(c) *Silico-manganese steel for use in the automobile industry*

Final analysis: C—0.450; Mn—0.600/0.650;
P—0.030; S—0.015; Si—1.900 to 2.100.

Complete withdrawal of slags, beginning 20 minutes and ending 30 minutes after connecting the electric power; Total time: 1 hour 45 minutes. Output: 14.8 tons per hour; power consumption: 150 kWh per ton.

(d) *Medium carbon steel for automobiles*

Final analysis: C—0.350; Mn—0.700/0.800;
P—0.030; S—0.010; Si—0.150.

Complete withdrawal of slags, beginning 15 minutes and ending 30 minutes after connecting the electric power. Total time: 1 hour 35 minutes. Output: 16.5 tons per hour; power consumption: 150 kWh per ton.

(e) *Medium carbon steel for forgings*

Final analysis: C—0.350; Mn—0.65/0.75;
P—0.035; S—0.020; Si—0.300.

Total time: 1 hour; output 26 tons per hour; power consumption 120 kWh per ton.

In all cases the time is taken from the commencement of the pour of the second half of the furnace, i.e., with 13 tons already poured, at which moment power is connected.

2. *Special steel made in the ladle*

After blowing the liquid Thomas pig iron until the phosphorus content is 0.07%, and partial removal of slags, cold lime is charged into the converter, followed by a short blow. The phosphorus content of steel thus treated in the converter may be as low as 0.025%. To manufacture higher carbon steel in the ladle, the following procedure may be followed:

(i) The empty ladle is heated to a high temperature: 982°C;

(ii) The oxidizing mixture, containing anthracite as well, is heated to 427°C. and charged into the ladle a few minutes before pouring.

The capacity of the ladle should be 50% higher than the amount of steel to be treated, because of the heavy boiling. It is most important to ensure that no Thomas slag reaches the ladle.

To obtain an average phosphorus content of 0.03% in best final analysis, Mr. Girod's process is only used in the basic Bessemer ladle. To obtain a phosphorus content not higher than 0.025%, it is better to use Mr. Girod's process, once in the basic Bessemer ladle, and twice in electric furnace.

There are several examples, as shown in the tables below.

Figure 2 compares the normal procedure with the two special Girod processes.

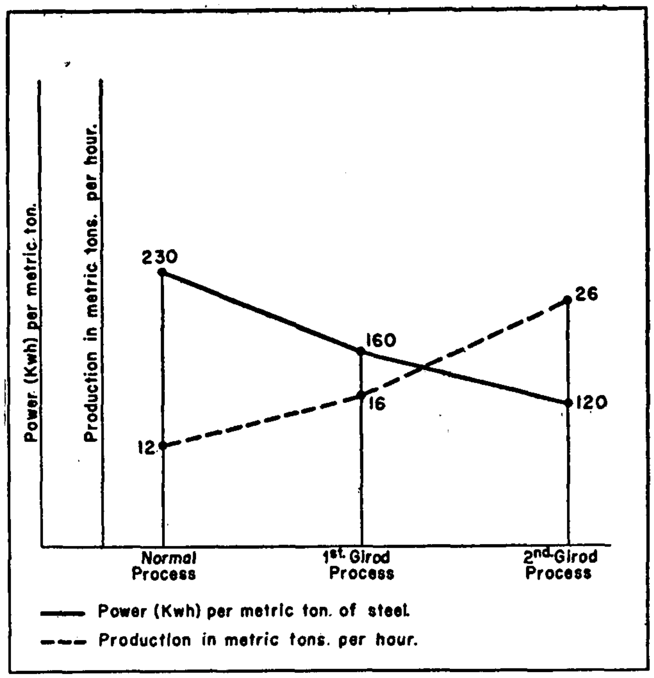


Figure 2.

Table 1

NICKEL-CHROMIUM CEMENTATION STEELS

Final analysis							*Time from electric power input until to top*	*Mechanical features after oil hardening at 1,562°F. without tempering*					*Basic Bessemer steel phosphorus content*	*Phosphorus content after reaction by Mr. Girod's double process*
C	*Mn*	*Si*	*S*	*Ph*	*Ni*	*Cr*		*R*	*E*	*A%*	*Resil. in length*	*Resil. across*		
0.110	0.360	0.280	0.016	0.020	2.61	0.62	1 h. 10 min.	108.5	96.3	13.2	21.7	9.28	0.043	0.015
0.115	0.370	0.260	0.015	0.021	2.62	0.74	0 h. 55 min.	112.8	98	10.5	18	6.29	0.046	0.014
0.110	0.360	0.260	0.017	0.019	2.60	0.68	1 h. 20 min.	110	82.7	13	18	6.29	0.051	0.009

Table 2

NICKEL-CHROMIUM STRUCTURAL STEELS

Final analysis							*Time from electric power input until to top*	*Mechanical features*					*Basic Bessemer steel phosphorus content*	*Phosphorus content after reaction by Mr. Girod's double process*
								after oil hardening at 1,562°F. and tempering to 482°F.			*after oil hardening at 1,562°F. and tempering to 1,202°F.*			
											Resilience			
C	*Mn*	*Si*	*S*	*Ph*	*Ni*	*Cr*		*R*	*E*	*A*	*length*	*across*		
0.350	0.400	0.220	0.015	0.023	2.87	0.69	1 h. 10 min.	168	154	7	16.3	6.58	0.043	0.017
0.310	0.390	0.220	0.604	0.024	2.89	0.65	1 h. 20 min.	172	158	8	15.2	5.46	0.060	0.019
0.350	0.370	0.260	0.015	0.018	2.75	0.84	1 h. 20 min.	173	159	8	12.4	4.25	0.063	0.015

Table 3

CHROMIUM—MOLYBDENUM HARD STEELS

Final analysis								*Time from electrical power input until to top*	*Mechanical features after oil hardening at 1,526°F. and tempering at 482°F.*					*Basic Bessemer steel phosphorus content*	*Phosphorus content after reaction by Mr. Girod's double process*
												Resilience			
C	*Mn*	*Si*	*S*	*P*	*Ni*	*Cr*	*Mo*		*R*	*E*	*A%*	*length*	*across*		
0.410	0.670	0.660	0.009	0.024	0.3	0.910	0.240	1 h. 20 min.	174	157	7	6.58	3.25	0.070	0.020
0.365	0.660	0.190	0.012	0.018	0.4	0.9	0.180	1 h. 15 min.	179	163	7	5.19	2.48	0.050	0.014

Table 4

NICKEL-CHROMIUM HARD STEELS

Final analysis							*Time from electric power input until to top*	*Mechanical features*					*Basic Bessemer steel phosphorus content*	*Phosphorus content after reaction by Mr. Girod's double process*
								after oil hardening at 1,526°F. and tempering at 1,202°F.						
											Resilience			
C	*Mb*	*Si*	*S*	*Ph*	*Ni*	*Cr*		*R*	*E*	*A%*	*in length*	*across*		
0.30	0.660	0.340	0.012	0.020	0.36	0.910	1 h. 10 min.	83	73.5	14.2	18	6.29	0.018	0.017
0.310	0.610	0.320	0.026	0.023	0.25	0.980	1 h. 15 min.	89.5	80.6	14	18	4.92	0.059	0.017

Summary of Discussion

The Acid Bessemer Process, *presented by the author*

Mr. COSTA LINO opened the discussion of the paper by questioning Mr. Philbrook on the type of blowing system used in the acid Bessemer process. He was particularly interested in ascertaining the composition of the blow and whether refrigeration was used.

Mr. PHILBROOK replied that the acid Bessemer process utilized normal or relatively low pressures, and neither heating nor refrigeration were necessary.

Mr. SANITER expressed special interest in the acid Bessemer process, inasmuch as he was associated with the only acid Bessemer plant in the United Kingdom. Mr. Philbrook had implied in his paper that the use of acid Bessemer steel for rails was limited. In the United Kingdom, however, 28% of the rails were rolled from acid Bessemer steel. He understood that the United States had, for some reason, stopped the use of that type of steel for rails, but had been unable to ascertain the precise reason.

With regard to duplexing, he stated that it seemed that Mr. Philbrook had in mind the process of blowing right down to low carbon. He was inclined to believe that that pattern was changing in favour of one which merely de-siliconized the iron. He wondered whether Mr. Philbrook had any comments to offer on the relative merits of those two methods of operation. A third point might, of course, arise to the effect that if de-siliconization were the objective, it might perhaps be best to de-siliconize in an active mixer.

Mr. PHILBROOK replied that he was aware that acid Bessemer steel was used quite extensively in Britain for making rails. One possible reason for that preference was that the phosphorus content was lower in British Bessemer steel. That might be one of the reasons why the United States railways had abandoned the use of that type of steel. Economic reasons might also govern the practices adopted in the United States, where open hearth facilities were so unlimited that it was possible to make rails more advantageously by that process. The combination of a high phosphorus and a high nitrogen content created some difficulty in impact strength and that factor would be dangerous for high-speed trains.

As to the duplex process, it was not possible to give a full treatment of the subject. One company in the United States practised high blowing, using high-blown metal and steel scrap. One of the difficulties there was that the carbon content could not be controlled and that the silicon deposits were inclined to build up. He was of the opinion that it was advisable to alternate high and low blowing. The charge should consist of basic iron, as high manganese iron did not blow very well.

On the question of active mixers, he had no experience whatsoever and preferred not to comment on the subject.

Mr. ALLARD, at that stage, proposed that discussion of Mr. Philbrook's paper should be postponed until the afternoon's meeting, since other papers dealing with the same or allied subjects were to be presented that day.

There were, however, one or two points that he wished to raise dealing specifically with the paper under review. First of all, as to the duplex process involving acid Bessemer blowing, he understood from Mr. Philbrook's paper that the duration of open hearth refining was about five hours for a 180- to 200-ton heat, and that the duplex process therefore did not allow a saving of time in relation to straight open hearth practice.

He also raised the question of the manufacture of high carbon steel by the Bessemer process. In utilizing the basic Bessemer process it was necessary to pass through mild steel and add carbon in the ladle or the converter. Concerning the nitrogen content of the steel when using the direct basic Bessemer process, it was his opinion that it was unlikely that nitrogen would be high if the steel were oxidized and consequently the nitrogen in the steel was a negligible problem in making rails by the basic Bessemer process in Europe.

The author had mentioned that tests had been conducted in the acid Bessemer vessel by side blowing, but there was no indication as to the yield by that process. He was inclined to think that there would be a considerable loss of iron in the side blowing process.

He was of the opinion that, for Latin America, consideration should be given to all the possibilities of the basic lining, even when the phosphorus content of the pig iron was low.

Table 3

ELEMENTARY FLOAT-AND-SINK DATA OF SOME CAUCA VALLEY HIGH-VOLATILE COAL SAMPLES

(Values in per cent on moisture-free basis)

Size 37 — 10 mm.

	Sample 2A			*Sample 2B*			*Sample 3A*			*Sample 4A*		
Specific-gravity increment	*Weight*	*Ash*	*Sulfur*	*Weight*	*Ash*	*Sulfur*	*Weight*	*Ash*	*Sulfur*	*Weight*	*Ash*	*Sulfur*
Float – 1.30	65.30	6.36	0.6	81.93	6.56	0.6	52.23	5.23	0.6	43.00	5.67	1.64
1.30 to 1.40	18.69	14.76	0.6	14.16	12.51	0.6	13.73	14.62	0.6	33.49	13.18	2.83
1.40 to 1.50	4.07	25.16	0.6	1.66	24.99	0.6	15.30	26.97	0.6	9.48	22.54	4.70
1.50 to 1.60	1.95	33.70	0.6	.72	32.45	0.6	10.62	35.28	0.6	4.35	30.00	6.76
1.60 to 1.80	3.98	46.76	0.6	.38	45.88	0.6	5.53	44.16	0.6	5.26	43.73	5.01
Sink – 1.80	6.01	67.02	0.6	1.15	62.78	0.6	2.59	68.55	0.6	4.42	58.60	10.71
Total	100.00	17.33	1.2	100.00	7.87	0.7	100.00	16.69	0.5	100.00	15.18	3.13

	Sample 5A			*Sample 6A*			*Sample 7A*		
Specific-gravity increment	*Weight*	*Ash*	*Sulfur*	*Weight*	*Ash*	*Sulfur*	*Weight*	*Ash*	*Sulfur*
Float – 1.30	82.43	5.56	1.73	70.33	5.94	2.45	50.01	5.50	1.73
1.30 to 1.40	14.16	12.51	2.97	14.54	13.54	4.37	24.75	12.98	2.92
1.40 to 1.50	1.66	24.98	3.97	4.65	24.71	4.72	11.24	23.66	3.97
1.50 to 1.60	.72	32.45	4.36	2.67	33.19	5.18	5.28	31.73	4.36
1.60 to 1.80	.38	45.88	6.87	1.87	43.70	4.71	3.86	40.71	6.87
Sink – 1.80	.65	62.78	13.76	5.94	70.24	3.93	4.86	63.14	13.75
Total	100.00	7.8	2.8	100.00	13.17	3.04	100.00	16.94	3.29

Table 4

ELEMENTARY FLOAT-AND-SINK DATA OF SOME CAUCA VALLEY HIGH-VOLATILE COAL SAMPLES

(Values in per cent on moisture-free basis)

Size 10 — 0.15 mm.

	Sample 2A			*Sample 2B*			*Sample 3A*			*Sample 4A*		
Specific-gravity increment	*Weight*	*Ash*	*Sulfur*	*Weight*	*Ash*	*Sulfur*	*Weight*	*Ash*	*Sulfur*	*Weight*	*Ash*	*Sulfur*
Float – 1.30	62.57	5.42	0.6	81.25	5.26	0.6	58.40	4.67	0.6	53.34	5.61	1.54
1.30 to 1.40	13.94	13.64	0.6	12.13	11.96	0.6	11.41	13.28	0.6	30.44	11.52	2.19
1.40 to 1.50	4.19	24.64	0.6	2.01	22.99	0.6	8.59	24.36	0.6	8.15	20.81	3.21
1.50 to 1.60	2.54	33.05	0.6	.91	32.67	0.6	7.87	34.07	0.6	3.01	30.59	4.06
1.60 to 1.80	4.86	43.78	0.6	.99	43.44	0.6	7.41	44.90	0.6	2.62	41.32	4.85
Sink – 1.80	11.90	67.53	0.6	2.71	56.40	0.6	6.32	67.42	0.6	2.44	56.34	10.68
Total	100.00	17.33	1.2	100.00	8.71	0.7	100.00	16.60	0.5	100.00	11.57	2.26

	Sample 5A			*Sample 6A*			*Sample 7A*		
Specific-gravity increment	*Weight*	*Ash*	*Sulfur*	*Weight*	*Ash*	*Sulfur*	*Weight*	*Ash*	*Sulfur*
Float – 1.30	81.75	5.26	1.63	68.08	5.78	1.59	54.75	4.73	1.63
1.30 to 1.40	12.13	11.96	2.66	13.54	11.58	2.84	17.79	12.36	2.66
1.40 to 1.50	2.01	22.99	3.33	4.08	23.17	4.07	8.93	21.94	3.33
1.50 to 1.60	.91	32.67	3.77	2.51	31.80	3.88	5.43	30.98	3.77
1.60 to 1.80	.99	43.44	3.78	3.30	43.17	3.42	5.85	41.80	3.79
Sink – 1.80	2.21	66.40	6.05	8.49	68.51	2.87	7.25	61.44	6.05
Total	100.00	7.8	2.8	100.00	14.49	2.09	100.00	15.33	2.53

To facilitate study of the technical difficulties of the job, the washability characteristics of a mixture of these samples has been established by a statistical combination of the individual sample data. The composite data are given in the two charts, Figures 5 and 6. These charts may then be taken to represent the average blended feed to the washery, assuming that the coal will come from the several mines in about the proportions being currently mined at these operations.

First, an estimate has been made of the performance of a Baum-jig washer operating on this blended feed and handling in the primary step the unsized plant-feed represented by the washability charts. Not having the washability data broken down by sizes, it has been assumed, in order to make this estimate possible, that the coarser sizes have the specific-gravity content shown by the data of Figure 5, and that the smaller sizes ranging below 10 mm. have the same specific-gravity content as shown in Figure 6. Although this is not strictly true, the assumption does not materially alter the hypothetical problem that we are examining for illustrative purposes to represent a typical washing situation.

Table 5

SIZE-CONTENT OF THE COMPOSITE OF SEVEN SAMPLES REPRESENTED BY TABLE 2

	Per cent of total by weight	
Size—millimetres	*Direct*	*Cumulative*
37 – 18	26.0	26.0
18 – 10	19.6	45.6
10 – 5	19.4	65.0
5 – 2.5	11.5	76.5
2.5 – 1.5	8.9	85.4
1.5 – 0.8	3.1	88.5
0.8 – 0.3	8.4	96.9
	3.1	100.0
Total	100.0	

Other advantages of the basic Bessemer process were the possibility of using oxygen-enriched blast and of performing desulfurization in the vessel.

The CHAIRMAN supported Mr. Allard's suggestion regarding postponement of the discussion of papers dealing with the Bessemer process until all the pertinent documents had been presented. He therefore called upon Messrs. BUELL, FITTERER and DECHERF to present their papers, entitled respectively:

Basic Open-Hearth Steelmaking Practice in the United States of America

Economics of the Modern Acid Open Hearth Practice

Manufacture of Thomas (Basic Converter) Steel at Paz de Río, Colombia

The CHAIRMAN announced after the presentation of the papers mentioned, that the afternoon's discussion would be divided into two parts; the first would cover the acid open hearth process, and the second would attempt to compare the Thomas with the acid Bessemer process.

STEEL MAKING AND FINISHING

Comparison of economics of different processes of steelmaking (*continued*)

23 October 1952—Afternoon

Chairman:

Joaquín PRIETO, Assistant Manager, Empresa Siderúrgica Nacional de Paz de Río, S.A., Bogotá, Colombia

Contributed Papers:

Acid Open Hearth Products and Their Specifications
G. R. FITTERER, Dean, Schools of Engineering and Mines, University of Pittsburgh, Pittsburgh, Pa., and Director of Research, Acid Open Hearth Research Association Inc., Pittsburgh, Pa., U.S.A.
Use of Phosphorus Ores for the Production of Pig Iron for Transforming into Steel by the Thomas Process, Società ILVA, Bagnoli, Italy
Perrin Process for Converter Steelmaking
Marc ALLARD, Directeur adjoint de l'Institut de recherches de la sidérurgie, Saint-Germain-en-Laye, France
Thomas Steel with Low Nitrogen and Phosphorus Contents
P. COHEUR, Director, Centre national de recherches métallurgiques, Liège, Belgium

Summary of Discussion:

Participants: Messrs. FITTERER, MERCIER, KALLING, CANGUILHEM, CHÁVEZ, BUELL, SANITER, PHILBROOK, COHEUR, COSTA LINO, ALLARD, GONZÁLEZ VARGAS

Acid Open Hearth Products and Their Specifications

G. R. FITTERER

When the manufacturers of acid open hearth steel are asked why they continue to use the process, some very interesting and direct answers are received. Some of the reasons are:

1. Economy and speed of operation;
2. Greater freedom from flakes;
3. Greater metal fluidity;
4. Consistent ease of refining within chemical specifications;
5. High percentage of recovery of alloy additions;
6. Ability to hold a heat without change in analysis.

Items 1, 4 and 6 were discussed at some length in another paper on the "Economics of the Modern Acid Open Hearth Process". However, it should be repeated here that the acid open hearth process has sufficient advantages as shown above that it will probably be continued indefinitely. In fact, its use should be reconsidered by many who generally think only in terms of the basic process. As will be shown in this paper, the process has sufficient advantages from the standpoints both of economics and with respect to steel quality that certain products are best made by this process. Although it will never replace the basic method for the tonnage steels unless some ingenious method for phosphorus and sulfur removal is devised, there are some special products and much bar stock which could be more readily produced by this method.

1. ACID OPEN HEARTH STEEL MANUFACTURERS

The variety of products made by the acid process is revealed immediately by the names of the producing companies which are listed in the following table.

Table 1

ACID OPEN HEARTH STEEL MANUFACTURERS IN THE UNITED STATES

Atlantic Steel Castings Company, Chester, Pennsylvania
Continental Foundry and Machine Company, Chicago, Illinois
Commercial Steel Casting Company, Marion, Ohio
Erie Forge Company, Erie, Pennsylvania
Heppenstall Company, Pittsburgh, Pennsylvania
Mackintosh-Hemphill Company, Pittsburgh, Pennsylvania
Mesta Machine Company, Pittsburgh, Pennsylvania
The Midvale Company, Philadelphia, Pennsylvania
Ohio Steel Foundry Company, Lima, Ohio
Railway Steel Spring Division, American Locomotive Company, Latrobe, Pennsylvania
John A. Roebling's Sons Company, Trenton, New Jersey
Standard Steel Works Division, Baldwin-Lima-Hamilton Corporation, Burnham, Pennsylvania
United Engineering and Foundry Company, Pittsburgh, Pennsylvania
United States Steel Company, Lorain Works, Johnstown, Pennsylvania (Also at Gary, Indiana)

All the above companies are members of the Acid Open Hearth Research Association, Inc. and are responsible for some 85% of all acid open hearth steel made in the United States today.

As will be indicated in the following table, the products of the acid open hearth vary considerably with respect to their chemical analysis, their physical properties and their uses.

2. VARIETY OF PRODUCTS MADE BY THE ACID OPEN HEARTH PROCESS

The tonnage of steel made by the acid open hearth process is equally divided between forged products and castings. In general the forgings made by this procedure are very large and are used for dynamic applications. This indicates that the acid open hearth is able to produce steel of high quality because it is well known that the larger the section, the greater the difficulty in producing quality steels. A large section when heat-treated has such differentials in temperature from outside to centre, that corresponding differences in stresses occur. Often the outside of a large section is under a very high tensile stress, while at the same time the centre of the section is under compressive stresses. Thus, any defects such as excessive segregation, ingotism, or flakes will cause cracking and rejection of the part. Manufacturers and customers alike regard the acid open hearth as a good source of such material free from such defects.

A similar situation exists in the production of castings. The acid open hearth furnace itself has many advantages in the casting shop as will be shown later in this paper. The process is responsible for most of the very large castings made in the United States today. In addition to the above, it will be shown at the end of this paper that in view of recent changes in acid open hearth technology, there are many products such as rails and miscellaneous bar stock which might be made economically by this process. The numerous items made by the acid open hearth process are listed in Table 2.

Table 2

PRODUCTS OF THE ACID OPEN HEARTH FURNACE

Forged and rolled products

High quality wire for bridges
Die blocks
Locomotive, car wheels, and tires
Rams and anvil blocks and bases for steam and drop hammers
Large marine crankshafts, etc.
Diesel locomotive crankshafts
Reduction gear rims
Gear blanks
Rolled steel rings and flanges
Pinion and turbine shafts
Grinding rings for cement mill pulverizers
Hydraulic press forged columns and cylinders
Miscellaneous forgings, shafts, etc.

Castings

Cast steel cinder or slag pots
Housings for ore crushing machinery
Pressure vessels for high pressure steam, the oil and chemical industries
Valves and fittings for high pressure service
Crane wheels
Hydroelectric power castings
Housings for rolling mills
Cylinders and glands for hydraulic presses
Rolls for steel mills (back up rolls), rubber mills, pipe mill rolls (lap, butt and electric), and printing presses
Parts for ingot mould cars
Open hearth charging boxes
Blast furnace bells
Side frames and bolsters
Miscellaneous castings

In addition to the above, those familiar with steel mill equipment will recognize, among the list of members of the Acid Open Hearth Research Association, most of the steel mill equipment manufacturers. These companies are responsible for most of the rolling mills (continuous and reversing), large forging presses and hammers, large screw feed lathes for rolls and shafting, straightening machines, etc., which are produced in North America today. Their equipment is to be found in every steel producing country and the quality of their products is well known.

It would be essentially impossible to estimate the relative tonnage of steel constituting each of the respective products in Table 2. However, the percentage of the above companies producing some of these items has been estimated for the purposes of this paper. This is shown in Table 3 where it may be seen that 70% of the acid open hearth plants in the United States produce some B-grade castings, etc. Also given in the table are the approximate chemical analyses of these materials.

Table 3

PARTIAL RANGE OF ANALYSES OF ACID OPEN HEARTH PRODUCTS

Description	*Companies producing per cent*	*Analyses, per cent*						*Remarks*
		C	*Mn*	*P*	*S*	*Si*	*Alloy*	
Grade B castings	70	0.15 to 0.45	0.85 max.	0.06 max.	0.06 max.	[a]		
Rolling mill housings	65	0.30 to 0.40	0.60 to 0.90	0.04 max.	0.05 max.			1030, 1040[b]
Pinions and gears	55	0.38 to 0.43	1.75 to 1.00	0.04 max.	0.04 max.		4140	Variable
Steel rolls	50	0.40 to 2.60	1.25 max.	0.06 max.	0.06 max.		Cr, Ni, Mo	1.50 max. 0.60 max
Press columns, cylinders	45							Variable
Crank and turbine shafts	40						Ni, Cr, Mo	Variable
Pressure vessels	35						Ni, Cr, Mo	Variable
Valves and fittings	30	0.15 to 0.45	0.50 max.	0.05 max.	0.06 max.	0.20 min.		
Hammer rams and anvil bases	20							Variable
Forged, rolled wheels and tires	20						1040	Variable
Heavy machine tools	15							Variable
High quality wire	7	0.85 max.	0.40 to 0.60	0.040 max.	0.040 max.	0.15 to 0.30		
Die blocks	7	0.50 to 0.60	0.65 to 0.95	0.040 max.	0.050 max.		Cr, Mo, Va Ni, Cr, Mo	

[a] If no value is given or indicated by code, the analysis is not specified or limited.
[b] AISI or SAE specifications.

3. FORGED AND ROLLED PRODUCTS

Wire

The parallel suspension cables for the Ramfis Bridge in the Dominican Republic were constructed of strands of wire produced by the acid open hearth furnace. This illustration is given to indicate that bar stock from the acid open hearth is sometimes rolled and drawn into high quality wire. As shown in Table 3 this material is high carbon but non-alloy. It is deoxidized with aluminium for grain size control and treated to about 230,000 lb. per sq. in. tensile strength and 160,000 lb. per sq. in. yield point. Besides visual inspection and surface treatment (chipping, scarfing, etc.), this material must withstand a bend test after galvanizing. Otherwise, the material is purchased as acid open hearth steel because of confidence in past performance over many years of experience.

Large forgings

Many large items are produced by the acid open hearth method, for instance a 57,000 lb. finished eccentric crankshaft for the world's largest forging press was made of acid open hearth forged steel. The press is used to manufacture automobile crankshafts at a very rapid rate and the stresses to which the eccentric is subjected are very high and rapidly repeated. The original ingot from which this shaft was made weighed 112,000 lb. and the shaft was produced after much "upset" forging and careful heat-treatment. The steel was a 4340 grade.

Turbine shafts

High velocity turbine shafts are some of the other forgings usually made of acid open hearth nickel molybdenum steel. Such parts operate under severe dynamic conditions as well as at high temperatures. The service requirements are rigid and difficult to meet.

Large gears

Large gears such as rolling mill dinions are obviously component parts of equipment which in turn are purchased on a quality of material and workmanship basis. However, it is well known that these machines which are products of the acid open hearth have good performance records.

Large crankshafts

Crankshafts designed for use in a blooming mill must, as is well known, withstand severe service. Diesel locomotive crankshafts are constructed by twisting a forged shaft to the angle of each of the six "throws". Any internal defect is accentuated by such treatment. The shafts must undergo individual Zyglo examination particularly at the fillets. These shafts are made from standard 4140 grade of steel.

Die blocks

Many die blocks of all sizes and description are made from acid open hearth steel. Generally, the steel is of a modified 4150 grade and the blocks are prepared at various hardness levels depending upon the ultimate use. Nearly all such blocks are supersonically tested. The current trend is towards blocks of larger sections.

Discs, wheels and tires

These products may be either forged or rolled. Rigid specification of the physical properties are imposed. The increased speeds and greater loads carried by our locomotives today indicate the confidence placed in acid open hearth steel.

Specifications for forged and rolled products

The variety of analyses and ultimate uses of acid open hearth forged and rolled products are so numerous that it would be difficult to list them all. However, a large portion of the tonnage of forgings is made from plain carbon steel containing less than 0.60% carbon. Generally, chemical specifications follow the American Iron and Steel Institute or the Society for Automotive Engineers code. However, it has been recognized by customers and manufacturers alike that these two classifications are not entirely satisfactory for forgings.

Consequently, an additional system of classification (1) was established by the Forging Manufacturers Association, the specifications for which are followed by the industry. The FMA system classifies the plain carbon grades of steel in a similar manner to the AISI and SAE systems but signifies forging grades with the insertion of the letter F. For example, the AISI 1040 grade is signified as the 10F40 grade by the forging industry. A similar system is used for the alloy grades.

The FMA system was established to distinguish forging grades from small diameter bar stock. The large sizes of most of the forgings have required this distinction because of the tendency for segregation and the difficulty in sampling large sections. The AISI (or SAE) system is satisfactory for sections not exceeding 200 sq. in. or 18 in. in width or for those weighing less than 10,000 lb. Above these sizes, the FMA specifications are used.

As shown in Table 4 certain limits and ranges of permissible variations have been established which are not listed by the AISI (or SAE) method. The table is only a partial list but is given here to illustrate the FMA specification. For example, in a 10F40 steel, the carbon content may vary within an 0.11% carbon range (i.e., +0.06 and −0.05%). Similarly, if the manganese were specified at 0.50%, it is permitted to vary from 0.40 to 0.60% or within ±0.10% range.

Table 4

FORGING MANUFACTURER'S SPECIFICATION(1)
(PARTIAL LIST)

Element	*Specified analysis exists between ranges, per cent*	*Total Permissible variation from specified value, per cent*
Carbon..........	To 0.25 inclusive	0.09
	0.25 to 0.40 inclusive	0.10
	0.40 to 0.55 inclusive	0.11
	0.55 to 0.80 inclusive	0.14
Manganese.......	To 0.50 inclusive	0.20
	0.50 to 1.65 inclusive	0.30
Phosphorus......	Basic open hearth steel	0.040 max.
	Acid open hearth steel	0.050 max.
	Basic electric steel	0.025 max.
Sulfur...........	Basic open hearth	0.05 max.
	Acid open hearth	0.05 max.
	Basic electric	0.025 max.
Silicon...........	To 0.35 inclusive	0.15
	0.35 to 1.00 inclusive	0.20
Nickel...........	To 0.50 inclusive	0.20
	0.50 to 1.50	0.30
	1.50 to 2.00	0.35
Chromium.......	To 0.40 inclusive	0.15
	0.40 to 0.80 inclusive	0.20
	0.80 to 1.05 inclusive	0.25
	1.05 to 1.25 inclusive	0.30
	1.25 to 1.75 inclusive	0.50
Molybdenum.....	To 0.10 inclusive	0.15
	0.40 to 0.80 inclusive	0.20

The phosphorus is set at 0.04% maximum for basic open hearth steel and 0.05% maximum in acid open hearth steel. It is important to note that this difference not only recognizes the difficulty of removing phosphorus in the acid furnace, but it also signifies that the quality of the acid steel is not impared with a higher phosphorus content than the limit placed on the basic furnace.

The sulfur content is specified at 0.050% in forgings made by both the acid and basic open hearths. The phosphorus and sulfur limit imposed on basic steel is 0.025% maximum in each case.

The physical specifications under which the forged and rolled products of the acid open hearth are purchased also vary appreciably. Many products are sold essentially on past and/or guaranteed service performance. However, in some of the forgings which are used under severe dynamic stresses, no chances are taken and considerable inspection of the product is usually made by the manufacturer and customer alike. These parts are given a detailed visual survey of the machined surface particularly at the sides of flanges and fillets where excessive ingotism and other internal defects are most likely to be revealed. Also, cross-sectional etch tests are made to inspect for cracks, flakes, excessive "porosity", etc. Sometimes such forgings are subjected to Magnaflux, Magna-glo and/or Zyglo examination. The latter is particularly good in revealing excessive non-metallic inclusions on the surfaces near the fillets.

More important than these, however, is the trend to examine all large forgings with the supersonic reflectoscope which will non-destructively reveal any internal burst or even minute defects in the entire length. The location of a defect can be determined from three dimensions and the possibility of its being detrimental to performance may be estimated. Besides supersonic tests, transverse and longitudinal physical properties are usually required.

4. CASTINGS

Types

As was shown in Table 3, castings constitute a large portion of the tonnage produced by acid open hearth furnaces. The variety of types of castings as well as analyses is considerable. Miscellaneous castings are made in great quantities by some companies. The size of a single acid open hearth casting may also reach several hundred thousand pounds as in blooming mill housings.

The casting may be intricate as in the case of the head for river bottom dredges, or it might be required to withstand very high pressures such as in large pressure vessels. Other large production items of the acid open hearth are steel rolls, rolling mills, and forging presses.

The wide composition range of castings may become evident in Table 3 by a comparison of the analysis of the miscellaneous castings with that of the "semi-steel" rolls made by the acid open hearth. The carbon content of the latter may vary from about 0.40 to 2.60%. In addition to plain carbon steels, alloy steel castings are also made. Rolls are examples of the alloy "semi-steels" made by the process.

The acid process is best for castings

It is shown herein that the acid open hearth has many advantages in a casting shop. The charge weight for a given furnace is often varied over a wide range as needed. As was stated in another paper at this Conference, a heat of acid open hearth steel may be either required to refine quickly or to be held for some time in the furnace without an appreciable change in analysis. This factor makes the process desirable in a casting shop because the production rate of the furnace may be adjusted to coincide with that of the moulding shop. If the moulding shop is not ready at the same time that the heat should be tapped, the acid open hearth may be held until the moulders are ready. Also, in a plant making very large castings such as rolls or housings, it is often the practice to schedule two furnaces to tap simultaneously so that the metal in both may be poured into the same mold. In such cases, it is usually necessary to retain the liquid metal in one furnace while the contents of the other are being transferred to the mould. Fortunately, this can be done without any appreciable change in composition. This coupled with the belief on the part of many operators that acid steel is more fluid at the same temperature makes the process quite suitable for casting manufacture. In fact, the acid open hearth and acid electric furnaces are responsible for the bulk of the castings made in the United States today.

Specifications for castings

For many years the chemical specifications (2) for castings were divided into two major classifications for the bulk of the production. Class A was specified as a steel having a maximum carbon content of 0.45%. No manganese, sulfur or silicon were specified. However, the steel made in the acid furnace was limited to 0.07% phosphorus, whereas the basic furnace was restricted to 0.06% maximum phosphorus. No physical specifications or heat-treatment were required for Class A castings.

The Grade B castings at that time were heat-treated to hard, medium and soft tempers, but the only chemical specifications were with respect to phosphorus and sulfur. The acid open hearth was permitted up to 0.060% of phosphorus as compared with a limit of 0.050% for basic open hearth steel. The sulfur content was 0.060% in both cases. The tensile strength of the hard, medium and soft B grade castings were 80,000, 70,000 and 60,000 psi., respectively. The specified yield point was 45,000 psi. for all tempers. The per cent elongation requirements were 17, 20 and 24%, respectively, and the minimum reductions of area were 25, 30, and 35%, respectively. Although these particular specifications were discarded some time ago by the American Society for Testing Materials, (2) there is still some steel being purchased on this basis in the United States today.

In recent years the specifications for steel castings have become somewhat more rigid. However, so many different agencies have established specifications that considerable confusion has developed. Because of this the Steel Founders' Society of America has recently published a summary (3) of all of the existing and recognized specifications which are currently being used for the purchase of steel castings. The specification systems in this excellent summary are given in Table 5 together with the ultimate use of the castings. This list has been made available for the purpose of this paper by the Steel Founders' Society.

Table 5

SUMMARY OF STANDARD SPECIFICATIONS FOR STEEL CASTINGS

Prepared by the Specifications Committee, Steel Founders' Society of America — Revised 1 April 1952

A.S.T.M.	A 27-50 T Mild to Medium-Strength Carbon-Steel Castings for General Application (Tentative).
A.S.T.M.	A 148-50 T High-Strength Steel Castings for Structural Purposes (Tentative).
A.S.T.M.	A 95-44 Carbon-Steel Castings for Valves, Flanges and Fittings for High-Temperature Service.
A.S.T.M.	A 216-47 T Carbon Steel Castings Suitable for Fusion Welding for Service up to Temperatures of 750°F.
A.S.T.M.	A 217-49 T Alloy Steel Castings for Pressure Containing Parts Suitable for High Temperature Service.
A.S.T.M.	A 352-52 T Ferritic Steel Castings for Pressure Containing Parts Suitable for Low Temperature Service.
S.A.E.	1946 Automotive Steel Castings (Tentative).
A.A.R.	M 201-47 Steel Castings.
A.R.E.A.	Specifications for Steel Railway Bridges 1949: Steel Castings.
FEDERAL	QQ-S-681b (11-6-1941) Steel Castings (Amended 8-24-1943).
MILITARY	MIL-S-15083 (Ships) April 1950 Steel; Castings. Superseding 49S1 (Ships). (Amended 11-1-1950).
MILITARY	MIL-S-870 (Ships) September 1949 Steel, Alloy, Molybdenum; Castings, Superseding 46S33 (INT).
MILITARY	MIL-S-15464 (Ships) December 1950 Steel, Alloy, Chromium-Molybdenum; Castings. American Bureau of Shipping Steel Castings 1952 Rules Edition—Machinery and Hull Castings. Lloyd's Register of Shipping 1951—Steel Castings.

(*Table 5 continues on next page.*)

Specification and heat treatment			*Mechanical properties—minimum*				
Specification	*Class*	(#) *Heat treatment*	*Tensile strength p.s.i.*	*Yield toint p.s.i.*	*Elong. in 2" %*	*Red. of area %*	*Other tests: bend, impact hardness*†
ASTM	N-1	—	—	—	—	—	—
A 27-50T	N-2	A or N or NT or QT	—	—	—	—	—
	N-3	A or N or NT or QT	—	—	—	—	—
	U-60-30	—	60,000	30,000	22	30	—
	60-30	A or N or NT or QT	60,000	30,000	24	35	—
	65-30	A or N or NT or QT	65,000	30,000	20	30	—
	65-35	A or N or NT or QT	65,000	35,000 (a)	24	35	—
	70-36	A or N or NT or QT	70,000	36,000	22	30	—
ASTM	80-40	A or N or NT or QT	80,000	40,000	18	30	—
A 148-50T	80-50	A or N or NT or QT	80,000	50,000	22	35	—
	90-60	A or N or NT or QT	90,000	60,000	20	40	—
	105-85	A or N or NT or QT	105,000	85,000	17	35	—
	120-95	A or N or NT or QT	120,000	95,000	14	30	—
	150-125	A or N or NT or QT	150,000	125,000	9	22	—
	175-145	A or N or NT or QT	175,000	145,000	6	12	—
ASTM							Bend—degrees
A 95-44	—	A or N or NT	70,000	36,000	22	30	[90]
ASTM	WCA	A or NT	60,000	30,000	24	35	[90]
A 216-47T	WCB	A or NT	70,000	36,000	22	35	[90]
ASTM	WC1	A or NT	65,000	35,000	24	35	[90]
A217-49T	WC4	A or NT	70,000	40,000	20	35	[90]
	WC5	A or NT	70,000	40,000	20	35	[90]
	WC6	A or NT	70,000	40,000	20	35	[90]
	WC9	A or NT	70,000	40,000	20	35	[90]
	C5	A or NT	90,000	60,000	18	35	[90]
	C12	A or NT	90,000	60,000	18	35	[90]
							Impact—ft.—lbs.
ASTM	LCB	N or NT or QT	65,000	35,000	24	35	15 (g)
A 352-52T	LC1	N or NT or QT	65,000	35,000	24	35	15 (g)
	LC2	N or NT or QT	65,000	40,000	24	35	15 (g)
	LC3	N or NT or QT	65,000	40,000	24	35	15 (g)
SAE							BHN
Automotive	0022	A or N or NT	—	—	—	—	—
	0030	A or N or NT or QT	65,000	35,000	24	35	131
	0050	A or N or NT	85,000	45,000	16	24	170
	0050	QT	100,000	70,000	10	15	207
	080	A or N or NT or QT	80,000	40,000	18	30	163
	090	NT or QT	90,000	60,000	20	40	187
	0105*	QT	105,000	85,000	17	35	217
	0120*	QT	120,000	100,000	14	30	248
	0150*	QT	150,000	125,000	9	22	311
	0175*	QT	175,000	145,000	6	12	363
AAP	A	Unannealed	60,000	30,000	22	30	—
M 201-47	A	A or N	60,000	30,000	26	38	—
	B	A or N	70,000	38,000	24	36	—
	C	NT or QT	90,000	60,000	22	45	—
	D	QT	105,000	85,000	17	35	—
	E	QT	120,000	100,000	14	30	—
AREA	—	A	65,000	33,000	24	35	—
FEDERAL							Bend—degrees
QQ-S-681b	X	—	—	—	—	—	—
	O	A	—	—	—	—	—
	1	A	60,000	30,000	24	35	120
	2	A or N or NT	65,000	35,000	20	30	90
	3	A	80,000	40,000	17	25	—
	4A1	A	75,000	40,000	24	35	—
	4A2	A	85,000	53,000	22	35	—
	4B1	N or NT	85,000	55,000	22	40	—
	4B2	N or NT	90,000	60,000	22	45	—
	4B3	N or NT	100,000	65,000	17	30	—
	4C1	QT	90,000	65,000	20	45	—
	4C2	QT	105,000	85,000	15	30	—
	4C3	QT	120,000	100,000	12	30	—
	4C4	QT	150,000	125,000	10	25	—
MILITARY	CW		55,000	27,000	15	25	—
MIL-S-15083	B	A or NT or QT	60,000	30,000	24	35	120
(SHIPS)	A-70	A or NT or QT	70,000	35,000	22	30	—
	A-80	A or NT or QT	80,000	40,000	18	30	—
	A-90	A or NT or QT	90,000	55,000	18	30	—
	A-100	A or NT or QT	100,000	60,000	15	30	—
MILITARY							
MIL-S-870	—	A or NT	65,000	35,000	20	30	120
(SHIPS)							
MILITARY	1	A or NT	70,000	40,000	20	35	—
MIL-S-15464							
(SHIPS)	2	A or NT	70,000	40,000	20	35	—
A.B.S.	1	A or NT	60,000	30,000	24	35	120
	2	A or NT	70,000	36,000	22	30	90
	Hull	A or NT	60,000	30,000	24	35	120
Lloyds	—	A	58,200 to 78,400	—	20	—	120

(*Table 5 continues on next page.*)

Specification and heat treatment			*Chemical composition %—maximum*								
Specification	*Class*	*(#) Heat treatment*	*C*	*Mn*	*P*	*S*	*Si*	*Cu*	*Ni*	*Cr*	*Other elements*
ASTM A 27-50T	N-1	—	.25 (b)	.75 (b)	.05	.06	.80	.50 (c)	.50 (c)	.25 (c, d)	W .10
	N-2	A or N or NT or QT	.35 (b)	.60 (b)	.05	.06	.80	.50 (c)	.50 (c)	.25 (c, d)	W .10
	N-3	A or N or NT or QT	—	1.00	.05	.06	—	—	—	—	—
	U-60-30	—	.25 (b)	.75 (b)	.05	.06	.80	.50 (c)	.50 (c)	.25 (c, d)	W .10
	60-30	A or N or NT or QT	.30 (b)	.60 (b)	.05	.06	.80	.50 (c)	.50 (c)	.25 (c, d)	W. 10
	65-30	A or N or NT or QT	—	—	.05	.06	—	—	—	—	—
	65-35	A or N or NT or QT	.30 (b)	.70 (b)	.05	.06	.80	.50 (c)	.50 (c)	.25 (c, d)	W .10
	70-36	A or N or NT or QT	.35 (b)	.70 (b)	.05	.06	.80	.50 (c)	.50 (c)	.25 (c, d)	W .10
ASTM A 148-50T	80-40	A or N or NT or QT	—	—	.05	.06	—	—	—	—	—
	80-50	A or N or NT or QT	—	—	.05	.06	—	—	—	—	—
	90-60	A or N or NT or QT	—	—	.05	.06	—	—	—	—	—
	105-85	A or N or NT or QT	—	—	.05	.06	—	—	—	—	—
	120-95	A or N or NT or QT	—	—	.05	.06	—	—	—	—	—
	150-125	A or N or NT or QT	—	—	.05	.06	—	—	—	—	—
	175-145	A or N or NT or QT	—	—	.05	.06	—	—	—	—	—
ASTM A 95-44	—	A or N or NT	.15–.45	.50 Min.	.05	.06	.20 Min.	—	—	—	—
ASTM A 216-47T	WCA	A or NT	.25 (e)	.70 (e)	.05	.06	.60	.50 (e)	.50 (e)	.25 (e)	Mo+W .25 (e)
	WCB	A or NT	.35 (e)	.70 (e)	.05	.06	.60	.50 (e)	.50 (e)	.25 (e)	Mo+W. 25 (e)
ASTM A217-49T	WC1	A or NT	.25	.50–.80	.05	.06	.60	.50 (f)	.50 (f)	.35 (f)	Mo. 45– .65
	WC4	A or NT	.20	.50–.80	.05	.06	.60	.50 (f)	.70–1.10	.50– .80	Mo. 45– .65
	WC5	A or NT	.20	.40–.70	.05	.06	.60	.50 (f)	.60–1.00	.50– .90	Mo. 90–1.20
	WC6	A or NT	.20	.50–.80	.05	.06	.60	.50 (f)	.50 (f)	1.00–1.50	Mo. 45– .65
	WC9	A or NT	.18	.40–.70	.05	.06	.60	.50 (f)	.50 (f)	2.00– 2.75	Mo. 90–1.20
	C5	A or NT	.20	.40–.70	.05	.06	.75	.50 (f)	.50 (f)	4.00– 6.50	Mo. 45– .65
	C12	A or NT	.20	.35–.65	.05	.06	1.00	.50 (f)	.50 (f)	8.00–10.00	Mo. 90–1.20
ASTM A 352-52T	LCB	N or NT or QT	.30 (e)	.70 (e)	.05	.06	.60	—	—	—	—
	LC1	N or NT or QT	.25	.50–.80	.05	.06	.60	—	—	—	Mo. 45–.65
	LC2	N or NT or QT	.25	.50–.80	.05	.05	.60	—	2.00–3.00	—	—
	LC3	N or NT or QT	.15	.50–.80	.05	.05	.60	—	3.00–4.00	—	—
SAE Automotive	0022	A or N or NT	.12–.22	.60 (b)	.05	.06	.60	—	—	—	—
	0030	A or N or NT or QT	.30 (b)	.70 (b)	.05	.06	.60	.50 (c)	.50 (c)	.25 (c)	W. 10
	0050	A or N or NT	.40–.50	.50–.90	.05	.06	.20–.60	—	—	—	—
	0050	QT	.40–.50	.50–.90	.05	.06	.20–.60	—	—	—	—
	080	A or N or NT or QT	—	—	.05	.06	—	—	—	—	—
	090	NT or QT	—	—	.05	.06	—	—	—	—	—
	0105*	QT	—	—	.05	.06	—	—	—	—	—
	0120*	QT	—	—	.05	.06	—	—	—	—	—
	0150*	QT	—	—	.05	.06	—	—	—	—	—
	0175*	QT	—	—	.05	.06	—	—	—	—	—
AAP M 201-47	A	Unannealed	—	.85	.05		—	—	—	—	—
	A	A or N	—	.85	.05	Basic	—	—	—	—	—
	B	A or N	—	.85	.05	.05	—	—	—	—	—
	C	NT or QT	—	—	.05	Acid	—	—	—	—	—
	D	QT	—	—	.05	.06	—	—	—	—	—
	E	QT	—	—	.05		—	—	—	—	—
AREA	—	A	.30 (b)	.70 (b)	.05	.06	.60	—	—	—	—
FEDERAL QQ-S-681b	X	—	.30	1.00	.05	.06	—	—	—	—	—
	O	A	.45	1.00	.05	.06	—	—	—	—	—
	1	A	.30 (b)	.60 (b)	.05	.06	.60	.30	.50 (i)	—	.25 each
	2	A or N or T	.35 (h)	.70 (h)	.05	.06	.60	.50	.50 (i)	—	.25 each
	3	A	.50	—	.05	.06	—	—	—	—	—
	4A1	A	—	—	.05	.06	—	—	—	—	—
	4A2	A	—	—	.05	.06	—	—	—	—	—
	4B1	N or NT	—	—	.05	.06	—	—	—	—	—
	4B2	N or NT	—	—	.05	.06	—	—	—	—	—
	4B3	N or NT	—	—	.05	.06	—	—	—	—	—
	4C1	QT	—	—	.05	.06	—	—	—	—	—
	4C2	QT	—	—	.05	.06	—	—	—	—	—
	4C3	QT	—	—	.05	.06	—	—	—	—	—
	4C4	QT	—	—	.05	.06	—	—	—	—	—
MILITARY MIL-S-15083 (SHIPS)	CW		.30 (k)	.70 (k)	.07	.06	—	—	—	—	—
	B	A or NT or QT	.30 (k)	.60 (k)	.05	.05	.20–.60	.30 (j)	.50 (i)	.20 (k)	Mo .20
	A-70	A or NT or QT	.35	—	.05	.05	—	—	—	—	—
	A-80	A or NT or QT	—	—	.05	.05	—	—	—	—	—
	A-90	A or NT or QT	—	—	.05	.05	—	—	—	—	—
	A-100	A or NT or QT	—	—	.05	.05	—	—	—	—	—
MILITARY MIL-S-870 (SHIPS)	—	A or NT	.25 (m)	.50–.70 (m)	.05	.05	.20–.50	.30 (n)	1.00 (n)	.20 (n, o)	Mo. .40–.60
MILITARY MIL-S-15464 (SHIPS)	1	A or NT	.20	.50–.80	.05	.05	.20–.60	.50 (n)	.50 (n)	1.00–1.25	Mo. .45–.65
	2	A or NT	.18	.40–.70	.05	.05	.20–.60	.50 (n)	.50 (n)	2.00–2.75	Mo. .90–1.20
A.B.S.	1	A or NT	—	—	—	—	—	—	—	—	—
	2	A or NT	—	—	—	—	—	—	—	—	—
	Hull	A or NT	—	—	—	—	—	—	—	—	—
Lloyds	—	A	—	—	—	—	—	—	—	—	—

(*See Explanation of Symbols and Notes on next page.*)

EXPLANATION OF SYMBOLS

Heat treatment: A—Full annealed; N—Normalized; T—Tempered; Q—Liquid quenched.

* Hardenability requirements when specified.

† Hardness tests when specified in contract or order.

[ ] Figures in brackets are expected values only. No test required unless specified in order.

(a) If full annealed specified, 33,000 psi yield required.

(b) For each reduction of .01%C below the maximum specified, an increase of .04%Mn above the maximum specified will be permitted to a maximum of 1.0% Mn.

(c) Total maximum content of undesirable elements is 1.0%. For each .10% below the specified maximum alloy content of 1.0%, an increase of .02% in the Cr plus Mo content and .06% in the Ni and Cu contents above the specified maximum will be permitted.

(d) Chromium plus molybdenum maximum .25%.

(e) Total maximum content of undesirable elements is 1.0%. For each reduction of .01% below the specified maximum C content, an increase of .04% Mn above the specified maximum will be permitted up to a maximum of 1.1%.

(f) Restrictions on unspecified alloy elements: W .10 for all grades; total maximum content of unspecified elements 1.0% for all grades except WC4 and WC5 where total maximum is .60%.

(g) Charpy "keyhole" notch impact test required at temperature specified by customer. Test temperature: -25, -50, -75, -100°F, etc.

(h) For each .01% C under the maximum, .025% Mn above the maximum may be added.

(i) For ordnance castings a maximum of 1.0% Ni is allowed.

(j) For ordnance castings a maximum of 1.5% Ni is allowed.

(k) For each reduction of .01% C under the maximum specified, an increase of either .04% Mn or .04% Cr above the maximum specified will be permitted, but in no case shall the Mn content exceed 1.0% or the Cr content exceed .40%.

(m) For each .01% C under the maximum, .04% Mn above the maximum may be added.

(n) Residual maximum permitted and shall not be added.

(o) For each .01% C under the maximum an increase of .04% Cr above .20% is permitted with maximum at .40% Cr.

MISCELLANEOUS REQUIREMENTS

Radiographic inspection—ASTM: A27, A148, A216, A217, A352; and SAE required when specified. MILITARY: MIL-S-870 required for castings of listed services and other services when specified; MIL-S-15083 all castings shall meet radiographic requirements for BUSHIPS only and for other agencies when specified; MIL-S-15464 all castings shall meet radiographic requirements.

Magnetic particle testing—ASTM: A27, A95, A148, A216, A217, A352; and SAE required when specified. ABS required for stern frames. MILITARY: MIL-S-870 required for castings of listed services and other services when specified; MIL-S-15083 all castings shall meet magnetic particle testing requirements for BUSHIPS only and for other agencies when specified; MIL-S-15464 all castings subject to magnetic particle testing.

Hydrostatic tests—ASTM: A95, A217, A352 required for all castings; A216 required when specified.

Destructive tests—ASTM: A216, A217, A352 required when specified.

Welding—Major defects over 20% of wall thickness repaired with consent of purchaser by welding with approved process: ASTM: A27, A95, A148, A216, A217, A352; SAE; FEDERAL QQ-S-681b; AAR M201; MILITARY: MIL-S-870, MIL-S-15083, MIL-S-15464.

General specifications for castings

A review of the detailed specifications in Table 5 indicates that there are certain types of castings for which chemical specifications are imposed but physical specifications are not included as a requirement. However, purchases of steel castings on the basis of chemical analysis alone undoubtedly involve agreements between the manufacturer and the customer regarding the ultimate performance characteristic of the parts. The general classes of castings for which no physical specifications are listed are the N-1, N-2 and N-3 classes of castings purchased under the ASTM (A-27-50-T) specification.

These parts represent many different products and their performance in use, therefore, is quite variable. Some castings are used in the unheat-treated or "as-cast" condition. Others are annealed, normalized, normalized and tempered, or quenched and tempered. In general, these are low carbon (i.e., 0.25-0.35%) and plain carbon steels and the phosphorus specification is 0.05% maximum, whereas the sulfur limit is 0.06% maximum. Some residual alloys such as copper, nickel and chromium are permissible but limited as shown in the table.

In contrast to the above group, other castings are specified according to certain physical property requirements instead of the analysis which is more or less left to the manufacturer unless some specific request is made by the customer.

The specifications of this second group list a gradation in physical properties from the soft and more ductile materials to the harder parts obtained by quenching and tempering. The specifications indicate the minimum tensile strength and the minimum yield point desired as well as the ductility which is indicated by the per cent elongation and per cent reduction of area. For example, the ASTM (A148-50T) specifications vary from the 80-40 class to the 175-145 class. These castings are designated as high strength structural materials and are heat-treated to develop the required physical properties. The 80-40 class, for example, is specified as having a minimum tensile strength of 80,000 psi., a yield point of 40,000 psi., an elongation of 18% in two inches and a reduction of area of 30%. The 175-145 grade on the other hand is required to have a minimum tensile strength of 175,000 psi., a minimum yield point of 145,000 psi., an elongation of 6% in two inches and a reduction of area of 12%. All the intermediate grades vary between these limits. The SAE automotive specifications are very similar to the above except that a Brinell Hardness Number is also required. The American Association of Railroads specifications (AAR-M201-47) are also similar to the above ASTM and SAE specifications except that a slightly higher ductility is required for a given hardness level. The Federal and military specifications (S-15083) and the American Bureau of Ships (ABS) specifications are also similar to the above. However, in the case of the ABS, castings which are used for machinery and hull parts must undergo a bend test of from 90-120° depending upon the specific applications. Where no analysis is specified in Table 5 it is assumed that the customers may request that certain analytical limits be placed on some of the elements.

There are a number of different types of castings which are purchased on both mechanical and chemical specifications as shown in the afore mentioned table. Some of these should be mentioned as special cases.

Specifications for cast valves, flanges and fittings for high temperature service

The ASTM specifications (A-95-44) are for carbon steel castings for these purposes and both the physical and chemical properties are specified. In addition, a bend test of 90° minimum is required. The manganese and silicon contents are specified at 0.50 and 0.20% minimum, respectively. Alloy steels for high temperature service are listed under ASTM (A217-49T) specifications. These are generally for high pressure vessels to be used under high temperature service.

Low temperature applications

High pressure vessels of cast steel for low temperature service are listed under the ASTM (A352-52T) specifications. These are ferritic steels and some contain from 2-4% nickel. In these cases a minimum of 15 ft./lb. impact strength is required. Impact testing temperatures, which simulate actual service temperatures of −25, −50, −75 and −100°F., etc., are specified.

Castings suitable for fusion welding

Certain carbon steel castings which are suitable for fusion welding and for services up to 750°F. are specified in the ASTM (A216-47T) specification. In this case the physical properties including a 90° bend test are specified. In addition certain restrictions are placed on the chemical analysis. For example, the limits with respect to carbon and alloy content are set in order to prevent air hardening of the welded section. The total maximum content of manganese is limited at 1.1%. The manganese content, however, is directly specified in accordance with the carbon content. The carbon specification for casting of 0.35% may be reduced somewhat and for each 0.01% that the carbon is reduced some 0.04% additional manganese is permitted. In this manner the manganese content may be increased up to the maximum stated above.

Special tests for castings

Radiographic and magnetic particle testing are required of some steels by almost all of the standarization agencies. The specifications requiring these tests are also listed at the bottom of Table 5. Certain specifications also require hydrostatic and destructive tests.

Surface defects

Also, surface defects may be repaired by welding up to 20% of the thickness without permission of the customer for many of the grades.

5. THE PHOSPHORUS AND SULFUR PROBLEM IN THE ACID OPEN HEARTH

The specifications for forgings and castings indicate that in general their phosphorus and sulfur contents should not exceed 0.05%.

Today, most of the industry would prefer to retain this limit on phosphorus indefinitely because of detrimental effects of phosphorus. However, as stated previously, the acid open hearth has been given an 0.010% higher limit on both phosphorus and sulfur. In recent years the phosphorus limit has been recognized as important but as shown in the AAR specification M201-47 acid steel is permitted up to 0.060% sulfur on certain items.

Although raw materials of low phosphorus and sulfur content are difficult to obtain, the acid open hearth industry has been meeting the specifications regularly. A recent survey of eight plants of member companies of the Acid Open Hearth Research Association regarding their ability to meet these specifications with respect to sulfur indicated that the average sulfur for two plants was from 0.033 to 0.034%. Four plants indicated an average of 0.040% and two others indicated an average of 0.043 to 0.044%.

Only one plant indicated difficulties due to surface hot tears from high sulfur content. Seven plants indicated that the sulfur should not exceed 0.050% and one stated that his product could attain 0.060% sulfur without detrimental effects. All plants indicated that there had, however, been a tendency for the sulfur content of the raw materials to increase in the last several years.

In general, the chief difficulty with sulfur is the tendency to produce hot tears in castings and in large ingots. It has been determined (4) that the percentage of rejections from hot-tears in low carbon thin walled castings varied from essentially none at 0.02% sulfur to about 1% at 0.05%. Above this value the rejections increased rapidly. Some difficulty with hot-tears is often overcome by changes in mold design.

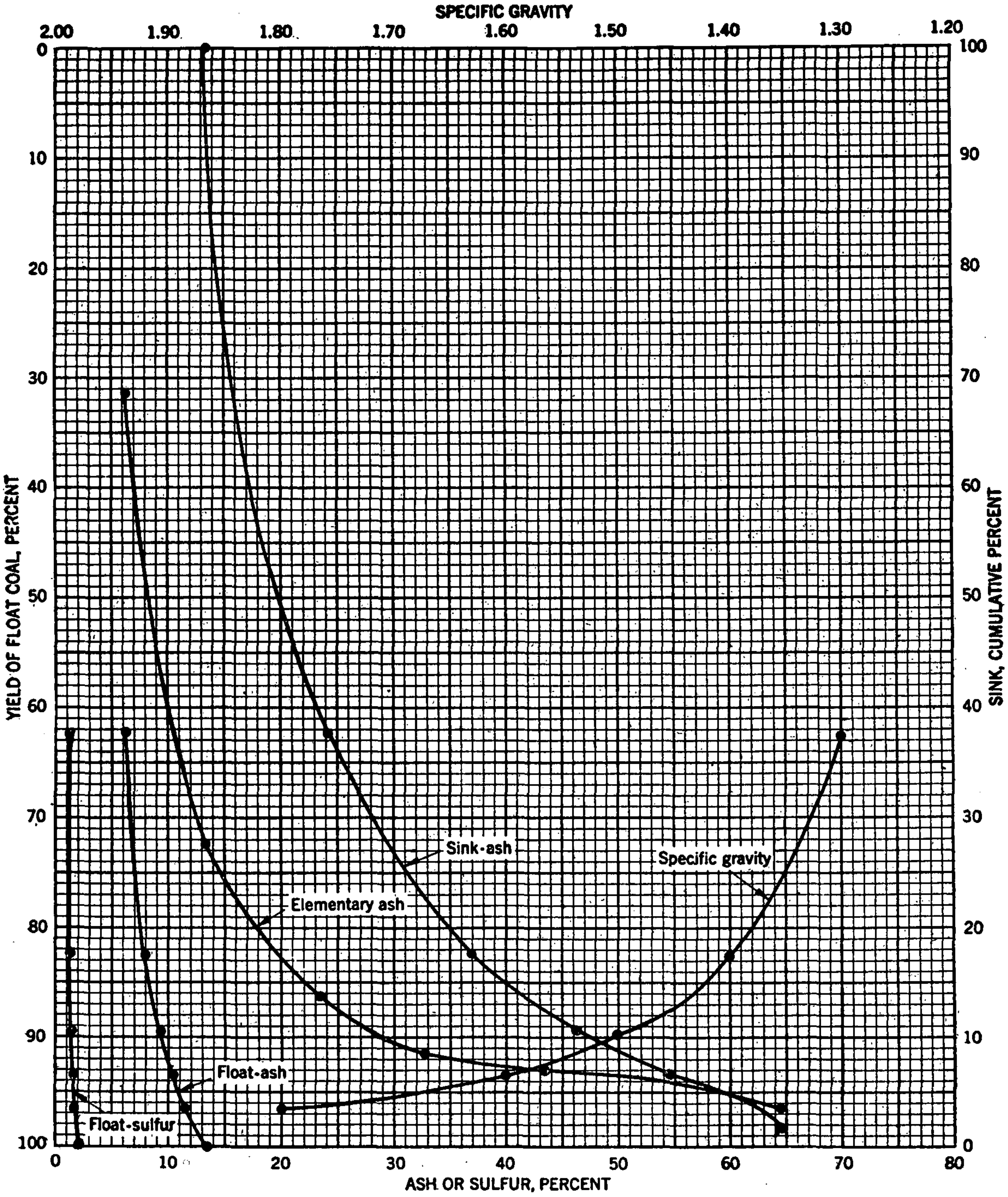
6. CONCLUSIONS AND RECOMMENDATIONS

A review of the information in preceding pages will indicate that the acid open hearth is probably much more widely applicable to the manufacture of different types of steel than is generally supposed. The acid open hearth and acid electric furnaces are largely responsible for all steel castings in the United States. The acid electric furnace is generally used in shops where the major products are small parts and where it is more economically feasible to co-ordinate the moulding shop with a melting furnace which will tap regularly every two hours. However, for shops in which not only small castings but some large forgings and/or castings are to be made, the acid open hearth is to be preferred. It may be seen from the foregoing discussion of specifications that the acid open hearth has continually produced steels within specification in spite of the fact the phosphorus and sulfur cannot be eliminated in the acid process. In other words there has been and probably always will be sufficient good scrap to permit the acid furnace to continue to make a wide variety of products as indicated in this paper.

The question might be raised at this time, however, as to why advantage has not been taken of the fact that the acid open hearth will not remove the phosphorus and sulfur. By this it is meant that there is no grade of high sulfur and phosphorus screw stock steel which the acid open hearth could not produce with ease and economy. The argument might be made that the acid open hearth is not used because the scrap will be high with respect to phosphorus and sulfur, but if the ultimate product is to contain from 0.09 to 0.13% phosphorus and from 0.10 to 0.18% sulfur as in standard screw stock the scrap problem is not important. Furthermore, many grades of bars, track spikes, and sheet and tin plate are made with relatively high phosphorus and sulfur. In fact, there is no Bessemer product listed in Table 6 according to Henning

Table 6
ACID BESSEMER PRODUCTS (5)

	Analysis, per cent			
Product	*Carbon*	*Manganese*	*Phosphorus*	*Sulfur*
Screw steel (standard)	0.08 to 0.16	0.60 to 0.90	0.09 to 0.13	0.100 to 0.180
Screw steel (high sulfur)	0.08 to 0.16	0.60 to 0.90	0.09 to 0.13	0.200 to 0.300
Screw steel (high carbon)	0.25 to 0.35	0.60 to 0.90	0.09 to 0.13	0.100 to 0.180
Skelp (electrical conduit)	0.08 max.	0.30 to 0.60	0.11 max.	0.08 max.
Bars, track spikes, sheet and tin plate	0.10 max.	0.30 to 0.50	0.11 max.	0.07 max.
Soft wire	0.12 max.	0.60 max.	0.11 max.	0.08 max.
Medium wire	0.10 to 0.20	0.70 max.	0.11 max.	0.08 max.
Reinforcing bars	0.15 to 0.35	0.70 max.	0.11 max.	0.08 max.
Structural soft	0.07 to 0.12	0.30 to 0.60	0.11 max.	0.08 max.
Structural medium	0.25 to 0.35	0.30 to 0.60	0.11 max.	0.08 max.
Structural high tensile	0.35 to 0.45	0.30 to 0.60	0.11 max.	0.08 max.



Specific gravity fractions	Elementary data				Computed cumulative data					
	Weight grams	Percent			Float, cumulative percent			Sink, cumulative percent		
		Weight	Ash	Sulfur	Weight	Ash	Sulfur	Weight	Ash	Sulfur
Float - 1.30		62.3	6.3	1.19	62.3	6.3	1.19	100.0	13.1	2.01
1.30 to 1.40		20.2	13.2	2.52	82.5	8.0	1.52	37.7	24.2	3.36
1.40 to 1.50		7.2	23.4	3.12	89.7	9.2	1.64	17.5	36.9	4.32
1.50 to 1.60		3.8	32.8	3.63	93.5	10.2	1.73	10.3	46.4	5.16
1.60 to 1.80		3.1	43.5	4.05	96.6	11.3	1.80	6.5	54.4	6.06
Sink - 1.80		3.4	64.3	7.89	100.0	13.1	2.01	3.4	64.3	7.89

Fig. 5. Washing characteristics of composite of 7 high-volatile Cauca Valley Coals of Table 2. Size 37 – 10 mm.

(5) that could not be made with considerable ease and economy in the acid open hearth. If the acid furnace were required to produce both low and high phosphorus and sulfur products, careful scrap segregation would be necessary.

The recommendation that is made herein, therefore, is that the acid open hearth might be used as a method for the production of steels with a wide variation in analysis. In countries where only a few open hearths are to be installed to manufacture diversified products, the acid open hearth might be a wise choice. Certainly, the acid furnace should be considered if the plant is dependent upon cold pig iron.

It might be argued that the basic open hearth is used for such steels as the screw stock or soft wire, for example, because of their low carbon contents. However, carbon contents as low as 0.10% no longer create any great degree of difficulty to the acid open hearth operator in view of the use of an oxygen lance for quickly reducing the carbon content. This can be done rapidly and without the degree of erosion which typified the manufacture of low carbon steel in the acid open hearth process through the use of ore additions.

As shown in another paper at this Conference, the successful use of the acid open hearth depends upon the availability of high B.T.U. fuel in order to permit this furnace to work under its most efficient procedure. When this is done, it will be found that the acid open hearth is relatively economical.

Acknowledgments

The author wishes to express his gratitude to the Officers and Members of the Acid Open Hearth Research Association, Inc. of Pittsburgh, Pennsylvania for permission to use some of the data given herein. Also, the assistance of Mr. J. B. Kopec, research engineer for the Association, has been greatly appreciated.

Furthermore, the assistance of Mr. Charles W. Briggs and the Steel Founders' Society of America in furnishing copies of their castings specifications is deeply appreciated.

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Use of Phosphorus Ores for the Production of Pig Iron for Transforming into Steel by the Thomas Process

SOCIETÀ ILVA

The Società Ilva, which forms part of the Finsider group, has an iron and steel plant at Bagnoli (Naples), in which there are three blast furnaces, a Thomas converter (Basic Bessemer), a Martin steel mill, one blooming mill and several rolling mills intended for the production of merchant sections and bars for reinforced concrete.

The Thomas converter was installed in 1939–41 in relation to the available supplies of iron ore in the vicinity of "Nurra", on the Island of Cerdena; this ore is composed of limonites and siderites with a phosphorus content of 0.60 to 0.70%. The pig iron to be produced in the blast furnaces for subsequent conversion into steel by the Thomas process, should have a phosphorus content of 1.80 to 2%. Utilization of "Nurra" ore in large quantities would not result in a pig iron with the phosphorus content indicated above.

It is therefore necessary to add other phosphorus-bearing materials to the fusion charge. For this purpose apart from Swedish Kuruna D 2% ore, it was considered desirable to use "phosphorites" from nearby Tunis (North Africa); they are derived from ancient deposits of a sedimentary formation, and are mainly composed of fossil remains of molluscs and other small marine animals. The average composition is as follows: CaO = 48%; P = 12%; SiO_2 = – ; Fe = 0.6%.

Since most of the minerals used in the Ilva blast furnaces are acid, the phosphorites, apart from phosphorus, contribute the limestone which they contain to give the slag the necessary degree of basicity. The average approximate fusion charge for the production of Thomas pig iron is summarized in the adjoining table, which also shows the average analyses of the materials used.

As an illustration, the average analyses of the pig iron produced and of the slag are shown here. As is known, in the blast furnace process all the phosphorus is reduced and passes integrally into the pig iron, except the very small portion corresponding to the dust eliminated by the gas in the washing.

Table 1

AVERAGE ANALYSIS

Thomas pig iron	*Slag produced in the blast furnace for Thomas pig iron*
C — 3.45%	SiO_2 — 32%
Si — 0.38%	Al_2O_3 — 13%
Mn — 1.20%	CaO — 46.7%
S — 0.04%	MgO — 3.5%
P — 1.90%	Fe — 0.3%
	Mn — 1.3%
	S — 2%

The molten pig iron which leaves the blast furnace is transported by cars to the Thomas steel plant and after being kept in the 800 ton mixer, it is heated with recovery gas, and filled into the basic converters for transforming into steel.

The Thomas steel mill has been in operation for some months and has suffered no handicaps in working which could be attributed, in particular, to the fusion charge prepared for the blast furnace. The progress of the blow is similar to that described in technical literature on the various technical steel plants installed in Europe.

It is therefore possible to produce almost all types of ordinary carbon steels and in qualities ranging from extra mild (C = 0.05) to hard steels (C ca. 0.50) which can be used in the manufacture of merchant sections, bars, sheet, and tinplate.

Another interesting point about this conversion process is the utilization of the slag, in which almost all the phosphorus contained in the pig iron is digested and, precisely on account of these elements and their particular chemical composition, it represents a fertilizer in demand by farmers. Precisely for these reasons, beside the Thomas steel mill there is an installation for grinding, crushing and packing the slag; a sizable profit is made from its sale, and this reduces the cost of the steel.

The composition of the slag is as follows: CaO = 48.0; SiO_2 = 0.0; P_2O_5 tot. = 18.0 (soluble in citric acid: 16.0); Fe tot. = 11.0; and output of it amounts to about 230 kg. per ton of steel produced.

Table 2

APPROXIMATE AVERAGE FERROUS CHARGE USED FOR THE PRODUCTION OF THOMAS PIG IRON

		Analysis								
Minerals	%	*H_2O*	*SiO_2*	*Al_2O_3*	*CaO*	*AgO*	*Mn*	*P*	*Fe*	*S*
Agglomaterate pyrite cinders and various fines	44	0.20	11.24	2.6	3.3	0.3	1.0	0.084	56.2	0.16
Nurra ore	20	7.00	15.00	7.3		3.22	0.21	0.70	43.00	0.24
North Africa	18	6.00	4.51	1.59		4.26	1.8	0.02	53.2	0.13
Kuruna	7	2.00	2.5	2.78	2.5	0.2	0.16	1.8	61.0	0.08
Other ores	6									
Pig iron scrap	5									
	100									
Limestone kg. per ton or pig iron: 240 kg.		1.00	0.4	—	55	0.4	—	—	0.15	—
Phosphorites kg. per ton of pig iron: 130 kg.		4.00	4.6	1.2	47	—	—	11.00	—	1.2

Perrin Process for Converter Steelmaking

MARC ALLARD

INTRODUCTION

The general scrap scarcity results in more and more importance being given nowadays to the direct transformation of pig iron into steel. Furthermore, it is evident that the refining of iron in the liquid state, as it is produced in the various kinds of blast furnaces (using coke, charcoal, or electric power) is a source of economy since advantage can be taken of the latent heat of the metal. Hence, the converter appears to be ideally suited for the refining of iron, necessitating but small initial costs as compared with those required for the installation of other refining furnaces. If the pig iron contains enough phosphorus, the basic converter process, or Thomas process, brings a ready solution to the problem. If the pig iron has a low phosphorus content, the acid Bessemer process can be used. For pig irons too high in phosphorus to be refined by the acid Bessemer process, but too low to be used in the basic Bessemer process, there are other alternatives:

Use of a basic converter and introduction of oxygen-enriched air to compensate for the lack of calories,

Use of an acid converter first and removal of the phosphorus in a second furnace (open hearth or electric furnace), or use of the Perrin process, with a dephosphorizing slag.

The converter process is particularly well adapted to smaller works since it permits a steady and diversified production, and the construction of small converters is cheap.

However, working conditions must be such as to provide the best quality steel along with low iron losses. In other words, highest yield must be obtained at all stages of the process, together with a uniform quality of the finished products giving steel assemblies of minimum weight.

It is obvious that one of the first metal savings consists in losing a minimum of it during the blowing in the converter, while at the same time the various heats are refined in the shortest possible time in order to obtain maximum output for the plant and minimum thermal losses for the metal. Thus, as regards the basic converter process in Europe, the conditions for good blowing have been carefully studied, attention having been directed towards correct dimensions of the converter, its lining and bottom.

It is well established that the greater the silicon content of a Thomas pig iron, the more difficult it is to have good blowing conditions, on account of the projections. The possibility of removing part of the silicon through oxygen blowing before introducing the iron in the converter has been investigated.

Furthermore, the removal of silicon from pig iron is also quite a usual operation in the case of the open hearth.

For these reasons, the French Iron and Steel Research Institute (IRSID) intends to publish the results of its research on these questions.

The quality of the steel is generally closely connected with the presence of those elements which are known for their ill-effects (sulfur, phosphorus, oxygen, nitrogen) and also with the micrography. There is no foundation for the frequent belief that the converter processes give steels of inferior qualities. That is why converter steels are refined a second time in the electric furnace or in the open hearth, in order to obtain high-grade steels.

It must be noticed first that the short time during which the converter steels remain in those furnaces, does not allow of any marked reduction in the nitrogen content. But there are of course other processes using oxygen, which enable production of low nitrogen steels in the converter. As for the phosphorus content, it can be brought down in the basic converter by, for instance, the use of two slags and in the acid converter by thorough mixing with a ferro-calcic slag, as previously pointed out.

In order to complete the steel refining, the sulfur and oxygen contents must be decreased and best micrographic cleanness should be produced. These results can easily be obtained through the Perrin processes, mixing the steel with an alumina-calcium slag. The steels thus obtained are killed ones, and through the proper use of ingot tops, 86–87% of the ingot can be taken as good metal. These methods require only a moderate amount of bauxite and a small monophased electric furnace for melting the slag.

IRSID feels that the publication of the results it has obtained on the basic converter, results which can also be applied to the acid converter, should be of interest and value to the South American steel-producing industry. These processes, it should be added, can be applied equally well in the electric furnace and open hearth.

1. PRECAUTIONS CONCERNING THE BLOWING IN THE BASIC CONVERTER PROCESS

(a) *Some recent observations regarding the lining and the bottom of the converter*

The following facts are taken from results obtained, published or to be published, by the IRSID. The corresponding list is shown at the end of the paper.

The amount of air blown in a basic converter per minute depends upon the appearance of projections. Research conducted by Mr. Malcor at the Homécourt Works has clearly shown that the projections usually start during the first half of the decarburization period, when a third of the amount of air necessary for the process has already been blown. A curve can be defined showing the variations of the limit-flow (amount of air per minute, which, when exceeded, leads to the appearance of projections) as a function of time, or of the amount of air blown since the beginning. This volume can be expressed in percentage of the total volume of air necessary for refining the charge completely. The limit-flow curve shows a minimum which, in fact, corresponds to about 30% of the total volume of air blown. The value of this minimum decreases rapidly as the silicon contents of the iron increase.

Furthermore, observation of a converter during blowing shows bunches of sparks, appearing periodically, as forerunners of the projections. Similar observations can be made for the rumbling of the converter. If the interior of a converter is examined through its mouth, when there are no projections (with projections, this observation is impossible), violent motions of the bath can be noticed which look like the sea running against a pier. These motions are also periodical.

All these phenomena seem to reappear regularly, in general every two seconds, and it was suspected that the projections might be connected with massive oscillations of the bath. The basic phenomenon should thus be of a hydromechanical nature and the physical chemical factors might well disturb these oscillations and modify their tendency to produce projections, without actually being their cause.

In 1949, IRSID asked the Laboratoire dauphinois d'hydraulique to make investigations on small-scale models. Afterwards, full-scale trials were made at the Mondeville works,[1] replacing the iron by water.

Recently, a systematic study has been undertaken in the shop, with iron, at the Moyeuvre works,[2] using the "volume-débitgraphe" (volume-flowmeter) designed by the IRSID. This flowmeter records the amounts of air with an automatic device for pressure correction, the speed at which the paper sheet unrolls being proportional to the flow of air. Hence, at every moment, the length of paper gives an indication of the volume of air which has been blown since the beginning of the operation. The apparatus also possesses a pressure-meter and recorder.

All these full-scale investigations confirmed the results obtained from the laboratory, on small-scale models, when the water is replaced by iron.

First it was proved possible to apply to actual size processes, and with iron, the results obtained on reduced models with experimental fluids. Similitude factors have been defined as follows:

Scale similitude: the measurements made on reduced models and on a full scale at Mondeville show that the critical air flows, and the number of rotations per minute, vary with the size of the apparatus, respectively according to a 5/2 power and 1/2 power law. This is in accordance with the hydrodynamic similitude law of Froude. It is assumed that only the inertia and gravity forces act. Just as the oscillation period of a pendulum is independent of the density of the material of which it is made, so the global oscillations of the bath in the converter must be independent of the density of the liquid. At Mondeville, this periodicity was found equal to two seconds, equivalent to the one observed with steel.

Fluid similitude: from what has been said, a change in fluid should have no influence on the appearance of the projections, when reasoning on a hydrodynamic basis.

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[1] Société métallurgique de Normandie.

[2] Société de Wendel.

This had been foreseen by results obtained with various fluids: mineral oils of varying viscosity, alcohol, acetic acid, heavy alloys, "cerrobend" alloy (lead, cadmium, bismuth, tin) of composition very similar to that of the Wood alloy (density : 9). Thus, density, surface tension and viscosity could be changed. It could be observed, in particular, that an increase of the viscosity necessitated a decrease in the critical flow above which global oscillations in the converter appear, but only for high viscosity values. The normal viscosity of a regular pig iron being about 0.02 poise, i.e., quite nearly that of water (0.01 poise), the viscosity range is one where this factor has no influence. Research carried out with cerrobend also gave only minor deviations from water.

Thus, the fundamental experiment with water alone, indicated that Froude's law of similitude could be applied and showed the importance of the experimental determination of the following variables:

(*a*) The "critical flow" (Q_c): flow of air at which global oscillations of water appear;

(*b*) The "limit flow" (Q_e): flow at which that circling wave jumps out of the converter's mouth, i.e., appearance of "bunchwise" projections.

Numerous runs were made with water on small models, and on a full scale. To reproduce actual conditions, such as lime and scrap additions in the converter, pieces of cork and plexiglas were thrown into the bath. It was found that critical flow and limit flow vary in the same way for a constant amount of material and a given height of the bath in the converter, and that it is advantageous to bring the critical flow to a maximum. This critical flow decreases as one goes from the centre to the outer circles of the moving bath. As the periphery is not usually provided with tuyeres, it appears that an extension of the blowing zone towards the periphery will be of interest.

The deleterious effects of silicon have been pointed out. Its action is mainly noticeable between 15 and 60% of the total amount of air blown. The minimum of the limit flow decreases as the silicon content increases (see Figure 1).

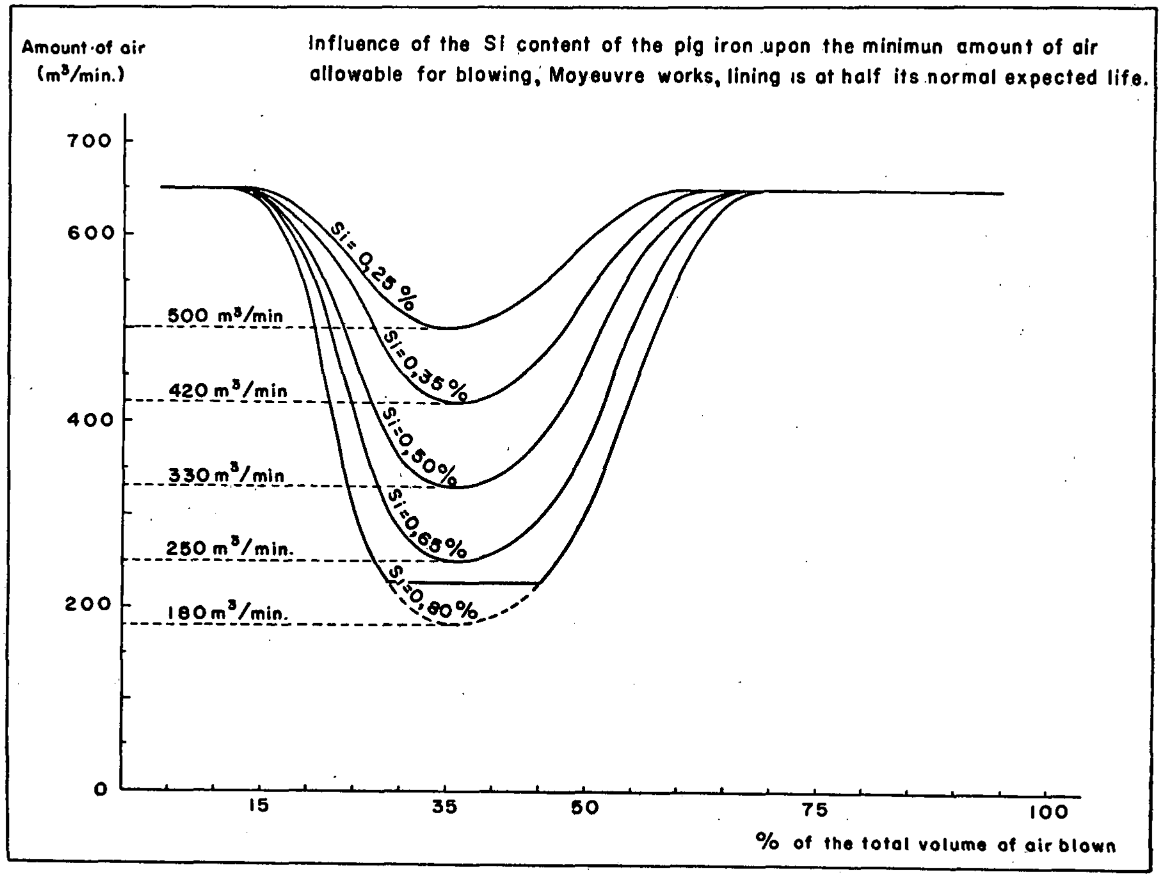

Figure 1.
Influence of the Si content of the pig iron upon the minimum amount of air allowable for blowing, Moyeuvre works.

A high manganese content does not seem to be beneficial.

The capital breaking action played by the solid additions like limestone and scrap has been demonstrated. The starting temperature of the iron has no real importance in this respect, but it allows a greater amount of scrap additions.

The blowability is evidently improved by a decrease in the height of the bath. However, since there is a limit to the charge C (in metric tons) and the interior diameter D of the lining (in metres) has been worked out as being: $D \geq 0.56 \times \sqrt{C}$ [for instance, D equals 2.10 m. for 14 tons, and D equals 3.05 m. for 30 tons], the suppression of the bevelled part of the body of the converter, joining the side wall to the bottom, must be aimed at.

As a general rule, the existence of a dead, inactive zone, often called the circulation zone, appears to be of no value, and is all the more harmful the nearer it is to the periphery.

The practical conclusion is that for a converter of a given diameter, the bottom with its holes must be as large as possible. As an example, Figure 2 shows the modifications brought to the converter, in application of the foregoing rules.

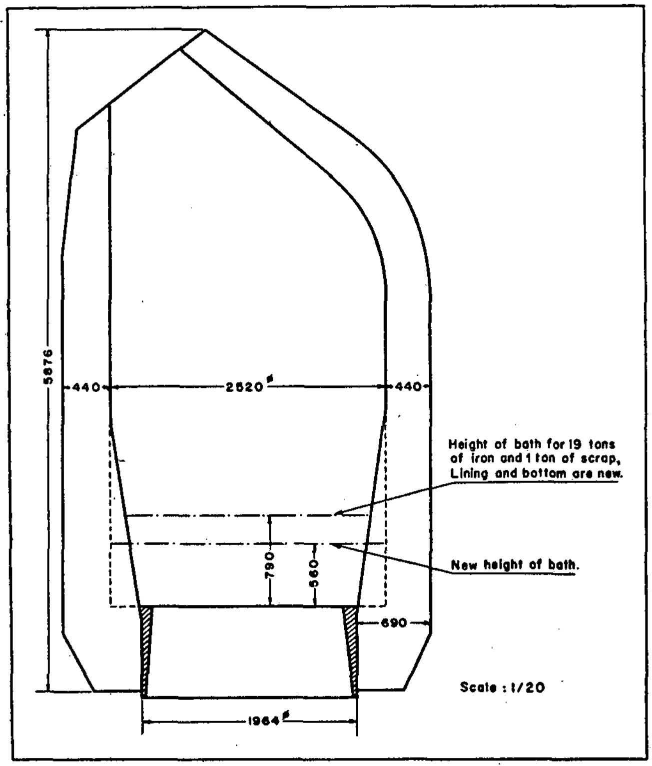

Figure 2. Moyeuvre works. Comparison of the usual and improved design.

A word remains to be said about the influence of the quality of the limestone on the blowability.

The silica introduced with the limestone is as bad as the silica coming from the silicon of the iron. Raw limestone, if it is not gas-baked, must be screened to remove the dust, which is mainly silica. But the lime must be reactive, and the big lumps, especially if they have been overbaked, are of poor quality. A crushing of lime is therefore advisable.

These two operations are not expensive and lead to definite over-all savings. It seems that a good, reactive lime rapidly neutralizes the ill effects of silica. Some results can be achieved through partial replacement of the lime by soda, but this is expensive.

Recent research has also led to a better understanding of the way in which silicon affects blowability in the basic converter. The conclusions are outlined below.

The presence of silicon in large amounts leads to the formation of a semi-slag having probably a higher melting point than the temperature existing in the converter. This semi-slag is thus changed into foam occupying a large volume and including up to 50% of its weight of small droplets of iron. The volume of the mass in motion in the converter becomes large and the circling movements, caused by the air, eject part of this spongy slag out of the converter, with a resulting loss of iron.

The conditions prevailing in the acid converter are quite different. The temperature, in particular, rises rapidly due to the combustion of a large amount of silicon, so that the early slags formed are easily fluidified. However, the fact remains that the motion study of the bath in a basic converter could help to improve acid converter technique too.

It is hoped that these indications will be of some help to converter builders and steel engineers.

(b) *Recent work on pre-refining pig iron in the ladle with oxygen*

In order to lower the silicon content of the basic pig iron (basic open hearth, or basic converter), several processes have been tried and, in particular, the blowing of oxygen in the ladle.

However, this operation is of a prohibitive length, more ladles are needed, there is a great loss of oxygen and the uniformity of the resultant iron is not satisfactory. Finally, after numerous trials, two techniques have been proposed by the IRSID.

(1) In the first one, the oxygen is injected in the ladle while the latter is at the blast-furnace during the tapping. Two injection tubes of 20 mm. inside diameter are used; they are coated with a silica-aluminous refractory material. The pressure is of 3 kg.

(2) In the other method, the oxygen is blown with a tube on the surface of the metal and at high pressure (15 kg.). The same idea has been applied in Austria to refine phosphorus-containing iron.

The drawbacks resulting from the sticking of slag on the injection tubes can be avoided by surrounding the main tube with a system of small vertical feeders opposing the running down of the projections of liquid slag along the tube.

The process is easy to apply, the tubes remain intact and the iron losses are small.

Another improvement was made possible by the simultaneous addition of lime (7 kg. per ton of iron). The results obtained by these various methods are summed up in the following table, and based on a decrease of the silicon content from 0.70 to 0.45% in all the cases (see Table 1).

Table 1

METHODS FOR LOWERING THE SILICON CONTENT OF BASIC PIG IRON

	A	*B*	*C*
	Using tube dipped into bath	*O_2 blowing during filling at blast furnace*	*Copper tube on surface and CaO additions*
Fe loss per ton of iron..........	12 kg.	8 kg.	3 kg.
Appreciable slowing down of the circulation of ladles..........	yes	no	no
O_2 consumption per ton of iron..	5m^3	4.5m^3	4.5m^3 [a]
Temperature rise of iron........	60°C	65°C	65°C
Ratio of the quantities of Si and Mn which are oxidized: Si/Mn	3	3	4

[a] Can be reduced by using compressed air instead of O_2 in the auxiliary feeders.

Methods B and C give a better homogeneity of the iron than method A.

Method C enables the removal if necessary of more silicon by blowing more O_2 without excessive iron losses. This is not the case with the other methods.

We think it worthwhile to publicize these processes which are conducive to a reduction in fabrication costs and to an increase in the productivity of the basic shops.

2. Rapid refining of steel through the Perrin process

The treatment of a steel melted in one furnace by a slag prepared in another furnace enables a steel quality to be obtained in the ladle which would otherwise need duplexing with the usual refining methods, in which the slag lies upon the bath. These duplex methods are rather slow and sluggish because the bath-slag interface is a plane whereas in the thorough mixing of the Perrin processes, the metal-slag contact is very intimate.

In many cases, the Perrin processes lead to economies of furnaces, of time and of materials.

(*a*) These processes have a particularly favourable field as regards removal of phosphorus, as in the case of acid converter steels (Bessemer steels) obtained from irons of high P content (often from 0.20 to 0.50%).

Mixing these steels with a fused slag containing 25–28% iron oxide and 45–50% CaO, a steel is obtained containing 0.025% P, as compared to 0.30–0.40% before the treatment. Starting with a P content equal to 0.40%, the slag amounts to about 7–8% of the weight of the steel to be treated.

The technique of the process can easily be developed in the case of production of non-killed low-carbon steels. The manganese additions can be made without difficulty in the ladle. The fabrication of higher carbon steels requires more care since the mixing is accompanied by a boiling effect. The process is then slower.

However, it may be of interest to produce in the liquid state, a low-carbon steel of high purity which could be re-used in an acid furnace for recarburation, if needed.

It should be mentioned that with an excellent P removal, there is also an appreciable removal of sulfur.

The molten slag required can be prepared in a reverberatory furnace starting with iron oxide (scale or pure ore) and lime, or in an electric furnace of very simple design. It can also be prepared by overblowing iron with lime additions, in a small basic converter.

After mixing, the slag formed is a phosphorus mineral of value.

It is difficult to establish a balance that would apply in every case when using this process. Each specific application must be studied separately.

It is therefore evident, that when starting with a given P-content in the local ores, it may be advantageous to have a basic converter installation, whose slag could partly be recycled in the blast furnace to attain the normal P-content of the Thomas iron. If oxygen is available as mentioned above, basic irons of low P-content can also be refined.

As an example, Table 2 gives the analysis of five heats obtained with a dephosphorizing mixing:

Table 2

Analysis of five heats obtained with a dephosphorizing mixing

	A. *Preblown steel* B. *Ingot steel*					A. *Slag before mixing* [a] B. *Slag before mixing*				
	C	*Mn*	*S*	*P*	*Si*	*SiO_2*	*FeO* [b]	*CaO*	*MnO*	*P_2O_5*
I (A)	.08		0.075	.352	.100	4.	37.	50.	2.3	.230
(B) [c]	.08	.35	.053	.025	.045	9.2	32.	40.3	10.12	6.360
II (A)	.08		.077	.288	.240	4.8	37.	46.2	1.5	.343
(B)	.08	.25	.050	.025	.057	10.2	33.	38.10	6.03	4.55
III (A)	.09		.076	.308	.240	4.68	39.21	42.95	1.92	.506
(B)	.06	.18	.040	.025	.070	11.6	31.83	37.7	5.77	4.600
IV (A)	.07		.072	.283	.188	5.6	38.96	43.75	1.83	0.542
(B)	.07	.32	.047	.025	.050	9.6	25.15	40.50	8.	5.980
V (A)	.58		.082	.318	.240	7.4	39.65	46.	.70	.288
(B)	.50	1.	.041	.045	.070	18.	18.85	43.	4.	5.080

[a] On the average, slag weight = 8% metal weight.
[b] Assuming that all the Fe is in the form of FeO.
[c] After addition of ferro-Mn in the ladle.

(*b*) For deoxidation and removal of sulfur, the Perrin processes have brought to the metallurgy of the converter (basic or acid) the essential certainty of a good deoxidation along with the possibility of a good sulfur removal. Simultaneously, they lead to clean steels, i.e., free from inclusions.

In practice, these results are achieved by mixing the steel from the converter with an aluminous slag previously melted in a monophased electric furnace. To prepare the slag, correct amounts of lime and bauxite are loaded into the furnace. The ladle containing the slag is placed under the converter, and the latter is rapidly tilted to ensure a violent mixing. Additions of aluminium or ferro-silicon are made during the tilting and the fully-killed steel is poured into the moulds.

The aluminous slags used have approximately the following composition: CaO : 53%; Al_2O_3 : 43%; SiO_2 : 3% maximum; FeO : minimum (about 1%).

The amount of slag used is 4–5% the weight of the steel. Its fusion in the furnace requires 60 to 70 kW. per ton of steel. The necessary raw materials are solely bauxite and lime. The most suitable furnace for melting the slag is a monophased electric furnace, with one electrode, with a carbon and dolomite lining.

The deoxidation products remain in the slag after mixing. For the S-removal, the completion of the reaction

$$CaO + FeS \qquad CaS + FeO$$

seems to be possible through the deoxidation reactions:

$2\ Al + FeO \qquad Al_2O_3 + 3\ Fe$; if Al is used
$Si + FeO \qquad SiO_2 + 2\ Fe$; if Fe-Si is used

The killed metal thus obtained is free of oxygen, sulfur and inclusions. It is a remarkably good deep-drawing steel, non-ageing and of uniform quality. It is later rolled on a continuous strip mill built in France by the Usinor Company.

Another French steel company uses the Perrin process to produce, from Thomas steel, a new type of steel of high purity for the charging of high frequency furnaces. The analysis of this steel is as follows:

C	*Mn*	*P*	*S*	*O*	*Az*
0.025%	0.05%	0.015% to 0.020%	0.008% to 0.010%	0.003% to 0.005%	0.006% to 0.009%

Finally, the same process is used for the production of large quantities of high carbon steels and low alloy, high strength steels of the following compositions:

Steel	*C*	*P*	*Mn*	*S*	*Si*	*Cr*	*Mo*	*Ni*	*O*
Cr-Mo................	0.18 to 0.45	0.020 to 0.030	0.5 to 0.7	0.005 to 0.015	0.10 to 0.30	0.85 to 1.	0.25 to 0.35	—	0.002 to 0.004
Ni-Cr................	0.18 to 0.30	0.020 to 0.030	0.30 to 0.35	0.005 to 0.015	0.10 to 0.25	0.6 to 1.	—	2.7 to 3.2	0.002 to 0.004

All the steels produced by this method, even the very low carbon steels, are fully killed and without segregation. The hardenable steels are of regular and uniform hardenability. They meet the most severe specifications and, in particular, possess excellent impact properties perpendicularly to the rolling direction.

This application to the converter steels is not limitative. Before the process was applied to Thomas steels, the intermixing of steel with an aluminous slag had been used for refining steel produced in the electric furnace and in the open hearth. At the Ugine works, for instance, the method enables an electric furnace steel of more than 0.060% S to be treated, which shows after treatment a S-content of less than 0.010%.

In addition to the steel quality thus achieved, there can also be considerable economy in manganese. For a good forgeability, the final manganese content must often be high. This manganese content can be much lower when the sulfur and oxygen contents can be kept at a low level.

We are therefore convinced that the present slag-mixing method is a very effective one for saving manganese.

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Thomas Steel with Low Nitrogen and Phosphorus Contents

P. COHEUR

Owing to its method of production, basic Bessemer steel has characteristics which differ from those of open hearth steel, with which it is usually compared, and often, wrongly, opposed.

It is certain indeed that for many applications such as free cutting steels, rails, structural steels, concrete rods, etc., Thomas steel is as suitable as Martin steel. For other uses, however, for which great ductility at room temperature is required (steel for deep drawing, for example), Martin steel seems to be preferred. That is why, for over half a century, numerous steelmen have tried to modify the properties of Thomas steel by reducing its nitrogen and phosphorus content. The various methods which they have proposed and which are actually used in daily practice, consist, in fact, in a careful control of the classical operation of conversion (temperature, pressure, duration, etc.). With such a control it is currently quite possible to produce types of steel with a nitrogen and phosphorus content which, in the case of Belgo-Luxembourgeoise plants, is around 70/10,000 to 120/10,000 for nitrogen and 40/1,000 to 50/1,000 for phosphorus.

These percentages are already quite sufficient to obtain mechanical properties which suffice for most applications requiring a certain ductility at room temperature.

In order to reduce the nitrogen content further, without radical modification of the principle of bottom blowing, it is necessary to resort to special processes of elaboration such as blowing with oxygen-enriched air, or mixtures of other gases. The purpose of this paper is to give a very brief account of the researches undertaken by the Centre National de Recherches Métallurgiques (CNRM) in that direction and of the results obtained (1).

First, it should be mentioned that the tests were undertaken with 15/18-ton converters in the three Thomas steelworks of the Liège region, that is: the Ourgrée-Marihaye, Espérance-Longdoz and John Cokerill companies, and in the Luxembourg steelworks of the ARBED company.

The products have been studied chemically and physically in the laboratories of these four companies and in some transforming factories.

Finally, the results, obtained from over 400 blows, or more than 7,500 tons, have been made the subject of statistical studies.

TESTS WITH OXYGEN-ENRICHED AIR

The first tests were undertaken to define the optimum conditions of the oxygen-enriched process. For that purpose, several methods of blowing were used in which modifications were introduced either in the process of oxygen enrichment, or in the charge, or in the nature, quantity and timing of additions. These tests (see Table 1 and Figure 1) have shown that:

(*a*) With an air enriched to 30% oxygen which, from an economic viewpoint, seems to be the optimum, and by using only scrap to cool the steel bath, the nitrogen content is reduced by 20%, as compared with the reference blows, which contained 100 ten thousandths of nitrogen.

(*b*) With the same air enrichment, but using ore instead of scrap, it was noticed that the nitrogen content fell by 40% as compared with the reference blows. The specific action of the ore is therefore appreciable and is of the same order of magnitude (20%) as that resulting from conversion with ordinary air (H.P.N. process).

(*c*) Another way of decreasing the nitrogen content is to introduce into the converter an element such as a carbonate, which is decomposed by heat, evolving a gas and a solid product which itself enters into the composition of the slag. From this point of view, limestone is the most advantageous, as apart from the fact that it is cheap, it produces sulfur-free lime. Moreover, to the dilution effect caused by the carbon dioxide is added a certain decomposition of the dioxide into monoxide and oxygen, which reduces the conversion time.

Table 1

	N_2	P	S
Reference Thomas steel	0.0100%	0.049%	0.031%
30% O_2 + scrap	0.0083%	0.041%	0.030%
30% O_2 + iron ore	0.0060%	0.037%	0.032%
30% O_2 + ore and limestone	0.0040%	0.032%	0.027%
O_2 + steam	0.0018%	0.035%	0.025%
O_2 + steam + double slag	0.0021%	0.017%	0.023%

In order to ascertain the effect of the limestone, various tests were made with ordinary air charged with increasing concentrations of limestone (castine), while decreasing proportionally the amount of lime and scrap. Figure 2 gives an experimental curve showing the variation of denitruration as a function of the amount of limestone charged per ton of pig iron. Admitting that some twenty kilograms of limestone per ton of pig iron has no effect (on losses by spitting, premature or tardy superficial action) the experimental curve of denitruration coincides exactly with Sieverts' law which Sawyer had introduced as early as 1923 in the study of nitrogen in steel. It may be said that limestone acts mainly on the nitrogen content by reducing the partial pressure of that gas. Moreover, it may be said that limestone has an additional and specific denitrurating effect, in the sense that its action is independent of the other procedures utilized at the same time to decrease the nitrogen content (oxygen-enriched blast, addition of ores, etc.).

Thus, by using simultaneously a 30% oxygen-enriched blast and ore while replacing in the charge 750 kg. of lime by 1,200 kg. of limestone, an additional denitruration of 35% was obtained as compared with the tests using oxygen-enriched air and iron ore. Proceeding in this way, a Thomas steel has been prepared having 40 ten thousandths of nitrogen, that is, a nitrogen content similar to that of Martin steels.

As regards phosphorus, it has been shown that by enriching the air with oxygen and by using scrap, ore or limestone, the content of that element is around 35 thousandths, that is to say it is lowered 20–30% as compared with the reference blows. In order to decrease the content still further, it is necessary to resort to a double slag process.

Very many experiments made along these lines have led CNRM to establish a very simple formula for current production of steel containing about 20 thousandths of phosphorus. This formula consists in adding sodium carbonate (5 to 10 kg. for one ton of steel) at the end of the operation, after having drawn off the calcic slag. The converter has to be erected for only 30 seconds in order to mix the carbonate with the steel and obtain a considerable dephosphorization. It may be remarked that this formula is at present being used on an industrial scale when needed in Belgium and Luxembourg.

From an economic point of view, CNRM has shown that the enrichment of the air with oxygen makes it possible to melt an excess of scrap and ore and to reduce by about 40% the time required for conversion. This increases the reproductivity of a converter. For example, with a 30% enrichment the productive

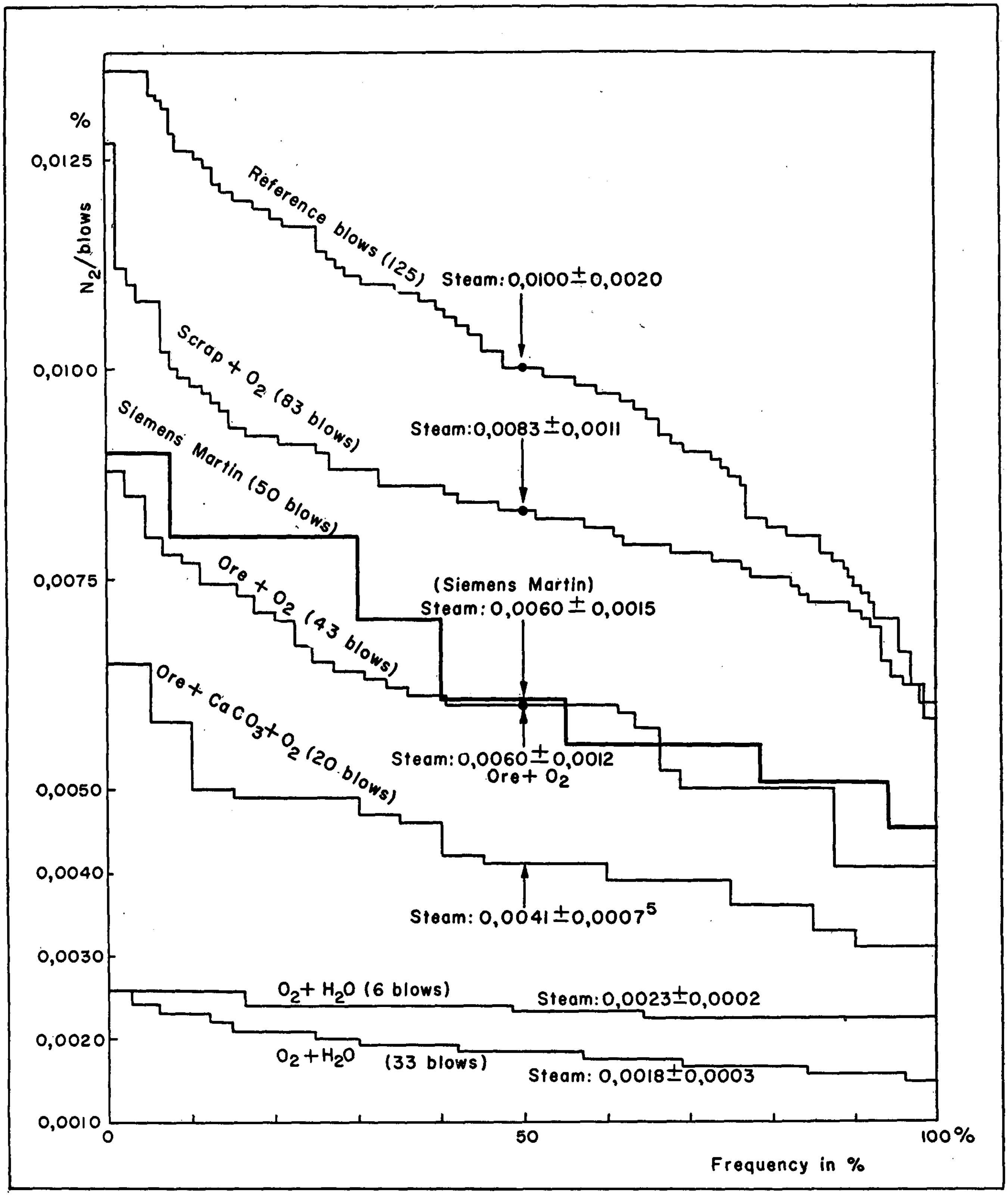

Figure 1. Results of tests using oxygen-enriched air.

capacity can be increased from 20 to 25%. This is a particularly important advantage at the present time; but it was theoretically demonstrated 25 years ago, and had already been experimentally verified by several researchers (3).

Another favourable circumstance is the fact that with oxygen-enriched air, it is possible to work pig iron of chemical composition differing from that required for satisfactory conversion with an ordinary blast. In particular, it is possible to blow pig iron relatively poor in thermogenic elements such as Si and P.

CNRM has calculated that, by the use of air enriched with 30% oxygen, it would in principle be possible to convert into steel pig iron with contents

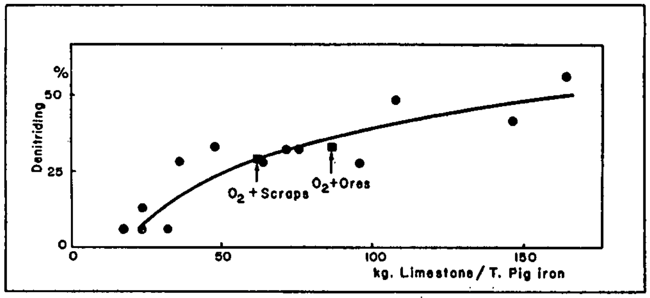

Figure 2. Experimental curve showing variation of denitruration.

of 3.6% carbon; 0.3% of Si; 0.1% of phosphorus. This may have outstanding economic consequences, all the more since steelmen have to deal with types of ores having compositions differing from those to which they are accustomed.

Tests with a mixture of steam plus oxygen

If it is desired to reduce the nitrogen content of steel even more, more drastic processes have to be used, i.e., to effect the conversion with mixtures of oxygen and a gas containing no nitrogen. The gas which is most advantageous from the economic viewpoint is superheated steam, the use of which had already been proposed as a method for controlling temperature during the course of conversion.

The first tests undertaken showed at once that the use of oxygen-steam mixtures lessened spitting, lowered the nitrogen concentration in the steels about 20 ten-thousandths, and shortened the conversion time by 40 to 50%. Furthermore, the quantity of scrap added could be increased to an extent that depended on the proportion of oxygen and steam chosen.

After numerous trials, CNRM has adhered to two methods of elaboration:

The first aims at reducing the total consumption of oxygen by using the mixture only during its period of maximum efficiency as far as nitrogen elimination is concerned, i.e., from the middle to the end of the conversion. The first part of the operation is conducted preferably with the use of a 30% oxygen-enriched air, which permits the charging of considerable additions of scrap or ore.

The second method aims at reducing the losses caused by spitting, by using the mixture shortly before these appear, or more simply, by using the mixture exclusively. This also offers the possibility of using pig iron of chemical composition differing from that required for the good operation of the Thomas process with ordinary air.

In particular, the working of the blast furnaces can be modified to obtain, for example, irons rich in Si but with improved desulfurization. It follows, therefore, that use of a mixture of steam and oxygen opens up vast technical and economic possibilities.

The phosphorus content of the steel produced by one or the other method is around 35 thousandths; that is to say similar to that obtained in the tests with oxygen-enriched blast. To reduce that content, it suffices to resort, near the end of the operation, to the double slag process with sodium carbonate, which was mentioned earlier. In fact, experience has shown that by combining the double slag process and that of the oxygen and steam mixture, it was easily possible to produce Thomas steels with 20 ten-thousandths of nitrogen and 20 thousandths of phosphorus.

Physical tests

The steels obtained with either oxygen-enriched air or steam oxygen mixtures, have been submitted to very many physical tests in laboratories and in factories. They may be summarized as follows:

As far as ductility at room temperature is concerned, the very object in view, the CNRM formulae considerably improve the behaviour of Thomas steel. They permit of the manufacture of Thomas steels of a quality at least equal to that of Martin steels.

Figures 3 to 6 give some examples of hardness curves corresponding to the several CNRM elaboration processes.

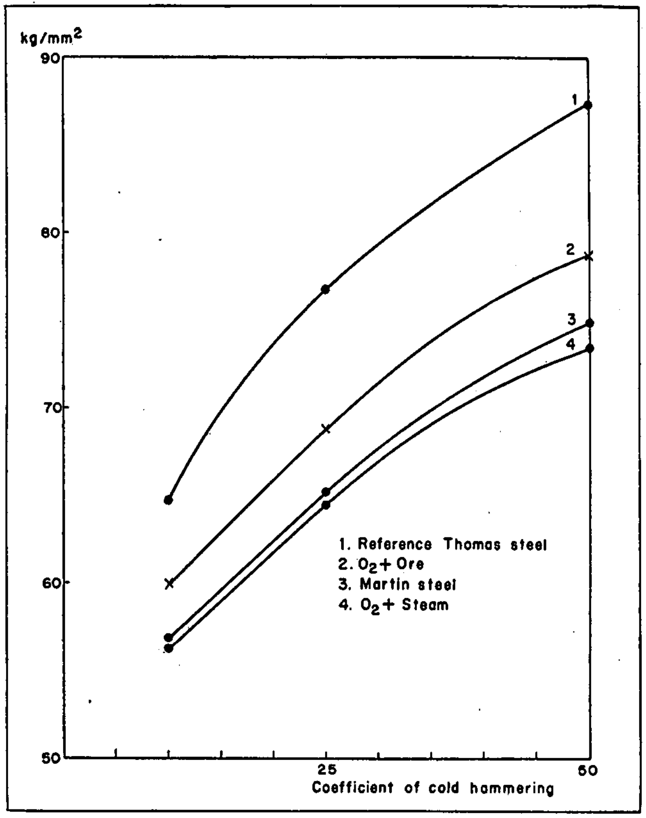

Figure 3. Hardness curves.

The curves in Figures 3 and 4 were obtained by plotting the variations in hardness of sheets deformed by simple compression, as a function of the coefficient of reduction. In the case of Figure 3 the initial thickness of the sheet was 1.5 mm. After cold deformation they were artificially aged. In the case of Figure 4 (Table 2), the original thickness was 2.5 mm., obtained by hot rolling in a semi-continuous mill. After cold deformation, the samples were not aged.

It may be seen immediately that steels produced with the use of $O_2 + H_2O$ mixtures harden less rapidly than the reference steels, and therefore have better ductility.

The curves of Figures 5 and 6 were obtained by plotting the variations in hardness of the sheets

A STUDY OF
THE IRON AND STEEL INDUSTRY
IN LATIN AMERICA

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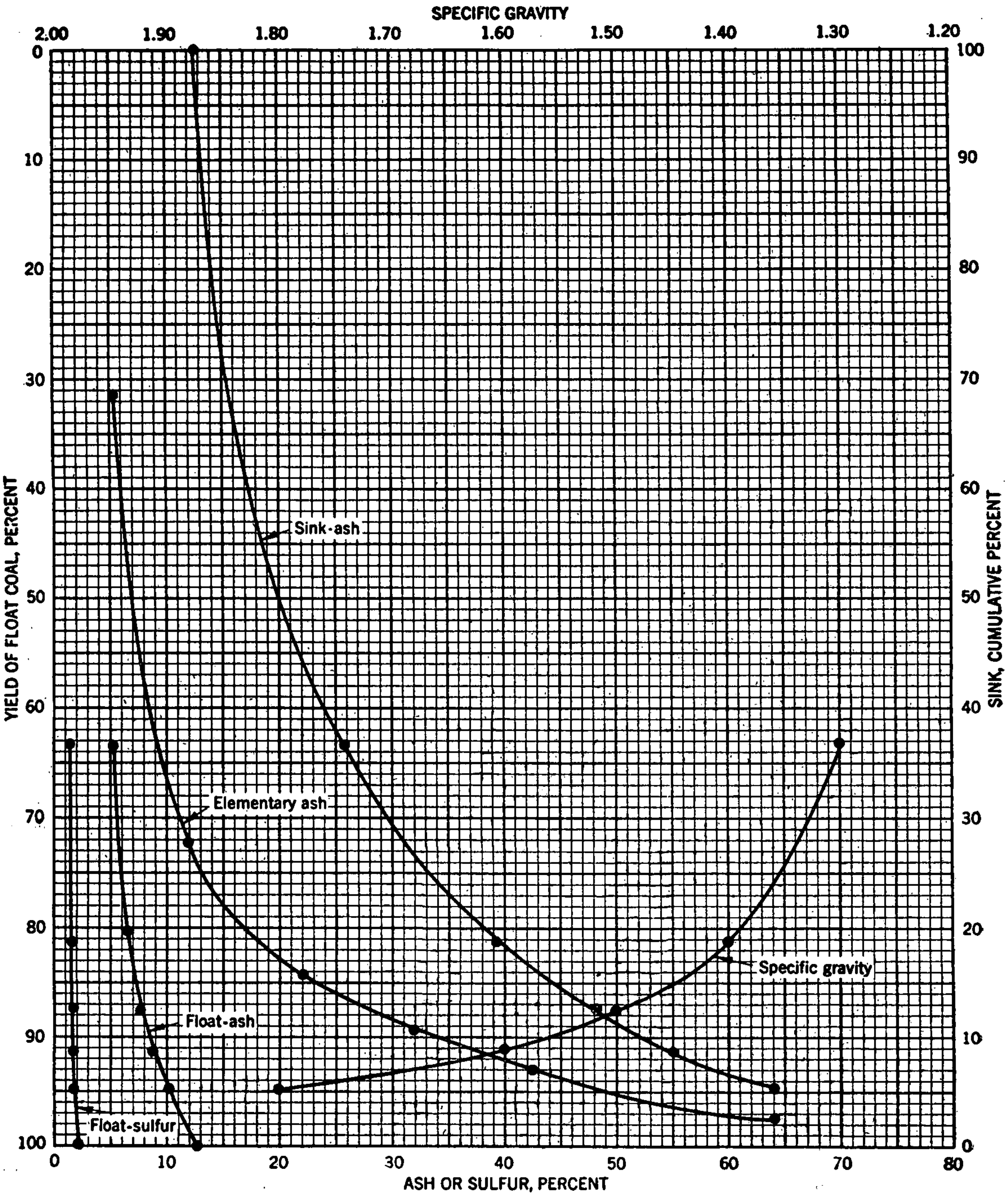

Specific gravity fractions	Elementary data				Computed cumulative data					
	Weight grams	Percent			Float, cumulative percent			Sink, cumulative percent		
		Weight	Ash	Sulfur	Weight	Ash	Sulfur	Weight	Ash	Sulfur
Float - 1.30		63.1	5.2	1.41	63.1	5.2	1.41	100.0	12.8	1.93
1.30 to 1.40		18.2	12.0	2.22	81.3	6.7	1.59	36.9	25.9	2.81
1.40 to 1.50		6.3	22.0	2.83	87.6	7.8	1.68	18.7	39.5	3.38
1.50 to 1.60		3.6	31.9	2.95	91.2	8.8	1.73	12.4	48.4	3.69
1.60 to 1.80		3.7	42.7	3.03	94.9	10.1	1.78	8.8	55.1	3.94
Sink - 1.80		5.1	64.1	4.60	100.0	12.8	1.93	5.1	64.1	4.60

Fig. 6. Washing characteristics of composite of 7 high volatile Cauca Valley Coals of Table 3.
Size 10 – 0.15 mm.

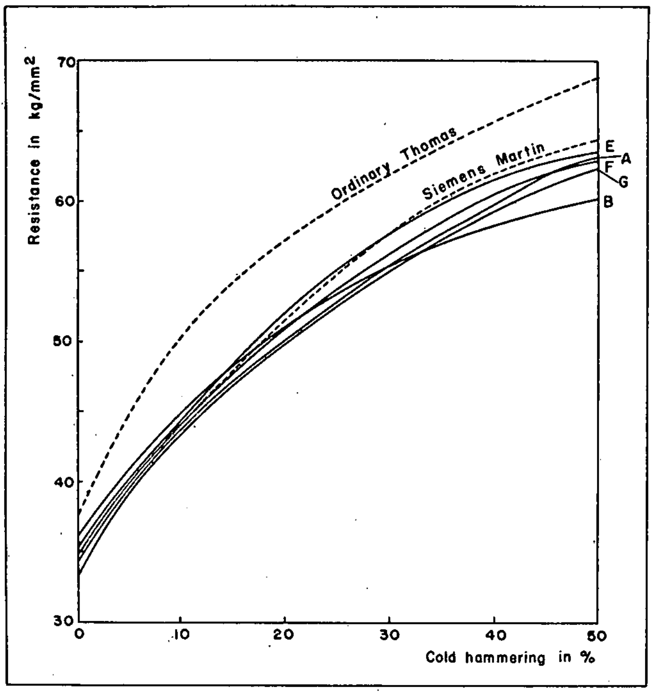

Figure 4. Hardness curves.

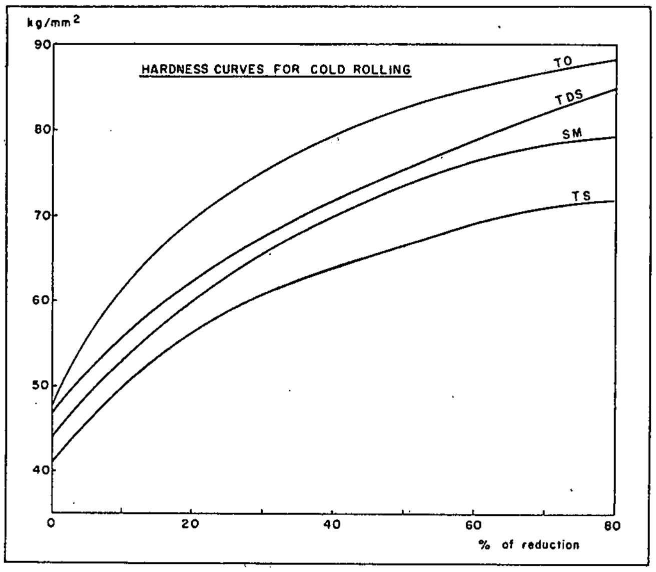

Figure 5. Hardness curves.

cold reduced as a function of the coefficient of reductions. Here, however, for the two figures the deformation was produced by cold rolling in a small laboratory mill (Figure 5, Table 3), and in an industrial rolling mill (Figure 6, Table 4). These curves fully confirm the above results.

The same applies to the curves of Figure 7, which show, according to the method of elaboration, the hardness of the sheets after hot rolling, cold rolling, annealing and finally, skin passing.

Figure 8 refers to wire-drawing and again corroborates the CNRM results.

Furthermore, the stamping properties of these steels have been studied by very numerous tests carried out in the laboratories and in the factories. For example, with Eriksen tests, the results given in Figure 9 were

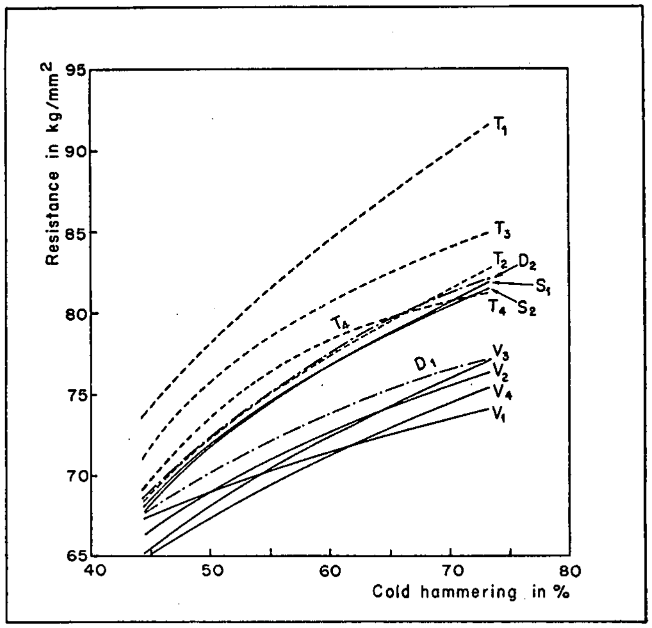

Figure 6. Hardness curves.

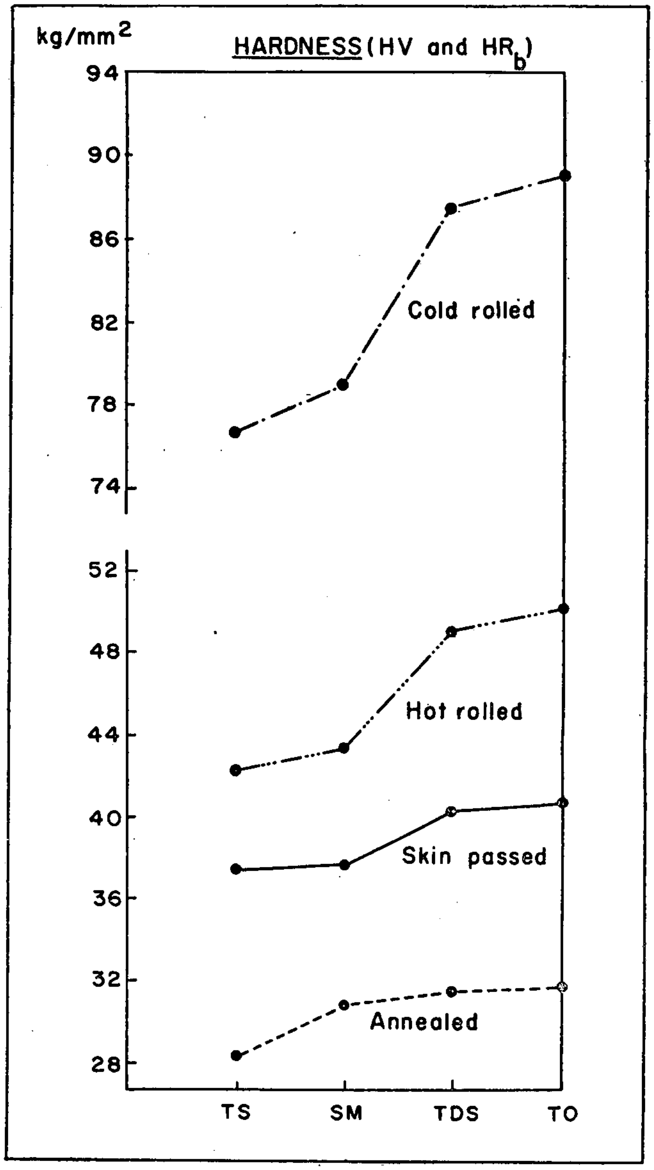

Figure 7. Hardness curves of sheets after different types of processing.

Table 2

	C	*S*	*P*	*Mn*	*Cu*	*As*	N_2
Thomas	0.034	0.027	0.035	0.27	0.065	0.042	0.0071
Reference	0.041	0.031	0.043	0.36	—	0.044	0.0074
Martin	0.040	0.024	0.007	0.26	0.054	0.045	0.0047
Reference	0.047	0.028	0.015	0.30	0.060	0.047	0.0078
A	0.033	0.026	0.020	0.21	0.065	0.044	0.0023
B	0.062	0.032	0.029	0.43	0.054	0.045	0.0036
E	0.042	0.028	0.013	0.40	0.059	0.045	0.0040
F	0.045	0.027	0.015	0.34	—	—	0.0026
G	0.037	0.025	0.015	0.25	—	—	0.0036

Table 3

ANALYSES ON SAMPLES DRAWN FROM THE MELT

Type	*Melt*	*C*	*Mn*	*S*	*P*	N_2	*P + 10 N*
Thomas: steam + oxygen: [T.S.]	1	0.05	0.32	0.022	0.017	0.0039	56
	2	0.052	0.33	0.018	0.015	0.0023	38
	3	0.049	0.43	0.026	0.022	0.0034	56
Siemens Martin: S.M.	1	0.07	0.37	0.030	0.010	0.0045	55
Thomas: double slag [T.DS.]	1	0.06	0.44	0.023	0.020	0.0092	112
Ordinary Thomas [T.O.]	1	0.07	0.35	0.033	0.044	0.0088	132
	2	0.07	0.44	0.030	0.044	0.0085	129
	3	0.07	0.49	0.024	0.046	0.0047	93

Table 4

	C %	*Mn %*	*P %*	*S %*	*N %*	*O %*
Thomas O_2—steam						
1. Blow V_1	0.065	0.44	0.027	0.020	0.0031	0.019
2. Blow V_2	0.055	0.36	0.032	0.029	0.0049	0.025
3. Blow V_3	0.08	0.56	0.034	0.033	0.0025	0.016
4. Blow V_4	0.06	0.41	0.017	0.020	0.0053	0.020
Ordinary Thomas						
Plant A						
5. Blow T_1	0.055	0.37	0.045	0.013	0.0104	0.021
6. Blow T_2	0.045	0.24	0.030	0.020	0.0074	0.030
Plant B						
7. Blow T_3	0.05	0.39	0.035	0.055	0.0088	0.024
8. Blow T_4	0.045	0.35	0.044	0.051	0.0070	0.022
Siemens Martin						
9. Blow S_1	0.05	0.27	0.011	0.025	0.0056	0.014
10. Blow S_2	0.05	0.34	0.011	0.030	0.0052	0.017
Duplex double slag						
11. Blow D_1	0.08	0.37	0.012	0.027	0.0092	0.012
12. Blow D_3	0.09	0.36	0.013	0.024	0.0084	0.007

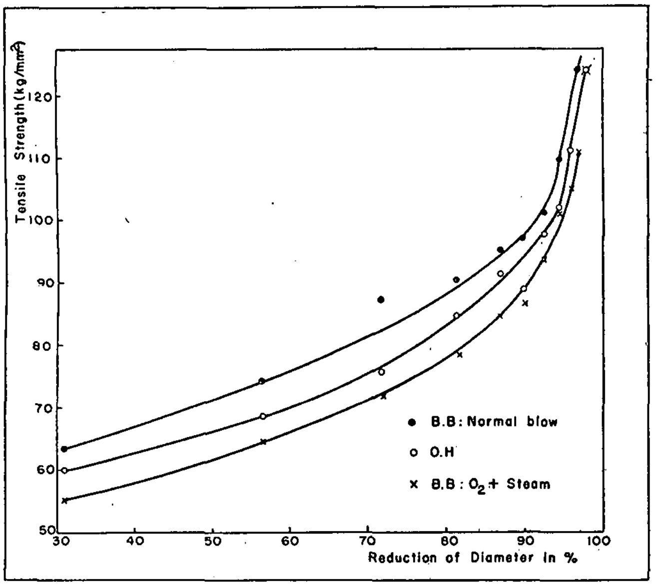

Figure 8. Results of wire drawing tests.

obtained, which show that the CNRM Thomas steels have stamping properties at least equal to those of Siemens Martin steel. The same conclusion may be drawn from kW.1 tests, and Figure 10 in particular shows the modulus of plasticity for the various samples tested.

Bending tests carried out with the Jenkins machine confirm the very good behaviour of CNRM "steamed" Thomas steel (Figure 11).

Practical tests by various clients such as manufacturers of nails, can-openers, metal container plugs, etc. have invariably confirmed the high ductility of this new steel.

Finally, with a view to reducing to figures the respective values of CNRM processes, the various results have been submitted to statistical analysis. This work involved more than 1,300 physical tests, as functions of over 500 several chemical analytical variations. It has shown, among other things, that the resistance to drawing and the hardness bore relation to the phosphorus and nitrogen concentrations

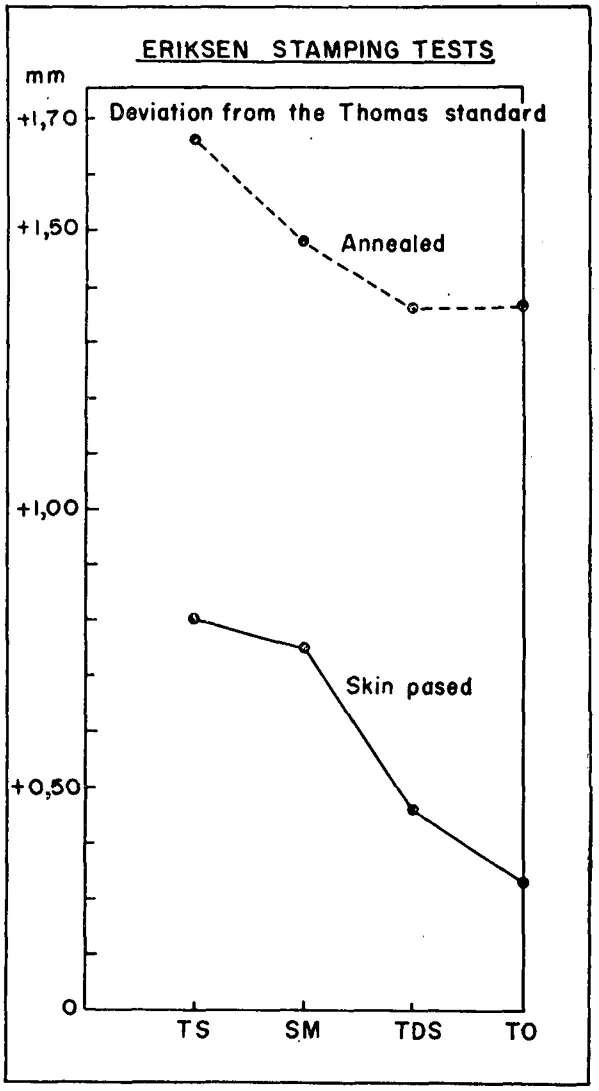

Figure 9. Results of Eriksen stamping tests.

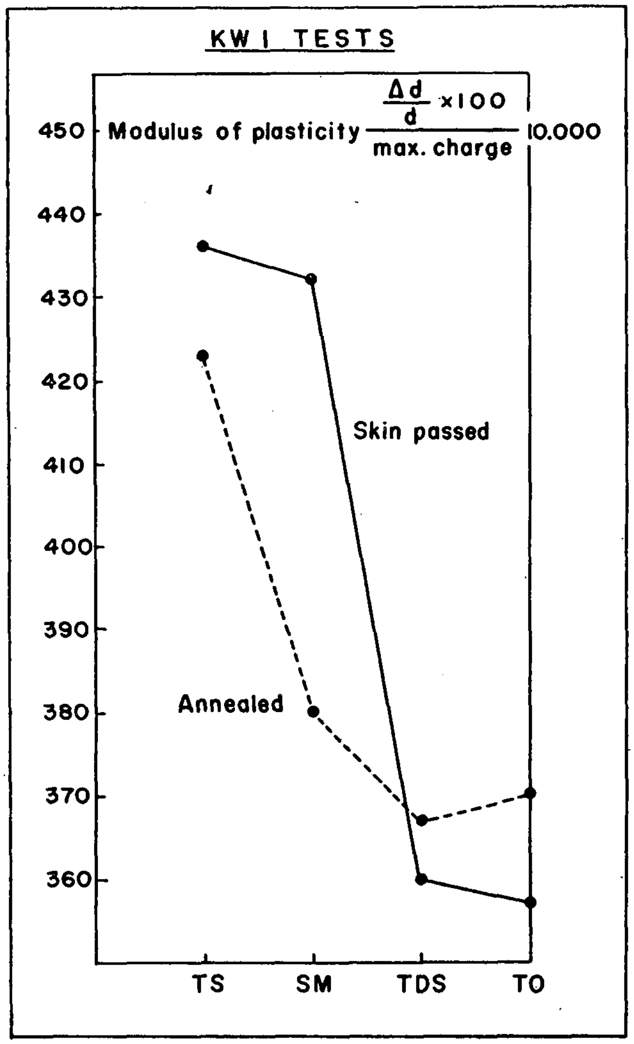

Figure 10. Modulus of plasticity for the samples tested.

by the formula P + 5 N, proposed by Stromeyer as long ago as 1917. It has shown also that the hardening, under the effect of deformation at room temperature or ductility, depended closely on the characteristic P + 10 N.

By applying these two symbolical formulae to CNRM's several elaboration processes, our experts have established the quantitative classification of Table 5, which is the numerical synthesis of the whole of the experimental research undertaken to evaluate the properties of our steels.

Table 5

	P + 5 N.	*P + 10 N.*
Reference Thomas steel	100	150
Thomas steel O_2 + ore	67	97
Martin extra soft steel	35	60
Thomas steel O_2 + steam	45	60
Thomas steel O_2 + steam + double slag	30	40

CONCLUSIONS

It follows from CNRM's tests that, from the point of view of the metallurgist, blowing with oxygen-enriched air or with a mixture of oxygen and steam permits of the production of Thomas steels with exceptionally low nitrogen and phosphorus content. The qualities of ductility of these new steels have proved to be excellent, and at least equal to those of Martin steels. From the economic point of view, these blowing formulae also afford remarkable possibilities. While ensuring an outstanding quality for the steel, they increase the capacity of the converters while retaining the principle of conversion via the bottom of the converter. They also permit the treatment of irons of abnormal chemical composition, which would be difficult to blow using the classical methods.

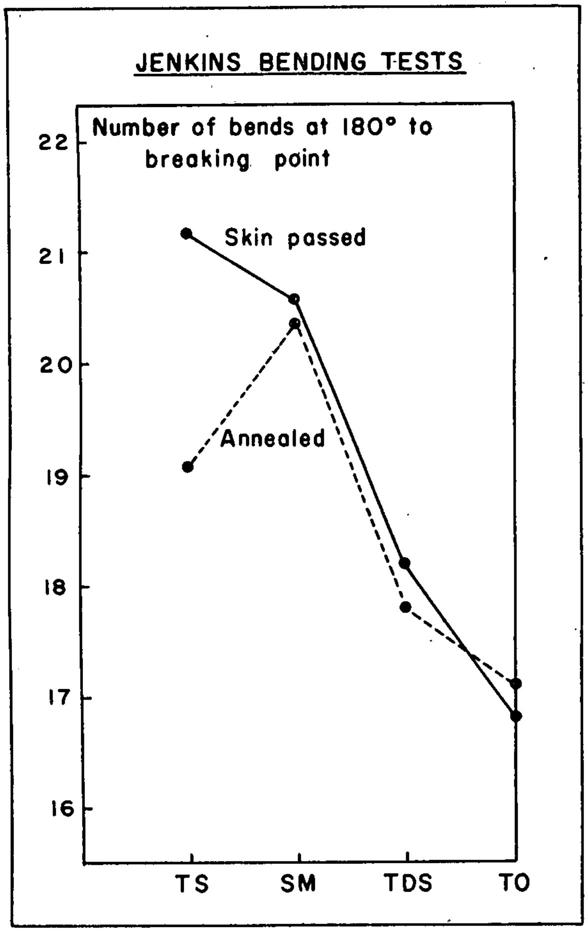

Figure 11. Results of bending tests on CNRM Thomas steel.

ACKNOWLEDGMENTS

It is a pleasure to point out that the results of these researches are essentially the fruit of teamwork in which the experts of Liège and Luxembourg steel plants participated, notably Messrs. Bosseler, Spautz, Metz and Langers, of the Aciéries réunies de Burbach-Eich Dudelange (ARBED); Messrs. Daubersy, de Radigues, Peeters, of the S.A. Métallurgique d'Espérance-Longdoz; Messrs. Grosjean, Marbais, Hotot, Poverman, of the S.A. d'Ougrée-Marihaye; Messrs. Van Campenhout, Nepper, H. Herbiet and Dor of the S.A. John Cokerill.

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Summary of Discussion

Acid Open Hearth Products and Their Specifications, *presented by the author*

Mr. FITTERER opened his presentation with a brief résumé of the salient points in his paper, drawing special attention to the types of products made by the acid open hearth process, as illustrated in the figures at the back of the paper. One of the things the paper had been designed to do was to show what was made, and who made it, but mainly what the specifications were and why they were made by the acid open hearth process. It therefore belonged elsewhere on the programme, but had been put forward in view of the speaker's departure. He explained that the acid process was considered best in the United States for castings. Small castings were made either by acid open hearth or acid electric, but for large castings acid open hearth was preferred, for the reasons given in the paper.

There were many different methods of drawing up specifications for castings and a complete list of specifications used in the United States was included in the paper.

In conclusion, he emphasized that although textbooks indicated that acid open hearth was difficult to operate economically because of the need for low phosphorus and low sulfur materials, there was no reason why high phosphorus or high sulfur steels should not be made by that process.

The CHAIRMAN thanked the speaker, and declared open the discussion on the three papers dealing with the acid open hearth process.

Mr. MERCIER remarked that he had been very interested in the papers by Messrs. Buell and Fitterer, the one dealing with the merits of basic open hearth and the other two with those of acid open hearth. Both had their respective uses, and he himself had had occasion to employ both systems in a French plant. The problem was not so simple in France, where the scraps were not so good, and acid open hearth scraps were much harder to find and much more expensive. However, it was his opinion that basic Martin steel could be made as good as that using the acid process.

He agreed with Mr. Fitterer concerning the flake formation on the castings, based on his own experience, and great precautions had to be taken in that respect with the basic process. It had been found in France that when castings of a certain resilience were required, it was easier to use the basic rather than the acid method, added to the fact that in Europe it was cheaper. He wondered whether that had been noticed by Mr. Fitterer. The acid method was not normally used in Europe for making special steels because the time element was longer and production costs were higher.

Mr. Kalling said that in Sweden some twenty years previously, the acid open hearth process had accounted for 40% of the total production, whereas currently not more than 15% was made by that process. It was still used, however, for most high-grade steels being produced in Sweden. Clean raw materials were required but currently some Swedish steel works were using the process on the basis of coke pig iron. The pig iron was desulfurized and the phosphorus was reduced by selecting coke with low phosphorus. Oxygen was used in Sweden to shorten the time of the heat in the acid open hearth process, and it was thought there that the use of oxygen was much more advantageous in the acid open hearth than in the basic open hearth process. It was being used just as the charge was melted down. For commercial steel the acid process could seldom compete with the basic one, but in comparing furnaces of the same size, the melting cost would be almost as cheap as in the basic furnace.

Mr. Fitterer, in reply to the questions raised by Mr. Mercier, said that there appeared to be general agreement that the purpose of the meeting was to state the advantages and disadvantages of the different processes. There had been no intention in any of his remarks of suggesting replacement of the basic open hearth by the acid open hearth. He had simply tried to state the use of that process and the possibilities opened up by the new trends occurring in the refining process.

With regard to impact strength, he said that a study had been made, making the same product by both acid and basic processes; identical types of samples were taken, and a difference had been found. The basic steel had higher impact properties, the reason being simply that acid steel was inherently large grained and no aluminium had been used in the acid heat. After comparing basic heats of the same grain size, it had been found that the impact strength was essentially the same.

He had been very interested in Mr. Kalling's remarks on the use of oxygen and his own experiences had been similar. Once the boil had started through the use of oxygen, the rate of carbon elimination continued, providing sufficient heat had been supplied to the bath. A similar phenomenon occurred in acid electric practice. At the start of a heat the refractories became hot, and as soon as the silicon and manganese had been eliminated and the boil started, the electrodes had to be raised out of the way of the boil because it was very violent, becoming so violent after some 5 to 10 minutes that the doors had to be opened.

Mr. Chavez said that in his view there was a very definite tendency in Latin America to consider the phosphorus in the iron entering the open hearth as an enemy instead of as a possible source of profit. He had two questions to raise: first, he wondered whether Mr. Saniter could tell him what percentage of phosphorus was required in the hot metal or in the scrap charge in the open hearth in order to use open hearth slag as a fertilizer; secondly, he asked Mr. Buell whether, in the plant in Canada where the hot metal contained 1.5% phosphorus, the slag had been used as a fertilizer, and if not whether there was any definite reason against so doing.

Mr. Buell, answering the second question, said that the iron ran about 1.7% phosphorus, but that the slag was not suitable for fertilizer because considerable quantities of fluorspar were used in processing and therefore the phosphoric acid was not soluble. He believed that in England and in the United States 12% soluble phosphoric acid was necessary in the slag if it were to be sold as a fertilizer.

Mr. Saniter said that the tilting furnace process to which Mr. Buell had referred at the previous meeting, and which was used in England, was very interesting from other points of view than fertilizers, because it was in fact a rival of the Thomas process and there was, in his opinion, very little to choose between the operating costs of the two processes. There was the benefit of producing low nitrogen steel. It seemed difficult to account for that fact, but the secret lay in the difference in the metallic yield of the two processes, namely under 90% for Thomas, compared with 102–104% for the tilting process, due to the large quantity of ore which was reduced.

On the phosphorus question, he remarked that an active mixer was used in that process in England, and the iron charge contained 1.6% phosphorus. The purpose was to remove the silicon, and the iron subsequently transferred to the open hearth furnace contained about 1.5% phosphorus; 75% of the hot metal was used and the resulting open hearth slag had a content of from 15 to 18% of P_2O_5. Fluorspar was not used, as the phosphorus was required in soluble form. In the ordinary costing system, instead of having a debit for slag disposal, there was a credit of five shillings per ton.

He wanted also to deal with one or two other points. First, for some years past, just a little oxygen had been used in the acid open hearth process in England. It saved 20 to 30 minutes per charge at a cost of 75/100 cu. ft. of oxygen per ingot ton. In so far as flaking was concerned, his experience had shown that acid open hearth steel was the least susceptible, followed by basic open hearth and basic arc electric, in that order.

Mr. Philbrook, referring to the phosphorus question, said that in the southern part of the United States, where the phosphorus content of the pig iron was 0.8%, a duplex process was practised which was semi-continuous. The furnaces were of about 300 tons capacity, but a heat of only 200 tons would be tapped, the other 100 tons being left in the furnace with the slag of the preceding heat. The furnace was then charged with alternating high blown and low blown Bessemer, and a kicker of high blown steel was added to start a high reaction with the metal in the furnace. New slag was made of burnt lime and iron oxide of some sort to finish the dephosphorization. The process continued for about a week, and periodically the furnace was drained dry for general repairs. It produced a slag high in P_2O_5 which was used as a basic soil conditioner.

Mr. Canguilhem referred to Mr. Buell's remarks on the cost of slag and believed it was considered economical even at 8%, although that was related to the question of transportation costs.

Mr. COHEUR said that three processes for producing steel had been described, all of which were good, but he believed from his own experience that each had limited applications. The one described by Mr. Philbrook was actually a combination of two, and in his opinion was an expensive solution, as either of the other two processes would be sufficient for the steel to be made in Latin America; nevertheless he would be interested to know Mr. Philbrook's views on the matter. There were limitations to the acid process in producing such items as tin plate, as he believed the phosphorus content was too high.

Mr. COSTA LINO requested Mr. Fitterer to explain the effects of rapid heat employed on the refractory material, and asked Mr. Buell to explain what kind of basic refractories were used.

Mr. FITTERER referred first to the question of tinplate, stating that it could not be made economically by the acid open hearth process. It would require special and costly raw materials. Referring to the high temperature and the high fuel input procedure, there had been no noticeable effect on the roof life. The effect on the bank and bottom refractories was actually advantageous. The erosion of any acid lining was not only a matter of chemical attraction of the iron oxide for the refractories but also of time. Thus the faster a heat was made, the less erosion.

Mr. BUELL, referring to basic refractories, said that most of the plants were using basic brick to some extent. The practice was proceeding slowly and the full basic end was finding some favour in certain quarters, but there was a current tendency to omit the end roof and the suspension on the end walls. The zebra roof was also being used considerably in some plants, but as was usual in the steel industry, there had been some controversy about its value.

Use of Phosphorus Ores for the Production of Pig Iron for Transforming into Steel by the Thomas Process, *presented by Mr. Allard in the absence of the authors*

Mr. ALLARD prefaced his presentation by explaining that the paper described a well known process for compensating a lack of phosphorus by the addition of natural phosphates; it was an interesting process for those companies having high phosphorus in their ores, but insufficient for use with the basic Bessemer process.

Perrin Process for Converter Steelmaking, *presented by the author*

Mr. ALLARD, after presenting his paper, added that the Perrin slag aluminous process was in operation at the famous old John Cokerill works (near Liège, Belgium) in order to obtain fine forged steel from basic instead of acid open hearth.

Thomas Steel with Low Nitrogen and Phosphorus Contents, *presented by the author*

Mr. COHEUR began his presentation by stating that the paper was a very brief account of the research undertaken by the Centre national de recherches métallurgiques, the main purpose of such research being, not to improve the quality of basic Bessemer steel, but rather to increase its field of applications, since the resultant Thomas steel was as suitable as other types for many uses such as free cutting steel, rails, structural steel and so forth. For other uses where great ductility at room temperature was required, it had seemed advisable to increase the mechanical properties of the steel.

After a brief summarization of the main points in the paper, he explained that two Belgian companies were currently commencing to apply the new processes (enriched air and mixtures of steam and oxygen) on an industrial scale, and a third would begin to do so at the end of 1952, in addition to a German company. During 1953 two Luxembourg steel companies, one in the United Kingdom and a fourth Belgian firm would also apply those processes, which was proof of the confidence that eight European steel companies had in the new methods of blowing described in the paper.

Mr. GONZÁLEZ VARGAS remarked that he had had the opportunity of observing a similar process a month or so previously in Austria. The plant concerned had an experimental 10-ton converter which had been in operation with very good results for several months. It was working with iron containing about 0.2% phosphorus and the resulting steel had between 0.015% and 0.020% phosphorus, some 50 to 55 cu. m. of oxygen being used per ton of steel. The oxygen was very cheap, as in Europe a cubic metre cost approximately the same as 1 to 1.1 kWh. The oxygen was injected through a refrigerated tube with copper point and a tuyere of Laval type at a pressure of about 12 atmospheres. The process took from 15 to 18 minutes, with losses from the charge to the casting of ingots of about 12% iron. It was very important in that process that the phosphorus be burnt before the coal, which did not therefore require over-blowing, so that once the flame subsided, the steel was ready.

Experiments were being conducted in Austria with very low carbon steels, about 0.04%, and the results were very promising. Experiments were also being carried out with a view to de-phosphorizing high phosphorus iron. There was great interest in that aspect in Mexico, and attempts were being made to obtain the co-operation of the plant concerned in order to have some tests made with Mexican ores.

Mr. COHEUR remarked that he was familiar with the Linz-Donawitz technique described by Mr. González Vargas. The results had been explained at a congress in Austria in 1951, and had, furthermore, been published in *Stahl und Eisen.* There was one point he would like to stress, however, and that was that the iron produced in Austria was of a particular type and the process could not be used with current Thomas pig. As to the phosphorus content, it could not be entirely eliminated by that process and very often more was left than was desirable. If an extra mild steel were required, losses in iron slag were somewhat excessive. It had been precisely with a view to overcoming those difficulties that Dr. Springorum had developed a process very similar to the Linz-Donawitz process.

He availed himself of the opportunity to announce that the Centre national de recherches métallurgiques, in collaboration with the Societé francaise de métallurgie, the British Iron and Steel Institute and the Verein Deutscher Eisenhüttenleute, was organizing a meeting in Liège from 7–9 May 1953, where processes for improving Thomas steel would be discussed. He cordially invited all participants to attend that meeting; every metallurgist naturally considered his own process to be the best, and that meeting would afford a good opportunity for comparisons.

Mr. KALLING said that as Mr. Coheur had spoken of metallurgists who all found their own methods to be the best, he would take the opportunity of adding a new method to the list of processes for converting pig iron into steel, as he considered that it offered good possibilities. At Domnarfvet, in recent years, experiments had been made using a rotary furnace for converting basic Bessemer iron into steel and the results had proved very promising indeed. The converter used was small, with a three-ton capacity, rotating up to 40 revolutions per minute. It had been found that by using that furnace, a much lower iron content in the slag resulted than by using other methods. The considerable loss of iron as smoke in the exhaust gas, usually obtained when blowing with pure oxygen, was almost negligible. Another point was that it was very easy to blow ordinary basic Bessemer pig iron in that way, although it had previously been difficult. Large-scale operations were to be commenced shortly.

In connexion with Mr. Coheur's statement that one of the advantages of using oxygen in basic Bessemer steelmaking was the possibility of handling pig irons also relatively low in phosphorus, it could be pointed out that the phosphorus content in the pig iron could easily be regulated by returning part of the basic Bessemer slag to the blast furnace. At the same time the lime and manganese content in the slag were made useful in the furnace. In Domnarfvet the phosphorus content of the pig iron, which was normally about 1.8%, would have been only about 0.7% without circulating the slag.

Tests had been made for many years at his plant, using a combination of oxygen and CO_2 in the blowing of basic Bessemer steel. Using that mixture only during the after blow, the nitrogen content was reduced to a very low figure. The use of a water vapour might be slightly cheaper, but to be certain of obtaining a hydrogen-free steel, the combination of oxygen and CO_2 might be preferable. Domnarfvet should be added to those plants using oxygen in the basic Bessemer in continuous operation.

Mr. CANGUILHEM wished to make a comment on the Ilva paper. Within two or three years, the Chilean steel plant would be using a 58% iron ore with 0.25% phosphorus, so that the pig iron would have about 0.45% phosphorus, and an alternative under consideration was the addition of natural phosphates to the charge of the blast furnace, as was apparently being successfully done in Italy. He knew that a great effort had been made to obtain the paper in question for the meeting. The situation in Chile was a special one, as there was a great shortage of phosphate fertilizer, and the natural appatites mostly contained a high proportion of iron, from 8 to 10%. There were large resources of such appatites so that their use in the charge of the blast furnace might be a solution, as they could not be used directly for making superphosphate because of their high iron content.

Mr. ALLARD pointed out that as he and Mr. Kalling had remarked, where the phosphorus content was about 0.2% the best method was to recirculate part of the Thomas slag. It was not very expensive as the Thomas slag contained some limestone and there was also the possibility of recovering iron and manganese from it, the iron content of that slag being about 10%, and manganese of the order of 3 to 6%. It was sufficient to consider the expense in coke consumption for reducing phosphorus and it was only a very small amount. The difference between the Thomas slag and the limestone having the same lime content, i.e., about 50%, was about 20 or 25 kg. more for 100 kg. slag refluxing than for 100 kg. limestone, so that exact calculations could be made.

Mr. CANGUILHEM thanked Mr. Allard for his explanation. Undoubtedly there were clear advantages in the alternative proposed, but the conditions at his plant might result in the process used in Italy being more convenient.

STEEL MAKING AND FINISHING

Comparison of economics of different processes of steelmaking (*continued*)

24 October 1952—Morning

Chairman:

Alfredo Gonzalez BALLESTEROS, Assistant Superintendent, Departamento de Altos Hornos, Compañía Fundidora de Hierro y Acero de Monterrey, S.A., Monterrey, Nueva León, Mexico

Contributed Papers:

Steel Production in Latin America
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The Balance of Materials and the Economic Comparison of the Different Steelmaking Processes
Ernst KREBS, Hüttenwerke Rheinhausen A.G., Rheinhausen, Germany

Comparative Investment Costs for Different Steelmaking Processes
Charles F. RAMSEYER, President, Ramseyer & Miller Inc., Consulting Engineers, New York, U.S.A.

Details of a Steel Plant for a non-Steel-producing Country
André MERCIER, Directeur de la Societé d'etudes et d'entreprises sidérurgiques (S.E.E.S.), Professeur a l'Ecole centrale des arts et manufactures, Paris, France

Summary of Discussion:

Participants: Messrs. BULLE, KREBS, WARING, SADA, LEUSCHNER, GONZALEZ VARGAS, VUCETICH, RAMSEYER, PRIETO, DECHERF, SCHERESCHEWSKY, COHEUR

Steel Production in Latin America

GEORG BULLE

1. POSSIBLE STEELMAKING PROCESSES

In the young countries of Latin America, most of the steel disappears at the consumers, and there is not much old iron to be picked up by the garbage scavenger. In the big industrial countries, the old cars, machines and steel structural parts are carefully salvaged and return as scrap to the iron and steel works for remelting in the steel furnaces. It is for this reason that in the big industrial countries scrap plays such an important part as a steel raw material (30–45%), while in Latin America probably 25% scrap at the most returns to the steel furnaces. Thus, nearly all the raw material for steel production in South America will be pig iron or ore—together approximately 85% of the steel produced. To date, steel can be made from ore only in the presence of pig iron as a reducing agent, i.e., in small quantities (maximum 15% of the pig iron tonnage), so that steel must be primarily produced from pig iron.

There are four methods in operational practice:

(*a*) Pig iron—ore process in the hearth furnace;
(*b*) Air-blow process in the converter;
(*c*) Air-blow process with oxygen in the converter;
(*d*) Duplex process: air-blow in the converter, after-melt in the hearth furnace.

(a) *Pig iron—ore process in the hearth furnace*

Scrap, limestone and ore are pre-melted in the hearth furnace and molten pig iron is added. With a violent reaction, this is converted under the oxidizing influence of the ore and possibly of the oxidizing flame (open hearth furnace) into steel, which is then finish-melted and refined in the furnace. Considerable amounts of slag are produced which are tapped partly during the heat and partly along with the steel. Molten pig iron of every kind can be used as raw material, but a certain manganese content (oxidation protection of the iron) is desirable, while the contents of silicon, phosphorus and sulfur should be as low as possible, as these elements slag uselessly and cost time and fuel. The ore used should be as rich in Fe and should contain as little gangue, especially silica, and accompanying undesirable elements (phosphorus, sulfur, etc.) as possible. Any amount of scrap and limited amounts of solid pig iron can be added. The steel produced by the pig iron—ore process is excellent from the quality standpoint, but steels with special properties should be produced preferably from scrap. As a rule, an open hearth furnace is used, generally one of the stationary type (seldom a tilting one), and it is heated with oil or gas (natural gas, coke-oven gas, producer gas, blast furnace gas, the latter only in mixture with other gases). Theoretically, the normal electric arc furnace could also be used, but due to operational difficulties (foaming, electrodes, etc.), this process has not been developed. Hearth furnaces of other types, e.g., rotary furnaces, rocker furnaces, the pot furnace with oxygen now in process of development, might also perhaps be made usable for the process.

(b) *Air-blow process in the converter*

Some burnt lime is charged into the hot converter and molten pig iron is added. Air is then blown through the metal bath in the converter until the pig iron is converted into steel. Except where they escape in the form of gas (carbon), the impurities along with lime and parts of the lining form a slag which is run off. The pig iron is converted into steel in the converter and sometimes it is finished later in the ladle.

As raw material only molten pig iron is used, which should be as hot as possible and of specified composition, i.e., either:

(1) Pig iron with a minimum 1–2% Si and a maximum 0.08% phosphorus. For this pig iron the converter has a lining of sand, ganister, etc., that is to say, it is acid, and is known as the acid converter or Bessemer process; or

(2) Pig iron with a maximum 0.2%—0.6% Si and at least 1.6% phosphorus. For this pig iron the converter has a lining of dolomite or similar material, i.e., a basic lining, and is known as the basic Bessemer or Thomas process.

Scrap and ore can only be added in very small quantities. The steel produced by the converter process is suitable, from the quality standpoint, for making all kinds of commercial steel and rails. Only in some pertinent cases is it inferior to the hearth furnace steel as, owing to the method of production, it contains more nitrogen. Notwithstanding this characteristic, however, a number of European countries cover the bulk of their steel requirements, amounting to several millions of tons, from converter steel only. The converter employed in big-scale practice is of fairly uniform shape. The Bessemer converter is blown from below, through tuyeres inserted in the detachable bottom, and the Thomas converter is blown through an interchangeable holed bottom. The acid or basic lining (see above) is either built up of refractories or stamped out of plastics.

(c) *Air-blow process with oxygen in the converter*

If a converter is blown with oxygen instead of with air, less blast is required and consequently, the amount of escaping waste gas is lower. The amount of heat lost in the flue gas is reduced and the percentage of the high reaction heat remaining in the converter is so great that a surplus of heat is produced here while the normal converter process is always hampered by the temporary lack of heat. It is possible, without affecting the liquidity of the steel, to make heat-absorbing additions during the blow in order to improve the steel or to reduce costs. Up to the present, two variations of the oxygen converter process have been successfully applied:

(1) *Oxygen addition with enriched air:* Oxygen is added to the blast streaming into the converter from below, so that a big percentage of the air oxygen (at least one third, sometimes one half and more) is replaced by pure oxygen (98–99%). Scrap, ore, steam or carbonic acid is then added as a cooling agent. The run of the charge is shorter but is otherwise just the same as in a normal converter, only the flame is hotter, and thick red fumes are caused by vaporizing iron.

(2) *Oxygen only:* 99% oxygen is blown from the top through a tuyere into the pig iron bath in the converter. The bath is cooled with large quantities of scrap (30% or more) or ore. Thick red smoke pours out of the converter.

Theoretically, every type of pig iron can be used as raw material for this process. In practice, however, use has been made to date only of Thomas pig iron with 0.2–0.6% Si, 1.3–2.0% P and approximately 1.0% Mn. (until now only experiments) and of stahleisen with 0.2–1.0% Si, 0.1–0.3% P and 2% Mn, which are converted by the "oxygen-only" process, but can also be blown in the oxygen addition process. There is no reason, therefore, why pig iron, the composition of which lies within the range of the tried-out kinds of iron, should not be blown by the oxygen-addition process. Meanwhile, two oxygen converter plants have been running for two years and two more are being built. Recently, too, Thomas and similar kinds of iron which produce large quantities of slag, were used with success in the "oxygen-only" process.

It will be possible to eliminate the big amounts of slag by structural changes in the converter. F. H. Springorum will report on this question at length at the General Meeting of the Verein Deutscher Eisenhüttenleute in November. The steel produced in the oxygen converter is as good as the open hearth steel in respect to quality and is suitable for all ordinary commercial purposes as, unlike the normal converter steels, it cannot absorb nitrogen during the production process. In the "oxygen-only" process, nitrogen is not present and in the oxygen-addition process, the absorption of nitrogen is avoided by cooling with scrap, ore, steam or carbonic acid.

The converters used for oxygen practice are of normal design, except that the "oxygen-only" converter has no perforated bottom. New design proposals are still at the experimental stage.

(d) *Duplex process: air blow in the converter, after-melt in the hearth furnace*

The duplex process was born before any oxygen converter existed, the idea being to combine the high output of converter steel works with the quality advantages of the open hearth steel works. Moreover, it was possible to use pig iron of every kind in the converter without running the risk of the steel becoming spoilt or cold as for instance, Bessemer pig iron with a high phosphorus content or Thomas pig iron with a too low phosphorus content. For Europe the method has now lost some of its importance. The process is as follows: The pig iron is blown in the converter to steel, the requirements in respect of composition and temperature being considerably lower than in the case of normal production of finished converter steel. This steel is charged without any pretreatment with ferro-manganese or other de-oxidizing agents into the open hearth furnace or electric arc furnace, like a sort of liquid scrap, and is then converted into open hearth or electric steel. In order to eliminate the nitrogen absorbed in the converter, a stahleisen is added in the open hearth furnace and coal, if necessary, in the electric arc furnace. Pig iron of every kind can be used as raw material (in the converter) and scrap in any desired quantities (in the hearth furnace). The charging of ore is unnecessary. The converters and hearth furnaces used in the duplex process are of standard type.

2. COMPARISON OF THE PROCESSES

If the Latin-American countries have to choose between the steelmaking processes as described above, the decision will vary according to local conditions and will have to be re-examined from case to case. Apart from the purely metallurgical conditions, economic aspects will have to be considered. The latter forms the subject of a special paper by Dr. Krebs.[1] In this paper only the questions of output, construction type and layout, and investment costs will be discussed.

The output pattern is as follows:

1 60-ton open hearth furnace produces in the pig iron—ore process a maximum of 50,000 tons of crude steel per annum.

1 60-ton converter produces—when pig iron is blown with air—400,000 tons crude steel per annum.

1 60-ton converter produces—when pig iron and scrap are blown with oxygen—400,000 tons crude steel per annum.

1 60-ton open hearth furnace produces in duplex process (66% converter metal) 100,000 tons crude steel per annum.

1 60-ton electric arc furnace produces in duplex process (66% converter metal) 180,000 tons crude steel per annum.

A comparison thus reveals the following picture:

(1) *Pig iron—ore process in open hearth furnace*

Uses pig iron of every kind and scrap in any desired quantity, produces every kind of common steel in good qualities, has a low output, and therefore needs a big plant with high investment and operational costs. In addition, it consumes considerable tonnages of heating fuel. Charge costs may be cheap as both pig iron and ore (up to 15%) can be converted into steel.

(2) *Converter process*

Uses only pig iron of specified compositions. A special furnace must be provided for the melting of scrap. Produces most common steels but not all, owing to nitrogen occurrence, has a high output; only requires, therefore, a small plant with low investment and operational costs; requires no heating fuel, but may have an expensive charge (only pig iron, hardly any ore).

(3) *Converter process with oxygen*

Uses pig iron of all kinds, major quantities of scrap or ore; produces every kind of common (unalloyed) steel in good quality; has a high output and therefore needs a small plant with low investment and operational costs. The costs of the oxygen plant represent only a small increase in the steelworks investment costs; the increase in costs for the oxygen consumed is offset by the lowering of the converter operational costs (higher output, less slopping, etc.). No heating fuel is required. The charge is not expensive (scrap or ore as cheap additions).

(4) *Duplex process: converter—open hearth furnace*

Uses pig iron of all kinds and scrap in any desired quantity, produces every kind of common steel in good quality, has an output about twice as high as that of a normal hearth furnace plant, but the need of a converter for the first stage results in high investment and operational costs. Fairly large quantities of fuel or power are required and the costs of the charge are high (no ore addition).

3. LAYOUT OF THE STEELWORKS IN LATIN AMERICA

The steelworks in Latin America which are to be expanded and/or erected for the expansion of the Latin-American steel industry will probably be of considerable size, that is to say, they may perhaps be planned for a production of 300,000 tons of crude steel per annum (at the first stage of building) and for 600,000 tons of crude steel per annum (at the second stage of building). It would appear advisable, moreover, to locate them as a central plant within big integrated iron and steel works where molten pig iron would be available to them from the blast furnace or the electric pig iron works. In a line with the processes described in the foregoing, some four construction types would be possible for the new steelworks for the raw material conditions as envisaged (25% scrap, 85% pig iron and ore).

I. Open hearth steelworks (a number of open hearth furnaces) (Fig. 1);

II. Converter steelworks with acid or basic converters and a small hearth furnace plant (open hearth or arc furnaces) for the melting of scrap (Figs. 2 and 3).

III. Converter steelworks with oxygen-enriched air, with basic converters and a small hearth furnace (open hearth furnace or arc furnace) for steels with special properties (Fig. 4);

IV. Duplex steelworks with converters and hearth furnaces (open hearth or arc furnaces) in tandem (Fig. 5).

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[1] The balance of materials and the economic comparison of the different steelmaking processes.

The estimate of washery performance following the above procedure has given the series of projected performance data shown in Table 6. This table gives the estimates of yield, ash content of washed coal, and sulfur content of washed coal by sizes and, finally, the composite average of washed coal. These estimates have been computed, starting with a washing gravity of 1.60 for the 37–10 mm. size and following the general washing-gravity gradient, which is more or less characteristic of the Baum jig.

It is evident from the data on estimated ash and sulfur content by sizes (Table 6), that this is a case where the ash and sulfur of the washed coal are both higher than is generally considered good for the production of metallurgical coke. This indicates a study of the effect of rewashing of the fine coal-size to correct the effect of the washing-gravity gradient and to bring the fines down into line with the quality of the coarser sizes. Therefore, we have proceeded with a study of the results obtainable by rewashing the 5 mm.–0 fines in a separate operational step which constitutes, then, a modified Baum integral washing system and gives the facility for controlling the separation on the fines separate and apart from the primary washing operation.

Table 6

ESTIMATED BAUM-WASHERY PERFORMANCE ON CAUCA VALLEY COMPOSITE HIGH-VOLATILE COALS

		Washed coal		
Size range—millimetres	*Washing gravity*	*Yield*	*Ash*	*Sulfur*
37 – 18	1.50	89.5	9.3	1.6
18 – 10	1.55	91.6	9.8	1.7
10 – 5	1.65	92.1	9.4	1.7
5 – 2.5	1.80	94.2	10.0	1.7
2.5 – 1.5	1.90	95.2	10.2	1.7
1.5 – 1.0	2.00	95.9	10.7	1.8
1.0 – 0.1	2.15	89.1	11.3	1.9
Total and average		89.4	10.0	1.7
0.1 – 0 mm. raw		3.1	12.8	1.9
Total, including sludge		92.5	10.1	1.7

This estimate of performance on the 5 mm.–0 size has been made by applying the performance curves of a table-washing operation which does not differ substantially from that of the feldspar-jig, the hydrotator or the rheolaveur, which are other commonly used methods of treating material in this size range. The computations have been set up on the basis of 1.50 washing gravity for the 5–2.5 mm. size and following the characteristic washing-gravity gradient into the finer sieve-size range.

This computation, carried out on the data for sizes below 5 mm., results in the table of estimated end-products by sizes given in Table 7.

Table 7

ESTIMATE OF PERFORMANCE TO BE OBTAINED IN REWASHING 5 MM. – 0 PART OF WASHED COAL OF TABLE 6

		Washed coal		
Size range—millimetres	*Washing gravity*	*Yield*	*Ash*	*Sulfur*
5 – 2.5	1.40	78.7	7.1	1.6
2.5 – 1.5	1.50	85.8	8.0	1.6
1.5 – 1.0	1.60	89.4	8.1	1.7
1.0 – 0.1	1.75	85.0[a]	8.1	1.7
Total and average				

[a] Per cent of raw plant-feed of this size.

This will illustrate general application of the rewash modification of the Baum system. It might be duplicated by initial screening and separate washing of the 37–5 mm. size and the 5 mm.–0 sizes. Choice of these methods as compared with the simpler integral washing operation is a matter of economy that would require the introduction of cost factors and produce-use appraisals not available for this study. The above analysis of the performance angle of the operation is sufficient to illustrate application of the modified Baum-washing system to the moderately difficult coals that may be encountered in developing the Andean coalfields.

Barro Branco coal of southern Brazil

Figures 7 and 8 are typical washability-test data sheets of a sample of the Barro Branco coal. This sample is a mixture of the top coal bench and the bottom coal bench of the Barro Branco bed near Criciuma in the state of Santa Catarina, Brazil, which furnishes coking-coal for the National Steel Company of Brazil. (3)

This unique coal deposit has come to be known the world over as the classic of coal-preparation problems. Judged by the pre-heavy medium-criteria of washing difficulty, such as the Byron-Bird (4), scale set up on the basis of percentage of near-gravity material in the raw feed, this coal is beyond the range of practicability with a near-gravity percentage exceeding that designated "formidable". The yield—specific-gravity curves are straight lines showing nowhere any tendency even to an inflection point that would indicate the washing gravity of most economic separation. It presents an ash-reduction problem and a sulfur-reduction problem for, although it easily yields a sulfur production far beyond the possibility of any other known coal-bed, the residual float-sulfur is generally around the top tolerance accepted in current standards for metallurgical use. The float-ash content at washing gravities commonly used is so high that the washing problem immediately falls into the field of low-gravity separation requiring special methods. Although the most serious problem currently under consideration is ash reduction, it appears likely that, when introduction of heavy-medium processes brings about a substantial improvement in this regard, the sulfur problem will then become acute. The extremely high sulfur content of the raw coal discourages the fine-crushing approach to the general problem because of the difficulty of handling sulfur in the projected treatment of slimes by froth-flotation. The washability curves show that, no matter what gravity-concentration system is used, the yield of acceptable metallurgical-grade coal must be so small that secondary-coal disposal will always play a predominating role in the over-all economy of the preparation operation.

Notwithstanding all these technical difficulties, the phenomenally strong coking quality, adequate reserves, and strategic location of this coal, with respect to the tremendous deposits of exceptionally high-grade iron ore of Brazil, may well place the Barro Branco coal among the most important coking-coal reserves of the world in decades to come.

The Barro Branco coal-bed consists of two entirely separate and distinct benches; the upper bench or Coberta is separated from the lower or Banco bench by a thick, strong clay and shale band and a bony band called the Quadração. Generally, the upper bench

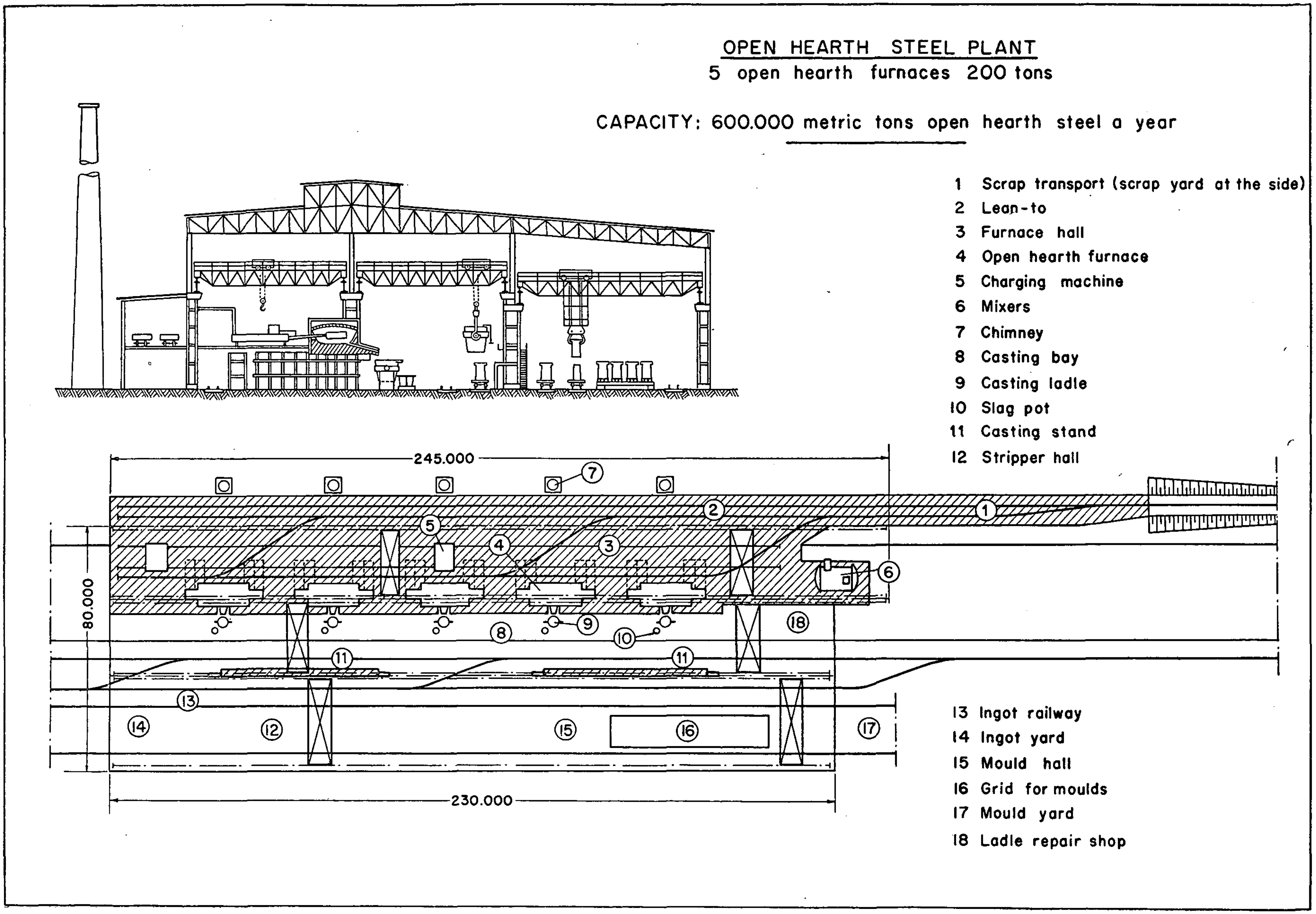


Figure 1. Lay-out for an open hearth steel plant of 600,000 tons annual capacity.

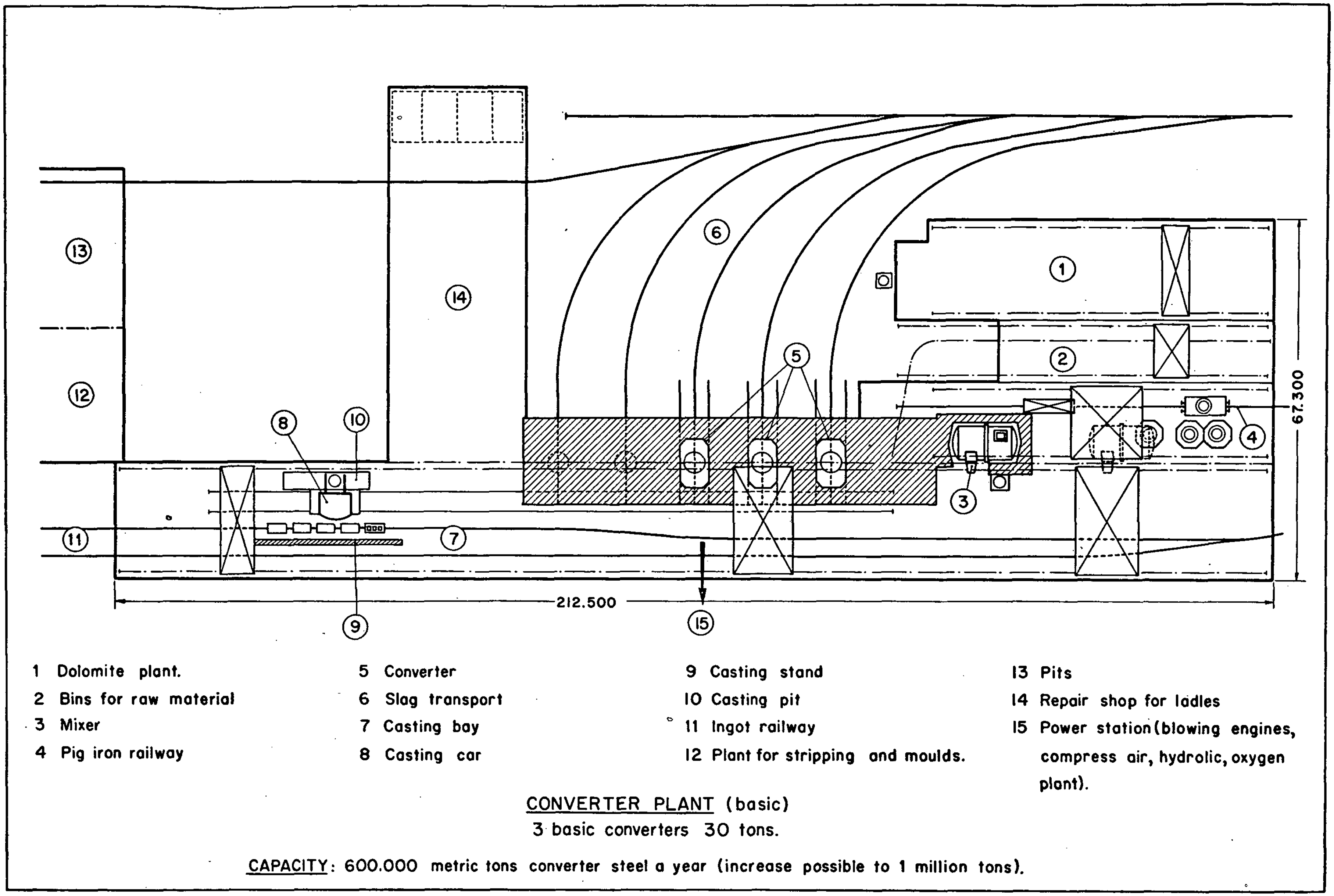


Figure 2. Lay-out for a basic converter plant having an initial capacity of 600,000 tons a year.

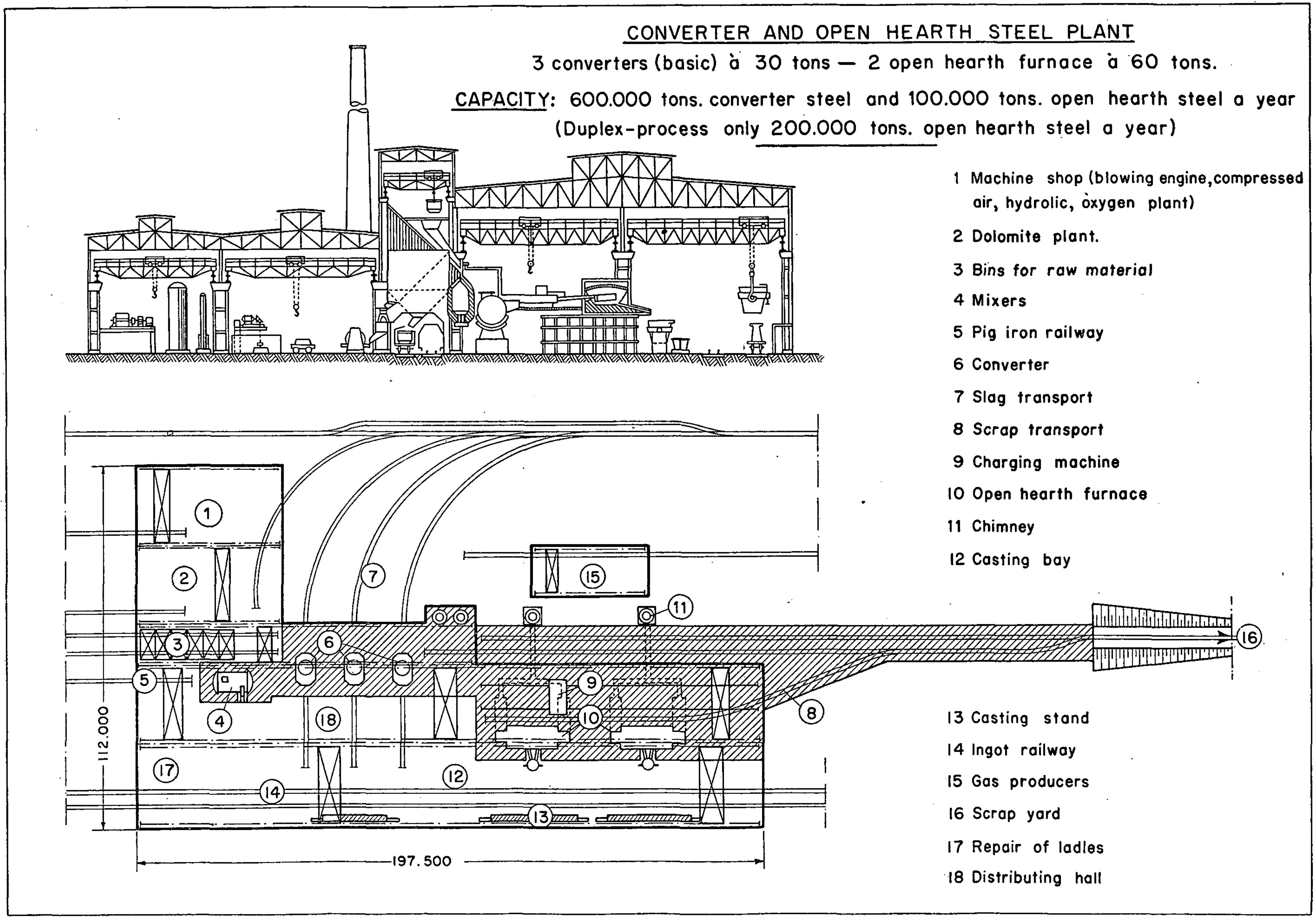
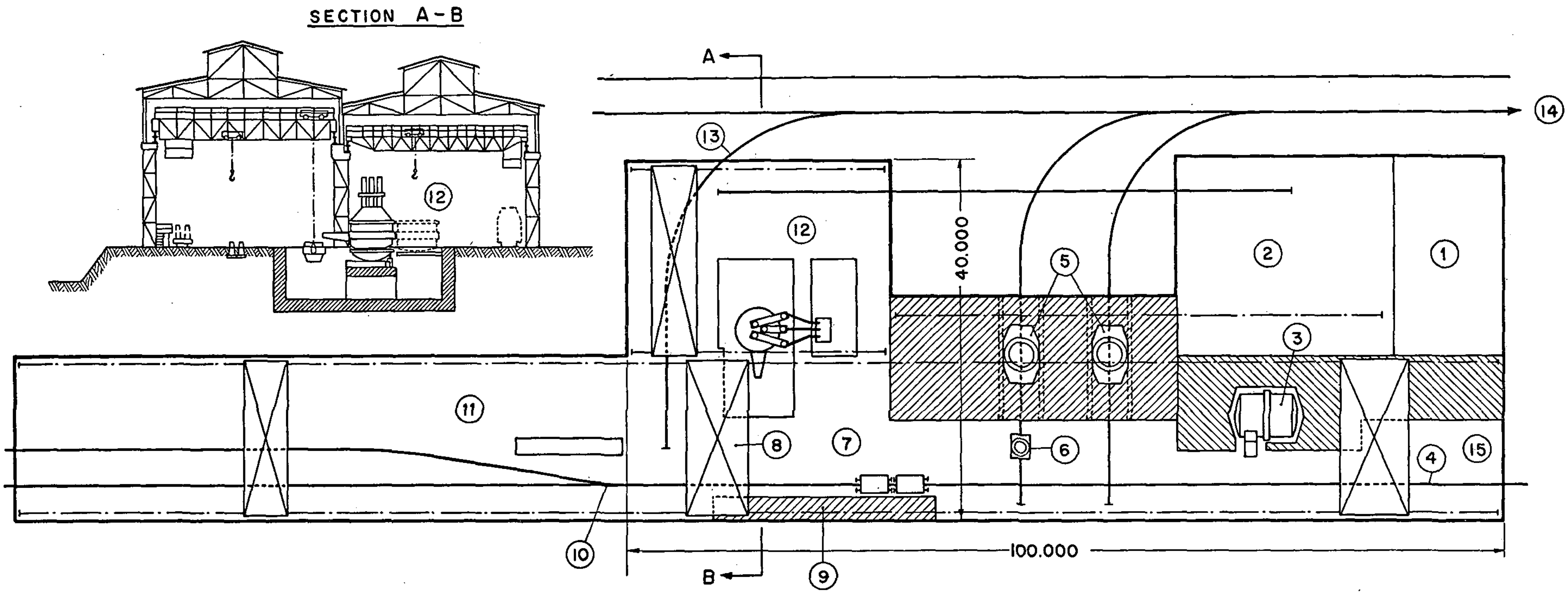


Figure 3. Lay-out for a converter and open hearth steel plant.



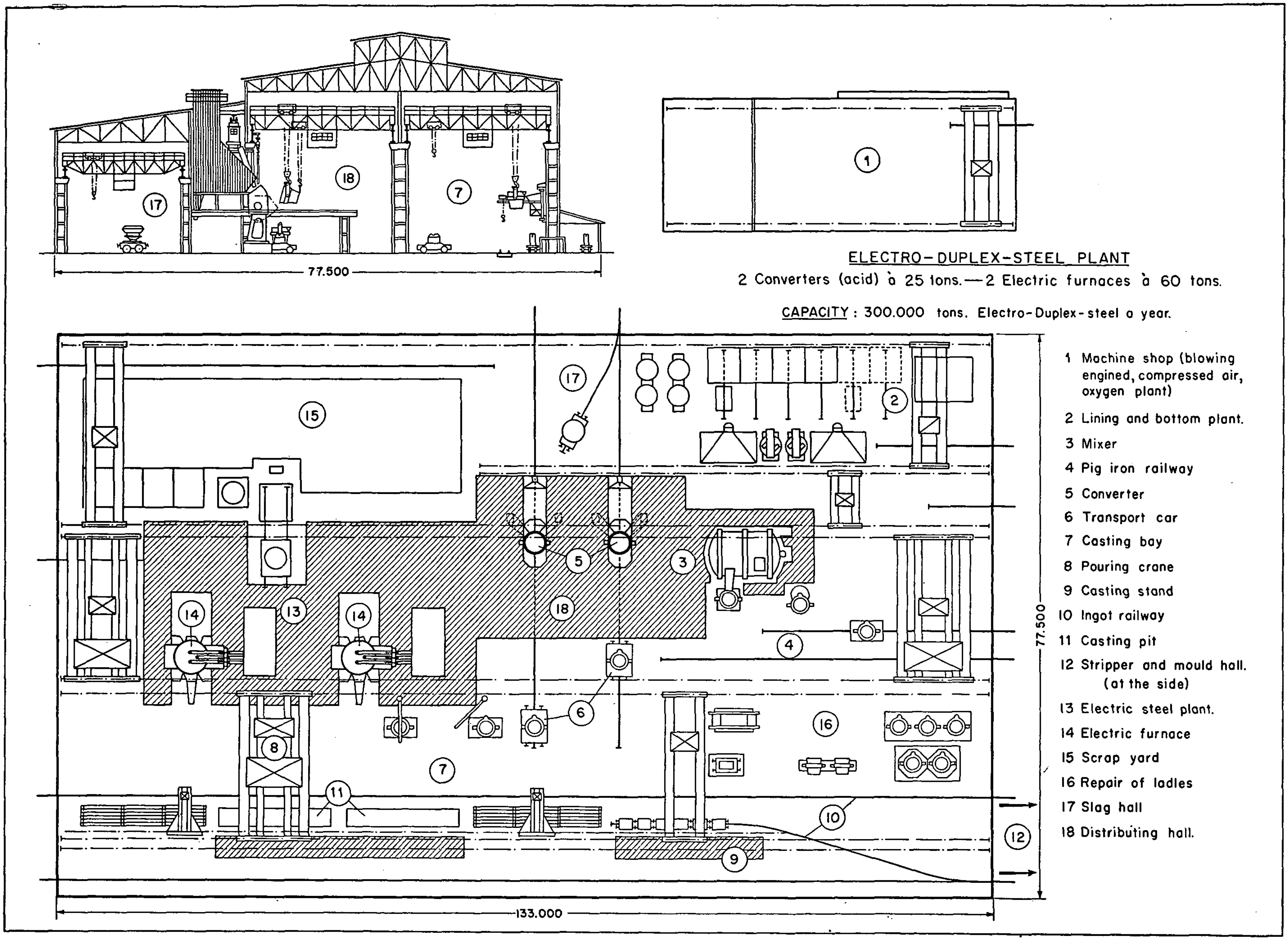
1 Power station (blowing engines, compressed air, hydrolic, oxigen plant.)
2 Dolomite plant.
3 Mixer.
4 Pig iron railway.
5 Converter.
6 Transport car.
7 Casting bay.
8 Pouring crane.
9 Casting stand.
10 Ingot railway.
11 Plant for stripping and moulds.
12 Hall with electric furnace.
13 Scrap transport.
14 Scrap yard at the side.
15 Ladle repair shop.

CONVERTER WITH OXYGEN, ELECTRIC FURNACE FOR SPECIALITIES

2 Converters (basic) à 30 tons — 1 Electric furnace à 20 tons.

CAPACITY: 400.000 tons. oxygen converter steel and 30.000 tons. special steel a year.

Figure 4. Lay-out for an oxygen converter with an electric furnace for special steels.



ELECTRO-DUPLEX-STEEL PLANT

2 Converters (acid) à 25 tons.—2 Electric furnaces à 60 tons.

CAPACITY : 300.000 tons. Electro-Duplex-steel a year.

1 Machine shop (blowing engined, compressed air, oxygen plant)
2 Lining and bottom plant.
3 Mixer
4 Pig iron railway
5 Converter
6 Transport car
7 Casting bay
8 Pouring crane
9 Casting stand
10 Ingot railway
11 Casting pit
12 Stripper and mould hall. (at the side)
13 Electric steel plant.
14 Electric furnace
15 Scrap yard
16 Repair of ladles
17 Slag hall
18 Distributing hall.

Figure 5. Layout for an electro-duplex-steel plant.

Table 1

STEEL PRODUCTION IN LATIN AMERICA

(300,000 tons commercial steel per annum)

A. *Steelworks plant*

	I	II	III	IV
Process	*Pig iron-ore-process* Open hearth furnace	*Normal converter process* for pig iron, open hearth furnace (arc furnace) for scrap	*Oxygen-converter-process* for pig iron; scrap open hearth furnace (arc furnace) for special conditions	*Duplex-process:* converter—open hearth furnace (arc furnace)
Plant	3 open hearth furnace of 200 tons	2 converters of 25 tons acid or of 30 tons basic, 2 open hearth furnace of 60 tons (1 arc furnace of 50 tons)	2 converters of 30 tons basic 1 open hearth furnace of 30 tons (1 arc furnace of 20 tons)	2 converters of 25 tons acid or of 30 tons basic 3 open hearth furnace of 70 tons (2 arc furnaces of 60 tons)
Raw material delivery				
(*a*) Pig iron	Rail transport from blast furnace with mixer wagon (car) or rail transport from low shaft furnace in pig iron ladle to 500-ton mixer in steelworks.			
(*b*) Scrap	Rail transport from scrapyard or rolling mill in boxes (or baskets for arc furnaces)			
Casting operation	Car casting with crane in casting bay	Converter steel transfer, car, then crane casting open hearth (electric) steel direct crane casting on ingot car	As in II	As in I
Stripping and ingot mould operation	Stripping bay with ingot mould operation			
Heating plant	Gas or oil plant	As I, but smaller (for electric transformer and switching station)	As in I but smaller	As in I but smaller
Mechanical	Conversion station	As I, in addition 2 blowers and possibly hydraulic station	As in I, in addition oxygen plant for 3,000 m^3/h, possibly	As in II
Auxiliary plant	Traffic facilities, quick laboratory, rest room, etc.			

Table 2

STEEL PRODUCTION IN LATIN AMERICA

(300,000 tons commercial steel per annum)

B. *Raw material consumption in tons per annum* (figures in brackets: arc furnace)

	I	*II*		*III*		*IV*
	O.H. furnace	*Converter O.H. furnace*	*Converter arc furnace*	*Oxygen converter special O.H. furnace*	*Oxygen converter special arc furnace*	*Duplex converter O.H. furnace (arc furnace)*
Pig iron						
For converter	—	232,000	(257,000)	254,000–246,000	(254,000)	233,000 (263,000)
For open hearth furnace	225,000	25,000		0– 8,000		30,000
		257,000		254,000		263,000
(For arc furnace)			(0)		(0)	
Duplex material						
For open hearth furnace	—		—			208,000
(For arc furnace)	—		—			(234,000)
Scrap						
For converter	—	—	—	75,000– 51,000	(75,000– 45,000)	—
For open hearth furnace	75,000	75,000		0– 24,000	—	75,000
(For arc furnace)			(75,000)		(0– 30,000)	(75,000)
Ore						
For converter	—	—	—	Part of the pig iron can be replaced by ore if the special furnace undertake the melting of the scrap		—
For open hearth furnace	50,000		—			—
			Steel production in tons per annum			
Open hearth steel	300,000	94,000	—	0– 30,000	—	300,000
(Electric steel)	—	—	(71,000)	—	(0– 30,000)	(300,000)
Converter steel	—	206,000	— (229,000)	300,000–270,000	(300,000–270,000)	
TOTAL	300,000	300,000	(300,000)	300,000	(300,000)	(300,000 (300,000))

Table 1 illustrates how a 300,000 ton steelworks can be laid out according to the four construction types. Figures 1–5 also show a number of steelworks for approximately 300,000 to 700,000 tons.

Table 3

COMPOSITION OF ORES IN LATIN AMERICA

Source: Mainly from G. Einecke "Eisenvorräte der Welt"

	Fe	*Mn*	SiO_2	Al_2O_3	*CaO*	*MgO*	*P*	*S*	*Other*	CO_2	*Wetness*	*Annealing losses*	*Remarks*
(1) *Argentina*													
Zapla (Jujuy)	46.2 [a]	—	18.7 [b]	4.8	3.2	0.2	0.7	—	—	—	0.9	1.4	100 m. tons
	49.7	—	15.1	7.0	2.5	0.4	0.65	—	—	—			
Sierra Grande (Río Negro)	55-65?		little				1.7?						
(2) *Brazil*													
Minas (Roteisenharterz)	60/70	1	0.3–2.7	—	—		0.3	0.1	0.10–75 Alk		—	—	—
Caue, Itabira	68.7	0.06	0.52	0.63	0.05	0.12	0.038	0.019	—		0.59–1	0.78 H_2O	Export ore 1948
Itabirite	46–43	0.18–16	30–36	0.6–0.46	—	—	0.011–17		0.10–11 Alk		0.4–0.46	—	—
Casa da Pedra	68.46	—	0.49	1.01	—	—	0.056	—	—		—	—	Used in Volta Redonda
Canga	63.5–57.0	0.2–0.06					0.2–0.06						
Santa Caterina Magnetite	55/67	?	11.5/1.5	1.8–0.4		0.8–0.2	0.03–0.02						
(3) *Chile*													
Tofo	67.5	0.2	2.5	—	0.0	0.0	0.112	0.12	0.8 TiO_2		—	—	60 m. tons
Algarrobo	64.5	0.2	6.4	—	0.5	0.18	0.189	0.04	—		—	—	120 m. tons
Romeral	60.0	0.07	9.74	2.38	1.75	1.0	0.34	0.15	—		—	—	20 m. tons
(4) *Colombia*													
Paz de Río	48.03	0.26	10.52	5.88	0.47	1.1	0.99	0.07			3.3	11.12	
(5) *México*													
El Mamey, Colima	63.44		5.68				0.167	0.025					40 m. tons
Aceros Ecatepec	61.4	0.6	3.22	0.84			0.073	0.02					
Cerro de Mercado, Durango average analysis	60/65	0.15–0.3	3.7				0.2–0.88	0.02–0.35					
Altos Hornos-analysis	60.9	0.20	3.67	0.3	2.35	0.32	0.9	0.08		1.85			106 m. tons
Monterrey-analysis	64/66	—	2.5–3.5	—	0.8–1.6	—	0.35–0.54	0.05		—	—	—	
Einecke-analysis	63/62	0.18	3.5	0.63	1.40	0.51	0.31	0.027		—	2.08		
(6) *Perú*													
Marcona	62.13	0.08	1.82(–11)	1.62	0.36	0.18	0.008	0.007	1.0 TiO_2				100 m. tons
Jatunkussi, near Chimbote (in neighbourhood of coalfields)	60		6.0				0.07						64 m. tons (Magnet)
(7) *Venezuela*													
Swiss Iron Mines of Venezuela	60.5/64	0.05/0.07	1.2–3.7				0.19(0.02)	0.06–0.3	0.3–1.4 Ti				
Corporación Venezolana de Fomento	68	0.2	0.5	little			0.03	0.01	0.1 Ti				300–500 m. tons
El Pao, Manao	67.5	—	0.6	0.28			0.03	0.06	0.2 TiO_2				35 m. tons
Cerro Bolivar	63.8	—	—	—			0.113	—	—				

[a] Often less till 39.

[b] Sometimes.

The erection layout of the steelworks will depend on local conditions and will not be discussed here. In preparing the layout plan, the first decision will concern the proper construction and arrangement of the hearth furnaces, converters, etc., but consideration will also be given to a practical system of transport routes and facilities, for instance, for the delivery of pig iron, scrap, ore and lime to the works, the removal of steel, the disposal of slags, etc. Finally the auxiliary plant for moulds, ladles, refractories, power, oxygen, etc., will also require careful planning.

4. RAW MATERIAL REQUIREMENTS

The requirements of the principal iron raw materials for the production of steel by the various processes are listed in Table 2 (more detailed data on the requirements of other raw and ancillary materials will be found in the paper by Dr. Krebs). The calculation of the amounts of raw materials is based on the assumption of well-ordered operational conditions such as may be achieved when the plant has been in operation for some time, i.e.,

Yield (1) In the conversion of pig iron into steel
In the open hearth furnace.................91%
In the electric arc furnace..................91%
In the converter...........................89%
In the oxygen converter...................90%
In the duplex process......................87%

Yield (2). In the conversion of scrap into steel (using poor quality scrap the yield is much lower).......................95%

Yield (3). In the conversion of ore (60% Fe) into steel (20% slagging losses)..........48%

In the beginning, lower yield figures are likely to result, but the fundamental differences (apart from the scrap) will not change.

On the basis of the yield conditions indicated above, the following figures are calculated in kg/ton of steel:

	Process I	*II*	*III*	*IV*
Pig iron	750	857	847	877
Scrap	250	250	250[a]	250
Ore	166[b]	—	—	—
TOTAL	1,166	1,107	1,097	1,127

[a] Can be partly replaced by ore.

[b] Ore consumption substantially lower with a strongly reducing flame or slow operation.

To reach any decision as to the best way to develop the steel industry in the individual Latin-American countries, a careful study of prevailing local conditions would be required. An analysis of the available iron ore is of some assistance in selecting the best process; such an analysis is contained in Table 3.

The Balance of Materials and the Economic Comparison of the Different Steelmaking Processes

ERNST KREBS

The first object in improvements in technique is that the aim must be achieved with a minimum expenditure. A number of methods have thus been developed for the production of steel, which vary according to local conditions. The difference between these methods is determined either by the charging materials, the operating procedures, or by the end uses of the steel.

This paper is intended to compare the various steelmaking processes with one another as regards their operational economy. No consideration has been given to the fixed charges (interest, taxes, amortization, etc.) or to the end use of the steel in industry or commerce, or its quality, and only ordinary commercial carbon steels are considered. Due to the decisive influence of the raw materials, including the fuels, the material balance-sheet per ton of crude steel forms the basis for determining the costs of the various processes.

OPEN HEARTH PROCESS

The open hearth process is still responsible for producing the major part of the world's crude steel requirements. Its broad raw material basis has led to a large number of changes in the process, depending either on the kind or on the availability of the individual raw materials.

The principal charging materials for the open hearth process are pig iron and scrap. Statistics show that the proportions of these two materials in the total charge differ considerably in the various countries, depending usually on the scrap situation. The bases of open hearth steel production must therefore be given in their most general form in order to permit a flexible alignment to very varied conditions.

Three different kinds of scrap are used in the open hearth furnace, the distinction between them being one of origin:

1. Circulating scrap, from the steel mill itself;
2. Processing (new) scrap, from machine and process shops;
3. Old, or dealers' scrap, from dismantled or obsolete plants, equipment and parts. Of inferior quality to scrap of types 1 and 2, it may take a few years or decades before it gets into the open hearth.

The open hearth process can be carried out with any given proportions of scrap and pig iron. Working exclusively on a scrap basis, 1,065 to 1,200 kg. of scrap are needed per ton of open hearth steel, the lower figure for first class scrap, the higher for poor grade, strongly oxidized scrap. The average requirement should be estimated at 1,130 kg. per ton of open hearth steel.

In the pure pig iron-ore process, with due allowance for the iron joining the metal from the necessary iron ore, 960 kg. of pig iron is sufficient to produce one ton of crude steel.

Steel production by the open hearth process is essentially one of oxidation, that is to say, the separation of the accompanying elements from the iron by transfer to the gas or the slag. According to the oxidation techniques used, different reaction heats are involved, making the process exothermic or endothermic, depending largely on the percentage of scrap in the charge. This determines the extent of the oxidation required and thus influences the amounts of the additives, fuel consumption, furnace output and hence the costs of the steel production.

For this reason, Figure 1 first shows the oxygen requirement of the various scrap ratios per ton of open hearth steel. The X-axis (horizontal) represents the charge of scrap and/or pig iron per ton of crude steel. The Y-axis (ordinate) indicates the amounts of oxygen required for oxidizing, and/or the oxygen surplus caused by the scrap, or the lack of oxygen caused by the addition of pig iron per ton of crude steel. On the ordinate, to the left, lies the pure scrap-carbon process without a pig iron addition. Since the open hearth furnace is not a closed crucible, the scrap absorbs oxygen to a greater or lesser extent, depending on the surface area, the oxidizing atmosphere of the port blocks, or the flame action over the charge and its charging sequence.

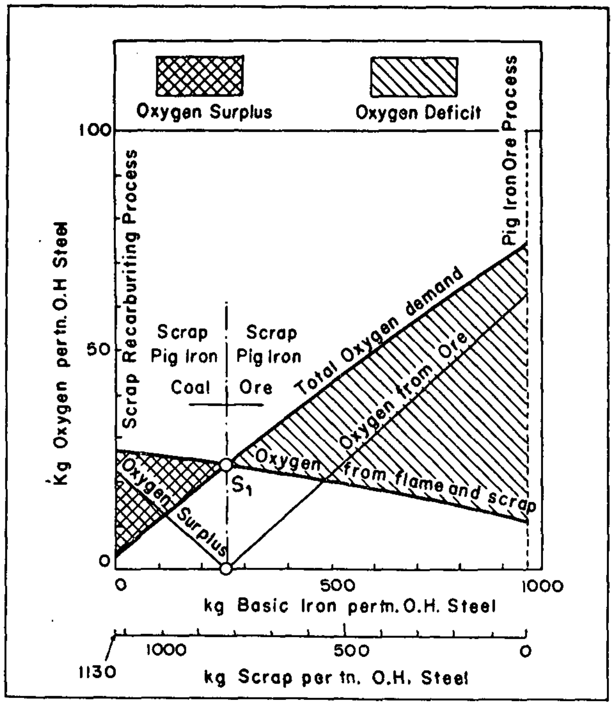


Figure 1. Oxygen balance in the production of open hearth steel for different basic iron or scrap rates.

Figure 1 is drawn with Bansen's figures for good scrap, i.e., for an average oxygen content and absorption of 24 kg. per ton of ingots with a normal pig-iron process with a charge of 260 kg. of pig iron per ton of ingots. When light sheet scrap and chips are used, the oxygen by absorption from the flame and the oxygen from the original rust may rise to 40 kg. per ton of ingots. The oxygen with rusty chips can amount to as much as 80 kg. per ton of ingots.

The absorption of oxygen decreases, especially in the case of a hot metal charge, with the increase in the proportion of hot metal. If the process is carried out with pig iron only, it decreases in Figure 1 to 11.5 kg. per ton of ingots. The oxygen from flame and rust per ton of ingots increases in direct proportion to the scrap, and amounts to 27 kg. per ton of ingots in the pure scrap-carbon process.

On the other hand, the oxygen demands vary also with the varying proportions of pig iron and scrap. Since with the pure scrap-carbon process there is only a small amount of iron accompanying elements, only the smallest amount is required for binding these elements, amounting to 3 kg. per ton of ingots. If the process is carried out with pig iron only, the result is that the requirement of oxygen is highest for the binding of the accompanying elements, i.e., silicon, manganese, carbon and phosphorus. It may vary according to the pig iron analysis between 70 and 90 kg. per ton of pig iron. In Figure 1 it rises to 74 kg. per ton of ingots in the pure pig iron-ore process. This refers to a pig iron with about:

4.1% O 0.9% Si 2.0% Mn 0.3% P 0.04% S

The curves of oxygen demand and oxygen supply intersect at Point S_1, indicating that the quantity of oxygen from flame and scrap just satisfies the oxygen requirement. Below and above the pig iron proportion corresponding to Point S_1 of 260 kg. per ton of crude steel, the differences between the oxygen requirement and the oxygen supply from flame and scrap must be made good by other means. Below the equilibrium point, i.e., below 260 kg. of pig iron per ton of raw steel, the oxygen surplus must be reduced by means of a carburizing agent. Above the equilibrium point, the oxygen deficiency must be corrected by an oxidant additive. The position of S_1 depends upon the quality of the scrap, pig iron analysis, etc., and varies widely between 150 kg. pig iron per ton of raw steel for low oxidizing conditions, and 700 kg. for high oxidation.

Melting the charge requires a fixed amount of heat. In the open hearth furnace, which is an endothermic process, heat must be supplied. The extent of the desired reactions thus depends upon the amount of heat supplied to the furnace. For any given open hearth furnace, the amount of heat which can be supplied is fairly constant, and is independent of the scrap-pig iron ratio of the charge. If, therefore, the heat requirements of the charge are known, the furnace output per hour can be calculated in tons of crude steel. Figure 2 shows the influence of the scrap-pig iron ratio on the requirements of effective heat. The continuous trace represents good, little oxidized scrap, and the interrupted line shows poor quality, heavily oxidized scrap. It is seen that the heat requirement is highest in the pure scrap-coal process. This heat requirement decreases with increasing molten pig iron additions in the charge (see point S_1 in Figures 1 and 2) down to a point where ore additions are necessary. From this point, the heat supplied by the increased molten charge and the heat required for the oxidation of the ore are fairly evenly balanced, and remain almost constant up to the pure pig iron-ore process.

It was from this course of the heat requirement that the output curve shown in the top part of Figure 2 was calculated.

Assuming, for instance, that the open hearth furnace is able to transfer 3.0×10^6 kcal-hour to the bath, then in a melt with 260 kg. of pig iron per ton of steel,

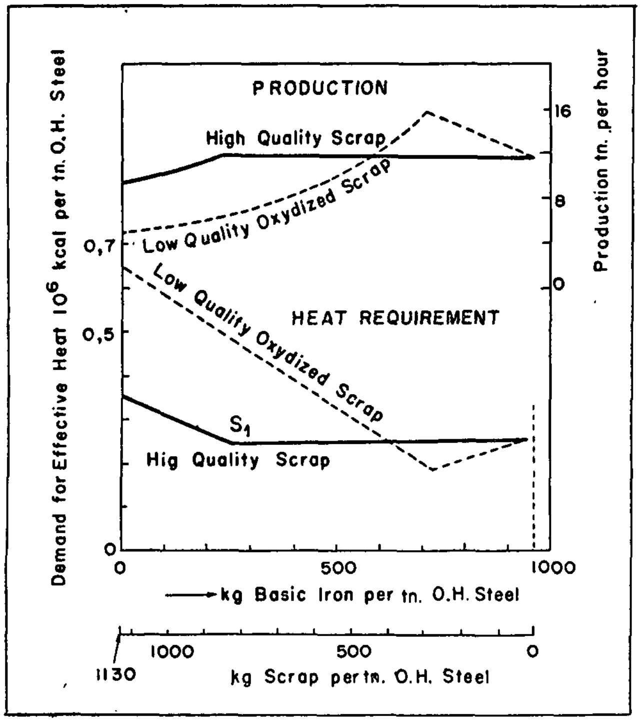


Figure 2. Demand for effective heat per ton open hearth steel and production per hour when using basic iron.

or more, the requirement of efficient heat for good scrap, according to Figure 2, amounts to 0.25 x 10^6 kcal per ton of crude steel. This results in an hourly furnace output of $\frac{3.0 \text{ x } 10^6}{0.25 \text{ x } 10^6}$ = 12 tons per hour. In this way the furnace output for the top part of Figure 2 was calculated from the requirements of efficient heat per ton of steel for the various proportions of scrap and open hearth pig iron.

The course of the efficient heat requirement and of furnace output in the other extreme case, i.e., when only poor quality, strongly oxidized scrap is used, is shown by the interrupted trace in Figure 2.

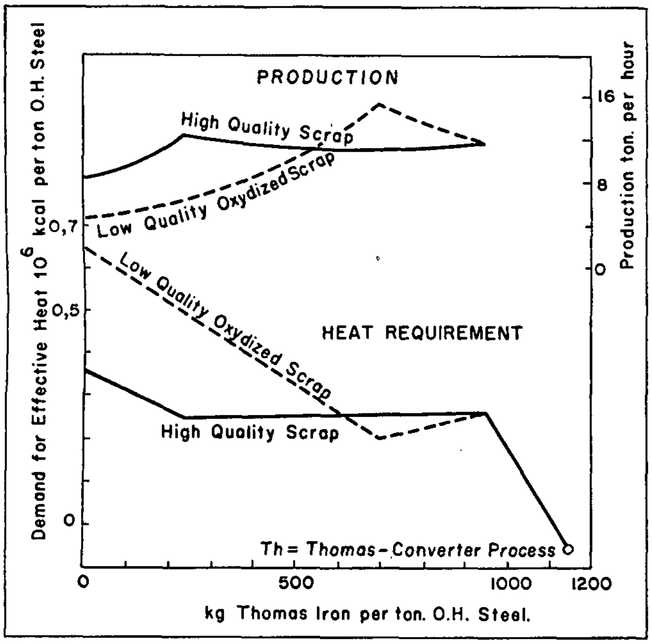


Figure 3. Demand for effective heat per ton open hearth steel and production per hour when using Thomas iron (2% P).

The kcal received by the bath of the open hearth is the useful or effective heat. In addition to this heat, the fuel must cover losses such as those from cooling and flue gases etc. For example, if the effective heat amounts to 27.5% of the fed heat, with a value of 3.0 x 10^6 kcal/hour, the heat input to the furnace must be $\frac{3.0 \text{ x } 10^6}{0.275}$ = approximately 10.9 x 10^6 kcal per hour. The actual consumption of heat per ton of steel per hour over a long period is higher, since the heat required for warming up the furnace and keeping it hot must be added.

Figures 1 and 2 are based on open hearth pig iron. Technically they are also valid for an iron rich in phosphorus, that is, Thomas pig iron. Heat consumption and output for a charge of molten Thomas pig iron and scrap between the extreme compositions are shown in Figure 3, whilst Figures 4 and 5 consider any pig iron or scrap charge. Coal and lime estimations are also inserted.

As a point of departure, a process is considered wherein the pig iron charge is 275 kg. and the scrap is 810 kg. per ton of steel. About 77% of the scrap is high grade. Figure 2 indicates an efficient heat requirement of about 0.3 x 10^6 kcal and the total heat is $\frac{0.3 \text{ x } 10^6}{0.275}$ = 1.09 x 10^6 kcal per ton of steel. With an average heat absorption of the furnace of 10.9 x 10^6 kcal, the output of the melt would amount to $\frac{10.9 \text{ x } 10^6}{1.09 \text{ x } 10^6}$ = 10 tons per hour.

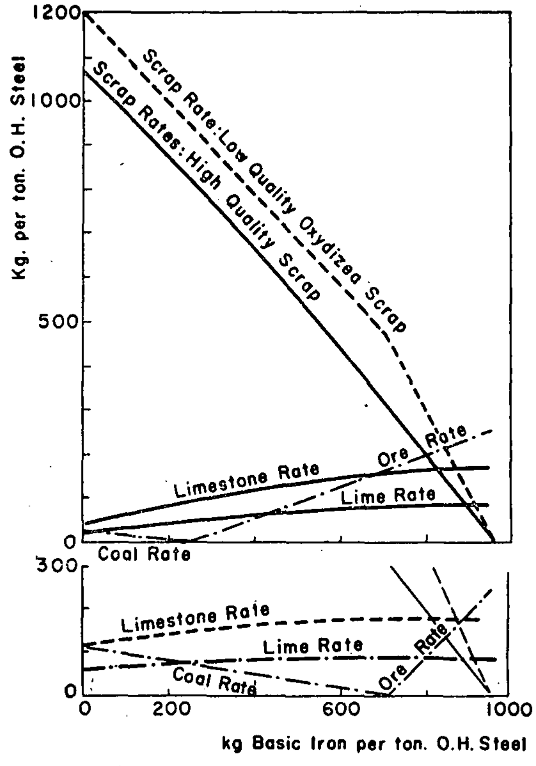
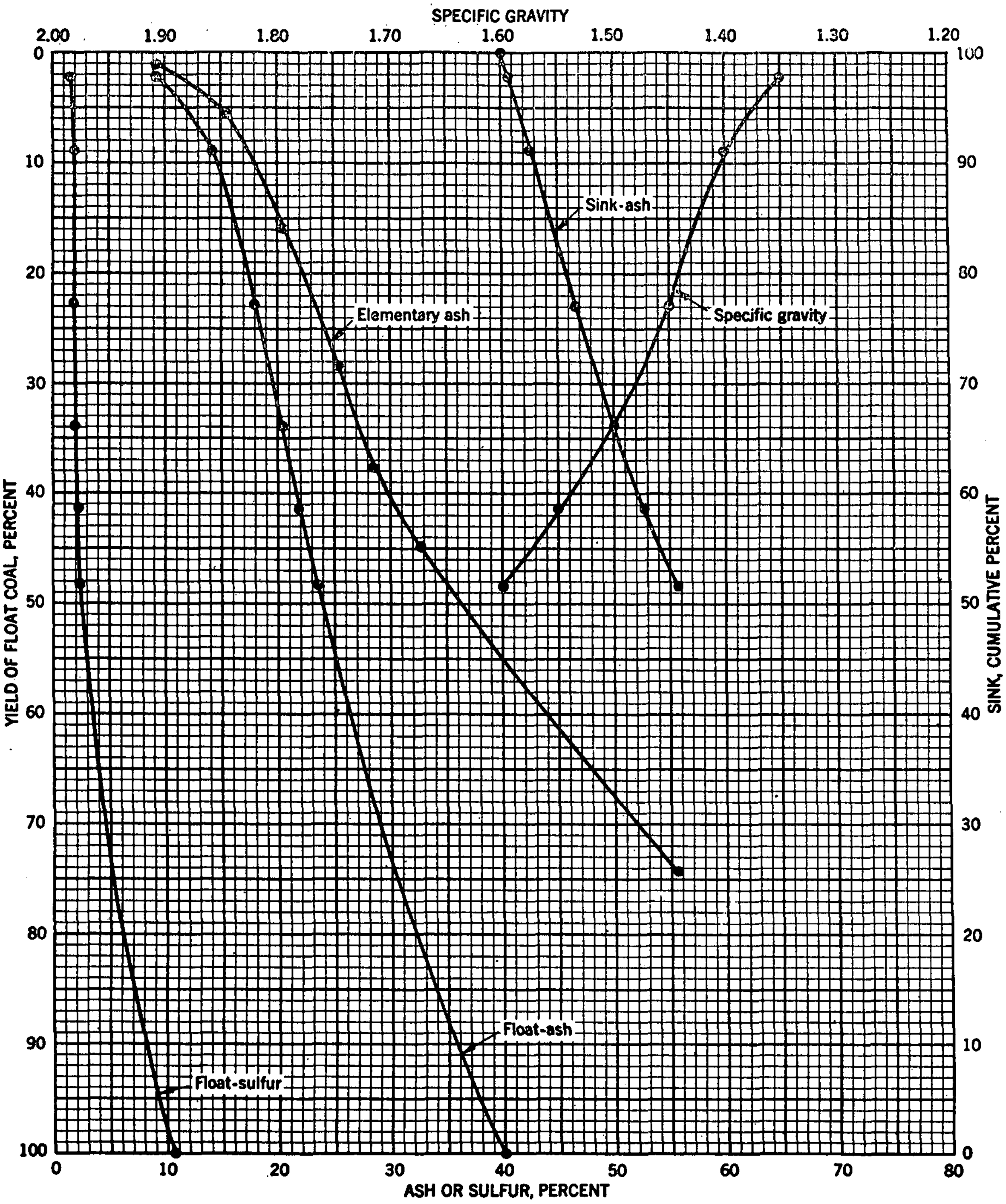


Figure 4. Materials per ton open hearth steel for different basic iron or scrap rates.

The conversion of the raw materials into steel may be subdivided into three stages: (1) Melting operation; (2) Ladle operation; (3) Casting operation.

These same stages are convenient for estimating costs.



Specific gravity fractions	Elementary data				Computed cumulative data					
	Weight grams	Percent			Float, cumulative percent			Sink, cumulative percent		
		Weight	Ash	Sulfur	Weight	Ash	Sulfur	Weight	Ash	Sulfur
Float - 1.35		2.2	9.3	1.76	2.2	9.3	1.76	100.0	40.0	10.58
1.35 to 1.40		6.7	15.7	2.03	8.9	14.2	1.96	97.8	40.8	10.78
1.40 to 1.45		14.0	20.6	1.81	22.9	18.0	1.87	91.1	42.6	11.42
1.45 to 1.50		11.2	25.5	2.16	34.1	20.5	1.96	77.1	46.6	13.17
1.50 to 1.55		7.3	28.5	3.40	41.4	21.9	2.22	65.9	50.1	15.04
1.55 to 1.60		7.1	32.8	2.48	48.5	23.5	2.26	58.6	52.8	16.49
Sink - 1.60		51.5	55.6	18.42	100.0	40.0	10.58	51.5	55.6	18.42

Fig. 7. Washability chart of the Barro Branco coal. Size 50 – 10 mm.

According to Figures 2 and 3, the output from the various charges is known. Therefore the hourly costs of the melting operation, which are the same for all charge conditions, need only be divided by the output in question in order to obtain the cost of melting one ton of steel.

On the other hand, the costs arising in the ladle and casting operations may be regarded as proportional to the quantities, and as being constant per ton of steel for the different charge conditions.

Table 1

BALANCE OF MATERIALS AND COSTS OF MATERIALS FOR THE PRODUCTION OF 1 TON OPEN HEARTH STEEL WITH A SCRAP CHARGE OF 810 KG. PER TON

	Quantity kg./ton ingot	*Price DM*[a] *per ton*	*Amount DM per ton ingot*	
Charge:				
Metal charge:				
Stahleisen, molten	275.0	200.—[b]		55.00
Scrap:				
Works scrap, 700 kg.........				
Outside scrap, 110 kg.........				
Total scrap........	810.0	170.—		137.70
Additions:				
Deoxidizing and alloying element additions......	8.0		5.49	
Ore, calculated in Fe............	20.0	150.—	3.00	
Total additions....			8.49	8.49
TOTAL METAL CHARGE	1,113.0			
Fluxes:				
White lime......	41.0	40.—	1.64	
Fluorspar.......	3.0	120.—	0.36	
Other...........	2.0	30.—	0.06	
TOTAL FLUXES	46.0		2.06	2.06
Output:				
Ingot butts, sprues, trumpets, skulls..	48.0	−170.—[c]	−8.16	
Slag..............	129.0	−10.—[c]	−1.29	
TOTAL CREDITS			−9.45	−9.45
Melting loss.......	65.0			
Ingot.............	1,000.0			
TOTAL COSTS OF MATERIALS IN CHARGE				193.80[c]

[a] German marks.

[b] Price of pig iron is arbitrarily assumed, in order to enable comparison; costs of steel are therefore fictitious.

[c] Credit.

The framework of quantities and costs in Tables 1 and 2 provides the basis for establishing the costs of open hearth steel under various pre-conditions. They are set up for stationary open hearth furnace with a melt weight of 90 tons with charge conditions as already indicated, and a weight of 5 tons per ingot.

Table 1 shows the costs of charging materials, and Table 2 the relevant conversion costs per ton of open hearth steel. The total, as indicated in Table 2, is DM 242.19,[1] which is an arbitrary figure assumed for comparative estimates.

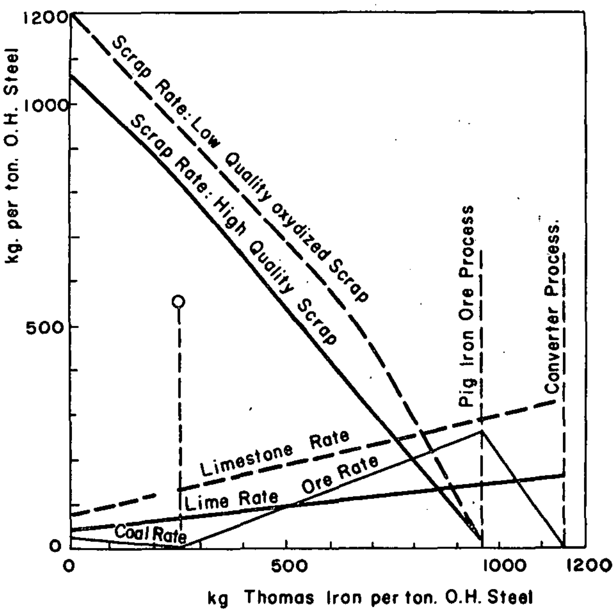


Figure 5. Materials per ton open hearth steel for different Thomas iron or scrap rates.

If oil alone were used as a heating agent, the figure would rise to DM246.28 per ton, due to the present high price of oil in Germany (October 1952).

Table 3 shows the man hours required, also subdivided into the three cost groups.

The above remarks deal only with the basic open hearth process.

THE CONVERTER PROCESS

Like the electric steel process, the ordinary converter process has a narrower raw material basis than the open hearth. By extending the range of Figure 3 to cover higher proportions of pig iron per ton of steel, the heat balance with an ordinary air blow is shown.

The charging materials for the Thomas process include a maximum of 50 kg. of scrap per ton of steel, the balance being composed of pig iron. The total of these two materials amounts to about 1,130 kg. per ton of steel.

The material balance sheet and the costs of charging materials for the Thomas process are shown in Table 4, and the conversion costs in Table 5. The total sum of DM236.44 per ton of Thomas steel indicated in Table 5, arising from charging material and conversion costs, represents the cost of production. The cost of converting the arising of Thomas slag into the fertilizer ground basic slag is indicated in Table 6.

Table 7 summarizes the labour required to produce one ton of steel, also divided into the three cost groups.

All these figures refer only to the Thomas basic converter process. The acid Bessemer converter process has a limited application, particularly in view of recent developments in surface blowing, which will be discussed later.

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[1] 1 DM—$0.2371.

Table 2

COSTS FOR THE PRODUCTION OF 1 TON OPEN HEARTH STEEL

			Melting operation		*Ladle working*		*Casting operation (casting crane, bottom casting)*	
	Unit	*Price per unit*	*Units per ton molten production*	*Costs DM/ ton molten production*	*Units per ton molten production*	*Costs DM/ ton molten production*	*Units per ton solid production*	*Costs DM/ ton sold production*
Fuels	10^6 kcal		1.09					
Coke oven gas; $H_u = 3{,}800$ kcal/Nm^3	1,000 Nm^3	60.00	0.244	14.64				
Oil; 10,000 kcal/kg	1,000 kg.	200.00	0.0163	3.26				
(Oil); " "	(1,000 kg.)	(200.00)	(0.109)	(21.80)				
Blast furnace gas; $H_u = 1{,}000$ kcal/Nm^3		8.50			0.040	0.34		
TOTAL				17.90		0.34		
Energy:								
Power	1,000 kWh.	60.00	0.004	0.24			0.006	0.36
Compressed air	1,000 Nm^3	7.00	0.012	0.08			0.003	0.02
Water	1,000 m^3	20.00	0.015	0.30				
TOTAL				0.62				0.38
Refractories:								
Furnace bricks				in quota for furnace relining				
Ladle bricks	1,000 kg.	170.00			0.010	1.70		
Runner bricks	1,000 kg.	165.00					0.007	1.16
Dolomite	1,000 kg.	60.00	0.020	1.20				
Others	1,000 kg.	25.00	0.005	0.13	0.005	0.13		
TOTAL				1.33		1.83		1.16
Moulds and accessories								3.00
Working materials				0.50				0.50
Other operating costs:								
Works railway				0.75				0.75
Laboratory				1.00				
TOTAL				1.75				0.75
Quota for furnace relining				7.00				
Maintenance and repairs:								
Working material and spare parts				0.75		0.05		1.75
Wages				1.77		0.11		0.80
TOTAL				2.52		0.16		2.55
Manufacturing wages:								
Wages, salaries, plus social obligations, etc.				1.81		0.16		2.40
CONVERSION COSTS				33.43		2.49		10.74

DM/ton solid production

Working costs melting operation .... $33.43 \times \frac{1{,}000+48}{1{,}000} = 35.04$

Working costs ladle work .... $2.49 \times \frac{1{,}000+48}{1{,}000} = 2.61$

Working costs casting operation .... = 10.74

Total working costs .... = 48.39
Total costs of materials in charge .... = 193.80

242.19

FOR OIL FIRING EXCLUSIVELY 246.28

Table 3

STATEMENT OF WORKING HOURS FOR THE PRODUCTION OF 1 TON OPEN HEARTH STEEL

	Working hours for starting up and first operating time		*Target working hours to be achieved*	
	Production hours/ton ingot	*Maintenance and repair hours/ton ingot*	*Production hours/ton ingot*	*Maintenance and repair hours/ton ingot*
Furnace operation...	0.97	0.95	0.65	0.633
Ladle working.......	0.08	0.04	0.054	0.0295
Casting operation....	1.3	0.27	0.868	0.178

ELECTRIC STEEL PROCESS

Computations are based on the performance of a 20-ton arc furnace, and charging materials are scrap and/or Duplex metal, in proportions ranging from 0 to 100% of the charge. In the pure scrap process, about 1,040 kg. of scrap will be the charge. If Duplex metal forms part of the charge, there is a saving in the heat required for the melting, and the charging time is shorter.

Table 4

BALANCE OF MATERIALS AND COSTS OF MATERIALS FOR PRODUCTION OF 1 TON THOMAS STEEL

	Quantity kg./ton ingot	*Price DM/ton*	*Amount DM/ton ingot*	
Charge:				
Metal charge:				
Mixer iron (scrap charge assumed =nil).........	1,129.2	200.—[a]		225.84
Additions:				
Deoxidizing and alloying element additions......	10.8[c]		5.06	
Anthracite......	0.25	70.—	0.02	
TOTAL ADDITIONS			5.08	5.08
TOTAL METAL CHARGE	1,140.00			
Fluxes:				
Lime...........	150.0	40.—	6.00	
TOTAL FLUXES	150.0		6.00	6.00
Output:				
Ingot butts, sprues and trumpets....	12.0	−170.—[b]	− 2.04	
Skulls, mill residues (80% Fe) calculated in Fe......	20.0	−170.—[b]	− 3.40	
Chimney skulls and converter dust (40% Fe) calculated in Fe......	12.0	−170.—[b]	− 2.04	
Melting loss.......	96.0	—	—	
Thomas slag (with 15% P_2O_5)......	240.0	− 45.—[b]	−10.80	
Credits...........			−18.28	−18.28
Ingot.............	1,000.0			
TOTAL COSTS OF CHARGE MATERIALS				218.64

[a] Pig iron price is assumed, to enable comparison; the costs of steel are therefore fictitious.

[b] Credit.

[c] Includes 92% spiegeleisen with 20% Mn.

The cost data for plain carbon steels are shown in Tables 8 and 9. Costs for a 60-ton unit will be used, determined from those of the 20-ton furnace. These furnace costs per hour will include all costs associated with the melting operation except those for power, electrodes and nipples, which in the smaller furnaces are proportionate to their capacity, and lower for a bigger furnace.

Table 9 shows the costs for the 20-ton unit as 84.27 − (48 plus 12.50) = DM23.77 per ton of electric steel. With an output of 4 tons per hour, the furnace costs would be 4 x 23.77 = 95.08, i.e., about DM100 per hour. As these costs rise only very slightly for a bigger furnace, those for the 60-ton unit might be some 20% higher.

As the power increases in proportion to the capacity, the time required for the heat is the same, so that the production rate of the 60-ton unit would be 3 x 4 = 12 tons per hour, and the melting costs, excluding electrodes and power would be $\frac{120}{12} = \text{DM}10$ per ton.

To this must be added the remaining costs as listed in Table 10. Hence the total production costs of the electric steel in the 60-ton furnace may be estimated at DM264.87 per ton. These costs are calculated on the high power cost of DM60 per 1,000 kWh.

OXYGEN CONVERTER PROCESS

Two production methods are favoured by the industry:

1. *Enrichment of the blast with oxygen up to 35%*

This process is applied with the aim of producing a steel which can to some extent replace open hearth quality steels. Bottom blowing is used.

2. *Blowing with practically pure oxygen (at least 98% O_2) surface blown process*

Although still in its early operating stages, this method is attractive to a country with little available scrap, since it is one of the cheapest processes, and furnishes high quality steel. The oxygen is not blown through the bottom, but through a top tuyere to the bath.

Of these two processes, only the second will be dealt with, as it is particularly suited for the raw material situation of countries poor in scrap. Moreover, it is claimed to produce better steel for several applications than that produced only with oxygen-enriched blast. Because of less nitrogen ballast, the surface blown system allows a lower grade of pig iron in the charge, and is, for instance, very successful with a pig iron analysis of 0.2 to 1.0 Si and 0.1 to 0.3% P. Tests are being carried out with prospects of success for pig iron even richer in phosphorus.

Experiences to date have shown that it is possible to remelt all the circulating scrap in the surface blown process. Compared with the open hearth, this process in some cases uses only 25% of the heat required for the open hearth, and costs half as much, according to preliminary work done in Austria.

Table 5

COSTS FOR THE PRODUCTION OF 1 TON THOMAS STEEL

	Unit	Price per unit	*Converter operation*		*Ladle working*		*Casting operation (casting car, bottom casting)*	
			Units per ton molten production	*Costs DM/ton molten production*	*Units per ton molten production*	*Costs DM/ton molten production*	*Units per ton solid production*	*Costs DM/ton solid production*
Fuels:								
Blast furnace gas; $H_u = 1{,}000$ kcal/Nm^3	1,000 Nm^3	8.50			0.040	0.34		
Coke	1,000 kg.	65.00	0.002	0.13				
TOTAL				0.13		0.34		
Blast	1,000 Nm^3	5.25	0.370	1.94				
TOTAL				1.94				
Energy:								
Power	1,000 kWh.	60.00	0.0015	0.09			0.0015	0.09
Pressure water	1,000 m^3	200.00	0.004	0.08				
TOTAL				0.17				0.09
Refractories:								
Dolomite bricks and tar mixture	1,000 kg.	120.00	0.008	0.96				
Dolomite bottoms	1,000 kg.	165.00	0.005	0.83				
Ladle bricks	1,000 kg.	170.00			0.010	1.70		
Runner bricks	1,000 kg.	165.00					0.007	1.16
Mass and clay	1,000 kg.	25.00	0.003	0.08			0.003	0.08
Others						0.13		
TOTAL				1.87		1.83		1.24
Moulds and accessories	1,000 kg.							3.00
Working materials				0.05				0.15
Other operating costs:								
Works' railway				0.50				0.75
Laboratory				0.50				
TOTAL				1.00				0.75
Maintenance and repairs:								
Working materials and spare parts				0.65		0.05		0.40
Wages				0.96		0.11		0.51
TOTAL				1.61		0.16		0.91
Manufacturing wages:								
Wages, salaries plus social obligations				1.00		0.16		1.07
WORKING COSTS				7.77		2.49		7.21

		DM/ton solid production
Working costs converter operation	$7.77 \times \frac{1{,}000+20+12}{1{,}000}$	= 8.02
Working costs ladle working	$2.49 \times \frac{1{,}000+20+12}{1{,}000}$	= 2.57
Working costs casting operation		= 7.21
Total working costs		= 17.80
Total costs of materials in charge		= 218.64
		236.44

To enable comparison with the other processes, the casting operation costs of open hearth steel working with crane ladle (Table 2) must be inserted here instead of the casting operational costs of Thomas steel. The casting operational costs are thus increased by 3.53 DM/ton so that the total production costs amount to 236.44+3.53=239.97 DM/ton.

Table 6

FRAMEWORK OF QUANTITIES AND COSTS PER TON GROUND BASIC SLAG

	Quantity per ton	*Price DM/ton*	*Prime costs DM/ton*	
Charge:				
Thomas slag......kg./ton	1,000	45.—[a]		
TOTAL CHARGE MATERIAL				45.00
Energy:				
Water............m³/ton	0.5		0.01	
Power and light..kWh/ton	17.0		1.02	
TOTAL ENERGY			1.03	1.03
Maintenance.............			2.68	
Ancillaries for operational maintenance............			0.18	
Transport with works railway, etc................			0.24	
Manufacturing wages (wages, salaries plus social obligations, etc.)..............			1.15	
Storage and shipping costs...			2.80	
			7.05	7.05
COSTS OF MATERIALS AND PRODUCTION COSTS				53.08

[a] The price of the Thomas slag is assumed.

Table 7

STATEMENT OF WORKING HOURS FOR THE PRODUCTION OF 1 TON THOMAS STEEL

	Working hours for starting up and first operating time		*Target working hours to be achieved*	
	Production hours/ton ingot	*Maintenance and repair hours/ton ingot*	*Production hours/ton ingot*	*Maintenance and repair hours/ton ingot*
Converter operation...	0.57	0.23	0.38	0.155
Ladle working........	0.56	0.01	0.375	0.008
Casting operation.....	0.59	0.17	0.39	0.111

The yield is put at 87%. Cost data and balance of materials are indicated in Table 11 on the basis of the results of this work. The costs of materials, based on the charging quantities according to K. Rösner, and the assumed prices, amount to DM221.97 per ton of steel.

The converter operation costs are assumed to be 50% of those for the open hearth furnace when using 220 kg. of good circulating scrap per ton of open hearth steel. According to Figure 2, for a charge of 220 kg. of scrap per ton of open hearth steel, furnace output would be approximately 12 tons per hour. Table 2 gives melting costs of DM334.30 per furnace hour, or a cost per ton of molten steel of $\frac{334.30}{12}=$ DM27.90. The converter operation costs would thus be 0.5 x DM27.90=DM13.95 per ton of molten steel, or $\frac{13.95}{0.975}$ = DM14.35 per ton of ingots. Adding the costs of the ladle and casting operations, as established in Table 2, total production costs would amount to DM249.63 per ton of ingots.

POSSIBILITIES AND COMPARISON OF COSTS OF STEEL PRODUCTION FOR COUNTRIES WHICH ARE POOR IN SCRAP

In the less industrialized countries scrap is scarce, and therefore, given an exclusive production of open hearth steel, no more than 300 kg. of scrap could be charged per ton.

The Bessemer process requires hardly any scrap, but releases scrap in the working of its product, and the scrap thus arising can be supplied to the open-hearth furnace.

Figure 6 plots the available amount of scrap against the percentage of converter steel of the total steel product. This graph is drawn for an own consumption of scrap of 50 kg. per ton of Thomas steel, which, for reasons of simplification and clarity, is not referred to in the foregoing and following remarks.

An iron and steel works with an annual capacity of 300,000 tons lies within the favourable range of costs, which, as seen from Figure 7, increases with a decrease in the size of the plant. Figure 7 shows the characteristic movement of the steel production costs in relation to the size of the plant for the German metallurgical works in 1930, according to Matuschka. The technical possibilities of producing these 300,000 tons of steel are shown in Table 12, and Figure 8 shows them in simplified diagrams.

I. *Open hearth furnace only*

Assuming that for underdeveloped countries the maximum amount of circulating and ordinary scrap available is 300 kg. per ton of steel, the 300,000 ton

Table 8

BALANCE OF MATERIALS AND COSTS OF MATERIALS FOR THE PRODUCTION OF 1 TON OF ELECTRIC STEEL IN THE ELECTRIC ARC FURNACE

	Quantity kg./ton ingot	*Price DM/ton*	*Amount DM/ton ingot*	
Charge:				
Metal charge:				
Scrap...............	1,043.6	170.—[a]		177.41
Additions:				
80% of addition costs for open hearth steel, due to better yield..	6.4			4.39
Ore, calculated in Fe content..........	35.0	150.—		5.25
TOTAL METAL CHARGE	1,085.0			
Fluxes and carb. agents:				
White lime..........	80.0	40.—	3.20	
Fluorspar...........	4.0	120.—	0.48	
Carburizing agents...	2.0	70.—	0.14	
TOTAL FLUXES AND CARB. AGENTS	86.0		3.82	3.82
Output:				
Casting waste material..	5.0	−170.—[b]	−0.85	
Electric slag..........	96.0			
TOTAL CREDIT			−0.85	−0.85
Melting loss...........	80.0			
Ingot................	1,000.0			
TOTAL COSTS OF CHARGE MATERIAL				190.02

[a] Price assumed, like other prices, hence the calculated costs of steel production are fictitious.

[b] Credit.

Table 9

MELTING OPERATIONAL COSTS FOR THE PRODUCTION OF 1 TON OF ELECTRIC STEEL IN A 20-TON ELECTRIC ARC FURNACE

	Quantity per ton ingot	*Price*		*Amount DM per ton ingot*
Energy......kWh	800	60.— DM/1,000 kWh		48.00
Water.........m^3	6	20.— DM/1,000 m^3		0.12
Refractories:				
Furnace bricks.....kg.	17	200.— DM/ton	0.34	
Dolomite....kg.	56	60.— DM/ton	3.36	
Steelworks tar.......kg.	4	150.— DM/ton	0.60	
TOTAL REFRACTORIES			4.30	4.30
Other operational costs:				
Works' railway.				0.75
Laboratory.....				1.00
Electrodes and nipples....kg.	6			12.50
Quota for furnace relining........				10.00
Maintenance and repairs.........				2.00
Manufacturing wages including salaries and social obligations (40% of total manufacturing costs for furnace operating = 0.4×14).......				5.60
				84.27
Costs of materials in charge.......				190.02
TOTAL PRODUCTION COSTS				274.29

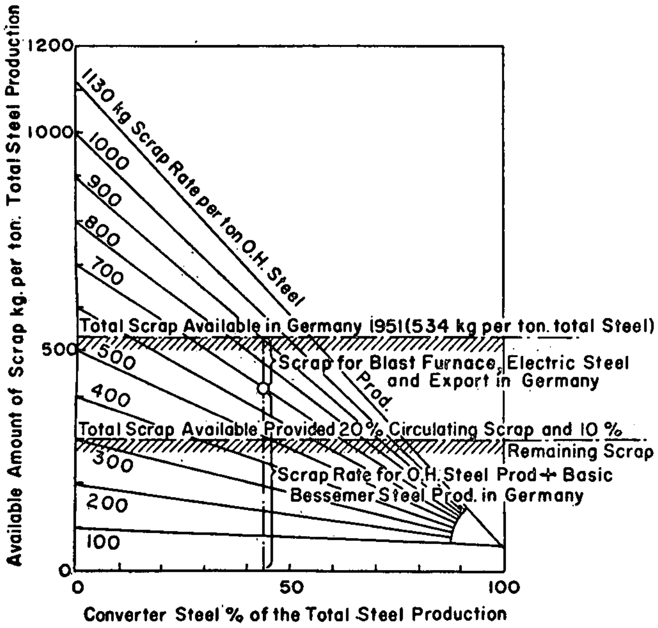


Figure 6. Available amount of scrap per ton open hearth steel production *versus* percentage converter steel of the total steel product.

Table 10

COSTS OF PRODUCTION OF 1 TON ELECTRIC STEEL IN THE 60-TON ARC FURNACE

	Quantity per ton ingot	*Price*	*Amount DM/ton ingot*
Melting operational costs..........			10.00
Power...........	650 kWh.[a]	60.— DM/1,000 kWh.	39.00
Electrodes and nipples........	6 kg.		12.50
Ladle working (costs same per ton as in open hearth steelworks).........			2.61
Casting operation (costs same per ton as in open hearth steelworks)			10.74
WORKING COSTS			74.85
Costs of materials in charge.......			190.02
TOTAL PRODUCTION COSTS			264.87

[a] Extrapolated according to "Anhaltszahlen Wärmewirtschaft".

plant would provide 0.3 x 300,000 = 90,000 tons of scrap per year, the amounts of pig iron and ore charged are shown in Table 12. Tables 13 and 14 show the costs, and total steel production costs, on the basis of these charge proportions, as DM249.90 per ton.

The electric furnace is not considered under I, as it would be uneconomical for transforming large quantities of pig iron into steel.

II. (a) *Converter and open hearth furnace*

The distribution of the pig iron between converter and open hearth furnace is on the basis that the

Table 11

BALANCE OF MATERIALS ACCORDING TO K. RÖSNER AND COSTS OF PRODUCTION OF 1 TON PURE OXYGEN STEEL

	Quantity kg./ton ingot	*Price DM/ton*	*Amount DM/ton ingot*
Pig iron......................	928	200.—[a]	185.60
Scrap.........................	220	170.—	37.40
Alloying and deoxidizing agents.	2	610.—	1.22
METAL CHARGE	1,150		
Lime and additions............	(50)	40.—	2.00
Melting loss..................	125		
Molten crude steel............	1,025		
Casting bay scrap.............	25	−170.—[b]	−4.25
Ingot yield...................	1,000		
INGOT YIELD IN % OF CHARGE	87		
Material costs................			221.97
Converter costs (=50% of furnace costs of the pig iron-ore process)..................			14.31
Ladle working and casting operation (as in open hearth process)			13.35
TOTAL PRODUCTION COSTS			249.63[a]

[a] Price of pig iron is arbitrarily assumed, to enable comparison; the costs of steel are therefore fictitious.

[b] Credit.

Table 12

RAW MATERIAL CONSUMPTION IN TONS PER ANNUM

	I	*II*		*IV*	
	Open hearth furnace	*a* *Converter and open hearth furnace*	*b* *Converter and electric arc furnace*	*a* *Duplex converter—open hearth furnace*	*b* *Duplex converter—electric arc furnace*
Pig iron					
For converter	—	212,000	242,000	235,000	263,000
For open hearth furnace	213,000	31,000	—	30,000	—
		243,000		265,000	
Duplex metal					
For open hearth furnace	—	—	—	208,500	—
For electric furnace	—	—	—	—	233,580
Scrap					
For converter	—	—	—	—	—
For open hearth furnace	90,000	90,000	—	90,000	—
For electric furnace	—	—	90,000	—	90,000
Ore					
For converter	—	—	—	—	—
For open hearth furnace	51,000	—	—	—	—
For electric furnace	—	—	—	—	—
Open hearth steel	300,000	112,500	—	300,000	—
Electric steel	—	—	86,000	—	300,000
Converter steel	—	187,500	214,000	—	—
TOTAL	300,000	300,000	300,000	300,000	300,000

furnace must work on the optimum proportion of scrap, which is about 800 kg. per ton of steel.

According to Table 12, under these conditions 112,500 tons of open hearth steel and 187,500 tons of air-blown steel will be produced, together forming the total of 300,000 tons of steel per annum.

(1) Costs of the air blown steel are, according to Table 5, DM239.97 per ton of Thomas steel;

(2) Costs of the open hearth steel, according to Table 2 are DM242.19 per ton of steel, or, for the 300,000 tons annual output:

$$\begin{aligned} & & \text{DM} \\ 242.19 \times 112{,}500 &=& 27{,}246{,}375 \\ 239.97 \times 187{,}500 &=& 44{,}994{,}375 \\ & & \overline{72{,}240{,}750} \end{aligned}$$

$$\text{or} \quad \frac{72{,}240{,}750}{300{,}000} = 240.80 \text{ per ton}$$

II. (b) *Converter and electric arc furnace*

The scrap is melted in the electric arc furnace, and the pig iron is charged only into the converter.

According to Table 12, under these conditions 214,000 tons of air-blown steel and 86,000 tons of electric steel will be produced, to make up the annual total of 300,000 tons of steel. A 60-ton arc furnace would be a suitable size.

(1) Costs of the air blown steel are those given in Table 5, i.e., DM239.97 per ton of Thomas steel;

(2) Costs of the electric steel are as given in Table 10, i.e., DM264.87 per ton.

The total cost would therefore be:

$$\begin{aligned} & & \text{DM} \\ 239.97 \times 214{,}000 &=& 51{,}353{,}580 \\ 264.87 \times 86{,}000 &=& 22{,}778{,}820 \\ & & \overline{74{,}132{,}400} \end{aligned}$$

$$\text{or} \quad \frac{74{,}132{,}400}{300{,}000} = 247.11 \text{ per ton}$$

III. *Oxygen air blown process*

The majority of the steel is produced in the oxygen converter by the surface blown process, the object

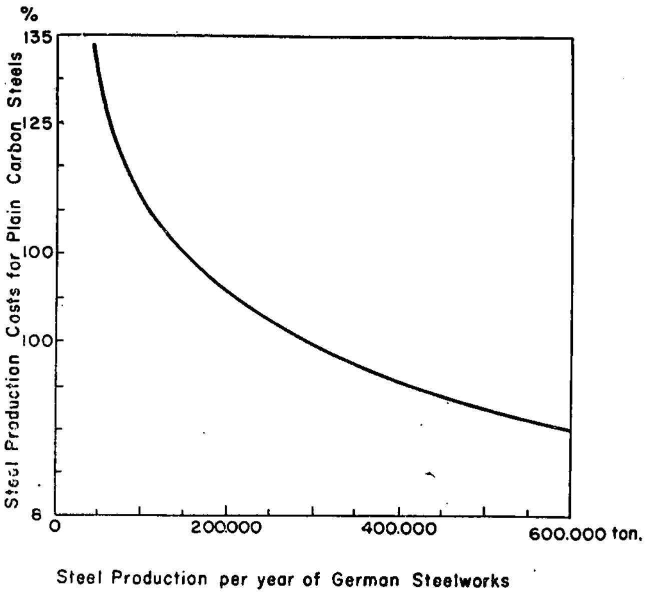


Figure 7. Steel production costs *versus* capacity of the plant.

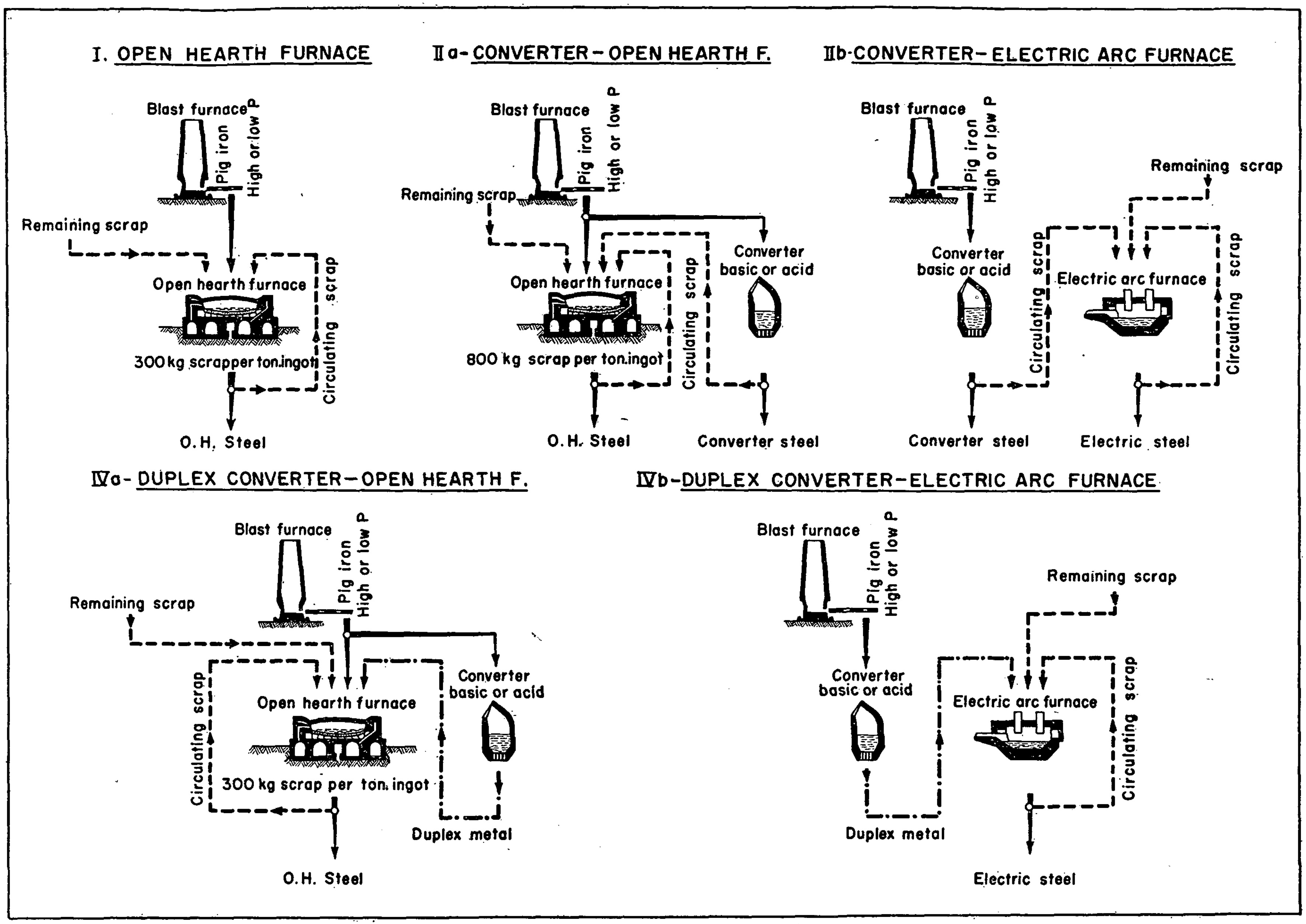


Figure 8. Sketch of the flow of materials for the different steelmaking processes.

Table 13

BALANCE OF MATERIALS AND COSTS OF MATERIALS FOR THE PRODUCTION OF 1 TON OF OPEN HEARTH STEEL WITH A SCRAP PROPORTION OF 300 KG. PER TON

	Quantity kg./ton ingot	*Price DM/ton*	*Amount DM/ton ingot*	
Charge:				
Metal charge:				
Stahleisen, molten	710	200.—[a]		142.00
Scrap	300	170.—		51.00
Additions:				
Deoxidizing and alloying element additions	8		5.49	
Ore, calculated in Fe	110	150.—	16.50	
Total additions			21.99	21.99
TOTAL METAL CHARGE	1,128			
Fluxes:				
White lime	75	40.—	3.00	
Fluorspar	3	120.—	0.36	
Others	2	30.—	0.06	
TOTAL FLUXES	80		3.42	3.42
Output:				
Ingot butts, sprues, trumpets, skulls	48	−170.—[b]	−8.16	
Slag	180	− 10.—[b]	−1.80	
TOTAL CREDITS			−9.96	−9.96
Melting loss	80			
Ingot	1,000			
TOTAL COSTS OF CHARGE MATERIAL				208.45[a]

[a] Price of pig iron is arbitrarily assumed, merely to enable comparison; the steel costs are therefore fictitious.
[b] Credit.

being to produce as much air-blown steel as possible for economy reasons and to produce in the open hearth or electric arc furnace only such proportion of steel as is absolutely essential to satisfy quality requirements. As no data is available, it is not possible to calculate the production costs for these steels. The steel costs amount to DM249.63 per ton.

Table 14

COSTS OF PRODUCTION OF 1 TON OPEN HEARTH STEEL WITH A SCRAP PROPORTION OF 300 KG. PER TON

Requirement of useful heat acc. to Figure 2		0.235×10^6 kcal/ton
Required supply of heat to furnace	$\frac{0.253}{0.275}\times10^6=0.918\times10^6$ kcal/ton	
Steel production/hour of furnace with heat supply of 10.9×10^6 kcal/hour	$\frac{10.9}{0.918}\times10^6=$	11.9 t/hour
Melting operational costs with furnace costs per hour of 334.30 DM/hour	$\frac{334.30}{11.9}=$	28.10 DM/ton
Costs of ladle working	=	2.61 DM/ton
Costs of casting operation	=	10.74 DM/ton
		41.45 DM/ton
Costs for charge materials acc. to Table 13	=	208.45 DM/ton
TOTAL PRODUCTION COSTS	=	249.90 DM/ton

Table 15

BALANCE OF MATERIALS, COSTS OF MATERIALS AND PRODUCTION COSTS FOR THE PRODUCTION OF 1 TON OF DUPLEX METAL

	Quantity kg./ton ingot	*Price DM/ton*	*Amount DM/ton Duplex metal*	
Charge:				
Metal charge:				
Mixer iron	1,128.0	200.—[a]		225.60
Additions:				
Lime	150.0	40.—		6.00
Output:				
Skulls, mill residues (80% Fe), calculated in Fe	20.0	−170.—[b]	− 3.40	
Chimney skulls and converter dust (40% Fe), calculated in Fe	12.0	−170.—[b]	− 2.04	
Melting loss	96.0			
Thomas slag (with 15% P_2O_5)	240.0	− 45.—[b]	−10.80	
Credits			−16.24	−16.24
Duplex metal	1,000.0			
TOTAL				215.36
Converter operating costs (per ton molten product; cf. Table 5)				7.77
Ladle working (per ton molten product; cf. Table 5)				2.49
TOTAL PRODUCTION COSTS				225.62

[a] Price of pig iron is arbitrarily assumed, to enable comparison; costs of steel are therefore fictitious.
[b] Credit.

Table 16

BALANCE OF MATERIALS AND COSTS OF MATERIALS FOR THE PRODUCTION OF 1 TON OF OPEN HEARTH STEEL WITH A DUPLEX CHARGE

	Quantity kg./ton ingot	*Price DM/ton*	*Amount DM/ton ingot*	
Charge:				
Metal charge:				
Stahleisen, molten	100.0	200.00[a]		20.00
Duplex metal	695.0	225.62		156.81
Scrap	300.0	170.00		51.00
Additions:				
Total deoxidizing and alloying element additions	8.0			5.49
TOTAL METAL CHARGE	1,103.0			
Fluxes:				
White lime	40.0	40.00	1.60	
Fluorspar	3.0	120.00	0.36	
Others	2.0	30.00	0.06	
TOTAL FLUXES	45.0		2.02	2.02
Output:				
Ingot butts, sprues, trumpets, steel skulls	48.0	−170.00[b]	−8.16	
Slag	100.0	− 10.00[b]	−1.00	
TOTAL CREDITS			−9.16	−9.16
Melting loss	55.0			
Ingot	1,000.0			
TOTAL COSTS OF CHARGE MATERIALS				226.16

[a] Price of pig iron is arbitrarily assumed, to enable comparison; the costs of steel are therefore fictitious.
[b] Credit.

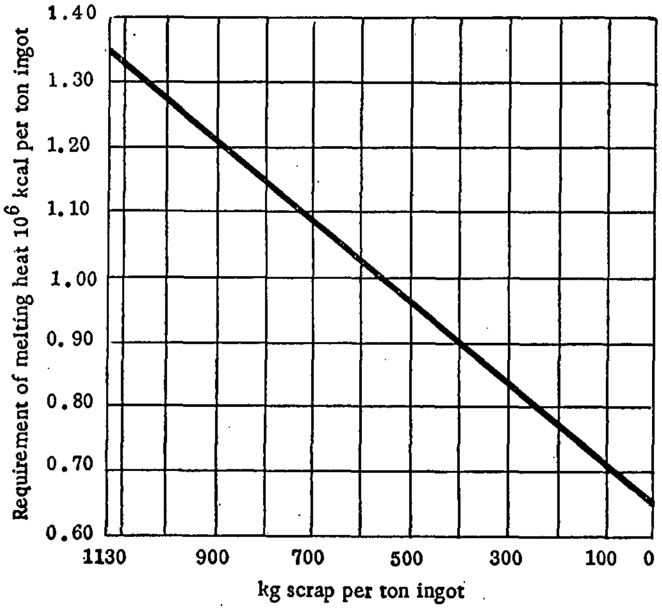
Table 17

The heat consumption per ton of open hearth steel for the charging conditions indicated in Table 16 must lie between the heat consumption of the Duplex process (0 kg. scrap; 100 kg. stahleisen) of 0.65×10^6 kcal/ton and that of the pig iron—ore process of 1.35×10^6 kcal/tons. Linear interpolation gives the calculation:

$$0.65 \times 10^6 + \frac{(1.35 \times 10^6 - 0.65 \times 10^6) \times 300}{1130}$$

$$= 0.836 \times 10^6 \text{ kcal/ton}$$

Steel production per hour of furnace with a heat supply of 10.9×10^6 kcal/hour..	$\frac{10.9 \times 10^6}{0.836 \times 10^6}$	=	13.04 tons/hour
Melting operational costs with furnace costs of 334.30 DM/hour...............	$\frac{334.30}{13.04}$	=	25.63 DM/ton
Costs of ladle working..................		=	2.61 DM/ton
Costs of casting operation..............		=	10.74 DM/ton
Working costs.........................		=	38.98 DM/ton
Costs of charge materials acc. to Table 16..		=	226.16 DM/ton
TOTAL PRODUCTION COSTS		=	265.14 DM/ton



Interpolation diagram for establishing the requirement of melting heat for the various scrap-Duplex proportions per ton of ingot (stahleisen proportion being approx. 100 kg).

IV. (a) *Duplex converter—open hearth furnace*

The minimum working amount of pig iron, 100 kg. per ton of steel, is melted in the open hearth, and most of the charge is molten Duplex metal from the converter, together with the arising of 300 kg. of scrap per ton steel as mentioned in process I.

The costs of the Duplex metal are shown in Table 15, whilst Table 16 shows the costs of the composite open hearth steel charge per ton of open hearth steel. Total production costs are shown in Table 17 to be DM265.14 per ton.

IV. (b) *Duplex converter—electric arc furnace*

The whole of the pig iron is charged into the converter, and blown into Duplex metal, and this, together with 300 kg. of scrap per ton of steel is charged into the electric arc furnace. Table 18 gives the costs of materials, and Table 19 the steel production costs.

Table 18

BALANCE OF MATERIALS AND COSTS OF MATERIALS FOR THE PRODUCTION OF 1 TON ELECTRIC STEEL WITH A DUPLEX PROPORTION OF 778.6 KG. IN CHARGE

	Quantity kg./ton ingot	Price DM/ton		Amount DM/ton ingot
Charge:				
Metal charge:				
Duplex metal......	778.6	225.62		175.67
Scrap...............	300.0	170.00[a]		51.00
Additions:				
80% of addition costs in open hearth steel; due to better yield........	6.4			4.39
TOTAL METAL CHARGE	1,085.0			
Fluxes and carburizing agents:				
White lime........	80.0	40.00	3.20	
Fluorspar.........	4.0	120.00	0.48	
Carburizing agents.	2.0	70.00	0.14	
TOTAL FLUXES AND CARBURIZING AGENTS	86.0		3.82	3.82
Output:				
Casting waste material	5.0	-170.00[b]	-0.85	
Electric slag.........	96.0			
TOTAL CREDITS			-0.85	-0.85
Melting loss.........	79.0			
Ingot...............	1,000.0			
TOTAL COSTS OF MATERIAL IN CHARGE				234.03

[a] Price assumed, like the other prices, hence the calculated costs of steel are fictitious.

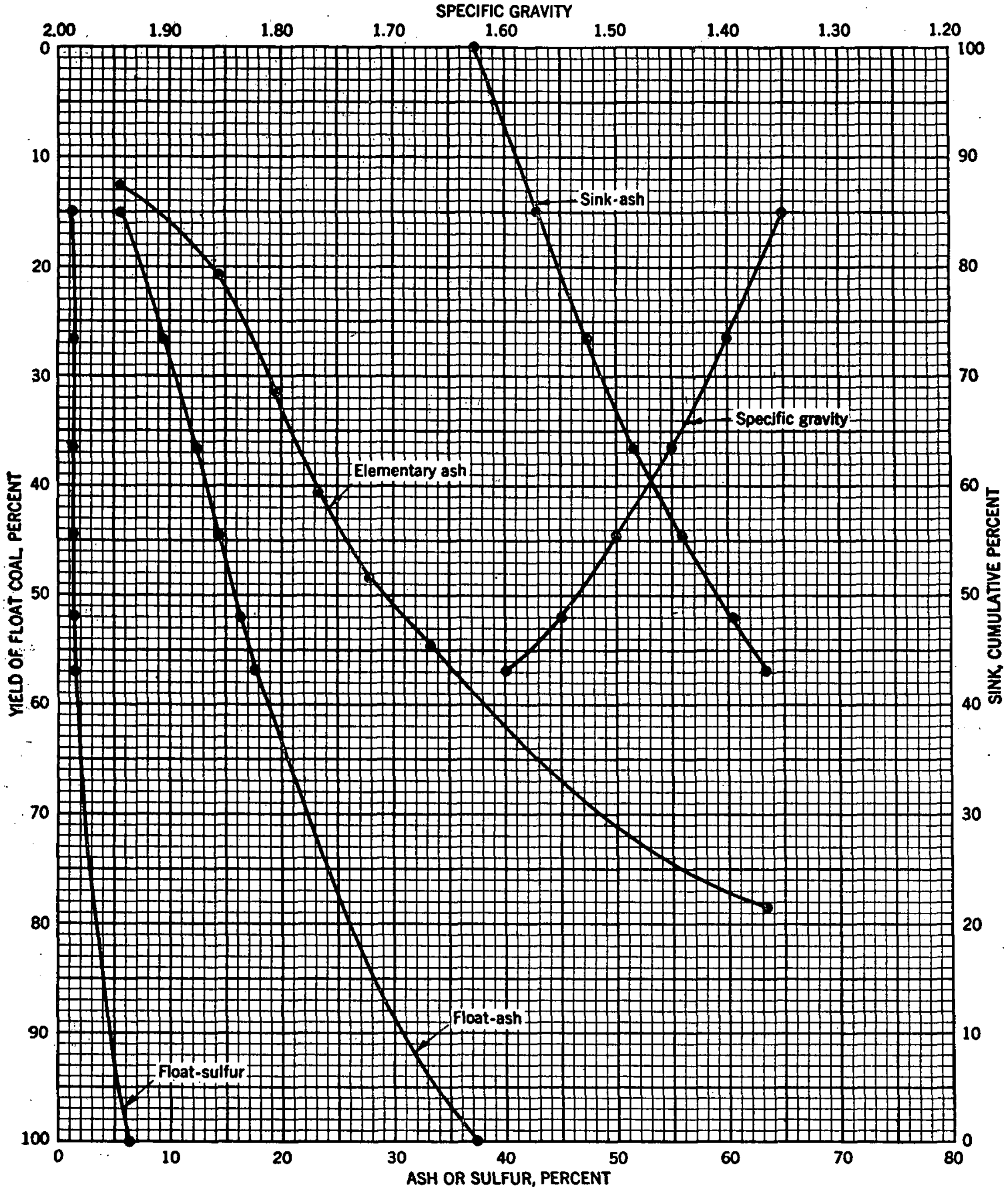
[b] Credit.

Table 19

PRODUCTION COSTS OF 1 TON OF ELECTRIC STEEL WITH DUPLEX METAL IN A 60-TON ELECTRIC ARC FURNACE

	Quantity per ton ingot	Price	Amouut DM/ton ingot
Melting operational costs......			6.00
Power.......................	400 kWh.	60.— DM/- 1,000 kWh.	24.00
Electrodes and nipples........	6 kg.		12.50
Ladle working (costs same per ton as in open hearth steelworks)..................			2.61
Casting operation (costs same per ton as in open hearth steelworks)...............			10.74
Working costs...............			55.85
Costs of materials in charge...			234.03
TOTAL PRODUCTION COSTS			289.88

According to previous calculations, the furnace costs of the 60-ton arc unit were shown to be DM120 per hour. The output is increased in the case of IV (b) by about two-thirds, i.e. from 12 to 20 tons per hour, thereby reducing the operational costs to $\frac{120}{20}$ = DM6 per ton. With an assumed power consumption reduced by an estimated 40% to 400 kWh. per ton, Table 19 gives a total production cost of DM289.88 per ton.



Specific gravity fractions	Elementary data				Computed cumulative data					
	Weight grams	Percent			Float, cumulative percent			Sink, cumulative percent		
		Weight	Ash	Sulfur	Weight	Ash	Sulfur	Weight	Ash	Sulfur
Float - 1.35		15.0	5.7	1.22	15.0	5.7	1.22	100.0	37.3	6.30
1.35 to 1.40		11.6	14.5	1.55	26.6	9.6	1.36	85.0	42.9	7.19
1.40 to 1.45		9.9	19.6	1.64	36.5	12.3	1.44	73.4	47.4	8.08
1.45 to 1.50		8.1	23.2	1.63	44.6	14.3	1.47	63.5	51.7	9.09
1.50 to 1.55		7.5	27.9	1.96	52.1	16.2	1.54	55.4	55.9	10.18
1.55 to 1.60		4.8	33.4	2.19	56.9	17.6	1.60	47.9	60.4	11.47
Sink - 1.60		43.1	63.3	12.50	100.0	37.3	6.30	43.1	63.3	12.50

Fig. 8. Washability chart of the Barro Branco coal. Size 10 – 0.3 mm.

Comparative Investment Costs for Different Steelmaking Processes

CHARLES F. RAMSEYER

The subject matter of this paper—a study of the relative capital investments required to make steel by different processes—was suggested by the United Nations Secretariat. Since we had recently made a detailed study of the economics of steelmaking by the open hearth as compared to the basic electric furnace process, the extension of our earlier work to cover other processes seemed likely to yield further information of considerable general interest. At the further suggestion of the Secretariat, the study was broadened to include figures for plants of several different sizes to show the effect of plant size on investment cost per unit of capacity. Besides developing tentative general layouts for plants of four different capacities for each of the six processes considered, a host of related items had to be taken into account; furnace and equipment capacities had to be determined, numerous engineering approximations—sometimes known as informed guesses—made, and many simplifying assumptions agreed upon, before we were in a position to arrive at the cost estimates which form the basis of this paper.

Originally we had hoped to be able to make our comparisons on the basis of actual capital cost figures for existing steelmaking plants of various types and sizes, but this turned out to be a very unsatisfactory method of approach. To cover all the processes meant that figures from more than one country would have to be used and really comparable cost figures from separate countries are very difficult to obtain because of differences in accounting practices, especially as to what costs should be included and what left out—as well as the influence of inflationary changes since, and even during, their construction. And this is not to mention the effect of diverse local political conditions, and whether a given plant was built on the basis of relatively limited private means or with ample government funds. In the end, we decided that the only way to do the job properly, was to estimate separately the actual building and equipment requirements of each size plant for each one of the six processes—and then to arrive at the desired over-all plant estimates from the sum of the parts required in each case.

While we believe the estimates reported herein give a valid comparison between different steelmaking processes from the capital investment cost point of view, and thus we hope will be found to be of considerable general interest, they certainly should not be used as the principal basis for favouring one type or size of steel plant over another. The factors involved in deciding on the most suitable process and layout for a new steel plant are so numerous and varied, that the best solution, i.e., the one that takes all factors into account, including the magnitude of the funds available, is usually arrived at only after many alternatives have been analysed, compared, and discussed at length from the differing viewpoints of all parties concerned.

SCOPE OF THE WORK

The whole reason for this paper lies in the plant capital investment cost estimates which it gives for 22 different types and sizes of steelmaking plants or "meltshops". The annual capacities of these plants are grouped into four sizes, for the production of 100,000, 250,000, 500,000 and 1,000,000 metric tons of ingots per year. The six different processes covered are listed below:

1. *The acid Bessemer process*—with auxiliary electric furnace capacity for melting the home or plant scrap, plus a small amount of purchased scrap;
2. *The basic Bessemer process*—also with electric furnaces for melting scrap;
3. *The basic open hearth process*—based on the use of stationary furnaces;
4. *The basic electric furnace process*—based on the use of top charge electric furnaces;
5. *The acid Bessemer-basic open hearth duplex process*—based on acid bessemer converters and tilting basic open hearth furnaces;
6. *The acid Bessemer-basic electric furnace, duplex process*—based on acid bessemer converters and basic electric furnaces.

LIMITATIONS OF THE ESTIMATES

As indicated, the 22 plant capital investment cost estimates were not derived from any existing steelworks investment cost figures, but were developed on the basis of separate unit building, equipment, and erection costs. They apply purely to the 22 hypothetical steel plants. The estimates thus cannot be said to represent any actual plants—an obvious disadvantage.

On the other hand, the results derived from the method employed probably afford considerably more accurate comparative figures for the different size plants and processes studied than could possibly be obtained from the original costs of various actual plants, since actual plants would of necessity have been built at different times, for different clients, in different countries, and therefore under anything but comparable conditions. Our figures, on the other hand, are all based on the same methods of calculation and estimation throughout.

Most of the building estimates were based on freehand sketches, developed only in sufficient detail to give the proper number, length and width of the various buildings required to house the furnaces and other equipment and facilities needed by each plant. The size and number of these were separately developed by a study of the metallurgical and operating requirements of each one of the 22 cases.

GENERAL CONSIDERATIONS

All the figures given are based on plants designed for Latin-American conditions. Due to the general scarcity of scrap in Latin America, the assumption was made that each meltshop would be part of an integrated steelworks having sufficient pig iron capacity to supply the full amount of hot metal each meltshop needed for its annual output. A small supply of outside purchased scrap, taken equal to 10% each plant's ingot capacity, was assumed to be locally available in addition to the plant's internal mill or home scrap. All the rest of the metallic charge going into the steelmaking furnaces would be molten pig iron or "hot metal". Presumably, in most cases this would come from blast furnaces of appropriate size, except that in some of the smaller plants it could also come from electric pig iron furnaces.

A meltshop capital cost estimate customarily includes a full complement of ingot moulds and cars for initial operation of the plant. Before their number and cost can be estimated, the ingot size for each plant must be established. The ingot size in turn is largely determined by the requirements of the rolling mills, and the size and type of the latter are set by the products to be rolled. Some assumptions therefore had to be made as to the kinds of products that were to be rolled from the ingots poured in our hypothetical meltshops.

The products to be rolled of course depend upon the local market. To have a common basis to work on, it was assumed that all 22 meltshops would be part of the first integrated steelworks to be built in that section of the country. This made reasonable the further assumption that the finished steel product mix would be roughly the same in all cases, i.e., reinforcing bars, wire rod and hot rolled (hand mill) sheets in the case of the smaller 100,000- and 250,000-ton plants with the addition of structurals, rails and an increased proportion of flat rolled products—plate, sheets, tin plate and welded pipe—in the case of the 500,000- and 1,000,000-ton plants.

Nothing is included in any of the estimates for the cost of land. Under Latin-American conditions, however, it was taken for granted that plant sites of ample size would be available. This factor influences plant layout, since in the case of the basic Bessemer plants, for instance, it permits the use of separate buildings where conventional European practice, due to space limitations, would be more likely to concentrate several different operations under one roof. While the more compact European design would perhaps permit some reduction in initial investment cost, it would certainly interfere with the large scale future expansion of the plants normally to be expected in Latin America. An attempt was therefore made to lay out all our hypothetical plants so that they could be enlarged in the future as the country's steel requirements rose.

METALLURGICAL CONSIDERATIONS

While the cost per ton of capacity of a new meltshop is an important point in the choice of the best steelmaking process for a new project, it must be remembered that in many cases the analysis of the raw materials available is likely to be the deciding factor. Also, while the meltshop is a most important department of an integrated steel works, it normally accounts for only from 8 to 14% of the total investment required to build a complete integrated steel works.

The composition of the available raw materials is such an influential factor in the selection of the steelmaking facilities to be used in any given case, that we propose to review the subject briefly here.

Case 1. The acid Bessemer process

For pig iron to be suitable for the acid Bessemer process, its phosphorus content must not exceed a maximum of approximately 0.080%. The phosphorus content of pig iron in most cases is determined by the amount present in the iron ore used since the majority of coals are relatively low in phosphorus and some source of low phosphorus limestone can usually be found without too much difficulty. On the other hand, the supply of low phosphorus iron ore throughout the world is relatively limited, although Brazil and Venezuela have vast quantities of this grade of ore and lesser deposits are known to exist in other Latin-American countries.

Since acid Bessemer steel is more difficult to fabricate than basic steel for many uses, it suffers under a considerable sales disadvantage, the steel industry being a highly competitive one. Because of this, no estimate for a million ton acid Bessemer plant was included in our study.

Since the scrap melting capacity of the Bessemer process is limited, electric furnaces were provided in all the Bessemer plant estimates, not only for melting the home mill scrap, but also the 10% of outside purchased scrap which was assumed in all cases. The resultant electric furnace ingots would provide an appreciable tonnage of low phosphorus steel for some of the products for which acid Bessemer steel is not suited.

The superior qualities of basic steel have also made acid Bessemer steel non-competitive except for a limited number of more or less special products, such as high sulfur screw stock and skelp for butt welded pipe.

Should an individual country desire to do so, it could undoubtedly satisfy the great majority of its steel requirements with acid Bessemer steel, especially if the steel from the 25% or so of all scrap electric furnace heats were reserved for those uses for which acid Bessemer was definitely unsuitable. In the face of international competition, however, with acid Bessemer steel excluded from the generally accepted specifications for such major products as rails and structurals as well as many flat rolled uses, it would be practically impossible to dispose of a large output of it for those uses unless the use of other types of steel for such products were embargoed or otherwise regulated.

Case 2. The basic Bessemer

As in the case of the acid Bessemer process, whether the basic Bessemer can be used at all or not in a given instance, is primarily dependent on the composition of the available iron ore. This is true, at least in the present state of development of basic Bessemer practice, since pig iron suitable for conversion into steel by the normal air blown basic Bessemer process must contain a minimum of from 1.75 to 2% of phosphorus. Oxygen enrichment of the blast is expected to make the handling of lower phosphorus content metal possible in the future however; in fact, this has already been proved to some extent in actual practice. In basic Bessemer plant capital cost estimates, this currently changing situation was taken into account by providing an oxygen plant in all cases with sufficient capacity to enrich the air blast going to the converters up to 30 or 35% oxygen.

It is probable that the use of 30 to 35% oxygen in the converter blast would permit pig iron containing as little as 1.50% of phosphorus to be used. Theoretically, with still higher oxygen enrichments, even lower phosphorus contents should be feasible. Up to the present time all attempts to use very high oxygen enrichments or pure oxygen have always resulted in very rapid destruction of the lining of the basic converter vessels, especially the bottoms, due to the extremely high local temperatures developed. The resultant high refractory cost combined with greatly reduced vessel availability have made it completely uneconomic.

In contrast to the finished product situation described in the case of acid Bessemer steel, we have assumed, in setting up our figures, that steel made by the basic Bessemer process, particularly with an oxygen enriched blast, would be comparable in quality to that made by the open hearth and therefore metallurgically suitable for the entire range of steel products.

The basic Bessemer plants require a greater number of auxiliaries than acid Bessemer shops. These include facilities for burning lime and dolomite, and a special plant to prepare the high phosphorus basic Bessemer slag for use as fertilizer. Such slag is widely used in western Europe for agricultural purposes, and the income from its sale represents an important credit to the cost of producing basic Bessemer steel.

Case 3. The basic open hearth

More of the world's steel is made by this process than by any other, primarily because most of the steel produced in North America is basic open hearth steel. One of the main reasons why the open hearth process replaced the Bessemer process in the United States was that, unlike the Bessemer, it proved able to make steel from scrap as well as from pig iron, and as the country became more highly industrialized, scrap became increasingly plentiful. In addition, the bulk of the iron ore on the North American continent had a relatively low phosphorus content, much too low for the basic Bessemer process, but on the other hand too high on the average for the acid Bessemer. Due to its ability to handle phosphorus contents up to 1% and to make steel from any and all proportions of scrap and pig iron, the basic open hearth process became, and still is, by far the most popular method of producing steel in North America.

Case 4. The basic electric furnace

While up to the present time the basic electric furnace has been used largely for scrap melting, during the past few years thousands of basic electric furnace heats have been made successfully in the United States from charges containing up to 40% or more of hot metal and there is apparently no reason why the refining of substantial amounts of hot metal in the electric furnace should not become a more widely spread practice in the future.

Due to the necessarily oxidizing nature of the combustion reactions taking place in an open hearth flame, blast furnace hot metal can be charged to an open hearth furnace in amounts up to 60 to 65% of the total metallic charge before it begins to have an adverse effect on the over-all heat time. In the electric furnace, on the other hand, the limited information available at the present time seems to indicate that when the amount of hot metal exceeds 40 to 50% of the total charge, heat times tend to lengthen and the productive capacity of a given furnace unit begins to taper off. This has been allowed for in arriving at the size and number of electric furnaces required for the different size electric furnace plants in our capital cost estimates.

Cases 5 and 6. The duplex processes

The influence of raw material composition, particularly as regards phosphorus content, on the selection of the steelmaking process to be employed in a given case has already been seen to be of primary importance in the case of the two Bessemer processes. If the phosphorus in the ore is low enough, the acid Bessemer process is technically feasible, while with a very high phosphorus ore, the basic Bessemer process is not only feasible, but practically mandatory, since both the basic open hearth and the basic electric furnace processes have increasing difficulty in eliminating phosphorus after its percentage in the pig iron begins to exceed 0.500% and these processes are not, to our knowledge, used at all when the phosphorus content in the hot metal exceeds 1%.

In cases where the phosphorus in the pig iron is too high to be readily handled by the basic open hearth or electric furnace processes, but still not high enough for the basic Bessemer, i.e., where the making of two slags for proper dephosphorization is required, present metallurgical thinking tends to favour the acid Bessemer—basic open hearth duplex process. The possibility of duplexing therefore needs to be considered primarily in those cases where the available iron ore would yield a pig iron with a phosphorus content of between 0.75 and 1.50%. To help simplify the handling of the two slags per heat normally required to refine such relatively high phosphorus metal down to the low limits of finished steel products, duplexing is usually carried out in tilting rather than stationary open hearth furnaces.

Scarcity of scrap, coupled with the availability of fairly high phosphorus hot metal, therefore favours the use of duplex practice. While duplexing from the acid Bessemer into the basic electric furnace has been known for at least 30 years, it has never become a very common steelmaking method, even though the fact that the electric steel furnace is normally a tilting unit would naturally be in favour of its use for this purpose. Since it was felt that the modern electric furnace was well suited for duplexing, particularly in a small plant, we included figures for it in the estimates.

Other steelmaking processes

The six conventional steelmaking methods for which plant capital cost estimates are given by no means cover the full range of possibilities, but there had to be some limit to the number of processes studied. No consideration, for instance, was given to the fact that in English practice, heated mixers, which perhaps might be considered as half-way between a normal pig iron mixer and a tilting open hearth furnace, are used in a number of plants to pre-refine blast furnace hot metal before it is charged into the open hearths.

Another recently developed practice for the conversion of pig iron into steel, which is about to be used on an industrial scale in Austria, is the use of a high pressure oxygen jet blowing from above down into a charge of molten pig iron held in a sort of closed bottom Bessemer vessel. Too little is known about this process for any figures to be given here. Nor was any consideration given to basic Bessemer duplex possibilities, it being assumed that with an oxygen enriched blast steel of a quality comparable to that produced in the open hearth furnace would be made directly in the basic Bessemer, without the need of duplexing. On the other hand, while oxygen plant facilities were provided for enriching the basic Bessemer air blast, none was provided in the acid Bessemer estimates. In the other four cases, small oxygen plants were included with sufficient capacity for lance decarburizing of the bath in the furnaces, but no more.

The above brief discussion of a few of the more important metallurgical factors influencing the choice of steelmaking processes, has been given, as mentioned earlier, to emphasize the fact that the process showing the lowest cost in our plant estimate tabulations is by no means necessarily the best process for a given new project.

ENGINEERING AND TECHNICAL ASSUMPTIONS

As indicated earlier, over-all plant metallic balances were first calculated for all 22 different hypothetical plants in order to establish the relative amounts of scrap and hot metal required in each case, and these figures in turn were used to fix the number and size of melting units needed. To simplify matters, a 70% yield of finished products from ingots was assumed throughout, and the amount of plant scrap returned from the rolling mills for remelting was set at 23% of ingot production. In actual practice both of these figures would vary somewhat with ingot size, type of rolling mill and heating furnaces, and especially with the kind of finished products made.

In setting up capital cost estimates, certain general assumptions were made as to the size of ingots, availability of power, ingot pouring practice, limiting number and sizes of melting units, etc.

In the case of the rolling mills, it was assumed that all plants of the same size would have the same type, number and size of mills, regardless of which steelmaking process the plant was designed for. For the smallest size plants, i.e., those with a capacity of 100,000 tons of ingots per year, a small 3-high mill was assumed capable of rolling 10 in., 700-kg. ingots into billets and sheet bar. For the 250,000-ton per year plant, a small 2-high bloomer was supplied and the billet and sheet bar mill expanded so as to roll some medium structural shapes. This plant would cast 16 in. 1,700-kg. ingots.

For the 500,000-ton plant, a somewhat larger 2-high bloomer was assumed, together with a single stand reversing hot mill for rolling strip steel directly from ingots. Three-ton, 20 in. ingots were assumed for all the 500,000 ton plants, with slab ingots of the same weight for the reversing strip mill. The 1,000,000-ton plant was provided with a combination blooming-slabbing mill and a semi-continuous strip mill. It was assumed that the meltshop would cast 8-ton ingots, either 30 in. square or equivalent slab sizes.

The above ingot sizes were chosen on the basis that the rolling mills provided would be able to roll the required annual ingot tonnages in 15 to 18 turns a week, at the rate of 15–20 ingots an hour. As already mentioned, the same size ingot was assumed for all plants of the same size, regardless of the steel melting process. In actual practice this would probably not be true; ingots of more than one size would undoubtedly be poured in the larger plants, and they would probably not have the same dimensions in the case of all six processes.

The number of steelmaking furnaces in each plant was fixed on the basis of over-all plant metallic balances. In the case of the straight acid and basic Bessemer plants, sufficient supplementary melting capacity was provided to melt both the 23% plant scrap as well as the assumed 10% of purchased scrap. It was assumed that only a relatively small amount of scrap would be melted in the vessel of the acid Bessemer. In the case of the basic Bessemer plants, on the other hand, due to the fact that the use of an oxygen enriched blast was contemplated in all cases, it was assumed that 15% of scrap, based on the total Bessemer metallic charge, would be melted in the converters. Fifteen per cent may be a somewhat conservative figure, but we did not feel justified in using a higher percentage in the present state of the use of oxygen in basic Bessemer practice. Electric furnace capacity was then provided to melt the remaining or surplus scrap.

In general a minimum of three open hearth units or two electric furnaces was assumed in all examples so as to provide a reasonably constant ingot output during necessary furnace outages. In the smallest size plant, however, providing three units would have meant using open hearths of only 43-ton capacity. These were thought to be too small for good practice under modern conditions, so two 65-ton furnaces were provided instead.

For each size plant and steelmaking process, we tried to lay out the individual meltshops so that, while they would be as modern and efficient as possible, they would not be overly elaborate or expensive under our assumed Latin-American conditions.

In the smallest size acid Bessemer plant, two or three-ton Bessemer units could theoretically be used. However, we decided to use a minimum converter size of 10 tons of finished steel per blow or heat, so as to improve the yield and avoid having to pour an excessive number of heats. With two 10-ton vessels and a single blower, the requirements of the 100,000-ton plant can readily be met by operating one shift per day, six days per week. Similarly, in the case of the 250,000 and 500,000 ton plants, somewhat smaller converters than those chosen could have been used by operating more shifts per week, but this again would have unduly increased the number of heats to be handled, so that two-turn operation with larger vessels was chosen as representing better, more economical steelmaking practice.

In the case of the 1,000,000-ton plant, seven 215-ton open hearths were selected instead of five 300-ton units to hold down the capital cost of buildings and cranes. On the other hand, in selecting the furnaces for the 1,000,000-ton size electric furnace plant, 22 ft. diameter units, the largest now operating in the United States, were used. Six furnaces of this size should easily accommodate the 106-ton high hot metal practice heats necessary to produce a million tons of ingots per year. The 106-ton figure is based on an average melting rate of 4½ heats per day. When melting 100% cold scrap charges, i.e., in the case of the auxiliary melting units provided for the various acid and basic Bessemer plants, a somewhat higher melting rate of five heats per day was assumed.

In connexion with the supply of power to the meltshops—especially in the case of the electric furnace plants—it was assumed, since they were to be part of an integrated steel plant operation in all cases, that a substantial amount of electric generating capacity would necessarily be available from the blast furnace power stations. Although it was recognized that a somewhat larger generating capacity might be necessary to cover the added plant power requirements for electric furnace melting, no allowance was made for this in the electric furnace estimates. Not only would it have been difficult to estimate this accurately, but if the steelworks happened to be so located that it could be readily connected to a large hydroelectric or other public power system, the small amount of additional plant power required for electric melting could be purchased, in which case no increase in the steelworks capital cost would be necessary at all.

Furthermore it must be remembered that an electric furnace operation based on a 60% hot metal charge and using high grade Latin-American decarburizing ore plus oxygen for working the heats would consume appreciably less power than conventional North American all-scrap charge electric furnace practice. The omission from the electric furnace estimates of anything for the cost of added power generating capacity was partly compensated for by providing a separate sub-station with sufficient capacity to supply the full requirements of the furnaces from an outside source of high voltage power.

The possibility of teeming steel into ingot moulds placed on stationary plates in the pouring pit versus the use of ingot mould cars, or "pouring on wheels", was given some consideration. Our conclusion was that while pouring on plates, with the resultant handling of all ingots and moulds in the pouring pit, was feasible for small productions up to perhaps 75,000 tons per year, pouring on cars would probably be preferable in most cases for larger plants. Although ingot outputs up to 250,000 tons do not justify provision of a separate stripper building, pouring on cars permits the ready removal of the ingots and moulds from the pit side pouring platforms to a separate stripping area. This can be at one end of the pit in an extension of the pouring bay. With the moulds on cars all mould preparation work can be done here, or kept out of the pouring pit entirely if desired.

The pouring of billet size ingots was also seriously considered in the case of the 100,000-ton plants but was discarded because of the need of being able to roll sheet bar for the production of hot rolled and galvanized sheets, two types of steel products likely to be of importance for any new integrated plant. Since the old hand hot mill method is still the only feasible way of rolling a limited tonnage of sheets in small plants designed primarily to roll bars and rods, the ingot size cannot be reduced below that permitting the rolling of sheet bar; 10 in. is about as small as it is feasible to go.

It will be noted that Table 1 shows the exact calculated sizes of the heats to be made by the major furnace units in the various cases, rather than just rounded off figures. This was done to keep the estimates as comparable as possible. On the other hand, more or less standard size units were used for most meltshop auxiliaries such as charging boxes, iron and steel ladles, cinder pots, etc.

Development of plant layouts

In the case of the two smaller size plants, the ingots are stripped in an extension of the pouring pit bay by the pit service crane using a portable stripper. Mould preparation is done either in a simple lean-to parallel to the pouring pit, or in a separate building, depending on the size and type of the plant. A separate stripper building with its own stripper crane or cranes was only provided for the larger 500,000 and 1,000,000-ton plants.

While all the layouts used as a basis for the estimates were made with a view toward the future expansion of the plants, it turned out actually that somewhat less provision for future expansion was provided in the case of the smaller plants than for the larger ones, since any appreciable expansion of the capacity of a small steel plant would be more likely to call for larger size furnaces rather than merely an increase in their number. The two smaller size plants were therefore kept as compact and inexpensive as possible, even though in actual practice, it might be better to provide them with somewhat bigger buildings and more space between units with a view to the installation of larger furnaces in the future.

In this matter of future expansion, large basic open hearth and electric furnace plants have an advantage over Bessemer and duplex plants in that their capacity can be increased merely by adding more units to the initial line of furnaces. Expansion is more difficult in the case of the Bessemer and duplex shops, due partly to the necessity of providing separate scrap melting facilities, and partly to the fact that in their layouts provision must be made not only for handling and pouring Bessemer steel, but also for charging scrap to the converters, blown metal to the duplexing furnaces, and scrap again to the auxiliary scrap melting units. Purely from the point of view of future expansion, it would therefore probably be best to place the Bessemer plant auxiliary melt-shops in an entirely separate building. However, for this study we have put them all under the same roof; the mixer, converters, scrap melting units and pouring facilities are all together. This results in a lower initial plant cost.

A flat plant site of ample size was assumed in all cases, and standard gauge railroad trackage was therefore used throughout in the estimates.

Cost basis of the estimates

All the costs given in the tables are based on United States prices as of the summer of 1952, F.A.S. nearest port, for the complete meltshop facilities indicated, erected and ready to operate, including suitable allowances for contingencies, engineering and overhead during construction. No allowance is made for working capital. Under Latin-American conditions the total

Table 1

NUMBER AND SIZE OF MAJOR FURNACE UNITS

Four different plant capacities and six different steelmaking processes

Steelmaking process	*Assumed total annual plant capacities (metric tons of ingots per year)*			
	100,000 tons	*250,000 tons*	*500,000 tons*	*1,000,000 tons*
1. Acid Bessemer				
(*a*) No. and capacity of acid Bessemer converter vessels	Two 10-ton converters	Two 15-ton converters	Two 15-ton converters	Omitted
(*b*) No., shell diameter, and heat size of auxiliary scrap melting electric furnaces (top charge)	One 13 ft. electric furnace tapping 17.3 ton heats	Two 14 ft. electric furnaces tapping 22.7 ton heats	Two 17 ft. electric furnaces tapping 42.5 ton heats	
2. Basic Bessemer				
(*a*) No. and capacity of Basic Bessemer converter vessels	Two 10-ton converters	Three 11-ton converters	Three 22-ton converters	Three 44-ton converters
(*b*) No., shell diameter, and heat size of auxiliary scrap melting electric furnaces (top charge)	One 11 ft. electric furnace tapping 10.3 ton heats	Two 12 ft. electric furnaces tapping 13.0 ton heats	Two 14 ft. electric furnaces tapping 26.1 ton heats	Two 18 ft. electric furnaces tapping 31.8 ton heats
3. Basic open hearth				
(*a*) No. and capacity of stationary basic open hearth furnaces	Two 65-ton open hearth furnaces	Three 118-ton open hearth furnaces	Four 182.3-ton open hearth furnaces	Seven 215-ton open hearth furnaces
4. Basic electric furnaces				
(*a*) No., shell diameter, and heat size of top charge electric furnaces	Two 16 ft. electric furnaces tapping 33.0 ton heats	Two 20 ft. electric furnaces tapping 81.8 ton heats	Four 20 ft. electric furnaces tapping 81.2 ton heats	Six 22 ft. electric furnaces tapping 106.7 ton heats
5. Duplex-acid Bessemer—Basic open hearth				
(*a*) No. and capacity of acid Bessemer converter vessels	Omitted	Two 25-ton converters	Two 25-ton converters	Three 30-ton converters
(*b*) No. and capacity of tilting open hearth furnaces		Three 70-ton tilting open hearth furnaces	Three 144-ton tilting open hearth furnaces	Five 176-ton tilting open hearth furnaces
6. Duplex-acid Bessemer—Basic electric furnace				
(*a*) No. and capacity of acid Bessemer converter vessels	Two 12-ton converters	Two 20-ton converters	Two 25-ton converters	Three 30-ton converters
(*b*) No., shell diameter, and heat size of top charge electric furnaces	Two 14 ft. electric furnaces tapping 22.3 ton heats	Two 18 ft. electric furnaces tapping 53.2 ton heats	Three 19 ft. electric furnaces tapping 70.4 ton heats	Five 20 ft. electric furnaces tapping 83.5 ton heats

costs shown would be increased by the cost of ocean freight and handling charges from North American ports. In the case of the basic Bessemer plants, European prices were used in part for some of the auxiliary equipment. These, however, were correlated with current United States prices as far as possible.

Building costs were developed on a cost per square foot basis; they are for structural steel, mill type buildings in all cases. The cost of building services is shown separately to emphasize the fact that this is an important item of cost. The cost of cranes, furnace units, and other auxiliary facilities are based on figures from various United States suppliers.

Steam driven turbo-blowers for the Bessemers were assumed throughout, using increasingly higher steam temperature and pressure conditions for each next larger size plant. This was done on the supposition that, as the size and total capital investment in a project increases, the higher priced supervisory personnel required to operate the more complicated, higher temperature and pressure units can be readily justified by the saving in fuel costs.

A final important point before going on to the tabulated figures, is that the estimates, while based on present day United States prices, assume normal business conditions under which the plants could be built without any costly delays in the delivery of equipment, and without having to pay premium prices for labour. During the past few years this has distinctly not been the case in the United States—which will explain why in certain cases our estimates may appear somewhat low to United States iron and steel men. By picking and choosing among the known capital cost figures of actual steel plants built since the war, it would probably be possible to find instances where plants approximating our hypothetical capacities, and employing one or another of the six processes studied, had been built for considerably more or less than the figures given in the estimates presented herein. This however does not necessarily detract from their value for comparative purposes.

THE ESTIMATES

The estimates of comparative investment cost finally arrived at for each one of our 22 hypothetical steel plant meltshops are given in a series of four tables. (Tables 2, 3, 4 and 5.) Each table gives figures for the six different assumed processes. The estimates are further divided into a series of major sub-items, the main ones covering buildings, building services, cranes, furnaces and auxiliaries, charging and pouring equipment, and scrap and material handling equipment.

Table 2

ESTIMATED PLANT CAPITAL INVESTMENT COSTS REQUIRED TO PRODUCE 100,000 TONS OF INGOTS PER YEAR BY DIFFERENT PROCESSES

Item	*Description*	*Case 1* *Acid Bessemer process with auxiliary electric furnaces*		*Case 2* *Basic Bessemer process with auxiliary electric furnaces*		*Case 3* *Standard stationary basic open hearth process*		*Case 4* *Basic electric furnace process*		*Case 5* *Acid Bessemer-basic open hearth-tilting furnace-duplex process*		*Case 6* *Acid Bessemer-basic electric furnace-duplex process*	
		$	%	$	%	$	%	$	%	$	%	$	%
1.	Buildings	571,300	14.3	687,900	11.6	1,011,500	21.6	674,700	20.4	Omitted		795,400	17.0
2.	Building services	112,900	2.8	143,700	2.4	250,300	5.4	129,400	3.9			163,500	3.5
3.	Cranes	488,700	12.2	607,400	10.3	525,500	11.2	363,900	11.0			472,300	10.1
4.	Furnaces and auxiliaries	1,723,500	43.4	1,933,100	32.7	1,350,900	28.9	916,800	27.8			2,135,700	45.7
5.	Charging and pouring	533,000	13.3	636,600	10.8	935,100	20.0	537,700	16.3			462,300	9.9
6.	Scrap and material handling	80,200	2.0	79,300	1.3	58,300	1.2	141,400	4.3			133,700	2.9
7.	Mould preparation and stripping	40,100	1.0	40,100	0.7	40,100	0.9	40,100	1.2			40,100	0.9
8.	Electric substation	171,300	4.3	206,300	3.5	160,000	3.4	202,900	6.1			227,800	4.9
9.	Miscellaneous	239,700	6.0	617,100	10.4	341,000	7.3	292,800	8.9			239,700	5.0
10.	Calcining and slag plants	2,800	0.7	966,200	16.3	2,800	0.1	2,800	0.1			2,800	0.1
11.	Total estimated cost	3,963,500	100.0	5,917,700	100.0	4,675,500	100.0	3,302,500	100.0			4,673,300	100.0
12.	Cost per ton of ingot capacity	39.64		59.18		46.76		33.03				46.73	

Table 3

ESTIMATED PLANT CAPITAL INVESTMENT COSTS REQUIRED TO PRODUCE 250,000 TONS OF INGOTS PER YEAR BY DIFFERENT PROCESSES

Item	*Description*	*Case 1* *Acid Bessemer process with auxiliary electric furnaces*		*Case 2* *Basic Bessemer process with auxiliary electric furnaces*		*Case 3* *Standard stationary basic open hearth process*		*Case 4* *Basic electric furnace process*		*Case 5* *Acid Bessemer-basic open hearth-tilting furnace-duplex process*		*Case 6* *Acid Bessemer-basic electric furnace-duplex process*	
		$	%	$	%	$	%	$	%	$	%	$	%
1.	Buildings	1,132,900	19.1	1,327,000	14.6	2,409,800	25.6	1,019,600	20.5	2,033,900	17.6	1,365,600	19.9
2.	Building services	214,600	3.6	261,000	2.9	570,600	6.0	184,800	3.7	436,200	3.8	273,800	4.0
3.	Cranes	697,800	11.7	751,000	8.2	798,100	8.5	545,400	10.9	1,176,000	10.2	706,700	10.3
4.	Furnaces and auxiliaries	2,354,500	39.6	2,767,700	30.3	2,752,100	29.2	1,217,800	24.4	4,870,400	42.1	2,712,600	39.6
5.	Charging and pouring	727,200	12.2	916,800	10.1	1,731,500	18.4	1,000,700	20.1	1,932,700	16.7	843,900	12.3
6.	Scrap and material handling	142,900	2.4	137,300	1.5	69,900	0.7	194,900	3.9	62,300	0.5	151,900	2.2
7.	Mould preparation and stripping	59,400	1.0	59,400	0.6	59,400	0.6	59,400	1.2	59,400	0.5	59,400	0.9
8.	Electric substation	240,400	4.0	261,000	2.9	177,600	1.9	251,600	5.1	198,200	1.7	282,900	4.2
9.	Miscellaneous	278,900	4.8	925,700	10.2	764,400	8.1	412,200	8.3	698,500	6.1	348,900	5.2
10.	Calcining and slag plants	95,200	1.6	1,699,000	18.7	95,200	1.0	95,200	1.9	95,200	0.8	95,200	1.4
11.	Total estimated cost	5,943,800	100.0	9,105,900	100.0	9,428,600	100.0	4,981,600	100.0	11,562,800	100.0	6,840,900	100.0
12.	Cost per ton of ingot capacity	23.78		36.42		37.71		19.93		46.25		27.36	

Table 4

ESTIMATED PLANT CAPITAL INVESTMENT COSTS REQUIRED TO PRODUCE 500,000 TONS OF INGOTS PER YEAR BY DIFFERENT PROCESSES

Item	*Description*	*Case 1* *Acid Bessemer process with auxiliary electric furnaces*		*Case 2* *Basic Bessemer process with auxiliary electric furnaces*		*Case 3* *Standard stationary basic open hearth process*		*Case 4* *Basic electric furnace process*		*Case 5* *Acid Bessemer-basic open hearth-tilting furnace-duplex process*		*Case 6* *Acid Bessemer-basic electric furnace-duplex process*	
		$	%	$	%	$	%	$	%	$	%	$	%
1.	Buildings	1,481,300	17.7	1,753,500	14.1	4,135,900	27.0	2,647,100	26.5	3,332,300	19.5	2,190,900	19.2
2.	Building services	267,200	3.2	333,700	2.7	946,300	6.2	472,200	4.7	675,100	4.0	437,800	3.8
3.	Cranes	874,700	10.5	929,600	7.4	1,163,500	7.6	1,027,900	10.3	1,728,300	10.1	1,398,600	12.3
4.	Furnaces and auxiliaries	3,284,400	39.2	3,637,700	29.2	3,917,200	25.6	2,453,700	24.5	6,546,600	38.4	4,298,500	37.7
5.	Charging and pouring	918,700	11.0	1,256,400	10.1	2,845,800	18.6	1,378,100	13.8	2,790,500	16.4	1,290,500	11.3
6.	Scrap and material handling	177,900	2.1	165,100	1.3	115,600	0.8	370,000	3.7	104,900	0.6	260,700	2.3
7.	Mould preparation and stripping	544,700	6.5	544,700	4.4	544,700	3.6	544,700	5.4	544,700	3.2	544,700	4.8
8.	Electric substation	284,800	3.4	325,700	2.6	196,500	1.3	341,400	3.4	233,500	1.4	335,200	2.9
9.	Miscellaneous	407,900	4.9	1,372,700	11.0	1,297,400	8.5	639,900	6.4	963,300	5.7	523,600	4.6
10.	Calcining and slag plants	127,700	1.5	2,146,700	17.2	127,700	0.8	127,700	1.3	127,700	0.7	127,700	1.1
11.	Total estimated cost	8,369,300	100.0	12,465,800	100.0	15,290,600	100.0	10,002,700	100.0	17,046,900	100.0	11,408,200	100.0
12.	Cost per ton of ingot capacity	16.74		24.93		30.58		20.01		34.09		22.82	

Table 5

ESTIMATED PLANT CAPITAL INVESTMENT COSTS REQUIRED TO PRODUCE 1,000,000 TONS OF INGOTS PER YEAR BY DIFFERENT PROCESSES

Item	*Description*	*Case 1* *Acid Bessemer process with auxiliary electric furnaces*		*Case 2* *Basic Bessemer process with auxiliary electric furnaces*		*Case 3* *Standard stationary basic open hearth process*		*Case 4* *Basic electric furnace process*		*Case 5* *Acid Bessemer-basic open hearth-tilting furnace-duplex process*		*Case 6* *Acid Bessemer-basic electric furnace-duplex process*	
		$	%	$	%	$	%	$	%	$	%	$	%
1.	Buildings			2,767,500	14.0	7,480,200	26.7	4,624,600	26.2	6,484,300	20.8	3,731,800	20.2
2.	Building services			516,600	2.6	1,624,400	5.8	794,200	4.5	1,427,300	4.6	732,400	4.0
3.	Cranes			1,331,600	6.7	2,492,700	8.9	1,841,400	10.4	2,985,400	9.6	2,004,200	10.8
4.	Furnaces and auxiliaries			6,403,200	32.3	7,748,400	27.7	4,814,900	27.3	11,390,000	36.5	7,101,300	38.4
5.	Charging and pouring			2,053,200	10.4	4,886,600	17.4	2,283,200	13.0	5,279,300	16.9	2,116,600	11.4
6.	Scrap and material handling	Omitted		282,600	1.4	158,800	0.6	563,300	3.2	158,800	0.5	433,100	2.4
7.	Mould preparation and stripping			986,200	5.0	986,200	3.5	986,200	5.6	986,200	3.2	986,200	5.3
8.	Electric substation			499,400	2.5	299,200	1.1	477,800	2.7	367,400	1.2	428,300	2.3
9.	Miscellaneous			1,946,300	9.8	2,084,700	7.4	981,400	5.6	1,821,000	5.9	707,700	3.8
10.	Calcining and slag plants			3,025,000	15.3	258,500	0.9	258,500	1.5	258,500	0.8	258,500	1.4
11.	Total estimated cost			19,811,600	100.0	28,019,700	100.0	17,625,500	100.0	31,158,200	100.0	18,500,100	100.0
12.	Cost per ton of ingot capacity			19.81		28.02		17.63		31.16		18.50	

Additional items are shown for the cost of mould preparation and stripping facilities, electric substation, calcining and slag plants and "miscellaneous". The cost of calcining and slag plant facilities are of importance only for the basic Bessemer process. Materials included in sub-items are listed in Table 6.

The total plant cost figures for all plants are summarized in Table 7. The last table (Table 8) shows the daily amount of molten pig iron or "hot metal" which each plant would require to produce its annual ingot tonnage.

Table 8 warrants some discussion, as it brings out a factor in the over-all cost of an integrated steelworks that is frequently overlooked, and which is not reflected in our estimates, i.e., the effect of the steel-making process employed on the amount of blast furnace and coke oven capacity that must be provided for a given ingot output.

Table 8 indicates that the duplex processes, due to the inherently higher metallic losses associated with the double handling of molten metal which they require, consume more pig iron for a given ingot production than any other process. The acid and basic Bessemer processes come next, with the basic open hearth and basic electric furnace processes showing the lowest hot metal requirement. Of the last two, the electric furnace requires the least. This is due to the

Table 6

Case 1—Acid Bessemer plants		*Case 2—Basic Bessemer plants*		*Case 3—Basic open hearth plants*	
Item	*Description*	*Item*	*Description*	*Item*	*Description*
1.	Buildings Mixer, converter and pouring bays, bottom and blower houses, mould preparation, scrap yard.	1.	Buildings Mixer, converter and pouring bays, bottom and blower houses, mould preparation, scrap yard.	1.	Buildings Charging and pouring bays, stock yard and mould preparation.
2.	Building services Sewer, water, gas and air-lines; lighting and power lines; railroad trackage; communications, etc.	2.	Building services Sewer, water, gas, and air-lines; lighting and power lines; railroad trackage; communications, etc.	2.	Building services Sewer, water, gas, oil, and air-lines; lighting and power lines; railroad trackage; communications, etc.
3.	Cranes Mixer, charging, steel pouring, pit service, scrap yard, mould yard, bottom house, etc.	3.	Cranes Mixer, charging, steel pouring, pit service, scrap yard, mould yard, bottom house, etc.	3.	Cranes Charging, steel pouring, pit service, stock yard, mould yard.
4.	Furnaces and auxiliaries Hot metal mixer, converters with blowers, electric scrap melting furnaces, bottom jack car, refractory pan mills, rammers, drying ovens, instrumentation, etc.	4.	Furnaces and auxiliaries Hot metal mixer, converters with blowers, electric scrap melting furnaces, bottom jack car, refractory pan mills, rammers, drying ovens, instrumentation, etc.	4.	Furnaces and auxiliaries Stationary open hearth furnaces with waste heat boilers, draft fans, chimneys, refractory mills, instrumentation, etc.
5.	Charging and pouring equipment Hot metal transfer ladles and car, slag pots and pans, steel ladles and relining pit, ingot moulds and cars, scrap charging buckets, etc.	5.	Charging and pouring equipment Hot metal transfer ladles and car, slag pots and pans, steel ladles and relining pit, ingot moulds and cars, scrap charging buckets, etc.	5.	Charging and pouring equipment Hot metal mixer type and transfer ladles, slag pots, steel ladles and relining pit, ingot moulds and cars, charging machines, boxes and buggies, material pans, etc.
6.	Scrap and material handling equipment Magnets, scales, transfer cars, dolomite machine, material bins, pans and buckets, lift trucks, oxy-acetylene equipment, etc.	6.	Scrap and material handling equipment Magnets, scales, car-pullers, transfer cars, dolomite machine, material bins, pans and buckets, lift trucks, oxy-acetylene equipment, etc.	6.	Scrap and material handling equipment Magnet, scales, dolomite machine, material bins and pans, lift trucks, oxy-acetylene equipment, etc.
7.	Mould preparation and stripping Stripper building and utilities, cranes, mould coating system, platforms, etc.	7.	Mould preparation and stripping Stripper building and utilities, cranes, mould coating system, platforms, etc.	7.	Mould preparation and stripping Stripper building and utilities, cranes, mould coating system, platforms, etc.
8.	Electric substation Transformers, high and low voltage switchgear, rectifiers, and control house.	8.	Electric substation Transformers, high and low voltage switchgear, rectifiers, control house, etc.	8.	Electric substation Transformers, high and low voltage switchgear, rectifiers and control house.
9.	Miscellaneous Office and sanitary buildings, switching locomotives, and rolling stock, trucks, air compressor, etc.	9.	Miscellaneous Office and sanitary buildings, switching locomotives, and rolling stock, oxygen plant, trucks, air compressor, etc.	9.	Miscellaneous Office and sanitary buildings, highline, switching locomotives and rolling stock, fuel oil facilities, oxygen plant, trucks, air compressor, etc.
10.	Calcining and slag plants Slag dump and slag handling equipment, including crawler crane, oxy-acetylene lances, drop ball, magnets, etc.	10.	Calcining and slag plants Slag dump and slag handling equipment, rotary lime-dolomite kiln with auxiliaries; slag crushing, grinding, bagging and storage facilities, etc.; all buildings.	10.	Calcining and slag plants Slag dump and slag handling equipment, including crawler crane, drop ball, magnets, oxy-acetylene lances, etc.

Table 6—continued

Case 4—Basic electric furnace plants		*Case 5—Acid Bessemer-basic open hearth-duplex plants*		*Case 6—Acid Bessemer-basic electric furnace-duplex plants*	
Item	*Description*	*Item*	*Description*	*Item*	*Description*
1.	Buildings Charging and pouring bays, scrap yard and mould preparation.	1.	Buildings Mixer, converter, pouring and charging bays, bottom and blower houses, stock yard and mould preparation.	1.	Buildings Mixer, converter, charging and pouring bays, bottom and blower houses, scrap yard, and mould preparation.
2.	Building services Sewer, water, gas, oil and air-lines; lighting and power lines; railroad trackage; communications, etc.	2.	Building services Sewer, water, gas, oil and air-lines; lighting and power lines; railroad trackage; communications, etc.	2.	Building services Sewer, water, gas and air-lines; lighting and power lines; railroad trackage; communications, etc.
3.	Cranes Charging, steel pouring, pit service, scrap yard, mould yard.	3.	Cranes Mixer, charging, steel pouring, pit service, scrap yard, mould yard, bottom house, etc.	3.	Cranes Mixer, charging, steel pouring, pit service, scrap yard, mould yard, bottom house, etc.
4.	Furnaces and auxiliaries Electric furnaces, transformers, regulators and controls, instruments, refractories, etc.	4.	Furnaces and auxiliaries Hot metal mixer, converters with blowers, bottom jack car, rammers, drying ovens, tilting open hearth furnaces with waste heat boilers, draft fans, chimneys, refractory mills, instrumentation, etc.	4.	Furnaces and auxiliaries Hot metal mixer, converter with blowers, bottom jack car, rammers, drying ovens; electric furnaces, transformers, regulators and controls; instruments, refractories, etc.
5.	Charging and pouring equipment Hot metal mixer type and transfer ladles, slag pots, steel ladles and relining pit, ingot moulds and cars, charging machines, boxes, buggies, buckets, material pans, etc.	5.	Charging and pouring equipment Hot and blown metal transfer ladles and cars, slag pots and pans, steel ladles, and relining pit, ingot mould and cars, charging machine, boxes and buggies, material pans, etc.	5.	Charging and pouring equipment Hot metal transfer ladles and cars, slag pots and pans, steel ladles and relining pit, ingot moulds and cars, charging machine, boxes, buggies and buckets; material pans, etc.
6.	Scrap and material handling equipment Magnets, scales, transfer cars, dolomite machine, material bins, pans and buckets, lift trucks, oxy-acetylene equipment, etc.	6.	Scrap and material handling equipment Magnets, scales, dolomite machine, material bins and pans, lift trucks, oxy-acetylene equipment, etc.	6.	Scrap and material handling equipment Magnet scales, transfer cars, dolomite machine, material bins, pans and buckets; lift trucks, oxy-acetylene equipment, etc.
7.	Mould preparation and stripping Stripper building and utilities, cranes, mould coating system, platforms, etc.	7.	Mould preparation and stripping Stripper building and utilities, cranes, mould coating system, platforms, etc.	7.	Mould preparation and stripping Stripper building and utilities, cranes, mould coating system, platforms, etc.
8.	Electric substation Transformers, high and low voltage switchgear, rectifiers and control house.	8.	Electric substation Transformers, high and low voltage switchgear, rectifiers, control house, etc.	8.	Electric substation Transformers, high and low voltage switchgear, rectifiers, control house, etc.
9.	Miscellaneous Office and sanitary buildings, switching locomotives and rolling stock, oxygen plant, trucks, air compressor, etc.	9.	Miscellaneous Office and sanitary buildings, highline, switching locomotives and rolling stock, fuel oil facilities, oxygen plant, trucks, air compressor, etc.	9.	Miscellaneous Office and sanitary buildings, switching locomotives and rolling stock, oxygen plant, trucks, air compressor, etc.
10.	Calcining and slag plants Slag dump and slag handling equipment, including crawler crane, oxy-acetylene lances, drop ball, magnets, etc.	10.	Calcining and slag plants Slag dump and slag handling equipment, including crawler crane, oxy-acetylene lances, drop ball, magnets, etc.	10.	Calcining and slag plants Slag dump and slag handling equipment, including crawler crane, oxy-acetylene lances, drop ball, magnets, etc.

fact that, in ordinary basic electric furnace practice, oxidation is supplied only by the iron oxide charged to the furnaces in the form of iron ore, while in the open hearth, part of the oxidation work is done by the flame. In the case of the acid and basic Bessemer, oxidation comes entirely from the Bessemer air blast.

If we assume that blast furnace and coke plant capacity would cost on the average in the neighbourhood of U.S.$75 per ton of annual pig iron output, it is obvious that the over-all cost of an integrated steelworks can be appreciably affected by the choice of steelmaking process, since more pig iron capacity must be provided in some cases than in others. This fact, however, is not taken into account in any way in our meltshop estimates, although the very considerable magnitude of the added investment cost involved can be approximated from the fact that a 100 ton difference in daily pig iron requirement is equivalent to nearly U.S.$3 million in terms of blast furnace and coke plant capacity.

DISCUSSION OF THE CAPITAL COST FIGURES

While the detailed explanation of the various cost differences shown by the estimates might be of interis much cleaner than the lower and very often the lower bench, at ordinary washing sizes, contains so small a proportion of metallurgical-grade coal that it is doubtful if any appreciable gain is made by routing this part of the coal through a metallurgical coal-washing plant. However, this characteristic is not uniform, and there are some instances where the beds are almost equal in quality. Furthermore, the simple idea of improving metallurgical-coal yield by handling the upper-bench coal only is complicated by the circumstance that the lower bench, though very difficult to clean, is generally a little lower in sulfur content than the upper bench; and this advantage, sulfurwise, is needed to keep the washed coal under the tolerance for sulfur content. Beyond these general observations, an intensive study of the characteristics of the bed with many local samples has indicated that the washing characteristics vary erratically from point to point, and therefore thorough blending is necessary, not only to prepare a plant-feed for effective operation of a washery but also to even out fluctuations in the inherent characteristics of the washed coal. Float-and-sink data of a number of these individual samples are presented in Table 8 to illustrate the erratic pattern of washing characteristics in this bed that are typical of all difficult coal reserves. These are important to show the essential role of the blending operation in preparing coking-coals and the need for wide knowledge of fluctuations in the characteristics of the coal in the initial design of a preparation plant. It is not enough to have an intensive study of a representative average sample mixture.

Table 8

FLOAT-AND-SINK DATA OF SOME BARRO BRANCO COAL SAMPLES

(Values in per cent on moisture-free basis)

	Sample 1			*Sample 2*			*Sample 3*			*Sample 4*		
Specific-gravity increment	*Weight*	*Ash*	*Sulfur*	*Weight*	*Ash*	*Sulfur*	*Weight*	*Ash*	*Sulfur*	*Weight*	*Ash*	*Sulfur*
Float – 1.35	2.4	7.5	1.64	7.9	1.6	1.11	8.3	4.6	0.93	0.8	9.5	2.36
1.35 to 1.45	26.6	17.3	2.29	13.3	8.8	1.69	23.3	16.62	1.09	15.6	18.8	3.02
1.45 to 1.50	18.7	21.1	3.96	8.2	16.4	1.82	10.0	23.2	1.01	11.4	22.6	4.11
1.50 to 1.55	7.3	22.7	6.15	8.9	21.2	1.85	6.8	27.8	1.10	16.0	25.6	4.43
1.55 to 1.60	5.7	24.9	9.09	7.3	26.7	1.94	3.9	33.0	1.37	4.8	29.5	4.51
1.60 to 1.70	4.9	27.8	11.48	7.8	32.8	2.14	4.2	37.4	1.53	15.6	33.6	4.64
1.70 to 1.80	3.7	32.4	16.15	4.4	40.8	2.60	3.0	47.0	1.40	8.1	36.0	5.68
Sink – 1.80	30.7	54.4	30.08	42.2	73.1	7.17	40.5	73.1	6.47	27.7	50.3	7.95

	Sample 5			*Sample 6*			*Sample 7*			*Sample 8*		
Specific-gravity increment	*Weight*	*Ash*	*Sulfur*	*Weigh*	*Ash*	*Sulfur*	*Weight*	*Ash*	*Sulfur*	*Weight*	*Ash*	*Sulfur*
Float – 1.35	13.7	4.9	1.79	2.6	7.0	1.52	6.8	5.1	1.57	22.6	6.9	0.92
1.35 to 1.45	18.5	14.5	2.07	28.4	15.5	1.22	15.4	15.3	1.62	34.4	11.1	1.38
1.45 to 1.50	12.3	21.0	2.18	15.2	19.7	1.25	8.6	22.5	1.65	4.8	22.3	3.42
1.50 to 1.55	12.2	25.5	2.27	11.3	28.8	1.46	6.3	26.6	1.65	4.2	25.9	4.89
1.55 to 1.60	9.7	30.3	4.85	10.0	34.2	1.32	4.8	30.4	2.17	1.6	30.6	3.86
1.60 to 1.70	9.0	34.0	4.24	13.1	40.3	1.89	6.6	36.2	2.59	2.3	31.8	7.60
1.70 to 1.80	3.6	38.5	7.85	5.0	49.4	2.64	5.4	45.8	3.63	1.4	41.7	8.15
Sink – 1.80	21.0	56.6	28.35	14.4	55.8	16.02	46.1	72.9	8.83	28.7	56.9	26.94

To explore the difficulties and possible solutions of the problem of recovering a metallurgical coal from the Barro Branco bed, Figures 9 to 13 have been introduced here to show the washing characteristics, by sizes, of a typical sample of a mixture of the upper and lower benches of this coal bed. These data have been used to make the following analysis of possible treatment methods.

It is obvious, on casual inspection of the data, that it would be impossible to obtain a practicable yield of metallurgical coal to meet the usual standards for coking practice in the United States or Europe. Therefore, it will be immediately necessary to set up some higher tolerance for ash content. Fortunately, the exceptionally pure iron ores of Brazil make this entirely feasible. However, even accepting an ash content of 18% and a sulfur tolerance of 1.60%, it is even then impossible to obtain an economical yield of coal of that grade. It is essential to make a three-product separation and to develop some economic use for the secondary coal. Therefore the various possibilities of combination treatment are being appraised to make a maximum recovery of metallurgical coal of 18% ash by a relatively low-gravity separation and recovering a secondary coal by a high-gravity separation to achieve the most complete recovery of usable combustible matter.

Preliminary study of the washability data indicate that, with the specifications suggested for metallurgical fuel, the raw material may be expected to yield around one third of metallurgical grade, with one third discarded as sterile refuse and around a third remaining as secondary fuel of high ash content.

Several plans may be followed in dividing the combustible material between the two grades, and the choice among these will depend upon the demands of the fuel-coal market as well as the physical conditions. Although the production of metallurgical coal is the primary objective, the largest volume of secondary fuel coal resulting unavoidably from the operation makes disposal of this product a prime economic factor. Several possibilities are:

1. Blending the entire production to feed to a central washery to make a two-gravity separation.

2. Separate treatment of the Forro and Banco coals with the Forro handled in a central washery for producing metallurgical coal and the Banco handled locally in small preparation plants for fuel coal.

3. Selection of most suitable raw coal sources for metallurgical coal and for fuel coal by mines instead of by beds.

The first plan has the very great advantage of simplicity in organization of the total job and standardi-

Table 7

ESTIMATED PLANT CAPITAL INVESTMENT COSTS IN TOTAL DOLLARS AND DOLLARS PER TON OF ANNUAL INGOT CAPACITY

Case No.	*Steelmaking process*	*Assumed total annual plant capacities (metric tons ingots per year)*							
		100,000 tons		*250,000 tons*		*500,000 tons*		*1,000,000 tons*	
		$	*$/Ton*	*$*	*$/Ton*	*$*	*$/Ton*	*$*	*$/Ton*
1.	Acid Bessemer (with auxiliary electric scrap melting furnaces) . .	3,963,500	39.64	5,943,800	23.78	8,369,300	16.74	Omitted	
2.	Basic Bessemer (also with auxiliary electric furnaces)	5,917,700	59.18	9,105,900	36.42	12,465,800	24.93	19,811,600	19.81
3.	Basic open hearth (using stationary furnaces)	4,675,500	46.76	9,428,600	37.71	15,290,600	30.58	28,019,700	28.02
4.	Basic electric furnace (using top charge furnaces)	3,302,500	33.03	4,981,600	19.93	10,002,700	20.01	17,625,500	17.63
5.	Duplex-acid Bessemer-basic open hearth (using tilting open hearths)	Omitted		11,562,800	46.25	17,046,900	34.09	31,158,200	31.16
6.	Duplex-acid Bessemer-basic electric furnaces (using top charge electric furnaces)	4,673,300	46.73	6,840,900	27.36	11,408,200	22.82	18,500,100	18.50

Table 8

DAILY PIG IRON REQUIREMENTS FOR THE PRODUCTION OF STEEL BY DIFFERENT PROCESSES

Case No.	*Process*	*100,000-ton plant*	*250,000-ton plant*	*500,000-ton plant*	*1,000,000-ton plant*
1.	Acid Bessemer (with auxiliary electric scrap melting furnaces) . .	221	545	1,070	Omitted
2.	Basic Bessemer (also with auxiliary electric furnaces)	223	550	1,080	2,150
3.	Basic open hearth (using stationary furnaces)	200	500	1,000	2,000
4.	Basic electric furnace (top charge)	189	473	945	1,890
5.	Duplex-acid Bessemer-basic open hearth (using tilting open hearths)	Omitted	580	1,160	2,320
6.	Duplex-acid Bessemer-basic electric furnaces (top charge)	227	568	1,135	2,270

est, it is doubtful whether the accuracy of the figures warrants such treatment. Too much significance should not be attributed to differences in the figures one way or another of a few per cent or a few dollars per ton of ingot capacity. This is because of the many assumptions and approximations that had to be made in setting up the basic data used to develop the figures. Comments on the estimates will therefore be limited largely to a short discussion of the figures as summarized in Table 7, which gives the total capital investment cost in each case as well as the cost per ton of ingot capacity.

In the case of the two Bessemer processes, the primary fact that stands out from the figures is that, in all three cases for which a comparison can be made, the basic Bessemer plants cost approximately 50% more than acid Bessemer shops of the same capacity. This is readily explained from the greater number of units resulting from the longer blowing time and shorter lining life of basic as compared to acid converters, plus the extra cost of the auxiliary oxygen calcining and slag plants included in the basic Bessemer figures. Even in the case of the basic Bessemer, however, the cost per unit of capacity drops off sharply as plant size and output increase. In the smallest group, the basic Bessemer shop is the most expensive of all, but in the case of the larger shops, with their bigger units and proportionately lower auxiliary equipment costs, a smaller investment cost results than for either the straight or duplex basic open hearth processes (Cases 3 and 5). The estimates show that the cost per ton of capacity decreases almost equally sharply in both Bessemer cases (1 and 2), the unit cost per ton of annual capacity of the million ton basic Bessemer plant for instance being only about one third that of the 100,000-ton one.

Although both the basic open hearth and basic electric furnace plants (Cases 3 and 4) show a similar definite downward trend in unit cost with increase in plant size, it is much less striking than for the two Bessemer cases. The unit investment cost of the 1,000,000-ton basic open hearth plant drops less than 50% from the figure for the smallest size, and the same is true, percentagewise, in the case of the basic electric furnace figures. As might be expected, the drop in unit cost in the case of the two duplex plants (Cases 5 and 6) lies between the Bessemer figures and those for the open hearth and electric furnace.

In the three plant size categories for which figures are shown, the acid Bessemer-basic open hearth duplex process shows the highest cost per unit of capacity, although in the case of the two larger size plants, the differential between it and the straight basic open hearth narrows considerably, differing only by about U.S.$3.50 per annual ton. This means that for large plants the two processes can be taken as more or less identical in investment cost. In other words, when it comes to a choice between the straight vs. the duplex open hearth process for a 1,000,000-ton plant, the question of relative capital cost does not enter into the picture. The composition of the iron ore and the pig iron made from it, plus the steel products to be rolled, will decide which one of these two processes should be used.

In recent years it has frequently been said (we have said so ourselves) that a basic electric furnace plant costs only about half as much as a basic open hearth plant of the same capacity. The figures presented in this paper bear this out only in the case of the 250,000-ton per year plants. For the other plant sizes the ratios are much lower—only 1.42, 1.53, and 1.59. Just why this should be so is not apparent from our detailed figures but it is probably due, in part at least, to the high hot metal charge practice we assumed under Latin-American conditions. To be on the safe side, the number of electric furnace units required, as already brought out in the metallurgical discussion, was based on an output of only 4½ heats per furnace per day with 60% hot metal in the total charge. In actual fact, however, American electric furnace plants are currently tapping 40 to 50-ton heats of plain carbon steel at an average rate of six heats per day from all cold scrap charges. It is therefore felt that possibly one reason for the relatively lower ratio between the capital cost of the basic open hearth and basic electric furnace plants shown by the present estimates is this assumption that each electric furnace would be able to produce only 4½ heats per day from a 60% hot metal charge.

The 1,000,000-ton per year basic open hearth estimate of over U.S.$28,000,000 is appreciably lower than the cost of some of the more recent large new open hearth shops built in the United States. This is perhaps mainly due to the fact that the newer North American open hearth furnaces are designed to tap somewhat larger heats than the 215-ton units we have chosen, and consequently would require heavier buildings and cranes than we have felt would be suitable for Latin-American conditions. In addition, as stated earlier, our estimating figures were based on the existence of more normal general business conditions, particularly as regards equipment delivery and prices, as well as the availability of construction labour, than has been the case recently in the United States.

Among the smaller size plants, the basic electric furnace shops show the lowest cost per ton of capacity, but when the output gets up to half a million tons, the acid Bessemer figures are even lower, and had a figure been developed for a 1,000,000-ton acid Bessemer plant, this would still undoubtedly have held true. This result of course is mainly due to the combined effect of larger converters and increased Bessemer plant operating time. The same downward trend in cost per unit of capacity is shown, as mentioned before, by the basic Bessemer plants; in the 1,000,000-ton group the basic Bessemer cost is down to about the same order of magnitude as the electric furnace and the acid Bessemer—basic electric duplex plants; all three are in a narrow range of from U.S.$17 to U.S.$20 per annual ton.

Also to be noted is the importance, both relatively as well as in absolute figures, of the cost of buildings and building services. Percentagewise, this item of cost varies from a low of 14% in the case of the smallest basic Bessemer plant, to a high of 33.2% in the case of the 500,000-ton open hearth shop, and 32.5% in the case of the 1,000,000-ton open hearth shop. In absolute values, the variation is even greater, ranging from a two-fold difference, from minimum to maximum, for the 100,000-ton plants to nearly three-fold in the case of the 1,000,000-ton meltshops.

In studying these building figures it should be remembered that they are for the main steel plant buildings and building services only; the office and stripper buildings are separate, as are also all buildings connected with the calcining kilns and slag plant in the case of the basic Bessemer.

Details of a Steel Plant for a non-Steel-producing Country

ANDRÉ MERCIER

Wherever foundry coke—either imported or locally produced—and iron ore with approximately 40% of iron content are available, it is possible to build up a complete steel industry involving reasonable investments and resulting in manufacturing costs which at normal amortization rates will permit the local market to be supplied with rolled products at a cost price substantially lower than that of imported products.

For the proposition to be attractive it is essential both that the yearly tonnage should not be too low and that manufacturing should be restricted to a small number of shapes without aiming at producing all such articles as are imported in the country at the time the mill construction is decided upon.

This paper describes briefly a two blast-furnace steel plant with a production capacity of 100,000 to 150,000 tons of rolled products per annum, and with provisions for expansions up to 200,000 tons.

The approximate construction cost and the basic cost price of the finished products are also given.

It has been assumed that phosphorous ore suitable for the basic Bessemer process is available.

The plant would consist of:

One river port with unloading wharves and facilities for storing the raw materials.

One junction-line connected to the main railway system.

One road suitable for lorry traffic.

Two blast furnaces producing 200 tons iron per day, including sloping track bucket charger, together with their auxiliaries.

One steam generating plant including steam boilers fired with blast furnace gas.

Two turbo-alternators, turbo-blowers for the blast furnaces and steel works.

The generated power can be linked to the main supply line, if one is available, thus making it possible to obtain power from the outside when desirable and to supply any excess available to the grid.

One basic Bessemer steel plant with a 500-ton mixer and three converters of 15 tons for turning the basic pig iron into liquid steel.

Provision has also been made for an open hearth furnace for production of steel, using either scrap, if any is available from the local market, or the crop ends from rolling mills, which is more attractive than sending them over to the blast furnace and converter.

Rolling mills for blooms, billets, bars, merchant products, sheets and medium plates.

One basic slag plant for fertilizer production.

One blast furnace slag grinding and screening plant for ballast or other requirements; this plant can of course be omitted in areas where cheaper quarried stone products are available.

The auxiliary facilities, including an iron foundry, one steel foundry, one forge, one machine shop for maintenance requirements, one laboratory, together with various buildings for offices, first-aid station dressing-room, canteen, etc.

Should the steel plant be located in an area where other machine shops are already available, the size of the machine shop can be reduced.

A subsequent addition to the plant may consist of facilities for sintering blast furnace dust and ore fines.

A coke oven may also be installed, providing coking coal is locally available.

Rolling mill facilities should include:

One reversing two-high 750 mm. blooming mill fed with 2,000/2,500 kg. ingots for the production of blooms and slabs.

One three-high 600 mm. mill for the production of billets, sheet-bars and heavy sections.

One strip and rod rolling mill.

One medium rolling mill for straight sections.

One sheet rolling mill.

One medium plate rolling mill.

The range of the finished products will include the production of:

(1) 6 to 120 mm. diam. rounds;
(2) Squares from 6 to 120 mm.;
(3) Flats 20 x 5 to 100 x 25 mm.;
(4) Angles from 25/25 to 130/130 mm.;
(5) Standard joists from 30 to 200 mm.;
(6) Standard channels from 30 to 180 mm.;
(7) Strip from 20 x 1 mm. to 75 x 2.5 mm.;
(8) Sheets from 0.25 to 2 mm. thick: max. length 3 m., max. width 1.25 m.;
(9) Galvanized and corrugated sheets;
(10) Medium sheets 2-8 mm. thick: max. length 8 m., max. width 1.4 m.

The production of semi-finished products as required by the manufacturing programme includes:

(*a*) Billets from 50 x 50 mm. to 130 x 130 mm.;
(*b*) Sheet-bars from 10 to 25 x 250 mm.;
(*c*) Slabs from 230 x 60 to 500 x 80 mm.;
(*d*) Blooms from 100 x 100 to 250 x 250 mm.

Such a plant will produce from 100,000 to 150,000 tons of finished products per annum.

It can be indicated that on a 130,000 ton basis, the production figures would be as follows: 60,000 tons merchant bars, 30,000 tons sheets, 10,000 tons medium plates, 10,000 tons strip, and 20,000 tons billets.

COST PRICES OF PLANT

The cost price including the supply of equipment, transport, civil engineering work, erection on site and designing and general contracting expenses under the economic conditions prevailing in September 1952, is about 12,000 million francs, viz. $34,200,000 approximately, at the rate of 350 fr. per one dollar.

No customs duties are included.

Costs involved in purchasing the necessary site, the preparation of same for building, construction, and the excavation work—all of which are closely related to local conditions—are not included in the above figures.

Here is an approximate breakdown per item involved:

	$
Blast furnaces	6,150,000
Basic Bessemer steel plant and lime kilns	3,700,000
Open hearth steel plant (one furnace)	1,700,000
Rolling mills	13,150,000
General service installations: Steam generating plant, shops, laboratories, tracks and rolling stock, water, gas, electricity and air supply systems, buildings, sewers, unloading wharf, lighting, etc.	9,500,000
TOTAL	34,200,000

Capital required for construction, and putting into operation

Cost of plant	34,200,000
Working capital, 20% of the yearly turnover	2,300,000
Raw material stock and spare parts	3,500,000
TOTAL	40,000,000

PRICES OF THE MANUFACTURED PRODUCTS

A. Basic cost price without amortization and interest on capital:

Assuming the cost of raw materials to be as follows:

$4.50 per ton of iron ore at charging stage;
$17 per ton of Mn ore at charging stage;
$0.90 per ton of limestone at charging stage;
$21.5 per ton of coke at charging stage.

These figures would lead to the cost prices shown below under Column A of Table 1.

B. Basic cost price with amortization, without interest on capital:

Assuming that the installation cost of the plant must be written off in 20 years, the amount to be paid off every year is $1,710,000, namely 23% approximately of the "A" cost price of the products sold.

Consequently the prices arrived at, including amortization, are those shown under column B in Table 1.

No consideration has been given to financial expenditure involved in the payment of interests on invested capital and taxes, as such items vary considerably from one country to another.

It will be easy, when such data and the possible selling price of the manufactured products are known, to predict the paying capacity of the plant to be installed.

Table 1

COSTS PER TON IN A 100,000–150,000 TON STEEL PLANT

Products	*(A)* *Without amortization*	*(B)* *With amortization over 20 years*
	$	$
Basic pig iron	30	37
Basic steel ingot	34.2	42
Bloom	37.1	46
Billets	43	53
Small sized merchant sections	53	65
Medium sized merchant sections	49	61
Strips	54	67
Medium plates	64	79
Sheets	78.5	94

Summary of Discussion

Steel Production in Latin America, *presented by the author*

Mr. BULLE, in introducing his paper, pointed out that Latin America was interested in manufacturing steel from pig iron, not from scrap, and he had therefore dealt with the various processes used for the manufacture of steel. He quoted the respective installation costs as well as the estimated production figures contained in the paper contributed by Mr. Krebs. He then referred to the steel-making prospects of certain Latin-American countries, the economic structure of which he had studied. Those countries were: Argentina, Brazil, Chile, Colombia, Mexico, Peru and Venezuela.

The CHAIRMAN postponed discussion of the paper until after the presentation of Mr. Krebs' paper, entitled:

The Balance of Materials and the Economic Comparison of the Different Steelmaking Processes *presented by the author*

Mr. WARING opened the general discussion of the paper by thanking Mr. Krebs for the very able presentation thereof. He was of the opinion, however, that hypotheses usually carried with them a mass of imponderables, and he was sure that Mr. Krebs himself would agree that the conclusions at the end of the paper concerning production costs should be treated with some reserve. He recalled the case of a team of British experts in the United States which had attempted to establish comparative costs. The recording methods adopted by the British were so different from those practised in the United States that although the basic figures were the same, it was practically impossible to determine exactly how the final cost figures were arrived at. Bearing that incident in mind, he was inclined to believe that it was very difficult to establish comparative costs. There were so many factors that might influence the figures one way or the other.

He hoped that his intervention would not be misconstrued, since he had not intended any unfavourable criticism of Mr. Krebs' paper, but merely to point out that hypothetical figures should be regarded with caution. He was speaking, of course, from the point of view of an economist. He thought, however, that the methods of assessment employed by Mr. Krebs would be of great value in appraising individual cases.

Mr. KREBS agreed entirely with Mr. Waring's comment. He had, however, attempted to present only a rough comparison based on data he had received.

Mr. SADA thought that the two papers under review contained very valuable information indeed. He then proceeded to discuss the merits and demerits of each of the processes described as applied to Latin America. The electric furnace would be entirely out of the question in those countries with few hydroelectric power resources, as well as for those countries where the cost of such power was prohibitive. As to the converter and open hearth furnaces, he agreed that the initial costs of the open hearth type were quite high. It should, however, be borne in mind that the converter process required the support of a blast furnace.

Furthermore the cost of refractories was very high.

He did not think that processes involving small tonnages could be used as a basis for the development of the steel industry before they had been thoroughly tested both technically and commercially, as most of the Latin-American countries were economically unable to experiment with untried methods. Each country had to adopt those processes that were most suited to prevailing conditions.

He referred to a statement in Mr. Bulle's paper to the effect that Mexican coke had an ash content of 19%, pointing out that cokes of 85% fixed carbon had been produced recently, with an ash content of 14%.

Mr. LEUSCHNER observed that Mr. Krebs had been requested to calculate the comparative cost data appearing in his paper on a hypothetical basis because it was the only basis that could be used for many Latin-American countries. He added that the virtue of the cost analyses given lay, not in their infallibility, but in their value as a guide for the calculation of actual costs. The analyses were to be regarded, therefore, merely as specimens.

Mr. GONZALEZ VARGAS remarked that the comparison of processes on the basis of cost data was very difficult to establish. He did not entirely agree with Mr. Sada's point to the effect that Latin America should adhere to traditional processes owing to the difficulties attendant upon experimenting with new processes; that attitude might not be economically sound, because the plants currently being constructed according to seemingly safe traditional methods might in the course of ten years be found to be obsolete. He did not therefore hold the view that Latin America should cling so desperately to the old processes.

Mr. VUCETICH expressed his particular interest in the papers under review, inasmuch as they dealt with one of Chile's main problems—that of the use of scrap.

Except for a few minor exceptions, he found that the costs shown in Mr. Krebs' paper compared with figures developed in Chile; the Huachipato plant operated two open hearth furnaces and one basic Bessemer converter, with a yield of 250,000 tons of ingots a year. Production had been increasing steadily and that upward trend was expected to continue. A second basic Bessemer converter was to be installed in the near future.

Mr. KREBS observed that the cost of hydroelectric power used for the calculations appearing in his paper was at the rate of DM60 per 1,000 kWh. That was a very high figure.

The CHAIRMAN announced that as the next two papers also dealt with costs, further discussion of the papers presented by Messrs. Bulle and Krebs would be adjourned until the afternoon meeting.

Comparative Investment Costs for Different Steelmaking Processes, *presented by the author*

Mr. PRIETO opened the discussion of the paper by asking Mr. Ramseyer whether he considered the operation of two 10-ton converters in the shop described in Table 1 as feasible.

Mr. RAMSEYER replied that, on the assumption of a 30% scrap availability, he was of the opinion that two converters would be adequate.

Mr. PRIETO said that he was interested not only in the capacity of the converters but also in the time schedule. He wondered whether Mr. Decherf would care to comment on that question.

Mr. DECHERF stated that the first consideration in conducting tests was the capacity of the converter. In his opinion, the lowest cost of Thomas steel could only be achieved if regular production were maintained. He therefore thought that three converters would be necessary for a production of 125,000 tons of ingots a year. Two converters seemed to him to be totally inadequate.

Mr. RAMSEYER thought that Mr. Decherf's reasoning was quite sound, under prevailing conditions at Paz de Río.

Mr. SCHERESCHEWSKY commented that Messrs. Krebs and Ramseyer had both emphasized that the figures given in their papers were of relative value. Under those circumstances, he thought it unfair to criticize those figures. Mr. Ramseyer had described 22 different ways of making steel. A possible twenty-third way should be of great interest under conditions prevailing in several Latin-American countries and merited being added to Mr. Ramseyer's list. He referred to the method described by Mr. Allard at a previous meeting, on the refining of steel, better known as the Perrin processes. That process had various advantages which were of value to Latin America: (1) it provided a high degree of refinement comparable to that of the best duplex operations, at a very small capital cost, reducing the phosphorus, oxygen and sulfur contents considerably in a very short time, by the use of two kinds of slag (iron-lime and alumina-lime); (2) it permitted the manufacture of only small tonnages of the desired high purity steel; (3) it was a proved industrial process being already used on a large scale in France and Belgium (several hundred thousand tons per year).

Mr. BULLE regretted the fact that he had not brought with him documents permitting a comparison between European and American investment figures.

Mr. RAMSEYER stated that, in the figures presented in his paper, the costs were increased by the addition of certain installations. Therefore, on the basis of small productions, using United States figures, the average resultant costs seemed to be very high. Large outputs would carry figures closer to Mr. Bulle's ratio.

Mr. COHEUR referred to the so-called Perrin process. Although the steel produced by that method was of high quality, it was killed steel. In Belgium more economical processes had been established, producing Thomas steel which was equal to Martin steel. The sulfur content was quite low and the product compared favourably with other types of steel.

Details of a Steel Plant for a non-Steel-producing Country, *presented by the author*

Mr. BULLE opened the discussion of the paper by remarking on the similarity between French and German cost figures presented at the meeting.

Mr. LANARI asked Mr. Mercier whether he thought it possible for a small plant to operate a quick-acting forging press to reduce ingots of 1,000 kg.

He also wished to know what type of sheet mills were provided for in the paper under review.

His third comment was directed to Mr. Leuschner. He wondered whether ECLA could undertake the compilation of cost data on the various processes. Such a study would be extremely helpful to the Latin-American countries interested in developing a steel industry.

Mr. MERCIER, replying to Mr. Lanari's first question on the possibilities of replacing the blooming mill with a quick-acting forging press, regretted not having in his possession a document dealing with that question. He would, however, send the pertinent papers to Mr. Lanari on his return to France. Where relatively small annual tonnages were required, the economics of operating a quick-acting forging press equipped with manipulators compared favourably with the blooming mill.

With regard to the second question, he said that the mills referred to were highly mechanized and suitable for rolling sheets and plates of 3.8 mm.

Mr. LEUSCHNER, replying to Mr. Lanari's suggestion, stated that ECLA had been conducting an iron and steel (cost analysis) programme for over a year. At a later meeting, ECLA would be presenting a paper on the subject.

Several other studies were also being undertaken and he requested the co-operation of those Latin-American participants present in order to bring the programme to fruition.

GENERAL SUBJECTS AND COMMEMORATION OF UNITED NATIONS DAY

24 October 1952—Afternoon

Chairman:

Roberto JARAMILLO FERRO, Manager, Empresa Siderúrgica Nacional de Paz de Río, S.A., Bogotá, Colombia

Later:

Juan URIBE HOLGUIN, Minister of Foreign Affairs, Colombia

Contributed Papers:

Approximate Comparative Production Costs and Investment Requirements of the Standard Blast Furnace, Electric Smelting Furnace, Sponge Iron Furnace and Tunnel Kiln Sponge Iron Furnace

P. E. CAVANAGH, Assistant Director, Department of Engineering and Metallurgy, Ontario Research Foundation, Toronto, Canada

Desulfurization of Pig Iron with Pulverized Lime in Rotary Kilns
Bo KALLING, Director of Research, Stora Kopparbergs, Bergslags Aktiebolag, Domnarfvet, Sweden

Some Notes on the Organization of Monclova Steel Works
Pablo SADA, Assistant General Manager, Altos Hornos de México, S.A., Monclova, Mexico

Commemoration of United Nations Day

Address[1] by Mr. Juan Uribe Holguin, Minister of Foreign Affairs, Colombia

Address[1] by Mr. Raymond Etchats, Permanent Representative of the Technical Assistance Board in Colombia

Brief Account of Iron and Steel Producing Processes Used in Brazil
Amaro LANARI, Escuela Politecnica, Universidad de São Paulo, Brazil

Summary of Discussion:

Participants: Messrs. CAVANAGH, NEWHALL, KALLING, LEUSCHNER, ALLARD, SANITER, SADA, BALLÓN, SHEROVER, Juan URIBE HOLGUIN, Raymond ETCHATS, LANARI, VARGAS MARTINEZ, VILLELA, VUCETICH, JARAMILLO

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[1] Included in the Summary of Discussion.

Approximate Comparative Production Costs and Investment Requirements of the Standard Blast Furnace, Electric Smelting Furnace, Sponge Iron Furnace[1] and Tunnel Kiln Sponge Iron Furnace*

P. E. CAVANAGH

Item	*Blast furnace*	*Electric smelting furnace*	*Sponge iron furnace*	*Tunnel kiln*	*Krupp-Renn*
Plant and furnace capacity					
Total yearly output (tons)	280,000	110,000	45,000	40,000	100,000
Number of furnaces	1	3	2	4	1
Daily output per furnace (tons)	800	100	64	30	300
Production costs per ton of iron (dollars) [2]					
Coke	9.35	5.12	3.60	4.50	5.00
Ore	14.50	14.50	20.75 [3]	20.00	14.50
Limestone	0.97	0.88	0.09	0.09	0.10
Electrodes	—	1.80	0.40	—	—
Power (0.3 cents per kWh.)	—	7.50	2.70	0.10	2.10
Gas for heating	—	—	—	0.75	—
Cooling water	0.42	0.33	0.06	—	0.05
Labour at $1.20 per hour	0.94	2.19	2.52	7.20	2.40
Repairs, maintenance, overhead, and miscellaneous (including casting)	4.15	4.75	4.90	7.00	4.50
Production cost	30.33	37.07	35.02	—	28.65
Gas credit at 25 cents per million B.T.U.	1.90	1.75	—	—	—
Saggers	—	—	—	5.00	—
Net production cost	28.43	35.32	35.02	45.00	28.65
Total investment (thousands of dollars) [4]					
Total	28,000	9,750	3,000	1,000	5,000
Investment per ton-year (dollars)					
Total	100	89	67	25	50
Write-off of investment at 15%	$15	$13.50	$10	$4	$ 7.50
	$43	$48.00	$45	$49	$36.15

[1] Tysland-Hole electric smelting furnace and Wiberg-Sodersfors sponge iron furnace.

[2] Production cost estimates based on typical operating conditions, reflecting 1948 cost of production factors in Canada. Production costs do not include profits or fixed charges on investment.

[3] Higher grade ore required than in case of other processes.

[4] Investment costs are not intended to be exact nor to apply to any particular location. They are representative costs in Canada in 1948 and are intended only for purposes of comparison.

Source: World Iron Ore Resources and Their Utilization, United Nations, 1950.

* This paper was prepared during the course of the meeting, for purposes of discussion.

Desulfurization of Pig Iron with Pulverized Lime in Rotary Kilns

BO KALLING

In the production of pig iron the sulfur content is dependent on the amount of sulfur present in the charge and above all in the coke used. It is usually possible to obtain the desired low sulfur content in the pig iron by adjusting the composition of the charge in such a way that a basic desulfurizing slag is formed. A high content of manganese in the charge is also favourable. In some cases, especially if the ore is poor and silicious, the amount of lime needed for attaining sufficient basicity of the slag will be so large that it is more economical to produce a pig iron high in sulfur and desulfurize it in a special operation after it leaves the blast furnace. If this treatment can be done cheaply and efficiently enough, it may prove economical to introduce such a process more generally in the production of pig iron. The reduced consumption of lime and materials containing manganese and the resultant decrease in coke consumption and increase in production might lead to savings far exceeding the cost of desulfurization outside the blast furnace.

The materials hitherto commonly used for desulfurization of pig iron have been soda or a mixture of limestone and fluorspar, in the latter case sometimes in combination with soda. The addition is done when tapping the pig iron into the ladle in order to obtain an intimate stirring effect between the pig iron and the molten basic slag formed by the added materials. It is thus possible to decrease the sulfur content of the pig iron appreciably, but without special procedures it is difficult to remove the sulfur completely. One reason for the often unsatisfactory results may be insufficient stirring, another that the remaining sulfur content of the pig iron is too high even at equilibrium with the molten slag. Due to these facts the desulfurization often has to be repeated.

The desulfurization process which will be described here is different from other previous methods in that here the pulverized burnt lime which is used as the desulfurizing agent does not melt but remains solid during the whole treatment. The process, which had its first stages of development at the Domnarfvet Steelworks in Sweden (1, 2), consists in bringing the molten pig iron into intimate contact with pulverized lime in a rapidly rotating furnace under strongly reducing conditions. In this way a very rapid and effective desulfurization can be obtained with a low consumption of lime.

This method is in principle closely related to the method of desulfurizing sponge iron that was developed at the Avesta Steelworks, Sweden (5). The only main difference is that the iron is in the fluid instead of the solid state. It should be mentioned that a method in many respects similar to the Domnarfvet one was invented by Eulenstein and Krus in Germany (3), and tried at the end of the 1930's (4). Interesting investigations regarding the reactions in question have been carried out by Oelsen (6) and also recently by Giedroyr and Dancy (7), and Rosenqvist (8).

If the pig iron does not contain alloying elements other than carbon, the reaction can be expressed most simply by the well-known formula:

$$FeS + CaO + C = Fe + CaS + CO \quad (1)$$

This reaction can go almost to completion due to the fact that there is very little mutual solubility between CaS and CaO in the solid state.

In practice, conditions are somewhat complicated by the fact that the pig iron to be desulfurized is very seldom free of silicon. In a pig iron containing silicon above a certain level, about 0.05%, the carbon is replaced by silicon as a reducing agent, and the reaction formula may be written:

$$2\ FeS + 2\ CaO + FeSi = 3\ Fe + 2\ CaS + SiO_2 \quad (2)$$

The silica formed will, however, probably react with the lime present to form a silicate, presumably Ca_2SiO_4 in the main, when there is an excess of lime.

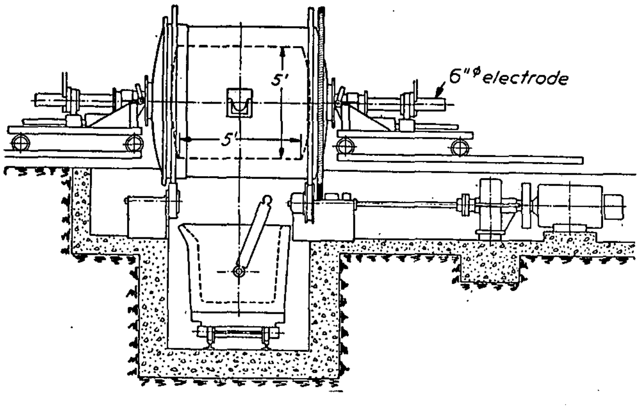


Figure 1
Design of rotating kiln used for the first large-scale tests at Domnarfvet.

The reaction equation will then be

$$2\ FeS + 4\ CaO + FeSi = 3\ Fe + 2\ CaS + Ca_2SiO_4 \quad (3)$$

The formation of silicate here implies that a large part of the lime is bound to the silica, resulting in a higher theoretical consumption of lime in this case than with silicon-free iron.

The first large-scale tests with this method were carried out at Domnarfvet (1) in a rotating kiln, the design of which is shown in Figure 1. After preheating of the furnace with electrodes these were drawn out and the kiln was charged with 2.5 tons of pig iron. The burnt lime was charged in a powder usually mixed with a certain quantity of coke breeze. The furnace was thoroughly sealed to prevent the entry of oxidizing gases and the rotation was started. The speed of rotation was normally 40 r.p.m. A lower speed of rotation gave a notably poorer result. The necessary time for treatment was normally about 10 minutes.

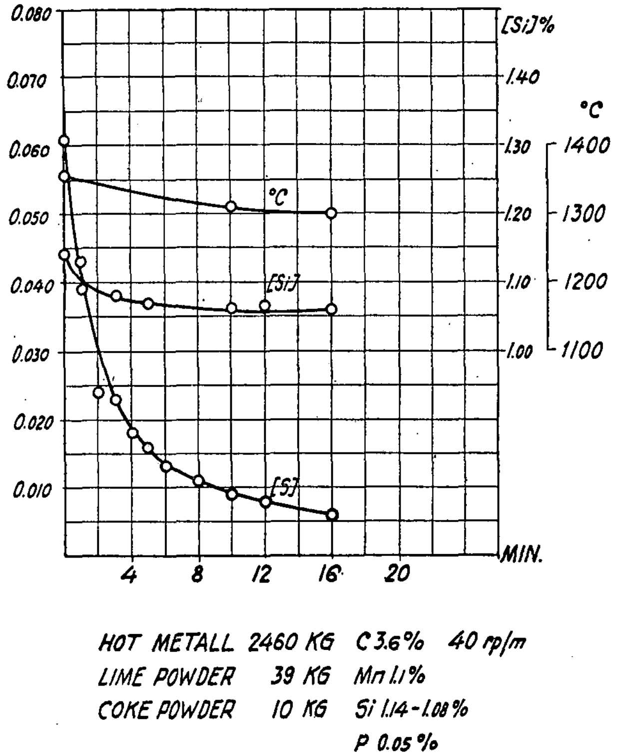
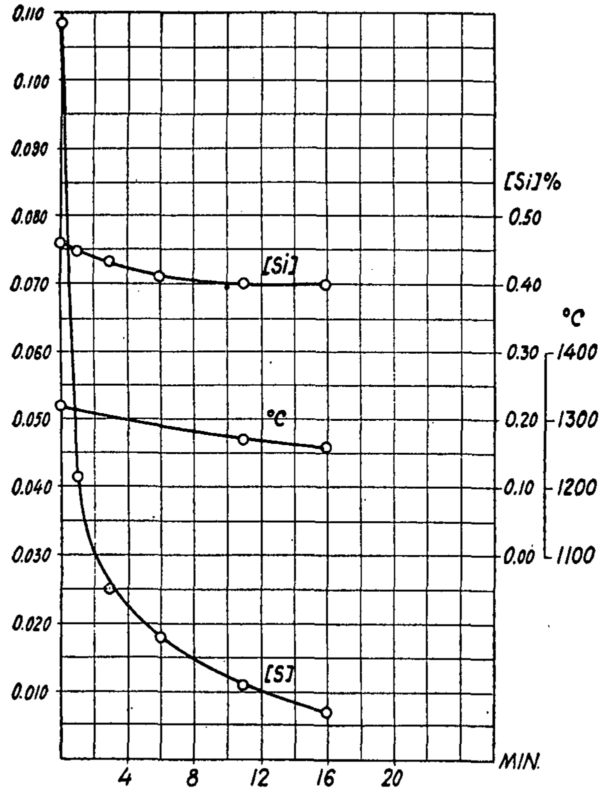


Figure 2
Results of normal desulfurizing in the rotating kiln.

Figures 2 and 3 show some results of normal desulfurizing in this kiln. The burnt lime had the following screen analysis:

Mesh	*mm.*	*%*
32	0.5	2.7
32–65	0.5–0.2	7.3
65–150	0.1–0.06	10.0
	0.06	71.8

The lime addition was in this case 2% of pig iron weight. As can be seen, the desulfurization is extremely rapid. The initial sulfur content was 0.061% and had after three minutes decreased to 0.025% and after 15 minutes to 0.007%. In another test (Figure 3) where pig iron with high phosphor and relatively low silicon content was treated, the sulfur content decreased just as rapidly during treatment, in spite of the high ingoing content in this case. Different tests have shown that the sulfur content under favourable conditions can be brought down to 0.001%.



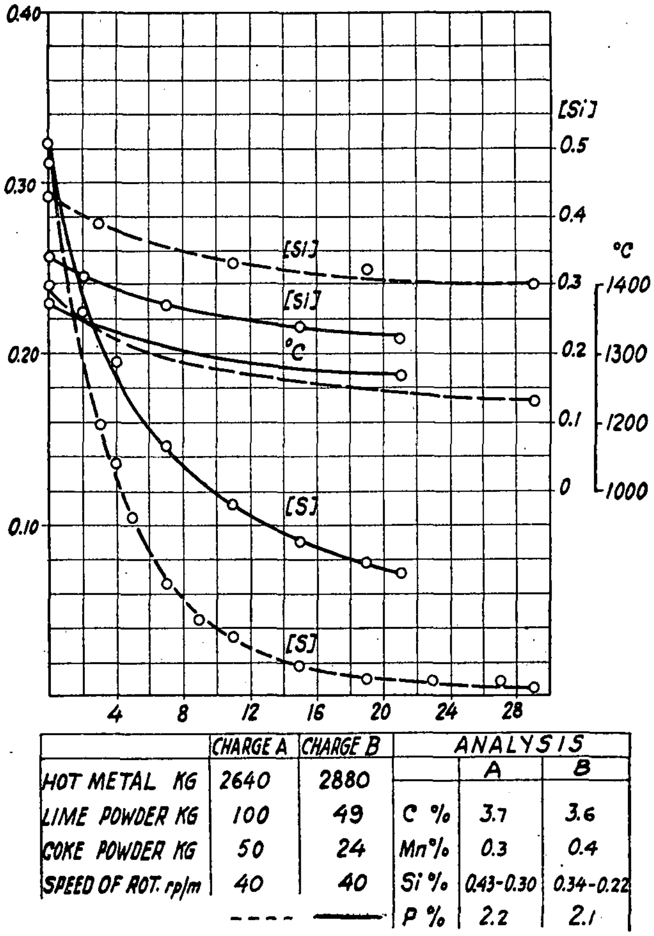
HOT METAL 2230 KG C 3.7% 40 rp/m
LIME POWDER 45 KG Mn 0.5%
COKE POWDER 45 KG Si 0.46-0.40%
P 1.7%

Figure 3
Further results of normal desulfurizing in the rotating kiln.

Figure 4 shows examples of how completely the sulfur can be removed, even in pig iron with very high initial contents. In charge A the sulfur content has been brought down from 0.31 to under 0.003% in one operation by treatment with 3.8% lime. In charge B, in which case the lime addition was only half as large, the desulfurizing was less complete but after 20 minutes the sulfur content was still decreasing. The composition of the powdery lime material after treatment and removal of the added coke and the shots of iron in the powder was the following:

	CaO	*CaS*	*S*	*SiO₂*	*MnO*	*Al₂O₃*
Charge A.....	57.4	23.2	10.3	13.8	0.3	4.0
" B.....	32.9	40.6	18.1	19.8	1.8	2.8

The first industrial application of the process took place at Surahammar Steelworks in Sweden (2), where a great deal of additional experience had been gained. In this plant the process has now been in operation for 2½ years treating about 20,000 tons a year. The kiln capacity is 14 tons and the speed of rotation 34 r.p.m. (Figures 5, 6). The method of working is as follows:



	CHARGE A	CHARGE B		ANALYSIS A	ANALYSIS B
HOT METAL KG	2640	2880			
LIME POWDER KG	100	49	C %	3.7	3.6
COKE POWDER KG	50	24	Mn %	0.3	0.4
SPEED OF ROT. rp/m	40	40	Si %	0.43-0.30	0.34-0.22
	----	——	P %	2.2	2.1

Figure 4
Chart showing the extent of sulfur removal.

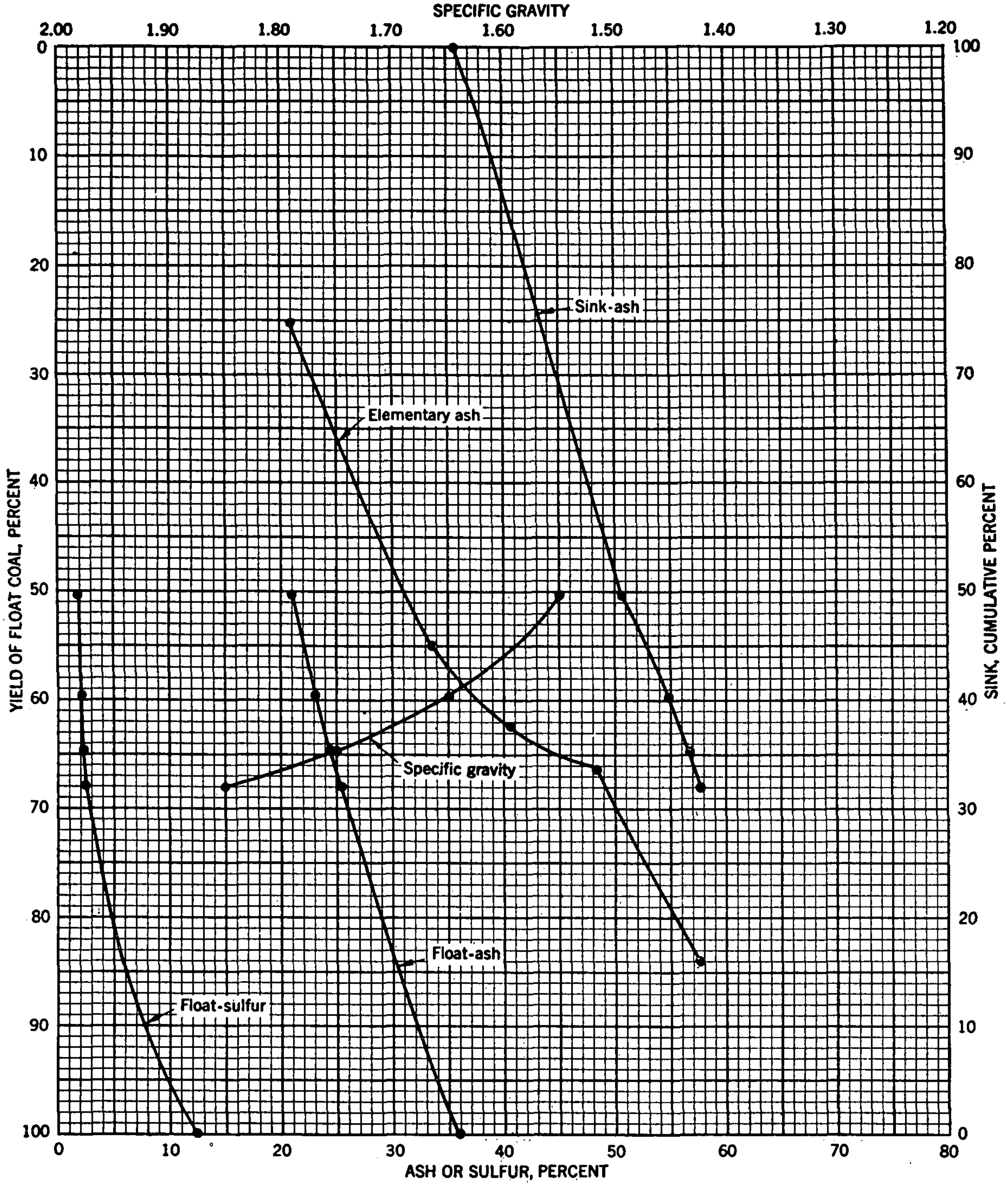
The kiln hanging in the crane is moved to the blast furnace and used as a pig iron container to replace the ordinary ladle. After tapping the blast furnace, the kiln is placed on the rolling stand where it is charged with 1.2% lime and 0.4% coke breeze and closed. Sufficient time of rotation is 15–20 minutes. The iron is tapped from the kiln directly into pigs or into foundry ladles. Table 1 shows the desulfurization results during five months in 1952, from which it can be seen that the average sulfur content is decreased from 0.088 to 0.007%. Table 2 shows that it is possible to obtain extremely low contents of less than 0.005% also during long working periods.

Table 1

DESULFURIZATION AT SURAHAMMAR, SWEDEN

16 April–7 September 1952

Before Charges *Number*	*Before* Charges *Per cent*	*Before* *Per cent S in pig iron*	*After* Charges *Number*	*After* Charges *Per cent*	*After* *Per cent S in pig iron*
50	8.5	0.020–0.050	319	54.1	0.002–0.005
392	66.5	0.051–0.100	160	27.1	0.006–0.010
123	20.8	0.101–0.150	82	13.9	0.011–0.020
22	3.7	0.151–0.200	18	3.1	0.021–0.030
3	0.5	0.200	5	0.8	0.031–0.040
			6	1.0	0.041
590	100.0	Average: 0.088	590	100.0	Average: 0.0075



Specific gravity fractions	Elementary data				Computed cumulative data					
	Weight grams	Percent			Float, cumulative percent			Sink, cumulative percent		
		Weight	Ash	Sulfur	Weight	Ash	Sulfur	Weight	Ash	Sulfur
Float - 1.55		50.2	21.0	1.93	50.2	21.0	1.93	100.0	35.9	12.27
1.55 to 1.65		9.6	33.8	3.01	59.8	23.1	2.10	49.8	50.8	22.69
1.65 to 1.75		5.0	40.7	3.90	64.8	24.4	2.24	40.2	54.9	27.39
1.75 to 1.85		3.2	48.3	4.42	68.0	25.5	2.34	35.2	56.9	30.73
Sink - 1.85		32.0	57.8	33.36	100.0	35.9	12.27	32.0	57.8	33.36

Fig. 9. Washability chart of composite of Barro Branco bed samples crushed to 38 mm. top size. Size 38 – 20 mm. increment.

In order to obtain the best results, the SiO_2 content of the lime should be less than 5% and the lime powder should be ground to less than 0.5 mm. It seems as if the addition of coke breeze could be avoided if high quality lime could be used. Lime consumption is 8 to 15 times the amount of sulfur in the pig iron, the lower figure being valid for high initial sulfur contents.

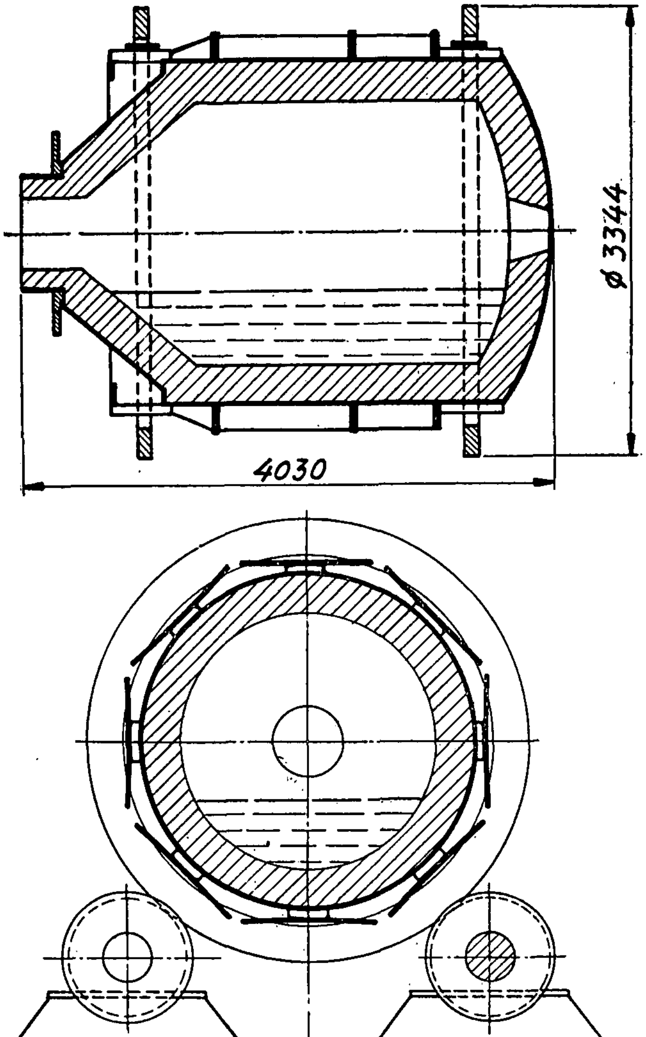


Figure 5
Plan and cross-section of the rotating kiln.

The efficiency of the desulfurization process is not dependent on temperature in the range of 1,200 to 1,400°C. Temperature loss during treatment will be very small if the kiln is used continuously. Generally it can be said that the sealed and well insulated rotating kiln will show the same or less temperature drop as that during transfer in a normal open ladle.

Table 2

Desulfurization results of pig iron at Surahammar, Sweden

17–27 August 1952

Date	*Before treatment*	*After treatment*	*Date*	*Before treatment*	*After treatment*
	% S	% S		% S	% S
Aug. 17....	0.090	0.002	Aug. 22....	0.020	0.002
	0.045	0.004		0.025	0.002
	0.050	0.002		0.025	0.004
	0.050	0.002		0.030	0.003
	0.055	0.007		0.030	0.003
Aug. 18....	0.060	0.003		0.040	0.002
	0.070	0.003	Aug. 23....	0.055	0.007
	0.050	0.002		0.050	0.004
	0.030	0.002		0.060	0.006
	0.035	0.002		0.085	0.003
Aug. 19....	0.045	0.002	Aug. 24....	0.075	0.006
	0.050	0.003		0.070	0.005
	0.035	0.003		0.070	0.005
	0.025	0.002		0.060	0.002
	0.055	0.002	Aug. 25....	0.055	0.002
	0.060	0.002		0.055	0.004
Aug. 20....	0.050	0.005		0.060	0.003
	0.045	0.002		0.050	0.004
	0.055	0.004	Aug. 26....	0.055	0.006
	0.050	0.004		0.055	0.004
	0.060	0.002		0.065	0.005
Aug. 21....	0.055	0.002		0.065	0.004
	0.045	0.003		0.070	0.005
	0.040	0.002		0.065	0.004
	0.045	0.002	Aug. 27....	0.080	0.004
	0.050	0.004		0.090	0.006
	0.030	0.003		0.095	0.007
				0.080	0.004

It is extremely important that reducing conditions are maintained during the treatment. Possible sources of oxygen entering the furnace are unburnt or hydrated lime, wet coke breeze, and slag with a high content of FeO following the pig iron in charging the furnace. The kiln should never be heated during continuous operation, but in some cases where it cools off too much, heating between the treatments might be needed. This is simplest with gas or oil, but in order to prevent oxide formation completely, electrical heating with electrodes may in some cases be preferable.

The desulfurization result is dependent on the contact between lime and pig iron and therefore the speed of rotation should be as high as possible. A speed at the periphery of between 3 and 3.5 m/s has been found necessary. With this speed the time of treatment will be 15 minutes when contents below 0.010% are wanted. Acid as well as basic linings have been used, and it has been found that the bricks are only negligibly eroded due to the fact that the lime stays as dry

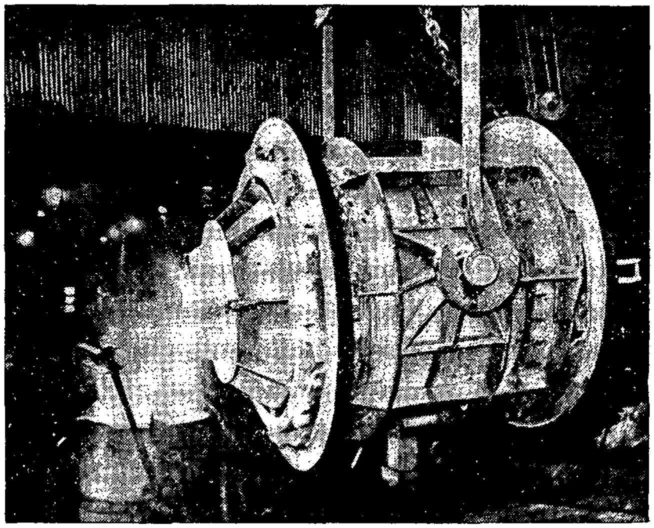


Figure 6
Photograph of the rotating kiln.

powder during the whole treatment and no molten slag is formed. Tests are under way with a carbon brick lining, which for many reasons should be ideal.

When installing a rotating kiln in a steel plant, it should if possible—as in Surahammar—be used as a ladle at the same time as it is used for treatment with lime, in order to avoid too much reladling of the iron. The desulfurization can be done between the blast furnace and the mixer or between the mixer and the steel furnaces, depending on local conditions.

In Fagersta, Sweden, a rotary kiln of 25-ton capacity will shortly be put into operation. About 100,000 tons of pig iron per year will be treated there. Domnarfvet Steelworks are planning an installation for about 200,000 tons a year. The first kiln outside of Sweden is under erection in Normandy, France, where high sulfur pig iron made with relatively acid slags in the blast furnaces will be treated.

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Some Notes on the Organization of Monclova Steelworks*

PABLO SADA

I

At the end of the first half of the sixteenth century, the intrepid Muñe de Guzmán had pacified what was called the Kingdom of Nueva Vizcaya in New Spain, and one of his captains, don Cristóbal de Oñate, had founded the city of Guadalajara. Another of them, Ginés Vasquez de Mercado, desirous of glory and riches, requested and obtained his chief's permission to organize an expedition and march to the north in search of a legendary mountain of gold and silver.

The adventurer set off for the unknown, and after innumerable setbacks reached the mountain of the supposed fabulous wealth. What must have been his chagrin upon finding that it was a hill composed of hematites, with no ores containing precious metals. During its return, the disappointed expedition was attacked and cut in pieces in Juchipila Canyon by the Tepehuan Indians. Only a few soldiers escaped, carrying with them the dying Vasquez de Mercado. In his delirium the captain exclaimed "Iron, iron, that is what we need; gold and silver conquered with iron."

Four centuries have passed since that time, but the words of Vasquez de Mercado, far from solely expressing the incoherent cry of a dying man, contain the cold sense of a prophecy. (From that time to this, mankind has, first uncertainly and weakly, now with assurance and speed, been walking to the present age which may well be called the Iron Age, the Steel Age). The words of Vasquez de Mercado certainly have a bellicose ring, but in a world seeking justice and peace, they express the desire to overcome time and distance and the wish for success in the fields of scientific research and economic development of the countries of the world.

At the present time, an era of a new industrial revolution, the development of the iron and steel industry is considered essential for the progress of the world, not only in the highly industrialized countries, but even in those where mining, stockbreeding and farming play a dominant role and where, in a rudimentary and perhaps domestic fashion, an elementary industry has just been commenced which is far from filling the minimum requirements of those countries.

Private enterprises and the governments have realized that industrialization has become an exact science which, when properly applied, taking into consideration the results obtained in other regions and the peculiar conditions in their own, may give a decisive concrete reply as to what may be expected from steadfast efforts. Governments have made industrialization a part of their programmes, and as a most important branch within these, the development or commencement, as the case may be, of the iron and steel industry has been taken into account.

Latin America, in general, is not outside this movement. Composed of countries whose economic and industrial development is not generally speaking at the level of other countries, it has sought the progressive expansion of its iron and steel industry, trying to satisfy its steadily increasing requirements with steel products made by national technicians and workers.

It may be said that sixty years ago Latin America had no commercial undertaking for the production of iron and steel for the market. At present seven Latin-American countries have units which, while not outstandingly large from a world standpoint, have, little by little, been developing and have tried to fulfill their essential function.

Those Latin-American countries which have iron and steel plants are considering the desirability of increasing their capacities and output, while those countries which have no such plants are giving serious thought to the possibility of establishing them.

When considering the possibility and desirability of an iron and steel plant, thought should be given to market requirements as well as to the quantity and

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* Due to the untimely decease of the author, this paper is published in its original version, unabridged and unrevised.

quality of the available raw materials. Technicians of highly industrialized countries are often inclined to think solely of the facility of reducing man-hours per ton produced, regardless of the initial investment, since they usually build with large sums of money obtained at low rates of interest. Consequently, they do not recommend the installation of an industry on a modest or small basis, in which the installation may expand and become progressively mechanized in accordance with the growing requirements of the market, without sacrificing the initial plant.

Once it has been exactly determined what products to make, and the quality and quantity required, not only the most modern methods of manufacture should be studied, but also those which are still in use in less modern plants. An attempt should be made to obtain used machinery in good condition which can begin to give immediate results, even when it implies a greater use of labour, which in our countries is proportionately cheaper. In this way, a plant can be obtained which is possibly less efficient at the beginning, but which has the following advantages:

(1) It stands on solid economic foundations;
(2) It is capable of producing within a relatively short period;
(3) The initial investment is relatively low;
(4) Our own resources and labour are used in a higher degree;
(5) It is capable of growing and expanding its activities, and of increasing its production in such a way that the profits can be invested in the enlargement and modernization of the installation.

II

It is of fundamental importance to carry out a plan for the integral training of the staff, so that within a short time there will be a sufficient number of "key men" capable of directing a heavy industry. It is not enough to rely on our universities, which produce excellent technicians who naturally lack experience, for the provision of first-class education.

It is necessary to select men with capacity and possibilities, who should if possible be sent to do postgraduate work in universities in the more industrialized countries, as these schools, with the co-operation of industry, can provide greater facilities for a harmonious combination of technical studies and practical work.

A periodic interchange should also be sought, on the basis of a scrupulous selection of technicians from among the different Latin-American countries, since they may find therein the solution to problems similar to those prevailing in their countries of origin. Although technical problems are always solved through the analysis of figures, it is found in practice that the application of methods for solving them has both distinctive and similar features in different countries.

Some important industries in other countries have been fortunate enough to develop through the genius of men guided by a fundamental idea which has been the reason for their lives and to the achievement of which they have dedicated all their efforts. Fortunate indeed is that country which has been blessed with such visionaries, since its progress and the development of its industry have been proportionate to that phenomenon.

Whereas all countries have not had the same good fortune, they can all, through a careful, systematic education programme, train their "key men" with technical knowledge to solve their problems and to overcome the difficulties which may arise in the construction of the plant, in staff administration, in the general maintenance of machinery and equipment, and in the control of costs and distribution of products. The training of this group of "key men" with a clear understanding of industry in general, could become the basis for building up an iron and steel industry with a low investment cost per annual ton of capacity. This is important in our countries, where financing charges represent a very high percentage of the cost.

Industry should give all kinds of facilities to young technicians and, according to their capacities, should periodically engage young professionals for the purpose of giving them essential experience, profiting in turn as their boards are renovated with young, properly trained men.

A training scheme such as the one proposed, complemented with study trips to iron and steel plants abroad and attendance at steelmen's conventions, will have given those technicians such versatility in their thinking that when planning new developments they can choose between using reconditioned equipment which is being replaced in the great steel producing centres, as their use will be possible in a new plant by means of certain changes or modifications; or else they can acquire equipment or machinery designed especially for their particular requirements.

III

Possibly a single example in which these suggestions have been carried out does not constitute final proof of their adaptability to all conditions; nevertheless, in one case at least, circumstances were propitious for obtaining the desired results.

During the years 1937–41 the following had been imported annually into Mexico:

7,000	tons of plate
16,000	tons of black sheet
17,000	tons of tinplate
9,000	tons of iron piping
49,000	

Sufficient deposits of iron ore, coal, limestone and fuel oil had been located within the country, as well as other raw materials necessary for the creation of a plant which would supply the demands indicated above.

Moreover, the Mexican Government had set up in the Nacional Financiera, an institution for assisting and fostering the country's industrialization, to strengthen private enterprise in large-scale projects which private firms could not achieve alone.

Plans included an installation with an annual output of 18,000 tons of tinplate, 15,000 tons of cold rolled sheet, 15,000 tons of plate and 7,000 tons of hollow iron tubing. This may seem small in comparison with industries in iron and steel producing countries, but for Mexico it meant a tremendous economic effort. Those Mexican bankers and industrialists who undertook this apparently Roman task, always had the support of the Government of Mexico, directly, and also through the Nacional Financiera, and it was the combination of all these elements which enabled the idea to be put into practice.

In December 1941 the Altos Hornos de Mexico S.A. were legally constituted, with a capital of 52,310,000 pesos.

The basic units for the plant had been ensured by means of a number of options obtained on certain essential installations for this type of industry, consisting mainly of a blast furnace located near the city of St. Louis, Missouri, and a universal mill for plate and strip which was found at Indiana Harbor, also in the United States. These units, as well as other essential equipment, had been held by option since the beginning of 1941 and were bought at the beginning of 1942, once the new company had been formally organized.

Had that equipment not been available, Altos Hornos de Mexico would not have been considered as an immediate possibility. Delivery dates promised for the heavy equipment manufactured specially for the projected plant were so remote and uncertain that they would have made the project unfeasible from the very beginning.

The first material for the construction of the blast furnace reached Monclova on 27 April, 1942, and construction formally began on 5 October of that same year. Twenty months later, on 2 June, 1944, the blast furnace went into production and four months later, on 9 October, 1944, the first basic open hearth furnace was lit, the universal plate mill beginning operations the following day. Altos Hornos de Mexico, notwithstanding the difficulties encountered, with staff trained as the work proceeded, and without the experience universally considered to be essential, under completely adverse circumstances due to the Second World War, and using equipment which had been reconditioned and readapted for the projected plant, had succeeded in carrying out the original idea.

Assistance was received from the Economic War Committee and priorities from the War Production Committee, both in Washington, through offering to supply the Allocations Committee with plate for ship-building; thousands of tons of plate were dispatched in fulfilment of that promise. It is a matter of pride for Mexico that inspectors of the United States Maritime Commission always found the material to be within the standards specified.

Once the war effort was over, the plate was put on the market in Mexico.

The cold rolling mill, for which entirely new equipment was purchased, could not commence operations until February 1946, when it also produced the first tinplate in Mexico.

Originally the Altos Hornos plant had consisted of a blast furnace with a maximum daily production of 200 metric tons of pig iron, the main limitation being the lack of coke; two basic open hearth furnaces with a capacity of 65 tons per charge, one hot rolling mill for plate and strip; three machines for the centrifugal castings of iron piping, and the essential maintenance and service departments.

Since that time Altos Hornos has been increasing and improving its production. At the present time the reconditioned blast furnace is producing an average of 350 tons daily and work has already commenced on the design of another blast furnace with a daily capacity of 800 tons. There are three basic open hearth furnaces with a capacity of 105 tons per charge, and by the end of the current year a fourth furnace will be completed which will produce 120 tons per charge. The universal mill, which currently produces 15 tons per hour of strip in rolls of an approximate thickness of 4.57 mm., is being complemented by a reversible type Steckel mill, which will increase production to over 35 metric tons of 2.54 mm. strip per hour.

Annual production of steel ingots since 1944 has been as follows:

	Tons
1944	4,300
1945	44,000
1946	74,300
1947	75,200
1948	85,900
1949	118,100
1950	123,100
1951	143,100
1952	162,700 (estimate)
1953	221,000 (estimate)
1954	260,000 (estimate)

The investment cost of the plant before beginning the current expansion programme was 106 million pesos, and as 143,167 tons of steel ingot were produced in 1951, the investment cost of the plant is 740 pesos per ton of steel ingot, equivalent to US$85.55.

Upon completing the expansion, the total investment will be approximately 180 million pesos, and 260,000 tons of steel ingot will be produced annually, giving a comparative investment cost for the plant of 693.71 pesos, equivalent to US$80.10 per annual ton of capacity of steel ingot. These unit prices compare favourably with the investment per annual ton of capacity of steel ingot in other more industrialized countries.

The preceding paragraphs and the data included therein provide in figures the brief history of a steel plant built under difficult—even critical—conditions, which at the beginning used reconditioned machinery and equipment and inexperienced staff who, by means of a training plan and much hard work have been able to achieve the outstanding results described. Although it is not possible to go into great detail concerning the work done, an idea has been given of the potentialities, which is precisely why this paper was prepared.

As a concrete suggestion, and in a spirit of collaboration with the other countries of Latin America, Altos Hornos de Mexico wishes to make the following proposal.

Since Latin America already has sufficient steel-producing installations to be used for the training of those aspiring to be the "key men" who will be needed for carrying out new and well planned installations which are economically feasible, it is only necessary to establish a method or system whereby these industries can be placed at the service of the other countries, and thus the advantages of low cost production will be obtained, to the benefit of their respective economies. Altos Hornos de Mexico is authorized to take the first step in this direction, and offers, as from this date, to accept in its plant a limited number of candidates from other Latin-American countries, in order to give them a general training in industry for several years, so that they will be duly prepared for positions of responsibility in the development of basic iron and steel industries in their respective countries. This offer naturally is limited as regards the number of persons concerned, subject to their aptitudes and their application to work, while carrying out their training. It is to be expected that this will be a very important step in meeting future requirements, in so far as the basic iron and steel industry is concerned. Moreover, the development of a group of key men is the most important step that can be taken for the preparation and training of a group of technical leaders with the ability to determine which are the most suitable products to manufacture, to select methods and equipment for the manufacture of those products in the quantities and qualities required by the market at a minimum production cost, always taking into account the future market requirements.

By means of this offer, the goodwill, the experience acquired on the basis of pitfalls and setbacks, the selective and progressive training plan implanted in this plant in Mexico are at the service of similar industries in Latin America.

Brief Account of Iron and Steel Producing Processes Used in Brazil

AMARO LANARI

I. Introduction

The modern phase of the development of the Brazilian steel industry began in 1888. This development was rendered possible by the penetration of the first railways to the central part of the state of Minas Gerais where the enormous deposits of very high grade iron ore are to be found. Two blast furnaces were built at that time in Esperança and Burnier, the latter one being still in operation at this time.

The two world wars created favourable conditions for the growth of the steel industry, whereas at the same time the improvement in the means of transport, the increase in the internal demand and the growing industrial experience allowed the steel industry to reach progressively more and more satisfactory levels in terms of quantity and of quality.

Table 1

Brazilian Iron and Steel Production

(thousands of metric tons)

	1920	*1925*	*1930*	*1935*	*1940*	*1945*	*1950*
Pig iron	14	30	35	64	186	260	729
Steel	—	7	21	64	141	206	789
Rolled products	—	0.3	26	52	135	166	623

Source: Ministry of Agriculture, Office of Production statistics.

II. Localization of the Brazilian Iron and Steel Industry

The iron and steel industry was first localized in the centre of the state of Minas Gerais, close to the iron ore deposits and to large forest resources which formed the basis for the charcoal used as reducing agent in the blast furnaces. Later, the industry became scattered over the country, with a primary view to location near the consuming centres.

In those regions close to iron ore deposits, such as the states of Minas Gerais, Rio de Janeiro and a part of São Paulo, steel is produced in integrated plants provided with reduction furnaces. In the plants which are far away from the ores—such as those of Pernambuco, Rio Grande do Sul, Santa Catarina and many of those in the state of São Paulo, steel is made directly from scrap, with or without an addition of solid pig iron.

III. Availabilities of Iron Ores and of Fuels

Ninety-nine per cent of the iron ore utilized by the Brazilian steel industry comes from the large deposits of the central area of the state of Minas Gerais, the so-called "ferrous quadrilateral" limited by the cities of Belo Horizonte, Lafayette, Ouro Preto and Itabira. The ore reserves of this region are estimated to amount to 13,000–15,000 million tons.

The ore is hematite and the deposits are generally covered by a layer of a ferrous conglomerate called "canga" which is itself an important ore. The canga is formed of pieces of ore held together by a limonitic cement and is very easy to reduce. Thus, although its iron content is lower than that of other ores, it is widely used in charcoal blast furnaces.

The iron bearing formations do not have a fixed composition. There is thus a progressive enrichment from quartzites, which contain no iron, to compact and extremely pure hematites. A very characteristic intermediate formation is the so-called "itabirite", an ore which, in its most typical form, is constituted by fine alternate layers of hematite and of silica. In the same way "filitos" evolve gradually, through enrichment in iron, into a pure hematite form.

This explains the variety of aspects and of chemical compositions found in the ores and which gives rise to various local denominations such as "chapinha" or quartzeous itabirite, clayey itabirite, etc. The chapinha is a very porous itabirite, partially limonitized, and takes the shape of fine layers. It constitutes, together with the canga, the preferred ore for small charcoal blast furnaces.

Practically pure hematite also presents variations in aspect, especially as far as hardness is concerned. Thus, there exists a compact hematite which is extremely hard but also a very friable hematite called "jacutinga".

The ores currently used for iron and steelmaking are of three main types:

1. Canga;
2. Itabirite under various denominations;
3. Compact hematite.

The following table shows a few typical compositions of the ores:

Table 2

TYPICAL COMPOSITIONS OF BRAZILIAN ORES

	Canga	*Silicious itabirite*	*Aluminous itabirite*	*Compact hematite*
	%	%	%	%
Fe	59	64.3	61	69
Mn	0.1	0.19	0.7	0.04
SiO_2	6.0	4.3	4.0	0.14
P	0.14	0.08	0.07	0.02
Fire losses	7.6	4.7	3.7	0.2
Humidity	2.5	1.2	1.5	—
Al_2O_3	—	—	4.8	—

As regards phosphorus content, it may be observed that:

(1) The canga is a typical intermediate ore, which is not adaptable to classical production, either of Bessemer or of Thomas pig;

(2) Itabirites generally have a phosphorus content too high for the Bessemer process. Sufficiently low-phosphorus itabirites are found but phosphorus content is variable so that the conditions required for a normal production of Bessemer pig do not exist on a stable basis;

(3) Compact hematites are generally sufficiently pure for processing by the classical Bessemer method.

Only Volta Redonda in the state of Rio de Janeiro uses metallurgical coke, all the other furnaces using charcoal. Some of them have large capacities, perhaps the largest in the world.

The importance of charcoal for the Brazilian steel industry should not be underestimated. Until now, only natural forests have been exploited for the manufacture of charcoal, but some large plants are beginning to exploit artificial forests containing selected varieties of eucalyptus. The Companhia Paulista de Estrados de Ferro in São Paulo has had lengthy experience in the cultivation of eucalyptus. This experience proves the exceptional advantages of this tree for the production of charcoal in Brazil, inasmuch as it can be grown on poor agricultural land and because it reaches the maximum level of production per hectare and per year in the short period of seven to eight years.

Charcoal pig iron constitutes approximately one half of total Brazilian production and is obtained in 32 blast furnaces, with capacities varying from 10 to 200 tons per day. The rapid growth of forests in the Brazilian climate, the existence of large areas which can be used for the creation of artificial forest resources and the long inland hauls are all economic elements which are favourable to the retention and even to the development of steelmaking based on charcoal. With an absolutely normal chemical composition—that is with 80–82% of carbon content—charcoal generally shows in Brazil a high specific gravity, always higher than 200 kg. per cu. m. but sometimes reaching 300 kg. (including natural humidity).

Brazilian mineral coal, which has a high ash content, produces a metallurgical coke with excellent physical properties. For economic reasons Volta Redonda uses a blend of domestic and imported coals, in proportions which vary with the conditions of the moment. Brazilian reserves of coking coal are currently estimated at 500 million tons.

IV. REDUCTION OF IRON ORES

All the pig iron produced at present in Brazil is obtained in blast furnaces. Electric reduction furnaces do not exist today, but at least two steelmaking enterprises will shortly initiate the construction of such furnaces. This initiative must be viewed in connexion with the great postwar effort to reduce to a minimum the consumption of carbon in pig iron production.

In addition to the increasing interest shown for the installation in Brazil of electro-metallurgical plants, there have been some very favourable results in the use of sintered ores such as in the Companhia Siderúrgica Belgo Mineira. The consumption of carbon is reduced to approximately 650 kg. per ton of pig iron.

The characteristics of the Brazilian market are not propitious to the development of steel industry in the direction of small size production of special products—as in Sweden—but rather to the large-scale elaboration of standard quality steel. This is why the blast furnaces which use easily reducible ores—even if they are of lesser purity, such as canga and itabirite—develop their capacity rapidly, in order to produce a cheaper pig iron. The larger charcoal furnaces are those of the integrated plants which produce bars; wire rod, sheet and galvanized tubes. The small furnaces produce foundry pig iron and such products as centrifuged iron tubes, sanitary and domestic equipment, agricultural implements, and iron and steel castings. Almost all these plants are attempting in addition to establish sintering installations which, taking advantage of the fines which often abound in the ore and of the siftings of charcoal which until now were wasted, can constitute a fundamental element in the production economy of every steel plant.

It has already been noted that in those countries in which industry is only at its incipient phase, new processes and new techniques may be of greater interest than in countries in which industry has established itself solidly on a large scale. The reason is that in more developed countries, a new process, in order to be applied on a commercial scale, must first dislodge older processes which, although they may be less efficient, can compete advantageously because they are already solidly established technically and commercially. Thus, for instance, commercial aviation can be established easily and rapidly in the countries which have scant means of transport such as Brazil, because other communications are difficult to make use of, or simply do not exist.

As far as new methods of iron ore reduction are concerned, it may be said that all those which involve a reduced consumption of coal or the possibility of using low-grade coals, are viewed with considerable interest in Brazil. Such are the Wiberg process, the oxygen low-shaft furnace, and the hydrogen reduction methods.

As soon as these methods can be applied on a reasonable scale and at a cost which would allow steel for current uses to be produced, their utilization in Brazil will probably not meet with any impediments. On the contrary, they will probably be considered with the greatest of interest.

The yearly congresses and meetings of the Associaçao Brasileria de Metais and the Mining and Metallurgical "Study weeks" of the Moraes Rego Centre, which bring together practically all the metallurgists of the country, have made valuable contributions to a better understanding of the specific problems of Brazilian steel industry. The debates have already resulted in the establishment of a clearer understanding and of a more or less uniform approach to these problems. Thus as far as the current methods for the reduction of iron ore are concerned, all agree in that both the charcoal blast furnace and the coke blast furnace or the electric reduction furnace can co-exist in the country, as favourable conditions exist for every one of these processes. The choice will depend on locational conditions, availabilities of electric energy and on the production programme of the various steel plants.

V. Steelmaking processes

The properties of those ores which are the easiest to reduce lead Brazilian integrated plants to the adoption of the basic open hearth process. There is one exception, a plant essentially engaged in the production of special steels, which utilizes a duplex system, i.e., converter plus basic electric furnace. Electric furnaces are generally used in non-integrated plants, located in large consumption centres, where there are certain availabilities of scrap. Steel production in electric furnaces in Brazil does not represent 20% of the total.

There are also a few small Bessemer converters used to produce steel castings, but they have little statistical significance.

Although the proportion of purchased scrap used in relation to total production of crude steel represents 27%, this scrap is preferably used in electric furnaces so that the major integrated plants practically use circulating scrap exclusively in open hearth furnaces. As a result, the working formula used in the open hearth furnaces is decidedly the ore process, compact hematite being used for oxidizing.

In view of the intermediate phosphorus content of Brazilian pig iron and of the interest shown by all plants in obtaining a maximum quantity of crude steel from every ton of pig iron, it will be understood that the utilization of the oxygen-enriched converter, which also allows for ore additions, is one of the possible directions in which steelmaking processes may develop in Brazil.

VI. Rolling operations

Approximately 30% of the Brazilian crude steel output is obtained in small ingots 100 to 150 mm. square with weights varying between 80 and 180 kg. Only Monlevade and Volta Redonda use ingots exceeding three tons which are rolled into bars, sections, wire-rod, strip and sheet according to the classical methods used in large-scale European and American industry.

As a result of the youth of the Brazilian steel industry there are a great number of small plants. This scattering is also due to a limited and diversified market and to the great distances separating the consuming areas.

It is to be noted that the inauguration of large plants like Volta Redonda has not handicapped the installation and growth of small producing units. A typical size for such small plants would be that corresponding to a yearly production of 30,000 tons of ingots, each one weighing some 130 kg.

The economics of production in plants of this type is seriously studied in Brazil. Assuming a production of 10 tons per hour of rolled products, which is satisfactory for a reasonable mechanization, it may be seen that an adequate size for such plants would correspond to an annual production of 50,000 tons of rolled products, work being performed in two shifts of eight hours, under perfectly acceptable technical conditions.

Although rolling losses of small ingots compare favourably with those observed in large plants provided with blooming mills, pouring operations and the handling of a large amount of small ingots constitute a serious handicap. For this reason, continuous casting of billets and blooms would be a very adequate solution. Eliminating the difficult casting work, this method would also eliminate the expenses connected with the blooming mill, and would provide billets 60 to 80 mm. square. This is why the development of continuous casting techniques has been followed with great interest in Brazil.

Another possible solution which involves low investment costs would be the forging of billets and blooms on rapidly operating presses.

Summary of Discussion

Approximate Comparative Production Costs and Investment Requirements of the Standard Blast Furnace, Electric Smelting Furnace, Sponge Iron Furnace and Tunnel Kiln Sponge Iron Furnace, *presented by the author*

Mr. Cavanagh began his presentation by explaining that he believed experiences in Canada in relation to iron ore might be of some interest, since conditions were closely similar to those in Latin America. There was a large market for iron and steel in the centre of Canada, but there was no coal, and iron ore had only recently been found there. The iron and steel industry in central Canada had therefore been based on imported coke and ore, and was still to some extent so based, but during the past ten years the possibility had been studied of expanding the existing industry and adding new units to it. Some very interesting conclusions had been drawn from that study.

One of the most interesting facts was that in Canada and the United States, steel companies were installed, not for the purpose of making steel, but for making money; it was thus the banker and not the metallurgist who in the last instance decided whether or not new equipment should be acquired. Since bankers were the same everywhere, they liked to examine operating costs over a period of years.

His own views on some of the processes that had been discussed during the meeting were not in agreement with some of the more recent statements put forward. He thought, for instance, that it was excellent that sponge iron processes were carried out in small units, because those units could be financed by the company concerned out of its own funds, without needing to approach a banker. At the other end of the scale was the blast furnace, which had often been installed because funds for it were easy to obtain, notwithstanding the fact that its output might be too great for the particular market.

Canada was a conservative country and did not believe in spending its money on development work when it could benefit by the results obtained by others. A great deal of the development work was therefore based on an exchange of information with other countries.

Reverting to the bankers' attitude, he did not believe that it made any difference to a banker what process was used, providing that it showed a good margin of profit. After a process had been proved to be profitable on a commercial scale, under specific conditions, there was no further argument from the technician's standpoint.

The sheet of figures which had been circulated to participants had been taken from a document prepared for the UNSCCUR meeting held in New York, and showed the results of investigations made in Canada. The figures did not represent any particular place or company, nor were they meant to be accurate in so far as investments or production costs were concerned; they were useful as comparative figures, however, and most of them applied to the central portion of Canada. They were for a hypothetical plant located somewhere in that part of Canada, some 90 miles from Niagara Falls, and having some open hearth furnaces and some electric melting furnaces. He then explained the various figures for plant and furnace capacity, production costs per ton, total investment, investment per ton year, and write-off investment, for a blast furnace, an electric smelting furnace, a sponge iron furnace and a tunnel kiln. The tunnel kiln sponge iron process, according to the figures, would not be suitable for that part of Canada.

With regard to the next process listed in the paper, the Wiberg Söderfors sponge iron process, about 90% of the engineers in the United States would be against the idea of using it, because the output was too low. He held the opposite view, as if the capacity of the installation were kept to the total yearly output shown in the figures, the process would be suitable for that particular location in Canada. Merely because sponge iron had not been made successfully on a commercial basis in the United States, that did not mean that the Swedish experience should be forgotten. His organization had investigated the process thoroughly and had discovered that it was a commercially feasible and reliable method, and might provide the best solution for that particular location.

If pig iron for making steel were also required, then the electric smelting furnace should be considered. The corresponding capital investment was about the limit that the hypothetical company could afford. Due to the snow problem, all the materials had to be assembled during one six-month period in order to continue operations during the following six months, so that storage facilities were required. Therefore for higher tonnage output, investment in stockpiling equipment and in coke ovens would eliminate the advantage of electric smelting at the output quoted. The figures showed that at that time (1949) the cost of electric pig iron was quite reasonable, compared with scrap and purchased pig iron, so one company in Canada had been quite prepared to go ahead and build its furnace. They were stopped because they could not get a firm contract for electric power as there was insufficient fluctuating power available between the peak loads.

The question arose as to whether to wait until sufficient electric power was available, or to install a blast furnace, which was easier because the money was available. The output of the smallest blast furnace was far above that required by the hypothetical company, yet the choice of process was influenced more by the availability of money.

He referred next to the Hojalata y Lámina S.A. plant at Monterrey, explaining that there the sponge iron tunnel kiln had been the solution because raw materials were cheap, the process would produce about the amount of material required, and more important still, the company could finance the process with its own resources. The blast furnace offered no solution there, because the money for it could not be found even if the resultant materials could be used. Power was expensive, which eliminated the electric furnace, while the Wiberg process, where labour cost was low and capital investment high, gave comparative production costs of approximately $36 as against $30 for the tunnel kiln, when both processes were making the same product. The results had proved satisfactory not only to the company, but to the financiers, and three more kilns were planned.

He emphasized that the size of a unit had nothing to do with making a profit, as it was quite possible to make a profit with a small unit.

In actual fact, when there was an apparent choice between processes, it eventually appeared that there was no such choice. The characteristics of the ore determined what steel could be made and how; costs determined how the ore should be smelted, and finally the comparative costs indicated which process should be selected. Ultimately, there was the question as to whether money was available.

By analysing conditions in a certain location, a clear-cut indication could be given as to whether or not to make a particular choice.

Development work on different processes should not be undertaken by a country which was only going to apply the process to establish a steel industry, as there were plenty of places in the world, including Canada, which were spending considerable sums of money on studying the processes on a commercial scale, and the results were eventually available. As an example, he could cite a new process which he would describe in another paper. Canada could not afford to develop the process on a commercial scale, but the results of the development would be available and Canada would participate therein as much as possible.

He stressed that the most important point was neither size nor profit. There were many places in Latin America where steelmaking was contemplated for the sake of making steel rather than money. A steel industry provided the basis for the industrialization of a country. That was very worthwhile and it had been proved in Canada that it often went hand in hand with making money, but always with raising living standards and with economic development; whether a beginning was made with 5 or 5,000 tons a day was therefore not important, since output should depend on the results of market studies.

Mr. NEWHALL opened the discussion of the paper by remarking that it had been prepared from the Canadian standpoint; there was only lump ore in Canada and therefore the comparison between ore for the blast furnace and for the electric furnace should be made for each locality. Mr. Cavanagh had pointed out that the tunnel kiln process was suitable in Mexico on account of cheap labour being available there. Conditions in other parts of Latin America might change due to the cost of ore.

Mr. CAVANAGH replied that he entirely agreed with the previous speaker, adding that the figures given were comparative ones for a certain location and would have to be changed for any other location. The amounts in the paper had to be multiplied by the prices prevailing at a given location.

Desulfurization of Pig Iron with Pulverized Lime in Rotary Kilns[1]

Mr. KALLING explained that he had been asked to say a few words about the method of desulfurizing pig iron developed by his company in Sweden. Blast furnace costs were sharply influenced by the sulfur question, and when it did not have to be considered, it was usually possible to make a much cheaper pig iron or hot metal. There were several means for desulfurizing pig iron, but the method he was about to describe was a very simple and effective one. The principle lay in tapping the pig iron directly from the blast furnace into a rotary furnace; some very fine grain burnt lime was charged into the furnace, which was then rotated at a fairly high speed, to achieve good contact between the lime and the hot metal. After a few minutes of rotation, the sulfur had passed completely into the lime, providing there was a sufficient quantity of lime. During the whole operation the lime was retained in the powdered condition, which was very important for obtaining an effective desulfurization, and the lining of the furnace was in no way attacked. Indeed, the furnace could be used for a year or more without re-lining. One very important point was that the process took place under strongly reducing conditions and the furnace had to be sealed during the operation to prevent oxygen from entering it. At the same time the furnace was used as a ladle for the pig iron, which was tapped from the blast furnace into the rotary furnace and after that the rotary furnace was transported by crane to the rotary equipment, rotated for some minutes and then again removed by crane and transported to the steel plant or mixer. It was thus possible to decrease the sulfur content in a few minutes down to 0.005%. The lime content would be about ten times that of the sulfur to be removed. The process had already been used for about two years at the Surahammar Steelworks in Sweden, where it had been developed from the pilot scale used by his company. Very good work was being done with it. One of the largest steelworks in Sweden was going in for the method and Domnarfvet was also building large furnaces for a desulfurizing plant following that method.

Mr. LEUSCHNER opened the discussion on Mr. Kalling's statement, and requested information as to the amount of lime used and under what conditions, and whether it was calcinated lime.

Mr. KALLING replied that the lime averaged about 1% of the pig iron production.

Mr. LEUSCHNER said that he understood that that amount was related to the specific sulfur content in Swedish pig iron, but he wished to know what the ratio would be if the treatment were applied to pig iron with more sulfur. Moreover, he wished to know the efficiency of the lime in the reaction.

Mr. KALLING answered that at his plant the calculation was about ten times as much lime as there was sulfur to be removed, or a slightly higher ratio when the pig iron had a low sulfur content; 0.1% of sulfur would require 1 to 1.2% of burnt lime, while 0.3% of sulfur would need about 3% of lime.

Mr. ALLARD remarked that generally speaking it seemed very expensive to prepare the burden for blast furnaces from both ore and coke, and in his view the best way was to obtain the reduced iron by the cheapest method, from the blast furnace or from other apparatus. From that standpoint, the process described by Mr. Kalling was a very good one, as it seemed to be very cheap and very easy to apply. He thought therefore that it would be of interest to Latin-American firms as they sometimes had to contend with very poor coke or no coking coal, or sulfurized coal and coke.

Mr. SANITER wished to know how important it was to prevent blast furnace slag from entering the reaction vessel.

Mr. KALLING replied that the question was an important one. It had been found at the Surahammar plant that care had to be taken to avoid blast furnace slag from entering the furnace; the result could be adversely affected if too much slag did enter the furnace.

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[1] In view of the interest to the meeting of Mr. Kalling's remarks under this heading, he was asked by the Chairman to prepare a short paper on the subject for inclusion in the printed proceedings. The paper he prepared is therefore included among the other papers presented at this particular meeting.

Some Notes on the Organization of Monclova Steel Works, *presented by the author*

Mr. Ballón opened the general discussion on the paper, and expressed his enthusiasm at the description given of the Monclova plant, and his appreciation of the offer made at the end of the paper, an offer which he felt sure would be gladly accepted.

Mr. Sherover said that he was particularly interested in the experiences of Monclova, because his own firm in Venezuela had also met with many setbacks in its four years of operation. There had been no possibility of installing a completely integrated industry, but an electric furnace was in use, together with a rolling mill for the most essential products, such as reinforcement rods.

Many people in Venezuela had opposed the idea of the plant, on the grounds that unless a big industry could be started, there was no point in entering the field. They had been proved wrong. The first furnace was currently working at full capacity, the second electric furnace and the second rolling mill had been purchased and would increase production by 150%. The generous offer made by the Monclova firm would certainly be gladly accepted in so far as the speaker's firm was concerned.

The meeting was suspended at 4.30 p.m. and resumed at 5 p.m., when Mr. Juan Uribe Holguin, Minister of Foreign Affairs, arrived for the formal celebration of United Nations Day.

Mr. Uribe Holguin explained that the meeting's ordinary work had been interrupted in order to celebrate the seventh anniversary of the constitution of the United Nations.

It was fitting that the eminent scientific experts of the iron and steel industry there assembled should express their gratitude to the United Nations, from which had emanated the idea of encouraging in Latin America the discussion and analysis of the basic problems of that industry. His country had been very gratified to open its doors to the participants and to co-operate in fulfilling a wish of the United Nations. It was not the first time that the organization had found lasting and loyal friendship in Colombia, as his country had not only formed part of the Preparatory Commission which met in London, but a compatriot had been the Chairman of it.

It was easy to understand that if the organization emerged after the last war with a view to the prevention of future conflicts, on the basis of understanding among the member countries, an undeviating policy was essential. It had been said, however, that the United Nations had become inoperative, rich in abstract policy, but without sufficient coercive measures for implanting and preserving peace. There was not, however, a lack of means but rather of a unanimous will to use them. Colombia believed that mere adherence to formulae had no meaning without practical support when required, and faithful to those principles, it had sent its heroic troops to war when the Security Council issued its verdict on the justice of the South Korean cause. Having been the first State in Latin America to answer the summons, Colombia had authority for stating that collective security would continue to depend on the speed with which countries put into practice the doctrine preached.

In the economic and social field the United Nations also had broad plans requiring the effective concourse of its member nations, either in the way of funds or through facilities offered for the execution of plans. Colombia had been in the forefront in both cases. It had contributed funds according to its obligations, and the iron and steel meeting was a further proof of its collaboration by all available means. On the other hand, twelve distinguished experts from the international Organization were engaged in studying Colombian problems and would provide valuable solutions for increasing agriculture, defending stock-farming, improving public health and the progress of statistical science.

The foreign policy of Colombia, the speaker concluded, was based on its faith in the future of the United Nations, and that policy took its outer form in the most eloquent manner, through positive action.

The people and Government of Colombia hoped that the new year then beginning would find the path to a better world less steep.

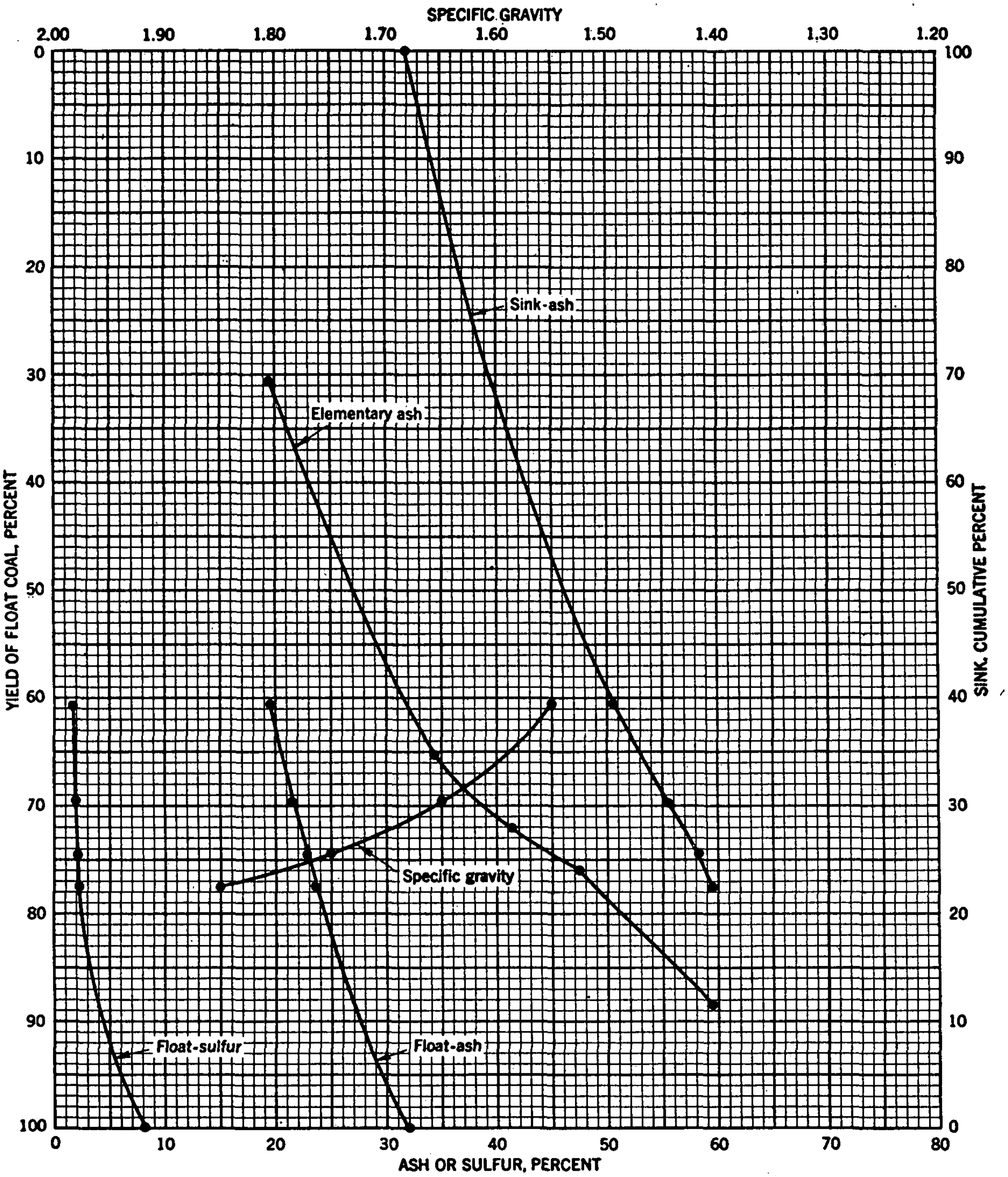
Mr. Etchats, Resident Representative in Colombia of the Technical Assistance Board, replied on behalf of the Secretary-General of the United Nations to Mr Uribe Holguin's speech.

Colombia, in taking the first steps towards creating the United Nations, in London, and in supporting it ever since, had confirmed its long tradition of Latin culture, the tradition of respecting the Roman concept of universality, of culture, science and human ideals.

It was important to stress the role played by economic factors in universal peace. Full employment, economic stability, were determining factors which had profound political and social repercussions and the United Nations had been dedicated to achieving them ever since the Charter was signed in San Francisco. The meeting of the Expert Working Group on Iron and Steel was a fulfilment of the mandate of that Charter, and had been possible owing to the generous hospitality of the Government of Colombia and the joint action of the Economic Commission for Latin America and the Technical Assistance Administration of the United Nations.

He then described some of the activities of ECLA, adding that the presence of European and North American participants showed that it was not only a regional body but a universal one through the fact that it was an integral part of the United Nations.

The first work of the Commission had been to prepare the first economic study of Latin America, which had shown that as modern technique increased productivity, so a labour surplus arose which was not required for the production of raw materials and it was therefore necessary for the countries to turn to industrialization in order to absorb that surplus. Therein lay the economic reason for the meeting. Moreover, that first survey of Latin America had stressed the disturbing fact that importing capacity, per capita, had not increased between the years 1925 and 1940. That fact would naturally have a bearing on decisions taken at the meeting concerning imports of capital goods and investments for internal capital formation.



Specific gravity fractions	Elementary data				Computed cumulative data					
	Weight grams	Percent			Float, cumulative percent			Sink, cumulative percent		
		Weight	Ash	Sulfur	Weight	Ash	Sulfur	Weight	Ash	Sulfur
Float - 1.55		60.6	19.7	1.79	60.6	19.7	1.79	100.0	31.9	8.05
1.55 to 1.65		9.2	34.4	2.70	69.8	21.6	1.91	39.4	50.6	17.69
1.65 to 1.75		4.7	41.4	3.55	74.5	22.9	2.01	30.2	55.5	22.25
1.75 to 1.85		3.0	47.3	5.09	77.5	23.8	2.13	25.5	58.2	25.70
Sink - 1.85		22.5	59.6	28.45	100.0	31.9	8.05	22.5	59.6	28.45

Fig. 10. Washability chart of composite of Barro Branco bed samples crushed to 38 mm. top size. Size 20 – 7 mm. increment.

The Commission's economic studies were already influencing the ideas of the new generation of Latin-American economists; moreover, they had provided a solid basis for United Nations Technical Assistance, which was the second agency which, with the generous help of the Government of Colombia, had sponsored the meeting.

Meetings of experts were one of the ways for providing technical assistance fostered by the United Nations, another being the sending of experts. By the end of 1951, the United Nations and its specialized agencies had sent 797 experts of 61 different nationalities to 75 different countries and territories, the programme being based on the idea that each country could give part of its knowledge to other countries. The third form taken by technical assistance was the granting of scholarships, 845 such scholarships having been granted in 1951 to 79 different countries.

The efficiency of Technical Assistance depended considerably on the collaboration between foreign staff and national staff. For that reason, he wished to express the profound gratitude of all the experts who had come to Colombia for the reception accorded them by the officials and people of Colombia, for the magnificent hospitality accorded to them, and in particular for the work which had been done so successfully by the Organizing Commission.

The CHAIRMAN then invited Mr. URIBE HOLGUIN to take the Chair for the remainder of the meeting.

Brief Account of Iron and Steel Producing Processes Used in Brazil, *presented by the author*

Mr. LANARI prefaced his presentation by explaining that the paper had been submitted as a contribution of the Associaçao Brasileira de Metais, which he represented. The purpose had been to demonstrate in a general manner the characteristics of the raw materials used in the iron and steel industry in Brazil. The metallurgical methods and processes used and the possible evaluation of those processes in Brazil, were described, account being taken of the specific conditions prevailing in that country.

Mr. VARGAS MARTINEZ opened the discussion on the paper, and remarked that one of the conclusions reached by the author in it lay in the limitation of coking coal resources in Brazil for use in the steel industry. That emphasized the importance of one of the most interesting prospects arising out of the meeting, which was the possible interchange of raw materials for the steel industry by the Latin-American countries. There was, for instance, the possibility that Colombia might furnish coking coal from the Cauca Valley or Atlantic coals which, although not coking by themselves, might well become so when blended.

Mr. VILLELA gave some detailed figures on Brazilian iron and steel production and consumption, which indicated that market requirements were considerably ahead of production capacity. By 1960, it was anticipated that the population of Brazil would have reached 66 million inhabitants, whereas steel consumption would have risen to 3,300,000 tons, representing a tenfold increase over about a ten-year period. In 1951 Brazilian production had been around 800,000 tons and consumption approximately 1,200,000 tons. That represented a per capita production of 15 kg. or 33 lb. and a consumption of 23 kg. or 51 lb. per capita per annum.

Demand was really much greater because in 1951 a special survey showed that several hundred thousand tons of steel could not be supplied either by Brazilian plants or through imports; that meant that consumption would be at least 28 to 29 kg. per capita (or 64 lb.) per annum.

The population was growing at a rate of 2.7 to 2.8% per annum and, if the yield of finished products from ingots were considered, the earlier figure of 3,300,000 tons of steel for 1960 would be reached.

Volta Redonda's current output was about half a million tons of steel a year, the programme for 1952 being exactly 478,000 tons.

Volta Redonda's installations were being expanded, and a second blast furnace was being built for a daily capacity of 1,200 metric tons, while the plans for a 1,000,000-ton plant had been completed.

Those figures, which were conservative estimates, served to indicate that the increase in consumption would far exceed existing calculations.

Mr. VUCETICH requested some information concerning the production of charcoal pig iron, in view of the fact that Brazil was one of the largest producers in the world of such pig iron. In particular, he wished to know how the cost of the charcoal pig iron compared with that for blast furnace coke pig iron, and whether the coke pig iron had a lower market value than charcoal pig iron for foundry purposes.

Mr. LANARI replied that in Brazil the consumption of pig iron in steel foundries consisted exclusively of charcoal pig iron. The greatest blast furnace unit belonged to an integrated plant, Volta Redonda, which required no market for its pig iron, as its output of coke pig iron was insufficient for its steel production. He could not provide definite costs for charcoal as against coke pig iron.

Mr. JARAMILLO asked what percentage of steel production in Brazil was based on charcoal.

Mr. LANARI, referring to the last question, said that Brazilian steel production was based on 50% charcoal and 50% coke.

STEEL MAKING AND FINISHING

Comparison of economics of different processes of steelmaking (*concluded*)

28 October 1952—Morning

Chairman:

Danilo VUCETICH, Administrator, Altos Hornos de Corral, Compañía de Acero del Pacífico, Corral, Chile

Contributed Papers:

Selection of Steelmaking Processes and of Locations for Integrated Iron and Steel Works
Wm. A. HAVEN, Vice President, Arthur G. McKee & Company, Cleveland, Ohio, U.S.A.

The Duplex Steelmaking Process at Monterrey
A. GONZÁLEZ BALLESTEROS, Assistant Superintendent, Departamento de Altos Hornos, Compañía Fundidora de Hierro y Acero de Monterrey, S.A., Monterrey, Nueva León, Mexico and
Narciso MORALES, Compañía Fundidora de Hierro y Acero de Monterrey, S.A., Monterrey, Nueva León, Mexico

The Acid Bessemer Process at Huachipato, Chile
Héctor CANGUILHEM, Assistant Superintendent, Metallurgical Department, Compañía de Acero del Pacífico, Huachipato, Chile

Considerations Concerning the Choice of Electrical Equipment for the Iron and Steel Plants of Paz de Río (Colombia) and Chimbote (Peru)
R. L. JUNG, Chief of the Mining and Metallurgical Department, Société ALSTHOM, Paris, France

Summary of Discussion:

Participants: Messrs. BUELL, ALLARD, PALMÉ, MERCIER, CAVANAGH, ARAMBURU, MACEDO SOARES E SILVA, MARTIJENA, YDSTIE, CORTÉS OBREGÓN, ANAWATE, MORALES, CANGUILHEM, BOULGER, DECHERF, JUNG

Selection of Steelmaking Processes and of Locations for Integrated Iron and Steel Works

WM. A. HAVEN

The economics of steelmaking are influenced in large part by the quality and costs of raw materials and of semi-finished materials, particularly pig iron and coke. Unless plentiful supplies and reserves of scrap for steelmaking are available, the principal source of metallics must be pig iron. (I am not one who has been willing to recommend large investments in plants that will be dependent for metal upon sponge iron). The steelmaking process will, therefore, be determined by the quality of the pig iron and the quality of the pig iron, in turn, by the qualities of the iron ores and coal.

The suitability of raw materials for steelmaking by one or another process is established primarily by their phosphorus content, inasmuch as no phosphorus is removed in the blast furnace. If the ores contain less than 0.02% phosphorus, they may, if the fuel and fluxes are not unduly high in this element, be used for the production of so-called "hematite" pig iron which is the only grade of pig iron that can be used in the acid open hearth steelmaking process. Not many countries are blessed with extensive reserves of such ore if the reserves are measured by present-day steelmaking capabilities. It is interesting to note in passing, however, that hematite pig iron was one of the foundations of the British iron and steel industry and that acid open hearth steel was the basis of England's early leadership in producing large tonnages of fine quality steels.

Next in the phosphorus range of ores are those that can be made productive of pig iron that contains less than 0.10% phosphorus. Such ores, unless exceptionally high in iron content, must contain less than 0.05% phosphorus. In North America, ores containing 0.45% phosphorus or less are known as Bessemer grade, and only pig iron made from such ores or a mixture of ores having a maximum of 0.045% phosphorus can be used in the acid Bessemer steelmaking process. Fifty years ago there were extensive reserves of such ores in the States and, at that time, more steel was made by the acid Bessemer (which some historians hold should be called the Kelley process) than by the basic open hearth process.

The great bulk of the iron ores of North America, however, are higher in phosphorus content than 0.05%. Ores of this class as a whole are designated as non-Bessemer to indicate that they cannot be used in acid Bessemer steelmaking, although some of them of intermediate content, say 0.045 to 0.065%, are sometimes mixed with Bessemer ores of very low phosphorus content to make an average mixture of less than 0.045%. For the most part, however, these ores are used to make pig iron of basic open hearth quality, and the abundance of these ores is one of the principal reasons why most of the steel in the United States is made in basic open hearth furnaces.

A fourth group of ores by the phosphorus content classification are those which are productive of pig iron containing not less than 1.75% and not more than 2.00% phosphorus. This is the range in phosphorus in the pig iron ordinarily required for making steel by the Thomas or basic Bessemer process. The limits of phosphorus content are imposed by the ability to remove phosphorus in the Bessemer vessel when lined with basic materials and by the amount of heat generated in burning out the phosphorus. Ores of this grade can also be used, although with increased costs, in the basic open hearth furnaces or by duplexing processes which combine the use of the converter or active mixer with basic open hearth furnaces, preferably tilting furnaces (see Tables 1 and 2).

Table 1

SPECIFICATIONS FOR VARIOUS GRADES OF PIG IRON [a]

Grade	*Per cent silicon*	*Per cent manganese*	*Per cent phosphorus*	*Per cent sulphur*
Low phosphorus	Under 2.00	Under 2.00	Under 0.035	Under 0.035
Acid open hearth	1.00/1.50	0.5/2.50	Under 0.05	Under 0.045
Acid Bessemer [b]	1.00/2.25	0.5/1.00	0.04/0.10	Under 0.045
Basic open hearth	Under 1.50	0.4/2.00	0.11/0.90	Under 0.050
Duplex iron	1.00/1.75	0.4/0.90	0.7/1.50	Under 0.060
Basic Bessemer [c]	0.50/1.00	1.5/2.50	1.9/2.50	Under 0.200
Foundry pig irons	1.00/4.00	0.2/1.50	0.1/2.00	0.04/0.10

[a] Specifications from "The Making, Shaping and Treating of Steel," by the United States Steel Company, 6th edition.

[b] Usually acid Bessemer pig iron contains silicon 1.00%, manganese 0.60%, phosphorus 0.09%, sulfur 0.05%.

[c] In continental practice, basic Bessemer, or "Thomas," pig iron contains silicon 0.05% and under, manganese 0.7%, phosphorus 1.7% or over, sulfur 0.05% or less.

Table 2

ALLOWABLE PERCENTAGES OF PHOSPHORUS [a] WITH VARIOUS PERCENTAGES OF FE IN IRON ORE WHEN PRODUCING DIFFERENT GRADES OF PIG IRON

	Allowable percentages of phosphorus in iron ore mixture					
Per cent Fe in ore	*70*	*65*	*60*	*55*	*50*	*45*
Grade of pig iron						
Low phosphorus	0.020	0.019	0.017	0.016	0.014	0.013
Acid open hearth	.030	.028	.026	.024	.022	.020
Acid Bessemer	.067	.062	.058	.053	.048	.043
Basic open hearth [b]	.176	.163	.151	.138	.126	.113
Basic open hearth [c]	.652	.610	.545	.515	.470	.422
Duplex (3) [c]	1.090	1.020	.940	.864	.785	.705
Basic Bessemer [d]	1.830	1.700	1.570	1.440	1.310	1.180

[a] Varying somewhat with coke rate and analysis, and with amounts and compositions of slagging materials.

[b] Lake ore practice in U.S.A.

[c] Southern ore practice in U.S.A.

[d] With pig iron containing 2.5% phosphorus.

Most readers need not be told that the suitability of pig iron for steelmaking is determined by its content of other elements as well as phosphorus. However, I have wished to emphasize the fact that, unless there are several kinds of ores by the phosphorus classification for him to work with, there is only a limited opportunity for the engineer to pick or choose the steelmaking process. This happens to have been the case in Colombia. With ores of a very low phosphorus content, any method except basic Bessemer can be used, but with very high phosphorus content, the ore becomes unsuited for any steelmaking process and can be used economically only for foundry purposes. The ores of Paz de Río in Colombia contain too high a percentage of phosphorus for making pig iron that can be converted to good steel by either the acid open hearth or acid Bessemer methods. They are not so high in phosphorus, however, as to preclude their conversion by one or another of the basic steelmaking procedures. The problem of selecting a steelmaking process for Belencito, therefore, was one of choosing between basic Bessemer, straight basic open hearth or some combination of duplexing either with an active mixer preceding basic open hearth furnaces or a basic converter followed by basic open hearth or electric furnaces (see Table 3).

Table 3

ANALYSES OF RAW MATERIALS FOR PAZ DE RÍO, COLOMBIA

Raw materials for initial operation at Paz de Río (analyses on dry basis)

Raw material	*% Fe*	*% Mn*	*% P*	*% SiO_2*	*% Al_2O_3*	*% CaO*	*% MgO*	*% S*
La Mesa ore	47.0	0.2	1.13	8.3	6.0	0.7	0.02	0.02
Limestone	.7	—	.06	2.0	1.5	53.0	.7	.05
Coke (estimated)	1.1	—	.07	7.4	2.4	.2	.03	1.00

Iron ores at Paz de Río (analyses on dry basis)

Deposit	*% Fe*	*% Mn*	*% P*	*% SiO_2*	*% Al_2O_3*
La Mesa	44 to 51	0.2 to 0.5	0.8 to 1.5	7 to 10	5 to 8.5
Coloradales	40 to 52	0.2 to 0.3	1.0	6 to 25	6 to 7
El Uvo	43 to 50	0.5 to 1.0	0.8 to 1.25	8 to 14	4 to 7

La Chapa coal at Paz de Río (analyses on dry basis)

Coal and coke	*% volatiles*	*% fixed carbon*	*% ash*	*% Sulfur*
Coal as mined (estimated)	33.4	52.4	14.2	1.4
Coal as washed (estimated)	35.5	56.3	8.2	1.1
Coke (estimated)	1.7	86.0	12.3	1.0

At this point it became necessary to consider the quality of the steel products that were to be made. The initial production schedule for the Colombian mills had been devised to meet the principal requirements of the country, which included rails, light structurals, plates, sheets and tinplate, merchant shapes, bars and wire. It was decided by the Colombians to omit provisions for the manufacture initially of sheets and tinplate, in order to reduce the first cost of the plant. Also, by this decision the need for a large tonnage of steel of basic open hearth quality was eliminated since a satisfactory grade for all of the other products could be made from basic Bessemer steel. There would be a demand for small amounts of high quality steel in semi-finished form, but, to supply this requirement, basic Bessemer steel could be upgraded in electric furnaces (see Table 4).

Table 4

ESTIMATED MARKET FOR IRON AND STEEL PRODUCTS IN COLOMBIA
(estimate of 1950)

Kinds of products	*Metric tons per Year*	
Pig iron		13,400
Cast iron pipe and fittings		32,200
Steel products		
Billets	2,400	
Wire and wire products	32,500	
Merchant mill products	46,200	
Structurals and large bars	9,400	
Rails and accessories	16,500	
Plates	7,700	
Sheets	17,900	
Tin plate	10,600	
Steel pipe ½ in. to 2 in.	4,700	
Total steel products		147,900
TOTAL IRON AND STEEL PRODUCTS		193,500

Selection of the basic Bessemer process for Belencito had the further advantages of being lowest in cost from the standpoint of both capital and operating charges and of yielding as a by-product large quantities of phosphate slag which in Colombia is a fertilizer of great need and value. Also, since the plants were to be fabricated by European firms—German or French—the suppliers would be in a position to provide the highest order of know-how for their operation since most of the steel in their countries is made by the basic Bessemer process.

In the case of Belencito, and in accordance with the usual practices of our company, provisions have been made for modifying or supplementing the initial steelmaking practices when the growth of markets will justify the production of larger tonnages, a greater diversification and products of superior quality. Plans have been made for extending the converter plant buildings to house basic open hearth furnaces so that either some form of duplexing or straight basic open hearth processes can be used. This will permit steels to be made of chemical and physical characteristics that will be more suitable for strip, sheets and other commodities where deep drawing, cold forming or other more severe treatment in later fabrication is involved.

In selecting the steelmaking process for the first large integrated plant that was to be built in Brazil, there was much more latitude of choice for the engineers than in the case of Colombia. The iron ore deposits of Brazil are so extensive that it was possible to find adequate reserves in a wide range of chemical analyses. The ores at Itabira, especially, are so low in phosphorus that they could be used for the production of "hematite" pig iron and could, therefore, be considered as a satisfactory basis for the establishment of a plant in which the steel would be made by the acid Bessemer or even the acid open hearth process. As a matter of interest, the first loans for improving Rio Doce facilities are understood to have been made with the supply of low-phosphorus ore for the British in mind. About 20% of the British iron and steel production at the beginning of the Second World War was still by the acid open hearth process and, during the war, the procurement of low-phosphorus ores had become a serious problem (see Table 5).

Table 5

REPRESENTATIVE ANALYSES OF RAW MATERIALS IN BRAZIL

Raw materials used at Volta Redonda								
	% Fe	*% Mn*	*% P*	*% SiO_2*	*% Al_2O_3*	*% CaO*	*% MgO*	*% S*
Iron ores (dry basis)								
Hematite	66	—	0.10	1.5	1.0	Trace	Trace	0.05
Itabirite—hard	64	0.5	.035	1.5	1.0	Trace	Trace	.03
Itabirite—silicious	49	—	.022	26.0	2.0	—	—	.05
Canga	58	.5	.15	6.0	3.0	—	—	—
Manganese ore (dry basis)								
Manganese	3.0	35.0	—	10.0	5.0	—	—	—
Fluxes (dry basis)								
Limestone	—	—	—	2.2	—	50.7	—	—
Dolomite	—	—	—	4.3	—	19.5	29.0	—
Coking coal from Santa Catarina (dry basis)								
			% volatiles	*% fixed carbon*		*% ash*		*% sulfur*
As mined			—	—		35.9		2.3
Washed			29.7	50.3		20.0		1.6
Open hearth lump ore exported from Itabira (dry basis)								
	% Fe	*% Mn*	*% P*	*% SiO_2*	*% Al_2O_3*	*% CaO*	*% MgO*	*% S*
Hematite—Caue Peak	68.5	0.1	0.03	0.5	0.6	0.05	0.1	0.02

Consequently, in the case of Companhia Siderúrgica Nacional, the selection of the steelmaking method was dictated more by the location that was to be chosen for the steel plant, and by the requirements of the Brazilian markets, than by the quality of the raw materials. The low-phosphorus ores from Itabira were not the most readily available to a plant at Volta Redonda. The logical and most economical source of ore for this locality is in the Lafaiete region on the western slope of the Serra do Espinhaco which is tapped by the Belo Horizonte division of the Central Railroad of Brazil. Volta Redonda is on the Rio-São Paulo division about 25 miles from the junction of this line with the Belo Horizonte division at Barra do Pirai. Direct wide-gauge railway connexion was, therefore, afforded between a plant at Volta Redonda and iron ore (as well as limestone) in the Lafaiete area. At that time the railway and dock facilities of Rio Doce were not sufficient to supply the new large plant and even if they had been, the cost of transportation from Itabira to Volta Redonda involved some 500 miles of rail haul and about 350 miles of vessel transportation as compared with 250 miles of rail haul, only, for the Lafaiete ores (see Map 1).

The ores of the Lafaiete area, as will be seen from the tabulations in this paper, are both chemically and physically of several varieties and it is quite likely that acid Bessemer grade ores could be obtained from selected mines or by selection of low-phosphorus ore in certain individual mining operations. The average of those which the engineers were given to work with are of near Bessemer grade and of course excellently suited for basic open hearth steelmaking. It will be seen that this situation required careful studies of the metallurgical and commercial merits of each in order to choose wisely between these two methods.

It should be kept in mind that the initial cost of an acid Bessemer plant to produce a given tonnage of steel ingots is considerably less than for a basic open hearth plant of the same capacity. Stated conversely, the same amount of money will provide a greater capacity for ingot production if invested in an acid Bessemer plant than in a basic open hearth plant. Furthermore, the operating costs, above raw materials, in producing steel by the acid Bessemer process should be only about one half as much as by the basic open hearth process. These advantages weighed heavily in favour of acid Bessemer, but in the case of Volta Redonda were, nevertheless, offset by other considerations.

In the first place, it was necessary, when Volta Redonda was built, to use a steelmaking process in which the mill scrap could be remelted and converted into saleable products. In 1951, Companhia Siderúrgica Nacional produced 465,000 tons of ingots in making 342,500 tons of finished products, indicating that about 120,000 tons of plant scrap was returned to the furnace for remelting. The production of pig iron was only 345,500 tons, hence it will be evident that in using the basic open hearth process the company was able to convert a very high percentage of the total metallics produced into saleable, highly finished form. With a Bessemer plant, the tonnage of finished rolled products from the same tonnage of pig iron would have been reduced by about 25%, and about 100,000 tons of steel would have had to be disposed of in the form of scrap except for relatively small amounts that could be remelted in the blast furnaces or the converters.

Another condition which made the choice of basic open hearth steelmaking almost mandatory was the market requirements. Other mills in Brazil were producing bars and light structurals but there was no production of heavy structurals and heavy rails, plates, strip, sheets or tinplate. If the latter products were to be made, it was essential that they should be of high quality. Otherwise, it would be difficult to combat foreign competition. Some of them could be made of Bessemer steel, but they would not be up to the standards of the best American or British-made products.

Consequently, it was decided that the initial steelmaking facilities would be basic open hearth furnaces. The advantages and desirability of making acid

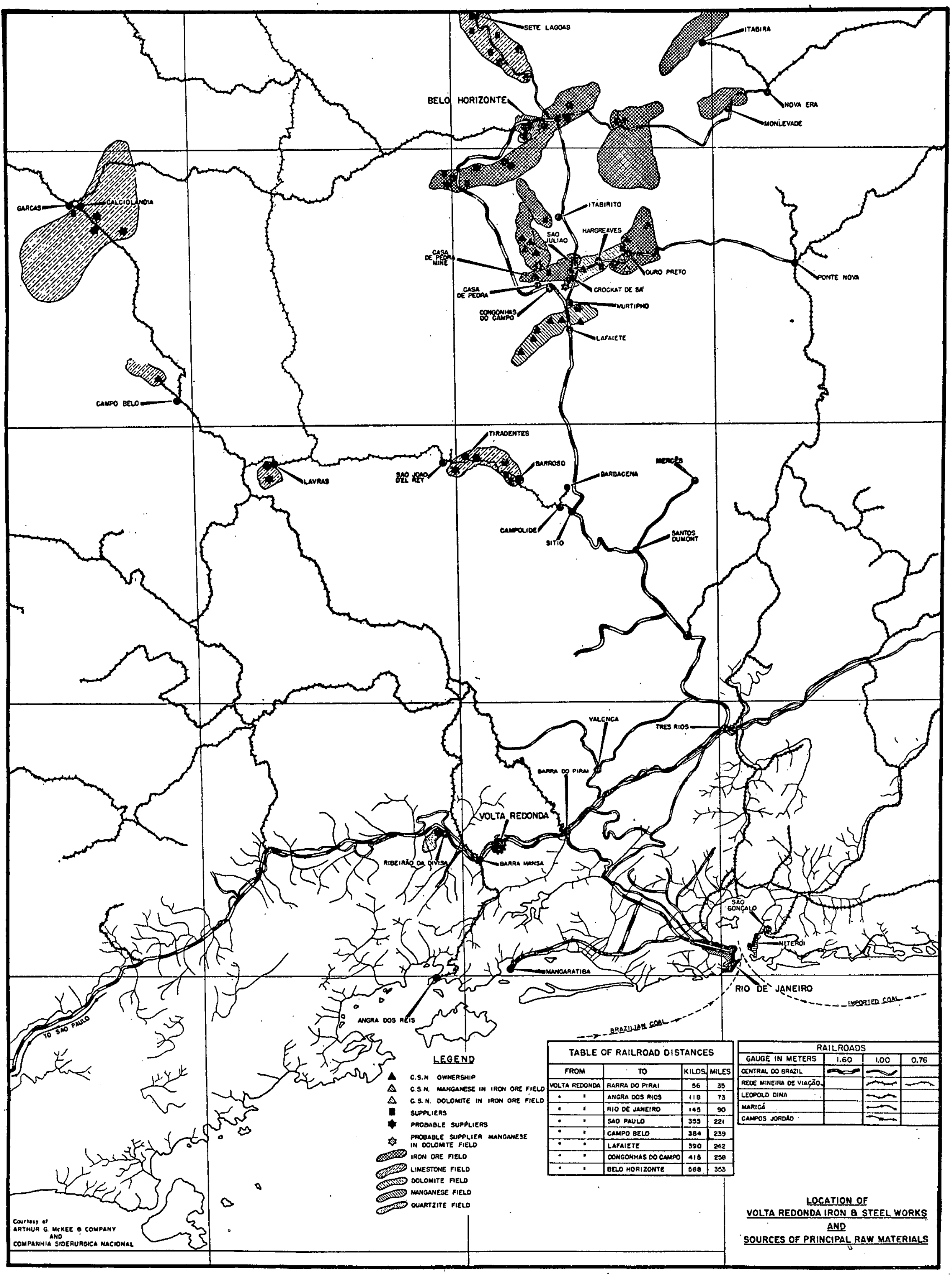


TABLE OF RAILROAD DISTANCES

FROM	TO	KILOS.	MILES
VOLTA REDONDA	BARRA DO PIRAI	56	35
" "	ANGRA DOS RIOS	118	73
" "	RIO DE JANEIRO	145	90
" "	SAO PAULO	355	221
" "	CAMPO BELO	384	239
" "	LAFAIETE	390	242
" "	CONGONHAS DO CAMPO	418	258
" "	BELO HORIZONTE	568	353

LOCATION OF
VOLTA REDONDA IRON & STEEL WORKS
AND
SOURCES OF PRINCIPAL RAW MATERIALS

Map 1.

Bessemer steel, however, were not to be lost sight of and adequate provisions were made for the installation of converters at a future time when it might be economical to manufacture both Bessemer and open hearth grades. The converters would be useful not only for producing Bessemer steel that would find its way into certain of the final products, such as reinforcing bars, tubular goods, railway accessories, agricultural implements, nuts, bolts and screw stock, but could also be used to increase the production of basic open hearth steel and perhaps reduce its cost, by combination of the converter with the open hearth furnaces in some form of duplexing.

For acid Bessemer production together with basic open hearth steel production in the same plant, it is almost essential that there be at least two blast furnaces, since two grades of pig iron would be continuously required. The second blast furnace is now being built at Volta Redonda and this fact, together with the intentions of C.S.N. to increase the capacity of the plant to 1 million tons of ingots yearly, may well result in acid Bessemer steelmaking in Brazil in the near future.

In closing the discussion of the quality of ore in its relation to the economics of steelmaking, it is important to note that ores with a high iron content are not so essential to the success of a plant as commonly believed. Some of the most important of the great pig iron producing districts of the world have had their beginning in, and owe their growth to, deposits of iron-bearing materials which, by the layman, would be considered as being of too low a grade to be smelted. Examples in Europe are the districts in Belgium, Luxembourg and the north of France and the Scunthorpe district in England, the former being dependent upon the Minette ores containing only 35% of iron and the latter upon the local ironstones of Lincolnshire and Northamptonshire which average only 30%. In the United States the Birmingham district uses mostly the red ores of Alabama which contain about 38% of iron. By comparison, some of the rich ores of India, Brazil and Australia contain more than 65% of iron. The requisites of a low-grade ore are that it can be smelted with a normal amount of fuel and that the delivered cost per unit of iron which it contains shall be low as compared with that of the richer grades. Thus, the mineralogical composition and physical characteristics of the ore, the cost of mining it, and the cost of transporting it to the furnaces are factors which sometimes transcend in importance the percentage of iron that it contains in its natural state. The districts previously referred to owe their international importance to the fact that their ores were amenable to smelting and that they occurred near accessible deposits of good coking coal (see Table 6).

Table 6

APPROXIMATE COST OF IRON ORE PER UNIT OF FE—AVERAGE GROSS TON BASIS

Location	*Equivalent average price in US dollars*	*Average per cent Fe (natural)*	*Cost per unit of Fe*
Lower Lakes, U.S.A.	$8.35	51.5	$0.162
Scunthorpe, England	.84	30.0	.028
Tata, India (1946)	.95	60.0	.016
Volta Redonda, Brazil (1949)	5.07	66.0	.077
Belencito, Colombia	2.77	47.0	.059

[a] Self-fluxing mixture.

It is also to be noted that modern methods of concentration and agglomeration have made it possible to convert many low-grade materials into high-grade ones. About 30% of the ores from the Lake Superior district in the United States are now enriched by some method of concentration and it is now certain that the enormous reserves of refractory low-grade magnetites and hematites, known as "taconite", can be converted to a product that can be easily smelted and that will contain 64% or more of iron as compared with 30% in the natural minerals.

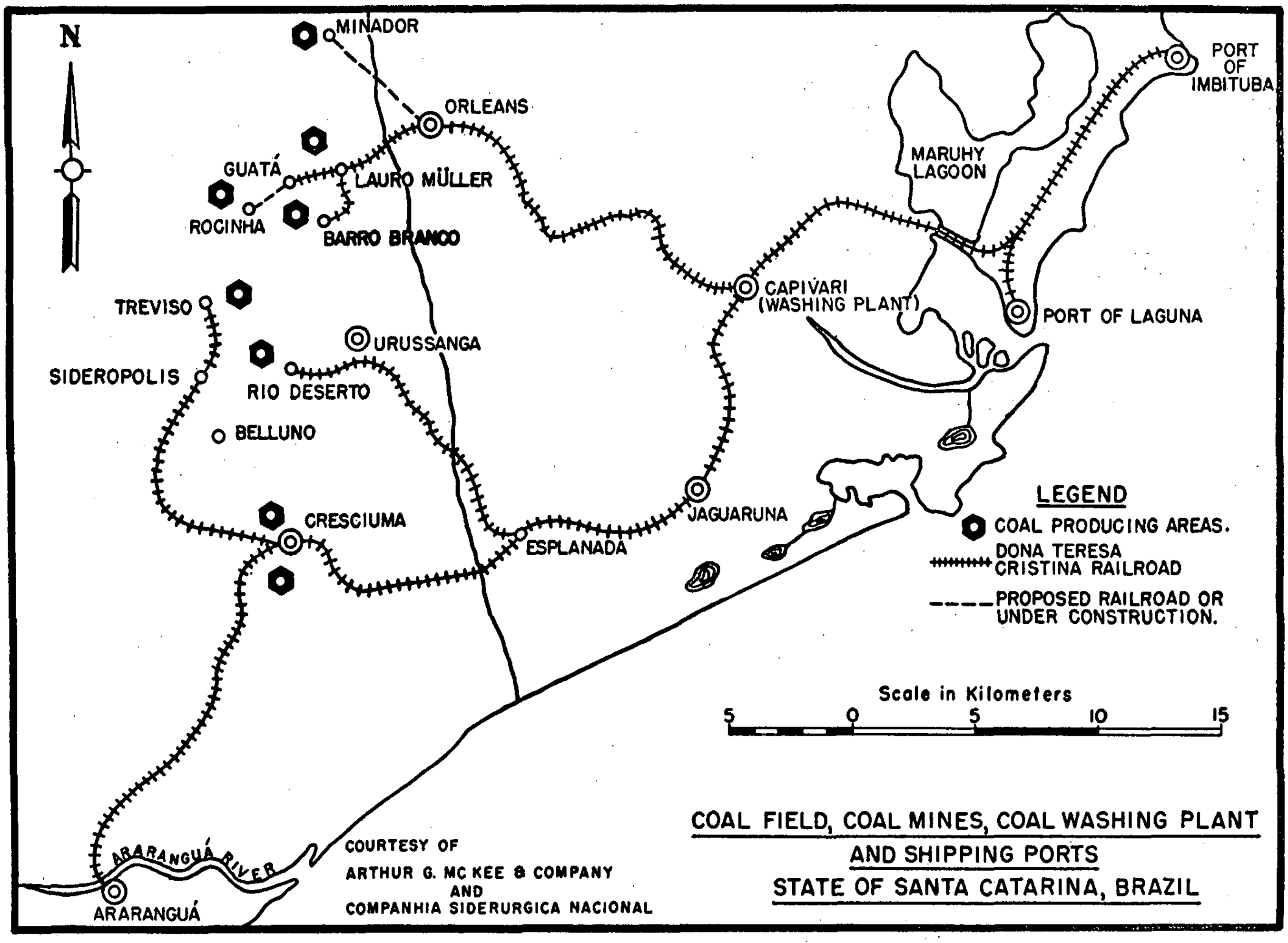
In most Latin-American countries, the need for concentration of ores may be of no immediate importance, although the industry in Colombia could not have been started if dependence upon high-grade iron ores had been necessary. The Paz de Río ores contain only about 45% of iron but, being limonitic, of coarse structure and porous, can be smelted more easily than most hematites and much more so than the denser magnetites. It will be well to note, however, that the rich natural ores of a country, as in Brazil, are not in themselves sufficient assurance that iron and steel can be produced there at costs lower than in other localities that are obliged to use low-grade ores in their natural state or beneficiated.

Lack of coking coal will more often prove to be a greater handicap to producing pig iron cheaply than lack of high-grade iron ore. An excellent example of this is to be found in Norway and Sweden which have not been outstanding in iron and steel production although their iron ores are the richest in all of Europe. In these countries the most persistent efforts have been made to produce pig iron by methods which eliminate or minimize the need for coal and coke, as by sponge iron processes or electric smelting furnaces, but such methods have been found to be economical only for the manufacture of special grades of iron and steel and in comparatively small tonnages.

The situation with respect to coal in Latin-American countries is not ideal in regard either to quantity or quality. However, except in the cases of Argentina and Venezuela, there seems to be enough coking coal in each of the larger countries to support a production of steel that will meet their present domestic requirements.

The feasibility of establishing iron and steel industries in Latin-American countries as well as in other localities wherein superior grades of coking coals have been hard to find has been enhanced by technological advancement in the methods and equipment for the beneficiation of all of the raw materials that are required for production of pig iron, iron ore and limestone as well as coal. Applications of various means of beneficiation, such as concentration, crushing, screening and agglomeration have so improved the smelting characteristics of iron ore that it has now become possible to have satisfactory blast-furnace performance with fuels that previously were held to be unfit for use. Improvements to the methods and equipment for washing the coal, for coking the coals, and for screening and handling the coke after it has been made, have also widened the range of quality in coals that could be used and thus have added hundreds of millions of tons to the reserve of coals that can be rated as being of metallurgical quality.

In Brazil, the national reserves are presently being conserved by the importation of foreign coal. The coke made from domestic coals, while physically of good quality, is much higher in ash than that made from coal that can be obtained abroad and, inasmuch as coal for Volta Redonda has to be transported by sea and railroad whether it be domestic or foreign, the penalty for using foreign coal is not so drastic, at least during peacetime, as might be expected. For emergency periods it has now been established by actual operations that coke of sufficiently good quality can be made from 100% Brazilian coal from the state of Santa Catarina (see Map 2).



Map 2.

The industry in Colombia, on the other hand, seems to be amply fortified in so far as reserves of coal are concerned. While the coal is not of the best chemical and coking qualities, carefully made coking tests indicate that the quality of the coke will be about equal to that of coke made from straight high-volatile coals in the Pittsburgh district of the U.S.A. It is hoped that further exploratory work will uncover beds of low-volatile coking coal, by the use of which, for blending, the physical quality of the coke for the Belencito plant can be considerably improved.

Other than the quality of raw materials and the market for its products, the principal economic factors that are involved in locating a new steel plant and in selecting the steelmaking and steel rolling processes would be the means and costs of transportation between the sources of raw materials and the plant and from the steel plant to the marketing centres, the availability and cost of fuels, water and electric power, and the supply, quality and cost of labour. Before discussing these items, however, some mention, perhaps, should be made of the fact that political, sociological and military considerations, which cannot always be classed as economic ones, sometimes play a predominating part in the selection of the location for a plant, the equipment of the plant and the products that are to be made. One case that can be cited wherein non-economic conditions have had their effect is in connexion with the Brazilian plant described above.

At the time location for the plant of C.S.N. was being discussed, the representatives of each of the three most influential political subdivisions of the country, Rio de Janeiro, São Paulo and Minas Gerais had rather strong reasons for requesting that the plant be located within their own state. The sources of iron ore and limestone were in Minas Gerais and the largest market was in São Paulo. Rio de Janeiro was closest to the sources of coal and represented the second largest market in the country. It was obviously impossible to satisfy all of these requests but the location at Volta Redonda represents in considerable part a good compromise. While the plant is in the state of Rio de Janeiro, it will be noticed from the maps that it is very close to the intersection of the borders of Rio de Janeiro with those of São Paulo and Minas Gerais.

In this case, also, the military viewpoint played its part. There were economic conditions related to transportation costs which favoured the selection of a coastal site, inasmuch as a plant, say near Rio de Janeiro, would be, in effect, about halfway between the sources of iron ore and coal. Keeping in mind, however, that the decisions were made just following the early military successes of Germany in the Second World War, it will be understood why some of the military people were desirous of having a location selected that would be less vulnerable to damage by naval and aeroplane action.

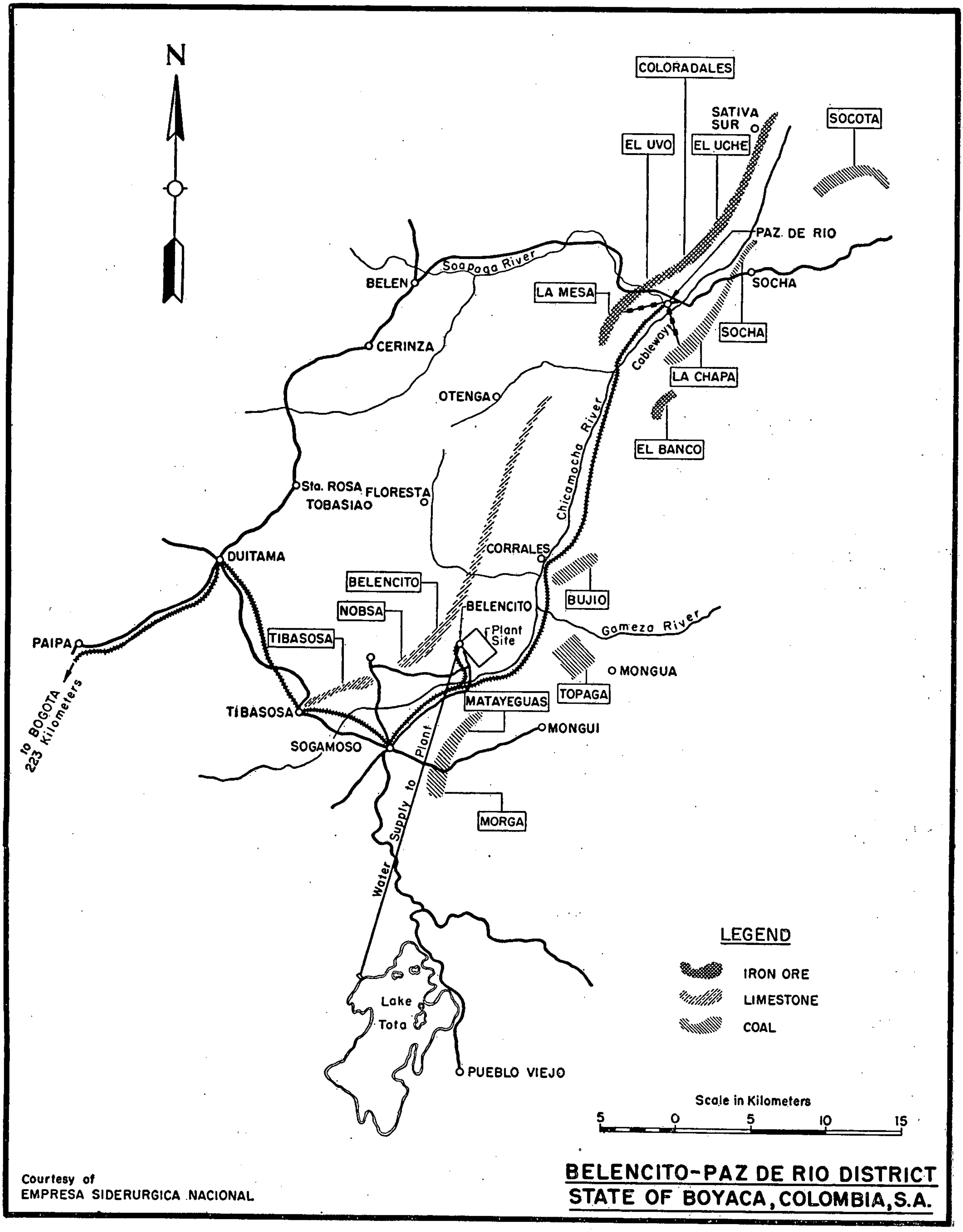
The principal reasons, however, for selecting Volta Redonda as the location for the first large integrated plant in Brazil were more truly of an economic nature. First of all, the transportation of very large tonnages of raw materials had to be considered. The coal deposits of Brazil are in the state of Santa Catarina, more than 1,000 miles from the ore fields, and in order to make use of domestic fuel, it would, therefore, be necessary either to locate the plant in Santa Catarina and haul the iron ore to this locality, to haul the coal to Minas Gerais, or, to select a location that would be a logical meeting place of ore and coal. The site also had to be within easy reach of the two principal domestic markets, namely, the cities of São Paulo and Rio de Janeiro. Examination of the map of Brazil (Map 2) will reveal that the site of Volta Redonda offers a solution of this problem that is also a reasonable compromise—the site being within 250 miles of the ore fields with an existing wide-gauge railway connexion thereto, and only 100 miles from the seaport of Rio where facilities for the transferring of coal from vessels to railway cars were also already in existence and available for use.

A comparison of transportation costs to the various proposed sites that was used to help in determining the optimum location of the plant from this point of view had to be based upon freight rates for ocean and railway transportation that were then in effect. If the same comparisons were made as of this date, some ten years later, the results might be different because railway transportation rates on the Central Railroad of Brazil have since changed from a low figure to a high one. In 1941 it appeared that the cost of hauling one ton of ore from Lafaiete to Volta Redonda would be approximately 20 cruzeiros, whereas the cost in 1949 had increased to about 50 cruzeiros per ton. A somewhat similar increase has occurred in the cost of transporting coal from Rio de Janeiro to Volta Redonda. Taking these changed conditions into consideration, the economic advantages of locating the plant on the seacoast would be more impressive. The savings in total railway transportation costs might be greater than the increased costs of railway freight on finished products inasmuch as about four tons of raw materials are required for each ton of finished products. It is also to be noted that since the plant was built, modern highways have been completed between Volta Redonda, Rio de Janeiro and São Paulo, as a result of which a large percentage of the transportation of finished products is now by trucks.

It might also be said that in order to meet the very extensive requirements for labour both in the construction of the plant and in its subsequent operations, it would have been preferable to build the plant in one of the large centres of population. Such a course would have saved great sums of money in the construction of housing and other requisite community facilities such as water supply, streets, sewers, hospitals, churches, and the like. As a matter of fact, building of the plant at Volta Redonda involved the creation of an entirely new municipality, the cost of which is sometimes mistakenly considered as an essential part of the cost of the steel plant itself inasmuch as these costs have been incorporated in the financial records of the company. If the project had been privately financed, and if the objective of the Commission had been limited to the maximum of profitability, the decision to build a new city might have been difficult to justify. Taking into consideration, however, that the undertaking was a quasi-Government one and the fact that the city of Volta Redonda has been planned and built with the purpose in mind of establishing a model for an industrial community, it will be readily agreed by anyone who has visited the site that the new municipality, apart from its substantial contribution to the success of the venture is, in itself, a national investment of tremendous sociological value. Over the long term, moreover, it is likely that the investment in the munipality will be self-supporting and profitable.

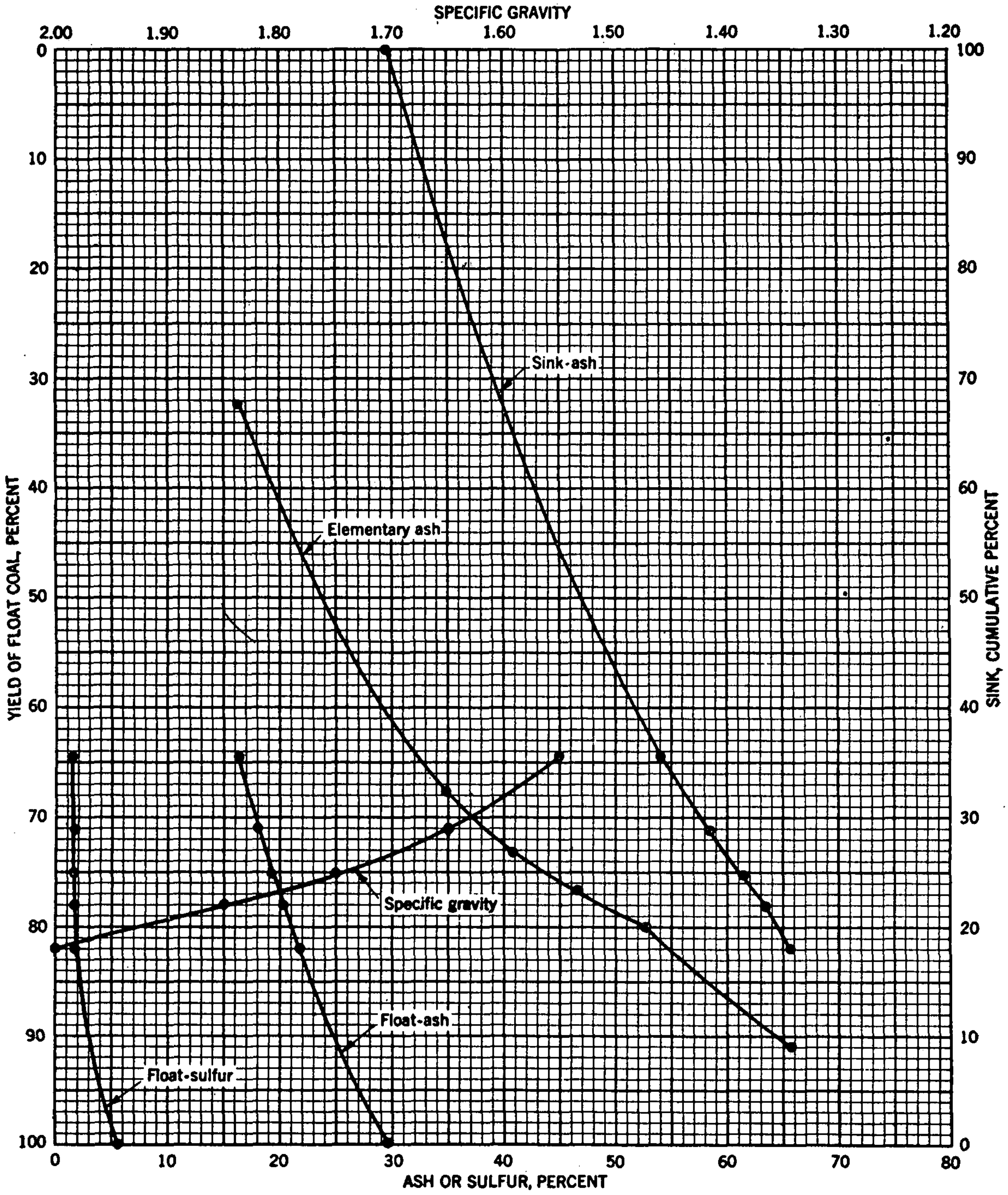
Turning now again to consideration of the integrated steel works in Colombia, it may be said that the selection of the location of the site for this plant was a far less difficult problem than in the case of Brazil. In the first place, many years of field work had failed to find any other location in Colombia at which the various raw materials could be assembled at reasonable cost. As a matter of fact, no large deposits of iron ore had yet been found other than those at Paz de Río. A particularly conscientious effort had been made to find ores more suitable for use in basic open hearth steelmaking but the deposits of those so far discovered were widely scattered and none of them of any great magnitude. Somewhat the same conditions seemed to exist with regard to coking coal. There are coal bodies in the western states, but few of these are known to be of coking quality. It is reported that there are extensive deposits of good coal in the extreme northeastern part of the country, but these are in remote regions where there is presently no means of access. It is believed that these may be of value in future years, either as a source of coal for export to other Latin-American countries or conceivably as a basis for a coastal iron and steel industry that could draw upon Venezuela, if necessary, for its supply of iron ore. The coastal areas of Colombia, so far as then explored, were quite devoid of iron ore deposits (see Map 3).

It has been advocated by some investigators, particularly the World Bank Commission, that the only location in Colombia where an iron and steel plant could be successful would be on the seaboard. There are many reasons why well-informed engineers cannot agree with this point of view, but it will probably suffice to mention, first, that such a plant could not be fully integrated or would have to depend upon the importation of satisfactory raw materials and, secondly, that such a location would be handicapped by the great difficulty and high cost of transporting its finished products to the industrial areas of the country adjacent to Medellín and Bogotá. It is not only the lack of direct rail and water routes to the interior, but the great elevation of these industrial



Courtesy of
EMPRESA SIDERURGICA NACIONAL

Map 3.



Specific gravity fractions	Elementary data				Computed cumulative data					
	Weight grams	Percent			Float, cumulative percent			Sink, cumulative percent		
		Weight	Ash	Sulfur	Weight	Ash	Sulfur	Weight	Ash	Sulfur
Float - 1.55		64.4	16.2	1.51	64.4	16.2	1.51	100.0	29.7	5.48
1.55 to 1.65		6.7	34.9	1.97	71.1	18.0	1.55	35.6	54.0	12.66
1.65 to 1.75		4.1	40.8	2.49	75.2	19.2	1.60	28.9	58.5	15.14
1.75 to 1.85		2.8	46.7	3.23	78.0	20.2	1.66	24.8	61.4	17.24
1.85 to 2.00		4.0	52.8	4.35	82.0	21.8	1.79	22.0	63.3	19.02
Sink - 2.00		18.0	65.6	22.28	100.0	29.7	5.48	18.0	65.6	22.28

Fig. 11. Washability chart of composite of Barro Branco bed samples crushed to 38 mm. top size. Size 7 – 1.5 mm. increment.

regions above the seacoast that makes such a proposition unfeasible. The present cost of moving products to the industrial areas from the coastal ports of Barranquilla and Buenaventura amounts to about $35 per ton. A plant located on the seaboard would be in competition with large plants in foreign countries where steel could be made at much lower costs. On the other hand, for a plant located in the interior, this high freight rate from the seaboard constitutes a protection created by the geography of the country. This protection is of such magnitude that it will probably preclude the need of import tariffs which oftentimes have had to be resorted to in order to support an infant industry.

The principal requisites, then, for the location of the plant in Colombia were that it should be close to and have reasonably inexpensive access to native raw materials and, secondly, that it should have means readily available for shipping its products to the principal industrial centers. In both of these respects, the location at Belencito, in the province of Boyaca, was, fortunately, satisfactory.

Those who attended the Bogotá meeting have had an opportunity to see for themselves the unusual geographical and geological features that characterize the Paz de Río region and combine to make it a "natural" location for an integrated steel plant, a location wherein there occur extensive deposits of iron ore, coking coal and limestone within a few miles of each other. On a small scale, there are examples of such occurrences in various parts of the world, notably in England, but there are very few in which large reserves of all the principal raw materials are to be found in such close proximity.

At Paz de Río the ore is a ferruginous cap for the mountains on one side of the narrow valley of the Chicamocha River, while the coal outcrops in several distinct seams from the slope of the mountains on the opposite side of the valley. The valley itself provides a fairly level route from Paz de Río to the site of the plant at Belencito where limestone is being mined from an adjacent hillside. The movement of ore and coal from mines to plant site is toward the principal markets of Colombia, and transportation of steel products from Belencito to Bogotá and to certain ports on the Magdalena River is provided by existing railways or highways (see Map 4).

In passing, it is interesting to note that, while the existence of iron ore at Paz de Río had been known for some years, there was no knowledge of the nearby La Chapa seams until 1947. The discovery of the latter was the result of geological studies and exploratory field work on the part of Dr. Benjamin Alvarado, Dr. Vicente Suarez and their associates. Following the clues provided by known outcrops and small mines in this coal bearing area, it was reasoned that there should be coal in this formation at Paz de Río and persistent trenching on the mountain sides finally proved this is to be the case. That the seams should be coking coal, however, was purely fortuitous.

The economic significance of finding the principal raw materials within such easy reach of a plant is obvious if it is kept in mind that, in most parts of the world, the cost of transporting these materials to the smelting furnaces is a major portion of the total cost of producing pig iron and steel. It is estimated that the cost of assembling these materials at Belencito will be $4.30 per ton of pig iron. At Pittsburgh, Pennsylvania, which is close only to the coal fields, the comparable cost is about $12.10 per ton. The percentage of total costs for such assemblage at Paz de Río is thus only 20% while at Pittsburgh, it is 40%. At Volta Redonda, which is between ore and coal, the present assembling cost is about $24 per ton and the percentage of total cost is understood to be about 60%. Considering that both Pittsburgh and Volta Redonda have proved to be satisfactory locations for the manufacture of steel products, it may readily be understood that the plant at Belencito is exceptionally well favoured, almost uniquely so, in this respect (see Table 7).

Table 7

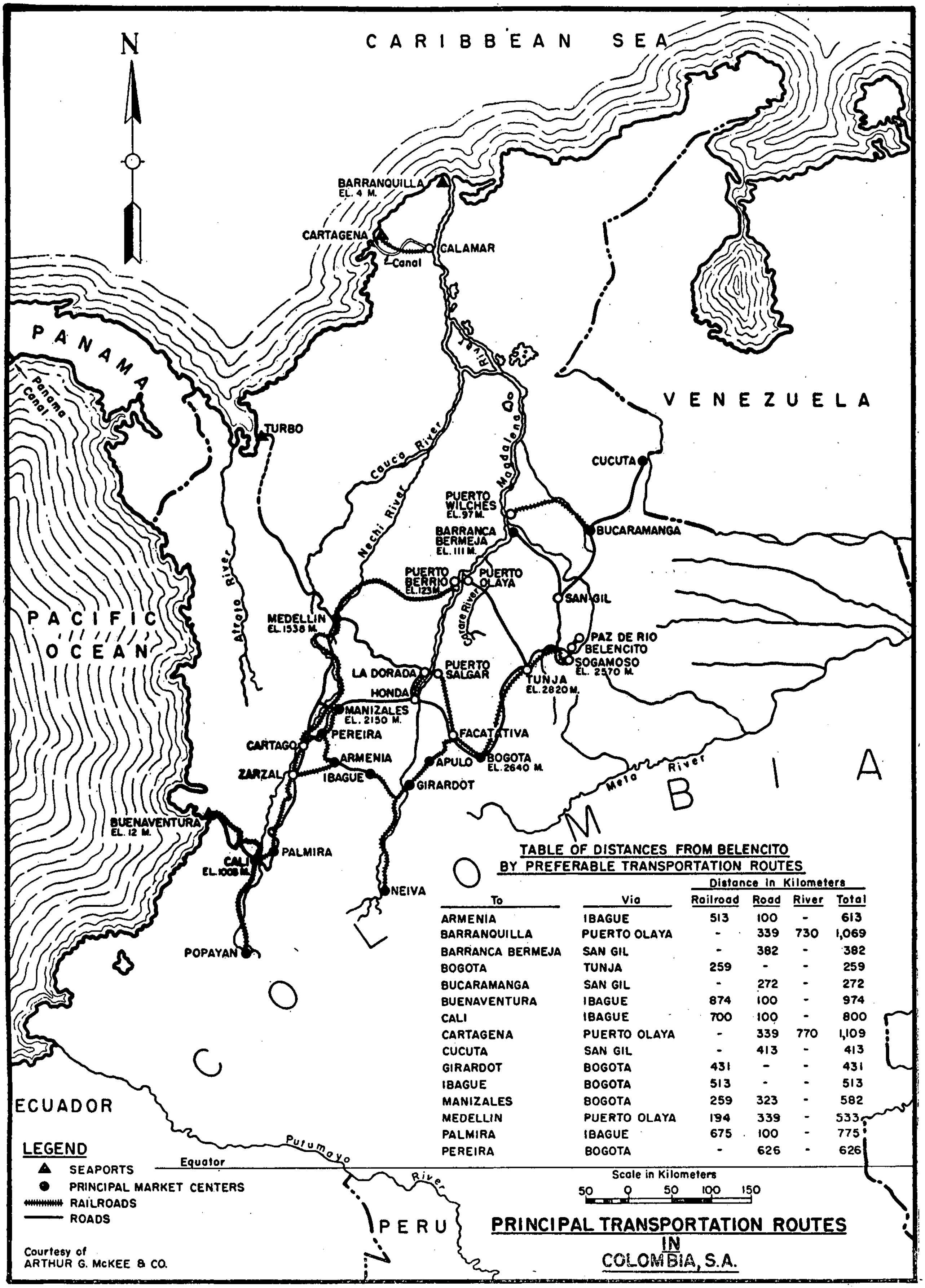
ESTIMATED ASSEMBLY COSTS OF RAW MATERIALS IN PRODUCTION OF PIG IRON

Location	*Cost of raw materials per metric ton of pig iron*	*Estimated percentage of total pig iron cost*
Pittsburgh, Pa., U.S.A.	$12.10	40.0
Volta Redonda, Brazil	24.20	61.0
Belencito, Colombia	4.30	20.0

So far in this paper the importance of three factors in the selection of steel plant locations has been stressed, namely, raw materials, markets and transportation. Others are fuels and labour, but it may be assumed that the fuel for the generation of power in various forms is provided as a by-product in a well balanced integrated plant. Selection of locations for non-integrated plants and of steelmaking processes for such plants is a problem to which the general formulae of this paper do not always or entirely apply, but detailed examination of the economics for non-integrated plants has been foregone in this paper because of their lesser importance on the international scene.

When speaking of labour costs, it is generally with reference to compensation of personnel required for the operation of the steel plant itself, including management and supervision, labour in operations and maintenance and in providing services within the plant, clerical and accounting, laboratory and other technical staff, but not including labour in connexion with the mining or transportation of materials. These latter are customarily incorporated in the cost of raw materials. The cost of labour is determined mainly by three considerations, viz: the modernity of equipment, the hourly or monthly wage rates and the proficiency of the individual workers.

With reference to the plants recently built or planned for in countries where wage rates are relatively low, it is frequently held that manual operations should be employed in so far as it is possible to meet the production requirements by manual procedures. The experience of the author's firm, however, is mostly to the contrary. Lack of experienced and technically skilled workers can best be offset by using methods and processes in which the operations requiring skill are entrusted to machinery to the greatest possible extent. Skill in maintenance, mechanical and electrical work seems to be more easily found or developed than that required for complicated manual operations such, for example, as in sheet or tinplate rolling. Also, the greater productive capacity of modern metallurgical units or rolling mills reduces to a minimum the number of skilled operators that must be trained to secure a given tonnage output.



PRINCIPAL TRANSPORTATION ROUTES IN COLOMBIA, S.A.

TABLE OF DISTANCES FROM BELENCITO BY PREFERABLE TRANSPORTATION ROUTES

To	Via	Distance in Kilometers			
		Railroad	Road	River	Total
ARMENIA	IBAGUE	513	100	-	613
BARRANQUILLA	PUERTO OLAYA	-	339	730	1,069
BARRANCA BERMEJA	SAN GIL	-	382	-	382
BOGOTA	TUNJA	259	-	-	259
BUCARAMANGA	SAN GIL	-	272	-	272
BUENAVENTURA	IBAGUE	874	100	-	974
CALI	IBAGUE	700	100	-	800
CARTAGENA	PUERTO OLAYA	-	339	770	1,109
CUCUTA	SAN GIL	-	413	-	413
GIRARDOT	BOGOTA	431	-	-	431
IBAGUE	BOGOTA	513	-	-	513
MANIZALES	BOGOTA	259	323	-	582
MEDELLIN	PUERTO OLAYA	194	339	-	533
PALMIRA	IBAGUE	675	100	-	775
PEREIRA	BOGOTA	-	626	-	626

Map 4.

Blast furnaces are now being built that produce 1,500 tons of pig iron daily whereas the largest hand-charged and hand-cast furnace can produce only about one fourth of this tonnage and about twice as many workers are required to operate the smaller, old-type furnace as the larger and more modern one. The personnel that have to be employed and trained for an output of 1,500 tons are, therefore, about 8 to 1. This would be more than sufficient to counterbalance any advantage that might result if the capital costs of a group of furnaces using a maximum of manually operated equipment proved to be less than that of a single modern one.

The extent to which fully mechanized, automatic and semi-automatic equipment can be used in countries like the Latin-American ones is, consequently, determined principally by the quantities of iron and steel products that can be used domestically or exported profitably. The market requirements in Brazil, for example, were favourable not only for the installation of large modern furnaces but, also, for a large blooming mill and a modern four-high continuous rolling mill for the production of strip, sheets and tinplate. The full capacity of the rolling mills was not immediately needed, but there was a sufficient demand for flat-rolled products to make its operations profitable and in the few years that have passed since the plant was started, the demand for these products has already overtaken its capacity to produce them.

When contemplating the construction of a steel plant that will produce basic materials for the first time in a country that has yet to be industrialized, it should be realized that the growth of fabricating industries that use these basic materials will be immensely stimulated by the ability of the consumer to secure his requirements from domestic sources and without the need for foreign currency. In establishing the initial capacity for the steel plant and in selecting the equipment that is to be used therein, the future as well as the immediate markets need to be given very careful consideration. To provide the variety of finished products that is needed, it is generally necessary to specify larger mills and a greater number of mills than would be justified merely by the total tonnage of steel that can be marketed at the time that the initial installations are completed. As a result, the cost of plant per ton of annual output is generally high for such pioneer plants. Subsequent rapidly increasing demand, however, may be depended upon to reduce the capital charges as the mills are brought up to capacity operations.

Herein, however, the main concern is with the relationship between the selection of equipment and the supply of labour. In this connexion, it is to be noted that our efforts to minimize the problem of employing and training labour in countries other than the United States has been somewhat thwarted by sociological and political conditions. The tendency toward patronage, paternalism and leniency on the part of all present-day governments has resulted in statute restrictions upon control of their working forces by management, has limited the productivity of individual workers, and made it mandatory upon the management of plants in foreign countries to augment their forces with large numbers of superfluous personnel.

Comparing the over-all manpower requirements for the industry in foreign countries with those in the United States, it is found that in England the ratio is 2 to 1, in India 3.5 to 1 and in Brazil 1.4 to 1. It will thus be seen that much of the benefit to be gained from modernization of plants and equipment has not yet been realized. Because of the unprecedented world-wide demand for iron and steel products during the past ten years, extravagance in the use of labour in these countries has not seriously impaired the earnings and financial standing of the producing countries, but it does constitute a potential threat to their competitive position, their healthy growth and their development in the future. Latin-American and other countries where the industry is in its infancy and where labour rates are presently much lower than in the United States have a unique opportunity to establish their plants on a sound basis by combining the advantages of modern equipment with that of relatively inexpensive labour. It is highly important to the future industrial growth, the economics and the sociological health of these countries that these advantages be recognized and used, rather than dissipated by overloading the payrolls of the new plants with supervisors, labourers, clerks, accountants, police or any others beyond the numbers that can be usefully employed.

However, the production of iron and steel within any country is the most requisite of all the so-called basic industries as a solid and permanent foundation for the development of manufacturing, transportation facilities and agriculture and can be justified at almost any cost. There can be no doubt that the building of steel plants which has featured progress in Latin-American countries during the past decade has contributed substantially to the over-all wealth and welfare of the people. As North Americans, we have been particularly pleased to have played a part in these enterprises because they are of a nature that helps our neighbours to help themselves.

The Duplex Steelmaking Process at Monterrey

ALFREDO GONZÁLEZ BALLESTEROS and NARCISO MORALES

In Mexico, as in any other country, the best method for producing steel is that which, by combining its own resources and the techniques best suited to its raw materials, permits the transformation of iron ore into steel, at a cost appropriate to the economic situation, and in quantities compatible with a widely varying demand for drawn and rolled sections.

As Mexico does not have sufficient scrap, it is necessary and economically desirable to use pig iron produced from ores mainly derived from the Cerro de Mercado, in the state of Durango. This is processed into steel at a higher pig iron-scrap ratio than usual (70% molten iron, 30% scrap); frequently even larger quantities of iron in relation to scrap are used, even though this means charges for the open hearth furnaces with a high phosphorus content (0.8%).

That is why in the specific case of the "Fundidora de Monterrey", the duplex method is preferred, as it permits the utilization of molten pig iron in Bessemer converters for the partial elimination of secondary impurities such as silicon and manganese; and with the same converters, in an operation not exceeding 15 minutes, to control the carbon down to a very low content (1%) which permits the subsequent removal of the sulfur and phosphorus.

The elimination of sulfur and phosphorus, requiring the use of basic-lined furnaces and basic-type fluxes, is carried out in the Siemens Martin open hearth furnaces, which receive the molten iron from the Bessemers.

The plant has the following equipment:

2 Blast furnaces with a total capacity of 900 tons per day; blast furnace No. 2 produces 550 tons, usually hot, for transforming into steel; the remaining 350 tons per day are generally low-phosphorus foundry pig iron, for billets, casting of railway wheels and occasionally ferromanganese and basic iron.
2 Mixers (thermo-recipients) for the storage and standardization of the blast furnace burden; one has a capacity of 300 tons, and the other of 150.
2 Bessemer converters for the initial refining of the molten metal, each with 17 tons capacity per "blow", equipped with their corresponding blowing apparatus.
1 Set of travelling cranes, drying ovens, ladles, transport cars for service in the Bessemer and mixing sections, and appropriate buildings.
5 Open hearth furnaces with a present capacity of 65 tons each per charge (but with their hearths and regenerator chambers prepared for future yields of up to 90 tons per charge by means of new crane equipment) for the final purification of the iron and its definite conversion into steel.
1 Yard with equipment of floor and overhead cranes, ladles for the charge of blown molten iron and extraction of steel, ladles for disposing of the residual slag, mechanical tipping cars, trains of billet cars and sets of box cars for handling scrap, fluxes and auxiliary material, buildings, etc.
1 Yard with cranes for handling and storing the billets and a "soaking pit" equipment.
1 Yard with its set of break-up cranes, loading cranes and mechanical shears, devices for crushing, classification, storage and loading of the materials necessary for conversion of molten pig iron into steel; limestone, quicklime, dolomite, fluorspar, iron ores, basic materials, bricks and alloys.

MIXERS AND BESSEMER CONVERTERS

These form the fundamental equipment for the "duplex" process which the "Fundidora de Monterrey" uses in its production of steel, and they operate as follows:

The pig iron from the blast furnace is stored in the mixers, both to have it at hand as required for the Bessemers and to level up the slightly varying analysis resulting from the periodic charges of the blast furnaces. In the mixers the iron remains fluid almost indefinitely, its loss by radiation being compensated by the combustion of natural gas injected through the tap hole. The mixers have tilting mechanisms which facilitate the charging and tapping of the molten pig iron.

The Bessemers at the Monterrey plant, which purify the pig iron by oxidizing the silicon, manganese and part of the carbon, require approximately 750 cu. m. of air per minute at 1.5 kg. pressure per sq. cm., which is injected through the bottom by tuyeres, to supply the necessary oxygen to the iron charge for the resulting reactions in its partial refining.

As the air is injected at the bottom and it is this element which practically serves as a bed for the molten metal charge, the Bessemer furnaces are necessarily of the tilting type, either for use in their horizontal position when charging and discharging the melt, or in the different positions (principally vertical) necessary for the blowing operation.

With the mixers and Bessemers available it is possible to "blow" whenever necessary. In the operations of the "Fundidora", these "blows" last approximately 12 minutes and are of 17 tons each, resulting in the silicon content being reduced from 1.50% to practically nil; the manganese content drops from 0.30% to 0.05% whereas the carbon in the pig iron is actually reduced from 4.25% to 1%. As these reactions generate heat any undue rise in temperature is controlled by additions of scrap or by steam injection.

The skill of the experienced operators is such that they can control the blowing process by observing flame colouration.

OPEN HEARTH FURNACES

The plant has five open hearth furnaces and, as indicated in the list of equipment, these furnaces are complemented by the machinery and accessories required for operating them.

Approximately 50 tons of blown molten pig iron are received from the Bessemer converters, with high phosphorus and carbon contents, wherewith to charge the steel furnace; the charge is made up with the scrap, iron ore and fluxes to average about 65 tons. The final figures of the impurities do not exceed 0.05% sulfur, 0.040% phosphorus, with the carbon appropriate to the type of steel desired.

Essentially, the obtaining of a tap according to specifications is controlled from the charge itself, which is composed of alternate layers of five tons of light scrap, four tons of limestone, five tons of scrap plus two tons of ore and three tons of limestone, with a final layer of 10 tons of scrap. Melting is initiated by the combustion of natural gas of 10,150 calories per cu. m. and crude petroleum of 9,150 calories per kg.; preference is given to the use of gas to prevent too much sulfur being absorbed from the crude oil when the melting of the charge begins.

Once the preceding charge is in process of fusion, 50 tons of blown molten iron are added so that elements such as the calcium from the limestone and oxygen from the iron ore, can enter into contact with the impurities in the iron, assisted by the boiling which takes place in the charge through the escape of carbon monoxide and dioxide gases. This results in the well-known violent chemical reactions which, increased by the high temperatures prevailing in the bath, also tend to produce a homogeneous mixture, with the oxidation of impurities, mainly phosphorus and carbon.

When special specifications so require, recarburation with coke fines is effected during the blow within the forehearth receiving the melt from the furnaces, if it is a question only of a carbon alloy. To meet specifications for commercial (normal) steels, approximately 150 kg. of ferrosilicon and 600 kg. of ferromanganese are added in the furnace for each 65-ton tap. For killed steels, these additions are slightly higher.

The furnace is tapped when the laboratory tests, which continue throughout the refining process, conform to the specifications required by Mexican standards of the Secretaría de Economía, Dirección General de Normas—equivalent to A.S.T.M., A.R.E.A. and the like. The temperature is also checked to see that it is right for tapping, in order to avoid the difficulties which arise from cold or over-heated steel both in the ladle and in the ingot moulds.

This process produces large quantities of slag, and constant vigilance is therefore required in removing it and making periodic analyses, for proper control of the final quality of the steel.

By weight, in relation to steel produced, the slag represents about 20%, composed of 35% of CaO, 9% of SiO_2m, 2% of Fe_2O_3 and 16% of FeO.

At present 26 to 27 ingots of 2,500 kg. each are obtained from each tap, each measuring 21 in. x 19 in. x 72 in. when poured into the ingot moulds (special moulds of low phosphorus iron) for teeming, and after solidifying these are passed to the soaking pit and prepared for rolling.

In addition to the general description above, the Monterrey plant has important auxiliary factors for steel production, which include:

(*a*) The obtaining of circulating scrap (re-utilization of broken pieces, steel scrap derived from them and especially mechanical scrap recovery from the slag heaps);

(*b*) Its own supply of ingot moulds, ladle-cars for molten iron and slag, ladles for collecting slag, cars and trucks for handling the materials for the charge, and similar equipment;

(*c*) Facilities for the manufacture of sundry spare parts for the mechanical maintenance of the plant, among which are: re-heating furnaces, rolls, rolling mills, and other items of heavy machinery all of which contribute to the efficiency of the processes in general.

The Acid Bessemer Process at Huachipato, Chile

HÉCTOR CANGUILHEM

Huachipato is the only Chilean integrated steel plant. Production covers, therefore, a large range of products: pig iron, round sections, light structural shapes, black and galvanized sheets, tin plate, skelp, wire rod and welded steel pipe.

Information contained in this paper is based on consideration of about 1,000 converter heats. They include those of the initial operations at the plant, thereby reducing over-all efficiency, as compared with recent practice.

RAW MATERIALS AND CHEMICAL SPECIFICATIONS

Ore is supplied by Bethlehem Chile Iron Mines Company from the El Tofo property. Chemical composition varies within the following limits:

	Per cent
Fe	54.4 –58.5
P	0.035– 0.085
SiO_2	7.1 –12.8
Al_2O_3	1.3 – 2.9
CaO	0.5 – 3.9
MgO	0.5 – 2.0
Mn	0.10 – 0.15
S	0.03 – 0.06
Ti	0.15 – 0.50
V	0.13 – 0.25

The Bessemer steel is duplexed in the open hearth plant when the P is over 0.11% in the blast furnace iron. Titanium varies from 0.05–0.14% and V from 0.25 to 0.40%.

Only semi-killed Bessemer steel is produced at Huachipato.

Table 1 shows the different grades made, the grade numbers corresponding to the internal standard practice at the plant.

Table 1

CHEMICAL SPECIFICATIONS OF HUACHIPATO BESSEMER STEEL

Grade	*Carbon*	*Manganese*	*Phosphorus*	*Sulfur*	*Silicon*
151	.05–.09	.20–.70	.099 max.	.06 max.	.04–.07
151-A	.05–.07	.20–.50	.100–.119	.04 max.	.04–.07
152	.08–.11	.20–.70	.099 max.	.055 max.	.04–.07
	.12–.14	.20–.50	.099 max.	.055 max.	
152-A	.08–.12	.20–.50	.100–.119	.04 max.	.04–.07
153	.12–.14	.51–.70	.099 max.	.055 max.	.04–.07
	.15–.17	.20–.50	.099 max.	.055 max.	

REFRACTORY PRACTICE

The converter is of the eccentric type, with a shell diameter of 10 ft. 11 in., and has a nominal capacity of 15 tons.

The lining is Chilean made. This silica brick has the following composition limits:

	Percent
SiO_2	94.6 –96.0
Al_2O_3	.9 – 1.4
Fe_2O_3	.7 – 1.4
TiO_2	.10– .20
CaO	1.6 – 2.3
MgO	.05– .10

This curve holds four hot ladles charged with 45/66m. tons.

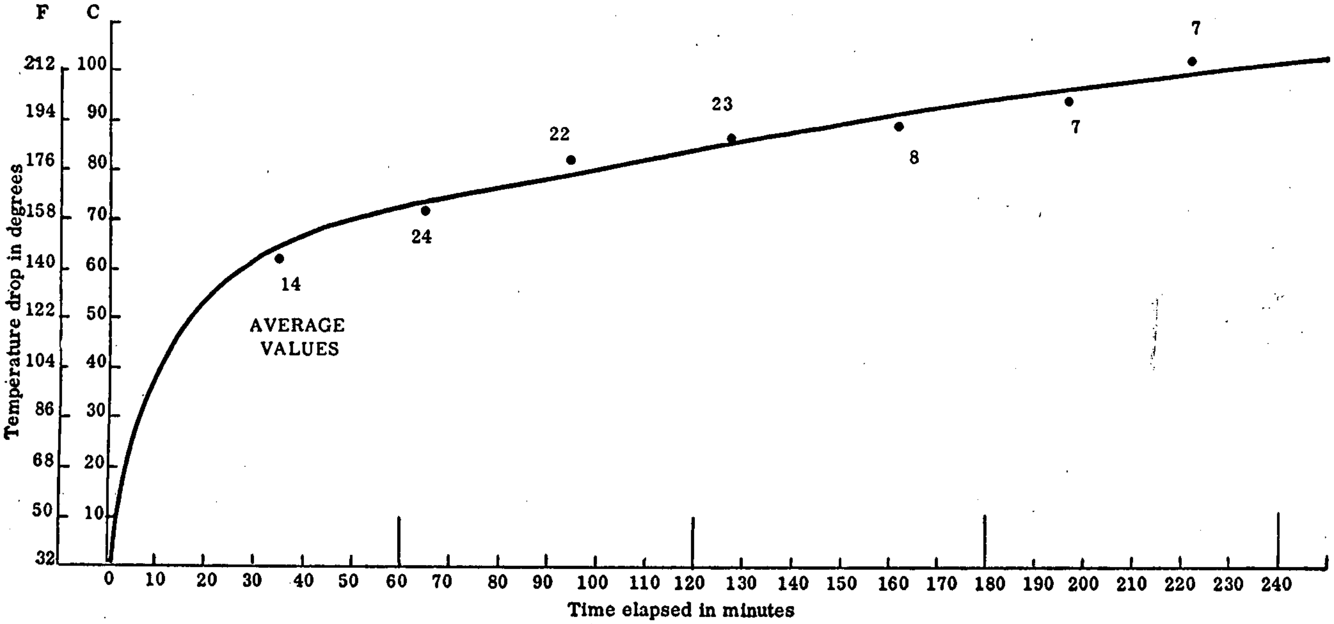


Figure 2.
Iron temperature drop in the blast furnace transfer ladle (Kling ladle type).

Some of the physical characteristics of the bricks are also shown below:

Apparent specific weight	2.25–2.35
Density (gr/cc.)	1.60–1.70
Linear expansion (5 hrs. at 1,500°C.)	1.0%
Expansion in volume (id)	2.93%

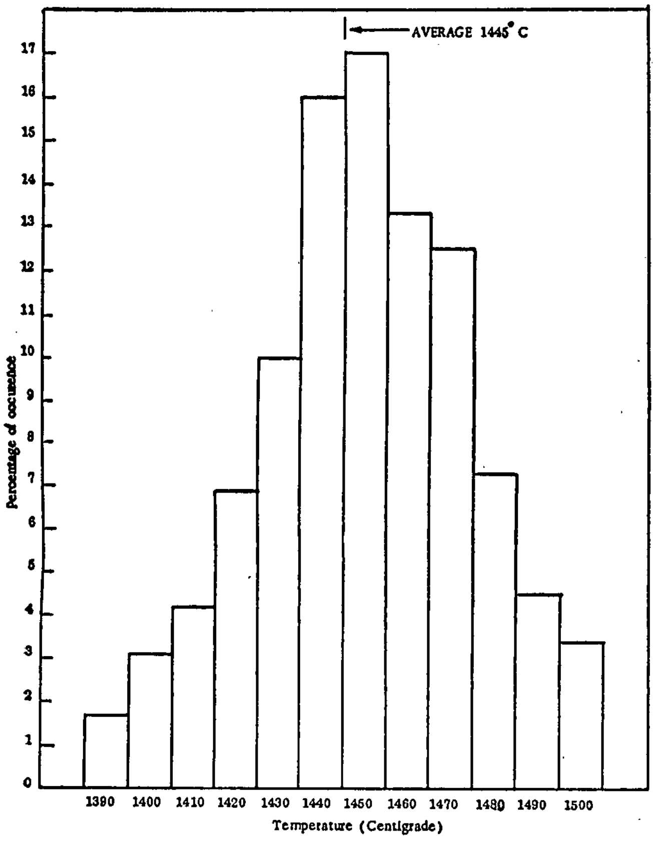


Figure 1.
Frequency curve. Blast furnace cast temperature.

The brick life in these Chilean-made silica blocks has proved excellent. Under rather heavy conditions the service performance is now about 1,000 heats per lining, while under normal conditions the bottoms last from 30 to 40 heats.

A careful drying sequence is necessary to obtain good results. An average of 600°C. heating over a 24-hour period should be followed by a maximum temperature of 1,000 to 1,100°C., during 25 to 30 hours.

The usual rammed-in bottom, 75% ganister–20% fireclay, is the plant practice.

Blowing practice

A blast furnace iron within the following limits is produced:

Si	1.25–1.60%
Mn	.30– .35
P	.085 max.
S	.035 max.

The aim is to keep the scrap constant in the charges, at around 2.1 tons, or 14%, in relation to a hot metal charge of 12.9 tons, or 86%.

The variation of the hot metal temperature (see Figure 1) is a function of the usual bell shaped curves obtained when tests are systematically repeated.

The variation of temperature of the pig iron (when the blast furnace is casting) is shown in Figure 2. The limits of this variation are 1,390°C. and 1,500°C. with an average of 1,445°C.

Usually blowing starts 20 to 40 minutes after the end of the cast, and sometimes the delay may amount to two or three hours, with a total loss of 55° to 95°C. (see Figure 2). Under these conditions the Huachipato average charging temperature in the converter is 1,350°C.

There are no plans to cool the Bessemer metal with scrap additions nor has steam blowing been found necessary for normal practice.

Among the factors influencing the temperature reached by the metal when blowing, the most important ones are the following: (1) percentage of silicon in the hot metal; (2) temperature of the hot metal charged; (3) condition of the lining and of the bottom of the converter.

It is attempted to blow "young", with an ideal C content of .06 to .07% and an "after blow" of five to eight seconds. Nevertheless a sluggish condition develops when the Si/Mn ratio becomes 3.7 or less (see Figure 3). The ideal ratio is between 4.0 and 4.5, although United States practice considers the limits 2.0 to 2.5 to be the best.

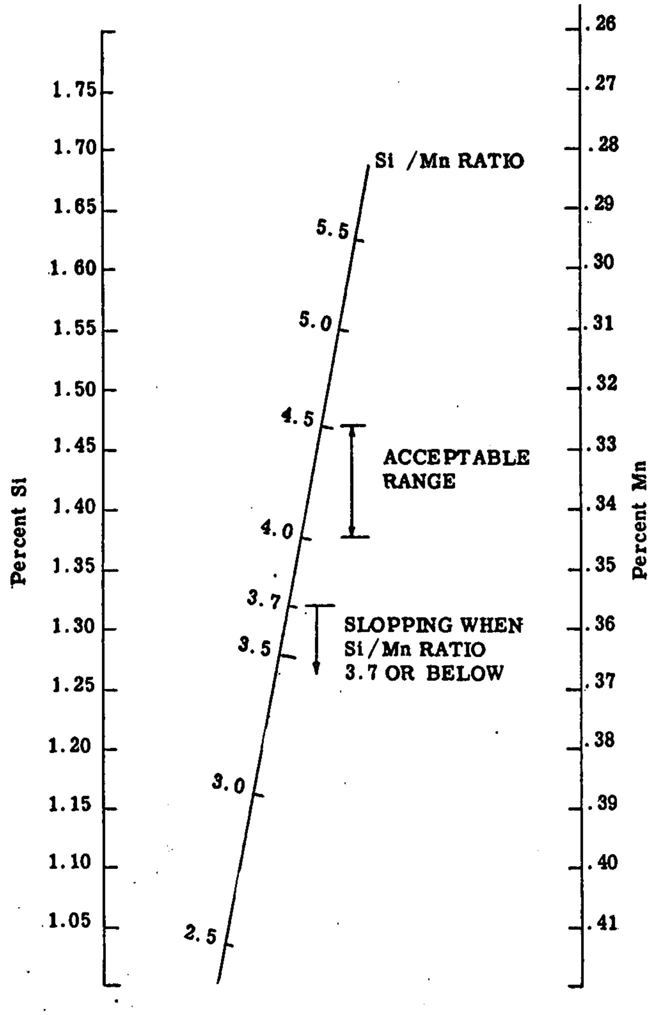


Figure 3.
Alignment chart Si/Mn ratio.

Huachipato slag practice

The slag is very fluid by normal Bessemer operation standards, its chemical limits being as follows, on a percentage basis:

SiO_2	FeO	Fe_2O_3	MnO
33–40	40–45	5.5–7.0	4.0–8.5
CaO	MgO	Ti	V
.20–.60 Max.	.35	.40–.60	1.7–2.4

It can be noted that there is a large proportion of FeO and Fe_2O_3 in relation to the SiO_2 content, and a low MnO. In fact 20% FeO is considered normal. Figure 4 shows how this slag compares with a slag considered normal by American standards.

The reason for such high FeO in the converter slag is not yet known, but it is thought to be the cause of the sloppy converter operation, together with the wrong Si/Mn ratio. This ratio is shown in Figures 3 and 4.

Ti and V are dissolved by the slag during the blow, although V recovery from the slag is under consideration because of the relatively high V_2O_5 content (3–4%) and low P_2O_5 (.02–103%).

Tapping and late additions

Deoxidation techniques include an addition of 50 kg. of 50% FeSi to the bottom of the ladle, together with 130 kg. of silico-manganese. Manganese recovery varies from 55 to 60%. In the case of Huachipato the low recovery is, of course, due to the very fluid slag and the metal-slag reactions during the reduction process. Several techniques have been tried in order to overcome the high Mn losses, the best results having been obtained by throwing silica sand to the nose of the converter with a shovel of special design, prior to pouring. The slag is kept in the converter until this is half full. The silico-manganese is thrown into the stream of steel early in the pouring as well as all other additions. Finally the Bessemer is emptied entirely.

With this present slag practice the recovery of Mn has been raised to the present figure of 55–60% which amounts to a 10% improvement.

Teeming

Whereas the converter metal temperature is from 1,710 to 1,690°C. the teeming temperature is around 1,610 to 1,590°C.

The travelling time from the converter to the teeming platform is about four minutes.

Ingot moulds are big-end-down of conventional design, for ingots weighing from 3,200 to 3,400 kg. A trail ingot is used to reduce metal losses. When pouring the ingot moulds a final deoxidation is made with 300 to 400 gr. of Al, in the shot form, per metric ton. The aim is to obtain a slightly convex ingot top.

Production tonnages and yields

The Bessemer plant production averages some 6,700 metric tons per month, but its actual capacity is much larger. The balance of the available capacity is used for blowing duplex metal and also to produce de-siliconized iron, to be refined in the open hearth.

The average yield of the converter operation is 88.6%. It has been calculated over an extended period of time covering about 1,000 heats of the plant's specifications, either by weighing the charges and the blown metal, or by comparing the percentage of phosphorus in the charge to that in the steel. Ingot butts amount to approximately 4%.

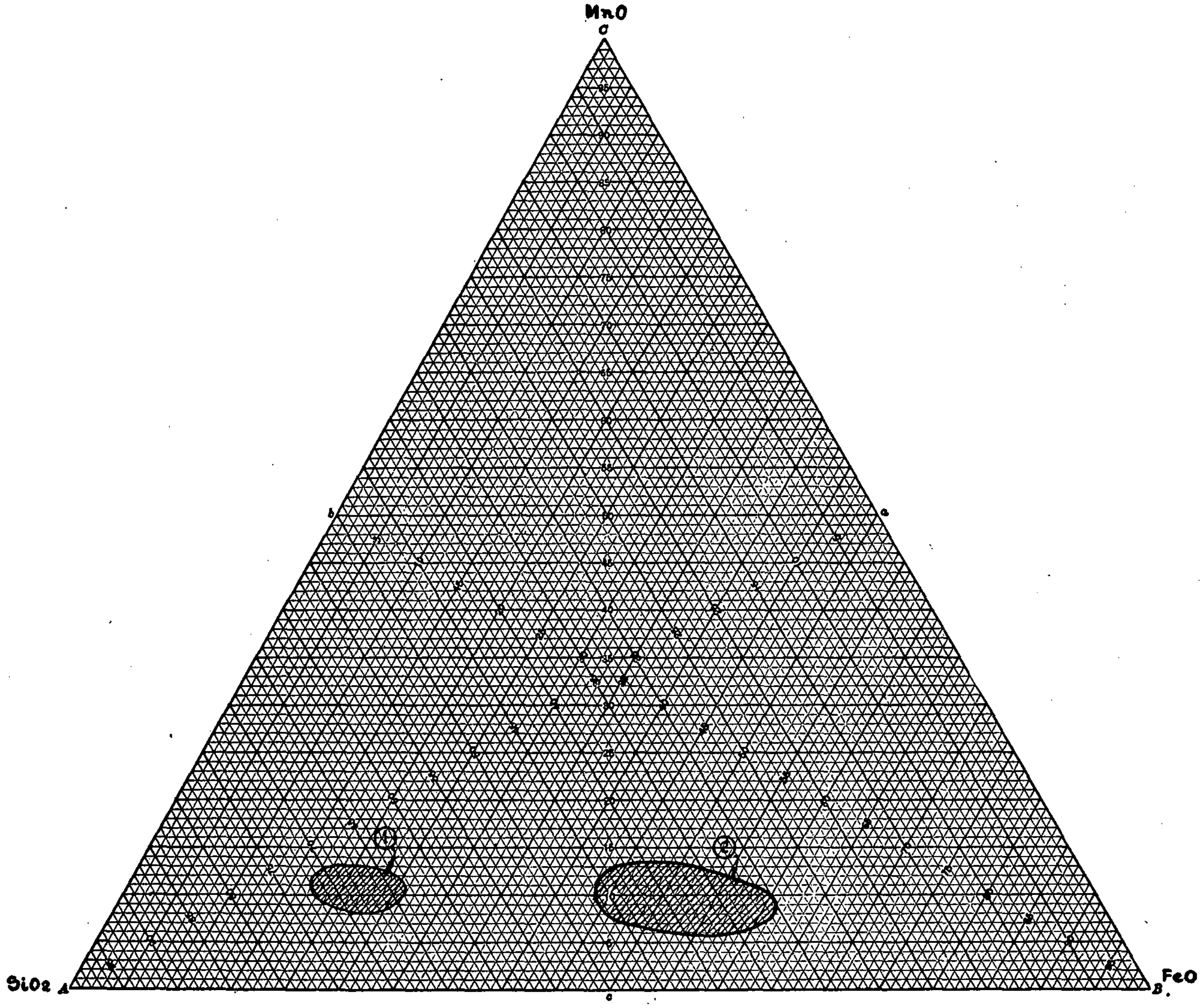


Figure 4.
Bessemer Slag Composition
Jones and Laughlin Steel Corp.
(R. E. Edwards—*Journal of Metals*, May 1950.)

Considerations Concerning the Choice of Electrical Equipment for the Iron and Steel Plants of Paz de Río (Colombia) and Chimbote (Peru)

R. L. JUNG

The modern steelmaking plant is, in its various parts, largely dependent on electrical energy. This energy is used either directly for smelting and reduction of ores and steelmaking or as motive power. The plant is thus a major consumer of electric current and the judicious choice of the form in which it is used has an important bearing on the general economy of the plant.

The problems concerning blast furnaces and electric smelting furnaces have been dealt with elsewhere. This paper will deal with application of electric equipment in the rolling mills and subsidiary equipment of two Latin-American steelworks, Paz de Río and Chimbote.

A special difficulty derives from the diversification of the products, i.e., blooms, slabs, wire, rounds, sections, and so forth, of many sizes, and the limited output of these plants, this second problem being common to most Latin-American enterprises. Thus the ideas developed in the description of these two plants can be said to have a general bearing on new plants to be built in Latin America.

Experience has proved that the control units should allow easy speed regulation, convenient power equalization by means of flywheels, proper energy synchronization, etc., thereby increasing the efficiency of the equipment.

1. EMPRESA SIDERURGICA NACIONAL DE PAZ DE RÍO

The rolling mills of this plant consist of two sections: one three-stand combined structural and sheet mill powered by a 1,500 h.p., 300–600 r.p.m. motor: one combined merchant mill made up of: a three-high roughing mill powered by a 1,500 h.p., 300–600 r.p.m. motor; a seven-stand intermediate mill powered by two motors, one being 1,000 h.p., 300–600 r.p.m., and another 800 h.p., with 250–625 r.p.m.; a continuous six-stand wire mill powered by a 1,300 h.p., 450–900 r.p.m. motor; a wire mill powered by a group of variable speed motors with a total output of 1,200 h.p.

It will be seen that, for the reasons stated above, motors with a wide range of speeds were installed. The best rolling-mill motor of this type is still a DC motor, since at the present stage of technical development it is the only one which combines sturdiness with the maximum flexibility required. Unfortunately this raises the question of current supply from modern three-phase AC power stations and grids.

Modern technology does, however, offer some solutions, such as the following: the use of high-power mercury-arc rectifiers; the application of electronic regulation to the current rectifiers; and the use of new static and rotating regulators.

The choice of rectifiers

Since the end of the First World War, European technicians have concentrated on the use of high-power metallic tank rectifiers for the direct feeding, not only of the general DC distribution networks of steelworks, but especially for rolling-mill DC motors of all outputs. The power per tank unit rapidly reached several thousand kW. These rectifiers have been very well received because of their fundamental advantages over the rotating converters which preceded them. These advantages include: simple low-cost construction; easy maintenance; small dimensions; low weight; cheap foundations; good power characteristics, that allow an average of 10% gain in industrial consumption, as compared to motor-generator sets.

Despite their 30 years of utilization in steel-making, the steelman often hesitates to adopt rectifiers and is embarrassed by the numerous types of equipment he is offered. It must be recognized, however, that rectifiers have improved considerably and that their evolution is far from complete. It is interesting to ascertain where they stand today in relation to the needs of the steel industry.

The first high-power, industrial metallic rectifiers were of the polyanodic type, that is, the transformation of multiphased AC current into DC current took place in a single tank of appropriate size. Shortly before the Second World War, the monoanodic rectifier (ignitron or excitron), containing a tank per phase, made its appearance, the rectifying block consisting of 6, 12 or 18 small tanks.

The monoanodic rectifier is claimed to offer the following advantages: a better yield for voltages below 300 v.; small tanks resulting in economical volume production; and a cheap reserve tank.

In the case of the polyanodic rectifier, the advantages are: greater safety up to 1,500 v. with normal construction; and greater simplicity.

More recently, both systems have been improved and tend to become similar in properties.[1] The choice between them therefore depends on the particular conditions prevailing in each special case.

Both types of rectifiers can be so constructed that cooling may be effected by water, for large sizes, or air, for smaller ones. This eliminates the difficulties relating to the circulation of cooling water with its pumps and its heat exchangers.

The general structure of water-cooled and the air-cooled rectifiers is similar. They differ, however, with regard to the disposition of the mercury-condensing surfaces, and to the size of the outside wall. Ventilation takes place either individually by a ventilator connected under the apparatus or by a conduct linking the bottom with a general ventilation installation.

The economic capacity of ventilated tanks is approximately 3,500 A, both for single polyanodic tanks as well as for groups of six monoanodic tanks of the same power corresponding to approximately 2,000 kW with 600 v. or 2,500 kW with 750 v.

However, in view of the power levels normally used in steelmaking, there appears to be no practical utility in trying to achieve greater unit outputs. In the case of the polyanodic rectifier in particular, when greater power is needed it does, in fact, seem more rational to multiply the number of tanks, thus creating cheaply a certain reserve. This is, in fact, the same practice as used with small monoanodic tanks.

As a result, and after a development which has lasted for more than 15 years, the ventilated rectifier has replaced the water-cooled polyanodic rectifier for metallurgical uses.

Finally the latest improvements in high-power metallic rectifiers consist in the suppression of the two vacuum pumps contained in the tanks. Sealed tanks are built for current of 500 to 1,500 amperes. A reservation regarding the utilization of certain tanks is that this makes the user dependent on the constructor, inasmuch as a tank which is out of order cannot be repaired on the spot, and that both reserve tanks and tanks which are not used are subjected to ageing according to the initial quality. Contrarywise, experience has proved that a tank provided with pumps can always be repaired on the spot and that it lasts almost indefinitely.

A problem to be considered in selecting rectifiers is the question of back-fire. This is essentially a matter of design, and providing a type of tank is chosen which is well-adapted to the service requirements, there will be no more difficulties than with the rotary converters.

All the above was taken into consideration in selecting DC rolling-mill motors and mercury-arc rectifiers for the production of practically all the direct current used at Paz de Río. In order to obtain an adequate elasticity in the functioning of the system as a whole, 11 air-cooled monoanodic tanks of 2,500–3,000 amps were selected. They are distributed as follows: two 600 v. tanks for the two 2,200 motors of the structural mill; two 600 v. tanks for the four motors of the merchant mill; two 250 v. tanks for the auxiliary motors of both rolling mills; two 250 v. tanks for the motors of

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[1] Augier, M. Monoanodic and polyanodic rectifiers. *Bulletin de l'Institut electrotechnique Montefiore*, Liège, Nos. 7, 8, 9; 1950.

the wire mill; two 250 v. tanks for the motors of the steel mill; and one reserve tank which is common to the whole plant.

Utilization of electronic regulation

The regulation grids of the rectifiers allow any value of the tension between zero and the maximum. The regulation is achieved electronically, and with the help of ultra-rapid static regulators. These regulators utilize the properties of "pre-saturated reactors" better known as "magnetic amplifiers, amplistat, transductor" which have small control currents on the grids, and can therefore be actuated rapidly and with precision.

The advantages thus obtained for the Paz de Río installation are the following:

(*a*) Maintenance of a constant DC tension at the poles of the rectifiers secured by the operation of the electrical regulating equipment;
(*b*) Starting of the rolling-mill motors, individually or in a synchronized group, by hand or automatically, through a static intensity regulator;
(*c*) Speed regulation, performed in the case of the normal rolling speed by the rheostat of the motor, and in that of the minimum speeds required for starting new types of work, by reducing the rectifier tension;
(*d*) A rational use of fly-wheels for power equalization, due to the electronic units.

Not only does this eliminate any return of current to the grid above the regulated level, but it also allows for a reduction in the size of the wheel. In order to obtain this result, it is sufficient to equip the rectifier with a static intensity limiting regulator. This method is used for the two 2,200 h.p. motors.

Utilization of the new static and rotary regulators

Rolling mill operations need simple, accurate regulating devices. They must be easy to maintain and operate. The new amplidyne regulator, developed shortly before the Second World War, is the first and most perfect example of such a device.

These new regulators were specially designed to meet metallurgical requirements. They are usually in the form of high-grade exciters with a very high degree of amplification. To increase their precision and reaction speed, they can be excited by the static magnetic amplifier described above. The regulation is performed in less than one tenth of a second, which is sufficient for most steel-making problems. The fact that they can play the role of a multiple regulator simplifies the control apparatus, consisting of relays and conductors.

In steel-making they are found often in relation with the "Ward-Leonard" coupling in the form of exciters of the generator or of the direct current motors, or of both at once. They also act as accelerators, sharply counteracting the slowness of the "Ward-Leonard" coupling. For the Paz de Río plant, several of these amplifier controls are contemplated, i.e., to synchronize the two last motors of the merchant mill, operating in tandem on the same line; to synchronize the wire-coiler operation with the corresponding rolling-mill motors; to control the shearing of the blooms; for the automatic control of the two hot saws; and for the automatic control of blast furnace skip.

2. CORPORACIÓN DEL SANTA—
CHIMBOTE PLANT (PERU)

The control principles adopted in the case of Paz de Río have also been applied here. Initially, the rolling mills will consist of:

One large two-stand mill for billets and blooms, powered by a 1,800 h.p. 515 r.p.m. motor.

One merchant and wire rod mill consisting of three parts:

(*a*) Two stands powered by a 1,000 h.p., 515 r.p.m. motor;
(*b*) Three stands powered by a 700 h.p., 225–450 r.p.m. motor;
(*c*) Two stands powered by a 500 h.p., 250–500 r.p.m. motor.

Six stands powered by a 800 h.p. 900 r.p.m. motor.

One sheet mill consisting of three stands, powered by a 1,800 h.p., 515 r.p.m. motor.

The constant speed motors are, of course, 4,000 h.p., 50-cycle, three-phase motors.

On the other hand, DC motors have been adopted for the two 700–500 h.p. variable speed motors, which are fed in parallel by two polyanodic grid-ventilated rectifiers. They develop 900 kW under 600 v. and are fed by a common tranformer. In case of damage to one of the tanks, the other one is practically sufficient to ensure the operation of both motors. This combination, which is very economical, also offers great security.

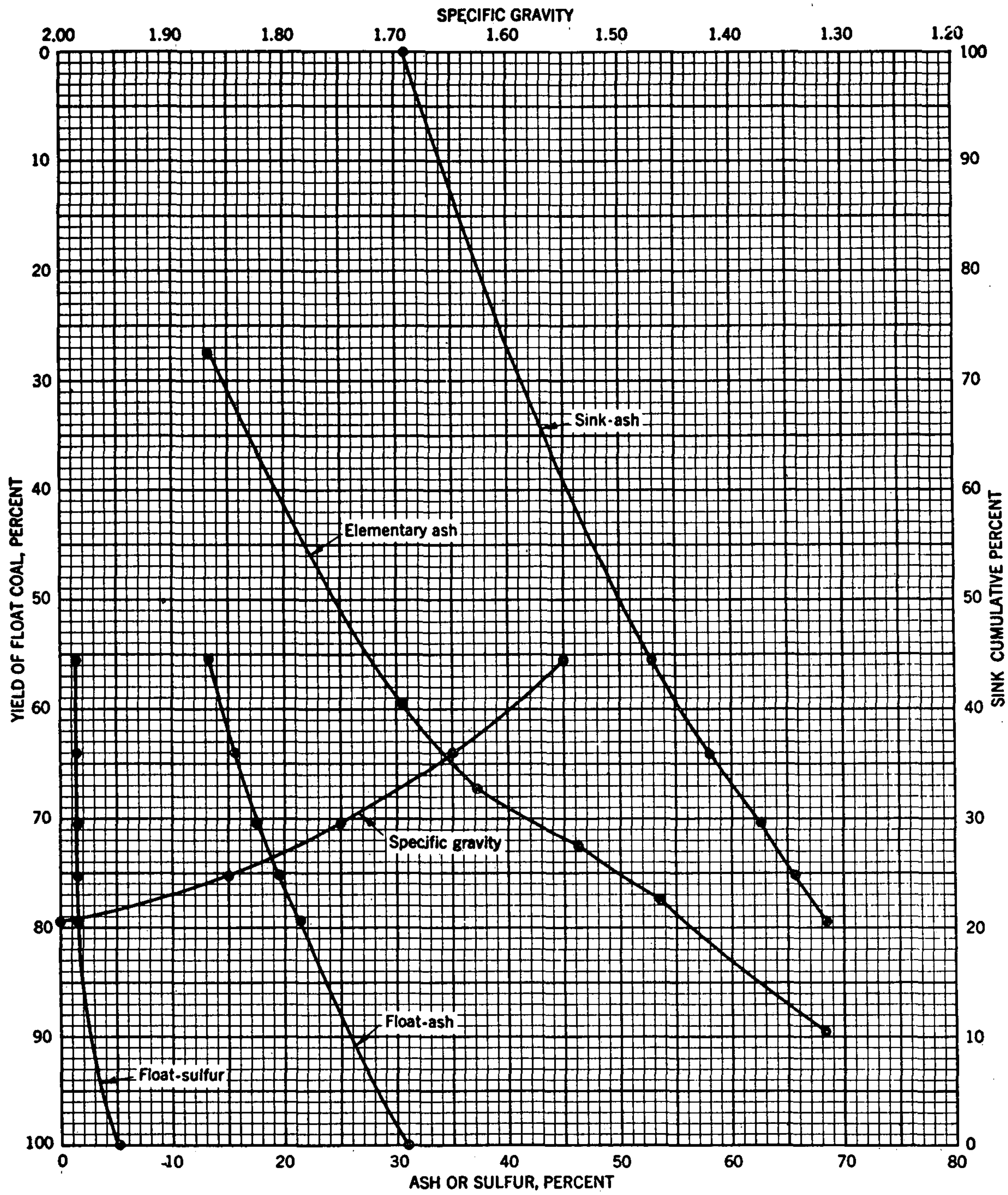
Amplidyne controls are foreseen for the synchronization between the two last motors of the wire rod mill working in tandem and for the synchronization of the wire coilers.

Summary of Discussion

Selection of Steelmaking Processes and of Locations for Integrated Iron and Steel Works, *presented by Mr. Buell in the absence of the author*

Mr. ALLARD opened the general discussion on the paper. He referred to statements made in it with reference to Thomas steel and affirmed that the major part of steel plate in France—and perhaps in Europe—was made by the Thomas process.

As both he and Mr. Coheur had stated earlier, some notable improvements had been made in the Thomas process. He thought, however, that the open hearth process had certain possibilities when rich ore was available.



Specific gravity fractions	Elementary data				Computed cumulative data					
	Weight grams	Percent			Float, cumulative percent			Sink, cumulative percent		
		Weight	Ash	Sulfur	Weight	Ash	Sulfur	Weight	Ash	Sulfur
Float - 1.55		55.6	13.3	1.52	55.6	13.3	1.52	100.0	30.9	5.10
1.55 to 1.65		8.5	30.8	1.82	64.1	15.6	1.56	44.4	52.9	9.58
1.65 to 1.75		6.3	37.2	1.94	70,4	17.6	1.59	35.9	58.1	11.42
1.75 to 1.85		4.8	46.2	2.28	75.2	19.4	1.64	29.6	62.6	13.43
1.85 to 2.00		4.5	53.7	2.88	79.7	21.3	1.71	24.8	65.7	15.59
Sink - 2.00		20.3	68.4	18.41	100.0	30.9	5.10	20.3	68.4	18.41

Fig. 12. Washability chart of composite of Barro Branco bed samples crushed to 38 mm. top size. Size 1.5 – 0.7 mm. increment.

Mr. BUELL stated that the paper did not intend to imply that satisfactory steel could not be produced with the open hearth and basic Bessemer processes. Mr. Haven had merely pointed out that the blast furnace had been found, in his experience, to be the best method of producing steel. The blast furnace still produced higher tonnages than any other processes in the United States.

Mr. PALMÉ wished to associate himself with the statement made by Mr. Allard. In Western Europe, practically all rails were made of Thomas steel. He expected to deal at greater length with that subject during the following day's meeting.

Mr. MERCIER observed that Mr. Haven had mentioned the development of minerals preparation for the production of iron. In France, the preparation of minerals at the mine and at the plant (screening, crushing, sintering) was fully developed. Results had been very successful and the plants adopting that method had decreased their coke consumption by at least 100 kg. per ton of iron.

Mr. CAVANAGH expressed the view that Mr. Haven's paper would be extremely useful. He wished, however, to know the reasoning behind Mr. Haven's opinions on sponge iron. He wondered whether Mr. Buell thought that a sponge iron production of 30 tons a day would be possible in a location without scrap.

Mr. BUELL replied that his firm would be very interested in any processes with sponge iron that offered reasonable opportunities for success under existing conditions. Personally, he would not wish to make steel from large quantities of sponge iron. However, he would be interested in sponge iron that could be adapted to Latin-American conditions.

Mr. ARAMBURU said that he had found the paper very interesting from an economic point of view. However, there seemed to be a contradiction in terms in Tables 1 to 5. On the question of the relationship of manpower to production, he was inclined to disagree with Mr. Haven. He considered that costs per ton of production were unusually high for the first plant (Table 7).

Mr. BUELL declined to comment on Mr. Aramburu's statements, saying that Mr. Haven would forward his comments by mail.

General MACEDO SOARES E SILVA thought that the paper was a very interesting one and that Latin America would derive great benefits from the points made in it. With regard to the location of the Volta Redonda plant, however, he wished to state that it had been governed by economic, not by military, considerations. Furthermore, the quality of the ores did not play the important role assigned to it by Mr. Haven. Volta Redonda had been selected because of its proximity to the markets.

Another point he wished to make was that although manpower was relatively cheap in Latin-America, earnest efforts were being made to raise the standard of living of the Latin-American worker. That prospect should therefore be borne in mind when estimating costs.

On the whole, the paper was a very useful one, and he expressed his gratitude to Arthur G. McKee and Company for their co-operation in that respect.

Mr. BUELL observed that Mr. Haven would also comment on General Macedo Soares e Silva's remarks by mail.

Mr. MARTIJENA pointed out that the use of oxygen converters under the right conditions would have a very great influence on steel production. He had in his possession statistics showing the proportion of scrap used in Europe and in the United States. In Europe about 30% of steel was produced by the Bessemer converter process and the amount of scrap used was about 57%. Undoubtedly, in Latin America, the proportion of scrap would be considerably lower. However, in view of the fact that Latin America produced, on the whole, much more than it consumed, there was no need for pessimism in that respect. It seemed obvious, therefore, that the evolution of the oxygen converter processes should be carefully watched. In Argentina, refractories involved heavy expenditure and had to be imported. Furthermore, in Brazil and Argentina, the question of fuel was very important.

Mr. YDSTIE remarked that Mr. Haven seemed to discourage the use of the electric furnace, and he wondered why the electric furnace should not be used in countries that were poor in metallurgical coke and rich in hydroelectric energy. Many European plants had been operating successfully with the electric furnace.

Mr. BUELL replied that no aspersions had been cast on the electric furnace in Mr. Haven's paper. Local conditions naturally governed the choice of furnace, but in the United States, the blast furnace had been found to be the most adaptable to prevailing conditions.

Mr. CORTÉS OBREGÓN said that the paper under discussion seemed to bring out the important aspects of Latin-American conditions. It was very useful to have an idea of the problems to be faced when a country was ready to install steelmaking plants.

Mr. ANAWATE intervened to remark on the statement made in the paper to the effect that Brazil had to import foreign coal. Actually, Santa Catarina coals were used, but there was a problem with regard to freight rates, which had been increased during the past year.

He then referred to General Macedo Soares e Silva's comment that the principal problem of Latin America was not the price of the material, stating that if the Rio de Janeiro market were taken into consideration in the establishment of Volta Redonda prices, as well as the cost of materials bought in the United States, Brazilian prices would be competitive.

The Duplex Steelmaking Process at Monterrey, *presented by Mr. Morales, co-author*

Mr. MORALES, after presenting the paper, said that Mexico was very interested in the interchange of ideas and techniques among the various Latin-American countries.

The Acid Bessemer Process at Huachipato, Chile, *presented by the author*

Mr. ALLARD opened the discussion of the paper by remarking that it was a particularly interesting one because it was based on the operation of an integrated plant. He wished to point out that there were certain steel plants under his direction where the distance of five to six minutes from the ladle did not create any great problems; in his opinion, Huachipato might be susceptible to a re-cooling phenomenon.

Mr. BOULGER requested some information on the nitrogen content of the material.

Mr. CANGUILHEM replied that the amount of nitrogen never exceeded 0.015, while the average was .09 to .011. It was considered uneconomic to study titanium recovery aspects because of the low grade of the slag.

Mr. DECHERF, referring to the question of ingots, said that he had not experienced much difficulty in that connexion.

Mr. CANGUILHEM replied that his firm had been studying the possibility of using another shape of ingot mould; the figures shown represented only the ingot moulds currently in use.

Considerations Concerning the Choice of Electrical Equipment for the Iron and Steel Plants of Paz de Río (Colombia) and Chimbote (Peru), *presented by the author*

Mr. JUNG, before presenting his paper, observed that little consideration had been given in the meeting's agenda to the question of electrical installations. That, however, seemed to be the trend in iron and steel circles. He held the view that close collaboration between the metallurgist and the electrician would lead to greater progress in the development of new processes. He hoped, therefore, that the next iron and steel meeting would devote more time to electrical problems.

There was no discussion on the paper.

STEEL MAKING AND FINISHING

Alternative methods to the rolling mill for finishing steel

28 October 1952—Afternoon

Chairman:

Danilo VUCETICH, Administrator, Altos Hornos de Corral, Compañía de Acero del Pacífico, Corral, Chile

Contributed Papers:

Continuous Casting of Steel by the Rossi-Junghans Process, as practised by the Allegheny-Ludlum Steel Corporation, Watervliet, N.Y., U.S.A.

T. Y. WILSON, Manager, Continuous Casting, Allegheny-Ludlum Steel Corporation, Watervliet, New York, U.S.A.

Direct Reduction Yields—Variable Density Steels. Reprint from *The Iron Age*, 24-31 January 1952

P. E. CAVANAGH, Assistant Director, Department of Engineering and Metallurgy, Ontario Research Foundation, Toronto, Canada

Notes on the Ugine-Séjournet Extrusion Process

J. SÉJOURNET, Directeur général du Comptoir industriel d'etirage et profilage de métaux, Paris, France

Some Aspects of Labour Problems in the Iron and Steel Industry

THE INTERNATIONAL LABOUR OFFICE, Geneva

Summary of Discussion:

Participants: Messrs. BUEHL, COHEUR, WARING, CAVANAGH, COSTA LINO, ALLARD, LEUSCHNER SCHERESCHEWSKY, DESPRÈS, CANGUILHEM

Continuous Casting of Steel by the Rossi-Junghans Process,

as practised by the Allegheny-Ludlum Steel Corporation, Watervliet, N.Y., U.S.A.

T. Y. WILSON

While continuous casting of metals and alloys is of only recent satisfactory accomplishment, the idea is quite old. Many attempts were made; some of them probably failed more through lack of continuing endeavour than want of technical inspiration. The patent literature, both American and European, is vast. Many of the inventions were obviously untried, embodying ideas now known to be impracticable. While Sir Henry Bessemer is generally credited with being the first to accomplish continuous casting of metals and in about 1868 cast a 40 in. length of wrought iron between rolls, J. Laing (U.S. Patent No. 3023, 1843) had been continuously casting soft metal tubing around a vibrating mandrel. Undoubtedly the idea of continuous casting had occupied the thoughts of many earlier metal makers. For an excellent history of the art, see T. W. Lippert's paper published in *Metals Technology*, vol. 12, No. 2, February 1945.

In 1937, Scovill engineers went to Germany to examine the methods developed by Siegfried Junghans. They arranged for shipment of or to buy one of his machines, which by 1938 was in operation at the Scovill casting shop and during the following year produced close to 60 million lb. of fine quality brass billets for hot extrusion of rod, wire and tube. Casting rates now exceed 20,000 lb. per hour.

The present discourse concerns only the Rossi-Junghans method as applied to steel. It should be pointed out, however, that an alternative method is in pilot plant operation by Republic Steel and Babcock and Wilcox together at Beaver Falls, Pa.

Continuous casting equipments of any considerable size and practicability (Figure 1) have certain elements in common: (1) a molten metal reservoir, (2) a mould, (3) withdrawing rolls, and (4) a cutting-off device. The reservoir must suit both the type and temperature of the alloy to be cast, designed to maintain both chemistry and temperature within required limits for the duration of a single cast, be it for minutes or days. The mould, in which the section is formed, must accomplish some minimum amount of solidification through removal of heat. The withdrawing rolls propel the solidifying casting downward at a rate equal to the entrance of the molten metal. Unless there be undue sticking in the mould, the withdrawing rolls allow the casting to fall at a controlled rate and so do not actually pull it out of the mould. The cut-off device may be automatic or manual and either a saw or oxy-acetylene torch, depending upon the metal involved.

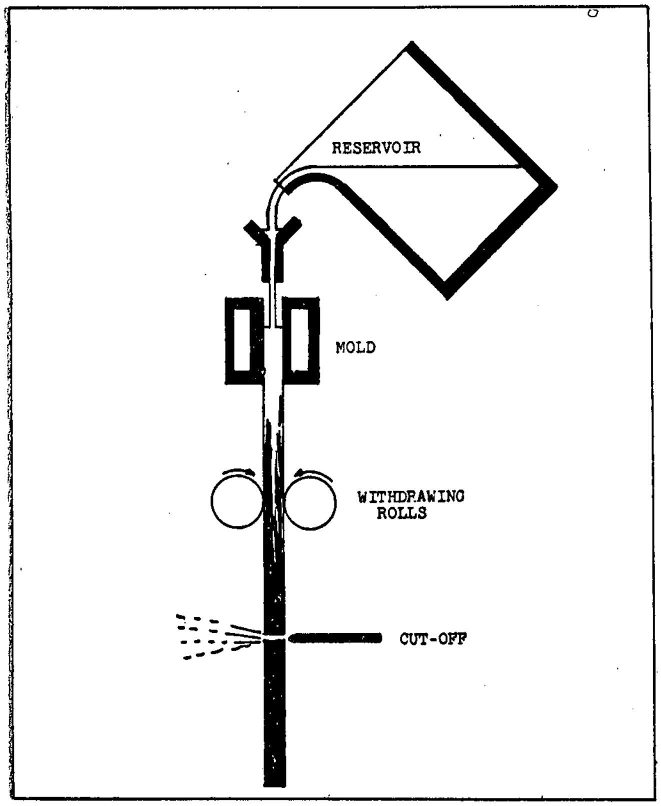


Figure 1

The one single invention of Siegfried Junghans which has contributed most to the successful development of continuous casting is the unique mode of reciprocation of the mould (Figure 2). Many early or contemporary attempts were doomed to failure through sticking of the casting in the mould. By the Junghans method, reciprocation of the mould is parallel to the axis of the casting, the mould descending at exactly the downward speed of the casting for a short distance and returning at an accelerated rate.

An amplitude of reciprocation of about ¾ in. (2 cm.) with the mould returning vertically at three times the rate of descent has been found quite as satisfactory for steel as it had been for copper and brass. During 75% of a cycle there is no differential motion between casting and mould, thus permitting solidification of the metal surface with a minimum of disturbance and maximum surface quality. The quick return has a stripping action, in which inertia of the metal being cast leaves it relatively undisturbed.

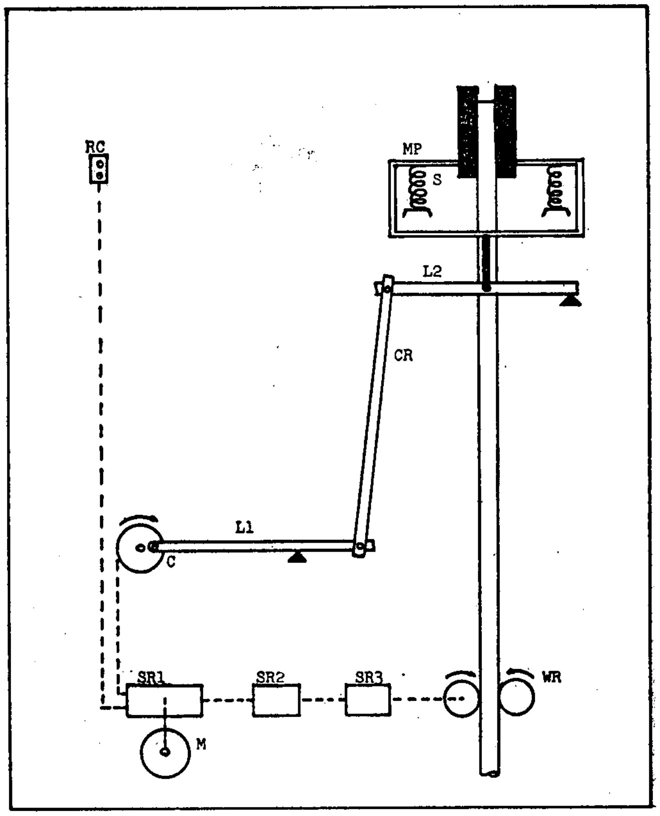


Figure 2
Continuous casting machine.

The mechanics of the oscillatory motion are simple: (Figure 2)

Single drive motor (M);

Stepless speed changer (S R);

Take off to cams (C), walking beams (LI, L-2);

Coil spring helpers (S), mould platform (MP);

Take off to rolls (WR) through synchronizing speed changer (SR_2), wherein exact correspondence between casting and mould motion is regulated;

Speed regulation of the entire unit is accomplished through electric control (RC) of the primary speed changer (SR_1) mechanical ratio.

Other drives, such as variable speed DC motors, are quite possible, but it is imperative that the circumferential speed of the withdrawing rolls be constantly that of the downward motion of the mould. This is the essence of the Junghans patent. Any other arrangement of rates of motion causes constant metal disturbance.

First operational plans for the continuous casting unit at Watervliet were built around a 15 in. x 3 in. (381 x 76.2 mm.) slab; and so the machine was designed to have a top speed of 40 in. per minute. That is: the maximum circumferential speed of the withdrawing rolls was 40 in. per minute. At a later period, when smaller sections were cast, it was found necessary to increase the top speed. This was accomplished by altering the pulley ratio between motor and primary speed changer to attain a top speed of 68 in. per minute. Greater flexibility has been attained in a machine recently designed by Freyn Engineering Dept. of Koppers Company, Inc.[1]

Before any casting could be done it was imperative to find refractory materials that would withstand the erosive and corrosive attacks of molten steels of various alloy contents and their slags with the temperatures in the 2,600–3,000°F. range. For various reasons, refractory metals for conducting molten steels were ruled out. While graphite has excellent thermal properties and is used to good advantage in the Asarco process for the continuous casting of small brass sections, it dissolves readily into and carbonizes steel and is rapidly oxidized itself.

These considerations left only ceramic refractories. The ceramic must have adequate thermal shock resistance without exacting preheating requirements. It must be resistant to erosion and, if possible, not wetted by the molten steel. Such a material was found in a peculiar grade of high-temperature stabilized zirconium oxide, now fabricated regularly by the Norton Company in Worcester, Massachusetts. Many tests of conducting molten steel through ceramic tubes were made in a shop where a 6,000 lb. arc melting furnace was available. Here the zirconia was proved before attempting to cast continuously even a small quantity of steel.

On 23 May 1949 the first cast was made: a 40-in. (1m.) length of 15 in. x 3 in. slab. Soon we were making casts up to 30 ft. in length with carbon and 18 Cr-8 Ni stainless steel poured from a bottom tap ladle, no holding furnace being installed, into an induction heated tundish. This, in turn, bottom tapped through a ½ in. diameter orifice and downspout into the casting mould.

One by one many of the difficulties which had beset the early investigators were overcome. The Rossi-designed casting mould (Figure 3) was quite efficient; and its peculiar oscillatory mode proved quite as satisfactory for steel as it had for copper and brass.

The slab mould, machined from a solid block of copper, has vertical water passes terminated top and bottom by integral manifolds. Each of the passages contains a concentric constrictor rod which confines the cooling water to an annulus. Water enters into the bottom manifold and leaves through the top manifold. For the 15 in. x 3 in. section about 100 g.p.m. (378.5 litres) has been used. The mould is 24 in. (61 cm.) high. A newer mould (Figure 4) for a 3½ in. sq. (9 x 9 cm.) section was built without the integral manifolds, an external manifold and a hose to each riser providing supply and separate hoses providing discharge.

Mould designs may vary in detail but certain principles are observed: (1) copper for good conductivity and corrosion resistance, (2) constrictor rods to minimize turbulence, and (3) hard chromium electroplate on the cavity walls for wear resistance.

The pattern and degree of cooling by water or air below the mould is governed by both the analysis of the alloy and the shape and area of the section. Thermal gradients made too steep will result in internal ruptures, either cracking the frozen shell and allowing molten metal to break out, or stress cracking the centres if all solid. Hot-shortness of the alloy also has a profound influence on this type of rupturing.

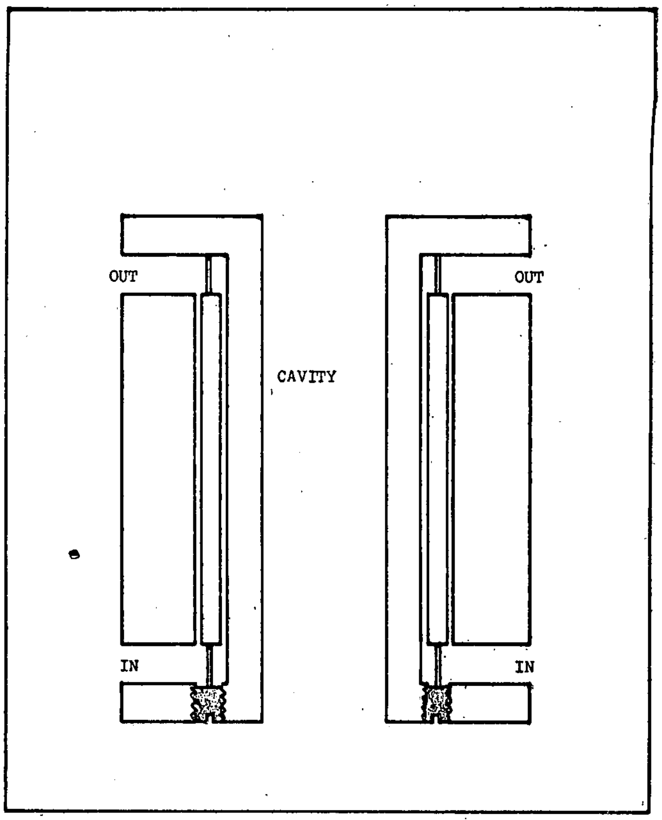


Figure 3
Continuous casting mould—vertical section.

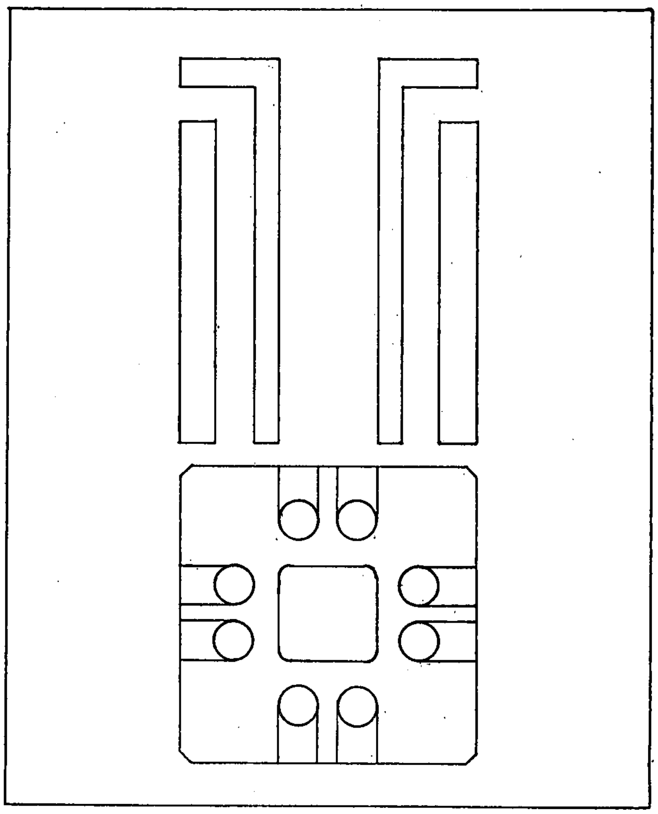


Figure 4

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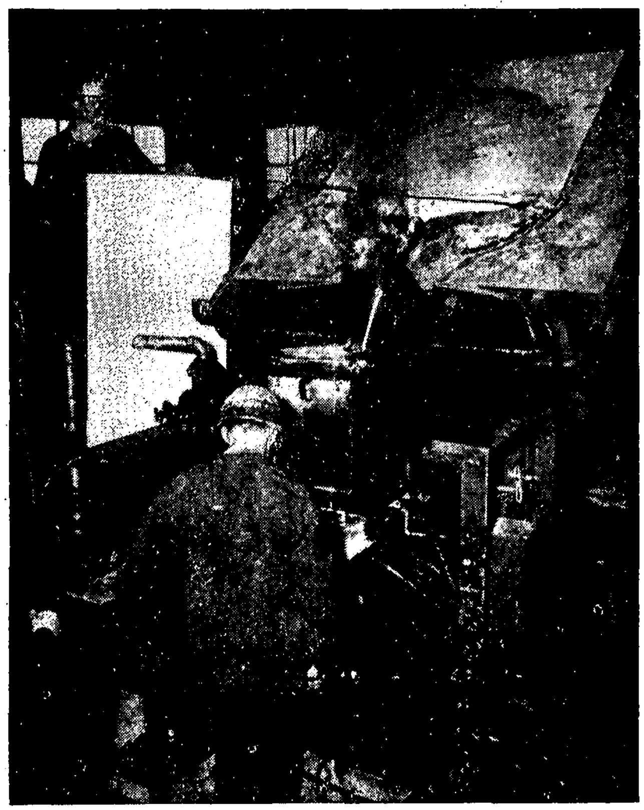
[1] The Freyn Engineering Dept., Koppers Co., Inc. has the exclusive right to design, build and erect continuous casting equipment operating under the Continuous Metalcast Co., Inc. licence agreement, and patents.

Cutting of the casting is done by oxy-acetylene torch. For the stainless steels and other oxidation resistant alloys, Linde powder cutting is used. Hot saws, such as are used to cut the non-ferrous materials, become ponderous and too slow for cutting carbon steel, if suitable at all for the work-hardening steels.

In the Allegheny-Ludlum experimental operation, the molten metal source is a 5,000 lb. capacity Ajax 1,200 cycle induction furnace which pours directly into a tundish which in turn, bottom pours through a nozzle orifice into the casting mould (see photograph 1). Control of the metal flow through the orifice is accomplished by maintaining a proper hydrostatic head above it. Early attempts were made to control the flow by needle valve, as is done in the non-ferrous operations, but ceramics of enough thermal shock and erosion resistance to sustain the required sharp needle taper could not be found. Other methods for automatic control are being studied currently.



Photograph 1

The induction furnace, while not suitable for long sustained casting runs, is eminently suited to the present operation, because temperature can be controlled so exactly. Pouring is done with power on—just enough power to maintain the desired temperature.

Experience has shown the need for close temperature control. Enough superheat is required in order to avoid freezing at the tundish nozzle, yet too high a temperature results in retarded solidification and undue oxidation. Oxidation of the metal in the furnace spout, tundish, and mould is minimized by an atmosphere of propane (C_3H_8). Molten steel, containing dissolved oxygen as it does, gives rise to some oxide slag development in the mould under the propane cover. This generally shows only on the casting surface.

As protective atmospheres, we have found argon inadequate and the natural gas available (1,200 BTU) not quite good enough.

The casting section range explored at Watervliet has been limited by both the product desired and the high cost of moulds. The sizes 15 in. x 3 in., 9 in. rd., 4½ in. rd., and 3½ in. sq. have been successfully cast. Rates of casting generally increase with the size of the section. The largest of the above is run at about 500 lb. per minute and the smallest at about 160 lb. per minute.

The maximum permissible casting rate is also limited by alloy composition, the solidification rates determining the time required to form a shell of strength adequate to resist deformation by the forces of high thermal gradients or high internal hydrostatic pressures.

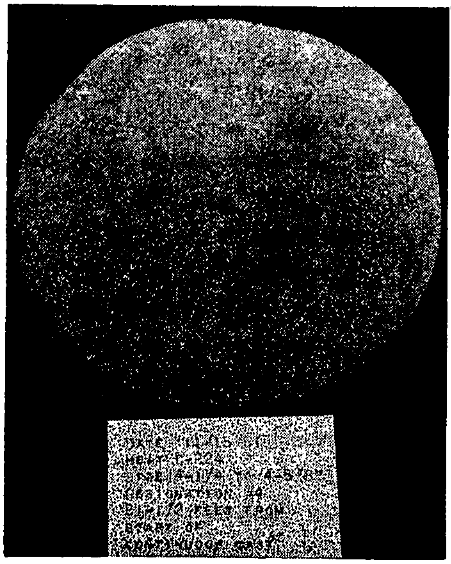
Plain carbon steels of from .07 to .80% carbon content, straight chromium non-hardenable steels of from 12 to 20% Cr, chromium-nickel stainless steels of the 18-8 types, an automotive valve steel, a cobalt base "superalloy", and a highspeed tool steel have been continuously cast (see Table 1). These represent a wide spread in physical and metallurgical properties. Melting temperatures range from 2,400 to 2,800°F. Some exhibit large temperature differences between liquidus and solidus, some air cool to martensite, some oxidize quite rapidly, some exhibit considerable hot-shortness, and some are notoriously susceptible to segregation. Each demands its own treatment in metal temperature, casting rate, after-cooling, and oxidation control for optimum results.

Table 1

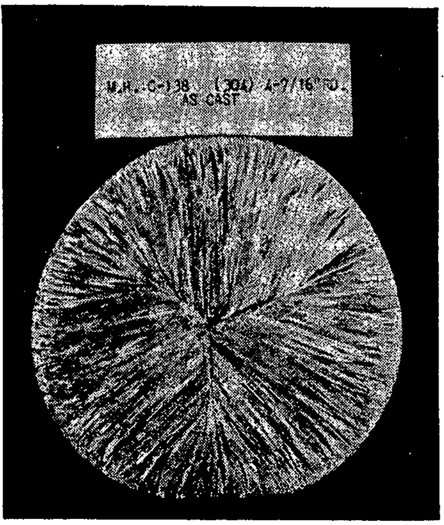
CONTINUOUSLY CAST STEEL ALLOYS

Type	*C*	*Mn*	*Si*	*Cr*	*Ni*	*Co*	*Mo*	*W*	*Cb*	*V*	*Ti*
1010	.10	.30	.20	—	—	—					
1080	.80	.35	.25	—	—	—					
403	.12	.40	.40	12.	.35	—					
442	.17	.35	.35	20.	—	—					
302	.10	1.50	.50	18.	10.	—					
304	.06	1.50	.50	18.5	10.	—					
321	.06	1.50	.50	18.	9.	—					.60
XB	.80	.40	2.20	20.	1.30	—					
S816	.38	1.50	.60	20.	20.	44.	4.	4.	4.		
DBL2	.83	.30	.30	4.	—	—	5.	6.		2.	

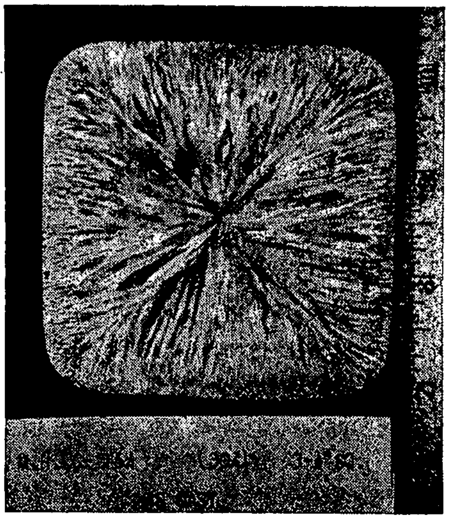
Proper temperature and proper cooling controls, together with certain mechanical aids, yield castings of excellent structure and surfaces frequently not requiring conditioning by grinding or machining. Typical examples are shown by the photographs of as-cast and of rolled products (photographs 2–5).



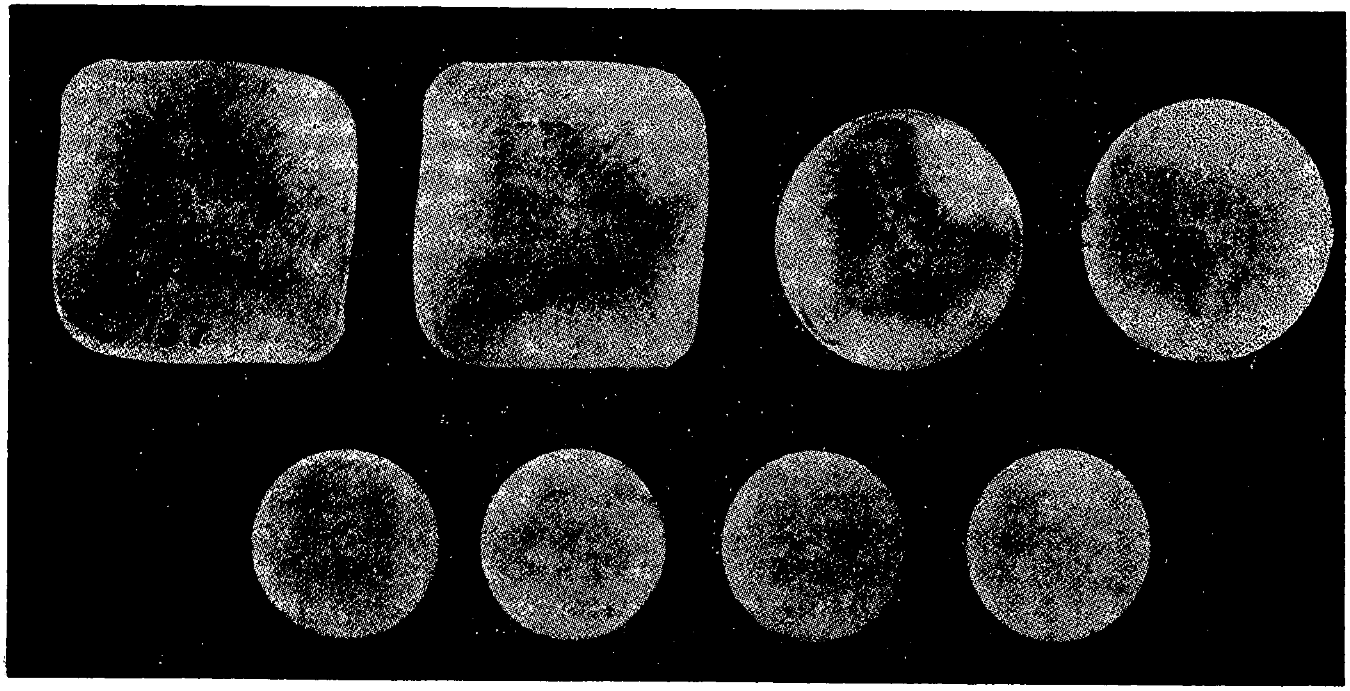
Photograph 2



Photograph 3



Photograph 4



Photograph 5

Carbon, 12% Cr, and 17% Cr cast slabs have been processed to good strip and sheet. Good 18 Cr-8 Ni seamless tube has been made from 4½ in. rounds after rolling to and piercing a 3¼ in. round. Round and square continuously cast ingots have been rolled to a variety of bar products without any special handling. Alloys 18-8 and 12 Cr have been successfully extruded from the as-cast condition by the Séjournet process, which employs glass as a lubricant.

Productivity of the process is determined by the alloy composition and section size, the maximum having been reached in Germany with mild steel cast into a 13 in. x 13 in. section. Quality and particularly yield generally show the greatest gains. For instance, 18 Cr-8 Ni reduced to rectangular bars in the size order of ¼ in. x 1 in. (6.35 x 25.4 mm.) show a 12% greater yield when rolled from 4½ in. rd. continuously cast ingots than when rolled from 12 in. x 12 in. conventional big-end-up ceramic hot-topped ingots. Molten metal to ingot yield approaches 100% as the length continuously cast increases, because the necessary bottom and top crop remains constant. It is obvious that the higher the intrinsic value of the alloy cast, the greater the economy of the Rossi-Junghans continuous casting method.

Continuous casting also promises enormous reductions in capital investment, as it can turn out a product not requiring soaking pit and blooming mill installations. Integrated with new melting facilities, the space requirements and continuing costs of ingot mould pits are eliminated. Some of the New England brass mills have rebuilt their facilities, centring upon Rossi-Junghans continuous casting machines.

The foregoing statements concern the Watervliet practice and experience, from which certain projections for future plant and operations of larger scale can be made. No actual operation has been pursued in the melt heat range of 10 to 200 tons, but considerable thinking has been done about it.

Special holding furnaces, designed but not built, are equipped with electric arc heating sufficient to hold temperature but not melt; and with induction stirring (Aros Electric, Inc.) to minimize thermal and chemical laminations. Simultaneous casting of two strands in order to double productivity has been designed also. Automatic pouring, as regulated by the altitude of the molten metal in the mould, is worked out, but, as the present small total of steel so cast does not now warrant the expenditures required, is as yet untried. Such automatic devices reduce both the amount and skill of labour required. Three to four men with a supervisor should be able, with proper mechanical aids, to operate the holding furnace, the casting machine, and billet discharge equipment. Masons for routine refractory maintenance might be drawn from the melting department force.

The productivity of a continuous casting unit as expressed in tons per hour depends upon the size and also upon the type of alloy cast. Currently much thought is being given to increasing casting rates. However, such increases may exceed available melting capacities, in which case there is little to be gained. Limitations therefore arise from melting available, desired cross-section of casting (as determined by subsequent hot-working facilities), and the nature of the solidification of the desired alloys (as determined by thermal conductivity, eutectic freezing, etc.). Over-all conditions and design of a continuous casting machine are determined by the nature of the business of the steel manufacturer.

Capital costs would vary with the aforesaid considerations, hence no general figures can be given. There may exist the question of entirely new plant *vs.* utilization of present equipment. Thus the cost might range from one quarter to several million dollars, not considering foreign shipment and import duty expenditures. Comparative costs of manufacture of the machine and its auxiliaries in one country *vs.* another are also unknown. Inquiries may be directed to Mr. Irving Rossi of 40 Wall St., New York City or Freyn Eng. Dept., Koppers Co., Inc., 109 N. Wabash Ave., Chicago 2, Illinois, both of whom have had to do with steel mill or continuous casting installations outside of the United States.

Operating costs may be estimated from the following schedule, using prices applicable to the particular country or area involved:

Rate of use of miscellaneous materials and supplies:

Cutting:	
Acetylene	.20 tank per hour of torch operation
Oxygen-preheat	90 cu. ft. per hour of torch operation
Cutting	10 cu. ft. per minute of actual cutting
Argon to propel powder	.5 cu. ft. per minute of actual cutting
#200 cutting powder	1 lb. per minute of actual cutting
Atmospheres:	
Propane in mould	1 lb. per hour of casting
In pouring box	2 lb. per hour of casting
In furnace spout	2 lb. per hour of casting
Or if natural gas is used:	
In mould	80 cu. ft. per hour of casting
In pouring box	200 cu. ft. per hour of casting
In furnace spout	200 cu. ft. per hour of casting
Pouring box heating:	
Oil	2 gal. per hour of heating
Gas	(no experience)
Cooling:	
Water in mould	80–200 gal. per minute of casting
In sprays	0–200 gal. per minute of casting
Electric power:	
Holding furnace	500 kWh per ton
Machine	5 kWh per ton

Estimates of labour costs should include an engineer, and also a supervisor and three operators per shift of operation. One of three operators should be of first helper quality.

Maintenance costs depend a good deal on availability of adequate ceramic materials such as fireclay brick, proper stabilized zirconia nozzles, sillimanite ramming cements, etc. This we cannot presage.

Direct Reduction Yields—Variable Density Steels[1]

P. E. CAVANAGH

The Ontario Research Foundation has developed a process for producing useful articles of steel directly from iron ore. The main characteristics are low cost and a wide choice of weight and strength.

The simple process involved was discovered by accident. While investigating the possibilities of producing sponge iron commercially (1), part of the work was devoted to the production of sponge iron suitable for processing into iron powder. To speed up reduction time sponge iron was produced in bars or wedges. Sponge iron bars intended to be ground into iron powder were reduced at a relatively low temperature. They were friable and easily broken up. However, a few bars accidentally produced were so hard, strong and tough that they could not be ground up.

A laboratory investigation was immediately started to try to produce this material deliberately. This did not seem a very promising project, since it is reasonably well known that direct reduction usually gives a density of 3.0 g. per cc. or less. It was thought desirable to produce a metal with densities of 4.0, 5.0, 6.0 or higher. All past experience suggested that any attempt to obtain such high densities directly from iron ore would result in pieces with the appearance shown in Figure 1. This cracked and broken piece is not a very useful metal bar.

Experience in producing extruded pellets from various iron ores (2), indicated the way in which this difficulty could be overcome. The success of the methods adopted is best illustrated by comparing the two bars in Figure 1. Both bars were made of the same ore and put through the same reduction process. But the screen size distribution of the ground ore was adjusted to provide for the behaviour of the individual ore particles during heating and reduction.

Laboratory investigation was then continued to a point where large pilot operation was indicated (Figures 3 and 4). The use of a pottery kiln in Oakville, 20 miles from Toronto, was obtained in September 1951, and pilot operation commenced. Since then, investigation of possible applications of this process has been the major activity. This exploratory work is now completed and pilot commercial operations are being undertaken for manufacturers interested in obtaining licences.

FACTORS DETERMINING STEEL DENSITY

The density of the steel is determined by several factors. Different iron oxides and different ore deposits of the same oxide behave differently during heating and reduction. Some ores will swell remarkably during this process, while others shrink as much as 50% in volume. This characteristic of various ores is one of the main factors in determining the final density of the product. The swelling or shrinking characteristics of the ore can be modified by adjusting the screen size distribution of the ground ore. This gives more or less voids in the final product and, therefore, some control of density. Final density of the steel can be increased by carrying on reduction and treatment at higher temperatures and for longer times (Table 1).

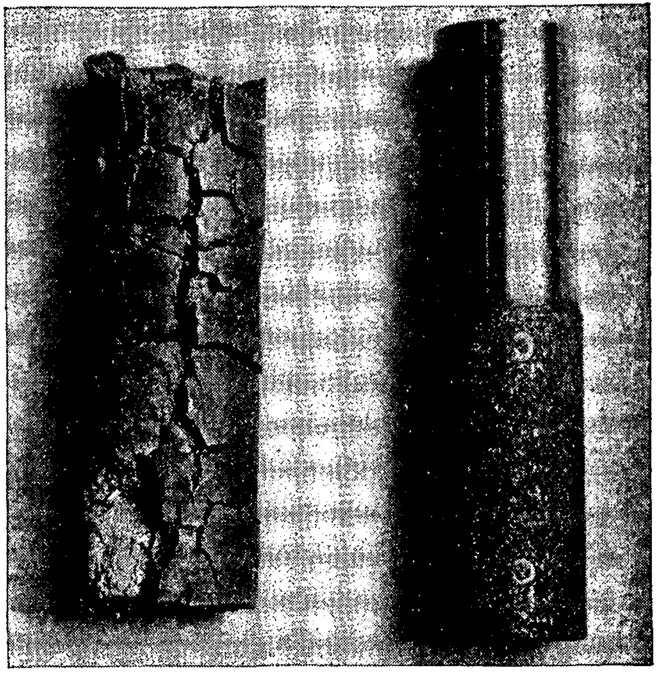


Figure 1. Two bars made from same ore and reduced in same way at 1950°F. At left, ore fines used were 25% −100 and +325 mesh, 75% −325 mesh. At right, 70% of −100 and +325 mesh fines, 30% of −325 mesh.

Carbon content of the steel products is determined by three factors. These are: composition of the reducing gas, reduction temperature and treatment time at temperature after complete reduction has been achieved (Table 2). Carbon content and density can, therefore, be separately controlled even with the same ore.

EXTRUDE ORE SAMPLES FOR STUDY

Success with the present method depends on a complete knowledge of the behaviour of the iron oxide raw material during reduction. In our laboratory studies, samples of ores are prepared in the form of ½ in. diameter extruded rods. A 500 g. sample is placed in the capsule of a reducibility furnace, where it is reduced. Weighing, examination and analysis follow at this point. Tables 3 and 4 show some of the results obtained with a variety of ores and illustrate the necessity of such preliminary studies. In addition, dilatometer studies of such extruded ore fines can be made at all stages of reduction (2).

If a given ore undergoes a volume increase of 10%, it is possible to reduce the amount of swelling markedly. This is done by grinding the ore to a lower bulk density (greater percentage of voids). An ore with pronounced shrinking tendencies can be blended with a swelling ore.

For production of parts with densities of 5.0 or higher, the use of a shrinking ore is indicated. The density of the product obtained from a given reduction

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[1] Reproduced, by courtesy of the publishers, from *The Iron Age*, 24–31 January 1952.

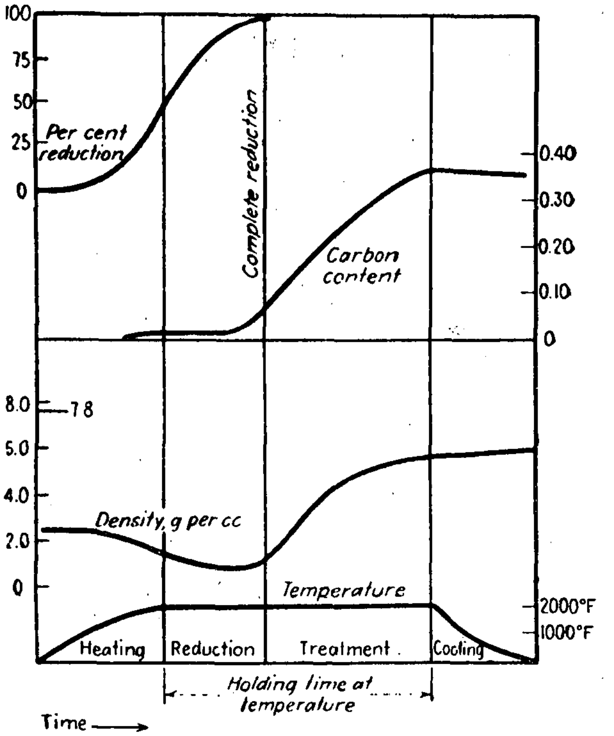


Figure 2. These curves show only trends in reduction practice. Specific ore characteristics, particle size and distribution, temperature, reducing medium and desired end product would influence actual operation of process.

process can be further increased by adding to the ore small percentages of salt, metallic iron powder and copper. The salt apparently acts as a cleansing agent. It keeps the reduced surface of the iron particles easily accessible to reducing gas and promotes sintering. The metallic iron takes up oxygen from the reducing gas and promotes faster reduction. This leaves more of the cycle time for sintering. The addition of copper appears to be beneficial for the same reasons as in iron powder metallurgy, when carried out above melting point of copper.

Some unexpectedly high densities are sometimes obtained by blending a swelling ore with a shrinking ore.

For many possible applications of controlled density steel it is obviously necessary to start with an extremely clean and pure iron oxide. The success which has been attained is due entirely to the 3-phase magnetic superconcentrator (3).

This equipment is in essence the stator of a 3-phase induction motor spread out flat. A travelling magnetic field is established on the under surface of the magnet. Fine particles of magnetic iron oxide fed underneath this magnet are rapidly transported through the air to the other end of the magnet. In addition to the forward motion of the particles a very rapid spin is imparted to them for the same reasons that an induction motor shaft turns in a similar field.

As long as a magnetic ore is ground sufficiently to free all the particles of gangue, these will fall into the concentrator's tailings bin. Magnetic particles travel under the magnet to the far end where they are dropped out of the magnetic field. In the case of some commercially available magnetite concentrates, one pass will reduce the gangue content materially, even when they are not ground any further.

Table 1

DENSITY CONTROL METHODS COMPARED

Method	*Ore*[a]	*Temperature*	*Hours at temperature*	*Density, g. per cc.*
Varying holding time at temperature	−60 mesh mill scale	1,950°F.	20	2.4
			24	3.7
			30	4.3
			36	4.5
Varying holding temperature	−60 mesh mill scale	1,900°F.	12	1.75
		1,950°F.	12	2.8
		2,000°F.	12	3.1
		2,050°F.	12	4.0
Ore blending	−60 mesh mill scale	1,950°F.	24	3.7
	With 25% blended fines[b]	1,950°F.	24	4.5
	With 50% blended fines	1,950°F.	24	5.3
	−60 mesh mill scale	2,050°F.	33	4.9
	With 25% blended fines	2,050°F.	33	5.3
	25% blended fines and 2% NaCl	2,050°F.	33	6.05

[a] In all cases prepared ore was charged in 1 in. paper tubes 12 in. long, packed in coke-limestone reducer in 12 in. diam. metal saggers.

[b] Fines used were scale from a continuous annealing furnace, all −325 mesh.

Table 2

HOW CARBON CONTENT[a] IS CONTROLLED

Method	*Temperature*	*Hours at temperature*	*Reducer*	*Carbon content, %*
Varying holding time at temperature	1,950°F.	20	Coke	0.68
		24		0.95
		30		1.42
		36		1.46
Varying holding temperature	1,900°F.	12	Coke	0.12
	1,950°F.			0.46
	2,000°F.			0.82
	2,050°F.			1.02
Varying proportions of ore to reducer	2,000°F.	15	Coke—twice ore weight	0.95
			Coke—equal to ore weight	0.76
			Coke—half of ore weight	0.07
			Coke—third of ore weight	0.03
Varying the kind of reducer	1,950°F.	24	Coke	0.95
			Coke+10% coal	0.61
			Coke+20% coal	0.38
			Coal[b]	0.46
			Pack carburizer[c]	1.13

[a] In all cases −60 mesh mill scale was charged in 1 in. paper tubes 12 in. long, and packed in solid reducing agent in 12 in. diam. metal saggers.

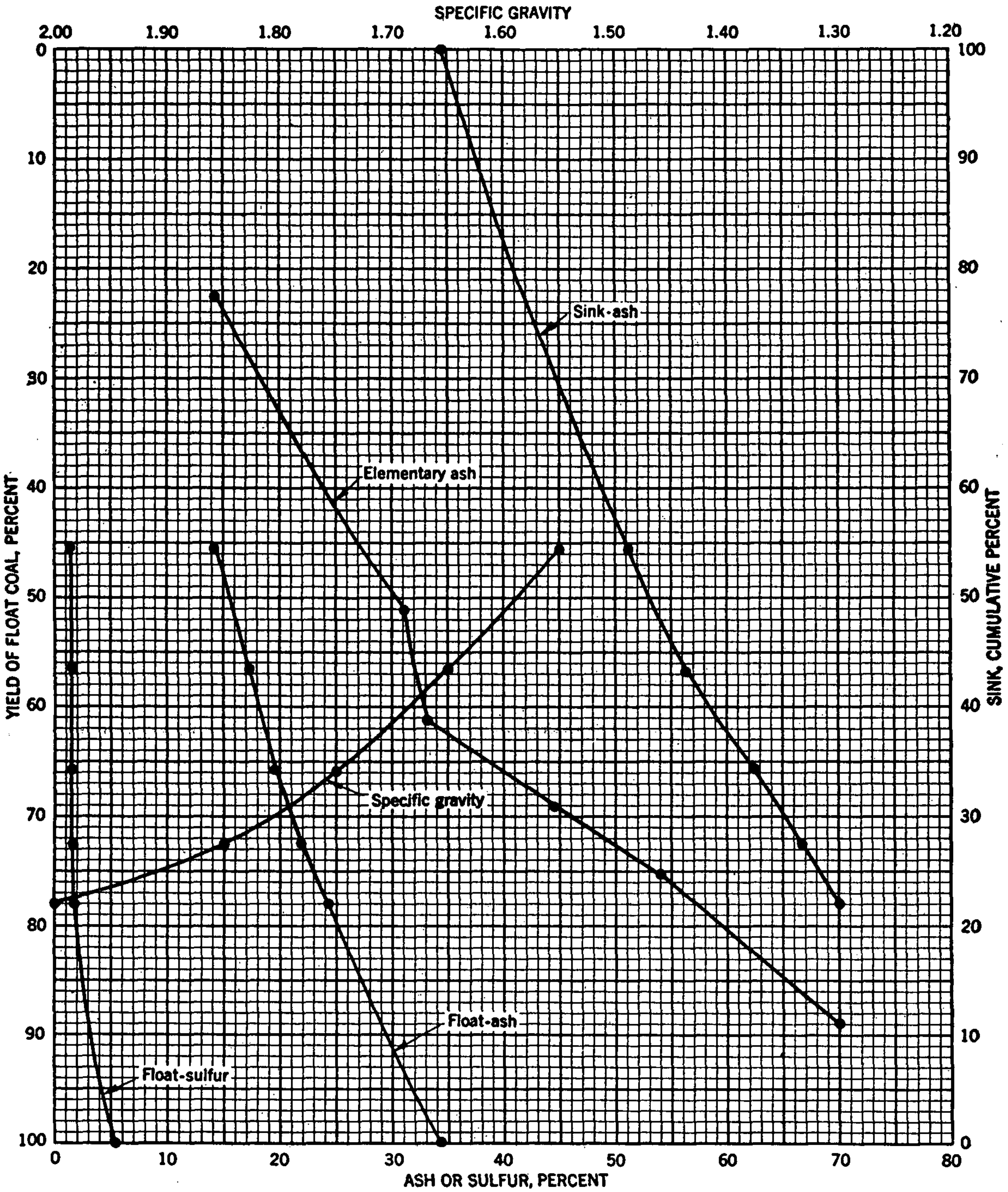
[b] High volatile bituminous coal.

[c] "Pearlite S" solid carburizing compound.

Mill scale is an ideal raw material after cleaning on the superconcentrator.

When hematites are available at a sufficiently low price it may be more profitable to carry out a magnetizing roast and then to clean the magnetic oxide with the superconcentrator.

The choice between these three materials will be determined by their price, special treatment costs for this process, and the suitability of the particular ore in producing the desired density and carbon content in the finished article.



Specific gravity fractions	Elementary data				Computed cumulative data					
	Weight grams	Percent			Float, cumulative percent			Sink, cumulative percent		
		Weight	Ash	Sulfur	Weight	Ash	Sulfur	Weight	Ash	Sulfur
Float - 1.55		45.6	14.2	1.48	45.6	14.2	1.48	100.0	34.3	5.24
1.55 to 1.65		11.1	31.1	1.83	56.7	17.5	1.55	54.4	51.2	8.39
1.65 to 1.75		9.2	33.2	1.87	65.9	19.7	1.59	43.3	56.3	10.08
1.75 to 1.85		6.5	44.5	2.33	72.4	21.9	1.66	34.1	62.6	12.29
1.85 to 2.00		5.6	54.0	2.87	78.0	24.2	1.75	27.6	66.8	14.63
Sink - 2.00		22.0	70.1	17.63	100.0	34.3	5.24	22.0	70.1	17.63

Fig. 13. Washability chart of composite of Barro Branco bed samples crushed to 38 mm. top size. Size 0.7 – 0.15 mm. increment.

Figure 3. Typical microstructures of controlled density steel: (1) Carbon, 0.32%, density 4.2 g. per cc.; (2) carbon 0.92%, density 5.13 (3) carbon 1.15, density, 6.6. 500X. All made from mill scale.

In many instances, controlled density steel articles are produced in usable form. In such a case the allowable level of impurities such as sulfur, phosphorus, silica or titanium oxide is determined by the service required of the article. The effect of these impurities in normal steel-working processes is not a factor. For example, if a certain part is not going to be hot worked, hot shortness due to a high sulfur content may be of no importance. The allowable sulfur content would then be determined by the mechanical or other properties required in the finished article.

This means that many ores which do not make desirable blast furnace feed, such as pyrite cinders and titaniferous magnetites, are perfectly satisfactory for this process. Steel has a density of about 7.8, depending on its composition and working history. Controlled density steel can be produced in a density range from about 1.0 to 7.2 directly from the reduction furnace. This new engineering material's future applications have been divided into two classes: light steel, density from about 1.0 to about 6.5 and full-density, hot-worked material with the same density as normal steel.

When reduction is carried out by packing in solid carbon, the reducing agent is carbon monoxide generated from the hot carbonaceous reducer. Carburizing compound, coke dust, coal charcoal, or even sawdust may be utilized as the solid reducing agent. About 10% by weight of limestone must be added to most solid reducing agents. This material takes up sulfur and prevents it from being absorbed by the steel.

Table 3

EFFECTS OF ORE CHARACTERISTICS, TIME, TEMPERATURE AND REDUCER ON DENSITY

Ore	Iron, %	Silica, %	Oxygen loss, %	Extruded density	Per cent change in density[a]							
					Hydrogen reduction				Carbon monoxide reduction			
					1,850°F.		2,000°F.		1,850°F.		2,000°F.	
					1 hr.	3 hr.	1 hr.	3 hr.	1 hr.	3 hr.	1 hr.	3 hr.
Cornwall magnetite	60.0	4.3	23	3.15	−24.0	−23.5	−22.5	−23.7	−42.5	−31.5	− 7.6	−28.8
Beneficiated hematite	60.5	9.1	26	2.75	−20.0	−16.0	−27.6	−20.7	−31.2	−27.3	−22.9	−31.3
El Pao	68.5	1.2	26	3.25	−24.1	−19.5	−33.5	−20.2	−53.0	−38.0	−45.0	−38.0
Pure magnetic	72.5	—	26.5	2.60	+ 4.0	+51.4	+ 7.2	+12.8	− 5.4	+ 7.2	+52.0	+28.0
Steep Rock hematite	54.4	4.0	28	2.70	−14.4	−10.0	−11.3	−16.0	−20.0	− 6.3	0	−11.0
Oliver hematite	54.5	12.5	23.5	2.40	−18.3	−21.3	−13.5	−18.0	−20.2	−16.6	+31.0	−11.5
Pure hematite	70.0	—	29	2.70	+36.0	+39.1	+48.5	+34.7	+19.6	+38.5	+31.1	+32.4
Hematite 20%, magnetite 80%	71.3	—	27	2.60	—	+19.0	+30.3	+33.0	− 5.7	+ 6.0	+34.2	+26.2

[a] If no change in sample volume takes place during test, density will decrease by percentage equal to oxygen loss. Swelling ores show greater decrease in density than their oxygen loss. Shrinking ores show less decrease in density than their oxygen loss. Increase in density results when volume shrinkage is more than enough to counterbalance the oxygen loss in weight.

Table 4

EFFECTS OF SIZE DISTRIBUTION, PARTICLE SHAPE, GRINDING METHOD ON DENSITY[a]

Method of grinding	Screen size distribution, per cent								Bulk density, g. per cc.	Surface area	Extruded density, g. per cc.	Final density, g. per cc.
	+28	+48	+60	+80	+100	+150	+200	−200				
1. Disc grinder	—	—	0.3	14.9	14.5	13.8	19.0	37.3	2.83	29.8	3.37	6.87
2. Pan mill	—	—	2.5	14.6	15.4	12.6	17.1	37.7	2.68	29.5	3.61	6.30
3. Rolls	—	—	0.3	13.8	17.9	19.8	14.9	33.4	2.68	27.7	3.30	6.06
4. Ball mill	—	—	—	11.0	11.7	22.4	19.7	45.0	2.75	33.3	3.38	5.06
5. Jaw crusher	6.5	57.0	22.7	13.6	—	—	—	—	2.78	6.8	3.21	4.21
6. Screened from jaw crusher	—	100.0	—	—	—	—	—	—	2.64	5.7	3.21	3.67

[a] El Pao Venezuelan hematite at 68.5% iron ground as shown, extruded with 0.5% starch binder packed in coke limestone reducer in metal sagger and held 24 hr. at 2,200°F.

The prepared iron ore is poured into a suitable mould and the mould is then vibrated. The main purpose of the mould is to form the loose fine ore into the desired shape and to hold it in that shape until the coke has been packed around it. After the mould is supported by the coke it does not matter whether the mould is burned away, since the fine, close-packed coke will now support the ore in the desired shape. It is possible, therefore, to use moulds of paper. Such moulds char during reduction but produce usable shapes.

Moulds must also be porous so that the reducing gas can pass through them to reach the iron oxide. Porous sand moulds, graphite and shell moulds of resin-bonded sand are all satisfactory.

The tunnel kiln process as carried on at the Foundation's Oakville pilot plant is quite simple. If bars are being produced, the prepared ore is poured into paper tubes. The closed tubes are then placed in a metal container or "sagger" and the fine reducing mix of coke and limestone is added (Figure 4). A 1 in. layer of reducing mix is placed over the top of the tubes and the sagger is sealed to exclude as much air as possible.

Saggers may be of clay or of alloy metal. They must be gas-tight and able to withstand the reduction temperature. The saggers are placed on kiln cars and pushed into the charging end of the kiln. In larger kilns many saggers can be charged on a single car (1). When the sagger and contents have cooled down to room temperature, the sagger is opened and the product removed.

The Oakville kiln is 65 ft. long and holds 26 cars. It is electrically fired by 12 "Globars" and will hold five carloads at 2,300°F. Pushing one car into the kiln each hour provides 5 hr. at temperature. A 3-hr. push will give 15 hr. at temperature.

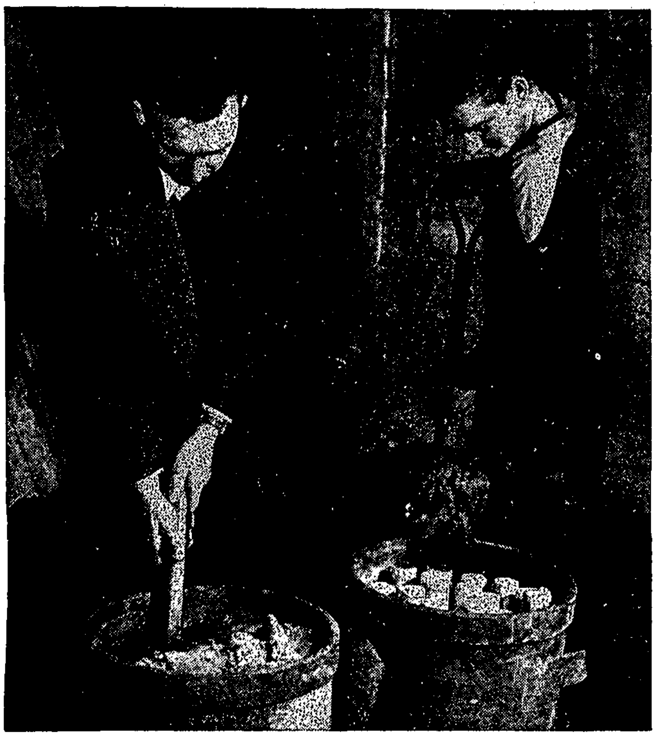


Figure 4. Loaded tubes are placed in enamelled sagger (at Oakville pilot plant, these are now made of ore directly reduced in desired shape in same kiln in which they are used). These containers' capacity is 24 tubes. When filled with coke-limestone reducer they are ready for reduction. On left, low-density steel bars are being removed from sagger after their trip through kiln.

The main characteristics of the new engineering material provided by this direct reduction process are low cost and light weight. The mechanical properties vary in proportion to density, (Figure 5). It must be emphasized that Figure 5 shows the minimum ultimate tensile strength as obtained on tensile test bars for one particular analysis in the "as-produced" state.

A peculiarity of the controlled density steel process is that the mechanical properties depend somewhat upon the shape of the finished article. During reduction and sintering operations different thicknesses will reduce and sinter at different rates. This fact must be kept in mind when evaluating the strength of light steel. It is desirable where possible, to measure the mechanical properties of the actual shape rather than those of specially prepared test bars.

The mechanical properties of the very low density material are comparable to those of wood or plastics. At the same densities, the ductility is considerably below that of magnesium or aluminium. The price per unit volume is also considerably lower than magnesium or aluminium.

As with all design problems, this would be a matter of comparative costs of the two different designs. A study of the costs and mechanical properties of, for example, controlled density steel and magnesium would be needed.

Light steel will rust more rapidly than normal steel but in most uses all steel is protected by paint or plating or enamel. The relatively rough surface of light steel as it comes from the furnace is an ideal surface for painting and particularly for enamelling. It is easy to machine. Nails and screws can be driven into it by hand.

As with ordinary steel, alloying, raising the carbon content, and heat treatment can all be used to increase properties at a given density.

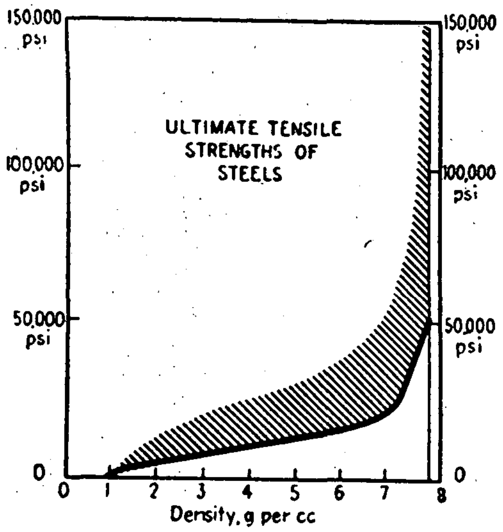


Figure 5. Solid line represents minimum tensile strength of satisfactory "low-carbon" bars "as produced." Shaded area indicates tensile at any density can be improved by increasing carbon content, heat treating or other means.

Table 5

LIGHT STEEL ANALYSES[a]

Ore	*Additions*	*C*	*S*	*Mn*	*Si*	*Cr*	*Ni*	*Co*	*Va*
Mill scale		—	0.010	0.39	0.38	—	—	—	—
Gallivare magnetite		—	0.005	0.06	0.02	—	—	—	—
Mag Iron Co. magnetite		—	0.011	0.06	0.23	—	—	—	—
Grangesberg magnetite		—	0.014	0.05	0.06	—	—	—	—
Mill scale	Nickel and chromium powder	0.14	0.007	0.17	0.34	2.00	24.14	—	—
Mill scale	Chromium powder	1.26	0.015	0.36	0.31	0.99	—	—	—
Mill scale	Cobalt powder	1.07	0.002	0.39	0.34	—	—	26.72	—
Mill scale	Vanadium powder	0.75	0.012	0.07	0.87	—	—	—	0.03
Mill scale	Nickel and chromium powder	2.42	—	—	—	8.1	5.90	—	—

[a] Ores ground to −60 mesh and reduced by packing in paper tubes in a sagger in coke limestone reducer and holding 36 hr. at 2,000°F.

With alloying elements such as cobalt and nickel, the metal may be added in oxide form. The oxide will be reduced and the resulting metal will alloy with the light steel during the process. An extensive investigation of methods of preparing alloys is under way and will be reported on later. Figure 6 shows two alloy compositions and Table 5 lists some of the alloys already produced.

The impact strength of low-density steel is limited. Therefore it is sometimes desirable to reinforce the material by placing rods of steel or other metals in the ore when it is moulded into shape. The lightweight hatchet, Figure 7, has a ⅜ in. steel bar in the centre of the handle. The edge of this hatchet and the blade of the knife also shown were hot forged and have the required properties for such cutting edges. The bond between such reinforcing rods, bars or sheets and the controlled density steel is excellent. As the very low-density steel slab in Figure 7 indicates, nails and screws can be driven with ordinary hand tools and the material can be worked with wood-working equipment.

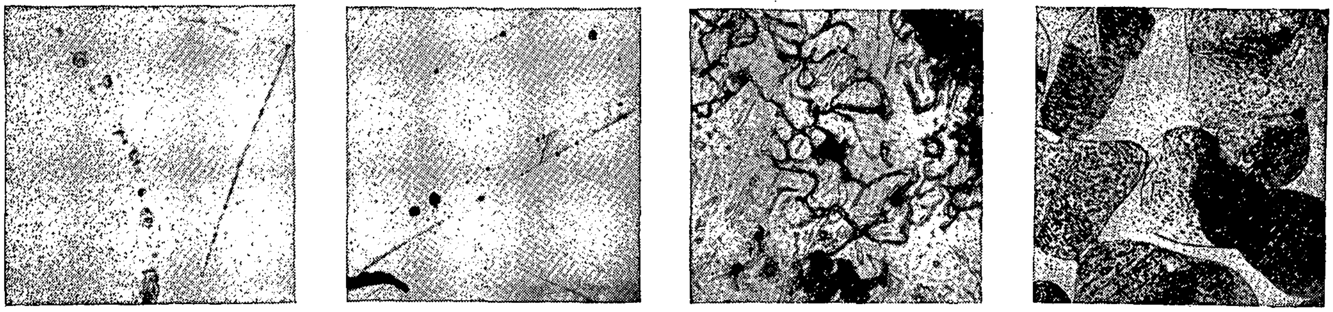


Figure 6. Types of alloy structures. At left, stainless steel: 0.14% carbon, 0.007 sulfur, 24.14 nickel, 2.00 chromium-density, 5.7 g. per cc. Note: inclusions in structure at far left do not occur mainly at grain boundaries. At right, copper-impregnated steel: 1% copper added to ore. Copper is in grain boundaries (phase contrast). At far right, 15.00 copper added. White constituent is copper in this structure. 375X.

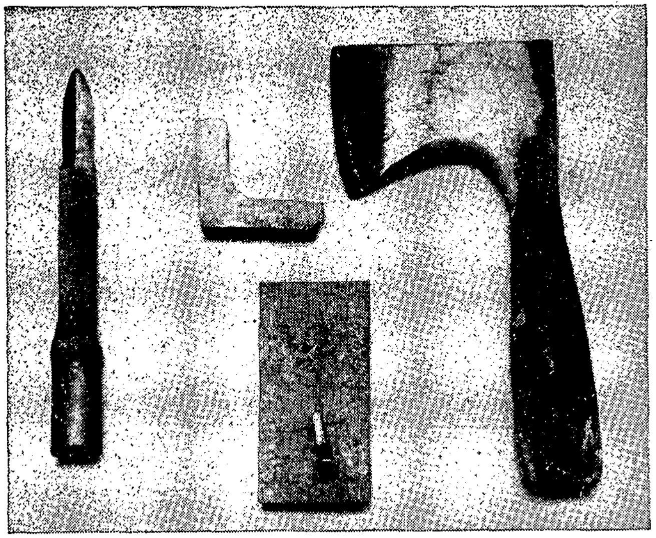


Figure 7. At left, knife blade forged to full density from 3.5 g. per cc. density and heated in blacksmith's forge. Top centre, bracket of 1.15 density steel. Bottom center, slab of light density steel into which a wood screw has been driven with regular screwdriver. At right, light-weight hatchet with ⅜-in. reinforcing steel rod in handle. Blade forged to full density and hardened.

A "sandwich" construction has been produced in which two thin sheets of stainless steel are placed ½ in. apart and the space filled with fine iron ore. One sheet is pierced with ⅛ in. holes placed about 1 in. apart to allow access of reducing gases. The light steel filler has a density between 1.0 and 1.5 per cc. The bond to the stainless sheets is satisfactory.

One of the annoying characteristics of any direct reduction process is that reduction takes place from the surface inward. It is quite common to find experimental sponge iron or controlled density steel parts which have not been reduced completely through. Sufficient time has not been allowed to complete reduction through the piece. This can be used to advantage in making hollow shapes. For example, Figure 8 shows experimental pipes made by filling a cylindrical mould completely with fine iron ore.

This solid bar was reduced at such a fast cycle that only the outer skin was reduced to controlled density steel. Afterwards, the unreduced iron ore in the centre was spilled out, leaving a hollow pipe. Under such circumstances the carbon content is always low.

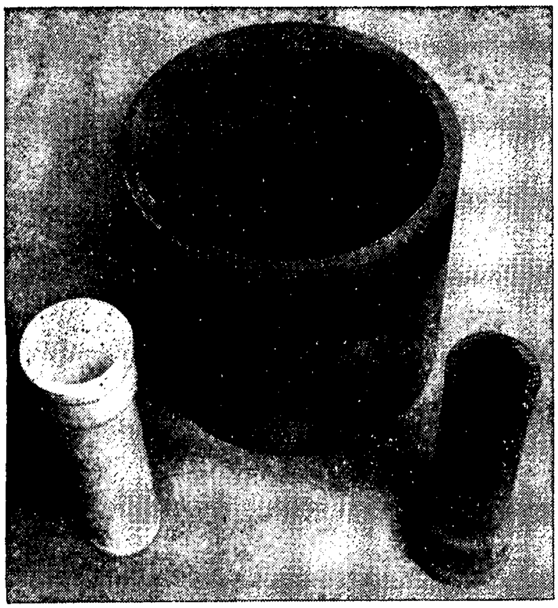


Figure 8. Light-weight steel saggers, enamelled, are being used in the Ontario Research Foundation's pilot production of controlled density steel. Together with the pipe (shown enamelled at left) which weighs ⅓ as much as cast iron, these hollow forms are made by giving a mould full of ore a fast pass through the kiln. The unreduced ore in centre is dumped out and the inside diameter reamed by hand.

If a higher carbon content is desired it is possible to re-pass the part through the furnace a second time after removing the unreduced centre. The density will then remain about the same but the carbon content will rise. Any unreduced material adhering to the pipe's inside surfaces is easily removed—hand reaming has been successfully employed.

Such pipes can be enamelled inside. Their weight is about one third that of cast iron pipe. They can carry corrosive liquids and should cost less than cast iron.

Tests made under gas pressure as high as 2,000 psi have indicated that very low-density steels (under 3.0) will leak to some extent.

This technique has proved useful in producing the hollow, light-steel saggers shown in Figure 8. Coated with a suitable high-temperature, vitreous enamel, these saggers withstand the temperatures in both the controlled density steel and sponge iron processes. Heat conductivity is greater than those made of ceramics. The cost is far below high-temperature, alloy steel saggers.

LAYER OF ORE SEALS SAGGERS

A simple method of sealing the saggers has been found. Instead of ceramic or metal lids with clay seal, a loose layer of finely ground iron ore about 1 in. thick is placed on top of the charge. During the process, the bottom portion of this layer is reduced to metallic iron. During the cooling portion of the cycle, contraction of the gas inside the sagger pulls some air into it, no matter how good the seal. However, when an ore seal is used, the entering air is pulled through a layer of metallic iron. This layer is re-oxidized to iron ore. This provides satisfactory protection against oxidation of the light steel further down.

As stated previously, control of swelling and shrinking is accomplished by the choice of ore, control of grain size distribution, blending of ores with different swelling characteristics, reduction temperature and/or treatment time. It is possible to control shrinkage in "moulded" parts to a remarkable degree. For example, Figure 9 (at lower right) shows tensile test bars produced from shell moulds with less than 1% shrinkage. Such control can be maintained from piece to piece within practical limits and makes possible production of such articles as those in Figure 9 directly from shell moulds.

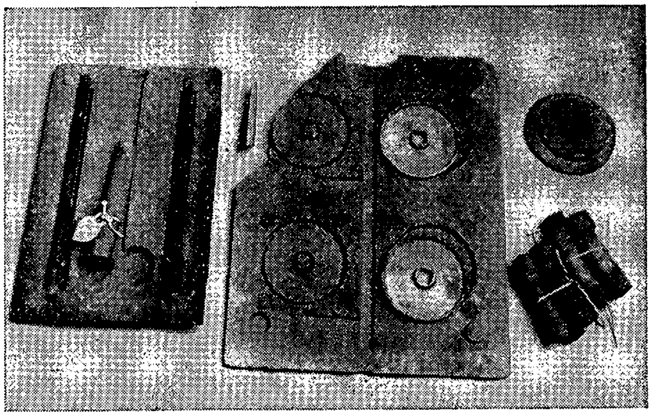


Figure 9. Tube mould at left and ratchet gear mould in centre were designed for liquid metal but were successfully used for the parts shown, made of ore. Oil seal at top right is ready for service immediately after drilling of bolt holes and painting. Tensile test bars at lower right were made in resin-bonded shell moulds.

SHELL MOULDING METHOD PREFERRED

The shell-moulding technique is the preferred way of producing light steel parts to close tolerances. The surface finish is as good as a cast iron finish. It can be considerably improved by using fine sand for the mould and finer maximum screen size in the ore. Experiments are being carried out with resin-bonded test moulds, replacing the sand with a reducing agent so that the mould performs a useful function in reduction.

In the production of simple parts such as bars, tubes and angles, the extrusion technique[2] developed for iron ore can be used. No supporting moulds are then needed. A binder such as starch will give sufficient green strength so that the extruded shape can be handled and packed in the sagger. The cereal type of binder provides some reducing action in the process. Other binders such as low-sulfur tar or thermo-setting resins can also be considered.

It appears possible to get slightly better mechanical properties in controlled density steel parts of the same analysis and density than in corresponding pressed powder parts. This is probably due to the fact that sintering takes place during and immediately following reduction. There is no chance for formation of any oxide films on the iron particles. The competition with iron powder would seem to be entirely where only a small amount of machining is necessary to provide the same tolerances achieved directly in pressed powder parts. The tolerances that can be expected from controlled density steel are somewhere between precision casting and ordinary casting. They do not approach the tolerances of pressed metal parts.

In order to replace a powdered metal part, the cost of finish machining on the controlled density part must

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[2] Cavanagh, P. E., "Pelletizing of Iron-Bearing Fines By Extrusion", AIME Blast Furnace, Coke Oven and Raw Materials Conference Proceedings, vol. 9, p. 54, 1950.

be less than the savings accomplished by this new process' ability to eliminate some of the production steps and equipment common to powder metallurgy. As an example, where a simple shape of light steel can be produced as a blank for cold coining, there are possibilities of significant savings.

Because the voids in controlled density steel are not interconnected, it is possible to reheat and hot work the material without any unusual provisions to protect the surface against oxidation. For example, the forged knife blade and hatchet of Figure 7 were heated in an ordinary blacksmith's forge and hand forged on an anvil. Bars and slabs of controlled density steel can be hot worked to the full density of normal steel. The new material cannot be hot worked in the same way as ordinary steel. Any attempt to put it through standard rolling mill passes will simply smash the cellular structure. In the first stages of hot working the structure must be gently compressed.

The best method of working a round bar is by hot swaging at about 1,900°F. The best method of rolling sheet is either in a closed pass mill or a mill equipped with following edge mills. The hot working of controlled density steel bears many similarities to the hot working of tungsten and molybdenum. There is one very important difference. It is possible to hot work controlled density steel above the welding temperature (about 2,400°F.) so that voids will be welded closed during the process.

Some of the voids will contain impurities or have surface conditions so that they will not weld. At some point in the hot working of the material these unwelded voids will work to the surface. The surface will then appear badly checked. However, the next pass, still above the welding temperature, will produce a good, hot-worked finish.

Figure 10 shows the steps in production of cold drawn bar stock. If long bars are desired, the short bars produced by reducing iron ore in paper tubes packed in saggers, may be joined. They are then welded together at the working temperature of about 2,350°F. in a swaging machine. Bars hot worked from 1.050 to .395 in. diam. can attain the full density of normal steel. They will stand a 180° cold bend test without any sign of cracking. After cold swaging down to 0.260 in. they have the same mechanical properties as a cold drawn bar of ordinary steel of the same analysis (see Table 6).

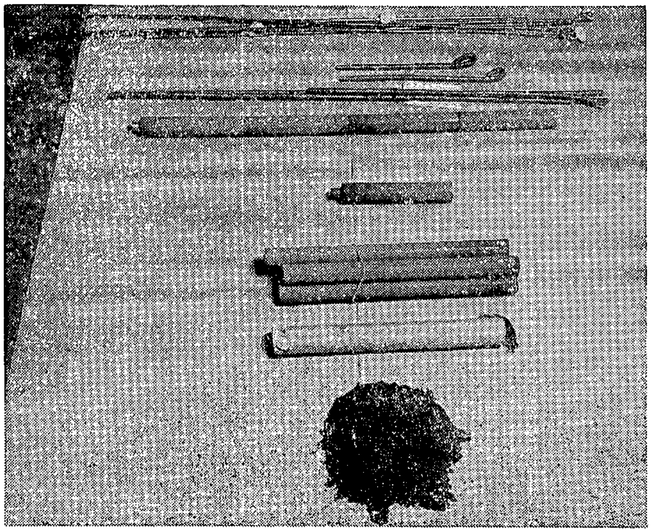

Figure 10. Steps in production of normal density bar stock: from bottom, −60 mesh mill scale, paper tube filled with mill scale, reduced steel bars (density 4.3; carbon, 0.45%), bar with male and female joints cut, joined length of bar (1.050-in. diam.), bars hot swaged at 2,400°F. to 0.375-in. diam. (note cold bends), bars cold worked from 0.375 to 0.260-in. diam.

To obtain competitive costs for plain carbon steel bars it is obviously impractical to carry on this process in the batch-type, tunnel kiln operation. Approaches to the continuous production of steel bar stock are illustrated in Figure 11.

Some of the unique possibilities of controlled density steel will be apparent when it is realized that alloy additions can be made, for example, only to the surfaces of sheet slabs. Controlled rolling of such slabs may well provide a satisfactory full-density surface while leaving the interior at low density. Decorative stainless sheet may be possible with satisfactorily corrosion-resistant surfaces only one quarter to one half as heavy as ordinary sheet. These possibilities are being studied.

At present, it appears that producing normal steel by this method would be more expensive than purchasing it in a steel producing district. In many remote locations, however, the difference in production cost may be more than offset by freight rates.

The problem in continuous production of normal steel from controlled density steel at a reasonable cost is very largely a matter of furnace design. If an extremely efficient furnace can be designed to handle large tonnages rapidly, it may be possible in future to revise present estimates of the production costs of such steel.

At present, with existing furnaces, this method for hot working controlled density steel to produce normal steel should be particularly applicable to production of some alloy steels, to production in remote locations and to provide a completely controlled source of small tonnages of steel.

Table 6

PROPERTIES AFTER WORKING TO FULL DENSITY

Carbon content, %	*Working history*	*Hot swaging temperature*	*Yield point, psi*	*Ultimate tensile, psi*	*Elongation, %*	*Redijn area, %*
0.38	Hot swaged 1.050 to 0.375 in.....	1,850°F.	36.400	51.800	22.5	32
0.85	Hot swaged 1.050 to 0.375 in.....	1,850°F.	68.600	98.200	1.0	4
0.38	Hot swaged 1.050 to 0.375 in.....	2,400°F.	38.000	53.500	15.0	44
0.68	Hot swaged 1.050 to 0.375 in.....	2,400°F.	51.000	86.000	12.0	22
0.38	Hot swaged 1.050 to 0.260 in.....	1,850°F. 2,400°F.	37.900 38.700	52.500 62.100	5.0 10.0	52 8
0.38	Cold swaged 0.375 to 0.260 in.....	—	—	103.000	—	30
0.95	Hot swaged 1½ in. to ¾ in. cold rolled to ½ in..	1,950°F.	—	141.800	—	5
0.88	Hot swaged 1½ in. to ¾ in. cold rolled to ½ in..	1,950°F.	—	123.400	—	9

In one respect the fact that "controlled density steel" is steel, and is called steel, is an obstacle to realization of its possibilities. The natural inclination is to compare light steel strengths unfavourably with ordinary steel strengths. If, instead, it were announced that: (1) a whole new family of metals had been discovered that ranged all the way from lighter than magnesium to the weight of steel; (2) that these metals were all readily available in huge quantities from operating mines; (3) and that some of the lighter metals were even cheaper than cast iron, consideration of these materials would start from a different angle.

In many applications cast iron is used in spite of its mechanical properties because it is the cheapest metal available. Cast iron soil pipe, for example, could be made of light steel, using the same patterns and at lower cost.

The Foundation's studies of the manufacture of sponge iron in tunnel kilns from magnetites and mill scales have established some cost figures. In Canada, such sponge iron costs between $40 and $50 a ton to produce, depending on local conditions, prices and equipment.

The extra costs above those in making sponge iron to produce light steel are in superconcentrating the ore and holding it longer in the furnace. Allowing a generous $20 per ton for these extras, the cost would be $60 minimum and $80 maximum per ton. This compares favourably with a figure of about $65 to $80 per ton (no overhead) for foundry iron in the ladle ready to pour into pipe moulds.

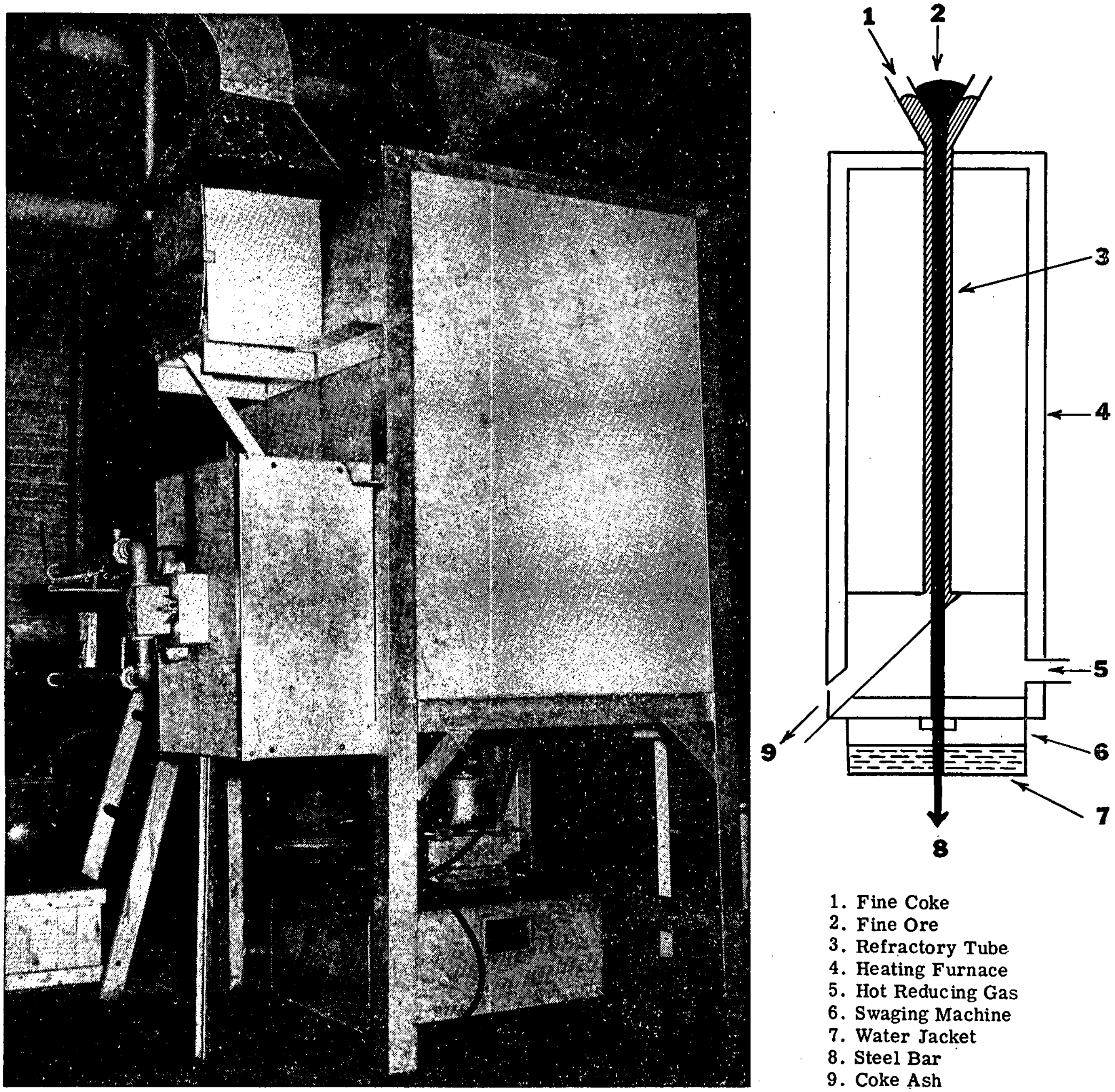

Figure 11. At left (photo), experimental vertical tube reduction furnace. Ore is fed into externally heated tube. Metal passes through cooling jacket into gas-tight discharge chamber. At right (drawing), proposed furnace for continuous reduction and swaging of steel bar stock, which would be produced ready for normal working.

Sand moulding costs will be about the same for both processes, with some advantage to the light steel because less sand is used. No gates, risers or sprues are required for light steel. The saving in cleaning and cutting costs could pay for a coat of paint or possibly even porcelain enamel.

So the main difference is in the cost of the metal per pipe. But if the light steel pipe has a density of 2.7 (the same as aluminium), there will be nearly three times as many pipes for the same weight of metal.

The sponge iron investigations from which this discovery grew were carried on by the Ontario Research Foundation under the Ferrous Metallurgy Sub-Committee of the Ontario Research Council. The sponge iron work was financed by the Government of Ontario and by contributions from the Steel Co. of Canada, Atlas Steel, Dominion Foundries & Steel and Algoma Steel. The assistance and advice of these organizations has been invaluable in the controlled density steel investigation.

The help given in hot and cold working of controlled density steel by Cleveland Wire Works of General Electric, by Burlington Steel, Atlas Steels and the Steel Company of Canada made available production equipment and advice which could not have been duplicated. In addition, many other companies and persons have helped make possible a very rapid advance in this new process.

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Notes on the Ugine-Séjournet Extrusion Process

J. SÉJOURNET

EXTRUSION PRESSES

There are three main types of steel extrusion presses in operation at present, or about to be put into service.

1. *Presses for non-ferrous metals, modified for the extrusion of steel*

These presses may be found in plants producing expensive, highly alloyed steels. The modifications have been directed to ensuring the right extrusion speed, the separation and sawing-off of the butts and the proper handling of the billets and extruded products.

Generally the speed of the operation is still too low. However, these presses may be used to process steel, but because of their low rate of production and high costs, they are reserved for the extrusion of expensive metals and are not used with ordinary steels.

A British manufacturer who had a press in operation on non-ferrous metals made a study to see if it could be modified for the extrusion of ordinary steel shapes and tubes. This study showed that it would be cheaper and quicker to build an entirely new press for the purpose than to start modifying the existing one.

2. *Presses specially designed for steel extrusion*

The general principles of these presses—design, sturdiness and safety—follow the pattern of the older ones.

The following problems have been specially studied and solved: the tool carrier and the guiding of the extrusion container have been modified; one or two powerful automatic saws for cutting the butts have been incorporated; the operating process may be simplified for rounds and tubes, and is otherwise normal; a special hydraulic equipment has been provided to permit the use of very high speed without hydraulic vibrations. The speed of the ram may be controlled from the start or during the operation.

Extrusion operations, such as billet loading, dummy block loading and handling of extruded products, are rapid and automatic.

Loading of the glass lubricant, automatic cleaning of the extrusion tools, cooling of the mandrel and the ram, are efficiently carried out. The extrusion container and the ram can be changed in less than an hour, while the dies can be changed in a few seconds. New devices provide for the removal of the discards in case of troubles, thus preventing production delays. The extruded products are placed on a run-out table. The speed of the operation cycle has been improved, by means of increased speed of the ram, a moving container, a die-holder easy to change. The cycle has thus been reduced to 30 seconds.

3. *New principles for special steel extrusion presses*

Future presses will most probably differ considerably from the already highly improved ones now in operation.

The trend will necessarily be towards fast-operating machines. Designers are ambitiously seeking to substitute displacements along the longitudinal axis of the extrusion press by lateral movements, so that new dies and tools can be moved in after the completion of each job. This would save 20 to 30% of the total operating time.

Tests are being made on such equipment at the plant of the Comptoir d'etirage at Persan (Seine et Oise), and in the United States. The presses have required careful studies and their adjustment is delicate, particularly for maintaining correct centring of the tools during the extrusion of tubes.

The question has been raised for some time past as to whether it will be practicable to use mechanical presses for the extrusion of steel. It may be possible to find a use for such presses in the production of great amounts of similar products, but there is as yet no technical experience in this field. The main advantage of mechanical presses over hydraulic ones is that they cost less and are cheaper to operate but they present some drawbacks which might affect the quality of the products. These drawbacks include the fact that the working pressure and speed are not constant, while difficulties arise in adjusting the stroke of the sliding block.

OPERATIONS

The size of the billet is first determined according to the extrusion specifications: grade of steel, extrusion temperature, extrusion ratio, etc. Circular saws of large diameter are used to cut the billets to the required lengths. Hot sawing has not yet been tried because the ends of the billets should be quite smooth and perpendicular to the axis.

Then, the billets are preheated to a suitable temperature in the furnace; gas furnaces are preferred.

Heating from 1,150 up to 1,250° C. being difficult without scale, final heating is performed at the present time in molten salt baths. This offers several advantages, such as heating without oxidation, removal of superficial preheating oxides, fast and regular heating which simplifies operations, protection of the material with a coating of molten salt.

There are, however, economic disadvantages in this heating process such as: high electric power consumption in the furnaces, heavy maintenance costs, and the introduction of a supplementary operation in the cycle.

Tests are therefore being made, which should result in considerable improvement of the heating methods.

Tube production by the Ugine-Séjournet process requires the forming of a hollow blank in a separate press before extrusion, with a diameter slightly over the size of the mandrel of the extrusion press. Better results are obtained with upsetting and the use of a glass lubricant.

Shearing may be done either on the same press or on a small separate one.

In some special cases, especially for thick walled tubes, piercing may be done on the extrusion press itself, but this considerably reduces the press output. For large productions it is better to perform the two operations on different presses.

Straightening and subsequent operations are made on special machines, which also remove the oxide layers mixed with the glass lubricant.

When a special finish is required for the extruded products, oxides and glass are removed by molten soda or hydrofluoric acid pickling

PRODUCTION

The 1,500-ton press of the Comptoir d'etirage at Persan, originally permitted 15 pushes per hour. In 1950 the average reached 20, in 1951 25 and at the beginning of 1952 36, with special performances of 60. The presses installed in the United States can reach 100 pushes per hour.

With mechanical presses the rate could be speeded up and might exceed 120 pushes per hour.

The total output naturally depends on the weight of the billets put in the extrusion container. That weight in turn varies according to the grade of steel, the extrusion ratio and the force of the press.

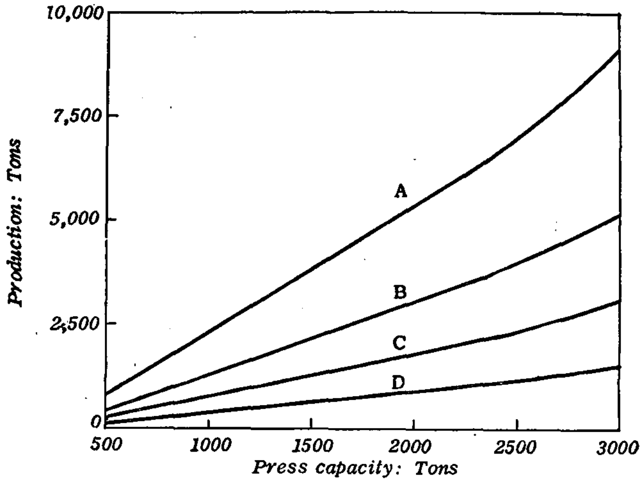
Figures 1 and 2 show the relation between maximum and minimum tonnages, in metric tons, and the force of the press, for 500 hours' work a month.

It is evident that extrusion presses are suitable for large production.

The field of application of extrusion presses for steel is almost unlimited. Stainless, chromium, nickel-chromium, chromium-manganese and more complex steels are easily extruded, and no longer raise any special problems. Ordinary low carbon and bearing steels are extruded without difficulty. Experimentally an extrusion ratio of 700 for carbon steel was obtained at Persan, that is to say that the extruded bar was 700 times longer than the billet put in the extrusion container.

High temperature, heat resisting steels of types 25/20, Inconels, Nimonics, S.816 alloy, Croloys, etc., have all been extruded at the Persan plant of the Comptoir d'etirage, as well as some high alloy steels previously considered as impossible to work. In the case of high nickel content alloys and high speed steels, some precautions have to be taken, and the extrusion ratio is limited to a value between 10 and 35.

Pure metals, including titanium, are easily extruded. Molybdenum at 1,400° C. and nodular cast iron have been extruded at Persan.



A - Maximum production for mild steel
B - Maximum production for stainless steel 18.8
C - Maximum production for high-speed steel 18.41
D - Minimum production for all grades of steel

Figure 1
Extrusion
Monthly production for 500 working hours
Number of pushes per hour: 50

A brief outline follows of the early applications of steel extrusion and of those towards which it is now directed.

Possibly because of the former activities of the Comptoir d'etirage, the first extruded products were raw products for its cold drawing plant. That application of extruded products is increasing and, as a result, several cold drawing mills are increasing their production capacity.

Close contact between designers, producers and consumers is required in order to select the shapes to be extruded. Figures 3, 4 and 5, show some of the new shapes and some examples of utilization.

Some parts of agricultural machinery are now made by extrusion; similar sections are also used as raw material for forging and stamping when they cannot be rolled easily.

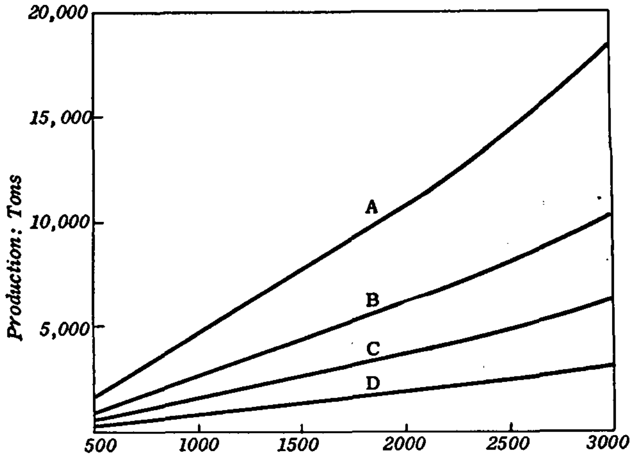
Fairly complicated products in nickel, chromium and chromium-molybdenum steels of any composition, can now be manufactured, which opens up an interesting possibility for builders. The alloyed steel sections make it possible to lighten mechanical parts by reducing their dimensions, demanding a new study of the design of parts and wholes.

Tubes with wings for boiler, hollow hexagonal bars for making nuts, stainless steel tubes for making watch-cases, etc., are products made from hollow sections.

The extrusion press also produces regular tubes in different alloys such as special alloy steels for ball bearings, heat resisting and stainless steels, chromium steels, and all tubes out of standard, or thick-walled tubes for special mechanical requirements.

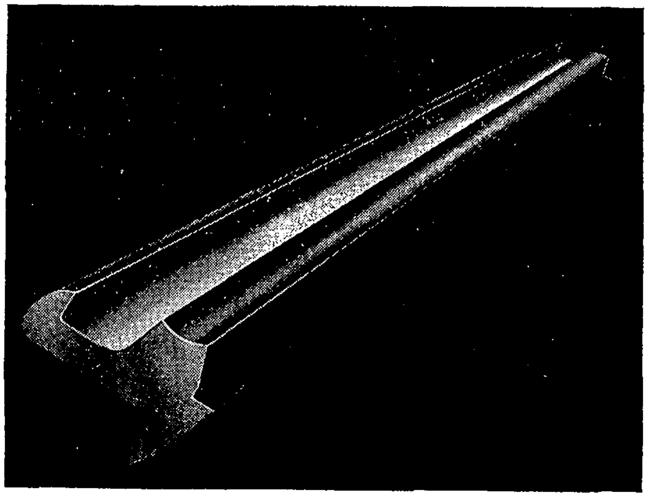
Finally, special mention must be made of the hollow turbine blades for jet engines. Their manufacture by the extrusion process from heat resisting steel is definitely of specific interest.

Bi-metallic tubes can be made by the extrusion process such as stainless steel tubes with an ordinary steel bore, or the reverse; ordinary steel bars with special steel cores, and so forth.



A - Maximum production for mild steel
B - Maximum production for stainless steel 18.8
C - Maximum production for high speed steel
D - Minimum production for all grades of steel

Figure 2
Extrusion
Monthly production for 500 working hours
Number of pushes per hour: 100



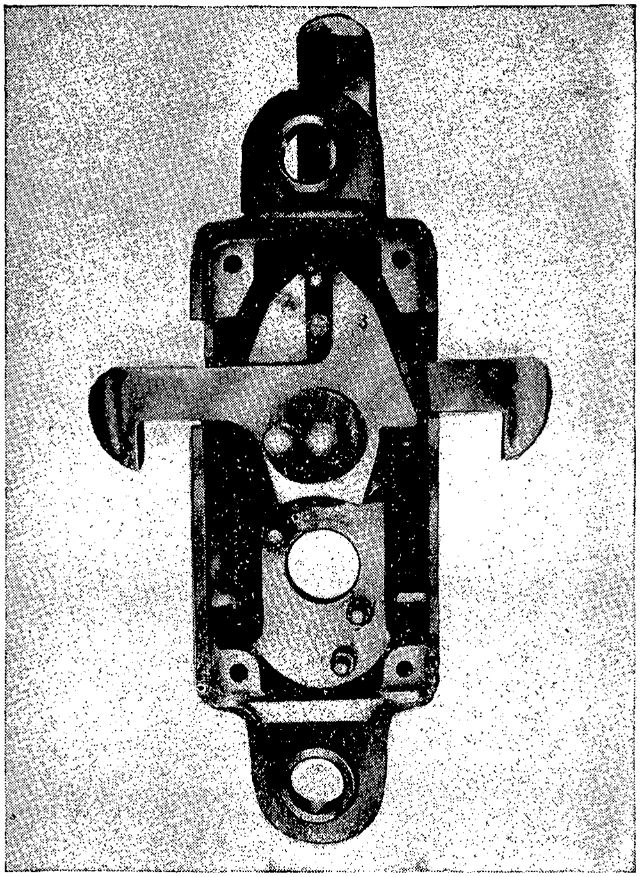


Figure 3

All the extruded parts can be made with varying thickness along the longitudinal axis, which covers, for instance, the manufacture of hollow cylinders with a conical exterior shape.

This short enumeration of extrusion possibilities is too brief to give an accurate idea of how this new process can help industry. Techniques are continually developing and improving, and when it is remembered that the Persan plant has been working for only three years, under industrial conditions which are still precarious, it will be easier to assess the future prospects for this new process for the transformation of steel and alloys workable at high temperature.

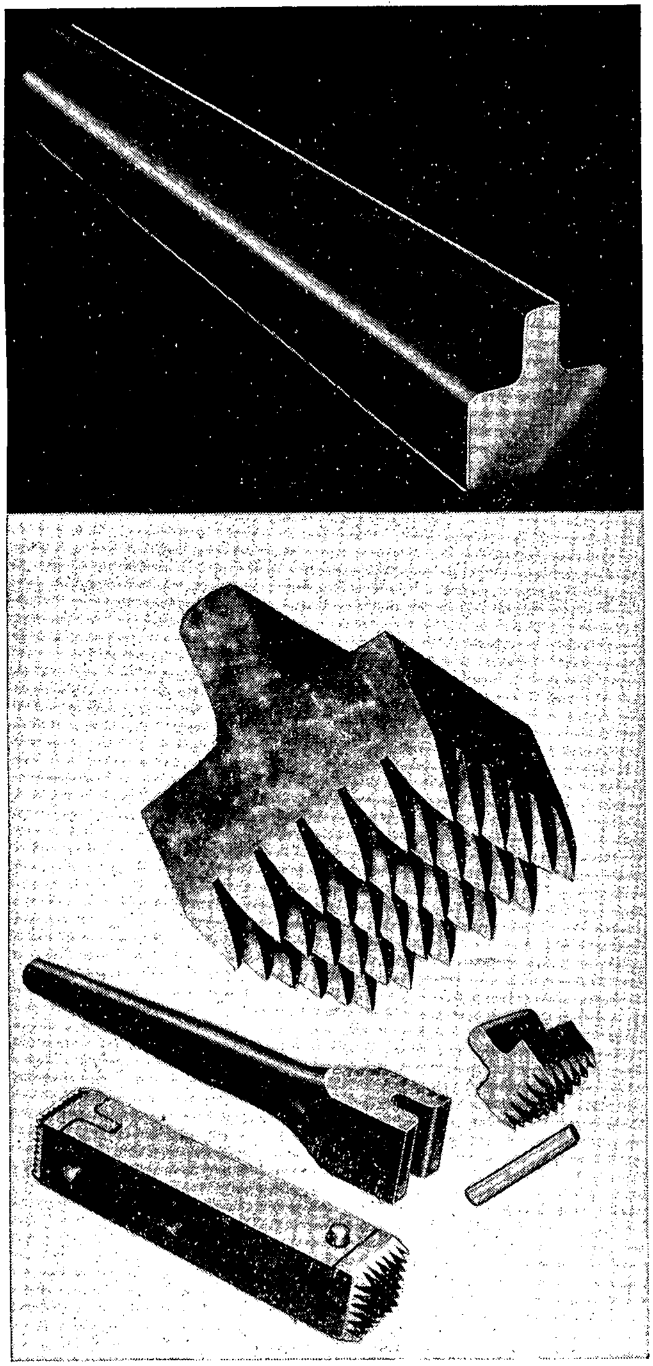


Figure 4

Production costs

While the designs of extrusion presses are being simplified and the speed of production increased, the cost is decreasing and is steadily approaching that of rolling mills.

Extrusion is of interest when special steel tubes and sections are required. The press cannot yet compete with the rolling mill in the mass production of ordinary carbon steel bars and simple sections, but in the case of unusual sections and tubes, it is always advisable to examine the advantages of extrusion as compared with rolling.

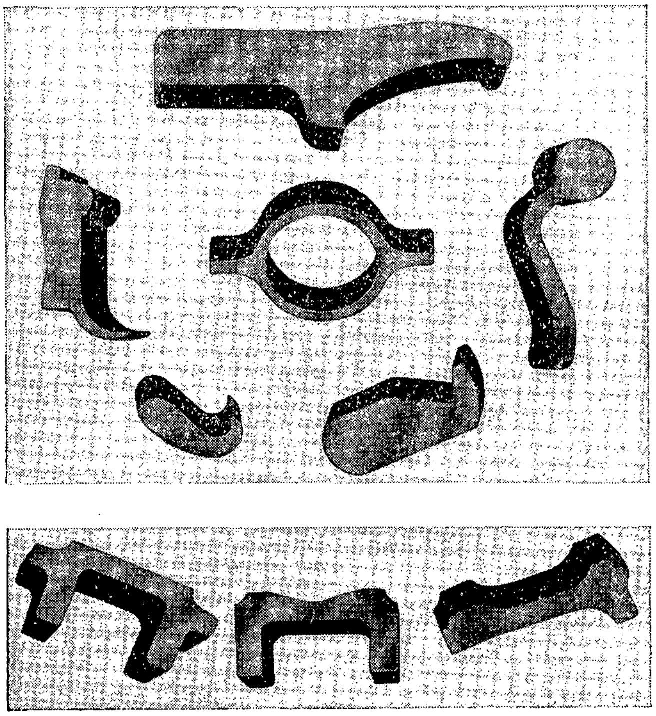


Figure 5

The investment required for the installation of extrusion equipments is undoubtedly less than in the case of rolling mill equipments. On the other hand, the production cost of extrusion is still slightly higher than that of rolling. In each particular case, a study must be made of the dominant factors, which are: period of amortization, total volume of production contemplated, and labour costs.

Many people think that the conventional hot forming equipment has now reached such a stage of perfection that any further progress will be slow, and that no radical changes are likely. They think on the other hand, that the extrusion process, which in ten years has taken its place in the field of high temperature transformation of metals, has a bright future, since those familiar with its details are aware of the innumerable improvements which technicians and experience cannot fail to introduce.

E/CN.12/293/Rev.1/Add.1
ST/TAA/Ser.C.16/Add.1

9 November 1954

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zation procedure in one operation. It has the disadvantages that the fuel-coal size, fixed by the crushing for release of maximum metallurgical-coal yield, is not well adapted to the combustion facilities available, and the blending of high- and low-sulfur coals into a common plant-feed makes it impossible to re-separate the low-sulfur part.

The second plan has the advantage that the fuel coal could be sent to market in the size best suited to the user, and the performance of the central metallurgical washery would be improved. There would be some economy in transportation, but the operating cost of the several washing plants would be higher.

In this scheme the slack, even from the Banco operation, should be sent to the central plant, since the float-and-sink data show a relatively good yield of low-ash coal in the fines. This method gives a much more flexible operation than method 1, hence the work could be easily varied to suit changing conditions; but this advantage will be offset in part by the cost of multiple small plants. Furthermore, the difference in quality of the two coals is not constant throughout the mining region. There is still another technical difficulty in that the Banco coal, though very high in ash, making for an extremely small yield of metallurgical coal, is generally a little lower in float-sulfur, and this advantage is greatly needed.

The third plan has the advantage of somewhat more flexibility than the second one. It would be based on frequent checking of the raw coal sources by sampling surveys.

Possibly, ultimate solution of the fuel-coal disposal problem lies in the construction of large central power plants with burning equipment designed for effective use of the secondary coal as it comes from the washery.

Turning to the strictly technological problem of washing these average Barro Branco coals, it has generally been agreed that 1½ in. crushing is necessary to free any metallurgical coal, and the basic data have been set up on that basis.

Maximum recovery of combustible dictates a very high primary washing gravity, because the waste rock, after the clay parting (Barro Branco) is discarded, is all bony. Primary washing gravities of 2.00 for the fuel-coal separation and 1.45 for the metallurgical-coal separation are suggested for the 38 to 10 mm. sizes. This will recover, in the steam-coal product, all the middling fractions up to approximately 50% ash content. The metallurgical coal at this size will exceed slightly the tolerances suggested for both ash and sulfur; but this can be compensated by a lower specification on the 10 mm.–0 size to bring down the average in the combined metallurgical-coal shipments.

These separations of the 38–10 mm. size might be made by two heavy-medium separating vessels, or by a Baum jig for the high-gravity separation and a heavy-medium system vessel for the low gravity. With the latter scheme, the entire plant-feed might go to the primary wash-box at 38 mm.–0 size, with the washed coal subsequently to be screened at 10 mm. for rewashing.

For the 10 mm.–0 size, it is suggested that the metallurgical separation be made at a washing gravity of 1.60 for the 10–5 mm. size. The normal washing gravity spread into the finer sizes will handle the ash and sulfur content to be safely under the tolerances down to approximately 0.5 mm. size. It is assumed that this separation will be made with a heavy-medium circuit in cyclone washers or by gravity washers backed up with cyclone rewash units. Precise separation of this size will result in delivery of a rather small proportion of steam-coal that may be absorbed in local thermal power plants to supply the energy needed for the mining operations and related community activities. To this end, the high-gravity separation for fuel-coal recovery in the fines, size 10 mm.–0, may be adjusted to the most economic performance, taking into account use requirements as well as the physical conditions.

For maximum recovery of metallurgical coal, application of froth flotation to the 0.5 mm.–0 size is very interesting but is complicated by the extremely high sulfur content. Experimentation in this field is recommended, not only for its role in slimes treatment but for its relation to promising possibilities of still further improvement in metallurgical-coal recovery by finer crushing.

Middlings re-treatment

The Barro Branco coal is exceptionally responsive to finer crushing for the release of sulfur minerals. The washability data of Figure 14 show the effect of 10 mm. crushing compared to 38 mm. crushing. This test was made on a duplicate of the sample represented by Figure 7. Still other tests of the sink portion of Barro Branco coal samples, not shown here, indicate that the material is unusual in the release of free pyrite and corresponding reduction in float-sulfur content. These data indicate that a substantial addition to the recovery of metallurgical coal can be obtained by crushing the coarse middlings (38–10 mm.) from the primary separation and rewashing the crushed material by the procedure recommended for the fine washery.

The choice among middlings-disposal schemes is an economic problem, depending upon the use application of steam-coal.

Similarly, a small improvement in sulfur content of the metallurgical coal might be obtained by finer crushing of the entire plant feed or by recrushing and rewashing of the primary washed coal. Here too, the solution is primarily economic. The improvement in metallurgical-coal sulfur (at a given yield) would be in the range of 0.10 to 0.20% for the total product. The advantage of this factor in blast-furnace operation must be balanced against the disadvantages of (*a*) finer size of industrial feed to be marked, (*b*) cost of finer crushing and rewashing, (*c*) increase in sludge recovery and water-clarification problems with hazard of larger sludge losses, and (*d*) the sulfur problem in the increased load of fines that should go to froth flotation.

The addition to metallurgical-grade product that may be attained by recrushing and rewashing middlings must likewise be balanced against the disadvantage of finer sizing in the steam-coal. There is still another factor to be taken into account—recovery of additional metallurgical coal from the middlings is necessarily reflected in a higher average ash in the remaining steam-coal, and so a breaking point may be reached where it is necessary to put back some first-grade coal to make the steam-coal economically usable.

Some Aspects of Labour Problems in the Iron and Steel Industry

THE INTERNATIONAL LABOUR OFFICE, Geneva

The difficulties created by the establishment of new industries in underdeveloped countries are not confined to those arising in connexion with raw materials, industrial technique and markets, etc., but include labour problems, or, more specifically social problems connected with labour; that is to say, with recruitment and vocational training of workers, conditions of employment, industrial relations, industrial safety, housing, welfare and social services. All these social problems are equally as important as those of an economic or technical order, since they exert direct influence on the productivity level.

The Iron and Steel Committee of the International Labour Organisation can make a positive contribution to the solution of social problems arising in connexion with labour in underdeveloped countries that have established iron and steel industries. This contribution is the subject of the present working paper, prepared by the International Labour Office.

The scope of this paper is necessarily restricted and only the more important aspects of labour problems in the iron and steel industry could be discussed.

1. INDUSTRIALIZATION AND SOCIAL POLICY

At the present time, in all countries, much attention is being devoted to industrial developments and the increase of production, both key factors in raising the standards of living. Simultaneously with the extending of industrialization, however, it is necessary to pay special attention to the social problems created by economic development. The expansion of industry multiplies the claims to be satisfied. This is particularly true in the case of newly-established industries, especially in the economically underdeveloped countries. The social conscience of the people has awakened in step with the progress of industrialization. The desire for a higher standard of living finds its expression in every country, whatever the stage of its economic development. A social policy should embrace the consideration of these claims, taking into account the limitations set by their economic resources, as well as the customs and institutions of different countries. The means which may be adopted to apply a dynamic social policy vary in accordance with the countries concerned and the problems arising therein. If governmental action is necessary, it does not follow that it alone will be all sufficient and final. Even when governments are called upon to build the framework of social policy and establish the minimum standards as well as to supervise their application, there nevertheless remains a vast field to be covered by the activities of employer and worker organizations.

In all the countries of Latin America it is believed that industrialization would promote the raising of the standard of living of their populations. Thus, in 1946, the Third Conference of American States Members of the International Labour Organisation adopted a resolution concerning industrialization. Its terms indicate that the Conference viewed industrialization as essential to Latin America for attaining a higher standard of living, better equilibrium of the economic structure, intensification of foreign trade and greater economic self-sufficiency.

The problems of social policy created by industrialization were re-examined by the Director-General of the Fourth Conference of American States Members of the International Labour Organisation held at Montevideo in 1949. The principal objective of industrialization was the increase of production and thereby the raising of consumption levels. It did not suffice to produce goods: the population must be able to purchase them. The income of wage-earners and of the rural population must increase at the same time as production in order that these sectors may obtain their share of the product of their own labour and of the advantages accruing from economic development as a whole. For wage-earners, this implies the existence of trade unions and, whenever possible, the determination of wage levels by collective rather than individual bargaining. When these unions are inadequate to safeguard the interests of the workers, they must have recourse to different mechanisms for fixing minimum wages by public authority, in accordance with the norms established by the 1928 International Labour Conventions. Social security is another of the claims arising out of industrialization. The greater the dependence of workers on industry for employment, the greater the risk of total loss of their livelihood if they should lose their jobs or become total invalids. Hence the importance attached to social security in the economic and social policies of all industrial countries. Housing is another problem created by industrialization, which forces a greater number of workers to live within the urban sector. The housing policy must fit into a town-planning scheme which takes into account the health of the working population, as well as their intellectual and leisure activities. In other words, from the workers' standpoint, the advantages of industrialization must represent, not only improved salaries, but also decent housing facilities in organized communities, that is to say, equipped with medical centres, schools, churches, recreational facilities, and all obviously within their means. Special attention should be given to matters of public health, since a worker cannot attain his optimum yield unless he is in good health. Moreover, it is essential to study the question of productivity since the success of every effort directed to raising the standard of living and ensuring adequate minimum wages, social security, housing and welfare depends ultimately on the production of each individual operative. The need to provide for the schools and vocational training of the worker, constitutes a basic requisite of any productivity planning. Lastly, the progress of industrialization calls for active co-operation between management and labour organizations, in turn involving observance of the freedom of association, collective bargaining, methods of consultation and co-operation as between management and labour within the enterprise, and at the industrial and national level. These are the most important comments made in the report concerning industrialization and social policy.

The chapter of the Director-General's Report on the progress of industrialization gave rise to a very useful exchange of views, culminating in the adoption of a resolution at the Fourth Conference of American States Members of the International Labour Organisation. This resolution concerns the social aspects of the economic development of the American continent and states that in order to obtain the economic and political benefits expected from programmes of economic development, the latter should be accompanied, at every stage, by a general impetus to social progress. With this object in view, the resolution calls for the establishment of a series of general directives, some of which are directly connected with social policy and labour problems. Moreover, the aforesaid resolution emphasized that technical assistance in connexion with the social aspects of economic development should form an integral part of every technical assistance programme carried out through the agency of international organizations. Specifically, the social aims of such economic development should envisage full and stable employment and the raising of the standard of living guaranteed by equitable treatment for foreign and domestic labour. The Conference established a list of points regarding which the International Labour Organisation could and should co-operate with the Latin-American countries who so desire.

There is a large sector of social policy in which joint action of labour and management may be efficiently utilized. This applies with particular force in the field of the iron and steel industry. Broadly speaking, it may be said that the iron and steel worker enjoys a standard of living and working conditions which place him at the top of the working class in every country. Labour and management in this industry have grouped themselves in solidly established professional organisations which by now have acquired long experience of industrial relations. This accounts for the fact that the Iron and Steel Committee of the International Labour Organisation has been able to attain the goals set for it. In economically underdeveloped countries, or in those in which iron and steel industries have been established in the course of the past few years, one notes a very marked tendency to take into account the claims of social policy. The iron and steel industry in countries undergoing a process of industrialization is faced with serious responsibilities in this connexion, since it is in a position to guide and influence the social policy of other industries having similar characteristics. It is therefore necessary for the iron and steel industries of Latin America to take into account the achievements of similar enterprises in countries with long experience in social problems. In this connexion, the Iron and Steel Committee of the International Labour Organisation might act as the link between the iron and steel industry of Latin America and that of other countries, thus contributing to the advancement of social policy.

2. THE WORK OF THE INTERNATIONAL LABOUR ORGANISATION IN THE IRON AND STEEL INDUSTRY

The work of the International Labour Organisation in the iron and steel industry is principally effected through the agency of its Iron and Steel Committee.

In 1945, the Governing Body of the International Labour Office decided to establish a number of Industrial Committees, among which figures that for the Iron and Steel Industry. In 1952, 19 countries made up the Committee: The Federal Republic of Germany, Australia, Belgium, Brazil, Canada, China, Czechoslovakia, France, Holland, India, Italy, Japan, Luxembourg, Mexico, Poland, Sweden, the Union of South Africa, the United Kingdom and the United States. As in the case of other industrial committees,[1] each country is represented by a tripartite delegation, comprised of two representatives of the respective governments, two representatives of management and two of labour. The meetings are also attended by representatives of United Nations and of the specialized international agencies, such as the World Health Organization, and observers from leading management and labour international organizations throughout the world.

The object of the industrial committees is the improvement of prevailing conditions in their respective industries and the furthering of world peace and social progress. The industries for which committees have been established are of international importance inasmuch as they are located in every country; they make use of imported raw materials; their products flow into the current of international trade and for like reasons. The Committee enable special and detailed attention to be directed to particular conditions in the leading international industries. It is thus possible to cope with problems before they get out of control, and to provide interested groups with a means of discussing their problems on the basis of accurate data, and facilities for reaching agreement regarding the measures to be adopted, and the best manner of carrying them out so as to obtain satisfactory results.

Prior to outlining the proceedings of the Iron and Steel Committee, it is important to begin by defining the limits of industry with a view to determining its representation at the meetings of the Committee. At its second meeting, the Committee adopted a formal definition of the term "industry," which term was to be accepted by the Governing Body. For purposes of representation at the meetings of the Committee, the "iron and steel industry" should be interpreted as: (*a*) the obtaining of iron from iron ores whether in the blast furnace or by any other process; (*b*) the production of steel ingots, wrought iron and steel for castings; (*c*) the operation of rolling steel and wrought iron, whether in the hot or cold state; (*d*) the manufacture of sheets coated with tin, zinc, or other metals; and (*e*) the drawing of iron and steel wire.

Employment

The problem of employment in the iron and steel industry was included in the agenda of the Second Session of the Committee (held in Stockholm, August 1947). On that occasion, the International Labour Office prepared a report on the regularization of production and employment at a high level in the iron and steel industry.

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[1] There are at present nine industrial committees to deal with the following branches of economic activity: Coal Mines, Inland Transport, Iron and Steel, Metal Trades, Textiles, Building, Civil Engineering and Public Works, Petroleum, and lastly, Chemical Industries.

This question appeared again on the agenda of the Committee's Third Session (held in Geneva, November-December 1949). The report then prepared by the International Labour Office dealt with technological improvements in the iron and steel industry and their effects on employment.

Vocational training

Vocational training and promotion in the iron and steel industry was included as an item in the agenda for the Fourth Session of the Committee (Geneva, May 1952) and, as at previous sessions, the International Labour Office prepared a report on this subject.

Minimum income security and guaranteed wages

Minimum income security (annual and other wage systems designed to provide assured earnings) appeared as an item of the agenda for the Second Session of the Committee. The session, however, did not reach definite conclusions regarding this subject, so that it was then transferred to the agenda for the Third Session. The International Labour Office prepared a further report on guaranteed wages in the iron and steel industry.

Industrial relations

The problem of industrial relations appeared on the agenda of the Second Session of the Iron and Steel Committee (Stockholm, August 1947). The report submitted by the International Labour Office dealt with labour-management co-operation.

Welfare services

The question of welfare services was included in the agenda for the Fourth Session of the Committee, when a report on the subject was presented by the International Labour Office.

Other studies and inquiries

In addition to the technical reports specially prepared in view of the inclusion of the respective subjects in the agenda of the sessions of the Iron and Steel Committee, the International Labour Office also prepared a general report for each of its sessions. This covers questions such as supply and demand and social conditions (employment, wages, working hours and industrial relations). It also contains studies and inquiries on matters which do not appear specifically on the agenda of the Committee, regarding which the Committee has nevertheless requested the International Labour Office to prepare preliminary papers, or alternatively on matters which are of undoubted interest to the Committee.

The listing of all these studies and inquiries would be far too lengthy a task, so that only the more important will be mentioned, among which figure the following:

(*a*) Accident prevention in the iron and steel industry (legislation, co-operation between state factory inspection services, employers and workers, statistics, education, propaganda, works safety organization, health protection, etc.);
(*b*) Safety and health in the iron and steel industry;
(*c*) Labour statistics;
(*d*) Fluctuations in the demand for steel in the United States and their relation to a programme for stabilizing employment in the industry;
(*e*) Iron and steel Bill in the United Kingdom;
(*f*) The "Basing points" controversy in the United States;
(*g*) Co-determination in the German iron and steel industry;
(*h*) Principal social provisions of the Treaty constituting the European Coal and Steel Community.

Conditions of employment of workers in the iron and steel industry in Latin-American countries

The Iron and Steel Committee of the International Labour Organisation could hardly ignore the activities of the Economic Commission for Latin America, nor those of the Economic Commission for the Far East, especially their economic and technical studies. In the course of its earlier meetings, the Committee has expressed its interest in regard to the question of technical assistance for the iron and steel industry in underdeveloped countries. At its Fourth Session, therefore, the Committee suggested that the Governing Body should authorize the Director-General of the International Labour Office to communicate with the Secretary-General of the United Nations, in order that he in turn might advise the Economic Commission for Latin America of the resolutions adopted during the Fourth Session of the Iron and Steel Committee and those of preceding sessions which might be of interest. The Committee also proposed that the Governing Body should request the International Labour Office to devote a part of its general report for the Fifth Session of the Committee to a preliminary study of conditions of employment of workers in the iron and steel industry in Latin-American countries, in view of the probable inclusion of the problem of conditions of employment in the iron and steel industry of underdeveloped countries in the agenda for the Committee's next session. This resolution which had been proposed by the two representatives of employers from Mexico was adopted unanimously.

3. CONCLUSIONS OF THE IRON AND STEEL COMMITTEE REGARDING PROBLEMS IN THE IRON AND STEEL INDUSTRY

The conclusions arrived at during the meetings of the Iron and Steel Committee take the form of suggestions involving a request for action either within the International Labour Organisation itself or by the different countries. These conclusions are communicated to the member countries of the International Labour Organisation and any country so desiring may adopt the recommendations of the Committee, whether or not it is represented on that Committee. The Governing Body of the International Labour Office has drawn up a method of procedure which facilitates the efficient analysis of the conclusions of the Committee in different countries. During the course of the sessions, moreover, a sub-committee is appointed to consider the action subsequently taken on these conclusions.

The conclusions of the Iron and Steel Committee cannot be analysed in this brief paper; consequently, only the titles of the different resolutions adopted will be cited, and reference may be made to the working papers themselves for further details.

Full employment and vocational training

A resolution concerning the regularization of production and employment at a high level in the iron-and-steel industry.

A resolution concerning technological improvements in the iron and steel industry and their effects on employment.

A resolution concerning vocational training in the iron and steel industry.

Guaranteed wages

A memorandum on the question of minimum income security in the iron and steel industry.

A resolution concerning guaranteed wages in the iron and steel industry.

Welfare services

A resolution concerning welfare services in the iron and steel industry.

Industrial relations

A resolution concerning freedom of association.

A resolution concerning collective bargaining.

A resolution concerning the observance of collective agreements.

A resolution concerning co-operation at the industrial level.

A resolution concerning the works committee.

Safety

A resolution concerning special safety services and joint committees.

Statistics

Table of labour statistics in the iron and steel industry.

FUTURE PROSPECTS

The Iron and Steel Committee of the International Labour Organisation has already shown its interest in the assistance which might be rendered to the iron and steel industry in underdeveloped countries. At its Second and Third Sessions, in fact, resolutions were adopted in this sense. The Governing Body of the International Labour Office, within the limits of its general policy of technical assistance, resolved to consider favourably any request which might be made by underdeveloped countries for such assistance in the iron and steel industry.

It is likely that in the future certain projects of technical assistance for Latin America will include requests directly concerned with the iron and steel industry. The project of assistance to Yugoslavia, concerning vocational training, is a typical example of the advantages which the iron and steel industry can derive from a project of technical assistance embracing all the industries whilst at the same time considering individual activities in certain industrial sectors.

When the plan was drawn up for a programme embodying the extension of co-operation by United Nations and its specialized agencies,[2] it was emphasized that the general service proposed by the International Labour Organisation might be supplemented, if necessary, by aggregate programmes in cases where industrial development for the time being is highly centralized, in the form of one or several important industries. By assisting in the solution of labour problems and by raising the productivity of certain industries of outstanding importance, aggregate programmes of technological assistance for certain industries might well facilitate the early stages of industrial development and would help to create favourable conditions for the healthy development of new industries.

The iron and steel industry is included among those for which aggregate technical assistance programmes might be formulated. These programmes would have to provide for the recruitment and training of labour, conditions of employment, industrial relations, particularly methods for the solution of industrial controversies, measures directed to ensuring safety and health services for labour, housing problems and general welfare services.

As may be deduced from sections 2 and 3, these matters have already been studied by the Iron and Steel Committee in the course of its first four sessions. As a result, definite conclusions were reached regarding most of these items. It is obvious that the form of action proposed by the Committee will have to be re-examined and adapted to new situations, as they arise, if the circumstances prevailing when they were established should change. This, in fact, is the reason for the Committee's constant interest in the trend of economic and social conditions within the iron and steel industry in the different countries. Yet, even when the form of action must correspond to prevailing conditions, there are nevertheless certain principles and methods which have been proved successful wherever they have been applied and these, to a certain extent, form a sort of social charter for labour and management in this industry.

The underdeveloped countries which have entered upon a trend of industrialization, particularly as regards the manufacture of iron and steel, may therefore profit considerably from the work of the Iron and Steel Committee. Moreover, as has been pointed out, the Committee has advised the Governing Body of its wish to have the International Labour Office prepare a study concerning conditions of employment of workers in the iron and steel industry in Latin-American countries. The scope of such a study would be relatively great, since it would have to envisage a series of questions affecting every aspect of labour in the iron and steel industry. To carry it out successfully, the International Labour Office[3] would have to have recourse to active co-operation on the part of the governments, labour and management in the interested countries. Moreover, in the opinion of the Committee, from such a study it should be possible to obtain some indication as to whether the question of conditions of employment of workers in the iron and steel industries of underdeveloped countries shoud be placed on the agenda of a future session of the Iron

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[2] *Technical Assistance for Economic Development*, United Nations, May 1949.

[3] A similar resolution was adopted with reference to Asia and the Far Eastern countries.

and Steel Committee. At all events, the study itself will permit an exchange of opinions and information during the general discussion to take place when the conclusions are made known to the Committee. At a later date, this discussion may offer an opportunity for the Latin-American delegates of governments, management and labour represented on the Committee to consider whether it would be desirable to recommend specific measures of technical assistance for the iron and steel industry in that part of the world.

The work of the Iron and Steel Committee, together with the existence of permanent contact between the International Labour Organisation and the industrial centres of those countries in which this industry is in active operation, have led to the accumulation of large stores of experience and technical knowledge which might well prove to the advantage of those countries in which the iron and steel industry is now being developed. The Governing Body has already resolved that the International Labour Organisation shall examine favourably all requests addressed to it from the underdeveloped countries with a view to obtaining technical assistance for their iron and steel industries, especially as regards the recruiting and training of labour, conditions of employment, industrial relations, industrial safety services, housing, welfare and social services.

Summary of Discussion

Continuous Casting of Steel by the Rossi-Junghans Process, *presented by Mr. Buehl in the absence of the author*

Mr. COHEUR opened the general discussion of the paper with the remark that the process was a very interesting one and should prove particularly so for the Latin-American countries. It had been studied very thoroughly in Europe, and whereas Junghans' researches were mentioned by the author, continuous casting had also been studied in France, Italy, Belgium and England. At Huckingen in Germany, a plant was operating with that process and was open for inspection.

Mr. WARING said that a firm in Bradford (England) had been working on continuous casting on a commercial scale, but it had not been too successful and there had been some unfortunate accidents. The United Steel Company was going to try out the process at one of its works. In Germany, manufacturers or major steel companies had formed an association which he believed had financed the Huckingen plant; it was rather more than a pilot plant, and the product was quite good.

Direct Reduction Yields—Variable Density Steels, *presented by the author*

Mr. CAVANAGH began his presentation by saying that the process he was about to describe was not in commercial production and should not occupy the attention of those present until it was; it was, however, a process which he thought to be interesting. It had been in operation on a pilot scale for one year, and one of the largest American steel companies was financing the development work. In another year there would be sufficient data on a commercial scale to enable those interested to formulate their own ideas on how useful it might be. The description he was about to give should, in the meantime, be regarded more as a news item.

His organization already had a furnace operation into which carefully prepared ore was fed. It travelled continuously through the furnace and emerged as steel sheet. He produced some examples of the work done with steel sheet, explaining that one operation before, they had been iron ore.

To those familiar with sheet steel operations, the production of deep-drawn sheet steel was a very difficult process. In the furnace being used at the Ontario Research Foundation, the sheet was being produced directly from iron ore; it then received one cold temper pass and became deep-drawn steel with an acceptable cupping test.

The only reason for interest in the process was that it was cheaper than ordinary methods. That was his own personal opinion, based on the pilot operation of one ton per year, but it was also the opinion of those who were putting a great deal of development money into the process. They had exclusive rights to it in the United States and non-exclusive rights in Latin America. Various ores were being used, including Mesabi hematite, Durango ore, magnetite concentrates and El Pao ore. One of the finest ores would probably be itabirite and in some locations the process would be very interesting indeed.

In practice the steel could be produced in one of two ways. The first method was to make it in batches in a tunnel kiln. After being hot rolled, sheet and wire could be obtained, having the same properties as any steel with the same chemical composition; the only difference was found in looking at the microstructure. Inclusions were not the usual elongated slag inclusions, but were the original silica particles, with a round shape which did not distort during rolling.

Alternatively, a special furnace could be made, obtaining the reducing gas from a very poor coal, from natural gas or from oil, and the steel could be produced directly.

One of the features of the new method was that a sheet of 0.005 in. or of 0.015 in. could be made directly, the only subsequent operation being the cold temper pass.

Steel sheet with copper cladding on both sides could also be made, or any other cladding could be used. It was easy to lay the cladding on the surface of the sheet as it was being formed. The second major class of product was formed directly in final shape. For instance, in a sheet furnace a slab of 3 ft. wide by 1 in. thick could be laid down continuously. The ore was reduced to metallic iron, the carbon content began to increase with time and a steel sheet of any required carbon content could be obtained.

A shape could be made and one edge could be hot formed. He produced a sample, explaining that it was a one-piece tool which was very light. It was a normal tool steel on the edge, providing a good axe blade. It had a reinforcing bar down the centre so that it would not break. In the case of another sample, the original bar had been heated in a blacksmith's forge, forged and made into a knife blade, as good as any tool steel available on the market.

There were many possibilities in taking the material from the furnace, whether it was to be a continuous production or a tunnel kiln batch type operation, followed by forging part of the material. It could also be formed into any shape desired by pouring the iron ore into moulds. It could be packed into an airtight container with coke, or reducing gas could be passed through the porous mould.

The material could be produced in weights ranging from that of wood to about the full density of steel, which was 7.8, and at the very low densities it cost about the same per volume as wood. It was an excellent insulator, was fireproof at low densities and could be worked just like wood. Applications in the low density field were so numerous that it had not been possible to follow up all the suggestions made in that connexion.

The Canadian Defence Research Board, and all the larger machining companies in Canada were assisting in the development work and it had been decided to concentrate on a few applications, because the more it was possible to learn about the material, the more profitable were the applications discovered.

He explained that apart from the publication that was in the hands of participants, there would be no further publications until the plant had been in operation long enough for production costs to be given with some accuracy.

Mr. COSTA LINO remarked that the process described would be of enormous advantage in the future. He would like, however, to know more about the swelling and shrinkage of the ores, and what properties were required in the ores in order to produce such a variety of materials as had been described.

Mr. CAVANAGH replied that the question of swelling and shrinkage of iron ores was not only interesting in connexion with the particular process described, although in that case it was vital. The matter had been studied in connexion with the blast furnace, the Wiberg-Söderfors process and the new method; if for instance the El Pao ore were reduced in carbon monoxide, it would swell extraordinarily. Once it was known how an individual type of ore was going to behave, it was possible to plan operations so that any particular part was not destroyed. If exact dimensions, + or − 1%, as well as light weight, were required, El Pao type ore could be used, but in order to do so, it was necessary to ensure that the swelling would not destroy the piece. Enough space had to be provided so that each particle could swell a little. His organization had discovered an experimental way to predict the behaviour of an ore, but a short investigation was required in order to learn how to grind it. Some ores shrank to half their original size, and a swelling and a shrinking ore could be blended with very great success.

Mr. BUEHL requested information on the corrosion resistance of the material, as he would be inclined to think that it would corrode throughout the entire piece instead of on the surface where, through high porosity, the density was lower than that of standard steel.

Mr. CAVANAGH replied that most steel in use was either plated, painted or protected in some way against corrosion, and so was the material described. The protective coating was more important in the case of that material than with ordinary steel.

Mr. ALLARD asked whether European ores such as, for instance, French or African ores, had ever been used in the process.

Mr. CAVANAGH answered that such ores had been used, one interesting feature of the process being that if the ore was not to be hot worked, high phosphorus, high sulfur or other such properties were not important. The only limitation of importance was the kind of service required from the product.

He explained that 99.5% of iron oxide was required for hot work. There were magnetic superconcentrators in use, one of which was operating in France, which concentrated ores to a very high degree.

Mr. LEUSCHNER remarked that there was bound to be a higher content of some metalloids with the process; that would not be injurious for pieces like the hatchet, but he wished to know the behaviour of those metalloids in forged and rolled material.

Mr. CAVANAGH replied that a glance at the deep drawing tests provided an answer. Many different types of material had been made and his organization considered that if the resulting products met the required service tests, mechanical and cupping tests, they would perform the same service as standard steels.

The final answer on the effect of the extremely high silica in some of the sheets as compared with ordinary steel sheets was whether or not the required service would result.

Notes on the Ugine-Séjournet Process, *presented by Mr. Schereschewsky in the absence of the author*

Mr. SCHERESCHEWSKY began by remarking that he had thought he would be presenting something extraordinary, but had become much more modest after listening to the still more extraordinary communication by Mr. Cavanagh. The extrusion of non-ferrous metal had been known for a long time, while the extrusion of steel had been unsuccessfully attempted for a long time. Interest in the extrusion process lay in the fact that it promised the production of small quantities of bar steel in complicated sections through a cheap apparatus, whereas if it was desired to prepare the same sections using the normal rolling process, expensive cylinders would have to be made, which would only be economic with a very high output.

By means of the process described, the commercial extrusion of various kinds of steel and even of cast iron had been made possible on a large scale and under economic conditions, through using glass as a lubricant both between the billet and the container and between the billet and the die. Extrusion conditions were quite different from those used in non-ferrous metals extrusion. The flow resistance of the various steels and their relationship to temperature had been experimentally measured. Viscosity and diffusibility of various types of glasses and glass-like materials successfully used had been tested; one type, for instance, contained 65% silica, 15% lime, 8% sodium dioxide and 4% alumina.

A surprising fact was that steel had to be extruded much faster than copper or aluminium alloys. The total duration of the extrusion cycle had already been reduced to 30 seconds and even shorter cycles were contemplated; for the time being commercial operations worked on the basis of 60 cycles per hour. That unusual speed had led to a substantial change in the design of the non-ferrous extrusion presses, the longitudinal motion of the sliding block, for instance, being replaced by a transversal motion. Simultaneously the very fast replacement of the die would be made easier. Additional speed could possibly be gained by using mechanical presses instead of hydraulic presses.

Mr. LEUSCHNER asked whether it were possible to give, very approximately, the investment costs and the size of the largest plant in existence.

Mr. SCHERESCHEWSKY replied that 2,500 tons was the standard extrusion press size. Tests in France, near Paris, had been made with a 1,600 ton extrusion press; that was a medium, rather small press, and had given better results. His earlier remarks concerning changes in design of the press did not necessarily mean a substantial change in the price of the press. The investment was extremely low because to change sections it was sufficient to insert a small die of hardened metal, just as in the extrusion of non-ferrous metals, whereas the cost of investment in a rolling mill cylinder was much higher.

Some Aspects of Labour Problems in the Iron and Steel Industry, *presented by Mr. Desprès on behalf of the International Labour Office*

Mr. DESPRÉS introduced the paper with an outline of the past and future activities of the Iron and Steel Committee of the International Labour Organisation. After having called attention to the study to be undertaken by the ILO on conditions of employment of workers in the iron and steel industry in Latin-American countries, and the role of the ILO in the implementation of the United Nations technical assistance programmes, he stated that the ILO would give sympathetic consideration to requests for technical assistance in the field of vocational training for the iron and steel industry in Latin America. Such requests should be transmitted by the respective governments, according to the procedure laid down by the United Nations and its specialized agencies. Furthermore, the next Latin-American Conference on Manpower, to be held in Lima in December, might give the governments the opportunity, if they so desired, of examining and discussing manpower problems in the iron and steel industry. That Conference might, for example, examine the possibilities of exchanging Latin-American workers from the steel industry for the purpose of vocational training, within the framework of the Technical Assistance Programme of the ILO. In that connexion, he was particularly interested in the offer made at an earlier meeting by Mr. Pablo Sada on behalf of the Monclova plant in Mexico, and requests for trainees to be sent would be welcomed.

Mr. CANGUILHEM remarked that the industry in Latin America would certainly be delighted to use the services offered, as he believed the idea of exchanging workers would be very interesting. To date, only engineers and technicians had been exchanged; the exchange of workers would be a further step towards increasing knowledge and understanding.

STEEL MAKING AND FINISHING

Range of application of steels made by different processes

29 October 1952—Morning

Chairman:

Augusto LEGRAND, Sociedad Mixta Siderúrgica Argentina, Buenos Aires, Argentina
Later: Evaristo ARAIZA, President, Banco de México, Mexico D.F., Mexico

Contributed Papers:

Some Effects of Minor Elements on the Characteristics of Plain-Carbon Steels
Francis W. BOULGER, Supervising Metallurgist, Battelle Memorial Institute, Columbus, Ohio, U.S.A.
Standardization in Relation to Control in Steel Production
G. WESTON, Technical Director, British Standards Institution, London, England, and
R. L. RICHARDS, Technical Assistant, British Standards Institution, London, England
The Control of Composition During Steelmaking
G. WESTON, Technical Director, British Standards Institution, London, England, and
G. R. BOLSOVER, Chief Chemist, Samuel Fox and Company, Ltd., Sheffield, England
Specifications of the Products Made by the Compañía de Acero del Pacífico, S.A., Huachipato, Chile
Oscar HERRERA SILVA, Assistant Superintendent, Metallurgical Department, Cia. de Acero del Pacífico, S.A, Huachipato, Chile.
Notes on Specifications of Steels for Different Uses
Federico FRICK, Consulting Engineer, Instituto Nacional de Tecnología y Normalización, Santiago, Chile

Summary of Discussion:

Participants: Messrs. BOULGER, ALLARD, COHEUR, COSTA LINO, SANITER, MERCIER, PALMÉ, LEUSCHNER, CANGUILHEM, MACEDO SOARES E SILVA

Some Effects of Minor Elements on the Characteristics of Plain-Carbon Steels

FRANCIS W. BOULGER

INTRODUCTION

The behaviour of steel is controlled by conditions imposed during processing or service and limitations established by its composition and microstructure. Small amounts of sulfur, phosphorus, tin, copper, nitrogen and hydrogen change some of the properties of steel. Whether the effect of a particular element be good, bad, or indifferent is determined by the conditions steel must satisfy in fabrication and in service. Much will be said at this meeting about the effect of variations in composition on the characteristics of steels. An even more difficult subject to discuss is the limitations these attributes impose on applications of the materials.

It cannot be emphasized too strongly that reductions in ductility caused by some impurities mentioned in this paper are not necessarily harmful. Steels which are inferior according to certain arbitrary standards are entirely adequate for many uses. Low ductility or even brittleness evidenced in laboratory tests need not bar materials from services not involving low temperatures, severe impact, or repeated stresses. In many cases the cheaper steel, even though inferior metallurgically because of its chemistry, is a sound engineering choice. Low ductility is undesirable, of course, in steels which must be severely cold formed during fabrication.

In the United States of America, specifications limit the maximum phosphorus and sulfur contents of a standard grade of steel as illustrated in Table 1. The limits vary for different types of steel and for grades made in different kinds of furnaces. The limits for phosphorus and sulfur are reasonably low but can be met without difficulty by ordinary practices and customarily available charge materials.

Table 1

MAXIMUM PHOSPHORUS AND SULFUR CONTENTS PERMITTED FOR CERTAIN AMERICAN IRON AND STEEL INSTITUTE GRADES

Type of steel	*Maximum sulfur %*	*Permitted, % phosphorus*
Basic open hearth carbon steel, forging or rerolling quality	0.05	0.04
Acid Bessemer carbon steel	0.06	0.11
Basic open hearth alloy steel	0.04	0.04
Electric furnace carbon steel	0.025	0.025

The limitations set on sulfur, phosphorus, and other chemical elements are intended to ensure uniformity in quality and properties of steels of the same grade. The chemical specifications are not intended to define the properties well enough to inform inexperienced designers as to suitable applications for the grade (1). Metallurgists work toward that end but the designer must establish the suitability of a material for his particular application.

Specifications in highly industrialized countries usually set relatively low limits for impurities in steel. Considerable benefits are derived when the product is used in mass production operations because narrower composition limits reduce the variation in properties characteristic of steels of the same grade. Variability of the raw material is less important when steels are fabricated in smaller quantities. Perhaps then, the Latin-American steel industry would be justified in setting higher limits for impurities than those used in other localities. The purpose of this paper is to outline the effects of sulfur, phosphorus and nitrogen on the properties of low- and medium-carbon steels. A knowledge of these effects will aid the metallurgist and production man to set useful, practical tolerances for impurities in different grades of steel. The information will help the metallurgist and designer to judge the suitability of steels made to such specifications for a particular application.

I. NITROGEN

The nitrogen content of a steel depends upon the melting process used in its manufacture. Ordinarily, the nitrogen content of basic open hearth steel is 0.005% or less. Bottom-blown Bessemer converters usually produce steel containing 0.010 to 0.020%. The nitrogen content characteristic of this process can be lowered by using converters with shallow baths or by diluting the air with oxygen or steam. Steels made in electric furnaces, in side-blown converters, or by duplex processes contain nitrogen in amounts intermediate between those typical of Bessemer and open hearth steel. Nitrogen is sometimes added to steels made for special applications such as open hearth free-cutting grades.

Nitrogen is a potent alloying element in steel. On a weight basis, it increases the yield strength, ultimate strength, and hardness of hot-rolled and cold-rolled steel, more than any element. Figure 1 indicates that each increment of 0.01% raises the yield and tensile strength, of semi-killed, hot-rolled 0.20% carbon steel, about 6,800 psi. Increasing the nitrogen content impairs the ductility and appreciably lowers the notched-bar values of such steels. Because of their lower ductility, rimmed, capped, or semi-killed steels containing more than 0.006% nitrogen are unsatisfactory for deep drawing. However, such steels are suitable for many uses not involving severe cold forming. As Faddis (10) points out, hot-rolled steels containing about 0.010% nitrogen are less susceptible to fluting, weld without difficulty, and make stronger barrels or shipping containers.

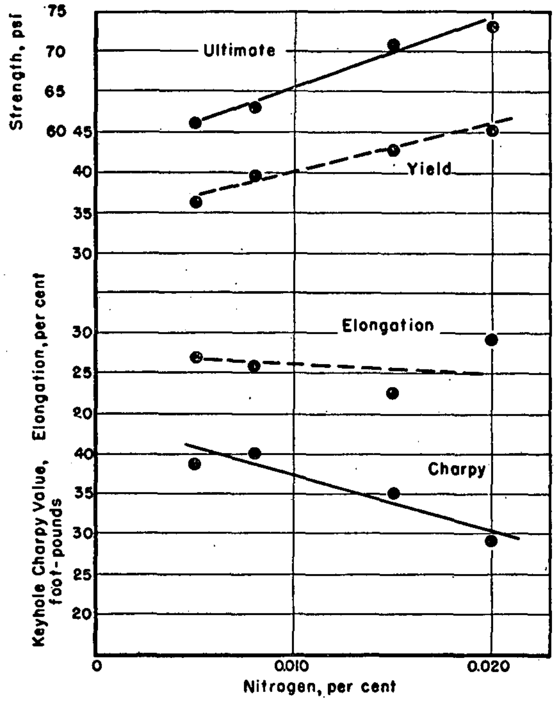


FIGURE I. TENSILE AND NOTCHED-BAR PROPERTIES OF 0.20 PER CENT C, 0.80 PER CENT Mn, 0.015 PER CENT P PLATE STEELS AT 24 C TESTED IN THE AS-ROLLED CONDITION

0-20668

Nitrogen contents, between 0.005 and 0.010, are useful when producing low-carbon, cold-reduced sheets and strips with high hardnesses (10). Such steels are used for signs, bands or cabinets where resistance to buckling is desirable and forming requirements are mild. For the same reason, some high-nitrogen tinplate is used for can bodies and ends. These materials reach the same hardness with less cold work than lower nitrogen steels (10).

The effect of nitrogen on ageing characteristics is often of much greater importance than its alloying influence on the properties of hot-rolled steels. Ageing is the term applied to the gradual change in properties of iron or steel, with time, after cold working or rapid cooling from high temperatures. Ageing can cause increases in strength and hardness, decreases in ductility and impact resistance, (38) and changes in magnetic properties. The rate of ageing depends on the composition of steel and the ambient temperature.

Figure 2 shows some of Cockritz's data (34) on the effect of ageing time and temperature on the strength of mild steel. These unkilled steels had been cold reduced 5% prior to the ageing treatments. The increase in tensile strength amounted to about 10% on ageing for 90 days at 20°C. The chart also shows that raising the temperature accelerates ageing, as would be expected for a precipitation reaction.

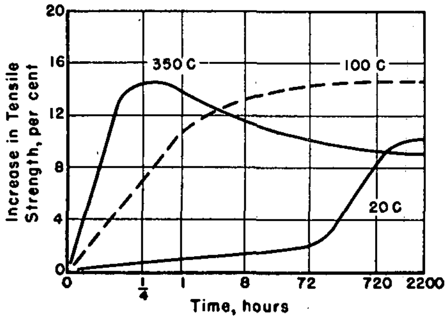


FIGURE 2. INFLUENCE OF TIME AND TEMPERATURE OF AGING ON TENSILE STRENGTH OF MILD STEEL (COCKRITZ)[34]

0-20669

Most weakly deoxidized steels containing less than 0.25% carbon are likely to "age". Like cold work, rapid cooling from elevated temperatures can set the stage for ageing reactions. Low (38), among others, presented evidence on the effect of quench ageing on hardness and toughness. He quenched samples of a 0.17% C, 0.75% Mn semi-killed steel from 690°C. and then aged them at room temperature. Within a few weeks the hardness increased from Rockwell "B" 75 to 90. The keyhole Charpy values at 25°C. decreased from 34 to 17 ft.-lb. on ageing at 25°C. for a year or more. Ageing at 25°C., for periods up to three years, gradually raised the temperature at which Charpy specimens broke with a brittle fracture.

For a particular testing method, the temperature or temperature range at which embrittlement occurs is a characteristic of the metal. This narrow range in testing temperature associated with a sudden loss in toughness is termed the "transition temperature". Low transition temperatures are preferable. They indicate that the steel is more resistant to embrittlement by rapid loading, localized stresses, or low temperatures. Recent investigators agree that the transition temperature is a better index of toughness than tensile ductility or notched-bar energy values. Therefore, the fact that Low (38) found that ageing raised the Charpy transition temperature from −40 to 0°C. is important. He also showed that over-ageing by reheating the specimens to 350°C., improved the toughness. The improvement was reflected both by energy values and by the transition temperature. This may partially explain the improvement in toughness imparted to weldments of semi-killed steels by heating after welding.

The consensus is that nitrogen is the primary cause of ageing (8, 11, 12). Since the solubility of nitrogen in ferrite is less than 0.001% at 20°C., most commercial steels are susceptible to ageing unless stabilized by some expedient. Ageing can be retarded or prevented by treating the liquid steel with proper amounts of aluminium (35), titanium (36), zirconium, or vanadium (37). All of these elements produce stable nitrides and all but vanadium are strong deoxidizers.

Annealed, rimmed or semi-killed steels free of strong nitride-forming elements are susceptible to strain ageing and exhibit some yield point elongation in tensile tests. That is, the metal stretches locally with no increase in load, so the stress-strain curve shows a flat region amounting to about 0.25 to 4% elongation in two inches. Low (11) showed that the amount of yield-point elongation increased with nitrogen content in tests on purified iron. During the deformation at the sharp yield point, jagged Luders lines appear on the surface of the specimen. Parts formed from steels of this type will often contain analogous surface defects which are called stretcher strains. Light cold-rolling treatments temporarily prevent the formation of stretcher strains in subsequent cold-forming operations. This treatment delays but does not prevent the ageing reaction which causes a return of the yield point elongation. Apparently (35) the prior cold work raises the rate of work hardening of the steel just beyond its elastic limit, enough to suppress the appearance of a definite yield point.

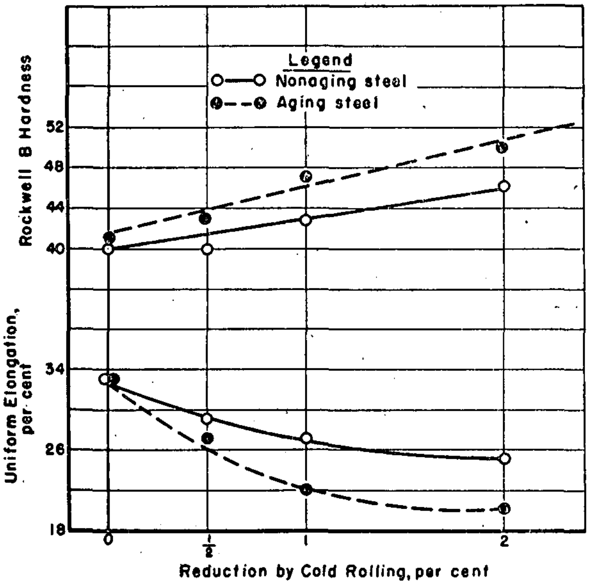
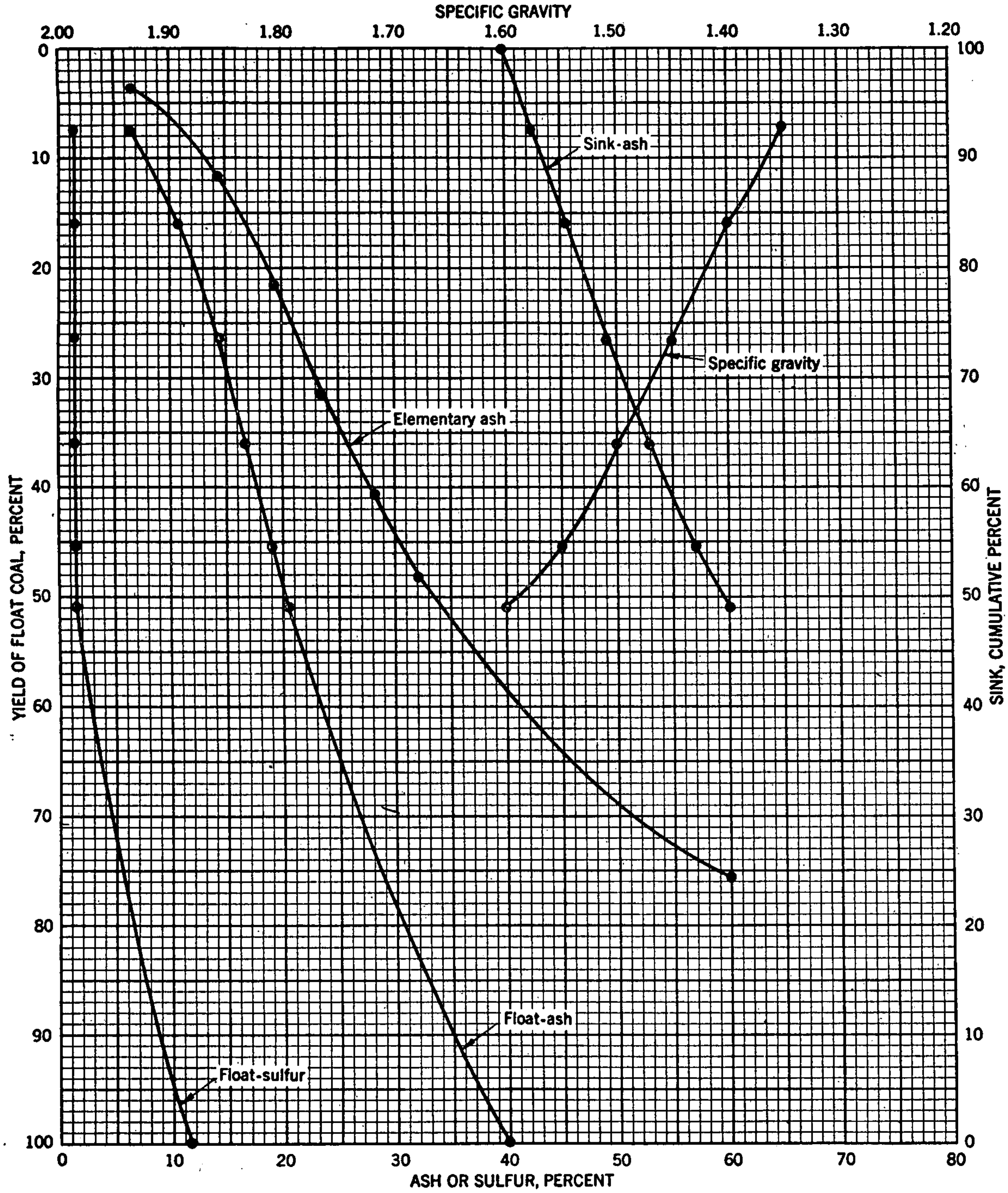


FIGURE 3. EFFECT OF TEMPER ROLLING ON HARDNESS AND ELONGATION OF RIMMED AND STABILIZED LOW-CARBON STEELS (EPSTEIN)[33]

0-20670

Even light cold-rolling treatment influences the mechanical properties of steels. Figure 3 shows some typical effects of temper rolling on the hardness and uniform elongation values of ageing and non-ageing steels reported by Epstein (33). The uniform elongation values are those for tensile specimens up to the point where necking started. Uniform elongation probably gives a better indication, than total elongation, of drawing quality. The differences in drawing quality between the two steels are reflected by the data in the chart. The differences of 5% in uniform elongation and four units in Rockwell "B" hardness after temper rolling 1% or more, are significant. The softer more ductile non-ageing steel would perform better in severe deep-drawing applications. The data in Figure 3 are for a typical rimmed steel, prone to ageing, and for an aluminium-killed non-ageing steel. Killed low-carbon steels often suffer the commercial disadvantages of higher ingot-product weight ratios and poorer surface quality than rimmed steel. Therefore, Epstein's use of vanadium to produce a rimmed non-ageing steel



Specific gravity fractions	Elementary data				Computed cumulative data					
	Weight grams	Percent			Float, cumulative percent			Sink, cumulative percent		
		Weight	Ash	Sulfur	Weight	Ash	Sulfur	Weight	Ash	Sulfur
Float - 1.35		7.2	6.7	1.24	7.2	6.7	1.24	100.0	39.9	11.46
1.35 to 1.40		8.9	14.3	1.25	16.1	10.9	1.25	92.8	42.5	12.26
1.40 to 1.45		10.5	19.5	1.39	26.6	14.3	1.30	83.9	45.4	13.42
1.45 to 1.50		9.5	23.6	1.33	36.1	16.8	1.31	73.4	49.2	15.15
1.50 to 1.55		9.2	28.2	1.35	45.3	19.1	1.32	63.9	53.0	17.20
1.55 to 1.60		5.6	32.1	1.85	50.9	20.6	1.38	54.7	57.1	19.87
Sink - 1.60		49.1	60.0	21.92	100.0	39.9	11.46	49.1	60.0	21.92

Fig. 14. Composite sample of the Barro Branco Coal crushed to 10 mm. top size.

(33, 37) is of considerable practical importance. The fact that vanadium, which is a weak deoxidizer comparable in power to manganese, prevents ageing is also of fundamental interest. It strengthens the conviction that nitrogen rather than oxygen is the principal cause of ageing.

It has been known for many years that aluminium-killed steels possess better low-temperature notched-bar properties than weakly deoxidized steels. Case (39) attributed the benefits of aluminium to the fixation of nitrogen as aluminium nitride. Recent work by Geil (40) and associates confirms this opinion. They made V-notch Charpy tests on aluminium-killed 0.30% carbon—1.7% manganese steels quenched and tempered to Rockwell "C". Geil, Carwile and Digges (40) concluded that increasing the aluminium nitride content from 0.01 to 0.08% lowered the transition temperature from −83 to −94°C. This improvement, although slight, shows the desirability of treating high-nitrogen steels with adequate quantities of aluminium. Their data indicate that nitrogen contents between 0.005 and 0.03% are not harmful, and may even be beneficial, in steels containing enough aluminium to form aluminium nitride after deoxidation is complete. The fact that nitrogen in this range lowered the Charpy values, at 20°C, from 65 to 48 ft.-lb. is probably of little practical importance.

2. Phosphorus

Phosphorus is an undesirable element in medium- and high-carbon steels because it causes embrittlement. Figure 4 gives data obtained by Baeyertz (7) illustrating the harmful effects of comparatively small amounts of phosphorus in quenched and tempered steels. These samples were cooled at the rate of 6.6° per minute to 370°C. after tempering at 620°C. Rapid cooling from the tempering treatment minimizes but does not prevent the embrittling effect of phosphorus (6, 7). Figure 4 shows that increasing the phosphorus content from 0.012 to 0.038% halved the room-temperature notched-bar values. The evidence that the temperature at which brittle fracture occurred in the Charpy tests was raised 80°C. is even more important. High transition temperatures, those at which the mode of fracture changes from tough to brittle, indicate unsuitability for services where impact loads, low temperatures or localized stresses are involved.

According to recent experience, the temperature at which the mode of fracture changes is a better criterion of toughness than the actual energy values.

Baeyertz and her co-workers (6, 7) also showed that replacing part of the manganese with molybdenum increased the tolerance for phosphorus. Molybdenum contents as low as 0.06% were beneficial, and steels containing 0.18% molybdenum and 0.035% phosphorus were tough in all laboratory tests. These data indicate that molybdenum additions are desirable for high-phosphorus steels which must be used in critical applications.

Phosphorus serves as a useful alloying element in low-carbon steels which need not be quenched and tempered. Each hundredth of a per cent of phosphorus raises the tensile strength of hot-rolled steel about 1,000 psi. Figure 5, from Epstein, Nead and Halley (2), illustrates the fact that phosphorus is much more effective than carbon in raising the yield strength.

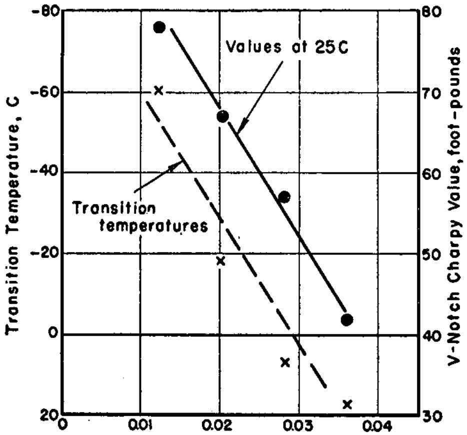


FIGURE 4. INFLUENCE OF PHOSPHORUS ON TOUGHNESS OF A 0.4 PER CENT C–1.8 PER CENT Mn STEEL QUENCHED AND TEMPERED TO ROCKWELL C 22(7)

0-20671

Consequently, phosphorus is frequently substituted for carbon in constructional steels which must be strong in the normalized or rolled condition. This substitution also improves the welding properties (5) and resistance to atmospheric corrosion (3). The characteristically high ratio of yield to tensile strength of phosphorus steels is advantageous in many applications. Very often, yield strength is a better basis for design than tensile strength.

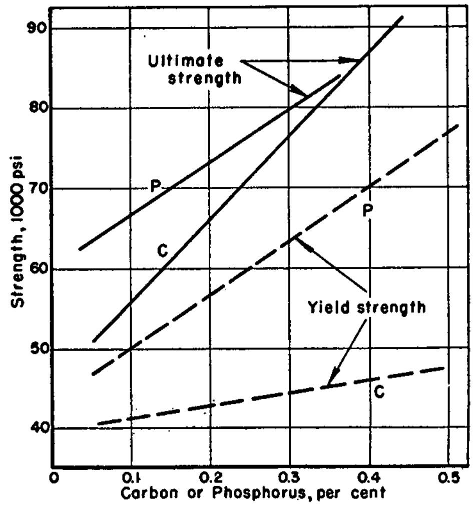


FIGURE 5. EFFECT OF CARBON AND PHOSPHORUS ON STRENGTH OF NORMALIZED STEELS CONTAINING 0.10 PER CENT C UNLESS OTHERWISE NOTED(2)

0-20672

Lorig and Krause (3) obtained good mechanical properties in some experimental high-phosphorus, aluminium-deoxidized steels in the normalized condition. However, phosphorus segregates badly in steels cast in large ingots and diffuses slowly. Consequently, such steels are likely to have a banded structure, with certain areas much richer in phosphorus than the average. Both carbon and phosphorus promote brittleness so the nominal total amount of these elements should be kept low, preferably below about 0.25% The combined effect of carbon and phosphorus on the Charpy notched-bar properties of Lorig's silicon-killed steels is shown in Figure 6. His work also showed that aluminium deoxidation benefited the toughness of such steels. Epstein (2) reported that a silicon-killed, normalized steel with 0.08% carbon, 0.12% phosphorus, and 1.0% copper had Charpy values of 47 and 34 ft.-lb., respectively, at 23 and −43°C. This steel combined 24% elongation, in an 8-in. gauge length, with yield and tensile strengths exceeding 60,000 and 72,000 psi respectively.

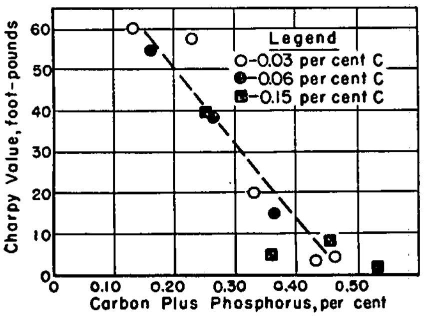


FIGURE 6. EMBRITTLING EFFECT OF PHOSPHORUS AND CARBON ON CHARPY SPECIMENS TESTED AT 23 C. SILICON-KILLED STEELS TESTED AS ROLLED(3)

0-70673

These and many other data show that properly deoxidized, mild alloy steels fairly high in phosphorus, have adequate toughness for many applications. Their high weight-strength ratios in the as-rolled condition and good weldability are desirable attributes for structural purposes. They are often used for truck, freight-car and earth-mover bodies and frames. Similar low-alloy steels have been used to a certain extent for automobile bumpers and bridges. Fine grain sizes are especially desirable in these low-carbon, high-phosphorus steels. The grain size is controlled by the deoxidation and rolling practice. Ordinarily, low finishing temperatures and fairly rapid cooling from the rolling temperature are preferable.

3. Phosphorus and nitrogen

All of the steels discussed in the preceding section contained about 0.004% nitrogen. This should be noted because phosphorus and nitrogen apparently have additive effects on the properties of steel. Enzian (8) presented considerable information on some silicon-killed, 0.14% carbon steels, containing various combinations of nitrogen and phosphorus. Room temperature Izod tests indicated that the toughness of normalized bars was unaffected by phosphorus in the range from 0.015 to 0.08% when the nitrogen content was below 0.006%. Similarly, the notched-bar toughness at room temperature of steels with less than 0.02% phosphorus was independent of nitrogen variations in the range from 0.005 to 0.013%. On the other hand, phosphorus in amounts exceeding 0.015% embrittled steels containing 0.012% nitrogen. Similarly, high nitrogen contents decreased the Izod values of normalized bars containing comparatively large amounts of phosphorus.

Most steels are subjected to at least small amounts of cold work during processing or service. Consequently, it is important to remember that rimmed and semi-killed steels are subject to embrittlement by strain ageing and that this tendency increases with the increasing amounts of nitrogen (9). Figure 7 shows some of Enzian's data (8) on the combined influence of phosphorus and nitrogen on the embrittlement produced by straining and then ageing one hour at 230°C. Apparently, these silicon-killed steels, with only 0.005% nitrogen, were relatively insensitive to the combination of straining and ageing embrittlement unless they contained 0.06% phosphorus or more. On the other hand, even the low-phosphorus steels containing 0.012% nitrogen were embrittled by the strain-ageing treatment employed in Enzian's experiments.

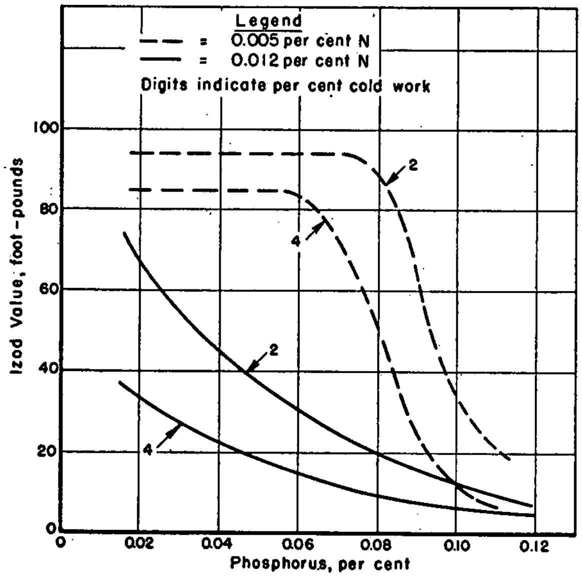


FIGURE 7. INFLUENCE OF COLD WORK AND NITROGEN CONTENT ON EMBRITTLEMENT CAUSED BY PHOSPHORUS. TESTS MADE ON BARS TEMPERED AT 230 C AFTER COLD DRAWING AMOUNTS INDICATED(8)

0-20674

Wright (5) demonstrated that aluminium deoxidation of steels produced in an acid-lined converter made them suitable for a variety of applications normally restricted to open hearth steels. His commercial Bessemer steels contained approximately 0.011% nitrogen, 0.09% phosphorus, and 0.15% carbon. Tensile data for a large number of such steels gave average elongation values of 35% in two inches with tensile strengths of 75,000 psi. These aluminium-killed Bessemer steels were as tough and ductile at normal and sub-zero temperatures as competitive open hearth steels of the same tensile strength. The killed Bessemer steel was no more sensitive to strain embrittlement than lower-phosphorus, higher-carbon steel of equal strength. Higher phosphorus steels have higher creep strengths (4, 5) at temperatures up to 530°C. which is sometimes an advantage.

4. Sulfur

Despite the poor reputation of sulfur, steels containing relatively large amounts of this element are widely used for many applications. The harmful effects of sulfur result from its influence on cleanliness, ductility and hot working properties. High sulfur contents cause trouble in hot working because some sulfides have comparatively low softening and melting points. The extent of such difficulties depends on the composition of the steel involved; low-carbon free-cutting steels containing 0.5% sulfur are processed commercially without unusual difficulty. Increases in manganese and decreases in carbon content help to minimize the hot-shortness of high-sulfur steels.

Ordinarily about half of the inclusions found in low-sulfur steel are sulfides. Obviously then, the proportion and the amount of inclusions increases with sulfur content. Sulfur can become important, therefore, in steels for applications requiring excellent surface finish, such as dies, moulds or bearings. Surprisingly enough, sulfur does not affect adversely the fatigue life or endurance limit of steel specimens (13, 14). The use of re-sulfurized alloy steels, containing of the order 0.06%, is an interesting recent development in the U.S.A. This fact is mentioned because the deleterious effects of sulfur have often been overemphasized.

Sulfur lowers the ductility and toughness of steel. Figure 8 shows some of Sims and Dahle's data (15) for normalized and tempered silicon-killed cast steels. This chart illustrates the effect of spherical sulfide inclusions. The deleterious influence of sulfur on tensile ductility and notched-bar values can be more pronounced in improperly deoxidized cast steels (15, 18) which contain film sulfides. Although sulfur usually lowers the breaking energy of notched-bar specimens, it seems to have little or no effect on the temperature separating tough and brittle fracture (18, 19, 20). This suggests that sulfur does not have a harmful influence on the behaviour of steels subjected to low temperatures or rapid loads.

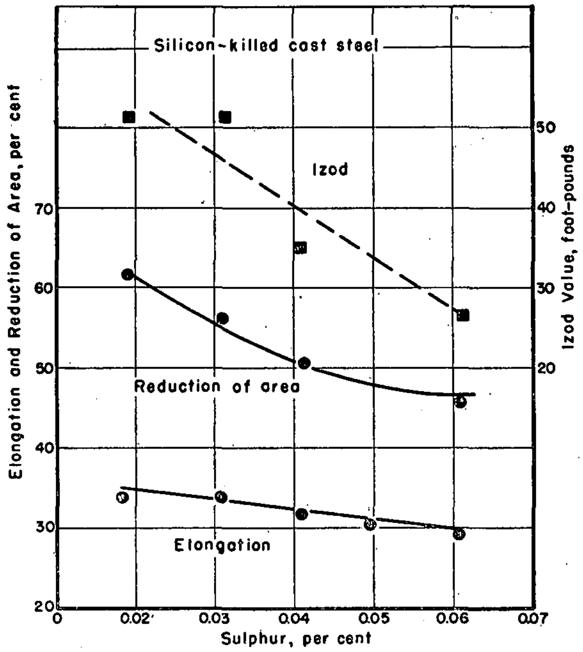


FIGURE 8. TENSILE AND NOTCHED-BAR PROPERTIES OF 0.30 PER CENT C, 0.75 PER CENT Mn CAST STEELS AT 24 C, NORMALIZED AND TEMPERED TO STRENGTHS OF 75,000 PSI(15)

0-20675

So far as wrought steels are concerned, tests at Battelle indicate that the orientation of the notch has a marked influence on test values of laminated steels. Samples with notches parallel to the surface, and the laminations, give abnormally high energy value. Rinebolt and Harris (31) agreed with this conclusion and noted that the tendency for laminations in their samples increased with sulfur content. Presumably, laminations interrupt the fracture and reduce the notch effect.. Boodberg and associates (32) also observed that sulfide inclusions can make steels tougher in notched-wide-plates. It appears, therefore, that the effect of sulfur on energy absorption in notched-bar tests on wrought steels depends on the nature and distribution of the inclusions as well as their quantity. This illustrates the general similarity of the sulfide shape effect on cast and wrought steels. It might also be mentioned that steels with the same level of sulfur content vary in machinability depending on the size of the sulfide inclusions present (27). Larger sulfides are preferred in free-cutting steels and their size depends on steel composition and rolling practice.

Mechanical tests on steels from 28 open hearth heats led one committee to conclude that sulfur contents up to 0.07% were permissible in wrought plain carbon steels containing less than 0.55% carbon and 0.70% manganese (21). The tests made during the investigation included tensile, bend, hot and cold upsetting, and notched-bar tests. No systematic variation in properties with sulfur was found in structural plate and forging grade steels containing up to 0.078 and 0.10% sulfur, respectively (17). The presence of sulfur in amounts exceeding 0.065% was accompanied by a decrease in transverse ductility of plate steels containing about 0.12% carbon (16).

The fact that sulfur reduces the ductility of steel is often unimportant because appreciable deformation destroys the usefulness of most articles. Ductility is a factor, of course, in fabricating operations involving cold forming. Low sulfur contents are desirable in steels intended for severe deep drawing (22), and many companies limit sulfur to 0.025% in steels of this type.

High sulfur contents are a disadvantage in steels to be fabricated by welding. Sulfur increases the likelihood of pinholes and porosity in the weld and of cracking during freezing or at high temperatures. These troubles are aggravated by segregation in the weld puddle, which can occur in shielded-arc or submerged-arc welding (26). Perhaps because the sulfides react with the flux, some of these difficulties are minimized by using electrodes with basic coatings (23). For example, lime-coated electrodes are useful for welding free-cutting steels containing around 0.10% sulfur (25, 26). It is impossible to set a sulfur limit which will avoid trouble in all cases. In some severe applications, as little as 0.015% sulfur in the base metal has been found injurious.

The incidence of weld-metal cracks depends on the welding practice and joint design as well as metallurgical or compositional factors. Heavy sections are more likely to crack because they impose more restraint on the weld and accentuate temperature differences. Stresses which cause hot cracks result from the temperature gradient between the weld and parent metal. Therefore, preheating or postheating operations may be desirable when welding high-sulfur steels.

The influence of sulfur on weld-metal cracking was investigated quite extensively by Reeve (24). His experiments were based on the reasonable assumption that the sulfur content in the weld, rather than that of the electrode or parent metal, is of primary interest. In his work, the volume of the weld metal was two or three times that contributed by the parent metal. Steel for welding rods should be low in sulfur, probably 0.035% or less, since electrodes usually furnish most of the metal in the weld bead. Usually, however, the type of electrode coating has a greater effect than the composition of the core wire (24, 30). Reeve's evaluations were based on the behaviour of 1 in. thick plates fillet-welded in tee joints; a design imposing comparatively little restraint. The sulfur limits given in Figure 9, therefore, refer to a particular welding operation and are not intended for specification purposes. Reeve's data in the chart indicate that carbon and silicon affect the sulfur limit which the weld metal can tolerate without cracking. Other factors being equal, increasing the carbon content of the parent metal, or the silicon, or sulfur content of the weld metal increased the crack sensitivity of the weld metal. It may be more than a coincidence that all three of these elements are deoxidizers. In fact, Reeve's data (24) show a correlation between lower oxygen contents and higher susceptibility to hot cracking in welds. Perhaps then, the sulfur limit for good welding properties is highest for rimmed steels and lowest for killed steels. Higher sulfur contents can be tolerated in lower-carbon steels destined for welding (24), because carbon also promotes cracking (25).

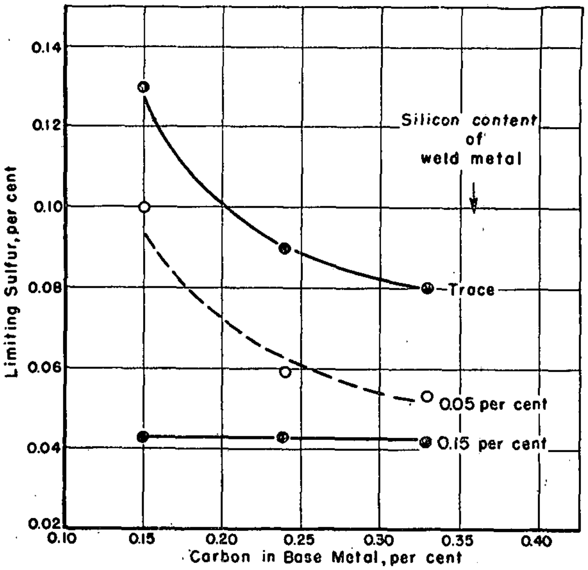


FIGURE 9. INFLUENCE OF CARBON IN PLATE AND SILICON IN WELD ON THE PERMISSIBLE SULFUR CONTENT OF WELD METAL IN WELD-CRACKING TESTS BY REEVE (24)

O-20676

Sulfur also has some beneficial effects on steel properties. The improvements in machinability resulting from higher sulfur contents are too well known to require any elaboration (27). Moreover, sulfur influences the austenitic grain-growth characteristics of steel. Baeyertz (28) showed that increasing the sulfur content from the normal level up to 0.10% raised the coarsening temperature of cast steel about 100°C. This was true for both silicon-killed and aluminium-killed steels. The effect of sulfur on grain growth was less marked in the few wrought steels studied. Sims also showed (30) that sulfur content influenced the grain coarsening temperature resulting from an optimum aluminium addition. His cast steels containing 0.038% sulfur had higher coarsening temperatures than steels with lower sulfur contents.

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Standardization in Relation to Control in Steel Production

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INTRODUCTION

Some measure of standardization and simplification is now regarded as indispensable for the efficient running of any major manufacturing industry. It is over 50 years since the first steps were taken to introduce simplification in steel production. This was a reduction in the number of sizes of various types of rolled sections such as structural shapes and tram rails and the effects of this are still felt and will continue to be felt for a considerable time to come.

Dealing with the subject of the meeting it is evident that there are many advantages to the steelmaker in having standard specifications and methods of test. Consider first the end of the steel rolling process which after all is the point at which the specifications must be met. If the steelmaker is working to a limited number of specifications he is able, in happier times than the present, to hold stocks sufficiently large to ensure prompt delivery of urgent orders. Moving backwards in the steelmaking process, the rolling mill and melting shops are able to plan their programmes more economically. From the dimensional point of view the number of roll changes is kept down and from the quality point of view the number of batches being processed can be kept high with less risk of mixing. In addition, when cold rolling, and to a certain extent hot rolling, strip, etc., is involved, the reduced number of qualities means reduced storage at the intake and, in the absence of continuous mills, reduced storage between mills.

In the melting shop the reduced number of types means that more attention can be given to the control of quality, and operators are able to develop to the full that judgment of temperature and condition of slag which is an invaluable adjunct to any system of technical control.

BASES OF SPECIFICATIONS

The form in which specifications are cast can now be considered and here a distinction must be made between private specifications and standard specifications. A private specification in its simplest form is a statement of what the customer requires and is only limited by what he knows the steelmaker can supply. A standard specification on the other hand is a statement of what the users and the steelmakers as composite bodies consider to be the minimum requirements which will ensure a suitable quality of steel. A review of standard specifications shows that the following factors are used:

(a) *Mechanical properties:*
  (i) Tensile strength;
  (ii) Proof stress;
  (iii) Yield stress;
  (iv) Elongation;
  (v) Reduction of area;
  (vi) Elongation at point of constriction;
  (vii) Hardness;
  (viii) Impact value;
  (ix) Bend ductility;
  (x) Upsetting ductility.

(b) *Chemical composition.*
  (i) Impurities;
  (ii) Alloying elements.

(c) *Response to heat treatment:*
  (i) Jominy hardenability;
  (ii) "Mass effect".

The determination of one or more of these properties is necessary, not only to show that the steel reaches the required standard but also to check the quality during production, irrespective of the terms of the specification; e.g., the specification may call for mechanical properties only but during melting the control of chemical composition within limits based on experience will be necessary to ensure that the steel is likely to have the specified properties.

It will be found that on the basis of these factors there are three large groups of specifications:

(*a*) Those specifying mechanical properties only;
(*b*) Those specifying chemical composition only;
(*c*) Those specifying mechanical properties and chemical composition.

In each group other factors are included but the three main subdivisions will serve as a basis for the following discussion.

Specifications based on mechanical properties only

This group is practically confined to carbon steels for use in the hot rolled condition and to a much smaller extent in the normalized condition. Such specifications cover by far the biggest tonnage of steel used, and the mechanical tests are designed essentially to reproduce conditions to which the steel will be subjected in practice.

All these specifications include tensile and ductility tests. The determination of the tensile strength by the tensile test is to show that the steel will withstand the stresses for which the structure is designed. The ductility test is to ensure that should the design stresses be exceeded in any particular part or member, plastic deformation will occur, the appearance then giving warning of overstress or the deformation allowing redistribution of the stress to take place without failure or with the part merely bending to unserviceability.

Tensile strength, which is always specified, is sometimes used as a primary basis of design but nowadays it is common to base designs on yield stress; e.g., B.S. 449 "The use of structural steel in building" in which all working stresses are in terms of the yield stress which is guaranteed by the two specifications for structural steel, i.e., B.S. 15 "Structural steel for bridges, etc. and general building construction" and B.S. 968 "High tensile (fusion welding quality) structural steel for bridges, etc., and general building construction". A minimum tensile strength is also specified for the material and the range between this and the yield stress will give some idea of the ultimate factor of safety.

Proof stress is not commonly specified for the type of steel being considered.

Ductility is generally expressed in terms of elongation in the tensile test but it is also common practice to specify a bend test in addition. The bend test is often done on a larger number of samples than the tensile test and gives some protection against accidental mixing of high tensile steel with the batch.

To meet the combination of minimum tensile strength and minimum ductility the steelmaker must control the carbon content, the minimum being fixed by tensile strength and the maximum by the ductility. It is also necessary to control the manganese and residual alloying elements. Further, in the rolling mill the finishing temperature must be controlled. Specifications generally allow reduced ductility for thicker sections where it is necessary to increase the carbon or manganese content, or both, to attain the tensile strength.

Steels in the higher tensile range are rarely specified by mechanical properties alone because generally speaking there are three ways of increasing mechanical strength each giving a different type of steel, and if any subsequent working is to be carried out the types are not interchangeable.

The three methods of increasing the tensile strength are first simply increasing the carbon content, secondly increasing the content of alloying elements and thirdly by cold working.

The first two are usually accompanied by heat treatment and in British practice the specifications for these normally include a temperature range for the heat treatment to be given to the particular steel. Similar information is included in American specifications in the form of recommendations.

Specifications based on chemical composition only

Turning now to the use of chemical composition alone, one leaves the field of structural work and is concerned with steel mainly used in the workshop. In this group there is a difference in practice between European countries and practice in the United States. Only in recent years have there been British standards for ordinary carbon and low-alloy steels based on chemical composition alone. The reason for the two practices is the tonnage involved. If a steel is to be bought on chemical composition alone the content of the various elements must be known fairly closely if the mechanical properties of the steel are to be in any way predictable. This applies particularly to the carbon content. The limits which can be allowed are generally close to or closer than the limits which can be worked to economically from heat to heat; for example, if a 0.3% carbon steel is ordered, the user needs to be sure that the carbon content will not vary beyond say 0.27 to 0.32% carbon, otherwise the strength and ductility will not be what he requires. These limits are very similar to those which can be expected on cast analyses from heat to heat and because of heterogeneity of the steel an occasional heat will fall outside the limits.

Examination of the American steel compositions shows that this is overcome by having a series of specifications with contiguous carbon ranges so that a heat having a composition outside that aimed at will come within another specification. This practice leads to a large number of specifications and is only practicable where there is a very large market as in the United States.

In British specification practice the use of chemical composition alone, in so far as carbon steels is concerned, is practically confined to sheet, strip and wire steels and the reasons for this are dealt with later in this paper.

In the above discussion we have considered steels for application where mechanical strength is the main requirement but there is wide application for steels where ductility is the overriding factor. This is so in the case of steel sheets for pressing, forming, drawing, spinning, etc. Up to the present time no satisfactory test has been developed for determining the amenability of steel to any but the simplest cold forming operations. Simple tests such as the bend test unfortunately do not bear any relation to the complex deformations which take place in spinning and deep drawing. In such processes the combined effect of ductility and work hardening is so complex that it cannot be analysed into terms which can be the subject of simple tests. However the chemical composition of the steel is probably the most important single factor, and in general, the more drastic the deformation, the lower must be the carbon content. Therefore the close control of chemical composition is essential. Hence there are a large number of specifications for low carbon steel sheet which are in terms of chemical composition only and these call for close control in steelmaking.

The earlier remarks on American practice apply not only to plain carbon steels but also to the full range of alloy steels, although where there are a number of alloying elements such as nickel, chromium and molybdenum, the number of specifications which would be needed to ensure that any composition would fall within a specification would be quite impracticable. Accordingly there are a number of steel types with alloying elements present within comparatively wide limits but with the carbon content restricted to close limits in each specification.

Specifications based on both chemical composition and mechanical properties

Specifications based on mechanical properties combined with limits for the impurities, sulfur and phosphorus, cover a large proportion of the steel made—even more than the proportion covered by specifications based on mechanical properties alone.

Having discussed mechanical properties in an earlier section, discussion in this section is confined to sulfur and phosphorus. Another paper in this meeting will deal with this subject more fully but broadly it can be said that for any given process, not only is the presence of increasing amounts of sulfur and phosphorus progressively detrimental but such an increase often indicates lack of cleanliness in the steel.

National standards vary somewhat in their requirements dependent mainly on the sulfur content of the coke available, the phosphorus content of the coke, ore, and fluxes and on the process used. In the United Kingdom the standard limits for structural steel and a large range of other steels are 0.06% each of sulfur and phosphorus, but in other countries where the acid Bessemer process is used to a large extent the phosphorus limits may be much higher.

Over many years there has been considerable argument on the effect of sulfur and phosphorus and just what limits are really necessary. No doubt the argument will continue for many more years but whatever limit is fixed there will always be cases of casts slightly above the limit. These can often be accepted by the inspecting authority when account is taken of the manganese content and the treatment to which the steel will be subjected but this procedure should not be taken as a reason for gradually raising limits.

While dealing with impurities slag inclusions should be considered. The proportion of such inclusion is rarely specified quantitatively although most specifications say that the steel is to be reasonably clean or include some such general statement. Requirements may be rather more stringent in the case of high duty steels such as spring steels. The A.S.T.M. has produced a series of standards for limits of slag inclusions and these take into account the form of the inclusion. These standards will be helpful in assessing the cleanliness of the steel when controlling production.

Limits for other impurities including adventitious alloying elements in carbon steels are not usually specified but in the earlier part of this paper reference is made to their control.

Alloy steels, including pearlitic manganese steels, which are generally heat treated and in which strength is the main requirement (i.e., excluding stainless and heat resisting steels) are generally specified in terms of comparatively full chemical composition in addition to mechanical properties.

It is difficult to discuss the limits of chemical composition and mechanical properties for these steels without bringing in the effect of hardenability. The way in which this is treated in the United Kingdom is different from its treatment in the United States.

Considering British practice as exemplified by the En specifications for engineering steels in B.S. 970 "Wrought steels for automobile and general engineering purposes" the limits for carbon and alloying elements are comparatively wide and serve to define the type of steel rather than to prescribe the limits of analysis for steel making. In the preparation of B.S. 970 the view has been taken that provided a steel is of a given type as defined by the broad limits of composition and provided it will give the mechanical properties when heat treated in a specified way it will be satisfactory. Thus the real control is through mechanical properties which must be obtained using a standard heat treatment. These properties are modified according to the so called "mass effect" which from a metallurgical point of view is the effect of the size of the part or test piece on the cooling rate during quenching. The heat treatment being specified, the variable is expressed in terms of the "ruling section", i.e., the diameter of a round section which will give a cooling rate equivalent to that which will be obtained in the most important part of the article when it is heat treated. By testing a sample having a diameter corresponding to the limiting ruling section of this part the user is assured that the strength of the part after heat treatment will be not less than that specified.

A method of arriving at the equivalent diameter is given in B.S. 971 "Commentary on British standard wrought steels En series" which includes some useful notes on the effect of variations in heat treatment conditions for most En steels.

In the United States until recently, control was entirely by means of close limits of chemical composition as mentioned earlier in this paper. During the last few years, however, a series of specifications in terms of Jominy hardenability have come into being. This method of specification is an attempt to obtain a range of rates of cooling with a single specimen whereas in B.S. 970 only one rate of cooling (i.e., that associated with the normal quenching of a bar of given diameter) is associated with any one test.

In principle the two methods described are similar in so far as the strength developed can be related to the rate of cooling. For example, take a case in which the American hardenability test is used and the distance between which a certain hardness is to be obtained as specified; this distance corresponds to a certain rate of cooling and hence, for any given quenching conditions corresponds to certain dimensions in the part to be manufactured. Thus one can expect that the requisite hardness (and therefore the tensile strength) will be developed in that part. It should be mentioned that this argument is not strictly accurate because it does not take into account the time rate or change of cooling rate which has an influence on the transformation temperature, but it is sufficiently true to indicate the similarity between the practices in the two countries.

An investigation of the hardenability of steel, mainly in terms of Jominy hardenability, was conducted in the United Kingdom between 1943 and 1945 under the aegis of the Technical Advisory Committee of the Special and Alloy Steel Committee (Ministry of Supply) and the Iron and Steel Institute (Iron and Steel Institute Special Report No. 36). No over-all conclusions were reached as a result of this investigation but it is significant that there has been no change in the treatment of mass effect in the 1947 revision of B.S. 970.

The shortage of steel alloying elements in Western Europe following the Second World War provides an illustration of the manner in which a limited range of alloy steels, controlled essentially by mechanical properties obtainable after heat treatment and in terms of the "ruling section", can be made to cover essential needs.

The British Ministry of Supply issued in April of this year a directive making certain alloy steels available only under licence. At the same time it also issued a tabular schedule, reproduced below, in which the appropriate alloy steel specification taken from B.S. 970, is listed under a range of tensile strength obtainable after suitable heat treatment against the ruling section operating.

By encouraging designers to select the British standard alloy steel from the table appropriate to the conditions they wish to meet, and discouraging them from selecting steels of enhanced properties, a voluntary effort is being made to use a limited supply of alloying elements such as nickel and molybdenum in the most economic manner possible. This is one of the ways in which the concept of "ruling section" is used in practice.

It is suggested that the adoption of similar principles in other countries, for example in Latin America, might be worthy of detailed consideration.

In certain cases some of the steels in B.S. 970 have been specified only in terms of close limits of chemical composition on the lines of American practice. In such cases it is the responsibility of the user to adjust his heat treatment so that he obtains the properties he desires.

Under this heading of specifications based on both chemical composition and mechanical properties cold rolled steels can conveniently be considered. Naturally the main criteria in specifying such steels are the mechanical properties, but from the point of view of the steelmaker the chemical composition must be controlled as this has a marked influence on the extent of cold reduction which will be necessary to meet mechanical properties. In general the chemical limits, of carbon in particular, are comparatively wide but this is to some extent due to the general practice of cold rolling being done in an organization separate from that responsible for the hot rolling. However, each individual cold rolling mill will need to specify comparatively close limits so that the extent of cold reduction can be fixed without producing too much variation in the final product. For example, A.S.T.M.A. 109-49T "Cold rolled carbon steel strip" specifies only a maximum carbon content whereas B.S. 1449 En. 2 "Ordinary quality steel plate, sheet and strip" goes even further, specifying only mechanical properties, leaving the choice of carbon content for arrangement between the cold roller and the steelmaker.

Specifications including reference to steelmaking process

Certain specifications restrict the steelmaking process to, say, the acid open hearth or electric process, but except for special steels this practice is declining. However, one aspect of steelmaking, i.e., deoxidation, is sometimes mentioned.

Fully killed steel is cleaner than "balanced" or "rimming" steels. In specifications for high duty steels this point is covered either by specifically stating that rimming steels must not be supplied or by fixing a minimum silicon content sufficiently high to ensure that deoxidation is complete. Where there are no such

Table "A"

STEELS FOR HARDENING AND TEMPERING EN SERIES

Properties as treated tons/sq. in.		*R 45/55*	*S 50/60*	*T 55/65*	*U 60/70*	*V 65/75*	*W 70/80*	*X 75/85*	*Y 80/90*	*Z 100 min.*
		15 or 18	15 or 18 –⅞ in.	18 or 100 or –111 –1⅛ in.	100 or 111	100 or 110	110 or 24	24	24	24 or 110[c] or 111[c] or 30[d, e]
			18	100 or 111	100[a] or 110	24	24[b]	25[e] or 26[e]	25[e] or 26[e]	25[e] or 26[e] or 30[d, e]
Ruling Section at time of heat treatment	2½ in.	18 or 100	100[a] or 111	110	24	25[e] or 26[e]	25[e] or 26[e]	26[e]	26[e]	26[e]
	4 in. 6 in.	100 or 111		24	25[e] or 29[e]	26[e]	26[e]			30[e]

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[a] B.S. 970 quotes S properties on En. 100 only up to 4 in. and U properties only up to 1⅛ in.
[b] B.S. 970 quotes W properties on En. 24 only up to 1⅛ in.
[c] B.S. 970 does not quote 100 ton/sq. in. properties on En. 100 or En. 111. To attain 100 tons/sq. in. a higher carbon content (0.35/0.45%) is necessary for En. 111.
[d] En. 30 may be used up to 2½ in. for air hardening and tempering only.
[e] The molybdenum content of En. 25, En. 26 and En. 29 must not exceed 0.60% and of En. 30B must not exceed 0.30% without an Alloy Steel Assent.

restrictions a "balanced" steel is normally supplied, particularly for structural steels, because of the small amount of cropping necessary. On the other hand the use of rimming steels is often preferred for steel sheet as this type of steel has a clean skin which gives a good finish when cold rolled and subsequently formed.

For steels which are to be used at elevated temperatures it may be expected that in the future, limits will be placed on the amounts of aluminium which may be added for deoxidation. It has been found that large additions of aluminium can produce steel with high creep rates ("Creep resistance of wrought carbon steels", Jenkins and Tapsell, *Journal of the Iron and Steel Institute*, vol. 171, part 4, August 1952, pp. 359–371). This effect of aluminium has been taken into account in the recently issued draft revision of B.S. 806 "Ferrous pipes and piping installations for and in connection with land boilers" in which it is proposed that for 1.2% molybdenum and low chromium-molybdenum steel pipes the aluminium addition, if any, should not exceed 0.5 lb./ton.

METHODS OF TESTING

In the foregoing discussion consideration has been given to the factors used in controlling quality, i.e., in defining a steel for the purpose of a specification. In this section the *modus operandi* will be dealt with and current practice in testing and analysis reviewed.

Tensile test

In tensile testing the most important factor is the form of the test piece which affects the elongation. It has been established that, for a test piece of circular cross-section, if the gauge length bears a constant ratio to the square root of the area of the test piece between the gauge marks the elongation figures are directly comparable. These test pieces are referred to as "proportional" test pieces. The ratio of the gauge length to the square root of the area differs in different countries, the common values being 4, 5.65, 8.16 and 11.3. At a recent meeting of the Technical Committee on steel, of the International Organization for Standardization, it was recognized that international agreement on one value for this ratio was desirable and although provision must still be made for values of 4, 8.16 and 11.3 it is strongly recommended that at the earliest opportunity a figure of 5.65 should be adopted. As many existing specifications include elongation figures based on the use of ratios other than 5.65 some time must elapse to allow correlation tests to be carried out before the elongation on the ratio 5.65 can be specified.

Non-proportional test pieces are particularly useful in controlling production of items such as plates in which various thicknesses are being produced. To machine test pieces with different widths according to thickness so as to obtain a uniform cross-section is rather expensive. If a flat test piece of standard width irrespective of thickness is used, the test pieces can be prepared by clamping together a number of test plates and milling in batches. For this reason a number of specifications for plates specify elongation for a fixed gauge length irrespective of thickness. This procedure is perfectly satisfactory because the elongations specified are generally those which can be met quite readily by a well-made steel and as mentioned earlier the exact elongation figure has no real meaning from the point of view of the end use of the steel. However, should it be necessary to convert elongations from one set of conditions to another, correlating graphs such as those in British Standard Handbook No. 13—Mechanical tests for metals (pp. 32–38) are in existence.

Proof stress is commonly specified for high grade steels, particularly for use in aircraft and in British practice is defined as the stress which is just sufficient to produce under load a non-proportional elongation equal to a specified percentage of the original gauge length. Methods for its determination are given in B.S. 18 "Tensile testing of metals." The direct method of measuring proof stress is from the load/extension diagram but in B.S. 3A4 "Test pieces and test methods for aircraft metallic materials" a "4 point" method is described. This involves taking only four readings and from these readings it is possible to calculate the proof stress provided the upper limit of the proof stress is specified or can be estimated. If, as is usually the case, only a minimum proof stress is specified a further simplified method which involves taking only three readings can be used as described in B.S. 18 and B.S. 3A4.

The yield point is readily determined on hot rolled steels by measuring the stress at which there is an increase in the elongation without any increase in the load. In higher tensile steels and cold worked steels the yield point becomes indistinct and in such cases British practice as described in B.S. 970 is to take the 0.5% proof stress.

The determination of elongation and reduction of area call for no comment apart from the reference to the use of proportional test pieces which is considered above.

In some European specifications the elongation at the point of constriction is specified. This is the ratio of the reduction in area after fracture to the area after fracture and is of course determined as easily as the ordinary figure for reduction of area.

In general the tensile test is one of the most important tests used in controlling the production of structural steel. However, it is comparatively expensive and for adequate control it is usually supplemented by hardness and bend tests which can be used more cheaply and expeditiously.

Hardness test

The existence of an approximate relationship between hardness and tensile strength enables hardness tests to be used as a means of control.

There are various hardness tests all of which with the exception of the scleroscopic test measure the resistance of the steel to penetration by a loaded indentor. The three most generally used are Birnell, Vickers and Rockwell tests which are described with little variation in most national standards. The Brinell test using a steel ball is generally the simplest but the Vickers test with a diamond pyramid indentor has the advantage of giving a more clearly defined impression for measurement. In addition the Brinell test can only be strictly called a Brinell test when hardened steel balls are used. The balls tend to distort when testing hard steel and this test should not be used for Brinell hardness numbers above 450.

The Rockwell test has the advantage of being direct reading and since the measurement is of the increase of depth under a given increase of load, the effect of elastic recovery of the steel being tested is eliminated.

The choice between the three methods is very much a matter of individual preference and is best related to the general practice in the specifications which must be met by the steel works.

In many publications there are tables correlating approximately the hardness obtained by the three methods and these can be used quite satisfactorily for most steels.

Impact tests

Impact tests are used to measure the susceptibility of a steel to the propagation of a fracture under the influence of and caused by shock loading. They are generally either the Charpy beam type or the Izod cantilever type. These again are adequately described in national standards. The most important factors in the test are the shape of the specimen and the form of the notch. It is urged that wherever possible the standard forms are used. For the Izod test the specimen should be 10 mm. sq. with a 2 mm. deep notch having an included angle of 45° and a root radius of 0.25 mm. For the Charpy test the specimen should either have the same dimensions as for the Izod test or else have a U notch or key-hole notch 5 mm. deep rounded off at the base with a radius of 1 mm. However, for control purposes other sizes and forms of notch are used, particularly when determining the transition temperature. Provided there is a reasonable idea of the relation between the value of the standard test piece and the value of the test piece used, non-standard test pieces are quite satisfactory as the transition temperature is probably of more interest than the exact value of the resistance to impact.

In future we can expect to see more specifications including low temperature impact properties. The low temperature notch embrittlement of steel has received considerable publicity over the last decade and the most general method of determining the susceptibility of a steel to such embrittlement is to determine the impact value at the lowest temperature likely to be met in practice. For example, B.S. 1500 "Fusion welded pressure vessels for use in the chemical and allied industries" calls for a minimum Izod impact value of 20 ft.-lb. at the lowest operating temperature when this is below 0°C.

The mechanism of low temperature embrittlement and its cause are not fully understood but two of the factors generally accepted as having an influence are grain size and extent of deoxidation. These may be related but it is generally accepted that fine grain steel deoxided with aluminium gives higher impact values and a lower transition temperature. This point should be borne in mind when making steel for low temperature service such as in the chemical industry and in shipping.

Bend test

The bend test is a quick and easy test of ductility. For works control purposes it can be done by clamping the specimen and former in a vice, hammering the test piece through an angle of something more than 90° and then clamping to a U form in a vice, the former being placed between the legs of the test piece. However, when testing to a specification rather more care needs to be taken to ensure that the test piece is in contact with the former so that the radius of the crown of the bend is not much less than the specified radius and that the edges are smooth enough to prevent undue stress concentration which might result in preliminary failure.

Upsetting test

For rivet steel the ductility can be determined by bending and also by upsetting the formed rivet head. B.S. 15 "Structural steel" specifies flattening the head hot until it is 2½ times the diameter of the shank. There should be no evidence of cracking after this flattening.

Chemical analysis

In a paper such as this it is not possible to consider in detail the methods of analysis used in the steel industry. However, attention is drawn to the methods published by the American Society for Testing Materials "Methods of chemical analysis of metals" and B.S. 1121 "Methods for the analysis of iron and steel". Both these publications set out standard methods which are intended for analyses used as a basis of acceptance of steel. In routine analyses simpler methods are available and such methods are published by individuals, societies and certain steel companies.

For the calibration of what might be termed secondary methods, for example, spectographic methods, the samples must of course be analysed by accurate methods such as those described in the two publications mentioned above.

There are available specially manufactured homogeneous samples which have been analysed by a number of first-class analysts by the best available methods and these are very useful for checking the method of analysis being used. These samples, being made by production methods contain all the elements usually present in a works sample and naturally include those elements which must be separated or otherwise taken care of in a normal analysis. Such elements are not usually present in synthetic samples which are generally based on pure elements.

CONCLUSION

This paper has outlined the principles underlying the control of steel types according to national specifications in highly industrialized countries, with special reference to the factors which operate in Great Britain. It has to be recognized that some of these factors are peculiar to geographical and temporal conditions which are not necessarily relevant in the case of Latin-American countries.

At the same time the outline does give the fruits of experience in a major steel producing country which, under present world conditions, has to forget the times of plenty and make all its resources stretch to the uttermost, while maintaining an international reputation for quality which automatically ensures a guaranteed minimum performance in its products. It may well be that in underdeveloped countries, endowed with rich natural resources, the rationalization achieved in an industrialized Western European country will set a first goal, admittedly not easy of rapid achievement, for the developing steel industries.

Hence, although the Barro Branco coal presents the most complicated technological difficulties, the final solution is still dominated by economic factors; and it is necessary to set up the most favourable possible tolerances for both metallurgical- and steam-coal products before definite criteria for technical operations can be established. The cost per unit of sulfur for handling this impurity in the blast furnace naturally constitutes the final referrent factor.

In considering the Barro Branco coal as an example of the more difficult problems, it is necessary to consider it unique in one major respect; that is, it is especially responsive to reduction in grain size for sulfur reduction.

Experience with most North-American coals and test data available on many coals from the Andean region indicate that fine crushing promises only an insignificant improvement in washed-coal quality, usually insufficient to justify the increased operating cost and hazard of sludge losses.

In the general case, it is probably most economic to select the best local source of coal, based on coke-structure and transportation cost, then wash this coal by well-proved conventional methods, and use it as delivered from the washery or blended with small proportions of special coals as may be necessary to obtain the necessary physical characteristics of blast-furnace coke.

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Modern Techniques and Installations for the Mechanical Treatment of Coal by the PIC Company

JACQUES TURPIN

DESCRIPTION OF MODERN EQUIPMENT MANUFACTURED BY THE PIC COMPANY

France being very poor in easily exploitable deposits, French engineers, specialized in coal preparation, have been obliged to solve very difficult problems and to carry out elaborate research work in order to determine what sort of washing equipment would treat satisfactorily the most difficult coal and to study washery circuits leading to the maximum recovery of saleable products.

Conversely, the easy coal washing conditions met with in the United States, and primarily in the United Kingdom, did not so often oblige engineers in these countries to modify their original washing equipment, whilst such equipment has almost totally and progressively been eliminated in France, Germany and Belgium, to be replaced by methods and units more appropriate to their coals.

A short description follows of the main modern cleaning equipment manufactured by the PIC Company, especially the washing devices.

(a) *The "Drewboy" large-coal heavy-medium washing process*

The washing unit manufactured by the PIC Company and known as the "Drewboy" is now in operation in France (a great number having been ordered by the Charbonnages de France—the French collieries), in the Saar, Belgium and Yugoslavia. Furthermore, a "Drewboy" installation is being built for the Empresa Siderúrgica Nacional de Paz de Río (Colombia). The main features of the unit are: none of the mechanical parts is bathed by the medium; the plungers climb through a slanted wheel which leads to a very large opening, permitting the treatment of coal lumps as large as one metre; the great evacuation capacity of the wheel for the plungers (buckets) permits the treatment of raw coal in which the proportion of shale can reach 80 or 90%; the raw products reach the liquid bath directly, with no risk of breaking the floating products or of enclosing them among the shales; the apparatus can be built in very large units, capable of an output amounting to 500 tons per hour of crude material.

(b) *Large-coal washing jigs*

For the purposes of this paper, products exceeding 10 mm. in size, are defined as "large coal", and those below 10 mm. are defined as "small coal". In France, the name of "Baum jig" is usually given to jig wash-boxes handling raw coal unclassified before washing.

For twenty-five years the PIC Company has been manufacturing, for the cleaning of large coal, jig wash-boxes fitted with automatic devices for the discharge of shale and middlings, by means of automatic units. The simplicity and safety of these units have induced most of the French mine owners to substitute this type of equipment for the ancient washing devices which were uncertain, difficult to adjust and of questionable accuracy.

In their modern design, the PIC wash-boxes include: compressed air pulsation and automatic controls fitted with hydraulic servo-motors.

Compressed air from the receiver is sent by an air distributing valve, at regular intervals, to the various air chambers, and then released, thus transmitting to

In the fullness of time it is possible that the Latin-American steel industry will be able to set itself even greater objectives in the variety and range of steels it is able to produce to meet the increasingly stringent demands of users.

Appendix

AMERICAN AND BRITISH SPECIFICATIONS FOR TESTING STEEL AND STEEL PRODUCTS

American

ASTM

E8-47T	Tension of metallic materials.
E10-27	Test for Brinell hardness of metallic materials.
E18-42	Tests for Rockwell hardness and Rockwell superficial hardness of metallic materials.
E48-47	Hardness conversion tables for steel.
E23-47T	Impact testing of metallic materials.
E16-39	Bend testing for ductility of metals.
E21-43	Elevated temperature short time tension tests of metallic materials.
E22-41	Conducting of elevated temperature long time tension tests of metallic materials.
E19-46	Classification of grain size in steels.
E45-46T	Determination of inclusion content of steel.
E2-49T	Preparation of micrographs of metals and alloys.
E3-46T	Preparation of metallographic specimens.
E14-33	Thermal analysis of steel.

British

B.S. 18	Tensile testing of metals.
B.S. 240	Part 1: Method and tables for Brinell hardness testing. Part 2: Steel balls for Brinell hardness testing.
B.S. 427	Diamond pyramid hardness numbers (Vickers).
B.S. 891	Direct reading hardness testing, Rockwell principle.
B.S. 860	Approximate comparison of hardness scales.
B.S. 131	Notched bar test pieces.
B.S. 1639	Notes on the simple bend test.
B.S. 485	Tests on thin metal sheet and strip.
B.S. 1271	Proof test for creep quality of carbon steel boiler plate.
B.S. 1686	Equipment and method for long period, high sensitivity tensile creep testing.
B.S. 1687	Equipment and method for medium—sensitivity tensile creep testing.
B.S. 1688	Equipment and method for the determination of time to rupture under stress, with or without measurement of creep strain.
B.S. 1121	Methods for the analysis of iron and steel.
B.S. 1837	Sampling of ferrous metals and metallurgical materials for analysis: Part 1: Iron and steel. Part 2: Ferro-alloys (shortly to be published).

The Control of Composition During Steelmaking

G. WESTON, B.Sc., in collaboration with G. R. BOLSOVER, A.Met., F.I.M., F.I.P.

The subject is one which lends itself to treatment in a variety of ways. Not only are methods of determination of composition involved, and these are now becoming relatively complex, but also the physical means of application of scientific knowledge play a very important part.

It is not proposed to deal in great detail with any particular branch of control, but rather to attempt to give an indication of present-day systems as applied to Great Britain. For this purpose it will probably be well to deal with the subject under three different headings.

(1) The control of raw material supplies.
(2) The control of composition in the melting unit.
(3) The control after leaving the furnace.

THE CONTROL OF RAW MATERIAL SUPPLIES

In a fully integrated works where fundamental raw material supplies are more or less constant in origin it is a relatively easy matter to keep these under control as compared with, for instance, cold metal works where large quantities of scrap from outside sources have to be used.

The control of scrap, particularly in these days of shortage where much has to be accepted which in normal times would not be brought into works, is in many respects more dependent on art and experience than on scientific methods of control. The latter of course must be subsequently applied, but first of all arises the question of satisfactory means of obtaining a reasonably representative sample. Whilst the sampling should in all cases be under the control of the chief chemist, he cannot personally select samples, and the most reliable means of obtaining these is to concentrate on the training of a staff, not necessarily scientifically trained, upon whom can be placed reliance for the physical examination of scrap and the collection of representative samples.

Where scrap can be obtained from consistent sources of supply, both as regards origin and intermediaries, the question is, in course of time, relatively easy of control, but even given the most desirable of conditions in this respect troubles will from time to time arise. For instance, despite the value of non-ferrous materials included in dismantled engines, the scrap suppliers frequently find it more easy to send steel scrap containing bearings, etc., rather than separate these prior to sending forward to the steel works. With this in mind it is an advantage to offer an incentive both to the scrap examiners and the loaders by means of a monetary return for any non-ferrous material which they find amongst the scrap as it comes into the works, or in loading up from the scrap pile for charging.

Quite apart from inclusions of non-ferrous material, there are many other aspects which in course of time the sampler will learn to look for in material prior to charging up. To mention only two of these, serious trouble has arisen at times from such sources as deposits on tubes which have been in use for boilers, and the high sulfur content of parts which have been exposed at relatively high temperature to sulfur-bearing gases.

The scrap having been received into the works, troubles are not by any means at an end, difficulties in storage frequently causing trouble in subsequent use.

For instance, too long storage of parts can result, under industrial conditions, in extra pick-up of sulfur by the oxidized surfaces of scrap material, and in the case of very light material such as turnings, combustion of the heap as a whole is not by any means an unknown factor.

Due to the incoming material being delivered as "scrap" there is, unless guarded against, a tendency to look on this as material of little value. This can lead to some carelessness in stacking and subsequent loading up, and the mixing of raw materials in this way giving trouble due to "incidental elements" in the steel produced.

From these points of view careful control is one of the very essential factors in the use of scrap materials and, well applied, not only guards against unsatisfactory usage in the ways shown briefly above, but also frequently leads to economies by being able to upgrade scrap particularly from the point of view of content of alloying elements which, whilst so extremely detrimental if not properly controlled, give a very high financial reward if properly utilized from the base material.

The examination of the sampled material is usually carried out in general by means of chemical analysis along well known and acknowledged lines. There are occasions, however, when physical methods can be applied with advantage, these varying from the old system of "spark testing" to the growing use, either at this stage or in the early stages within the furnace, of spectroscopic means of examination. The use of the "spark test" both at this stage and in subsequent operations is recommended as being well worthy of serious attention. Not only is it possible for an experienced man examining straight carbon steels to judge the carbon content within 0.05%, but where alloying elements are concerned, either within the composition of normal steels for hardening and tempering, or even as applied to stainless and heat-resisting steels, remarkably accurate estimations can be made.

The use of the spectroscope may be looked upon as a refined means of applying the "spark test", the only difference being that in the latter, experienced observation of spark formation on grinding is applied, whilst with the spectroscope an arc is struck with the material to be examined, the spectrum examined for lines corresponding to the various elements, and deductions based on more scientifically prepared and acknowledged experience are made.

Emphasis has been laid on the question of scrap examination in this portion on raw materials. This has been done deliberately, (*a*) because of the added complexity of this compared with any of the other raw materials, and (*b*) in an endeavour to get a modified conception of what the word "scrap" should convey. The essential point here is that what is scrap at a previous stage, becomes, when transferred to the melting department of a steel works, one of its most valuable raw materials, and should be looked on at every stage as this, even if the terminology cannot at any time be modified. All other raw materials must of course be equally meticulously examined, but methods of doing this and the simplified conditions of sampling make it probably unnecessary to dwell any further on this point.

THE CONTROL OF COMPOSITION IN THE MELTING UNIT

Practice varies very considerably according to the type of melting unit concerned and also the ultimate purpose in mind for the product. For instance, in furnaces run in the manner in which induction furnaces are mainly used, namely, as purely melting units, the problem is very simple. Conditions are very different in furnaces where any appreciable refining is to be put under control. In all such cases the emphasis is moved from the question of sampling to one of determining the most rapid, as well as accurate, methods of estimating the quantity of the various elements present. This has for the past few years brought seriously into question the use of the spectroscope with either a photographic or a direct recording finishing operation. The direct recording spectrometer is the one which naturally makes the greatest appeal, in that once a satisfactory sample has been obtained from the furnace little else remains to be done, except pass this sample into a slot and watch certain dials for an indication of the quantity present of the various elements for which the set up has been arranged.

It might be well at this stage to give some consideration to the points for and against the use of spectrographic methods. There is little doubt that for speed of working, for lack of necessity for analytical skill, and for economy of staff, everything is in favour of the spectrometer. Against this the cost of installation is high, the capacity of the instrument is limited to metallic elements only, and the original check for the instrument is still in general dependent on chemical estimations. Mention has been made of the high cost of installation, and this definitely applies even when only one instrument is to be installed, but it would in most cases be necessary to have at least two instruments, as apart from any question of a breakdown having disastrous effects in the case of one unit, there are regular occasions in steel works' practice where it is worth while to have the added safeguard of carrying out duplicate tests by different men on separate apparatus. In chemical analysis as generally practised, this is often done to the extent of carrying out initial tests on casts in the melting shop laboratories, but in any case of doubt having this checked in a separate general laboratory. Whilst the advantage of this availability of checking procedure is greatest after the steel has left the furnace, there are numerous points where advantage can be taken of a separate staff and laboratory as an adjunct to the melting shop group.

The conclusion from all this is that for large plants, and particularly if concerned with alloy steel as well as plain carbon steels, a spectrometer is a well worthwhile investment. For smaller concerns a start should be made with the less elaborate spectograph, or still simpler types of spectroscope. The latter have been found to be worthwhile instruments even in non-alloy steel producing works as a means of rapidly checking the raw material coming in for undesired incidental elements.

It has been mentioned that spectrographic examination can be made only for metallic elements, and this unfortunately cuts out the biggest number of estimations in steel works in general during the refining operation. In acid lined furnaces the element which has to receive the greatest attention, and the one which is being essentially refined, is carbon, whilst in basic lined furnaces repeated estimations have to be made in addition for the elements sulfur and phosphorus. The determination of carbon in all melting shop practices can be, and usually is, carried out automatically by means of the carbometer. This simple apparatus first of all depends on obtaining a satisfactory piece for electromagnetic testing in the carbometer itself. This requires considerable practice, but the art can be developed to make the carbometer a satisfactory means of determining carbon content with the desirability, at least on the special steels, of a final check by the more routine chemical method of determination. This latter varies throughout steel plants of the country, but the most generally accepted practice is the burning off of drillings in a stream of oxygen and the subsequent determination of carbon content from the CO_2 produced, either by absorption and weighing, or by the collection of gases and treatment along the ordinary lines for gas analysis. The latter gives a more rapid determination, but, in the opinion of many, is not so strictly accurate as the former method.

Sulfur, which has been determined for many years by solution methods, now tends to be more regularly, and possibly more accurately, determined by combustion in oxygen at high temperature, in the neighbourhood of 1,300°C., and absorption and measurement of the gases evolved by one of the accepted means.

Phosphorus must also still be determined chemically, and whilst for preliminary estimations determination of turbidity of solutions with precipitated phosphorus-containing salts is a satisfactory method, for strict accuracy titration or gravimetric methods must still be applied.

Where alloy steel is involved it is recommended that stable elements such as nickel, molybdenum, tungsten, etc., should be estimated at leisure by samples taken in the early stages after melting if chemical methods are to be applied, whilst for oxidizable elements such as chromium and manganese additional later estimations will have to be carried out after, or towards the end of, the oxidizing process.

It is assumed in these comments that the accepted method of sampling will be by pouring a spoon sample into a mould of suitable design, with the addition of aluminium where fully killed samples are not naturally obtained, in order to avoid both blow hole formation and segregation.

The composition of the bath and its variations will of course essentially be affected by the slag-metal reactions which are dependent essentially on slag composition and temperature. So far as the latter is concerned, general practice in Great Britain is to determine temperatures of the bath by means of immersion pyrometers consisting of platinum-platinum/rhodium thermocouples. There are various modifications of build up to enable these couples to be satisfactorily applied, but details of all these can readily be obtained in recent literature on this subject.

Slag control is still to a large degree dependent on the visual observation of the slag in the furnace together with the examination of samples rapidly cooled after taking from the furnace. Possibly the only general means applied, other than these, is that of iron content in basic lined furnaces, which can readily be determined with speed. Work is being carried out on physical means of rapidly determining slag compositions, but none of these has up to the present reached a stage of general application.

The effect of the hearth as distinct from the slag cannot be ignored in basic lined furnaces, in that, particularly under the highly reducing slags sometimes obtained in electric furnaces, appreciable sulfur absorption by the hearth, as distinct from the slag, takes place. If for any reason, however, a slag is subsequently removed and an oxidizing slag reintroduced, then the sulfur in the hearth is reabsorbed into the bath and sampling must commence all over again.

Other examples of this type could be included, but short of a volume on the subject it is probably inadvisable to go further than briefly to indicate in this manner the type of thing which must be looked for in the question of control of composition during melting and refining.

Similarly, serious consideration is not possible of other accepted items in procedure, such as the necessity of stopping reactions in the bath, usually by silicon or manganese additions, or both, after the final sample has been taken on which the finishing alloys are to be added to make for reasonable composition and conditions of metal.

THE CONTROL AFTER LEAVING THE FURNACE

Precautions are again necessary almost immediately after the liquid steel has left the furnace and as these are most necessary with basic lined furnaces, it is proposed to concentrate at least on ladle practice in this connexion.

With certain exceptions, such as tapping only a portion of the charge from a tilting furnace, there is always a certain amount of basic slag coming from the furnace with the last metal to leave, and some covering of this type is in addition desirable in order to retain all the heat in the metal during the time it is in the ladle. Ladles up to now have not been satisfactorily produced with basic linings, although promising results in this direction are now being obtained. Work in this direction should be pressed forward as the advantages to be derived, as applied to basic steel practice, are very great. When the basic slag comes in contact with the normal lining of the ladle there is a tendency for reaction between the two. This reaction becomes greater towards the bottom of the ladle where this has been heated to higher and higher temperatures by prolonged contact with the liquid steel. In certain cases reaction becomes so great as to start action between metal and slag which apart from the action on the ladle would have ceased for all time. In some cases this reaction is sufficient to cause in the last metal coming from the ladle modifications in composition of the steel, particularly as regards reduction in carbon and manganese contents and increase in phosphorus content.

In order to ensure there should be no lack of uniformity of composition of steel due to this effect, it is general practice to take three samples during the teeming of a normal cast of basic open hearth steel. These should be at the beginning, the middle and the end of the teeming operation in order to check whether any change in composition has occurred. Very rarely has this been found to happen up to the middle of the cast, but if any trace is found of increase of phosphorus content beyond the specification requirements on the last sample taken, then each ingot from that, coming forward in the order of casting, must be drilled and examined for phosphorus content until normal limits obtain. It is of course impossible to predict just when the last full ingot will be made, so that usually there are from one to three ingots cast after the last pit sample has been taken. These are separated from the rest of the cast until all the pit samples have shown whether there is any indication of phosphorus return to the liquid steel, in which case the ingots beyond the sample showing this action are of no use.

In electric steel practice there is little chance of this taking place due to the fact that the phosphorus bearing slag has been taken off prior to the finishing slag being added and the latter contains little or nothing of an oxidizing nature, so that the carbon and manganese elements again cannot be affected in the same way.

Again, as in the case of certain of the samples taken at the melting stage, it is advisable to treat all pit samples with a small quantity of aluminium for the same reasons as stated previously. Whilst the metal in itself should not give any tendency to blowing, it is possible that the condition of the sample pot may give a tendency in this direction, and in any case the aluminium can do no harm and will tend to unify the whole of the metal during solidification. This applies particularly in the case of "rimming steels" where it is almost impossible to get a true sample from either ingot or billet subsequently. The aluminium addition to the test bit melt should be sufficient with such steels to give a fully solid steel when a true composition of the cast as a whole can be determined. It is probable that the real carbon content of the metal in the ingot will be of the order of 0.01% to 0.02% and the manganese up to 0.05% lower than the test bit due to the action in the ingot mould, but this is not always the case and it is much better to rely on the test bit analysis due to the difficulties in sampling on the subsequent material.

It is possible for disputes to arise as to composition of steel in its ultimate form due to a variety of surmountable difficulties in the sampling operation. For instance, quite wrong compositions have been found due to the inclusion of bottom plates or turnings used in the bottom of the mould, to hanger loops at the top of the ingot, or to ladle drip or other pieces falling in the ingot moulds due to carelessness in pit conditions. In addition segregations have been analysed which may occur due to bad design of ingot, unsatisfactory mill discards or similar causes, but these are so well known, and presumably in efficient works satisfactorily guarded against, that little more need be said regarding them.

In conclusion it can be stated that if suitable precautions along the lines mentioned, and others which can be regarded as accepted practice, are applied, there is likely to be little variation shown in composition as between the test bit samples taken during teeming, and the same material examined in billet form, or in subsequent stages of production.

Specifications of the Products Made by the Compañía de Acero del Pacífico, S.A., Huachipato, Chile

OSCAR HERRERA SILVA

The tables included in this paper show the chemical composition, physical properties and usual dimensions of the various products manufactured by the Compañía de Acero del Pacífico. Where possible indications are given of the use for which the product is intended, and whether the product is made by the Siemens-Martin process (basic open hearth), or the Bessemer converter process (acid).[1]

Table 1 shows the classification of the pig iron from the blast furnace, according to its chemical composition.

A list is given below of the products manufactured according to one or more conditions laid down by the customers:

2.[2] Wire rod for bolts. Requirements as to roundness only.[3]

4. Wire rod for welding. Requirements as to chemical analysis only.

5 and 6. Concrete bars. In Chile, all concrete bars must, by law, meet the following requirements:

Yield point: 48% of the rupture fatigue.
Tensile strength: minimum 37 kg./mm^2 (R).
Elongation: 18% minimum for 10 mm. Ø bars or over; for smaller rod, elongation must be larger than 12+0.6 D(A).

Table 1

COMPOSITION RANGES OF THE BLAST FURNACE IRON MADE BY COMPAÑÍA DE ACERO DEL PACÍFICO

Pig iron company classification	*Silicon range %*	*Sulfur range %*
155	1.25–1.75	Less than 0.05
308	2.75–3.25	More than 0.07
307	2.75–3.25	0.05–0.07
305	2.75–3.25	Less than 0.05
258	2.25–2.75	More than 0.07
257	2.25–2.75	0.05–0.07
255	2.25–2.75	Less than 0.05
208	1.75–2.25	More than 0.07
207	1.75–2.25	0.05–0.07
205	1.75–2.25	Less than 0.05
158	1.25–1.75	More than 0.07
157	1.25–1.75	0.05–0.07
108	0.75–1.25	More than 0.07
107	0.75–1.25	0.05–0.07
105	0.75–1.25	Less than 0.05

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[1] *Author's note:* At the beginning of 1953, CAP discontinued the production of Bessemer steel.

[2] The numbers correspond to the relevant item number in table 2 (pages 410–413, inclusive).

[3] *Author's note:* Other specifications were issued in 1954.

Table 2

CHARACTERISTICS OF THE PRODUCTS MADE BY THE COMPAÑÍA DE ACERO DEL PACIFÍCO S.A.

			Chemical composition						Physical properties				
Name	Use	Usual dimensions	C %	Mn %	P %	S %	Si %	Quality of steel	Yield point $kg./mm.^2$	Tensile strength $kg./mm.^2$	Elongation %	Hardness[a]	Cup test 1/100 mm
Wire rod													
1.	For nails and wire		0.07–0.11	0.35–0.50	0.040 max.	0.050 max.	0.020 max.	Siemens-Martin	20–28	33–40	24–32	44–52	
2.	For bolts	Ø 5.5 mm.	0.07–0.11	0.35–0.50	0.040 max.	0.050 max.	0.020 max.	Siemens-Martin	20–28	33–40	24–32	44–52	
3.	For steel wool	6, 7, 8 and 10 mm.	0.13–0.18	0.70–1.00	0.040 max.	0.040 max.	0.020 max.	Siemens-Martin	33–39	49–55	18–24	73–79	
4.	For welding electrodes	6, 7, 8 and 10 mm.	0.10–0.15	0.30–0.60	0.040 max.	0.035 max.	0.030 max.	Siemens-Martin	24–30	36–42	24–30	53–59	
5. *Coiled concrete bars*		Ø 5.5 mm. 6, 7, 8 and 10 mm.	0.13–0.18	0.30–0.50	0.040 max.	0.050 max.	0.04–0.07	Siemens	27–33	41–47	21–27	64–70	
6. *Round concrete bars*		Ø 10, 12, 16, 18, 20, 22, 26, 28 mm.	0.13–0.18	0.30–0.50	0.040 max.	0.050 max.	0.04–0.07	Siemens-Martin	24–32	38–44	26–34	64–70	
			0.05–0.09	0.20–0.70	0.099 max.	0.060 max.	0.04–0.07	Bessemer	34–41	45–53	20–27	74–83	
			0.05–0.07	0.20–0.50	0.10–0.119	0.040 max.	0.04–0.07	Bessemer	35–44	46–54	19–28	76–85	
			0.08–0.11	0.20–0.70	0.099 max.	0.055 max.	0.04–0.07	Bessemer	39–46	49–56	18–25	79–86	
			0.12–0.14	0.20–0.50	0.099 max.	0.055 max.	0.04–0.07	Bessemer	39–46	49–56	18–25	79–86	
			0.08–0.12	0.20–0.50	0.10–0.119	0.040 max.	0.04–0.07	Bessemer	37–45	48–56	18–28	78–85	
		Ø 32, 35, 38, 44, and 50 mm.	0.18–0.25	0.30–0.50	0.040 max.	0.050 max.	0.04–0.07	Siemens-Martin	26–32	39–45	27–33	64–70	
			0.12–0.14	0.51–0.70	0.099 max.	0.055 max.	0.04–0.07	Bessemer	39–45	49–55	19–25	79–85	
			0.15–0.17	0.20–0.50	0.099 max.	0.055 max.	0.04–0.07	Bessemer	41–47	52–58	19–25	82–88	
7. *Round bars*	For bolts	Ø ⅜ in., ½ in., ⅝ in., ¾ in., ⅞ in., 10 mm. 12, 16, 18, 20, 22, 26, 28 mm.	0.18–0.23	0.30–0.50	0.040 max.	0.050 max.	0.04–0.07	Siemens-Martin	25–33	39–47	27–32	63–72	
		Ø 32, 35, 38, 44, 50 mm.	0.23–0.28	0.30–0.50	0.040 max.	0.050 max.	0.04–0.07	Siemens-Martin	33–39	45–51	21–27	74–80	
8.	For rivets	Ø ⅜ in., ½ in., ⅝ in., ¾ in., ⅞ in., 10 mm. 12, 16, 18, 20, 22, 26, 28 mm.	0.13–0.18	0.30–0.50	0.040 max.	0.050 max.	0.04–0.07	Siemens-Martin	25–30	37–45	28–35	59–69	
		Ø 32, 35, 38, 44, 50 mm.	0.18–0.23	0.30–0.50	0.040 max.	0.050 max.	0.04–0.07	Siemens-Martin	27–33	41–47	26–32	66–72	
9.	For structural use	Ø 10 mm. to Ø 28 mm.	0.18–0.23	0.30–0.50	0.040 max.	0.050 max.	0.04–0.07	Siemens-Martin	25–33	39–47	27–32	63–72	
		Ø 32 mm. to Ø 50 mm.	0.23–0.28	0.30–0.50	0.040 max.	0.050 max.	0.04–0.07	Siemens-Martin	33–39	45–51	21–27	74–80	

[a] Rockwell B, unless otherwise specified.

Table 2 (continued)

Name	Use	Usual dimensions	Chemical composition					Quality of steel	Physical properties				
			C %	Mn %	P %	S %	Si %		Yield point kg./mm.^{2}	Tensile strength kg./mm.^{2}	Elongation %	Hardness[a]	Cup test 1/100 mm.
10.	For chain making........	Ø 5.5 mm. to Ø 24 mm.	0.18–0.23	0.30–0.50	0.040 max.	0.050 max.	0.04–0.07	Siemens-Martin	25–35	39–50	20–34	60–75	
11.	For grinding............	Ø 50 mm. and less	0.90–1.05	0.30–0.50	0.040 max.	0.050 max.	0.15–0.25	Siemens-Martin	50–60	80–100	5–8	25–32RC	
12.	For sale without special specifications..........	Ø 10 mm. to Ø 50 mm.	0.05–0.30	0.30–0.50	0.040 max.	0.050 max.	0.04–0.10	Siemens-Martin	23–39	33–51	21–38	53–77	
			0.05–0.17	0.20–0.70	0.140 max.	0.060 max	0.04–0.07	Bessemer	31–52	44–63	15–31	65–92	
13. *Square bars*	For structural use.......	⧄ 10, 12, 14, 16, 18, 20, 25 mm.	0.18–0.23	0.30–0.50	0.040 max.	0.050 max.	0.04–0.07	Siemens-Martin	27–33	40–46	29–35	63–69	
			0.05–0.09	0.20–0.70	0.099 max.	0.060 max.	0.04–0.07	Bessemer	28–42	40–55	25–31	69–75	
			0.08–0.11	0.20–0.70	0.099 max.	0.055 max.	0.04–0.07	Bessemer	35–41	46–52	21–27	76–82	
			0.12–0.14	0.20–0.50	0.099 max.	0.055 max.	0.04–0.07	Bessemer	35–41	46–52	21–27	76–82	
		⧄ 30, 35, 40, 50 mm.	0.23–0.28	0.30–0.50	0.040 max.	0.050 max.	0.04–0.07	Siemens-Martin	27–33	40–46	29–35	63–69	
			0.08–0.11	0.20–0.70	0.099 max.	0.055 max.	0.04–0.07	Bessemer	33–48	45–56	18–28	75–85	
			0.12–0.14	0.20–0.50	0.099 max.	0.055 max.	0.04–0.07	Bessemer	33–48	45–56	18–28	75–85	
14.	For sale without special specifications..........	⧄ 10, 12, 14, 16, 18, 20, 25, 30, 35, 40, 50 mm.	0.08–0.30	0.30–0.60	0.040 max.	0.050 max.	0.04–0.10	Siemens-Martin	23–39	33–55	20–40	55–75	
			0.05–0.17	0.20–0.70	0.140 max.	0.060 max.	0.04–0.07	Bessemer	31–52	44–63	15–31	65–92	
15. *Rectangular sections*	For nuts................	Thickness: 3.2 to 20 mm. Widths: 20, 25, 32, 38, 50, 56, 63, 75, 90, 100, 112 mm.	0.13–0.23	0.30–0.50	0.040 max.	0.050 max.	0.04–0.07	Siemens-Martin	27–34	41–50	23–30	64–73	
16.	For structural use.......	Thickness: 3.2 to 20 mm. Widths: 20 to 112 mm.	0.13–0.23	0.30–0.50	0.040 max.	0.050 max.	0.04–0.07	Siemens-Martin	25–33	37–47	27–35	59–72	
			0.05–0.09	0.20–0.70	0.099 max.	0.060 max.	0.04–0.07	Bessemer	29–35	40–46	25–31	69–75	
			0.08–0.11	0.20–0.70	0.099 max.	0.055 max.	0.04–0.07	Bessemer	35–41	46–52	21–27	76–82	
			0.12–0.14	0.20–0.50	0.099 max.	0.055 max.	0.04–0.07	Bessemer	35–41	46–52	21–27	76–82	
17.	For sale without special specification..........	Thickness: 13.2 to 20 mm. Widths: 20 to 112 mm.	0.08–0.30	0.30–0.50	0.040 max.	0.050 max.	0.04–0.10	Siemens-Martin					
			0.05–0.17	0.20–0.70	0.140 max.	0.060 max.	0.04–0.07	Bessemer					
18. *Light sections or skelp*	For pipes and electrical wire protection........	Thickness: 2 to 6 mm. Widths: 40 to 190 mm.	0.07–0.11	0.35–0.50	0.040 max.	0.050 max.	0.020 max.	Siemens-Martin	26–32	33–39	30–36	57–63	
19.	For pipes of heating slabs.	Idem.	0.05–0.06	0.30–0.40	0.025 max.	0.040 max.	0.010 max.	Siemens-Martin	25–30	30–37	30–36	45–60	
20.	For cold rolling..........	Idem.	0.05–0.18	0.30–0.60	0.040 max.	0.050 max.	0.020 max.	Siemens-Martin	24–31	32–39	25–32	56–66	

[a] Rockwell B, unless otherwise specified.

Table 2 (continued)

			Chemical composition						Physical properties				
Name	Use	Usual dimensions	C %	Mn %	P %	S %	Si %	Quality of steel	Yield point kg./mm.²	Tensile strength kg./mm.²	Elongation %	Hardness[a]	Cup test 1/100 mm
21. *Angles*	For structural use.......	20 x 20 x 3 to 80 x 80 x 12 mm.	0.13–0.18	0.30–0.50	0.040 max.	0.050 max.	0.04–0.07	Siemens-Martin	28–36	38–46	23–30	65–71	
			0.18–0.23	0.30–0.50	0.040 max.	0.050 max.	0.04–0.07	Siemens-Martin	32–38	42–48	23–30	70–76	
			0.05–0.09	0.20–0.70	0.099 max.	0.060 max.	0.04–0.07	Bessemer	32–39	42–50	22–28	70–78	
			0.08–0.11	0.20–0.70	0.099 max.	0.055 max.	0.04–0.07	Bessemer	39–45	48–55	18–25	84–90	
			0.12–0.14	0.20–0.50	0.099 max.	0.055 max.	0.04–0.07	Bessemer	39–45	48–55	18–25	84–90	
22.	For sale without special specification..........	Idem.	0.08–0.30	0.30–0.60	0.040 max.	0.050 max.	0.04–0.10	Siemens-Martin	25–41	35–52	20–33	62–80	
			0.05–0.17	0.20–0.70	0.140 max.	0.060 max.	0.04–0.07	Bessemer	39–48	49–59	16–25	84–92	
23. *Window frame sections*........................		4 types	0.08–0.18	0.30–0.50	0.040 max.	0.050 max.	0.04–0.07	Siemens-Martin	34–40	44–50	22–30	73–79	
24. *Hot rolled sheets and plates*	For shovels............	1.6–4.8 mm. thick 300–1,200 mm. wide Up to 3,500 mm. long	0.45–0.55	0.40–0.60	0.040 max.	0.050 max.	0.10–0.15	Siemens-Martin	46–55	61–70	16–21	88–96	—
25.	For abrasion resistant uses	1.6–4.8 mm. thick 300–1,200 mm. wide Up to 3,500 mm. long	0.13–0.23	0.30–0.50	0.040 max.	0.050 max.	0.04–0.07	Siemens-Martin	22–34	37–50	17–27	55–75	—
26.	For mechanical forming..	0.4–4.8 mm. thick 300–1,200 mm. wide 3,500 mm. long	0.05–0.09	0.30–0.40	0.025–0.08	0.040 max.	0.010 max.	Siemens-Martin	—	—	—	—	—
27. *Hot rolled sheets and plates, cold finished*	For ordinary and special nails.................	0.4–4.8 mm. thick 300–1,200 mm. wide Up to 3,500 mm. long	0.08–0.18	0.35–0.50	0.025–0.06	0.050 max.	0.020 max.	Siemens-Martin	34–43	46–55	12–21	72–78	—
28. *Annealed sheets and plates*	For deep drawing........	0.4–1.6 mm. thick 300–1,200 mm. wide Up to 3,500 mm. long	0.05–0.09	0.30–0.40	0.040–0.08	0.040 max.	0.010 max.	Siemens-Martin	—	—	—	15–35 R30T	800–880
29.	Without special specifications.................	0.4–4.8 mm. thick 300–1,200 mm. wide Up to 3,500 mm. long	0.05–0.09	0.30–0.40	0.025–0.08	0.040 max.	0.010 max.	Siemens-Martin	—	—	—	—	—
30. *Annealed sheets and plates, cold finished*	Without special specifications.................	Same as No. 29	0.05–0.09	0.30–0.40	0.025–0.08	0.040 max.	0.010 max.	Siemens-Martin	—	—	—	—	—
31. *Pickled sheets and plates*	For deep drawing........	0.4–1.6 mm. thick 300–1,200 mm. wide Up to 3,500 mm. long	0.05–0.09	0.30–0.40	0.040–0.08	0.040 max.	0.010 max.	Siemens-Martin	—	—	—	35–41 R30T	775–825
32.	Without special specifications.................	0.4–4.8 mm. thick 300–1,200 mm. wide Up to 3,500 mm. long	0.05–0.09	0.30–0.40	0.025–0.08	0.040 max.	0.010 max.	Siemens-Martin	—	—	—	—	—

[a] Rockwell B, unless otherwise specified.

Table 2 (continued)

			Chemical composition						*Physical properties*				
Name	*Use*	*Usual dimensions*	*C %*	*Mn %*	*P %*	*S %*	*Si %*	*Quality of steel*	*Yield point kg./mm.^{2}*	*Tensile strength kg./mm.^{2}*	*Elongation %*	*Hardness*[a]	*Cup test 1/100 mm*
33. *Pickled sheets and plates, cold finished*		0.4–1.59 mm. thick	0.05–0.09	0.30–0.40	0.025–0.08	0.040 max.	0.010 max.	Siemens-Martin	—	—	—	27–34 R30T	825–875
		1.6–4.8 mm. thick	0.05–0.09	0.30–0.40	0.025–0.08	0.040 max.	0.010 max.	Siemens-Martin	21–27	26–40	23–49 (2 in.)	32–42	—
34. *Galvanized sheets*	Flat..................	0.4–1.6 mm. thick 300–1,200 mm. wide Up to 3,500 mm. long	0.05–0.09	0.30–0.40	0.025–0.08	0.040 max.	0.010 max.	Siemens-Martin	—	—	—	27–34 R30T	825–875
35.	Corrugated............	Same as above	0.05–0.09	0.30–0.40	0.025–0.08	0.040 max.	0.010 max.	Siemens-Martin	—	—	—	—	—
36. *Tinplate*	D-1....................	(*a*) 0.30–0.76 thick	0.05–0.09	0.35–0.50	0.090 max.	0.040 max.	0.010 max.	Siemens-Martin	—	—	—	47–53 R30T	620–670
	D-3....................	(*b*) 0.30–0.76 thick	0.05–0.09	0.35–0.50	0.090 max.	0.040 max.	0.010 max.	Siemens-Martin	—	—	—	50–56 R30T	550–600
		(*c*) Less than 0.30 mm. thick	0.05–0.09	0.35–0.50	0.080–0.11	0.040 max.	0.010 max.	Siemens-Martin	—	—	—	—	—
37.	D-5....................	0.28–0.30 mm. thick	0.08–0.11	0.35–0.50	0.080–0.11	0.040 max.	0.010 max.	Siemens-Martin	—	—	—	57–63 R30T	545–595
38. *Heavy plate*	Commercial quality......	Over 5.0 mm. thick Up to 1,500 mm. wide Up to 3,500 mm. long	0.05–0.14	0.20–0.70	0.140 max.	0.060 max.	0.04–0.07	Bessemer	36–43	47–53	18–24	78–84	—
39.	Structural quality.......	Over 5.0 mm. thick Up to 1,500 mm. wide Up to 3,500 mm. long	0.13–0.23	0.30–0.50	0.040 max.	0.050 max.	0.04–0.07	Siemens-Martin	22–34	37–47	23–30	60–70	—
40.	Boilermaking quality....	Same as above (No. 39)	0.13–0.23	0.30–0.50	0.035 max.	0.040 max.	0.04–0.07	Siemens-Martin	23–28	37–43	23–33	62–68	—
41.	For tubular structures...	Same as No. 39	0.18–0.23	0.30–0.50	0.040 max.	0.040 max.	0.04–0.07	Siemens-Martin	26–32	42–48	45–51 (2 in.)	64–72	—
42.	For welded pipe.........	Same as No. 39	0.13–0.18	0.30–0.50	0.040 max.	0.050 max.	0.04–0.07	Siemens-Martin	24–33	36–47	39–48 (2 in.)	57–70	—

[a] Rockwell B, unless otherwise specified.

Quality coefficient (RxA): 900 minimum for 10 mm. bars or over, and for smaller bars it should be 300+60D.

7. Round bars for bolts; requirements as to roundness only[3].
8. Round rod for rivets: minimum tensile strength, 34 kg./mm^2.
9. Round bars for structural use: there are various types, with different limits for the minimum tensile strength, such as 34, 37, 42, 47 kg./mm^2.
10. Round bar for chain making. Welding steel to be used.
11. Mill rods; abrasion-resistant steel to be used.
13. Square bars for structural use: same requisites as for 9.
16. Flat bar for structural use: same requisites as for 9.
18. Light sections or skelp, for pipes: maximum hardness, 70 Rockwell B.
19. Light sections or skelp, for pipes of radiant heating slabs: only requirements as to chemical analysis.
21. Structural shapes and sections: the same as for 9.

The following products from the sheet mill are manufactured as indicated:

24. Sheet for shovels: hardness between 88 and 96 Rockwell B.
27. Sheet for the manufacture of special nails: various hardnesses depending on the thickness of the sheets.
28. Sheets for deep drawing: they must have the proper ductility.

34 and 35. Zinc coated plates, flat and corrugated: the company delivers the sheets with a coating of 1.5 oz. of zinc per sq. ft.

36 and 37. Tinplate of varying ductility. Coating weight equal to 1.5 lb. of tin per base-box.

39. Heavy plates for structural use: the same as for 9.
40. Boiler plate:
    Grade A: Tensile strength: 38.7–45.7 kg./mm^2.
    Yield point: minimum 50% of the tensile strength.
    Minimum elongation: 27%.
    Grade B: Tensile strength (R): 33.8–40.8 kg./mm^2.
    Yield point: minimum 50% of R.
    Minimum elongation: 27%.
41. Heavy plates for tubular structures:
    Tensile strength: 42 kg./mm^2., minimum.
    Yield point: 23 kg./mm^2., minimum.
    Minimum elongation: 22%.
42. Heavy plates for welded pipes, manufactured according to chemical analysis and physical properties. They are determined by the company's metallurgical department.

Table 2 (pages 410–413, inclusive) shows that the variations in chemical composition and physical properties of the various products of the plant are relatively narrow.

The Compañía de Acero del Pacífico has adopted the policy of delivering the most uniform possible product to its customers, and the table shows that this has been satisfactorily attained by the company.[4]

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[4] *Author's note:* A number of specifications for other products, and some new specifications for products shown in this report, have been issued by the Compañía de Acero del Pacífico since this information was given in 1952.

Notes on Specifications of Steels for Different Uses

FEDERICO FRICK

1. Conditions peculiar to Latin America

In this study, it is proposed to discuss certain aspects of iron and steel specifications, which may contribute to the solution of similar problems in Latin America.

The limited Latin-American market, combined with the shortage of capital, renders imperative the adoption of every measure which may result in a reduction of production costs. A prime necessity is to restrict the variety of products to the minimum compatible with manufacturing requirements.

The high grade and wide variety of ores in the region, together with the shortage of scrap, and the impurity of the coking coal, lead to the conclusion that:

(*a*) Manufacturing processes should not call for a stricter specification than is absolutely necessary;

(*b*) Tolerances of chemical specifications should be as wide as possible.

It is proposed to examine specifications of carbon steels only, and it must first be stated that even in the highly industrialized countries there is no uniformity of opinion regarding the types and definitions of the various iron and steel products, even though this must be the foundation of any standardization.

One of the difficulties in the classification of steel lies in the fact that it is used in almost every manufacturing process, giving rise to a multiplicity of individual requirements, which, summarized, fall into four main groups: (*a*) mechanical properties, (*b*) chemical properties, (*c*) physical properties, (*d*) manufacturing properties.

The steel specifications in use in the industrialized countries form a solid background, based upon the experience of engineers, producers and consumers, for any study of the uses of different types of steel.

2. DEFINITIONS OF IRON AND STEEL PRODUCTS

For the purposes of this study, the following terms are defined:

1. *Steel.* An iron whose carbon content is higher than 0.05%, but lower than the saturation limit (about 1.8%). It may be forged without further treatment.

2. *Carbon steel.* A steel in which, normally, the additional elements (with the exception of iron, carbon and later copper) have not been deliberately added, which has a maximum content of: Mn −1.65%, Si −1.00%, P −0.12%, S −0.10%, P + S −0.20%, Cu −0.60%. The minimum Cu content if needed or specified, should not exceed 0.20%.

The remaining elements contained in the steel are considered as incidentals, but the content should not exceed the following: Ni −0.80%, Cr −0.25%, Mo −0.10%, V −0.05%, W −0.30%, Co −0.30%, Ti −0.30%, Al −0.30%. Other elements, (separately considered) −0.10%.

3. *Copper-bearing steel.* All carbon steel in which a minimum copper content is specified, but in no case should a sample from the ladle show more than 0.60%.

4. *Silicon steel.* All carbon steel in which a minimum silicon content is specified, but a sample from the ladle must not show more than 1.0%.

5. *Steel alloy.* A steel in which the content of one or more elements is greater than that shown in 2. above. This may include elements deliberately added to give required qualities to the steel.

6. *Cast steel.* Molten steel, obtained by any process, ready for forming into moulds.

In amplification of an earlier statement regarding the lack of uniformity of opinion the following examples may be quoted:

Spanish specification UNE 36001 of 1949 states that steel is an alloy of iron and carbon, in which the carbon content does not exceed the saturation limit on solidification.

Mexican specification B 19 of 1946 considers steel to be an alloy with between 0.06% and 2% of carbon, all of which should be soluble in the gamma iron at eutectic temperature.

Occasional definitions are met in the United States showing steel to be an iron with up to 1.9% carbon, and with no slag in its structure.

The Committee on German Specifications describes steel as all iron which may be forged without prior treatment, whilst the Metallurgical Association of the same country states that steel is an iron with less than 1.7% carbon, which can be forged, rolled or pressed either hot or cold.

The International Standards Association agrees more or less with this latter definition, but fixes the carbon limit at approximately 1.8%.

Carbon steel is described by both Spanish and Argentine specifications as an iron and carbon alloy which contains other elements as impurities, although the Argentine designation gives definite percentages.

3. CLASSIFICATION OF STEELS

Consumers' orders for carbon steel may indicate one or more of:

(*a*) The use, and any special conditions regarding the manufactured goods;
(*b*) Mechanical characteristics required;
(*c*) Chemical composition required.

If the user gives only (*a*), then the steel manufacturer must decide on the specification to be supplied, taking into account the following qualities of the steel:

(*a*) Cold and hot working properties;
(*b*) Weldability;
(*c*) Processing;
(*d*) Heat treatment;
(*e*) Corrosion resistance;
(*f*) Heat and chemical resistance;
(*g*) Metallographic.

Steels are not classified with regard to the raw materials used in their manufacture, since the same end can be achieved from a wide variety of raw materials. In some cases, however, the method of elaboration must be specified. In this connexion, a distinction must be made between acid or basic ratio blast furnace steels, Siemens-Martin acid or basic steel, arc or induction type electric furnace, duplex process, etc.

Conditions of a heat can be divided into four classifications:

(*a*) *Killed steel.* There is a discharge of gases from molten steel, notably hydrogen, nitrogen and oxygen in the form of carbon oxide. The hydrogen and nitrogen are given off as the temperature and the solubility fall. The carbon oxide does not dissolve, and causes considerable effervescence, which may be reduced by the addition of silicon or aluminium. These two elements combine with the oxygen and so prevent the formation of carbon oxide. In semi-killed or killed steel the greater proportion of gas discharged is hydrogen.

The process of "killing" steel is therefore deoxidation, using powerful oxidants which prevent the formation of carbon oxide, and gives a steel of very uniform composition, suitable for forging and a variety of heat treatments.

(*b*) *Semi-killed steel* is less uniform than killed steel.

(*c*) Between killed and semi-killed is a further type, produced either by pouring into covered moulds, thus controlling the time of gas development, or by adding a deoxidant on the surface of the molten metal in an open mould, thereby calming the surface and speeding up solidification.

(*d*) *Unkilled steel.* Shows a wide variation of chemical properties in each transverse section, and the heart of the ingot has a much higher carbon, phosphorus and sulfur content than the surface. The carbon content is usually below 0.25%, and manganese less than 0.60%, and the metal is suitable for cold working.

Considerable attention is being paid at present to the grain of killed steels, subclassifying the fine and coarse grains, a point of great importance in the field of tool steels.

the mass of water a pulsation movement. The raw coal feed goes successively from compartment to compartment; the heavier products are immediately and the lighter products are successively discharged by adequate gates. The last compartments produce middlings and, finally, washed coal is delivered with the water above the last sill. The amplitude of the pulsation can very easily be regulated in each compartment through the air distributing mechanism which makes it possible to effect a very prompt and accurate stratification. The form of the pulsation may also be regulated. Discharge of the heavy products is regulated by automatic control units which are, at present, fitted with hydraulic servo-motors.

The main characteristics of the compressed-air large-coal jig and of the servo-motor automatic control are as follows: dissymmetric pulsation—the superiority of which was pointed out in Mr. Cheradame's paper, as compared with the symmetric pulsation of the mechanically driven wash boxes; very easy regulation of the pulsation in each compartment, even when in operation (shape and intensity); all mechanical parts are placed externally and thus are very easily accessible; the servo-motor automatic control is, at the same time, very sensitive and very accurate.

(c) *Pulsation jigs for small coal*

The PIC Company has manufactured, for the last twenty-five years, wash-boxes with filtering bed for small coal, fitted with automatic control units.

The principle of these wash-boxes is well known. The classification takes place both through pulsation and flowing current. It is known that in rheowashers, classifying occurs very easily, but the difficulty comes with the discharge of heavy products, as one must discharge these without interfering with the classification. This discharge is most effective only when it involves a small quantity of products.

In the case of filtering bed wash-boxes for small coal, the discharge takes place on the whole surface of the wash-box, across the filtering bed, the quantity of products discharged at each point of this bed being quite small. It can be said that the filtering bed wash-box acts in respects of evacuation like a rheowasher having an infinite number of outlets, each of them evacuating only a very small quantity of products.

The modern PIC jigs are compressed-air wash-boxes, fitted with servo-motor automatic control units.

The main characteristics are, just as in the case of large coal: the advantage of dissymmetrical pulsation; the adjustment of the pulsation which can be effected on each compartment, very quickly and during operation (shape and intensity); same access to all mechanical parts, located externally.

The selection of the nature of the filtering bed, of its size and thickness, enables the classification, with the maximum of efficiency, of products of quite different natures.

(d) *Washing units for washing small coal in heavy medium*

The cyclonage in a heavy media permits washing even the finest products with great accuracy. PIC has developed a multi-tubular apparatus which uses the cyclonage between tubes of small diameter (enlargement of $\frac{V2}{r}$) grouped together. All the parts subject to wear are cheap and easily dismantled.

Two experimental workshops have just been put into use, and their industrial results will be known in a few months' time.

(e) *Flotation cell*

The PIC Company manufactures cells of a great capacity, specially suited for coal treatment. Several installations are in operation in Poland and France, particularly as far as coking coal is concerned, and the flotation froth obtained—having an ash content of less than 10%—is reincorporated in the coking mixture. The refuse, in such plants, has an ash content of about 70%.

INDUSTRIAL RESULTS

"*Imperfections*". Essential data are based on CERCHAR's definition. It must be stressed that these results correspond to industrial operation, under the supervision of qualified workmen and foremen of the plants. The sampling, heavy-medium "float-and-sink" tests and ash percentages, have been made or determined by the mining staff, or under its control.

(a) *Heavy-medium washing plants—for large coal*

The probable mean deviations range from 0.02 to 0.04 owing to size treated, medium density and load on the wash-box. As far as washing results in Drewboys are concerned, no abnormal curve terminals were noted, which is easily explained by the care taken with regard to the two following points: avoidance of entrapping the floats with sinks taking place when raw coal is plunged into the heavy medium; in the Drewboy, the raw coal is conveyed to the bath and washed coal need not be hoisted from the bottom of the bath—avoidance of floating aggregates of refuse. To obtain this, the products are turned upside down on the bath surface, in order to oblige refuse to sink.

(b) *Large- and small-coal wash-boxes*

"Imperfections" are summarized in the following table:

Size	*Mean value of I*	*Value of I in some instances*
+50 mm	0.09	0.08
50 to 30 mm	0.11	0.09
30 to 10 mm	0.13	0.11
10 to 0.5 mm	0.15	0.10

For units of the same type, the "imperfection" increases when the products treated become smaller. Even if tests are made with identical size ranges, different imperfection values will often be found from one installation to another.

The imperfection depends on the load of the jigs. PIC wash-boxes tolerate certain variations in the composition of raw coal treated. For a given output of products to be discharged (refuse and middlings), in a wash-box of a given width, the imperfection varies but little, but beyond this limit it increases.

In the various installations on which trials were effected, the loads on the units were sometimes larger than the optimum. Moreover, regularity of feed and the amount of treated products go in hand with a small imperfection.

Spanish specification UNE 36020 of 1949 groups steels according to their mechanical strength, elasticity, hardness, tenacity, corrosion resistance and heat resistance, and establishes the following groups:

(*a*) Fine carbon;
(*b*) High tensile;
(*c*) Carburizing;
(*d*) Nitriding;
(*e*) Furniture;
(*f*) Stainless;
(*g*) Heat-resisting;
(*h*) Special uses.

These groups are then divided according to their chemical composition, and subdivided into types depending upon their ultimate use.

French specifications classify carbon steels by mechanical properties, and consider the working properties, and to a certain extent the chemical also.

The English BS.971 of 1950 is based upon mechanical properties as indicated by test samples under a specific heat treatment. Sulfur and phosphorus content are fixed, but apart from this the producer is free to vary the chemical properties to meet the consumer's requirements.

The Tentative Argentine IRAM 600 P of 1949 analysed various methods of classification, and finally rejected all except chemical composition. The resulting specification, IRAM 600, finally covered three groups:

(*a*) Hypoeutectoide, with less than 0.80% carbon;

(*b*) Eutectoide, with between 0.80 and 0.85% carbon;

(*c*) Hypereutectoide, with a carbon content greater than 0.85% and a theoretical maximum of 1.7%.

The Belgian NBN 147 of 1946 also covers chemical composition, but subdivides one group into six types according to their ultimate uses. This classification is based upon phosphorus and sulfur content, and upon the grain.

4. SPECIFICATIONS BASED UPON CHEMICAL COMPOSITION

During the period of development of the iron and steel industry, each consumer produced a specification to meet his particular needs, and this resulted in an impossibly wide diversity of types. In an attempt to simplify matters for both producer and consumer, the American Iron and Steel Institute carried out elaborate studies, working upon the five main elements in carbon steel: carbon, manganese, silicon, phosphorus and sulfur.

Open hearth basic and acid Bessemer, for hot rolled bars, are given S max. of 0.04%, and P max. of 0.05%, with the exception of spec. B1010, which has S max. of 0.06% and P max. of 0.12%. C varies from 0.10 to 0.44%, Mn from 0.25 to 1.65% and Si from 0.10 to 0.30%, according to the carbon content.

Steels for forgings are limited to open hearth, because of the uniform chemical composition required, and to 200 sq. in. cross section, because of segregation produced in large ingots.

Open hearth basic sulfur steels are shown with C up to 0.55%, and a common P max. of 0.04%. S ranges from 0.08 to 0.33%, and the silicon from 0.10 to 0.30%. For basic Siemens-Martin, the silicon is limited between 0.10 max. and 0.30 max. in a series of specifications. In Bessemer acid steels P varies between 0.07 and 0.12%.

Studies carried out by the Canadian Society of Automotive Engineers produced specifications very much in line with those of the American Iron and Steel Institute, with the exception that the Canadian specifications show, on the whole, a lower manganese margin in the open hearth series, and a lower carbon content (0.05 to 0.10%).

Influence of chemical components on the properties of the steel

The chemical composition affects:

(a) *Segregation.* Segregation at any moment changes the specification of the steel. Sulfur has the greatest tendency, followed by carbon, phosphorus and manganese. Silicon and copper have little tendency to segregate.

(b) *Tensile strength* is increased by carbon, up to 0.85%, and phosphorus produces the same effect as carbon. Manganese has but small effect, and silicon even less. Sulfur has a very slight detrimental effect.

(c) Hardness is mainly achieved by varying the carbon and phosphorus content. Manganese, silicon and copper have little effect, and again sulfur is a slightly detrimental factor.

(d) *Ductility and malleability* are reduced by carbon, phosphorus and sulfur, almost disappearing when the carbon approaches its maximum of 1.7 to 1.8%. If the molten metal has not been completely deoxidated, then the presence of iron oxide will reduce the malleability.

(e) *Weldability* is reduced by carbon, manganese and sulfur, and by copper in lesser degree. Carbon is permissible up to 0.7% provided the silicon is less than 0.5%, but for construction steels this latter should not be more than 0.3%. Normal carbon content should be 0.4 to 0.5% and manganese up to 0.5%.

(f) *Toughness* is adversely affected by the presence of phosphorus and sulfur.

(g) *Working properties* are improved by increasing the phosphorus and sulfur. Copper should be low, especially for hot working.

(h) *Corrosion resistance* is raised by phosphorus and especially by copper.

(i) *Surface quality* is lowered by carbon in the case of unkilled steel, but the reverse is true for killed steel. Manganese improves the surface quality, especially in unkilled steels with low carbon content. An important consideration in killed low carbon steel is the amount of sulfur present and its relation to the manganese content.

(j) *Hardening properties* are improved by manganese.

In all the above considerations, account must be taken of the effect of the different elements upon each other, and therefore the subject is complex in the extreme, and makes even more manifest the necessity for reducing the number of different types of steel.

Limits and margins of chemical composition

Specifications generally impose upper and lower limits of the common elements—carbon, silicon, sulfur, manganese and phosphorus. Very low limits of phosphorus and sulfur would prohibit the use of certain raw materials, and restrict manufacturing processes, but many consumer processes do not require very fine limits. In view of this, various countries have proposed or adopted a small number of grades of purity; the ISA, for instance, has five only, whilst Belgium has six, and the Chambre syndicale des producteurs d'aciers fins et speciaux of Paris has adopted five. In any case the situation requires careful attention, in order to avoid unduly severe grades of purity, and each element must be considered separately in the light of production and manufacturing processes.

The American Iron and Steel Institute gives a carbon tolerance of 0.05 to 0.12 for heavy sections, and 0.05 to 0.15 for plates. For acid Bessemer the lowest value which might be specified for C max. is 0.08%, and 0.12% for basic open hearth. Mn tolerance for the range 0.50 to 1.65 is 0.20 to 0.35%; the lowest normal P for basic open hearth is given as 0.04, and as 0.06 to 0.11 for acid Bessemer. The lowest normal S value for open hearth is 0.05 and for acid Bessemer is 0.06. Si for basic open hearth is 0.08–0.30%, and the lowest normal Cu value is 0.10. Bessemer acid steel is not supplied with a specific silicon content.

The British specification BS.971 of 1950 makes the point that maximum tolerances in the various elements depend upon the total percentage content of that element, and gives lowest limits of tolerances for various cases.

Limits in chemical composition and manufacturing processes

The chemical reactions produced by the linings of blast furnaces necessitate the use of pig iron with phosphorus and sulfur content in direct proportion to the purity required in the steel.

In the Bessemer acid process, the elements which produce the most heat in the blast furnace are silicon and phosphorus, and since the phosphorus content should be less than 0.06%, it is often necessary to use up to 2 or 3% silicon. In cases where steel with a low silicon content is required, the blast furnace iron must be raised to a high temperature in the converter, thereby reducing the amount of silicon required in the Bessemer process.

Basic processes

In these processes dephosphorization and desulfurization are required. The pig iron must have a low silicon content (0.4 to 0.6%, although up to 1% is permissible at times) to avoid the slag being too acid, which would affect the basic requirement, attack the lining, and deter the emission of the gases.

Manganese content of the pig iron should be about 1%. In specifying limits for this element, the difficulties of elimination should be remembered.

Since dephosphorization can be achieved only by complete decarbonization, carbon must be added to produce any required content, although this does not necessarily apply in the case of basic open hearth operation.

Converter processes, using oxygen

The mixture of air and iron in the converter gives rise to the danger of high percentages of oxygen and nitrogen. The nitrogen content rises to 0.01–0.03%, whilst in an open hearth furnace it is only 0.001 to 0.01%. This can be avoided by blowing with oxygen, a process which allows the use of a wide variety of pig iron in the converter, to produce a large number of types of steel which otherwise could only be obtained in an open hearth furnace.

5. CLASSIFICATION OF STEELS, BASED ON MECHANICAL PROPERTIES

For steels which are classified solely by their mechanical properties, the requirements to be considered are bending, yield point, tensile strength, notch bar test, and elongation.

The bending test is normally applied for commercial steels, of no great strength, which must be suitable for pressing, shaping, etc.

To mention a few mechanical specifications, the Austrian M 3112, 1950, for construction steels specifies a tensile strength of 37–65 kg. per mm.^{2} and elongation from 25 to 20. The French NF A 36–203, 1945, for constructional plates, establishes maximum and minimum tensile strengths from 56–65 to 33–40 kg./mm.^{2} The Argentine IRAM 507, 1946, for general purpose steel plates, shows a tensile strength range from 37 to 62, and a minimum elongation of 20 to 19% respectively, for a 10 mm. plate. This specification also includes a bending test for all steels.

The Austrian M 3111 of 1950, for carbon steel for machine manufacture gives yield strengths of 34–70 kg./sq. mm., and also specifies tensile and elongation figures. The Austrian M 3121, 1951, covers boiler steel, whilst M 3181, and also the German DIN 1681, 1942, are both for cast steel.

6. CLASSIFICATION OF STEELS ACCORDING TO BOTH MECHANICAL AND CHEMICAL PROPERTIES

Although the chemical composition should not normally be specified when stating mechanical requirements, at times both are necessary.

The various mechanical properties mentioned in the preceding section define fairly closely the chemical composition, but the possibilities of variation in carbon steels are wide, when only the fundamental properties must be provided. On the other hand, when special consumer requirements must be met, the precise chemical composition must also be taken into account.

In many cases, the purity as regards phosphorus and sulfur is sufficient. In others, such as French specification NF 35–001 of 1946, a mechanical specification, weldability is guaranteed only when certain limits of carbon, silicon, phosphorus and sulfur are observed.

A number of specifications cover steel for bridge and general construction. The American ASTM A7–50T, for carbon steels with tensile strength 42–50 and rupture strain 22, specifies a maximum S content of 0.05 for open hearth basic and acid, Bessemer acid and electric furnaces, whilst P varies from 0.06 to 0.11%. ASTM A94–50T for silicon steel, tensile strength 56–76 kg./sq. mm., and 17% elongation, establishes that the steel must be open hearth, and requires a C max. of 0.40% and Si min. of 0.20%. Many other specifications, British, French, Spanish and German, for steels for these uses also follow the principle of defining both mechanical and chemical properties, and it is common practice when specifying steels for rolling stock.

7. CLASSIFICATION OF STEELS ACCORDING TO USES

Basically these are divided into three user groups:

(*a*) Rolling;
(*b*) Forging;
(*c*) Casting.

Since only carbon steels are being studied in this paper, they can be divided into the following general classifications:

A. *Construction steels*
   1. General use;
   2. Structural;
   3. Bridges;
   4. Rolling stock;
   5. Rails;
   6. Ships;
   7. Machinery;
   8. Boilers and pressure vessels;
   9. Bolts;
   10. Rivets;
   11. Chains.

B. *Spring steels*

C. *Tool steels*

D. *Heat treating steels*
   1. Carburizing steels;
   2. Special refined steels.

There is an obvious tendency to reduce the number of groups, and a rigorous comparison of specifications would surely lead to the formation of more general groups.

Relation between mechanical and chemical properties

Although the chemical composition alone will not determine the mechanical properties, in the case of carbon steels the tensile strength depends mainly upon the carbon content, and secondly upon the manganese content. Phosphorus also has a quite important effect, but the remaining elements have but little bearing on this mechanical property.

In fixing the chemical composition, it is easy to produce an entirely erroneous solution. For instance, by establishing the carbon content for a material required for welding, the resulting tensile strength may be totally inadequate.

Many specifications cover the mechanical requirement and add merely for information the approximate chemical composition. Others give the relation between hardness and tensile stress, as an added information.

9. SELECTION OF CLASSIFICATIONS OF STEELS

It has already been seen that the principle of classifying steels solely on the basis of chemical composition originated in the industrialized countries, and it has been pointed out that the manufacture to individual needs resulted in a very considerable number of types of steel. In a country such as the United States, the present chemical specifications as prepared by the American Iron and Steel Institute, and the American Society of Automotive Engineers, are well justified, because of the many and varied uses of steel in a highly industrialized nation. Even so, it is recognized in that country that it would be advisable to work towards a greater standardization of specifications, in order to achieve further improvement in the manufacturing techniques of each type of steel.

Under the conditions ruling in Latin America, however, it is considered that it would be preferable to work on a basis of mechanical properties, complemented only when strictly necessary by purity requirements as regards phosphorus, sulfur and occasionally other elements.

Adherence to a strict chemical composition demands a supply of uniform raw material, whereas a mechanical specification allows greater elasticity in this respect. Errors or differences in chemical composition between one heat and another can occur easily, and whilst this is no great matter in an industrialized country, where all production can be disposed of with ease, it is of considerable importance where the market is limited. A further point is that a mechanical specification avoids possible errors in ordering by inexperienced buyers. Finally, worldwide knowledge is greater in the field of mechanical specification than that so far achieved in the chemical composition.

10. STANDARDIZATION OF STEEL SPECIFICATIONS IN LATIN AMERICA

At the first Pan American Engineering Congress the opinion was generally expressed that Latin-American specifications should conform to world standards, and it was pointed out that the first step should be standardization within the continent. Failure to take action now, during the development period of the Latin-American iron and steel industry, would inevitably lead to the same multiplicity of local specifications which had earlier arisen in other continents.

The task should be undertaken by the standards institutions of the various countries, but it was considered that positive assistance was required from governments, producers and users. Basic standards should be established, clearly defining all iron and steel products in general, and particularly steels, and this could only be done by adopting a uniform, clear and precise terminology.

Product specifications must be standardized, and, in the light of the consumer importance, the study should commence with carbon steels.

Following the earlier argument of this study it is considered that a classification based on mechanical properties is the most advisable for Latin America, with complementary requirements of chemical composition when necessary. The mechanical specifications should be as wide as possible, and sulfur and phosphorus contents should be specified only to meet the proposed consumer requirements.

Great care must be taken to avoid an unnecessarily large range of types of steel, and the specifying of the steel-making processes must be limited to essential cases.

Product standards must be supplemented by inspection and test standards, and no specification should be laid down which cannot be verified by established test procedure. It is obviously important that test equipment and methods must be standardized.

Dimension standards are also necessary, but these, as in the case of other specifications, must be kept to a minimum.

Such standardization of mechanical, chemical and physical qualities within the framework of the consumer requirement of the limited Latin-American market would allow a greater production volume of a smaller number of products, with a consequent reduction of costs. A widening of the chemical specifications would ease the difficulties arising from the dissimilarity of the various ores, and from the shortage of good coking coal. Freedom from limitations as regards steel-making processes would allow full use to be made of the most convenient local methods, and would compensate to some extent for the scarcity of capital, and eventually for the shortage of scrap. The reduction in production costs would also tend to offset the effect of using high grade ore.

Under the conditions peculiar to countries of growing industrial development, it is considered that iron and steel specifications should also have an instructional role. It is essential that they should convey to the users a clear picture of the accuracy to which a specification may be met, and indicate to them the necessity for a serious study prior to ordering, covering all the conditions to which the steel will be submitted during manufacture and in use.

Summary of Discussion

Mr. LEUSCHNER introduced the subject of the meeting by stating that it was of paramount importance that steel specifications should be uniform throughout Latin America. Dissimilarities in terminology and measurements invariably led to misunderstandings which, in turn, involved unnecessary expenditure.

At this point, the CHAIRMAN *invited Mr.* Evaristo ARAIZA *to take the Chair.*

Some Effects of Minor Elements on the Characteristics of Plain-Carbon Steels, *presented by the author*

Mr. ALLARD opened the general discussion of the paper, saying that the paper under discussion added greatly to the interesting contributions already made at the meeting. The technical nature of the paper, however, would warrant lengthy and scientific discussion. He was in accord with the conclusions drawn by E. C. Wright, in view of his own experience with the beneficial effects of aluminium deoxidation in Europe.

There was one point that had not been mentioned in the paper, i.e., the influence of oxygen with a combination of phosphorus and nitrogen on the making of steel. It was, of course, perhaps irrelevant to speak of oxygen in connexion with effervescent steel. He wondered whether Mr. Boulger was of the opinion that the effects of oxygen and nitrogen were interrelated?

Mr. BOULGER thanked Mr. Allard for his kind comments and was gratified by his agreement with the conclusions put forward in the paper. In his opinion, both oxygen and nitrogen could have similar effects, but nitrogen was more important. The oxygen content was more nearly controlled by the amount of carbon, manganese and silicon present in the steel. In the United Kingdom, the trend was to consider those two elements separately, but the separation seemed difficult to conceive. Vanadium was quite effective in the prevention of ageing. He considered that with commercial steels, nitrogen was probably the prime cause of ageing, as oxygen and nitrogen were difficult to isolate in the very small dosages occurring in steel.

Mr. COHEUR shared Mr. Boulger's views on the relative importance and effects of phosphorus, sulfur and nitrogen. In Belgium, the effect of phosphorus and nitrogen on ductility had been the subject of detailed statistical studies, and it had been found that Stromeyer's formula, developed in 1907, gave the best results for checking the hardness, but for ductility, the influence of nitrogen was more important.

His experience with steels containing as little as 0.002% nitrogen had led him to conclusions differing somewhat from some of the statements made in the paper under review with regard to the effect of nitrogen as an ageing element and of a very low concentration of nitrogen in relation to rimmed steel. Chemical analyses of oxygen and nitrogen were difficult to carry out, however, and, as Mr. Allard had pointed out, it was not practical to discuss oxygen in connexion with effervescent steel.

Mr. BOULGER agreed with Mr. Coheur's view that it was difficult to distinguish between the effects of oxygen and those of nitrogen when both elements were present in small quantities. Analytical values for small amounts of carbon, oxygen and nitrogen should be accepted with some reserve, but it was only natural to accept values which agreed with one's own theories. He thought, however, that he reflected the majority opinion in the United States in his theory that nitrogen was the most potent ageing element.

Mr. COSTA LINO wished to know, first, the effect of inclusions on the fatigue life of high-carbon steel; secondly, if Epstein's high-phosphorus steel was so highly satisfactory, why did it not meet ASTM (American Society for Testing Materials) specifications?

Mr. BOULGER said, in reply to the first question, that popular opinion held that inclusions were undesirable in steels used at high strength levels. However, evidence to that effect was scanty and not very convincing. Some unpublished investigations carried out twenty years ago had indicated that steels with a low sulfur content and few inclusions produced better bearings. If that theory were correct, the results would indicate an indirect effect of inclusions on the surface finish. Smooth finishes ensured a longer fatigue life.

As to the second question, he stated that steels containing 0.10% phosphorus, sometimes called low-alloy high-yield-strength steels, were widely used in the United States. They were patented steels sold under trade names and with guaranteed properties rather than on the basis of chemical composition.

Mr. SANITER thanked the author for his very valuable and interesting paper. It was valuable because although attention was being focused on the importance of raw materials, the paper threw a different light on the subject by emphasizing that the finished product was also to be taken into account. For the manufacture of welded steel tubes, the Bessemer process seemed suitable. On the other hand, if a wide strip mill were to be installed, low-sulfur steel might be used. For the manufacture of a wide variety of products, however, processes which would allow the production of steel low in sulfur, phosphorus and nitrogen content should be used.

He was in entire agreement with Mr. Boulger's statement that despite their undesirable properties, nitrogen and phosphorus could be made to perform a useful function.

One of the disadvantages of nitrogen not mentioned in the paper under review and which was pertinent, was the problem created by attempting to add boron to steel. However, that difficulty could be overcome by the use of titanium or aluminium.

As to the effect of inclusions on fatigue resistance, Mr. Boulger and other participants might be interested in the findings of Frith, published in the *Iron and Steel Institute Journal*.[1]

Mr. BOULGER remarked that he was pleased to find a fellow heretic who believed that phosphorus, nitrogen and sulfur were not always harmful. He agreed that a high nitrogen and oxygen content precluded the hardening tendencies of boron if certain precautions were not taken. However, if aluminium were present in amounts exceeding those required for deoxidation, a stable nitride would be formed. In such steels, the hardening effects of boron could be obtained regularly and consistently.

Mr. MERCIER referred to a statement in the paper under the heading "Sulfur", in respect of the sulfur content of forged steel. In Europe, forged steel with so high a sulfur content was undesirable. He wondered whether Mr. Boulger was referring to segregated parts of ingots.

Mr. PALMÉ asked whether Mr. Boulger had taken into account the influence of inclusions on micro-structure.

Mr. BOULGER, in reply to Mr. Mercier's question, stated that the data referred to were developed in investigations in which high-sulfur steels were selected. Those investigations were conducted at the request of a special committee. However, although it was agreed that high sulfur contents were permissible, such steels were not widely used in the United States.

As to Mr. Palmé's comment on the effect of inclusions on micro-structure, he confessed a weakness for discussions on inclusions. Sulfide inclusions could be obtained. If inclusions were small and round, they were, by and large, innocuous. On the other hand, if they were elongated in shape, they could present serious problems. However, there were certain processes for the treatment of steel to reduce the incidence of elongated inclusions.

The CHAIRMAN asked Mr. Boulger to comment on the noxious effects of arsenic on steels, and on whether the effects of arsenic should be considered as cumulative to the action of the sulphur content.

Mr. BOULGER replied that as he was quite unfamiliar with the subject, he preferred not to answer the question. The general impression seemed to be, however, that arsenic was just as harmful to steel as it was to human beings.

Mr. SANITER stated that he could not contribute much information on the matter except to say that a number of specifications in the United Kingdom limited the arsenic content of steel and, although some study of the subject had been undertaken, he knew of no case where arsenic created any particular problems.

Mr. ALLARD stated that in Europe a limited, but very interesting, study had been undertaken which showed that arsenic was diffused in steel more readily than phosphorus.

Mr. MERCIER pointed out that in the Dwight Lloyd process, a certain percentage of arsenic was eliminated in the form of arsenic acid.

Mr. LEUSCHNER wondered why nitrogen and oxygen were not included in the great majority of steel specifications.

Mr. BOULGER thought Mr. Leuschner's point was well taken. The reason for excluding those elements from United States specifications was that those specifications called for either converter or acid Bessemer steel, in which the oxygen and nitrogen contents were fairly constant. A high oxygen content was desirable for better machinability.

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[1] United Kingdom publication.

Standardization in Relation to Control in Steel Production, The Control of Composition During Steelmaking, *presented together by Mr. Saniter in the absence of the authors*

Mr. PALMÉ opened the general discussion of the two papers under review by recalling that during the Second World War, certain Anglo-Saxon countries had developed methods of statistical dimensional control. He wondered whether those methods had been extended to include the quality control of the material itself in respect of specifications.

Mr. SANITER replied that he was under the impression that statistical control had not yet crept into specifications. However, there was a very interesting manual published in the United Kingdom on the application of statistical control within the steelworks proper, although that aspect did not appear in published specifications.

Mr. COHEUR wished to comment on the statements made in the Weston papers on spectrographic chemical analyses. It was stated that no spectrographic analyses were made of elements other than metallic ones. One Belgian plant would, in the very near future, employ that method for the determination of phosphorus in about 2½ minutes to control the end of the conversion. It was hoped that oxygen analyses would be conducted by spectro-chemical methods and a paper dealing with that subject had been delivered the previous year at Detroit by the Comité national de recherches métallurgiques. Carbon analyses were also planned by the same process when no nickel was present.

Mr. SANITER observed that the two Weston papers did not attempt to include processes which were still under investigation, but it was gratifying to hear that spectrographic methods had been developed and put into practice for the determination of phosphorus.

Mr. MERCIER remarked that spectro-chemical analyses were being developed more and more. The majority of analyses in the Le Creusot Steel Company were made by spectro-chemical methods to control the production of steel. Procedure for spectro analysis of carbon was currently being perfected.

Mr. ALLARD referred to the question of the contamination of steel by scrap. In Latin America, scrap was scarce and therefore the problem was automatically solved. In Europe, however, scrap was plentiful and the problem was a very real one.

Mr. LEUSCHNER called attention to one point in the Weston papers which had not been mentioned. That point was that the papers contained specifications on carbon content and the saving of alloying elements, a feature which was of special interest to Latin America.

Specifications of the Products Made by the Compañía de Acero del Pacífico, Huachipato, Chile, *presented by Mr. Canguilhem in the absence of the author*

Mr. CANGUILHEM, at the end of his presentation, suggested that information, similar to that contained in Mr. Herrera's paper, from other Latin-American countries would be very interesting for purposes of comparison.

Mr. ALLARD opened the general discussion of the paper, observing that he had not had time to examine the paper in detail. He had noted, however, in the specifications for ordinary steel that the sulfur content was between 0.050 and 0.060%, whereas the manganese content was quite low. According to an old method, steel should have a manganese content ten times that of sulfur.

Mr. CANGUILHEM was inclined to agree with Mr. Allard that a higher manganese content was desirable, and thought that there might be an error in that respect in the paper. Inasmuch as the paper was not his own contribution, however, he was not in a position to clarify the issue officially.

Mr. COSTA LINO observed that the Volta Redonda specifications were based on ASTM specifications and that, so far, no difficulties had been encountered.

Notes on Specifications of Steel for Different Uses, *presented by Mr. Leuschner in the absence of the author*

Mr. ALLARD opened the general discussion on the paper, and expressed the opinion that under certain conditions the customer had to provide the manufacturer with specifications based on the use to which the steel would be put. It was dangerous, in his view, to rely on chemical analyses only.

General MACEDO SOARES E SILVA commented on Mr. Frick's paper, with regard to the classification of steels. He was fully aware of the difficulties attendant upon such a classification. In Brazil, national terminology had had to be developed for the Volta Redonda plant as there was no Portuguese heritage to look to, in view of Portugal's lack of an iron and steel industry. A dictionary of about 6,000 words had been compiled by various technicians and technologists at Volta Redonda. With the assistance of the Technological Institute of São Paulo, a classification of terms by categories had been undertaken, with fairly good results.

He agreed with Mr. Allard's view that it was dangerous to rely exclusively on chemical analysis. Brazil had proved that point by bitter experience.

It was the hope of Brazil that problems would be solved gradually and it was his view that the Frick paper would be very significant in the standardization of the iron and steel industry in Latin America.

Mr. BOULGER observed that it would indeed be ideal if specifications could be written on a simple basis. Given the various factors involved, however, he was inclined to believe that specifications should include both chemical composition and mechanical properties.

Mr. LEUSCHNER thanked General Macedo Soares e Silva for his remarks, which brought the problems of Brazil into focus. He advocated the compilation of a dictionary similar to that published in Brazil for use in the Spanish-speaking Latin-American countries—a suggestion put forward at an earlier meeting of the Working Group. The Economic Commission for Latin America (ECLA) would probably be willing to cooperate in that venture if the Latin-American countries concerned would contribute their share to the project.

The CHAIRMAN remarked that it seemed a fairly simple matter to adjust the quality of the steel to suit the demands of the consumer. However, for orders of a certain magnitude, that possibility might turn out to be a rather complicated issue. In his view, specifications should be divided into categories. For that reason, Mexico had adhered to ASTM specifications to eliminate complications. In Latin America, the problem seemed to be related to prices and time schedules.

Mr. LEUSCHNER pointed out that the statements made by Mr. Frick in respect of specifications were not intended to convey the idea that those specifications would be imposed by steel companies; they were intended rather as a basis for discussion at meetings of producers, consumers and government buyers and would have to be approved with the consensus of such groups.

STEEL MAKING AND FINISHING

Range of application of steels made by different processes

29 October 1952—Afternoon

Chairman:

Augusto LEGRAND, Sociedad Mixta Siderúrgica Argentina, Buenos Aires, Argentina

Contributed Papers:

Thomas Steel Rails in France
  Jean PALMÉ, Ingénieur de la Société nationale des chemins de fer français, Paris, France
The Problem of Energy in Steel Plants
  Gabriel PÉRIN, Société Alsthom, Paris, France
Construction of the Thomas Steel Plant at Paz de Río and Production Costs of Thomas Steel
  Edouard DECHERF, Chief Engineer, Empresa Siderúrgica Nacional de Paz de Río, S.A., Bogotá, Colombia
Foundry Iron by Recarburization of Steel Scrap
  Roberto JARAMILLO, Manager, Empresa Siderúrgica Nacional de Paz de Río, S.A., Bogotá, Colombia
  Joaquín PRIETO, Assistant Manager, Empresa Siderúrgica Nacional de Paz de Río, S.A., Bogotá, Colombia, and
  Jaime RUDAS, Plant Superintendent, Empresa Siderúrgica del Muña, Bogotá, Colombia
Consumption of Iron and Steel Products in Mexico
  Marcelo ARAMBURU, Departamento de Investigaciones Industriales, Banco de México, México D.F., Mexico

Summary of Discussion:

Participants: Messrs. PALMÉ, ALLARD, MERCIER, BOULGER, BULLE, COHEUR, COSTA LINO, PÉRIN, ALBALA, KREBS, VUCETICH, SCHERESCHEWSKY, MACEDO SOARES E SILVA, DECHERF, CANGUILHEM, PRIETO, COCK, BUEHL, UCHOA, ARAMBURU

Thomas Steel Rails in France

J. PALMÉ

INTRODUCTION

The countries of Western Europe, and France in particular, are in a fairly unique situation as regards rails for their railways. In fact, the available sources of ore and the equipment of many of their steel mills induce them to use the basic converter (Thomas) for manufacturing rails.

Furthermore, railway traffic is very heavy, so that high quality rails are required.

This paper will therefore deal with the question of the rails used in France, including any special precautions taken in their manufacture and any differences between French and foreign specifications.

STATISTICAL COMPARISONS

The French railway system extends over 41,000 km., and is one of the longest in the world, so that it is considerably more developed than several of the largest North American systems, such as the Santa Fé railroad, and is 10,000 km. longer than the British railways.

Traffic on the French railway system is also among the heaviest in the world, some lines such as Charleville-Thionville having a daily traffic of approximately 100,000 tons (60,000 in one direction and 40,000 in the other). With regard to passengers, the French railway system also approaches a load carried by the largest North-American system, i.e., the Pennsylvania railroad.

In addition there are numerous wagons with 18-ton and even 20-ton axle loads travelling at speeds of from 60 to 75 km. per hour (the axle load is very close to that for locomotives, so that it is particularly high).

The load per axle of the locomotive material has recently been raised from 18 to 21 tons for steam engines type 242 AL and to 22 tons for electric locomotives type 2D2. However, in the case of very high speed electric locomotives, the load has been reduced to 17 tons. The 242 AL steam engines develop 5,000 h.p., and are the most powerful in Europe, whereas the electric engines of the 2D2 type used on express trains are even more powerful. The CC engines (5,450 h.p. in one hour; 4,480 h.p. for continuous running) hold the European speed record. Many French railway lines are electrified due to abundant supplies of hydroelectric power.

The power of the locomotives, coupled with good rail maintenance services, enables high speeds to be used: ordinary trains frequently run at 130 km. per hour; in the case of diesel powered cars this figure may be raised to 160 km. per hour and tests are currently being made to reach speeds of 180 km. per hour.

Briefly, then, the French railway system carries a very large traffic load, the loads per axle are heavy and the speed of both passenger and freight trains is very high.

THE THOMAS STEEL RAIL [1]

It is obvious that in order to stand up to such traffic, rails must be of high quality both for economic and for safety reasons. A rail does in fact have a complete task to perform as it is both a beam and a roadway. It must therefore resist the pressure and abrasion caused by wheels on the vertical and on the transversal plane, as well as the very intense compression developed on the tiny surface which makes contact with the wheel. The friction at high speeds frequently causes temperature rises sufficiently sharp to cause changes in the structure of the metal. Braking and slipping of the locomotive wheels have similar effects which are particularly destructive. The rail must be easy to weld, and finally, for safety reasons, it is essential that it has excellent shock resistance.

All these contradictory requirements are even more difficult to combine since the rail should be relatively cheap; the problem is further complicated by the high speed specifications.

From all the foregoing it is obvious that the rail is subject to constant research by both railway engineers and metallurgists. It is in fact a key product, and the study of the Thomas steel rail is thus very interesting, above all in a system where high speeds are used.

SPECIFICATIONS

Current French specifications lay particular emphasis on mechanical tests. Chemical analyses are made only for information purposes. It is in fact felt that the objective is a mechanical result and that the means used to achieve that result, and in particular the composition of the steel, should be left to the supplier's judgment. Furthermore, requirements as to the homogeneity of the steel are very strict, so that a macrographic test is specified.

Mechanical tests comprise:

(*a*) A shock test per ingot (on the head of the ingot which is the most fragile part);

(*b*) A tensile test every three heats (from the foot of the ingot which is the strong part);

(*c*) A Brinell test on each heat (at the foot of the ingot).

The shock test is a very special one: the rail is placed head downwards between two supports. A deep cut is made in it corresponding to two fifths of the thickness of the head made with a 100 mm. diameter milling cutter. The test results in tension at the head rather than at the base. In fact the head is less elastic. Furthermore, the cut opens up that part of the head where defects are usually evident.

A 300 kg. weight is then dropped on the rail.

This is a severe shock test and imported rails made from heats which have successfully passed the test of other specifications have often proved unable to resist it.

COMPARISON BETWEEN FRENCH AND FOREIGN SPECIFICATIONS

By contrast with some foreign specifications, the French ones do not specify any ratio between chemical composition and average weight, which is logical since the chemical composition is not fixed. French specifications are the only ones which impose minimum 12% head [2] rejects. Moreover, they are among the only ones demanding hardness and macrographic tests. French shock tests tend rather to ascertain the breaking point and approach the rupture test conducted on a small scale. This test is now being perfected so that the breaking work can be determined and measured.

French specifications are also the only ones which include a guarantee clause to the effect that "any rail taken off the track which breaks or deteriorates, for other causes than normal wear during the guarantee period" shall be reimbursed by the supplier at the price of the most recent order.

The tests affect a much higher proportion of the material since the sampling unit is either the ingot (average 4,200 tons) or the heat (average 20 tons), whereas in many foreign specifications the sample is taken from a 50- or 100-ton unit; French quality control is thus much stricter.

Finally, French tolerances on dimensions are considerably closer than most foreign ones.

STANDARDIZATION OF SPECIFICATIONS FOR THE SUPPLY OF RAILS

The study of standard specifications for the supply of rails is currently being undertaken through the Union internationale des chemins de fer. These studies are so far advanced that it is expected to apply the new specifications in 1953.

The new international specification is in close accord with current French specifications and it is noteworthy

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[1] Belgium and Germany, which are very highly industrialized and have very heavy railway traffic, both use Thomas steel rails.

[2] This in practice exceeds 20%.

that no discrimination has been made in the quality of steel used for rail manufacture, i.e., Siemens-Martin, Thomas and Bessemer, so that similarity of these quality steels is implicitly recognized.

MANUFACTURE

Such strict specifications require careful manufacture and in this connexion the reader is referred to the papers prepared by M. Decherf. No manufacturing details will therefore be entered into here, except when dealing with particularly interesting points.

SPECIAL PRECAUTIONS FOR THE MANUFACTURE OF THOMAS RAILS

First of all uniform hot liquid pig iron must be obtained with a Si content below 0.60 and at least 0.500 Mn. The S content should not exceed 0.100. Rails are not made when there are several unfavourable factors.

Desulfurization takes place by means of sodium carbonate, usually at the mixer outlet. If the pig iron is particularly high in sulfur, an initial desulfurization takes place at the moment of tapping the blast furnace. It is well known that desulfurization is all the more efficient when the pig iron is more fluid. This fluidity also facilitates the converter operation so that some French factories use a fluidity test.

Special care is taken in deoxidization which takes place both in the converter and in the ladle using various deoxidizers, well weighed. Tapping is done at the lowest possible temperature to restrict segregation and checks on cooling. Only new smooth ingot moulds are used as any roughness might hamper the removal of the ingot and might cause scabs. They are lacquered and covered with a heat-resisting preparation. The number of passes in the blooming mill is higher than for ordinary products, with a view to avoiding excessive crushing and rolling defects.

Rolling temperatures are controlled and the operation is finished at the lowest possible temperature to improve the steel's mechanical properties. The rolling operation is performed in a special way so as to avoid the appearance of longitudinal lines which occasion very dangerous, and previously very common, surface defects.

Care in the straightening process prevents deterioration; slight defects of alignment are later corrected.

In order to reduce this straightening operation to the minimum, some factories use a special device when the rail leaves the rolling mill, giving the still red-hot rail an inverse curve to that which it would naturally follow during cooling. When the device is well regulated the rail is almost straight on cooling.

The rails are finished on special machines which simultaneously cut and perforate the two ends. In several factories the perforating is now done with a special tool combining a drill and a milling machine, so that the hole, once opened, is polished by the miller. Laboratory tests and practical service results have shown that these holes have considerable resistance to cracks.

It can thus be seen that the manufacture of Thomas steel rails is subject to careful precautions.

COMPARISON IN MANUFACTURE OF SIEMENS-MARTIN RAIL WITH THOMAS RAILS

Although Thomas rails are more widely used in France, there are nevertheless large tonnages of Siemens-Martin rails, coming from two different sources: first, rails imported during the First and after the Second World Wars; and second, rails made in France by some southern plants. Most of the old "Midi" system uses these rails; some other French mills have also made Siemens-Martin rails from time to time.

All these rails were made in basic open hearths and can therefore be compared with Thomas rails from the manufacturing viewpoint. First of all, tapping temperatures are lower in Thomas than in Siemens-Martin, which tends to reduce harmful factors accompanying the solidifying of the steel. Then, tapping speeds are more uneven in the case of Siemens-Martin, due to the higher tonnage of the heats.

There are considerable differences in the chemical composition of Thomas and Siemens-Martin steel having the same mechanical strength. Thomas steel has considerably less C and more Mn, as well as more phosphorus (0.04 to 0.07).

The results of the foregoing are as follows:

(*a*) With the same strength, Thomas steel is much less prone to scratches and internal fractures than Siemens-Martin. That is why the transversal fissures so common in the United States before the introduction of special processes for controlled cooling, are rarer in France;

(*b*) Shock resistance should be lower in Thomas steel because of the presence of phosphor, but as it has less carbon and more manganese,[3] and as special care is taken in manufacture, this is offset and Thomas steel is often more resistant to shock than Siemens-Martin. This has been observed both in the very severe factory tests and during service.

Naturally there are advantages in the Siemens-Martin process, particularly in the fact that the quality of the raw materials need not be so high, but there appear to be equal advantages in both processes.

Briefly then, high quality Thomas steel can be made and has been made in France for a long time. Thomas rails leave the plant almost free from the internal cracks so frequent in Siemens-Martin steel, and are more shock resistant.

SECTION AND WEIGHT

Until recently the old south-west and western systems of France specified bull-headed rails but their manufacture has now been suspended. The standard sections today are all for "flat-based rails".

Standard French sections have the following weights:

Kg/metre	46	50	55	62
Pound/yard	93	101	111	125

There is no great apparent difference between French and other European rails, taken as a whole.

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[3] With such a low C content, the Mn improves rupture strength.

Furthermore, details of French sections which have been the subject of very careful studies are only interesting to specialists. The study of a rail section is rather complicated as the frequently contradictory specifications of the railway and of the rolling mill engineer have to be taken into account.

The new sections now being studied by the U.I.C. with a view to standardizing European rails, combine a careful equilibrium of the masses of the base and the head to prevent deformation during rolling and thus facilitate straightening.

Size of bolt holes

The bolt holes are a delicate part of the rail as it is here that most of the ruptures and damage appear during use. These ruptures occur in directions to 45° according to the mathematical theory of elasticity.

Static tests have shown that the strain at the edge of the holes increases rapidly with the diameter and consequently all bolt holes have been reduced from 32 to 25 mm. diameter for the last 20 years and, as a result, cracks have become very scarce.

Length of rails

French rails are usually 18 m. long. This reduces maintenance costs and has improved the life of the rail. (France has a moderate climate with no sharp temperature changes and consequently no great space is required between these 18 m. rails, as would be the case in the United States).

Very long bars

Welding of Thomas rails was studied extensively and this has led to their being assembled in great lengths—800 m. This type of rail has now emerged from the experimental field as 100 km. of main lines of different kinds have been welded during the last four years.

This welding naturally gives rise to new requirements due to the contraction of the steel during cold weather. In order to resist such strains the rail must be free from any outside defect, especially at the base.

From this standpoint, French Thomas rails rolled so as to form the base in the first passes have given very good results.

Welding of the rails

Two methods are used:

Electric welding, in specialized shops with automatic machines. The operation is followed by kiln annealing to prevent fragility.

Alumino-thermic, carried out on the railway bed. This process has been perfected and simplified in the last 20 years and is in great use because it is easy and the weld has considerable shock resistance. Annealing takes place on the terrain with a muffle furnace.

By these processes old rails can be used again after cutting and good results are obtained in reassembling rails which are worn or have different sections. Furthermore, as indicated previously, this process is useful for preparing very long rails.

In practice, lengths of from 54 to 96 m. are welded electrically in the shop and are then assembled on the railway track by the alumino-thermic process.

Rail defects

Rails may have different defects which are fairly common to all administrations. Some of the more common or dangerous ones are indicated below, together with the measures adopted in France to overcome them.

Fractures in the bolt holes for fishplates. The number of these fractures has been reduced by adopting a bolt hole of 25 mm. instead of 32 mm.; furthermore, a notable improvement is expected as a result of the milling operation which is being steadily adopted in the various factories. One factory which, some 12 years ago, put out 60,000 tons using this method, has to date experienced no fracture of the bolt hole, which is a very unusual achievement.

Internal transversal fissures. This defect is much more common in Siemens-Martin than Thomas rails and is therefore very rare in France. Nevertheless it is serious because several fractures may occur simultaneously in the same rail. To overcome this danger, mobile equipment is being assembled with a view to effecting systematic inspections of worn networks.

This defect is sufficiently rare in Thomas rails to avert any need in France for controlled cooling during manufacture, as is the case in the United States.

Crushing of rails. This may occur at the joints, at the switches or in the lower rails in curves over which slow, heavy rolling stock circulates.

The last example is tending to disappear in France due to higher speeds as a result of the widespread adoption of continuous brake in freight trains. With regard to the first two, the steady increase in loads and speeds make them particularly dangerous. This is why various methods are currently under study for hardening the rail unions, all of which consist in tempering the rail, either after rolling or on the track, by using various fluids such as water, compressed air, etc. Another quenching process by means of high frequency currents is also under study.

In addition, it is usual to strengthen worn unions by electric welding.

Fractures and cracks due to segregation have practically disappeared since the inclusion of macrographic tests in the specifications.

Finally, the relatively serious defect of *longitudinal cracks at the base* degenerating into transversal fractures, has disappeared almost entirely from French products through the use of new rolling methods. This type of defect formerly caused many rail fractures during cold weather periods and therefore its elimination constitutes obvious progress.

In other words, a very great effort has been made and as a result the main types of defects have almost entirely disappeared.

In the installations where PIC tests have been effected, the same regularity was not always observed, some tests corresponding to washeries treating only a certain quality of coal, whilst others, on the contrary, treated several qualities successively. There cannot be a single imperfection value for equipments working under these conditions.

In the small-coal wash-boxes the imperfection, furthermore, depends on the mesh at which samples are de-slimed and on the concentration in solid products in the washing water. Lower imperfection values are generally obtained with units handling a low percentage of refuse and middlings, i.e. over-dimensioned units, and with a fairly regular raw feed.

When washing fairly easy products, a very economical solution consists in using a jig treating products unscreened before washing, as is frequently the case in England. On up-to-date jigs made lately in the United Kingdom, and fitted with numerous improvements over Mr. Baum's original conception, very good results are obtained.

The separation gravity differs between large and small coals, but the difference is rather small. The +16 mm. being washed at 1.6, the −16 mm. are washed at 1.7 or 1.8. Imperfections registered in such a two-product washing jig are similar to those which would be obtained in a separate jig.

STANDARD DIAGRAM OF INSTALLATIONS REALIZED BY THE PIC COMPANY

In the case of products very difficult to treat it is absolutely necessary to use techniques and units including the most recent improvements. Such a product is the Bousquet d'Orb coal, mentioned by Mr. Cheradame, the nature of which is very similar to that of some difficult coals encountered in South America.

A standard installation should include:

(*a*) A pre-classification at about 10 mm.;

(*b*) For the washing of +10 mm. coal, heavy-medium washing units.

(*c*) For the treatment of −10 mm. coal, units giving very small imperfections and a great working stability, i.e. compressed-air jigs with filtering bed, fitted with automatic controls;

(*d*) Provisions for the intermediary products which should be submitted to very severe crushing tests, not forgetting to ascertain the following points: the amount of washed products and refuse—taking into account the mesh to which they are crushed; whether the re-cycling of recrushed products gives a sufficient quantity of saleable goods, taking into account the imperfection of the unit utilized.

Owing to coal texture, solutions may be quite different. In some cases, crushing frees a sufficient quantity of washed products to justify their recycling. Unhappily, in many cases, crushing middlings still contain 70%, and often a larger percentage, of mixed products. When this is so, taking into account the imperfection of the washing unit, it may not be advisable to rewash the middlings, since rewashing of these products necessitates the installation of supplementary washing units, in view of the fact that recrushed middlings are very difficult to wash.

As has been mentioned, French washeries generally produce middlings which are burned in thermic installations specially equipped to that effect.

(*e*) Provisions for the flotation of slurry to obtain clean products which can be reincorporated into the washed small coal;

(*f*) The water circuit shall be carefully designed and the settling tanks of suitable size, so that the percentage of solids in the washery water is kept sufficiently low not to degrade the washed small coal.

The products, discharged outside the washery, will consist of flotation tailings which, practically, do not contain any saleable products.

In some installations, with no flotation, the circulation water circuit includes a bleed chemically flocculated, so that very small coal or clay particles are eliminated from the circuit. Slurries are recovered on vacuum filters.

For less difficult products encountered, as for instance in Colombia and Chile, jig washing may be employed for large and small coal.

In order to ensure a flexible installation, the maximum of precision and the independence of the separating gravity for large and small coal, PIC plants include a pre-screening of raw coal at about 10 mm., and separate treatment of large and small coal.

A standard flow-sheet of the installations carried out several times in France, and proposed in some tenders for South America, is shown in Figure 1.

When a maximum of precision does not constitute the main objective, and for an easy coal, the size and specific gravity of which are fairly regular, the combination of a Baum jig for the unsized raw coal and of a small-coal wash-box with a filtering bed for retreating the sizes under 10 mm. produced by screening of the washed products from the primary wash-box, may be used.

An example of a flow-sheet frequently used by the ACCO Company in England, and put forward in certain instances for South America, appears in Figure 2.

STATISTICS

It is quite easy to compare the Thomas and Siemens-Martin rails used on the French railway lines:

(a) *Siemens-Martin rails made in France from Lorraine ore* are exactly the same as Thomas rails made from these same ores;

(b) *Siemens-Martin rails made in France by some plants* using foreign ores or particularly pure ones from the Pyrenees, have for several years past given rise to fairly frequent fractures during use;

(c) *The Siemens-Martin rails imported during the 1914-18 war* have given good results in service. Unfortunately, no conclusions can be drawn, because first, the tonnage was relatively small, and second, as the sections were fairly light (40 kg./m.) they were used on secondary lines;

(d) *Siemens-Martin rails imported in fairly large quantities in 1945, 1946 and 1947*, and used on main lines, gave rise at the beginning to a certain number of incidents during unloading, so that some changes in the specifications were requested of the manufacturers. The tests made by the French railways on a specific number of them showed somewhat unsatisfactory macrographies and some fragility. Subsequent deliveries have improved, but no opinion can yet be formed of these rails because of their short period of use.

From these comparisons, the Siemens-Martin rail does not show any superiority.

PRESENT TRENDS

With regard to sections, the trend is to increase the weight per metre and to use sections with better equilibrium. As stated above, sections are being standardized throughout Europe, and in the relevant studies being made by the S.N.C.F., modern laboratory tests are being used more and more, while strain-gauge tests are being made on the track.

Consequently, progress has been made with regard to the quality of the steel over the last 20 years under the aegis of the Joint Commission on Rails, founded in 1925 by the French Ministry of Public Works. This Commission, composed of S.N.C.F. engineers and producers of rails with recent assistance from the very important steel research institute, studies all problems connected with the manufacture of rails and their behaviour in service. It has at its disposal the laboratories of the various members of the Commission and in particular that for rails of the S.N.C.F. and the large I.R.S.I.D. laboratories at Saint-Germain-en-Laye, which constitute the most complete combination to be found in France for the study of metals.

Under the very active influence of the Joint Commission, the tendency with regard to quality is to make an ever more detailed inventory of all the factors affecting the value of the finished product (quality of raw materials, technical controls at all stages of manufacture), while at the same time the methods used are being steadily improved.

For its part, I.R.S.I.D., which has very considerable resources, has begun a vast research programme affecting the whole French steel industry in general, from which the rail in particular will undoubtedly benefit.

CONCLUSION

In view of the foregoing, it may be affirmed that those Latin-American countries having high phosphorus ores can quite confidently use the Thomas process for making their rails.

The Problem of Energy in Steel Plants

GABRIEL PÉRIN

The purpose of this paper is to discuss an essential factor in the operation of steel mills, i.e., power. The manufacture of primary or semi-finished products implies the use of a considerable amount of energy in various forms and it is therefore essential, when attempting to reduce costs, to pay attention to its rational use.

This problem will be discussed here only at a theoretical level with a view to showing that, in most cases, steel mills have tended to complement their specific steelmaking equipment with thermal plants. A thorough coverage of this subject would require the study of recent improvements leading to a higher efficiency and safety. Such research would, however, be beyond the scope of this paper, which will therefore be confined to outlining this energy-producing technique.

Current steelmaking practice provides, as a by-product, energy which can be used directly by the various departments of the plants concerned.

Actually most installations, including those of recent construction, continue to make steel in two stages: first, there is the reduction and carburizing of the ore in order to obtain pig iron, followed by decarburization of that pig iron to produce steel. In other words, there are two contradictory but currently inevitable operations.[1]

Nevertheless, the first stage of the operation, which is carried out in the blast furnace, provides as a by-product, a hot gas containing carbon monoxide and known as blast furnace gas.

Furthermore, there is a previous operation which may be considered as an integral part of the steelmaking process, i.e., the transformation of coal into metallurgical coke. The coking of the coal also gives rise to a hot, combustible gas, coke by-product gas, which has a higher heating value than the blast furnace gas.

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[1] See F. Guéry: "Evolution de l'energie dans les etablissements sidérurgiques", *Revue générale de mécanique*, April 1950.

It should not be forgotten that coking coal cannot be used as such on leaving the mine, but it is washed in order to concentrate the C needed by a coking coal, thus furnishing by-products in the form of middlings and tailings which have some heating value. Even though this value may be low, it can be recovered.

It can be seen therefore that at this stage there are various by-products having values which were unknown during the last century. Nevertheless, research into industrial energy over the last 50 years has resulted in these by-products forming part of the manufacturing process from which they themselves originated. Nowadays the energy is described as a "product" which is too valuable to be wasted as it is a vital economic factor; it is the constant concern of the builders or users of apparatus producing such energy, with a view to preventing losses through a careful study of efficiency and consumption.

SOURCES OF ENERGY IN IRON AND STEEL MILLS

In principle, the only source of energy in iron and steel mills is coal, which is transformed into coke and coke by-product gas, with the eventual production of middlings. Subsequently the coke itself supplies the energy required for producing pig iron from the ore, while it also produces the blast furnace gas. As in principle the output of coke is based solely on the production requirements of cast iron, there is no surplus of coke apart from that used in the blast furnace. Thus, the available sources of energy for the various services are: raw coal, middlings, coke by-product gas, and blast furnace gas.

It is therefore natural that the last three which are obtained "gratuitously" are given priority use, the coal only being used as an eventual complement.

There is, however, a wide difference in the heating value of coke by-product gas and of blast furnace gas, and they therefore have different uses. Coke by-product gas has the higher heating value, which may range from 3,000 to 4,500 calories/m^3. Blast furnace gas only has a heating value of 500 to 1,000 calories/m^3. With regard to the amounts of each obtainable from a ton of coal transformed into coke and later incorporated into the ore, the following figures might serve as an example: a ton of coal with 25% of volatile matter will provide some 700 kg. of coke and 300 m^3. of coke by-product gas. The 700 kg. of coke in the blast furnace provide 3,000 m.^{3} of gas by means of some 2,200 m.^{3} of air blast. Finally, each ton of coal gives rise to 1.2 x 10^6 calories of coke by-product gas and 2.4 x 10^6 calories of blast furnace gas. The typical composition of coke by-product gas is as follows:

	Per cent	
CO	7	
CO_2	2	
N_2	6	a.h.c. = 4,000 calories/m.^{3}
H_2	55	
CH_4	30	

and of blast furnace gas:

	Per cent	
CO	30	
CO_2	10	
N_2	56	a.h.c. = 900 calories/m.^{3}
H_2	3	
CH_4	1	

It may be seen from the above that the essential combustible element of this second gas is the carbon monoxide, derived from the reduction, by means of the excess carbon, of the carbonic gas released at the blast furnace tuyeres.

The differences between these two kinds of gas emphasize the importance of making the right selection for different uses. Formerly, the poverty of blast furnace gas led to its being considered as a residual or waste product, but subsequently it was rehabilitated through the use of gas motors. Although these motors are currently tending to disappear, the use of blast furnace gas continues to be indicated for heating at a low temperature, as in the case of the air to be blown into the blast furnace or into the chambers of the coke plants. Furthermore, a very important use for this gas has been found as a result of the considerable progress achieved in the realm of boiler combustion, as boilers have been so modified as to obtain their energy from many waste products. Nevertheless, for certain purposes as in the case of heating open hearth furnace gas and rolling mill furnaces, it is necessary to use coke gas (or a blend sufficiently rich in coke gas). Since the gases produced in the coke plants and in iron and steelmaking are, as will be seen below, usually surplus products, the coke gas has to be reserved for certain of the plant's requirements, for which it is the only generally acceptable product.

ENERGY REQUIREMENTS IN IRON AND STEEL MILLS

The distribution of energy requirements depends on a variety of factors and primarily on the manufacturing programme itself. For example, in a plant producing pig iron only, mechanical energy requirements would be very low. The importance of such energy will, however, increase in relation to the number of rolling mills used and the size of the steel produced. For instance, the manufacture of thin sheet by cold rolling, compared with other products, requires a relatively large quantity of kilowatts.

Nevertheless, some indication may be given of the amount valid for an average factory. The table on page 428 gives an approximate indication of it. This example covers an average plant manufacturing all the coke it requires but having no electric melting furnaces. It is assumed that the coking ovens are heated with blast furnace gas, whereas the open hearth furnaces and those of the rolling mills are heated with blended gas, 70% of the heating power being provided by coke gas and the remainder, i.e., 30%, by blast furnace gas. The relationship between mechanical energy and the heating energy has been established on the basis of 3,000 to 4,000 calories per kWh.

THERMAL PLANTS

The table shows quite clearly that over a quarter of the energy used is mechanical. This figure would be lower in the case of a plant producing only pig iron and higher in the case of other specialized plants, as for instance those manufacturing sheets of the higher gauges. Since there is a surplus output of energy in the form of combustible gases, it is obvious that this has to be recovered and transformed into mechanical energy without having to resort to the use of more coal. It therefore follows that the apparatus or devices capable of effecting such a transformation

Table 1

ENERGY DISTRIBUTION OF IRON AND STEEL BY-PRODUCT GASES

Source of energy	*Energy produced as a percentage of total energy used*	*Uses for energy*	*Energy used as a percentage of the total used*
1. Coke by-product gas	45	Coke by-product production	4
2. Blast furnace gas	90	Coke oven heating	22
		Blast furnace stoves (Cowper)	24
		Various furnaces	2
3. Gas mixture, 70% of its heating value obtained from coke by-product gas	Mixture of 1. and 2.	Open hearths	6
		Heat treating furnaces for rolling	15
4. Mechanical energy	Included in items 1. and 2.	Blast furnace blowers	7
		Bessemer converter blowers	4
		Rolling mill equipment	14
		Maintenance and miscellaneous uses	2
TOTAL	135	TOTAL	100

are of great economic importance for the iron and steel industry.

The first devices used during the second half of the last century were boilers and reciprocating steam engines. Although they were of considerable interest at that time, their use was limited due to certain aspects such as their poor yield, insufficient recovery of blast furnace gases and the attempts made at that time to burn very poor gases in the boilers, as a result of which coal had to be used as well.

Some fifty years ago, great progress was made through the introduction of gas motors, which gave an excellent yield and were perfectly adapted for the use of blast furnace gas; they were used to operate the blowers of the steel mills. Nevertheless, since ever-increasing demands were made on them, gas motors began to present more and more drawbacks. The main factor emphasizing the limitations of gas motors has been the progress made by electricity. Actually the alternating current generators require machines with a very uniform rotation speed, in order to prevent the creation of parasitic alternating currents, and their regulation must be easy and rapid to facilitate coupling. In this connexion, notwithstanding the excellent yield provided by the gas motors, the superiority of steam turbines soon became evident and considerable progress was made in this field within a very short time. Furthermore, an increase in their unit power was soon achieved, although it had proved impossible in the case of the gas motors.

This gave rise to the present position which appears to be a kind of intermission in the evolution of ideas, since fairly specific solutions had already been found in relation to present operating conditions in iron and steel mills. Most modern installations produce the required energy in the mechanical or electric form by means of turbo-alternator groups fed by boilers using combustible gases.

Consequently, the progress achieved in the field of thermo-dynamic and electric material in the period between the two wars and in particular since the last one, have converted these power plants into high-precision instruments, closely adapted to the technical and economic requirements of the industry.

UNBALANCED OUTPUT AND INTERCONNEXION

It is, however, necessary to specify the conditions for maintaining a balanced output of energy in a steel mill. This raises difficult practical problems, but some solutions thereof are interesting.

In the first place, the summarized balance shown in the table indicates that the energy produced in the form of combustible gases is usually in excess of the plant's requirements. There is therefore interest in seeking means of reducing the production of such gases. With regard to the output of blast furnace gas, there are few possibilities of reducing it, as it bears a direct relationship to pig iron production. The only means of reducing the amount of gas produced would be through a development in technique which would, for instance, lead to a reduction in the amount of coke required for a given output. Such development is currently becoming evident through the improved preparation of the blast furnace burden. It is unlikely, however, that this will progress very far. The surplus of blast furnace gas constitutes a complicated problem, because, as it is such a poor gas, it cannot be economically stored in gasometers nor transported over any great distance.

This is not the case with the coke by-product gas, as it has much higher heating properties and can be used to advantage in a great variety of applications. Steel mills normally encounter no obstacle in selling this surplus, since the plants are usually located in fairly large industrial centres and they can therefore distribute the gas, either to other industrial plants or to the neighbouring towns. Nevertheless, the problem is often solved more easily by limiting coke production to the amount which will provide exactly the quantity of gas required for its installations and obtaining any complementary supply—when necessary—from a commercial metallurgical coke plant. There are even some plants which produce no coke at all; in this case they have to install pipelines for the gas, a process which is not always profitable.

Another difficulty which is not shown in the summarized table but which does tend to constitute a problem, is the fact that there is no synchronization between the production of gases and their use. In principle, blast furnaces and coke plants operate continually without any daily or weekly interruption, whereas the other installations are subject to a much more irregular timetable, operating at a lower rate at night time and stopping almost entirely at the end of the week. Consequently, there is at times an output of gas which cannot be used immediately. There is no serious problem in the case of coke gas, but in the case of the blast furnace gas it is almost impossible to solve the problem within the confines of the plant.

This problem has nevertheless of late been very satisfactorily solved by electrical experts, by means of interconnexion. It is a well known fact that the greatest advantage in the use of electric power is the ease with which it can be transported, and there is no better example of this than its use in iron and steel mills. In other words, instead of stressing the impossibility of storing the continuously produced gases, it is much easier to send all of them to the power plant which, by means of an interconnecting system, can sell current to nearby factories, should these require supplementary electric power, or can send such current to the country's general network. This solution does, however, have certain defects, from the general viewpoint. At night and on Sundays or holidays, i.e., when the steel companies have a maximum supply of energy, the general network has lower demands to meet. This occasions frequent stopping and starting at regular intervals, of a certain number of thermic groups inserted within the whole of the network—except in those cases where there is a fairly liberal supply of hydroelectric power, which is seldom the case. Thus, the present trend of technical progress lies in research into conditions of frequent stopping and starting without causing inconvenience to the power plants, including the large ones.

THE SEARCH FOR LARGE UNIT POWERS

A study of power plant material indicates that all the devices used, particularly the turbo-alternator groups, provide a much higher yield as their power increases. Therefore, in view of the interest in producing energy as cheaply as possible, present efforts are directed towards providing the power plants with groups having the highest possible unit power.

It is seldom possible in steelmaking to consider the adoption of groups similar to those which currently provide the greatest yield, with a unit power standardized at 50,000, 100,000 or 125,000 kW. The power required for operating most of the steel mills is in fact much lower and the power plant installations are usually made up of units of 10,000, 20,000 or 30,000 kW. Thus, modern technique cannot be employed to the best advantage in the case of an isolated plant.

Some interesting experiments have been made with a view to solving these difficulties. The idea is to concentrate, in one single electric power plant, the production of energy necessary for several steel mills which are not too far distant from each other. The main drawback of such a system lies in the construction of the gas pipelines from the various producing factories to the common power plant. This initiative is naturally very attractive and there is already one large-scale installation of this type in France. Four very large steel mills in the district of Longwy (Lorraine), have changed their individual power plants to one single joint power plant: the Herserange power plant. This plant recently entered into operation and, once in full operation, will produce 160,000 kW.

Construction of the Thomas Steel Plant at Paz de Río and Production Costs of Thomas Steel

EDOUARD DECHERF

1. *The capacity of the converter*

In a new industry, where provision has to be made for future expansion, it is difficult to determine the capacity of the converter. To achieve reasonable production costs, a converter with a minimum capacity of 55 tons per hour is necessary.

Although in Europe 13- to 15-ton capacity converters are still extensively used, Paz de Río will be equipped with 22-ton converters which later, by a reduction in the thickness of the lining, might take 24-ton charges. This 22-ton converter, working at the rate of three blows an hour, would give an output of between 60 and 66 tons per hour.

2. *Height of the working platform*

At the altitude of Paz de Río, the dimensions of a 22-ton converter are approximately the same as those of a 28-ton converter at sea level. The height chosen for the working platform is 6 to 6.5 m., so that the tilting axis is from 7 to 7.5 m. above the ground.

3. *Distance between pillars in the converter building*

This will be of 12 m. in order to allow for the substitution of the 22-ton for larger converters.

4. *Working platform and chimneys*

Where there is only one pouring crane, as in Paz de Río, a rotating platform is required in front of the converter, so that a bridge crane can be used during maintenance or repair of the pouring crane. The chimneys should be wide and high, and lined with cast iron plates. Waste recovery is important; its effect on output and cost per ton of Thomas steel is appreciable.

5. *Location of the dolomite shop*

This shop must be near to the converters, since it handles the disposal of used linings, converter bottoms and dolomite bricks.

6. *Handling of the limestone*

The screened limestone comes from the storage bins over a light railway, in buckets which discharge through a conical chute in the bottom. These buckets feed the hoppers of each converter, and the same method is used to supply lime to the working platform.

7. *Handling of scrap*

Scrap coming from the rolling mill is lifted magnetically by a crane to a chute which gravity-feeds it into the converter. The slope angle of this chute must be arranged with great care, so that falling scrap does not damage the converter lining.

8. *Location of Thomas slag storage*

This must not be too far from the converters. The car which carries the slag ladle can be equipped to discharge the liquid slag in the storage yard. Two possibilities are being contemplated: in the first, the slag ladles are dumped directly on the yard, the disadvantage being that the rail switches have to be kept clean; in the second this disadvantage disappears, but an installation for transhipment of the ladle is necessary.

9. *Pouring basin*

A single pouring crane gives the following advantages:

(i) It is easier to pour in the centre of the ingot mould;

(ii) After the mould is filled, the ladle remains on the crane hook, where it can be cleaned with ease, and fitted with a new seat and stopper for the next heat.

It is difficult to pour small ingots of unkilled Thomas steel. From a 12-ton converter, the ingot should not weigh less than 700–800 kg., and from a 20-ton converter, not less than 1,250 kg. (Paz de Río). If the output per hour increases, the bridge crane is used in addition. Pouring into the ingot moulds is done directly from a ladle carried by the bridge crane; during this operation, the pouring crane is not used, but an additional crane delivers a new ready prepared ladle to the pouring crane to be used during the next blow.

APPROXIMATE COST PER TON OF PRODUCTION OF THOMAS INGOTS AT PAZ DE RÍO

Calculations are based on raw material prices as given by the Empresa Siderúrgica Nacional de Paz de Río indicate the cost of one ton of Thomas steel as being US$ 25, excluding fixed overhead.

The main factors affecting the production cost are:

1. *Ferrous material per ton of Thomas pig iron*

Table 1 shows the influence of this factor on cost:

Table 1

CONSUMPTION OF FERROUS MATERIAL PER TON OF THOMAS PIG IRON

(metric tons and US dollars)

Consumption of ferrous material per ton of pig iron	1.085	1.100	1.120	1.150
Direct cost per ton of pig iron	27.125	27.500	28.000	28.750
Increase in direct cost, not considering larger consumption of dolomite [a]		0.375	0.875	1.625
Working conditions of the blast furnace	Basic slag, low silicon in pig iron (0.4%)	Basic slag, low silicon in pig iron (0.4%)	Basic slag, silicon 0.4 to 0.5%	Acid slag, high silicon content in pig iron

[a] Higher operating expenses increase cost from three to four times these figures representing only additional cost of raw material.

Furthermore, when the consumption of ferrous raw material is high, pig iron is of poor quality, resulting in a low ingot yield in the blooming mill.

2. *Price of scrap produced by the plant*

Since the war, the price of scrap has been low in Europe, but this is not the case in South America. If the scrap costs more than US$25 per ton (the same as the Thomas iron), the cost per ton of ingots will increase in spite of a higher daily output of ingots for the same input of Thomas iron.

3. *Price of Thomas slag*

The slag is crushed to a fine powder and the price obtained depends on two factors:

(a) The total P_2O_5 content, which must be greater than 16%;

(b) The solubility in citric acid, which must be greater than 75%. In general, solubility increases with the fineness of the grain.

The content of SiO_2, CaO and P_2O_5 must be high in order to form the greatest possible quantity of ortho-phosphosilicate of lime which has a 99% solubility in the citric solution. The amount of lime in the slag is generally more than sufficient, but often there is not enough silicon, since a certain amount combines with the oxides of iron and aluminium. The higher cost of the pig iron resulting from an increase of the silicium contents, as shown in Table 1, is not compensated by the additional value of the slag due to a solubility above 75% resulting from such increase in silicium content.

A slow rate of cooling of the slag, resulting from the use of large ladles, gives a better citric solubility. Tests of slag cake invariably show a better solubility towards the centre.

Thomas pulverized slag is sold in France, in 300 kg. sacks, at approximately US$17 per ton. This same slag, exported to Colombia, would cost approximately US$60 per ton.

Price estimates of US$30 per ton of slag in the forecasts appear, therefore, to be reasonable. Considering US$30 per ton of slag and US$25 per ton of rolling mill scrap, the approximate cost of a ton of Thomas ingot will be US$30.78 (not including fixed overhead).

Assuming that, as an alternative to the Thomas process, Martin tilting furnaces were erected at Paz de Río, using the procedure of the plant at Appleby, Froddingham (England), then the Thomas blast furnace iron at Paz de Río will have a phosphorus content of 2%, whereas the Appleby has only 1.5%. There would be no gain through the better price of Martin slag in Colombia, as its cost would be higher than that of the slag produced by Thomas converters, owing to the small scale of operations at Paz de Río.

Furthermore, it is not feasible to use this procedure at the Paz de Río plant, for two reasons:

(i) To avoid the initial outlay in a blooming mill and a soaking pit, it is planned to produce small ingots of 1,250 kg. It is not possible, therefore, to install small tilting Martin furnaces, whose operating cost would be high. Investment in Martin furnaces would also be much higher than is the case with the Thomas converters;

(ii) The omission of the blooming mill makes it highly desirable at Paz de Río to have the regular hourly production that can be obtained from a Thomas steel plant.

Foundry Iron by Recarburization of Steel Scrap

ROBERTO JARAMILLO, JOAQUÍN PRIETO and JAIME RUDAS

A few special conditions, such as availability of cheap steel scrap, high price of cast iron scrap, customs duties, taxes, etc., may justify synthetic cast iron manufacture.

All the preceding conditions apply in the case of Bogotá's local market, i.e., the lack of an iron and steel industry which makes pig iron by the standard methods, the maximum use of foundry scrap by the numerous small and antiquated foundries and the lack of industries utilizing steel scrap. Therefore, and especially a few years ago, the price of grey iron castings was sufficiently attractive to justify an effort to start its production by any suitable method.

GENERAL TECHNICAL ASPECTS

Studies on the solubility of carbon in grey iron show that the rates of solution, at high temperatures, are governed by the following factors:

(1) Temperature of the liquid metal;
(2) Time of contact with the carburizing agent;
(3) Type of carburizing agent;
(4) Active surface, concentration effects and effects of mixing;
(5) Effect of the atmosphere above the liquid metal;
(6) Effect of silica and other elements in solution in the bath.

1. *Temperature*

Results obtained by Ruer and Birens (1) and by Chipman (2) agree very closely, and indicate a gradual increase in the C solubility in cast iron, from over 4% for 1,150°C., to over 12% for 2,650°C.

The important point here is that at the temperature of 1,500°C. the solubility of carbon in molten grey iron is approximately 5.10 to 5.15. Under these conditions, therefore, the bath is susceptible to being carburized theoretically well above the contents usually desirable in cast iron.

2. *Time of contact*

Riley (3) demonstrated the importance of the time of contact between the carburizing agent and the bath. Keeping the temperature at 1,500°C., several carburizers were used, obtaining after 2 hr. 15 min. 2.09% C for coke, 3.40% C for bituminous coal and 4.7% C for graphite, using half the time in the latter case, however.

3. *Type of carburizing agent*

The effectiveness of the carburizing agents is also illustrated by the above discussion, but in analysing this problem it is necessary to keep in mind also the physical form, and active surface of the carburizer. Other factors being equal, the order of effectiveness is: graphite, petroleum coke, anthracite, bituminous coal, and coke among the commercial materials. But factors such as density of the material evolution of gases in the bath, active surface and others may change considerably. The order of effectiveness is consequently only an approximation.

Experiments have been made with lamp black, potassium ferricyanide, calcium carbide, and rich carburizing gases like acetylene, but actually no commercial operations have been developed with them.

4. *Active surface, concentration effects, and effects of mixing*

The active surface is considered of the greatest importance, although if the carburizing agent is too divided, the beneficial effect may be counterbalanced by the resulting floating action. Pulverized graphite has been shown to be ineffective after two hours of contact. Solubility is a diffusional process to a considerable extent, and, therefore, the formation of inert films and concentration gradients due to oxides, ashes, slags and common impurities present in grey iron, as well as the presence of other elements in solution in the iron, is of considerable importance in the process of solution. Therefore the agitation, either mechanical or thermal, is effective in reducing such interferences.

5. *Atmospheric effects*

Much experimenting is being done in the field of vacuum melting, or under controlled atmospheres, such as hydrogen, nitrogen, air, oxygen, etc. Nitrogen, for example, hampers C solubility. It is also important to differentiate between laboratory researches and industrial conditions.

6. *Silicon effect*

It is interesting, on the other hand, to consider the protective action of silicon at low temperatures on carbon oxidation, and inversely the preferential oxidation of carbon at high temperatures. Lange and Heine (4) have analysed this phenomenon and have demonstrated that the effect of temperature on the direction of the oxidizing action is predominant. At temperatures under 1,300°C. (2,371°F.), carbon does not oxidize in the bath, but silicon does. The ability of silicon to react with oxygen decreases as temperature increases, and above 1,400°C. (2,550°F), the protective action ceases.

Above 1,400°C. another reaction starts, namely the reduction of silicon from the refractories and the slag.

The reduction of silicon: $SiO_2 + 2\ C = Si + 2\ CO$, has been analysed thermodynamically by Heine (5). Theoretically there is equilibrium at 1,466°C. (2,671°F.), with standard concentrations of silicon and carbon in the bath, namely 1%. At temperatures in the neighbourhood of 1,427–1,482°C., (2,600 to 2,700°F.), near the equilibrium temperature, oxidation of carbon may proceed independently of silicon pick up, and thus carbon may be oxidized up to as low as 2%, parallel with silicon reduction. Carbon concentration in the bath is of course of importance in the action, and in most of the cases pertaining to cast iron there is sufficient carbon concentration above 2% to promote silicon reduction which, of course, is most true in cases where there is an excess of coke in the bath. In this case there may be even an increase of carbon in solution while silicon is being reduced.

The action which is under discussion is important from another angle, concentration of silicon in the bath. If there is no silicon present, or if its concentration is greatly decreased, below equilibrium concentrations at the temperature under consideration, it is clear that there will be a carbon boil and evolution of CO, the equilibrium being displaced to the right. Therefore, at a given temperature, there should be in the bath a minimum silicon concentration if oxidation of carbon from this reaction is to be avoided.

Manganese behaves much in the same way, protected by carbon at high temperatures and oxidizing easily at low temperatures. Reduction of Mn also takes place, but of course to a lesser degree as there is no appreciable concentration or source of MnO present.

THE RECARBURIZATION PROCESS AT THE FOUNDRY OF EL MUÑA

(a) *Description of the installation*

The installation simply consists of a 2.5 tons electric furnace, with fixed roof, 6 in. electrodes automatically controlled, door charging, and acid lined with a mixture of ganister and clay from the locality. There are in addition the common facilities of a modern foundry for moulding, sand mixing, core drying, laboratory control, and a small machine shop. The foundry produces steel or grey iron castings according to needs, and the capacity is about five tons of finished castings per 8-hr. shift. For the last four years it has been making grey iron castings from steel scrap by recarburization, with economical success and having developed a uniform operating practice.

(b) *Scrap and other materials*

First quality scrap is heavy and clean scrap, which is being used only for steel castings. Second and third quality scrap is used for grey iron heats, and consists of light plate, borings, turnings and the like. Coke analyses an average of 10–12% ash, 0.5–0.7% sulfur, and 85%–87% of fixed carbon. Rammed refractories are made with a local mixture of about 80% ganister and 20% clay, adding molasses as a binder. Roofs last about 40 heats, and relining of the walls and bottom is made about every 150 heats. Duration of linings and roofs is therefore below normal practice, but considering imports of refractory bricks would result in four or five times the cost, this practice is well justified.

(c) *Consumption of energy and materials*

The table below shows consumption of electric energy, ferro-alloys, coke, wood and other materials, and is self explanatory.

Table 1

AVERAGE CONSUMPTION OF ENERGY AND MATERIALS PER TON OF GREY IRON

Electric energy	800 kWh.	
Electrodes	10 kg.	
Coke	60 kg.	
Wood	16 kg.	
75% Ferrosilicon	2.5–3.0 kg.	
1st quality scrap	0.0 kg.	0.0 kg.
2nd quality scrap	600 kg.	700 kg.
3rd quality scrap	200 kg.	300 kg.
Grey iron scrap	200 kg.	0.0 kg.

(d) *Recarburizing process*

Table 2, on page 433, shows the heat sheet control. Average total time is about four hours. Melting starts with a cold furnace and lasts 1¼ hours approximately, and one hour if charging is done in a hot furnace. At the beginning approximately two thirds of the scrap is charged and melted with about 75 kg. of coke, placed at the bottom and centre of the furnace.

At the appropriate stage of melting, the remaining one third of the scrap is charged, and another 75 kg. of coke are added, so that a total average of 150 kg. of coke are used per heat, or approximately 60 kg. of coke per ton of grey iron. When melting ends, the resulting slag, high in iron oxide from scale scrap is taken out, to avoid its oxidizing effect. During the same period silicon has been oxidized, and carbon is at 2.0 at 2.3%. At this moment the bath is cold. Ferrosilicon is added to restore the metal lost by oxidation and to have approximately 1% in the bath in order to take advantage of the protective action of silicon on carbon at low temperatures and to avoid carbon boil by absence of silicon in the bath. On the other hand at this moment the bath is oxidized, and silicon has at the same time a deoxidizing action. Temperature is raised in order to have the metal at the proper conditions for the final recarburization, in some 20 minutes, up to around 1,500°C. measuring the temperature with an optical pyrometer. At this point wooden poles are introduced through the furnace door and a violent boil is produced throughout the bath for some 5 to 10 minutes, depending on the amount of recarburization desired. A sample is taken and carbon analysed. If the metal has not yet reached the specified carbon content, the poling action is repeated. In this fashion, in a short time and cheaply, recarburization is completed raising the carbon content up to 4% if desired. In the operation the bath is cooled somewhat and therefore a final heating period to the correct pouring temperature is made if necessary. It is preferred at El Muña to pour at high temperatures, around 1,450°C. or higher in order to take advantage of the superheating effects on the physical properties of the material. Ferroalloys are added mostly in the ladle. Occasionally aluminium has been used as deoxidizer and grain refining element. The quality of the material obtained is exemplified by the following analysis:

C	*Si*	*Mn*	*S*	*Cr*	*Ni*
3.70	1.29	0.54	0.038	1.31	1.42
3.70	2.12	1.09	0.052		

Table 2

HEAT SHEET FOR CONTROL

Date: 16 September, 1952 Material: Grey iron Consumption: 2,420 kW. kW ton: 971

Charge			*Operation*		
			Time	*Power*	*Cold furnace*
Scrap—2nd class		1,850 kg.			
Scrap—3rd class		650 "			
TOTAL		2,500 "	11.00 a.m.	off	
Ferrosilicon		46 "	12.00 a.m.	on	Melting 1st charge
Aluminium		—	1.30 p.m.	off	2nd charge
Coke		150 "	2.00 p.m.	on	Melting
			2.45 p.m.	off	Slagging–Sample C = 2.5 T = 1,300° C.
Production:					
Finished castings, wt.		2,060 "			
Yield	82.7%		2.50 p.m.	on	Ferrosilicon
Residue return scrap		50 "	3.00 p.m.	off	T = 1,450° C. Poling
Metal yield	99.6%		3.05 p.m.	off	Sample C = 3.05
		2,490			Poling
			3.10 p.m.	off	Sample C = 3.18
Time:			3.15 p.m.	off	Poling
Charging		1½ hrs.	3.17 p.m.	on	Heating
Melting and recarburizing		3 hrs.	3.25 p.m.	off	T = 1,460° C.
Analysis:					Pouring 1st ladle
Final carbon		3.21%	3.37 p.m.		Pouring 2nd ladle
Si		2.12%			
P		Traces			
S		0.040%			
Mn		0.50			
Tensile strength		42.500 psi			

During the latter part of the heat, when temperature is raised, the reducing action of carbon and silicon pick up is experienced. As a rule from experience a gain of some 0.3 to 0.4% silicon is obtained in the operating practice described.

Occasionally in times of shortage manganese and chromite ores have been used to gain some manganese and chrome.

In order to protect the bath to a certain extent from oxidation by the atmosphere, the second slag, which forms from the residual coke after the first slagging is practised, is not removed during the poling operation and heating until tapping the heat.

The action of the wooden poles fulfils the conditions of agitation, mixing, boil in the bath and elimination of diffusional processes, permitting the fast carburization desired.

From the economic standpoint it is easily understood that this procedure presents considerable advantage and saving in time and cost over the use of graphite, petroleum coke, and other carburizing agents. In this fashion recarburization accomplished by wooden poles has been estimated at 60–70% depending on conditions, and calculated by weighing the wooden poles and knowing the amount of carbon introduced into the bath. Coke carburizing efficiency is given at 40–50%, weighing the amount of slag produced and analysing its carbon content, as well as determining the percentage of carbon dissolved in the metal.

The amount of carbon pick up from the electrodes is small as 10 kg. per ton of grey iron is the electrode consumption, a figure which on the other hand is desirable for reasons of cost.

Consumption of Iron and Steel Products in Mexico

MARCELO G. ARAMBURU

I. INTRODUCTION

The present development of the Mexican steel industry is based on the general economic conditions of the country. These conditions were undoubtedly accelerated by the economic phenomena resulting from the Second World War, one of the most important being the rapid loss of balance between the supply and demand of commodities. This situation was common to all Latin-American countries, which imported a large proportion of the manufactured products which they consumed.

Efforts to increase the volume of supplies on the part of exporting nations and local manufacturers were not sufficient to meet the increasing demand,

and consequently new industries were installed, and old ones expanded, in view of this prospect of a larger market.

Economic conditions changed with the end of the war, but the industrialization of Mexico had reached a stage of development easily adjustable to former conditions.

Another important factor was the new structure of industrial investments in Mexico. This was created with relatively limited public savings, and by national resources allocated to industrial projects of some magnitude; foreign loans were obtained for undertaking projects in decentralized branches, or for the creation of new State-sponsored industries, and direct investments were made by foreign concerns to establish manufacturing or assembly plants in the country. Table 1 gives some measure of the industrial development of the country:

Table 1

Base: Quarterly average for 1929 = 100

Year	*General*	*Crude oil*	*Oil refining*	*Metallurgical mining*	*Electric power*	*Transforming industries*
1935.....	106	89	144	71	126	134
1940.....	120	99	162	75	154	154
1946.....	155	100	238	60	203	240
1951.....	206	173	334	72	300	298

The national income increased from 5,670 million pesos in 1939 to 42,000 million in 1951. Once deflated by the wholesale price index in Mexico City, the corrected figure for 1951 was 10,881 million, or twice that of 1939. The income per capita in 1942 was 447 pesos, and in 1951 it was 1,582, which deflated by the price index, give 368 and 410 pesos respectively. This increase in income naturally increased the demand for manufactured goods.

A comparison of the iron and steel production figures for 1942 and 1951 shows a sharper rise in some of the indices of Table 1.

Table 2

Year	*General index*	*Oil refineries*	*Electric power*	*Transforming industries*	*Iron and steel production*
1942......	100	100	100	100	100
1951......	156	193	187	165	268

The favourable conditions created by the economic factors discussed above promoted the implementation of plans for the expansion of the steel industry. The larger domestic market for iron and steel products made possible the installation of new plants, and the expansion of some existing ones, in both the basic and secondary industries. Notwithstanding these favourable conditions, domestic production does not entirely satisfy consumption, and it is still necessary to import certain products in large quantities (see Figure 1).

II. PRODUCTION

The volume of primary steel products manufactured in the country, flat sheets and sections, as well as iron and steel castings, depends upon the following factors:

(1) The amount of pig iron produced;

(2) The supply of domestic or imported scrap for the manufacture of steel ingots;

(3) The ingots, billets and blooms, domestic or imported, which are rolled;

(4) The supply of scrap railroad rails and axles, domestic or imported, which may be rolled.

A. *Pig iron*

Pig iron is at present produced in Mexico in three blast furnaces, which have a maximum daily capacity of 1,420 tons. Output rose from 122,156 tons in 1942 to 253,894 tons in 1951, the capacity for the two years being 140,000 and 497,000 tons respectively, giving an efficiency for the latter year of 51.1%. The generally low efficiency is attributed to difficulties in the transportation of raw materials, and to the relative scarcity of coking coal. However, the production of pig iron shows a rising trend,. and once technical improvements are made, the established capacity will be used to better advantage.

The Mexican industry has lately been interested in the development of other methods for the reduction of iron ore. One plant is already in operation in Monterrey, with an average sponge iron production of 700 tons per month. Other methods under consideration are low shaft furnaces and the Krupp-Renn rotary kiln.

B. *Steel ingots*

Steel is produced in Mexico in 10 open hearth furnaces having at present a maximum daily capacity of 1,510 tons, and 14 electric furnaces with a maximum daily capacity of 330 tons, giving an annual capacity of 644,000 tons of steel ingots. In general, the open hearth furnaces are used to greater advantage than the electric, although in 1951 the electric furnaces showed for the first time a higher percentage utilization than the open hearth.

There has been a constant increase in the capacity of the steel furnaces, and the percentage utilization is also rising; for a capacity increase of 1.24% between 1942 and 1951, the production rose 1.64%. The figures for total steel production and finished products are shown in Table 3.

Table 3

Year	*Tons*		
	Steel ingots	*Finished products (14)*	*Percentage of yield*
1942..................	172,459	138,114	80.0
1943..................	163,152	135,668	83.2
1944..................	174,732	141,168	80.8
1945..................	231,282	173,745	75.1
1946..................	257,924	208,482	80.8
1947..................	290,163	233,352	80.4
1948..................	291,826	229,010	78.5
1949..................	372,826	297,227	79.7
1950..................	390,070	305,438	78.3
1951..................	455,863	354,927	77.9

From 1942 to 1951, finished products rose by 157%. For those two years the percentage yield varied somewhat, but the average for the period was 79.47%, which is considered satisfactory.

C. *Finished products and iron and steel castings*

The figures in Table 3 show the volume of steel products finished in Mexico with steel produced in the country, but do not include products rolled from scrap axles and rails, nor steel ingots and ferrous alloys which are imported. The full totals are given in Table 4.

Table 4

	Tons				
	Rolled from steel				
Year	*From domestic steel*	*From scrap axles and rails and imported steel*	*Sum*	*Iron castings*	*Total steel and iron*
1942	138,014	20,850	158,864	14,252	173,116
1943	135,668	21,501	157,169	13,046	170,215
1944	141,168	23,270	164,438	14,194	178,632
1945	173,745	23,769	197,514	19,863	217,377
1946	208,482	21,981	230,463	26,006	256,469
1947	233,352	22,152	255,504	24,004	279,508
1948	229,010	30,344	259,354	20,397	279,751
1949	297,227	47,709	344,936	23,812	368,748
1950	305,438	52,633	358,071	25,156	383,027
1951	354,927	85,706	440,633	24,050	464,683

The table does not show the total capacity of rolling mills in Mexico, because of lack of complete information, but it is estimated that 72% of the capacity was used.

The output of iron and steel products is quite considerable, and the manufacture of many products not as yet made in the country is under study. A great variety of products such as high calibre heavy sections, commercial sections of low domestic consumption, and special steel products, have not yet been attempted.

The domestic production of many items has only recently been commenced:

1. Steel sheet. Production was commenced in 1940, and although only 2,600 tons in 1942, it had risen to 90,809 tons by 1951 (see Figure 7).

2. Steel plate. Is manufactured only by Altos Hornos de Mexico S.A. The initial production was 32,548 tons in 1945 rising to a maximum of 46,226 in 1947. There has since been a decrease, offset by an increase in the amount of sheet and tinplate (see Figure 6).

3. Tinplate. Again, Altos Hornos de Mexico is the only producer. Starting with an output of 2,373 tons in 1946, production rose to 13,668 in 1951 (see Figure 8).

4. Pipe. The manufacture of pipe has been developed on a large scale in recent years (see Figure 9). The figures in Appendix 1, however, only include centrifugal cast pipe production, since under the headings of plate and sheet no deduction is made for the amounts destined for the manufacture of pipe, neither have the total import figures for flat strip been separated. Table 5 shows in detail the production of iron pipe and steel welded pipe:

Table 5

	Tons				
	Welded pipe				
Year	*Large dia.*	*Small dia.*	*Conduit*	*Centrifugal cast iron*	*Total iron and steel*
1944	894	—	—	—	894
1945	904	—	—	8,185	9,089
1946	1,615	—	775	14,837	17,227
1947	7,889	414	895	13,639	22,837
1948	4,813	2,565	1,186	9,856	18,420
1949	21,890	5,083	1,695	10,998	39,666
1950	22,544	6,242	1,875	12,445	43,106
1951	15,863	11,400	1,900	11,700	40,863

The steel welded pipe of large diameter requires steel plate as raw material, especially those of 3/16 in., 4/16 in. and 5/16 in. thickness, which cover 80% of the consumption; the greater part of the steel plate is domestic. The low production in 1951 was due, mainly, to a scarcity of plate.

The small diameter welded steel pipe requires steel sheet or strip, which are also used for the conduit pipe. The waste obtained in the domestic production is 10%, owing to defects in the edges and points; on the other hand, imported strip gives a waste of about 3%. The plate used for large diameter pipe gives waste of from 1 to 2%.

The increase in pipe production is considerable, as the total rose from 894 tons in 1944 to 40,863 tons in 1951.

The utilization of the capacity is relatively low, being only some 16% average. The highest is that for cast iron pipe and conduit pipe, followed by the large diameter pipe, and the lowest for small diameter pipe.

5. Heavy or structural sections are manufactured only by the Fundidora de Fierro y Acero de Monterrey, S.A. The volume produced in 1942 was 24,306 tons and in 1951 it was 48,748. The same rolling mills used in the manufacture of heavy sections are used for the manufacture of rails. The annual production volume of the last item amounts to about 20,000 tons, including accessories, indicating that for economic reasons, the rolling capacity is used to a greater degree to produce heavy sections rather than increasing rails production (see Figure 2).

6. The group including wire, bars and commercial sections is of great interest. Various rolling plants produced these materials, and in 1942, 77,354 tons were produced, increasing to 104,589 tons in 1951 (see Figure 5). Two of the larger plants, Fundidora de Fierro y Acero de Monterrey, S.A., and Consolidada, S.A., manufactured 75,003 tons in 1942 and 101,219 tons in 1951, that is, 97% of the total output.

However, there are several small plants equipped with electric furnaces and rolling mills, or mills only, which make or purchase steel ingots and make use of scrap railway axles and rails for rolling into bars and commercial sections.

It is estimated that 60% of the production of electric steel is used for castings, mill balls, straps, springs and other special steel products. The remaining 40% is used for the manufacture of bars and commercial sections. Corrugated bars and other light sections are made using scrap railway axles and rails as raw material.

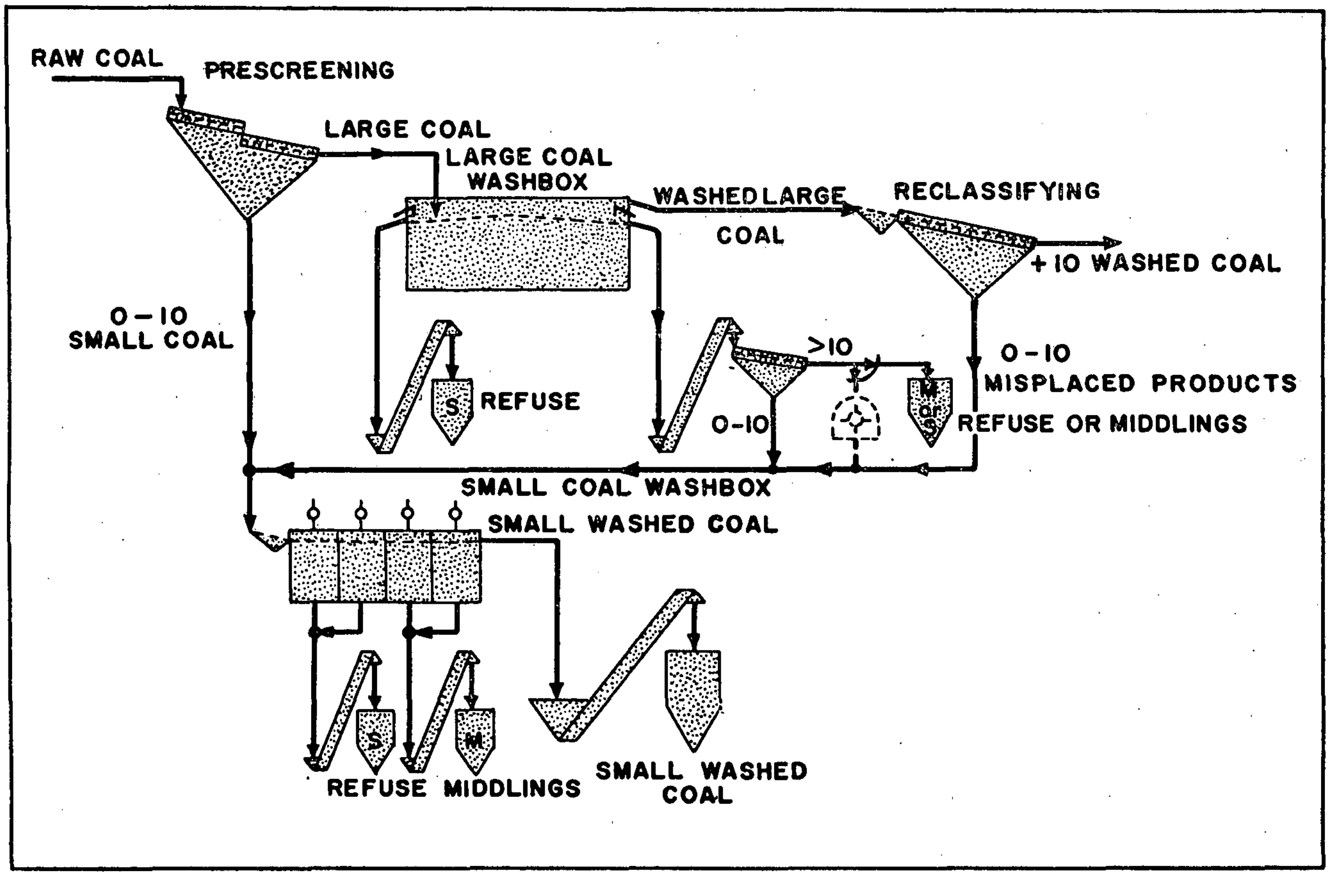

Fig. 1. Separated Washing for Large and Small Coal.

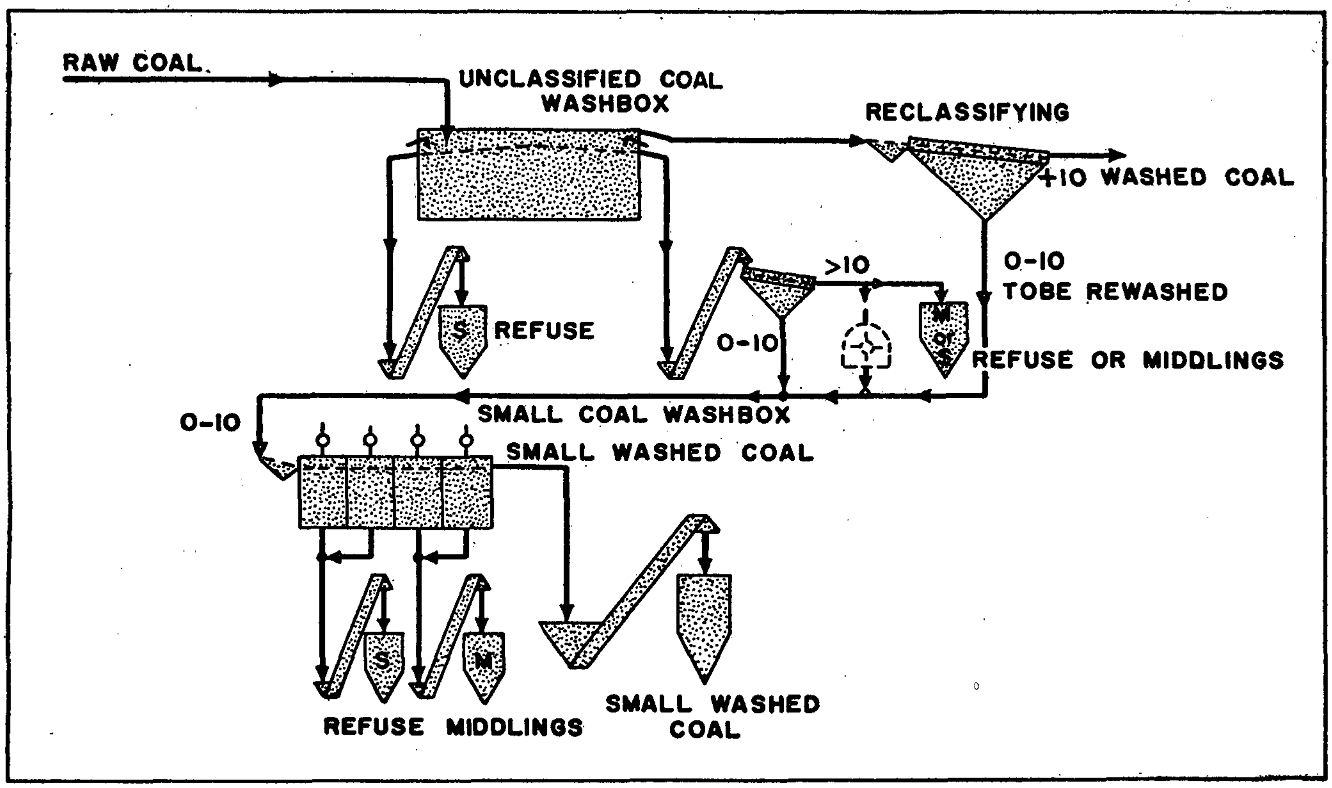

Fig. 2. Unclassified Coal Washing with Small Coal Rewashing.

Based on the foregoing information, it is estimated that the 1951 domestic production of wire, bars, light sections was 203,876 tons; the volume produced by the two steel plants just mentioned therefore represented 49.7% of the total for that year.

7. Cast iron wheels are produced only by Fundidora de Fierro y Acero de Monterrey, S.A., and are used for railway freight wagons only. The average annual volume produced is 5,225 tons (see Figure 4).

8. Balls for grinding mills are manufactured by a few firms. The Fundidora de Fierro y Acero de Monterrey, S.A., is the major producer, but it is impossible to determine the amount produced by other firms, for lack of correct information.

9. The cast iron pieces produced are used mainly to meet the needs of the same plants which manufacture them.

III. IMPORTS

The volume of iron and steel products imported into Mexico is quite considerable, and continues to rise in spite of the increase in domestic production, which remains insufficient to meet the demand. The figures given in Table 6 below refer only to the requirements of the steel industry, and do not include considerable imports of finished products, such as industrial machinery, locomotives, turbines, motors, and a great variety of other products which require an advanced manufacturing process. Certain of these items will soon be manufactured in the country—a plant for railway wagons is being installed, and another for diesel motors—but the majority will continue to be imported.

Table 6

	Total				*Total*
Year	*Value in thousands of pesos*	*Amount in tons*	*Scrap tons*	*Steel and alloy ingots*	*Minus scrap and ingots*
1942........	36,698	82,236	13,779	1,961	67,496
1943........	54,919	128,996	17,118	1,859	110,019
1944........	94,819	235,415	13,976	4,035	217,404
1945........	104,253	250,277	22,088	3,728	224,461
1946........	130,389	309,433	45,887	1,617	261,929
1947........	180,749	347,472	39,255	2,333	305,884
1948........	136,281	214,064	40,288	5,120	168,656
1949........	236,715	290,982	119,733	8,446	162,803
1950........	308,441	393,760	128,187	19,063	246,510
1951........	537,021	573,028	171,474	11,471	390,083

The value of foreign purchases of steel products represents 8% of the total value of imports for 1951, a figure which well illustrates the importance of the imports of iron and steel products in Mexico's economy.

The structure of the imports has been radically changed, however, by the development of the domestic steel industry. In 1942, 82% of the total imports were manufactured products, whilst only 18% represented scrap and ingots. By 1951, the value of manufactured goods had fallen to 68%, and that of scrap and ingots had risen to 32%.

Two of the most important imported items are scrap and steel, and ferrous alloy ingots.

1. Scrap. Is used in electric and open hearth furnaces for the manufacture of steel. The Consolidada S.A. steel plant, for instance, is totally supplied by scrap, and produced 20,430 tons of steel ingots in 1942 and 70,400 tons in 1951. Not all the scrap, however, is used for steel manufacture, since it includes rails and axles, which are used directly in some rolling mills.

It is estimated that scrap collected in the country outside of the steel plants amounts to about 60,000 tons annually, 20,000 tons of which are scrap rails and axles, and the rhythm of industrial development may accelerate the supply of larger quantities of scrap.

2. Steel and ferrous alloy ingots. Any failure to import these raw materials would have had immediate repercussions on output, reducing it by about 146,000 tons in 1951. The domestic output of finished products would have been reduced to 294,000 tons, thus creating a critical domestic situation, since a large number of steel plants would have been affected, and many small rolling mills would have ceased operations completely.

3. Wire imports for 1942 amounted to 1,916 tons, as against 27,881 tons in 1946; there was a drop to 11,716 tons in 1950, with a slight rise to 16,802 tons in 1951. These fluctuations are due to increased domestic production, which allowed for a reduction of imports after 1947. Imports after that year consisted in the main of types of wire not manufactured in the country.

4. Flat strip. Imports increased from 337 to 5,535 tons between 1942 and 1951, mainly for use by pipe manufacturers, who have no adequate domestic source of raw material.

5. Imports of merchant bars and sections were amongst the heaviest from 1944 to 1948, because of the increasing demand for public and private works. During the years 1949 and 1950, domestic production appeared to meet most of the demand, since imports amounted to only about 14,000 tons but rising demand to complete certain public works brought imports up to the high figure of 58,370 tons in 1951.

6. Tinplate was not produced in the country until 1946. Previous to that year the imports had been steady at about 15,500 tons, and in the first year of domestic production the figure fell to 8,783 tons. Later, however, imports commenced to rise again, to a total of 19,448 tons in 1951 (see Figure 8).

7. Sheet imports were 21,004 tons in 1942, and rose to a maximum of 67,573 tons in 1944, decreasing to 21,627 tons in 1948, and increasing again to 35,947 tons in 1951. The decrease in sheet imports is due partly to domestic plate production, and partly to the larger quantity of home manufacture of sheet from 1945 (see Figure 7).

8. Pipe imports rose from 18,015 tons to 80,254 tons between 1942 and 1951. This great difference was due to the requirement for seamless pipe and boiler tubes, as well as for sizes and diameters of pipe not manufactured in the country (see Figure 9).

9. Heavy section imports showed their highest figures from 1945 to 1947, the latter year being 32,462 tons. These imports were mainly for government

works, railway and road bridges, and some commercial and industrial building. There are many heavy sections which are not manufactured in Mexico, although the volume used is not very great (see Figure 2).

10. Imports of tanks and containers amounted to 850 tons in 1942, and increased to 4,562 in 1951. The manufacture of these items in Mexico has acquired a reasonable volume in recent years, using both domestic and imported sheet and plate, and the imports were mainly for the benefit of new firms, which enjoy freedom from customs duties.

11. Mining materials. A large increase was recorded during the years 1942 to 1951. In 1942, foreign purchases amounted to only 7,350 tons, but by 1951 this had risen to 147,323 tons (see Figure 10).

12. Railway materials. The large purchases from 1942 to 1945 were due to the renovation programme of the Ferrocarriles Nacionales, which was intensified during the 1946–52 administration. Heavy rails (112 lb.) are being imported to replace lighter rails on the main lines, and the lighter rails are then used to replace those of even lesser weight on the branch lines, so that only rails worn beyond use are available as scrap (see Figure 3).

IV. EXPORTS

The export figures include 30 of the most important tariff items, and contrary to those pertaining to imports, include some items of a higher grade of manufacture, since foreign sales are relatively small.

Exports of iron and steel products are low, representing only 0.8% of the total exports in 1951, and slightly more than 1.08% of total production. Table 7 shows the value and volume of exports.

Table 7

Year	Value (thousands of pesos)	Quantity (tons)
1942	1,650.5	1,515.5
1943	5,105.0	2,510.6
1944	11,082.7	10,236.3
1945	12,839.1	10,062.4
1946	8,667.7	18,126.2
1947	11,549.8	20,688.8
1948	9,162.9	8,028.5
1949	7,830.7	4,155.9
1950	8,609.9	5,170.9
1951	6,700.7	4,690.8

V. CONSUMPTION

The iron and steel products consumed in Mexico consist largely of those requiring only limited manufacturing processes, but the growing demand for a wide variety of items, due to the greater capacity of the domestic market, is giving rise to a high consumer potential. This favourable situation for the development of industry allows, within the economic possibilities of the country, for the installation of new plants to fill the gap between domestic production and consumption, and this in turn creates a demand for new types of iron and steel products, diversifying the structure of production and consumption.

Table 8 compares figures for production, net imports and consumption.

Table 8

Year	Production (tons)	Net imports (tons)	Total consumption (tons)	Consumption per capita (kilogrammes)
1942	175,116	65,981	239,097	11.571
1943	170,215	107,669	277,884	13.110
1944	178,632	207,170	385,802	17.731
1945	217,377	214,690	432,067	19.330
1946	256,469	243,803	500,272	21.774
1947	279,508	285,246	564,754	23.901
1948	279,751	160,680	440,431	18.114
1949	368,748	158,664	527,412	21.073
1950	383,027	242,008	625,035	24.250
1951	464,683	387,000	851,683	32.081
Estimate for 1960	1,200,000	370,000	1,570,000	45.300

The figures of this table are illustrated in Figure 1. The greater divergence between consumption and production in 1948 is due to the depreciation of the peso, which affected imports in general.

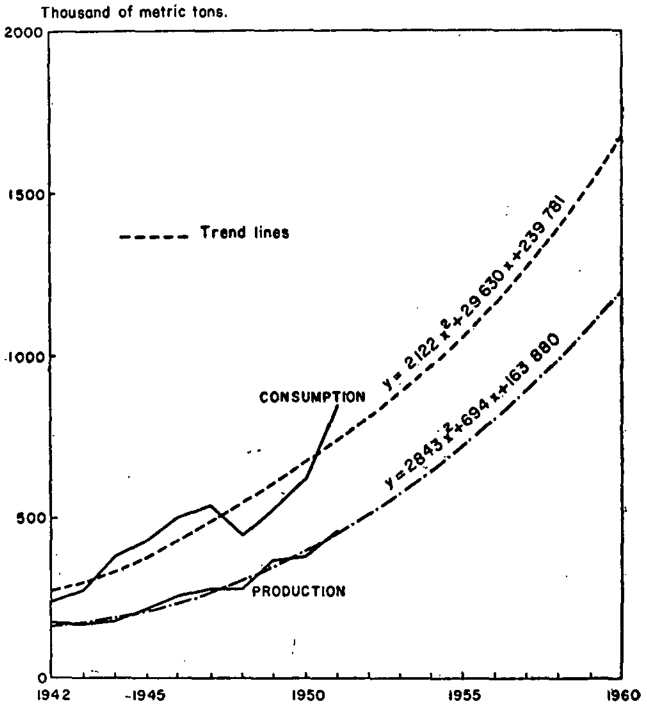

Figure 1

Mexico: steel production and consumption of iron and steel products.

It may be concluded, therefore, that the consumption of iron and steel products is more subject to cyclical movements of the domestic economy than the production, provided that there is a difference to minimize the fluctuations, which in this case is the volume of imports.

In Figure 1, the curves seem to show a certain parallel trend. However, if these lines are drawn to a semi-logarithmic scale, they approach each other towards the end. An analysis of the curves and the data on the production and consumption of iron and steel products, as well as an appraisal of general economic factors, indicates that the estimates made are reasonably accurate.

The relationship of production and net imports to consumption is quite interesting. In the first place, it may be noted that, whilst production for 1942 represented 72.4% of consumption, it tended to decline, until it reached 46.3% in 1944, maintained a level of about 50% from 1945 to 1947, rose to 69.9% in 1949, and fell in subsequent years; however, it is estimated that by 1960 it will reach 76.4%. In the second place, the adjusted net imports complement production as regards consumption, and therefore, the percentages of this item move in the opposite direction to those of production. These percentages are as follows:

Table 9

Year	*Consumption*	*Production*	*Net imports (26)*		*Estimated production, without imported scrap or ingot*
			Adjusted	*Not adjusted*	
1942.....	100.0	72.4	27.6	33.8	66.2
1943.....	100.0	61.3	38.7	45.6	54.4
1944.....	100.0	46.3	53.7	58.4	41.6
1945.....	100.0	50.3	49.7	55.7	44.3
1946.....	100.0	51.3	48.7	58.2	41.8
1947.....	100.0	49.5	50.5	57.9	42.1
1948.....	100.0	63.5	36.5	46.8	53.2
1949.....	100.0	69.9	30.1	54.4	45.6
1950.....	100.0	61.3	38.7	62.3	37.7
1951.....	100.0	54.6	45.4	66.9	33.1
1960.....	100.0	76.4	23.6	—	—

A column for non-adjusted net imports was included in the above table; this percentage is quite high in relation to consumption although it does not reach the level of production.

Consequently, the output of finished products, not including scrap and steel and ferrous alloy ingots, would be, in consumption percentages, the difference in the preceding column.

Such a situation could arise if foreign supplies of these materials were to be suspended, given the relative scarcity of scrap prevalent in the majority of the industrial nations.

1. Heavy sections, rails and rail accessories. Heavy sections production amounted to 95.4% of consumption in 1942, and to 79.4% in 1951. Although this production covered the major part of consumption from 1945 to 1948, net imports were fairly high. The increase in production in the last two years of the period was not sufficient to satisfy consumption requirements; nevertheless, net import figures were lower than those registered from 1945 to 1948. The increase in the consumption of this item is a consequence of the higher demand for sections for industrial plants and buildings, for private enterprises, and for government-owned railway and road bridges, brought about by the economic development of the country.

Rails and accessories show an opposite trend, as production for 1942 was 96.6% of consumption whereas in 1951 it represented only 14%. Although consumption has increased abnormally, due to the railroad renovation programme, domestic rail manufacturers did not increase their production, as they prefer to use their steel supplies, and their limited

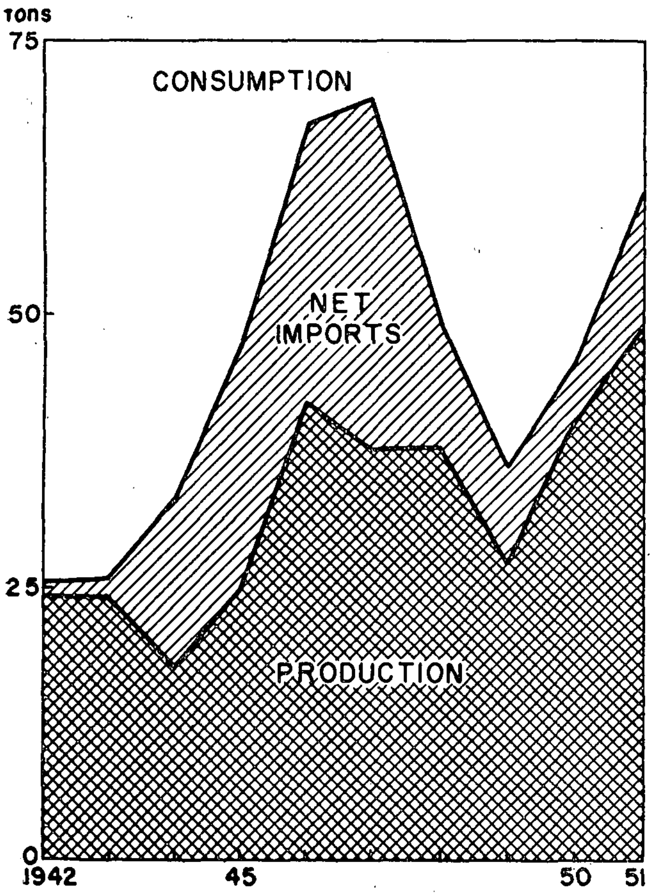

Figure 2
Heavy sections.

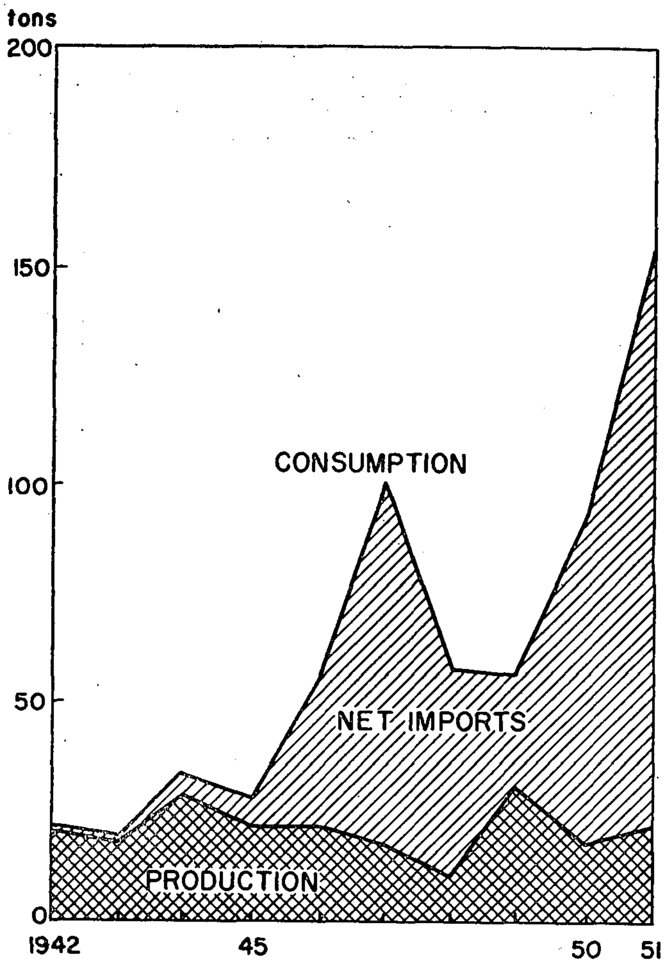

Figure 3
Rails and accessories.

rolling installations, for the manufacture of structural sections which yield a greater profit. Under these conditions, it is preferable to import rails and rail accessories, because the outflow of foreign exchange is reduced and the rolling mills are used to better advantage, both factors being beneficial to Mexico's economy.

2. Railway wheels, axles, wire, bars and light sections. Only cast iron wheels are produced, and these are used in railway freight wagons. The volume is quite regular; in 1942 it represented 49% of consumption, and in 1951, 35%. Net imports comprise steel wheels and axles for locomotives and passenger wagons, and registered sharp downward trends in 1944 and 1948, because of the economic conditions of those years.

Wire, bars and light sections are the most important items by volume of production and consumption. Production for 1942 was 97.9% of consumption, and in 1951, 73.3%, although domestic production was double that of 1942. This unusual consumption is due to the demand for a great variety of products included in those categories, especially corrugated bars, which are used in large quantities in the building industry in general. The remaining products are used in secondary industries, such as the manufacture of windows, structures, etc. Net imports were relatively high during 1944, 1945, 1946, 1947, and 1951, due to the increase in consumption, domestic production being quite inadequate to meet the demand.

3. Plate was not being produced in Mexico in 1942, and when its manufacture was commenced in 1945, it represented 47.6% of consumption and, consequently, the major net imports were registered in 1945, 1946 and 1947. Plate is used in fair quantities for the manufacture of large-diameter welded pipe, tanks and structures.

Sheet production amounted to 27.3% of consumption in 1942; by 1951 consumption increased twelvefold and, although production also rose, it met only 75.9% of consumption. The constant rise in the consumption and production of sheet is due to the variety of uses to which it is applied, such as: the manufacture of refrigerators, washing machines, gas stoves, kitchen cabinets, office equipment, etc. With the exception of 1944, net imports maintained a close relation to production tendencies.

4. Tinplate and tubes. Production of tinplate was commenced in 1946; output for that year amounted to 21.3% of consumption, and in 1951, to 41.3%. Net imports were more or less constant, but decreased in 1946, when domestic production commenced, and decreased again in 1948 due to the depreciation of the Mexican peso and other external factors. Although the production increase was very slight, larger quantities were imported to meet the requirements of the expansion of the canning industry.

5. Mining material, castings and other products. The production of mining material in 1942 represented

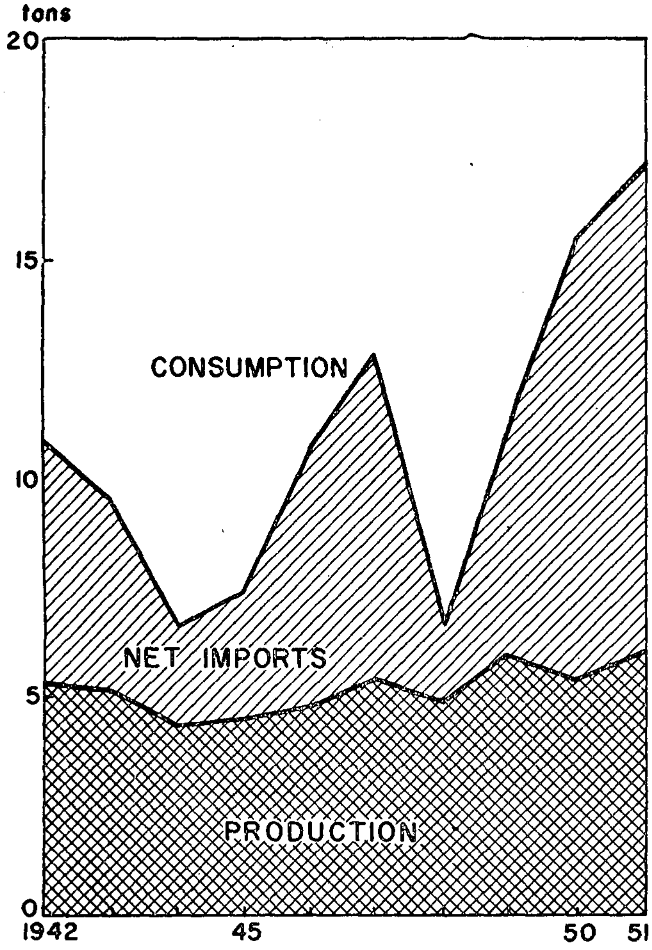

Figure 4
Railway wheels and axles.

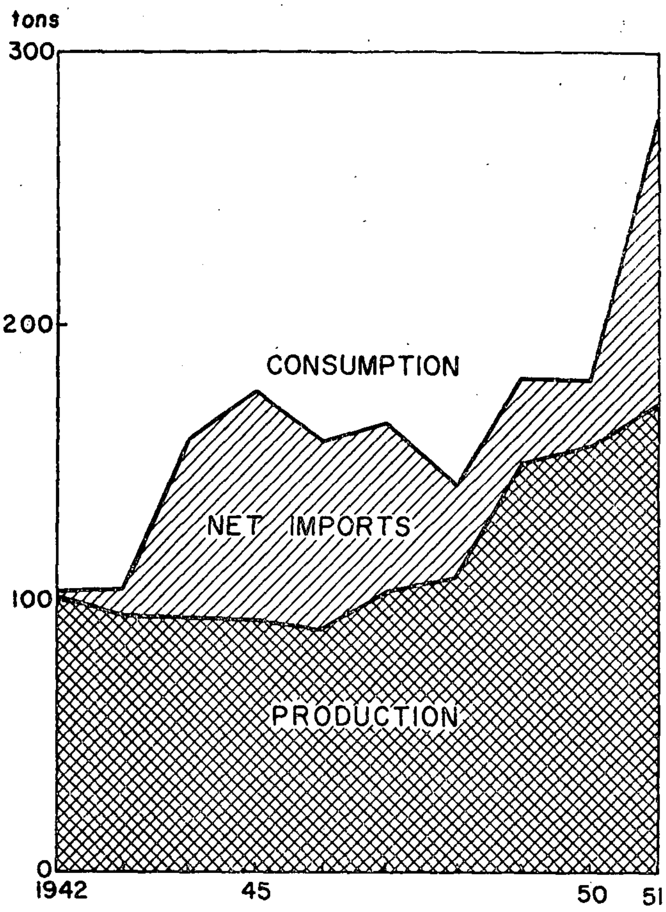

Figure 5
Wire, bars and light sections.

94% of the total consumption, as against 20.8% in 1951. There is a marked decrease in production and an increase in net imports. Consumption is more or less stable, since there has been no increase in mining activities, the volume of minerals remaining constant.

VI. Markets

An analysis of the information on production and consumption shows no indication of the structure of the market which could serve as a guide to manufacturers of iron and steel products in channelling production along practical lines, thus avoiding the accumulation of large stocks, and the consequent economic problems.

In 1942, however, the Banco de Mexico S.A. undertook a direct research in all secondary iron and steel plants in the country. A summary of a part of the data obtained is given in Table 10 below:

Table 10

ESTIMATE OF CONSUMPTION OF FINISHED PRODUCTS MANUFACTURED BY SECONDARY INDUSTRIES IN 1949

	Tons					
Consuming centre	*Sheets*	*Tinplate*	*Plates*	*Wire and wire rod*	*Bars and merchant sections*	*Structural sections*
Federal district	6,501	11,097	1,350	15,750	3,720	4,793
Tlanepantla	7,219	—	12,669	8,899	443	5,000
Monterrey	19,573	4,019	165	2,004	1,277	1,572
Puebla	109	—	—	—	325	—
Saltillo	4,527	—	140	48	1,045	185
Guadalajara	1,848	1,816	—	1,808	949	—
Chihuahua	171	—	806	253	—	35
Tampico	1,403	—	—	632	1,541	—
Veracruz	1,912	25	—	—	121	—
Other cities	1,312	95	124	447	907	4
TOTAL	64,574	17,052	15,268	29,856	10,328	11,589

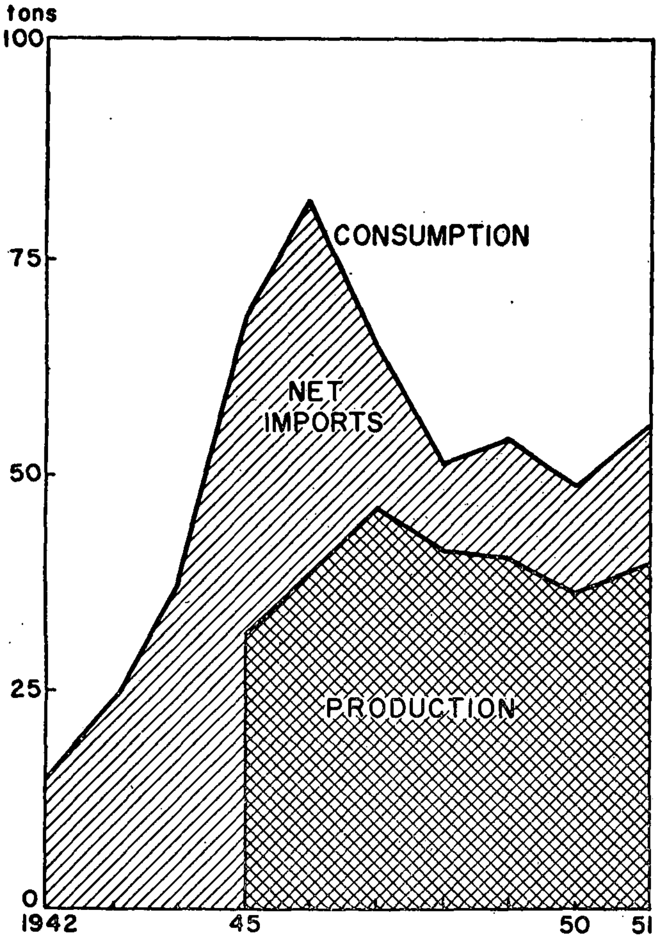

Figure 6
Plate.

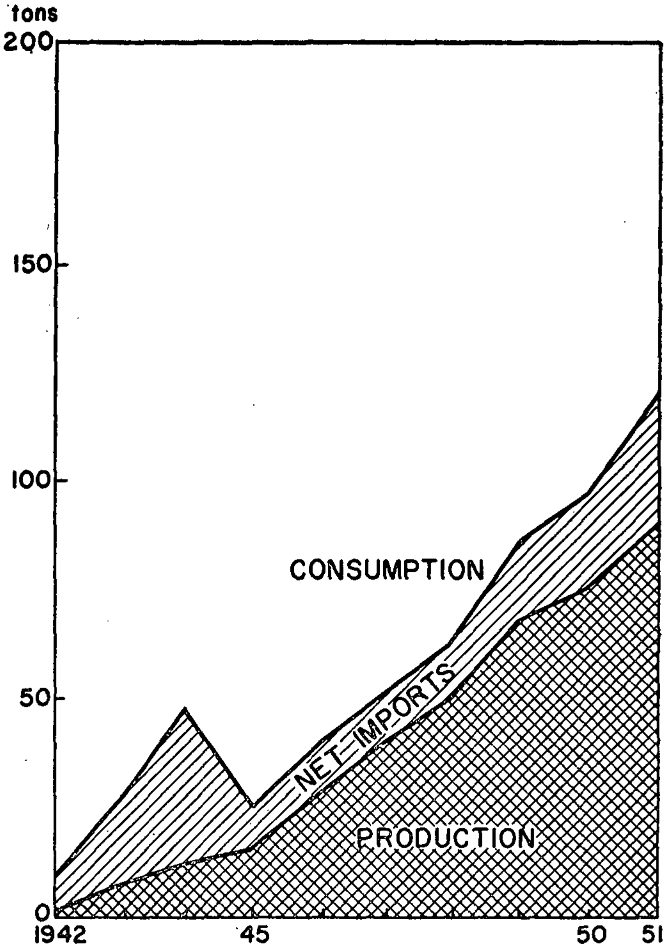

Figure 7
Sheet.

There are two principal groups of consumers in the iron and steel market; the first is the Government and the decentralized agencies, and the second is formed by the transforming industries together with the construction and mining industries. With this general classification in mind, the following table was prepared, showing the consumption of these two groups, by products.

Table 11

Groups of products	*Government and decentralized agencies*	*Industries*
Rails and parts	98.0	2.0
Railway wheels and axles	97.5	2.5
Structural profiles and structures	24.5	75.5
Plates	45.0	55.0
Sheets	4.0	96.0
Tinplate	1.0	99.0
Cast iron pipe	95.0	5.0
Welded steel pipe	60.0	40.0
Seamless steel pipe	90.0	10.0
Wire, bars, and merchant sections	40.0	60.0
Mining material	2.0	98.0
Castings and other materials	30.0	70.0

The State Railways (Ferrocarriles Nacionales de Mexico) is responsible for the high percentages of railway materials on the Government side and Petróleos Mexicanos S.A. is the chief purchaser of pipe and plate.

Apart from the known data illustrated by the two foregoing tables, there is a definite need for market research, in order that the capacity of the steel mills may be used to better advantage.

VII. Factors affecting production and consumption

A. The factors affecting domestic production are:

1. Degree of yield obtained from the raw materials of the country, iron ore and coking coal.

Proved reserves of iron ores are some 300 million tons, which would yield about 180 million tons of pig iron. Coal reserves are estimated at 2,000 million tons, 500 million tons of which could be converted into coke.

The chief handicap in the steady supply of both ore and coke to the industry is the lack of adequate transport facilities.

2. The degree of utilization of the installed capacity of the iron and steel industry.

In 1951, the utilization of the blast furnaces was 51.1% but this will tend to increase as the transportation system improves the deliveries of raw material. Altos Hornos de Mexico proposes to install a new blast furnace which will increase the plant capacity to 630,000 tons of pig iron by 1955, and to 777,000 tons in 1960. The shortage of scrap will be overcome by the use of a larger proportion of pig iron in the open hearth furnace.

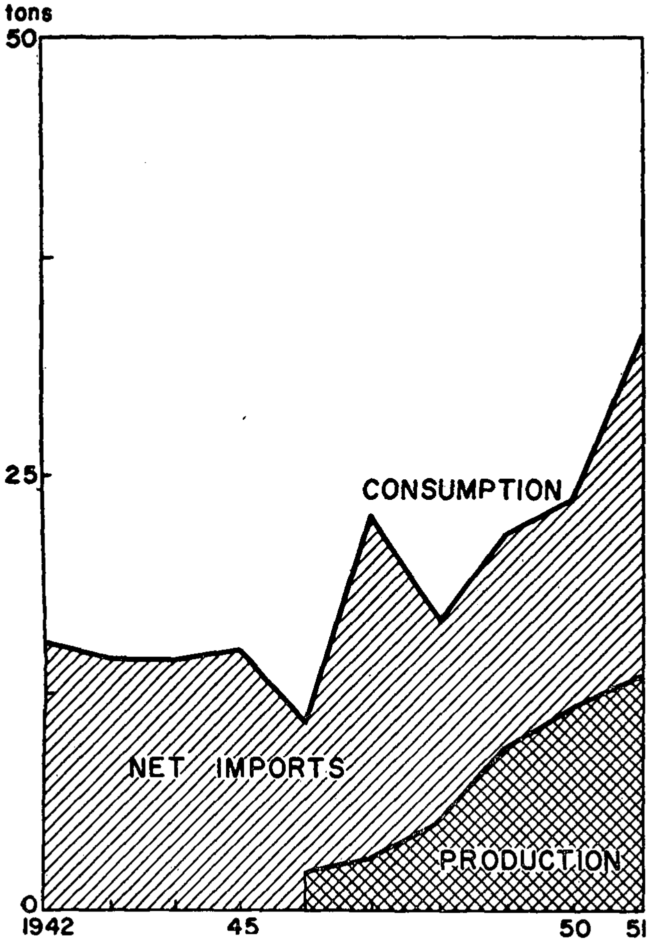

Figure 8
Tinplate.

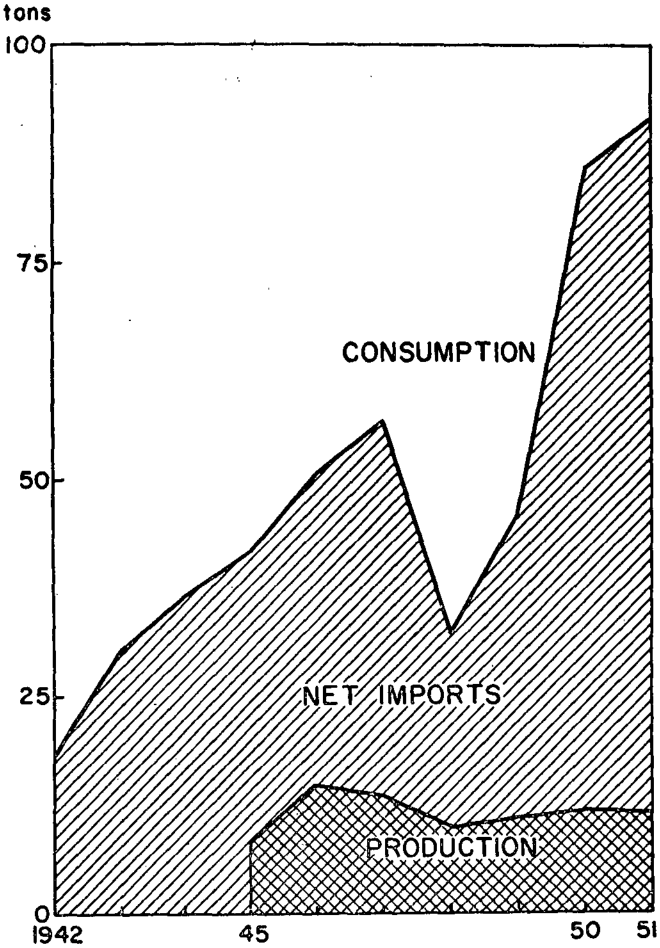

Figure 9
Pipe.

It must also be remembered that by 1960 other methods will have been developed for the reduction of the iron ore, and that the present sponge iron installations will have been enlarged. Should present plans mature, installed capacity might well increase to 400,000 tons. Although the costs per ton of reducing iron ore by other systems is higher than blast furnace operation, in the view of the scarcity of pig iron and scrap, these higher costs are justified.

Amongst the expansions of steel plants in process or planned are: Altos Hornos is building a fourth open hearth, and increasing the capacity of two others.

Fundidora de Fierro y Acero de Monterrey will install a sixth open hearth.

Consolidada, S.A. is considering the increase of one of its open hearths.

The implementation of these projects will increase the maximum yearly capacity to 798,000 tons of steel ingots, and a possible installation of new electric furnaces would bring this total up to one million tons by 1960.

Hot rolling mills, present and projected, will give a capacity of 1,200,000 tons by 1960.

B. Foreign supplies, especially scrap, are becoming critical, and a sharp decrease in imports may affect consumption.

C. The price structure of iron and steel products is interesting, but will only be briefly mentioned, without detailed analysis.

1. Economic location of steel plants. Although the plants are close to the raw material supplies, they are distant from the principal markets to the south.

2. There is not sufficient information available to analyse the economic efficiency of production.

3. The Mexican steel industry enjoys high tariff protection. Only a detailed study would show whether a decreased tariff protection would result in increased efficiency of the plants.

D. The public works programmes have a very considerable influence upon the demand, as have also the requirements of new industries.

E. Finally there is the effect of the normal growth of the established industries, in keeping with the domestic, and possibly the foreign, markets.

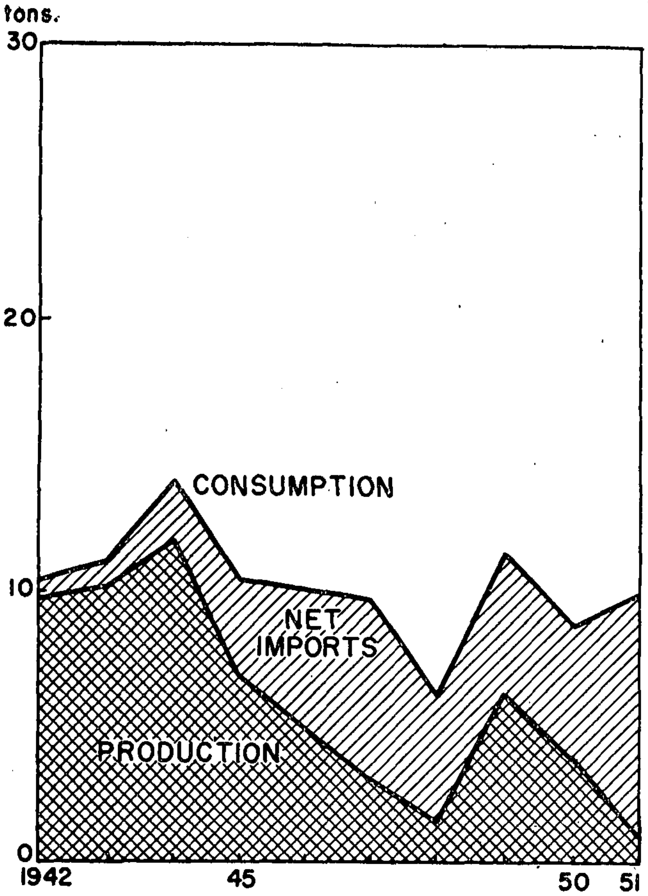

Figure 10
Mining Material.

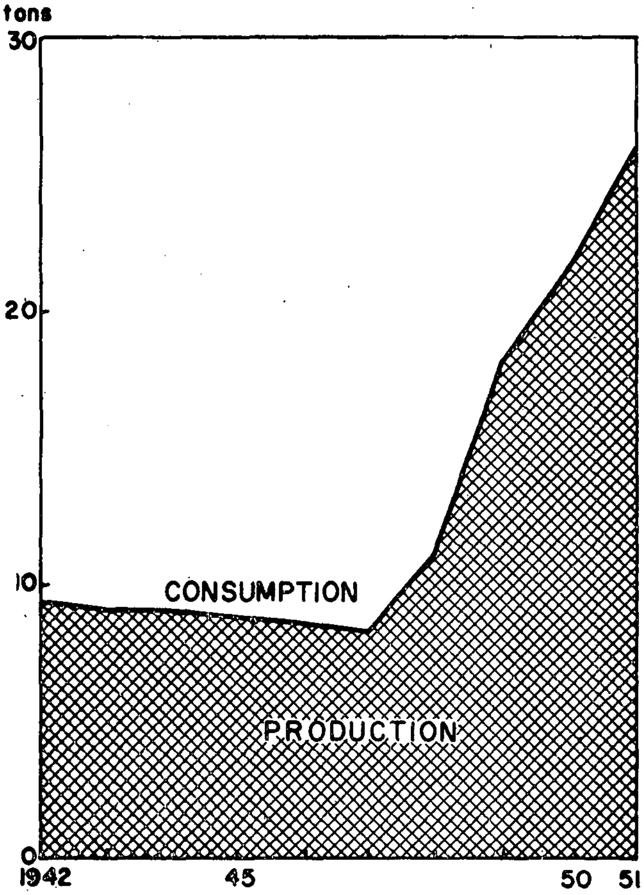

Figure 11
Castings and other products.

Appendix 1

ESTIMATED PRODUCTION OF FINISHED AND SEMI-FINISHED IRON AND STEEL PRODUCTS

(metric tons)

								Other steel products			Iron cast into				
Year	*Heavy sections* 1	*Rails and accessories* 2	*Wire, bars and merchant sections* 3	*Plates* 4	*Sheets* 5	*Tinplate* 6	*Bars of chrome-alloy steel* 7	*Steel ingots and castings* 8	*Used rolled wheels and axles* 9	*Total steel products* 10	*Wheels* 11	*Balls* 12	*Miscellaneous castings* 13	*Pipe* 14	*Total iron and steel* 15
1942......	24,306	20,577	77,354	—	2,600	—	5,943	7,234	20,850	158,864	5,357	3,815	5,080	—	173,116
1943......	23,756	18,040	70,949	—	8,410	—	6,703	7,810	21,501	157,169	5,143	3,487	4,416	—	170,215
1944......	17,630	29,052	68,068	—	12,690	—	7,082	6,646	23,270	164,438	4,382	4,780	5,032	—	178,632
1945......	24,604	22,115	66,925	32,548	16,303	—	4,864	6,386	23,769	197,514	4,555	2,071	5,052	8,185	217,377
1946......	42,030	21,642	65,066	38,219	29,132	2,373	3,185	6,835	21,981	230,463	4,826	1,775	4,568	14,837	256,469
1947......	37,571	18,314	77,122	46,226	41,246	3,169	2,203	7,501	22,152	255,504	5,496	972	3,897	13,639	279,508
1948......	37,682	9,824	74,428	41,131	50,012	5,112	1,222	9,599	30,344	259,354	4,954	183	5,404	9,856	279,751
1949......	26,521	30,566	94,526	40,587	68,916	9,557	4,923	21,631	47,709	344,936	6,019	1,384	5,411	10,998	368,748
1950......	40,133	17,962	93,199	36,424	76,443	11,785	2,233	27,259	52,633	358,071	5,459	1,699	5,353	12,445	383,027
1951......	48,748	22,183	104,589	39,324	90,809	13,668	1,654	33,952	85,706	440,633	6,066	434	5,850	11,700	464,683

NOTES TO APPENDIX 1

Figures relate to steel plants possessing blast furnaces and electric or open-hearth steel furnaces. Data are also included relating to enterprises utilizing ingot iron or steel produced in the country or scrap which can be rolled directly.

The production figures appearing in columns 1, 2, 3, 4, 5, 6 and 7 correspond to the following enterprises: Fundidora de Fierro y Acero de Monterrey, S.A., La Consolidada, S.A., Altos Hornos de México, S.A., Hojalata y Lámina, S.A., Herramientas México, S.A., and Fundidora y Laminadora "Chapultepec", S.C.L.

Column 8 registers the production of electric steel of the following enterprises: Aceros Tlanepantla, S.A., Aceros Topeyac, S.A., The Tezuitlan Copper Co., S.A., El Rosario, S.A., La Consolidada, S.A., Fundición de Hierro y Acero, S.A., and Fundidora de Fierro y Acero de Monterrey, S.A. In this last enterprise the production of open-hearth steel ingots for the market is included.

Column 9 includes used rails and axles, domestic and imported, which are rolled directly, as well as imports of steel ingots and ferro-alloys which are bought in order to be rolled in the country (commercial bars and sections are produced preferably).

Columns 11, 12, 13 and 14 correspond to the production of cast iron of Altos Hornos S.A., and Fundidora de Fierro y Acero de Monterrey, S.A.

Appendix 2

IMPORTS OF THE MAIN IRON AND STEEL PRODUCTS

(metric tons)

Year	*Ferro-alloys* 1	*Ingots* 2	*Scrap* 3	*Wire* 4	*Strip and flats* 5	*Bars of all shapes* 6	*Tinplate* 7	*Tubes* 8	*Sections* 9	*Sheets* 10	*Tanks and containers* 11	*Railway material* 12	*Railway material* 13	*Total* 14	*Total 14 − 3* 15	*Total 14 − 1, 2 and 3* 16
1942........	957	4	13,779	1,916	377	735	15,512	18,015	1,161	21,004	1,850	6,301	625	82,236	68,559	67,426
1943........	1,859	17	17,118	5,744	748	3,299	14,506	30,092	2,119	45,184	1,177	6,211	939	128,996	111,895	110,019
1944........	1,143	2,892	13,976	23,118	1,066	52,526	14,603	36,700	10,443	67,573	1,637	7,463	2,275	235,415	221,492	217,404
1945........	1,878	1,850	22,088	19,481	371	69,159	15,118	33,660	23,188	47,588	2,814	9,438	3,644	250,277	229,172	224,461
1946........	1,226	391	45,887	27,881	1,299	58,076	8,783	36,254	25,466	52,558	5,053	41,356	5,203	309,433	264,405	261,929
1947........	2,258	75	39,225	21,213	1,141	54,092	19,730	49,773	32,462	25,520	4,584	91,087	6,682	347,472	308,247	305,874
1948........	2,218	2,902	40,288	12,479	8,224	23,671	11,701	27,820	11,800	21,627	3,633	49,816	5,285	214,064	181,586	168,256
1949........	4,065	4,381	119,733	17,640	1,895	14,765	12,634	36,895	10,011	26,282	5,557	31,807	5,317	290,982	171,249	162,803
1950........	4,422	14,641	128,187	11,716	3,382	14,065	12,051	74,347	5,904	30,235	3,002	86,904	4,904	393,760	287,301	246,510
1951........	3,187	8,284	171,474	16,802	5,535	58,370	19,448	80,254	13,874	35,947	4,562	147,323	7,968	573,028	401,554	390,083

Source: Dirección General de Estadística.

Notes:

The data of this table correspond to 40 customs items.

Columns 6 and 7 correspond to one item each, columns 1, 5 and 13 to two items each, columns 2, 9 and 12 to three items each, columns 4 and 11 to four items each, and columns 3, 8, 10 and 5 to five items each.

Total imports were calculated excluding the figures of columns 1, 2 and 3, because scrap is incorporated into the production of steel ingots and a part of it (used rails and axles), with ingots of columns 1 and 2, is rolled directly within the country; this production is accounted for in the national production figures.

Appendix 3

EXPORTS OF PRINCIPAL IRON AND STEEL PRODUCTS

(metric tons)

Year	*Scrap* 1	*Ingots* 2	*Wire, cables and bars* 3	*Sheet and tinplate* 4	*Strip, chains and screws* 5	*Tubes* 6	*Heavy sections* 7	*Railway material* 8	*Tanks, boats and containers* 9	*Furniture and sanitary implements* 10	*Unspecified goods* 11	*Hand tools* 12	*Total* 13	*Total, excluding scrap* 14
1942	0.3	—	12.4	1.6	132.0	8.4	[a]	—	985.2	25.1	347.7	2.8	1,515.5	1,515.2
1943	161.1	—	2.1	12.6	528.5	8.4	—	17.2	1,203.2	62.2	490.0	25.3	2,510.6	2,349.5
1944	2.7	7,815.6	2.0	683.4	128.3	4.5	4.7	[a]	1,158.8	22.4	360.0	53.9	10,236.3	10,233.6
1945	290.9	2,916.3	1.5	4,372.0	209.3	49.6	—	—	661.1	407.2	940.0	214.5	10,062.4	9,771.5
1946	[a]	15,381.2	0.9	318.1	61.6	138.4	71.2	98.3	1,001.3	287.6	638.2	129.4	18,126.2	18,126.2
1947	50.7	11,497.4	7.5	68.2	217.9	6,450.5	34.7	115.8	934.7	582.2	635.0	94.2	20,688.8	20,638.1
1948	52.5	—	11.4	885.0	347.2	5,213.2	10.9	—	814.9	208.0	413.0	72.4	8,028.5	7,976.0
1949	16.8	[a]	2.5	0.2	21.9	2,146.0	256.3	—	925.7	221.7	465.3	99.5	4,155.9	4,139.1
1950	668.9	3.7	59.4	60.8	9.4	341.8	40.6	1,041.2	1,090.0	764.7	992.9	97.5	5,170.9	4,501.9
1951	1,608.0	494.4	2.5	0.2	10.7	39.5	1,232.6	—	573.3	370.1	326.0	33.5	4,690.8	3,082.8

Source: Dirección Nacional de Estadística.

Notes:

The data of this table correspond to 30 customs items.

Columns 1 and 6 correspond to one item each; columns 2, 7, 10 and 12 to two items each, columns 3, 4, 5 and 8 to three items each, and columns 9 and 11 to four items each.

[a] Less than 50 kg.

Appendix 4

CONSUMPTION OF IRON AND STEEL PRODUCTS

(metric tons)

Year	*Heavy sections* 1	*Rails and accessories* 2	*Railway rails and axles* 3	*Plates* 4	*Sheets* 5	*Tinplate* 6	*Tubes* 7	*Wire, bars and merchant sections* 8	*Mining material* 9	*Castings and other products* 10	*Total*
1942	25,467	21,299	10,936	15,285	9,534	15,512	18,007	103,254	10,383	9,420	239,097
1943	25,875	19,846	9,531	24,812	29,429	14,506	30,084	104,070	11,129	9,102	277,884
1944	28,068	34,206	6,691	32,134	48,968	14,603	36,694	161,281	14,137	9,020	385,802
1945	47,792	28,531	7,577	63,319	25,865	15,118	41,795	177,607	10,579	8,884	432,067
1946	67,425	56,997	10,729	82,995	41,659	11,156	50,953	159,526	10,163	8,669	500,272
1947	69,998	101,810	12,971	65,949	51,183	22,899	56,561	165,128	9,857	8,398	564,754
1948	49,471	57,930	6,664	51,803	63,506	16,803	32,473	143,918	6,690	11,163	440,431
1949	36,276	57,114	11,278	54,603	87,486	22,191	45,747	182,703	11,624	18,390	5,527,412
1950	45,996	93,651	15,633	49,933	97,637	23,836	86,450	181,355	8,836	21,708	625,035
1951	61,389	158,223	17,349	55,669	119,565	33,116	91,914	278,181	10,056	26,221	851,683

Notes:

In order to estimate the consumption by groups of articles, the figures of appendices 1, 2 and 3 were used in the following manner:

Column 1 includes column 1 of the production table plus column 9 of the imports table minus column 7 of the export table.

Column 2 includes 2 of production plus 12 of imports minus item 9.56.72 (axles and wheels) minus 8 of exports.

Column 3 includes 11 of production plus item 9.56.72 (axles and wheels).

Column 4 includes 4 of production plus item 3.53.63 of column 10 of imports plus 11 of imports minus 9 of exports.

Column 4 includes 5 of production plus 5 and 10 of imports minus item 3.53.63 minus 4 and 10 of exports.

Column 6 includes 6 of production plus 7 of imports.

Column 7 includes 14 of production plus 8 of imports minus 6 of exports.

Column 9 includes 7 and 12 of production plus 13 of imports.

Column 10 includes 13 of production and 60% of column 8 of production.

Summary of Discussion

Thomas Steel Rails in France, *presented by the author*

Mr. Palmé began his presentation by explaining that the use of Thomas converters was interesting in Latin America, where there were large reserves of phosphorus ore, and he therefore wished to comment briefly on the use of Thomas steel in France for the manufacture of rails.

Mr. Allard opened the general discussion of the paper; in support of Mr. Palmé's statement, he pointed out that in his view the advantage of the Thomas converter was that oxygen was not dissolved in very soft steel, and it was therefore possible to blow very moist air, or even steam, into a converter. There was no danger of hydrogenation of the metal, and with high carbon content rails there was a stage where the metal was very soft and oxidized. It was usually deoxidized through the addition of liquid spiegels in contact with the hydrogenated mass, and therefore the blow was dry. He would like to hear Mr. Coheur's opinion as to the superiority of Thomas over Martin rails.

Mr. Palmé, in reply, said that he had had the opportunity of assisting in the manufacture of rails for foreign countries, but they were Martin rails and showed numerous shatter cracks.

Mr. Mercier did not agree with Mr. Allard's view as to the superiority of Thomas rails. Very few Martin rails were made in France, but the big plants specializing in Martin steel could certainly produce a rail of as good a quality as the Thomas rail.

Mr. Boulger explained that although he had no direct contact with the relative hydrogen contents of Thomas and Martin steels, nevertheless his organization had made several laboratory experiments, from which he suspected that the hydrogen content would be lower in steels made in a converter, as by that process the hydrogen content was reduced to a very low level. Steels with a low hydrogen content would therefore show very few flakes.

Mr. Bulle said that many papers were available in Germany showing the quality of Thomas and Martin rails to be the same, although most rails in Europe were made by the basic Bessemer process. Many open hearth rails were also made in Germany and France, but those were mainly for export. Nevertheless, from the Latin-American point of view, good rails could be obtained with the basic Bessemer process.

Mr. Coheur, in support of the remarks made by Messrs. Allard and Boulger, cited three figures that had been reached by dosing hydrogen in extra mild steel. Those figures were:

Around 4 cu. cm. in open hearth steel
3 cu. cm. in Thomas steel
2.7 cu. cm. in Thomas steel blown with water and oxygen.

Mr. Costa Lino said that rails were being made at Volta Redonda by basic open hearth, and he would be interested to know what procedure had been set up for 0.07% phosphorus in Thomas steel, and if that were a fair figure for phosphorus. He also wished to know whether good basic open hearth steel could be made to the same specifications as those used in France.

Mr. Palmé replied that in so far as specifications were concerned, they were exactly the same for both types of rails. The European International Union of Railways was currently revising the specifications so that there would be a European scale for rails, which would be exactly the same for both steels. The only difference would lie in the commercial analysis. Slightly more phosphorus was admitted in Thomas than in Martin steel. As to the first question, relating to the highest phosphorus content admitted, the figures he had given were only of relative importance, serving merely as indications, since the analysis was never a cause for rejection. It could be said, however, that the average content of phosphorus was about 0.07%, although it was sometimes higher; the impact test permitted up to about 0.08%.

The Problem of Energy in Steel Plants, *presented by the author*

Mr. Périn, in introducing his paper, said that before giving a brief explanation of it, he wished to emphasize one point, which was that the steel industry possessed a very valuable product, i.e., energy. It would be criminal not to try and use that energy for the steel plant's requirements, as it was thermally essential as a complement to the plant.

Mr. Albala opened the general discussion on the paper, and described the way in which the Huachipato plant had tackled the question of having the best possible thermal balance. The plant was using blast furnace gas as much as possible, its uses being the following:

1. Heating of blast furnace stoves;
2. Heating of coke ovens;
3. Consumption in steam boilers;
4. Mixing with coke oven gas to a heating value of 4,000 cal/m^3, for use within the plant and for household uses in neighbouring cities.

Blast furnace gas could be used to advantage in any modern plant, although not all of it could be disposed of; on the other hand, coke oven gas was wholly utilized within the Huachipato steel works and for domestic purposes. In general, the disposal of blast furnace and coke oven gas at the Huachipato plant was quite satisfactory.

Mr. Krebs said that at his firm's Rheinhausen plant there had been the choice between blast furnace and coke oven gas for the reheating furnaces in the rolling mills; blast furnace gas had been selected because

Summary of Discussion

Considerations on Coal Washing in Europe and its Possible Application to Latin-American Coals, *presented by the author*

Mr. CHERADAME began his presentation by explaining that his paper gave the results of work done by the Centre d'études et recherches des charbonnages de France (CERCHAR) which he had the honour of directing.

Some 300 persons worked in the laboratories, there being fifty engineers in the coal washing department, while work was also done at the collieries. His paper dealt with the results of four years' work by this team specializing in the study of coal preparation and was not, therefore, a bibliographic document.

He stressed that the results of that research were valid for all countries and for all coals. Tests made with hundreds of apparatus had first shown that the partition curves obeyed a normal logarithmic mathematical law. That had been of assistance in the interpretation of all measurements, and it had then been possible to prove a fundamental washing characteristic; as a result it was possible to know *a priori* how a new coal would be cleaned by any given apparatus. The formula varied from one process to another only by the coefficient of imperfection which had been defined in the paper.

After explaining some points raised in his paper, Mr. Cheradame extended a cordial invitation to those Latin-American countries which might be interested in the processes developed by CERCHAR to submit their specific problems to the organization or to send their engineers to visit the laboratories, where they would be very welcome to see how the work was conducted, and how to work with coals from their own countries.

Mr. PRIETO opened the general discussion on the paper with a reference to the opening paragraph of the section entitled "Cleaning of Latin-American coals", and asked whether, in fact, it was impossible to obtain clean coals in the large number of cases apparently covered by the statement, or whether the type of curve indicated would apply only to a few instances.

Mr. CHERADAME replied that quite possibly CERCHAR'S research on the coals covered only a few types which presented particularly unfavourable characteristics, but that many of the washability curves resulted in a very low yield of coal, even if the cut was made at 15% ash. It was unfortunate that he could not know how representative the coals he had studied really were, nor illustrate his point graphically at that stage, but he would gladly assist by studying any specific washability curves in detail, either in Bogotá or in France.

Mr. PRIETO said that he understood that French power plants were using coal middlings with an ash content which might be as high as 50%, and asked whether there were any special technical problems involved in their utilization.

Mr. CHERADAME replied that no particular problem arose, providing that the coal middlings were used in pulverized form.

Mr. FRASER pointed out that he had heard some scepticism expressed by Latin-American colleagues as to the applicability of the coefficient of imperfection to their coals, as they naturally felt that their coals might be different from European ones. He asked whether it would be possible to set up a range of types of coals for which that index had been determined; he also wondered what the author's experience had been with regard to the consistency of the index in coals with a low percentage of refuse, and contrariwise those with a high percentage of refuse. He personally had found some variation in the value of the imperfection coefficient as between those two extreme cases.

Mr. BELUGOU pointed out that French coals were very variable in their characteristics but that even so it was very difficult for him to generalize on the Latin-American situation. No doubt if coals of sharply differing washability characteristics (as defined by the curves) were compared, some small variations might be observed in the imperfection coefficients, but their total amplitude would never exceed 20% at the utmost, whereas the variations in the organic yield were drastic. The assumption of the constancy of the imperfection coefficient for a given washer was therefore a good practical working hypothesis.

Problems in the Preparation of Metallurgical Coal in Latin America, *presented by the author*

Mr. ALBALA opened the general discussion of the paper, saying that he was extremely interested to learn that Peru and Colombia possessed low-volatile coals similar to Pocahontas No. 3, as that would be important for Chile's iron and steel industry, which used that coal to blend with its own.

Mr. FRASER said that the Bureau of Mines had made some standard AGA carbonization tests on some low-volatile coals of the Cauca Valley, of the Cundinamarca area near Bogotá, and of the Oyón region in Peru. More complete data appeared in another paper to be presented at a later stage in the proceedings by the Instituto de Fomento Industrial.

A short discussion followed, in which Messrs. CHERADAME and BELUGOU participated, on the use of the term "Baum jigs" in the paper under review. In France the term "Baum jigs" referred to integral washing units, treating various sizes simultaneously. Such units were virtually unknown in France, while pneumatic jigs cleaning sized coals tended to easier distribution of the heat in the furnace could be achieved, so that there was less scaling. Furthermore, experiments had proved that the scaling by using blast furnace gas was very low compared with the scaling by coke oven gas or mixed gas. That was due to the low hydrogen content of the blast furnace gas which caused a low content of the oxidizing water vapour in the combustion gases. These results had been confirmed by practical experience. Since then blast furnace was used exclusively in the Rheinhausen soaking pits and reheating furnaces. It was, of course, necessary to preheat the blast furnace gas by a recuperator.

It was generally economical to use the blast furnace gas wherever possible within the steel plant and to sell the coke oven gas at favourable prices to smaller manufacturing firms and for domestic purposes, even when a long distance pipe line was required.

Mr. PÉRIN replied that the question of the production of energy in a steel plant was related to particular cases, some solutions being better than others. Plants having no blast furnaces and therefore little or no gas, had to take certain factors into consideration in turning to electricity. A hydraulic plant was essential but it was desirable to complement hydroelectric plants with thermal power plants to ensure regular current supply, and those plants using electricity had to use coal as well. Products were made which could be utilized in auxiliary thermal plants.

Mr. VUCETICH referred to the Huachipato plant, which had a special lay-out using a blast furnace blower and a Bessemer blower driven by hydro-electric power. The electric power supply was fairly reliable and there was a good market for the gas. The purpose was to save fuel and use it to the best advantage in accordance with the country's economic structure.

Mr. MERCIER pointed out that Mr. Périn had chosen a complete plant, with blast furnace, steelworks and rolling mill. In that case there was no need for a hydraulic plant as the blast furnace and the coke oven produced sufficient gas to supply the energy for the whole plant. In the case of accident, poor coal or refuse could be used in the power plant.

Mr. SCHERESCHEWSKY thought Mr. Périn's paper to be very complete, but it might perhaps be of interest to give a few figures concerning conditions in northeastern France. Mr. Périn had mentioned the use of high ash coals obtained from the coal preparation plants. That had led the French coal industry to associate with the steel industry in building washing plants which produced three instead of two products—commercial, middlings and rejects. The amount of middlings available for power plants naturally depended on the distribution of ash in the coal bed. Coal production in France being about 55 million tons, the amount of middlings anticipated for 1953, and practically available in 1952, was close to 4 million tons. In northeast France, there were already two 100,000 kW units powered with middlings, and four 60,000 kW units in the north, all of which were in operation. To complete the picture he would give two other figures; regarding the energy from blast furnace gas, the new plant of Herserange had been put into operation a year ago, using that gas entirely. The question of gas was of special interest for a city like Bogotá, which was fairly close to the steelmaking district. In the case of Paris, one million cubic metres of coke oven gas per day would be collected and shipped through a 350 km. pipeline to the city. That pipe line would be completed in 1953, and was unique, being the only one in the world to carry coke oven gas at a pressure of 50 atmospheres; such pressure was normally used only for natural gas pipe lines.

General MACEDO SOARES E SILVA pointed out that in washing one ton of Santa Catarina coal, between 27 and 30% of metallurgical coal was obtained, and 25% of steam coal, the rest, called "local coal", being used to produce electric power. As to the blowers at Volta Redonda, steam turbines were used which received the steam from boilers heated with blast furnace gas. As a standby for heating the boilers, steam coal from Santa Catarina was used. There was insufficient energy for firing the reheating furnace.

Construction of the Thomas Steel Plant of Paz de Río and Production Costs of Thomas Steel, *presented by the author*

Mr. ALLARD opened the general discussion of the paper, and asked why 1,200 kg. ingots had been selected, as he had often seen ingots of 3,000 kg. produced from a 20-ton converter.

Mr. DECHERF agreed that he had worked with 15-ton converters in a Thomas steel plant producing larger ingots with blows every 15 minutes and conveying them straight to the rolling mills.

Mr. BULLE found the paper very interesting. In Germany 800,000 tons a year had been made with six converters of the same size, so that with the converters indicated for Paz de Río a production of 400,000 tons could be anticipated. The distance between converters was not very great and might lead to difficulties in the future. He thought further consideration should perhaps be given to the charging of materials. In a discussion with French steelmen some four years previously, the possibility had been foreseen of charging 30% of scrap into the vessel, and that might be interesting for the Paz de Río plant. He did not consider that the small size of the ingots would lead to difficulties, as at one of the German plants, ingots of 1,000 kg. were being produced.

Mr. DECHERF explained that the distance between converters could advantageously have been more than 12 m., but as he did not believe that the plant would, in the future, have converters exceeding 22 tons, the capacity being increased rather by the number of converters, he considered 12 m. was sufficient.

Mr. CANGUILHEM wished to know what system would be used in the converter to obtain the slag, in order to prevent it falling in the box, and whether that system could be used in an acid converter, as in Chile falling slag had proved to be a problem.

Mr. DECHERF explained that lime of a special size was used.

Mr. CANGUILHEM wished further to know what importance was attached to the production of Thomas slag in relation to the economic development of Colombia.

Mr. DECHERF replied that the question had been studied; the slag at Paz de Río would have a high phosphate content, so that it would be useful as fertilizer, particularly in view of the high cost of such fertilizer in Colombia. Thomas slag in fact paid for a large part of Thomas production costs.

Foundry Iron by Recarburization of Steel Scrap, *presented by Mr. Prieto, co-author*

Mr. PRIETO prefaced his presentation by explaining that the paper dealt with the recarburization of steel scrap to make grey iron. The topic was a rather unusual one, as in most places the process was of an academic nature, yet little literature existed on the subject. The paper mainly aimed at showing that it was possible to prepare techniques and methods of work in less developed countries which would be completely unacceptable in countries with more developed industries. The purpose was twofold, first to show what could be done, where there was plenty of scrap and no steel industry, and second, to contribute to the literature on the subject.

Mr. BULLE opened the general discussion of the paper, pointing out that other countries had also studied the question; in Germany, during the last war, 70% of hematite pig had been replaced by scrap melted in cupola furnaces. A higher coke consumption was required. In Germany preheated air was used, the use of 60 kg. of coke resulting in a saving of 900 kWh. Methods were being studied to provide a cupola with a basic acid lining so that it would be a small blast furnace as well. There were three main methods of making synthetic pig iron, as it was called: the new electric furnace, the blast furnace, or the cupola converter.

There was, however, a difficulty in using the process, which was that silicon had to be added.

Mr. PRIETO disagreed with Mr. Bulle on the question of silicon, as recent papers had shown that silicon was reduced at high temperatures, and in his own experience he had found that 0.3 to 0.4% silicon could be put into the pig. Manganese ore and chromite, which were more easily reduced than silicon, had also been put in. The temperature of the cupola was considerably lower than that of the electric furnace. The time of contact of the decarbonizing agent was not long.

Mr. MERCIER pointed out that four methods had been described, and there was a fifth, the rotary furnace, which was perhaps more economical than the electric furnace, as it was fed by pulverized fuel or fuel oil.

Mr. PRIETO replied that the table showed that the consumption of electric power was about the same as that required in foundry practice.

Mr. COCK described earlier experiences in Colombia in making grey iron, at the Empresa Siderúrgica of Medellín.

Mr. PRIETO agreed that of course the method had been used for a long time, but pointed out that existing literature on it was scanty..

Mr. BUEHL explained that a small experimental blast furnace had been operated in the United States by the Bureau of Mines, that furnace being very similar to the cupola. The ratio was 6 or 7 metal to coke, with temperatures at around 1,600°C., so that it was possible to use all scrap in a cupola and get high metal temperatures.

A mixture of low carbon scrap and coke had been fed into a rotary furnace, and it was possible to produce a high carbon metal at temperatures in excess of 1,300°C. which meant that steel could be carburized in the solid state very rapidly at temperatures just below the melting point. In the liquid state the coke tended to float up, whereas in the solid state there was rather intimate contact.

Mr. DECHERF asked three questions:

1. Had pig iron with 4.25 and 4.50% carbon been produced without damage to the refractory lining and roof? As he had explained in his first paper, that was easy with basic converters;
2. As the ferro-silicon was very important in that process, had malleable white pig iron been produced?
3. What was the cementite (Fe^3C) content in a pig iron with the following analysis: Mn = 0.500, Si = 2 to 2.2?

The third point was important when making ingot moulds.

Mr. UCHOA complimented Mr. Prieto on his excellent paper.

Mr. PRIETO, referring to Mr. Buehl's remarks, stated that it had been possible to obtain grey iron with about 2.5% carbon and enough silicon. Mr. Buehl's remarks, he felt, really referred to the old cementing process by direct contact between the metal and the solid carburizing agent. Replying to Mr. Decherf's questions, he pointed out that occasionally the carbon content had been increased to 4.2, but no experiments had been made, because it had been possible to obtain sufficiently high temperatures without damage to the refractories. As to the second question, it was possible to produce white iron, as that only depended on carrying out the carbonization without boiling so that the carbon content was low, while keeping the silicon low enough so that the iron was white. The cementite content in the solid metal had not been studied.

Consumption of Iron and Steel Products in Mexico, *presented by the author*

Due to the lateness of the hour, there was no discussion on Mr. Aramburu's paper.

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disappear. Feldspar jigs were being used increasingly, but their characteristics differed from those of the Baum jigs, the tables and other devices.

Mr. FRASER replied that the reference in his paper was to a jig actuated by compressed air, and he believed such jigs were used in France. Comparing French conditions with those in all of America, on the basis of coal prices, any imperfections in performance cost about three times as much in France as they did in America, thus having an important bearing on plant design.

Mr. CORTÉS OBREGÓN remarked that two Mexican deposits had been omitted from the map in the paper; those deposits were included in the paper which he intended to present on the following day.

Mr. FRASER replied that Mr. Cortés Obregón's remark merely emphasized the fact that important coal reserves existed in Latin America beyond those shown on the map. The map was only intended to show the important producing fields and some well-known reserves.

Modern Techniques and Installations for the Mechanical Treatment of Coal by the PIC Company, *presented by the author*

Mr. TURPIN, after presenting his paper, availed himself of the opportunity to draw some conclusions from it. One of his firm's aims had been to construct a series of apparatus that would meet the needs of different types of coal or fractions thereof. Several types of washers were constructed by the firm, due to the fact that from the washing point of view, a large assortment of different coals were found in France as in Latin America, ranging from "easy" to "difficult". The usual procedure was to construct, whenever possible, a custom-built washer to deal with specific problems as they arose.

Mr. CHERADAME noted that the quality of washing was improved if an oversize jig were used instead of an exactly calculated jig. The mines had, however, made the criticism that the additional cost was high. He therefore wished to inquire what percentage of the cost was involved, and in how many years the original cost would be recovered.

Mr. TURPIN, whilst agreeing with Mr. Cheradame's general statement, pointed out that it had been found that up to a certain charge (quantity of material treated per hour by a given washer), the variation in the imperfection was slight, but that once such a limit had been exceeded, it increased rapidly. The amortization of oversize apparatus was very rapid, however, particularly since the cost of the washer in itself represented only 10–20% of a washing installation.

FUELS

Reduction of coal impurities (*continued*)

15 October 1952—Morning

Chairman:

Joaquín PRIETO, Assistant Manager, Empresa Siderúrgica Nacional de Paz de Río S.A., Bogotá, Colombia

Later

Eduardo PYLES LOZANO, Companhia Mineração Geral do Brasil, São Paulo, Brazil

Contributed Papers:

The Tromp Heavy Media Coal Washing Process
John GRIFFEN, The McNally Pittsburg Manufacturing Corporation, Pittsburgh, U.S.A.

The Coal Used in the Mexican Iron and Steel Industry
Salvador CORTÉS OBREGÓN, Oficina de Investigaciones Industriales, Banco de México, S.A., México, D.F.

Description of Chilean Coals Used in the Steel Industry
Alfredo GONZÁLEZ, Chief Engineer, Fuel Section, Depto. de Minas y Petróleo, Ministerio de Economía y Comercio, Santiago, Chile.

Problems Related to Colombian Coal Used for Steelmaking
Joaquín PRIETO, Assistant Manager, Empresa Siderúrgica Nacional de Paz de Río, S.A., Bogotá, Colombia

Jaime LÓPEZ T., Chief, Belencito Chemical Laboratory, Empresa Siderúrgica Nacional de Paz de Río, S.A., Colombia

Benjamín ALVARADO, Assistant Manager, Empresa Siderúrgica Nacional de Paz de Río, S.A., Bogotá, Colombia, and

Vicente SUÁREZ HOYOS, Chief, Cleveland Office, Empresa Nacional de Paz de Río, S.A., Colombia

Summary of Discussion:

Participants: Messrs. WOODHEAD, CHERADAME, TURPIN, ALBALA, CORTÉS OBREGÓN, PRICE, BELUGOU, LEUSCHNER, OSPINA HERNANDEZ, PRIETO

The Tromp Heavy Media Coal Washing Process

JOHN GRIFFEN

The Tromp heavy media process was developed in Holland prior to the Second World War by Mr. K. F. Tromp, then Chief Engineer of one of the private coal mining companies in the south of Holland. It is now in use in the important coal producing countries of Western Europe, viz., Holland, Belguim, France, Germany, Poland and England. There are 40 Tromp vessels now in use in Europe and 12 additional are being installed. These vessels are cleaning a wide variety of size ranges of coal. The largest coal handled is 12 in. (300 mm.) top size and the smallest being 1/5 in. (5 mm.) bottom size.

The Tromp process is now being installed in the United States. One large installation has just been put into operation in Illinois. Two other plants for the cleaning of coking coal are now under construction —one in eastern Kentucky and the other in Alabama. These three plants contain nine Tromp vessels with an input capacity of 1,160 tons per hour.

The medium used to effect the separation consists of finely ground magnetite or other heavy iron-bearing materials, such as spathic iron ore, suspended as a pulp in water. The media can be varied in specific gravity at will between the limits of about 1.30 to 2.0.

The coal floats on the bath of medium while the impurities sink to the bottom of the bath.

The Tromp process is distinguished from other heavy media processes by the use of coarser medium solids—substantially minus 180 microns—and the virtual absence of clay or other stabilizing constituents. The medium thus has a low percentage of solids by volume at a given specific gravity, giving it a high degree of fluidity and thus promoting sharpness of separation. The coarser medium solids and greater fluidity result in a semi-stable medium in which the coarser and heavier medium solids can slowly sink. No ascending or descending currents are used within the vessel to offset this settlement of the coarser medium solids as such currents would alter the gravity of separation. Avoiding these disturbing influences, the Tromp process utilizes carefully controlled horizontal currents of the medium across the bath to effect the separation of coal from the refuse at the desired specific gravity. The separations made closely approach the theoretical.

The process has the further feature of automatic medium density-control regulators which adjust the returned medium to the desired specific gravity for re-use in the bath with a very high degree of precision.

The recovery of the medium solids from the rinsing water used to remove medium adhering to the products, as generally practised, is by screens and gravity settlement methods. Experience with gravity settlement has shown that the medium solids can be efficiently and very completely recovered and the medium circuit quite effectively purged of contaminating solids from the feed so that the viscosity of the medium is kept at low levels. Low viscosity of the medium promotes a sharp separation. If magnetic solids such as magnetite are used, magnetic recovery and cleaning of medium solids can be utilized.

Two types of Tromp separating vessels are built. The two-product vessel separates the feed into coal and refuse, while the three-product vessel produces a coal product, a middlings product and a refuse product.

The three-product vessel is fairly deep, being triangular in a longitudinal section and is provided with a float or clean coal conveyor at the top of the vessel, a middlings conveyor disposed along one side of the triangle and a refuse elevator reaching the bottom of the triangle. Due to the nature of the medium, settlement takes place so that the specific gravity of the medium is progressively greater with depth. Advantage is taken of this gradation in density within the bath to effect a separation of the lighter coal, which is carried by the top layer of medium to the top conveyor which drains the coal of medium and elevates it from the vessel. Both middlings and pure refuse sink through this top layer. The middlings are supported by middle layers of higher density medium which flow towards the middlings conveyor. This conveyor is provided with a pervious bottom, allowing the middle layers of medium to flow out of the vessel for continuous return as transporting currents. The pure refuse sinks through to the bottom and is removed by the refuse drainage elevator. The density of the various strata in the bath are definitely controlled to the desired values by the continuous circulation of three or four circuits of the desired densities which are returned to the vessel at appropriately varying depths.

The three-product vessel discharges the coal and refuse products, drained free of medium, to rinsing screens of about 2 mm. opening where adhering medium is removed by water sprays. The middlings as discharged carry medium which is drained on the first section of a screen and on the second section are rinsed. The rinsings from the three screens are combined and pass to a slurry screen, making about a 180 micron separation, where the medium solids and water pass through and any plus 180 micron contaminating solids are removed and delivered to the fine coal cleaning section of the plant. The dilute medium from the slurry screen is processed for recovery and cleaning of the medium in a manner similar to that described later for the two-product vessel.

The economics of coal cleaning in Europe make the three-product vessel desirable, as the clean coal can be made of excellent quality for the market, the refuse product can be almost entirely free of combustible material and the middling product can be used locally or crushed and recleaned. Of the 44 vessels in use or being installed in Europe, 30 are three-product vessels. Performance typical of three-product vessels is given in Tables 1 and 2. In Table 3 are summarized the performance data of additional three-product vessels, showing the high precision of separation of the washed coal from the combined middlings and refuse. The data in Table 3 covers a considerable variety of size consists of coal treated and of density of separation, yet the misplaced material is quite small in amount—varying from 15 to less than 30% of the material in the feed coal plus or minus 0.02 S.G. of the density of separation.

An examination of the middlings in Tables 1 and 2 show them to be substantially a true middling product, containing only minor amounts of coal and pure refuse. In this respect the Tromp three-product vessel is believed to excel any other unit making a three-product separation.

TROMP TWO-PRODUCT VESSEL

In the United States of America, coal cleaning practice is largely based on a two-product separation and the three plants in operation or under construction employ the Tromp two-product vessel, which is illustrated in Figure 1. In this rather shallow vessel, the float coal is removed by a flight conveyor passing over 2 mm. wedge wire, which drains the medium from the coal, so that only rinsing is done on the coal screen. The refuse is removed by the lower flight conveyor and is accompanied by a stream of medium, which is drained on the first section of the refuse screen followed by rinsing with water sprays. The recirculated medium is introduced back of the feed point and provides horizontal currents to float the coal into the coal conveyor. These currents continue to and overflow at the refuse discharge point.

FUELS

Reduction of coal impurities

14 October 1952—Afternoon

Chairman:

Roberto JARAMILLO FERRO, Manager, Empresa Siderúrgica Nacional de Paz de Río, S.A., Bogotá, Colombia

Contributed Papers:

Considerations on Coal Washing in Europe and its Possible Application to Latin-American Coals
R. CHERADAME, Directeur général adjoint, Centre d'études et recherches des charbonnages de France, Paris, France

Problems in the Preparation of Metallurgical Coal in Latin America
Thomas FRASER, Coal Mining Engineer, Bureau of Mines, United States Department of the Interior, Washington, D. C.

Modern Techniques and Installations for the Mechanical Treatment of Coal by the PIC Company
Jacques TURPIN, Chief Engineer, Société de préparation industrielle des combustibles, Fontainebleau (S.-et-M.), France

Summary of Discussion:

Participants: Messrs. CHERADAME, PRIETO, BELUGOU, FRASER, ALBALA, CORTÉS OBREGÓN, TURPIN

Considerations on Coal Washing in Europe and its Possible Application to Latin-American Coals

R. CHERADAME

INTRODUCTION

While it is impossible to speak of European coal-washing trends, owing to the use of differing techniques imposed by tradition or by varying technical or commercial considerations, there is one essential feature common to all European countries. This common feature is the relative scarcity of resources, imposing the use of a large number of seams which would not be exploited in, for instance, the United States. Consequently it is imperative in Europe to use both dirty and clean coals, those that are difficult, as well as those which are easy, to wash. As a result, great experience has been acquired in the field of coal preparation.

The essential result of such experience extends far beyond the determination of which process to use, and lies in the fact that it has been necessary to determine accurately beforehand the fractions to be obtained by cleaning any coal by any process, under any separation conditions; these can now be determined.

On the basis of the imperfections of the various processes, therefore, the whole range of products obtained and the yield for each one can be compared, thus permitting a choice according to prevailing market conditions.

Before making such a choice, the following should be ascertained:

First, for a given process, the cuts which give the maximum yield. Furthermore, it is instructive to verify whether these optima cuts are also the most logical ones.

Second, given various possible processes, a comparison of the yields will have to be weighed against the cost of preparation using each process. Thus the economy of the various solutions can be judged.

Although France has practically no new mines, it should be emphasized that such a study is being systematically conducted on all French coals. The decision is then taken, as the case may be, to replace or improve the installations which may be either

Table 1

PERFORMANCE OF TROMP THREE-PRODUCT VESSEL, WASHING 80 TO 18 MM. COAL, PLANT C, HOLLAND (1)

Specific gravity					
Sink	*Float*	*Feed*	*Washed coal*	*Middlings*	*Refuse*
	1.55	76.57	98.99	0.09	
1.55	1.60	0.70	0.90	0.16	
1.60	1.65	0.66	0.09	12.90	
1.65	1.70	0.94	0.00	20.75	
1.70	1.75	0.84	0.02	18.20	
1.75	1.80	0.88		19.40	
1.80	1.85	0.89		18.00	0.31
1.85		18.52		10.50	99.69
		100.00	100.00	100.00	100.00

Specific gravity		*Per cent weight feed*				*Per cent in*	
Sink	*Float*	*Washed coal*	*Middlings*	*Refuse*	*Reconstituted feed*	*Washed coal*	*Refuse*
	1.55	76.568	0.004		76.572	99.995	
1.55	1.60	0.696	0.007		0.703	99.004	
1.60	1.65	0.070	0.587		0.657	10.654	
1.65	1.70	0.000	0.944		0.944	0.000	
1.70	1.75	0.016	0.828		0.844	1.932	
1.75	1.80		0.883		0.883		
1.80	1.85		0.819	0.056	0.875		6.400
1.85			0.478	18.044	18.522		97.419
		77.350	4.550	18.100	100.00		

Table 2

PERFORMANCE OF TROMP THREE-PRODUCT VESSEL, WASHING 50 MM. SQ. TO 12 MM. RD. COAL, PLANT B, FRANCE (2)

Specific gravity		*Per cent weight*			
Sink	*Float*	*Feed*	*Washed coal*	*Middlings*	*Refuse*
	1.50	57.12	93.20		
1.50	1.55		1.13		
1.55	1.60		1.57		
1.60	1.65	5.57	1.45		
1.65	1.70		1.59	1.01	
1.70	1.75		1.00	1.55	
1.75	1.80		0.01	19.32	
1.80	1.90		0.05	55.92	
1.90	2.00	37.31		20.62	
2.00	2.10			0.86	3.63
2.10				0.72	96.37
		100.00	100.00	100.00	100.00

Specific gravity		*Per cent weight feed*				*Per cent in*	
Sink	*Float*	*Washed coal*	*Middlings*	*Refuse*	*Reconstituted feed*	*Washed coal*	*Refuse*
	1.50	55.510			55.510	100	
1.50	1.55	0.673			0.673	100	
1.55	1.60	0.935			0.935	100	
1.60	1.65	0.864			0.864	100	
1.65	1.70	0.947	0.040		0.987	95.947	
1.70	1.75	0.595	0.062		0.657	90.563	
1.75	1.80	0.006	0.777		0.783	0.766	
1.80	1.90	0.030	2.248		2.278	1.317	
1.90	2.00		0.829		0.829		
2.00	2.10		0.035	1.322	1.357		97.42
2.10			0.029	35.098	35.127		99.92
		59.560	4.020	36.420	100.000		

Table 3

PERFORMANCE DATA—TROMP THREE-PRODUCT VESSELS (3)

Plant	Size of coal washed mm.	Specific gravity of separation	Washed coal		Middlings and refuse		Total misplaced material per cent of feed	Per cent weight ±0.02 S.G. from separating gravity of feed	Misplaced material as per cent weight of ±0.02 S.P. in feed
			Per cent weight	Per cent sink	Per cent weight	Per cent float			
A	50 to 8	1.750	49.20	1.10	50.80	0.90	0.996	5.98	16.67
B	50 to 12	1.733	59.56	0.13	40.44	0.20	0.160	0.60	26.70
C	80 to 18	1.610	77.35	0.07	22.65	0.22	0.105	0.55	19.10
D	18 to 9 [a]	1.439	72.07	0.65	28.93	1.62	0.940	6.00	15.70
D	9 to 6 [a]	1.435	72.28	1.17	27.72	3.07	1.750	6.15	28.50
C	90 to 22	1.358	93.18	0.56	6.82	7.74	1.050	4.26	24.60

[a] These two sizes were cleaned together in one vessel.

TROMP MEDIUM RECOVERY CIRCUIT

Figure 1 is a flow sheet of the complete installation of a Tromp two-product vessel with its medium recovery circuit. It will be noted that the equivalent of counter-current rinsing is employed to minimize the volume of rinsings which must be handled in the recovery circuit and also to minimize the amount of clear spray water required for the final rinse which in turn fixes the volume of effluent from the recovery

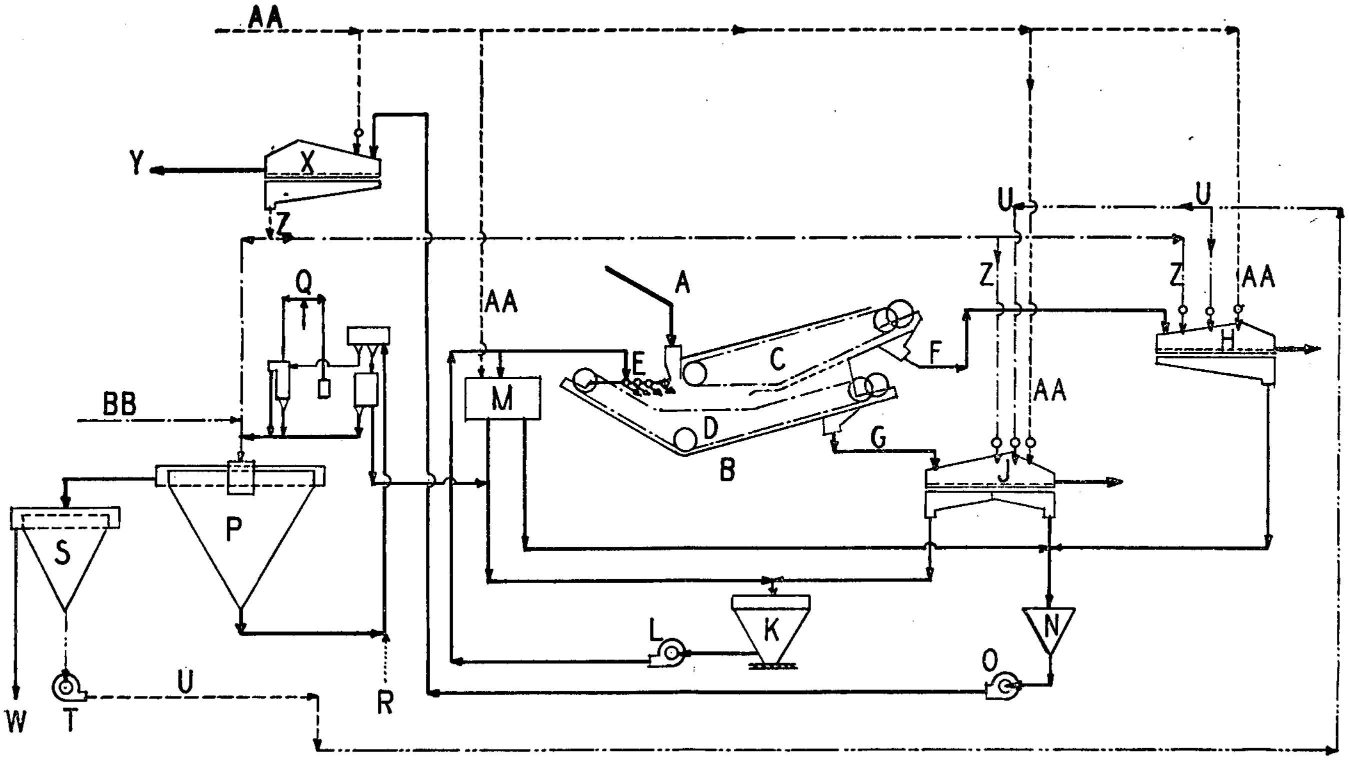

FIGURE 1-FLOW SHEET OF TROMP HEAVY MEDIA COAL CLEANING PLANT, SHOWING TWO-PRODUCT VESSEL AND MEDIUM RECOVERY AND RECONDITIONING EQUIPMENT

Legend Figure 1

A. Feed coal, prescreened
B. Tromp two-product vessel
C. Floats conveyor
D. Sinks conveyor
E. Medium to vessel
F. Floats to rinsing screen
G. Sinks to medium drainage and rinsing screen
H. Floats rinsing screen, 2 mm. openings
J. Sinks screen, ¾ mm. openings for drainage and 2 mm. openings for rinsing sections
K. Medium storage and recirculating sump
L. Medium recirculating pump
M. Bath automatic regulator and thickening cone
N. Dilute medium sump
O. Dilute medium pump
P. Medium recovery cone
Q. Medium recovery cone automatic regulator
R. Air lift
S. Classifying cone
T. Clarified water pump
U. Clarified water
W. Waste water carrying contaminating solids to the water clarification system
X. Slurry screen, 0.18 mm. openings
Y. Contaminating solids plus 0.18 mm. to fine coal cleaning section of the washing plant.
Z. Dilute medium
AA. Clean water make-up
BB. Make-up medium solids, minus 180 microns.

system. By these means the amount of clear water required for final rinsing sprays and the effluent from the plant can safely be kept to about 10% of the total volume of rinse application. An examination of this flow sheet will show that care has been taken that the slimes of any contaminating solids are forced to move to the final effluent from the system, at the same time assuring maximum recovery of the medium solids which are more granular as well as higher in density and thus settle in the medium recovery and classifying cones.

The rinsings or dilute medium not used for initial rinsing of the products is delivered to the medium recovery cone, where the medium solids settle to high-density pulp while the excess water, carrying any minus 180 micron contaminating solids, overflows to the classifying cone. The classifying cone acts to purge the medium system of contaminating slimes. Its overflow is substantially the volume of the clear water used as third or final rinsing sprays. The underflow of this cone is partially clarified water which is used for the second stage of rinsing.

RECOVERY CONE AND AUTOMATIC DENSITY REGULATOR

The medium settled in the recovery cone is withdrawn from its bottom by an air-lift and delivered to the recovery density regulator located above this cone. In this regulator a small stream of the medium is continuously diverted to flow through a chamber containing an hydrometer element suspended from one end of a scale beam, which, at the opposite end, carries a sliding weight for adjustment and a pole operating within two parallel coils. Measured impedance activates a modutrol motor, which actuates airvalves controlling an air cylinder moving a splitter gate to govern the disposition of the recovered medium. With the recovered medium at the desired density, the splitter gate directs it to the medium storage and recirculating sump for return to the vessel. Should the recovered medium be lower in density than the desired value, the splitter gate diverts it to the recovery cone for thickening to the desired value. Should the recovered medium have a density greater than desired, the modutrol motor acts to increase the amount of air to the air-lift which increases recovered medium withdrawal volume and thus dilutes the medium to the desired value.

BATH AUTOMATIC DENSITY REGULATOR

Where prescreening of the feed is done wet, and this is usually required, the feed carries water which dilutes the medium in the vessel, so the density of the recovered medium is raised above the desired value in the bath sufficient to compensate for such dilution. As the amount of moisture in the feed will vary from time to time, a similar automatic regulator is provided at each vessel to correct for such variations—this is called the bath regulator.

The bath regulator is similar in principle and construction to the recovery regulator and works with a small thickening cone. A small proportion of the medium returning to the vessel is diverted to the thickening cone where it is thickened and its underflow is sent to the medium storage and recirculating sump where it tends to increase the density of the recirculated medium. The overflow of the small thickening cone is dilute medium, which flows to the bath regulator. A small stream of recirculated medium is diverted to the hydrometer chamber of the bath regulator for continuous measurement of density. With the recirculated medium at the desired density the overflow of dilute medium is diverted to the storage and recirculating sump and the main flow of the medium to the bath remains at the desired value. Any lowering of the density causes a corresponding amount of the dilute medium sump, bringing the recirculated medium to the desired value. Should the recirculated medium increase in density above the desired value, the raising of the hydrometer not only diverts the dilute medium to the storage and recirculating sump but opens a valve port to admit the required amount of water to dilute the medium to the desired value.

The recovery and bath regulators combine to keep the density of the medium in the vessel almost always within 0.01 S. G. of the desired value and usually within 0.005 S. G. of this value. As operating results show that the separations made in the Tromp vessel are accurate to a small percentage of the feed coal found in the range of plus or minus 0.02 S. G. of the separation density, it is quite important that a uniform density of the medium in the vessel be maintained. To date, the Tromp system is the only heavy medium system which has had automatic density regulators in use for a number of years. This feature, which largely eliminates operator attention, is quite valuable in maintaining continuously very sharp separations at the desired density.

TROMP INSTALLATIONS IN THE UNITED STATES OF AMERICA

The first of these installations is in a new preparation plant at the No. 22 Mine, Old Ben Coal Corporation, Valier, Illinois, and was designed for an input of 730 tons per hour of 150 mm. to 0 raw coal from the Illinois No. 6 seam (4). Run-of-mine coal from No. 22 Mine is hoisted through a shaft and raw coal from other mines, also working the Illinois No. 6 seam, can be delivered by railway cars. Any run-of-mine coal is crushed to plant feed size and stored in a four compartment blending bin of 730 tons capacity.

As Mine No. 22 is being reopened and further developed, initial operation was started in December 1951 on a 75 mm. to 0 coal from other mines. By April, 1952, regular operation on coal from Mine No. 11 was being done, with the washing plant feed being minus 150 mm. The feed coal is screened into three sizes before cleaning—150 to 50 mm., 50 to 8 mm. and 8 mm. to 0. All screens have round holes or make the equivalent of round hole separations, except the 8 mm. screens which are square. The 8 mm. to 0 is cleaned in Rheolaveur fine coal units and the plus 8 mm. sizes are cleaned in two-product Tromp vessels as follows:

Primary separations

150 to 50 mm. Feed amounts to 285 tons per hour which is cleaned in two vessels 2.438 metres wide each and separated at about 1.36 S.G.

50 to 8 mm. Feed, consisting of about 233 tons per hour of raw coal plus about 27 tons per hour of crushed middlings or a total of 260 tons per hour, is cleaned in two similar vessels and separated at about 1.43 S.G.

Table 4

FLOAT AND SINK DATA ON VARIOUS SIZES OF RAW COAL. MINE NOS. 11 AND 22, OLD BEN COAL CORP., VALIER, ILLINOIS
(No crushed middlings included)

Size—mm.	*150 to 75*	*75 to 50*	*150 to 50*	*75 to 50*	*50 to 8*	
Mine No.	*22*	*22*	*22*	*11*	*22*	*11*
Ft. 1.30	9.49	16.87	12.65	12.72	33.69	28.45
Sk 1.30; " 1.35	54.09	53.22	53.72	49.28	35.89	39.69
" 1.35; " 1.36	21.06	14.80	18.38	8.24	13.31	9.33
" 1.36; " 1.40				7.35		
" 1.40; " 1.43	5.00	4.76	4.90	5.02	4.87	2.78
" 1.43; " 1.45						1.08
" 1.45; " 1.50	2.36	1.83	2.13	3.05	2.00	2.58
" 1.50; " 1.55	1.83	1.09	1.51	1.26	1.24	1.34
" 1.55; " 1.60	1.07	0.82	0.96	1.43	1.07	1.09
" 1.60	5.10	6.61	5.75	11.65	7.93	13.66
	100.00	100.00	100.00	100.00	100.00	100.00

Secondary separation

150 to 25 mm. Feed amounts to about 110 tons per hour which is cleaned in one vessel 1.829 m. wide at about 1.60 S.G. The feed to this vessel consists of the plus 25 mm. size of the sinks from the four primary vessels. The minus 25 mm. contains so little intergrown coal values that its retreatment is uneconomic and it is discarded as refuse. The floats from the secondary vessel are crushed minus 38 mm. and sent to the blending bin where they mix with the raw coal and are reprocessed through the plant.

Table 4 gives the float and sink data on various sizes of raw coal from Mines Nos. 11 and 22 which are representative of raw coal cleaned in the Tromp vessels. In Tables 5 and 6 are shown the float and sink data on the products from the primary vessels, i.e., on the 150 to 50 mm. and 50 to 8 mm. sizes. Unfortunately, the full range of specific gravity fractions was not determined on the products so that the specific gravity distribution of the reconstituted feeds cannot be calculated nor the exact distribution of the various gravity fractions established from which the density of separation and the sharpness of separation are determined.

Table 5 indicates that the density of separation of the 150 to 50 mm. coal was at 1.36 S.G. or very slightly above that point. The data in Table 5 shows that the feed contained 68.566% of coal floating at 1.36 S.G. of which 68.122% was recovered in the cleaned coal or a recovery of 99.35% of the float 1.36 S.G. in the feed. Table 5 also shows that 93.39% of the sink 1.36 S.G. was removed in the sinks product. Taking 1.36 S.G. as the separating density, the total misplaced material was 2.522% of the feed. The data in Table 4 on the 150 to 50 mm. raw coal from Mine No. 22, when plotted on a specific gravity graph, indicates that it contains 21.1% by weight of material between 1.34 and 1.38 S.G. and it is probable that the actual feed during the period covered by Table 5 was closely similar in its specific gravity distribution. From this we can assume that the total misplaced material for the separation of the 150 to 50 mm. coal was about 12 per cent of the 1.34 to 1.38 S.G. material in the feed coal.

Table 5

SPECIFIC GRAVITY TESTS ON PRODUCTS FROM TROMP PRIMARY VESSELS CLEANING 150 TO 50 MM. RD. COAL, MINE NO. 22 WASHING PLANT, OLD BEN COAL CORP. AVERAGE FOR PERIOD, 3 TO 21, APRIL 1952 (4)

		Cleaned coal		*Primary sinks*	
Sink	*Float*	*Per cent weight*	*Per cent weight of feed*	*Per cent weight*	*Per cent weight of feed*
	1.30	97.04	68.122	0.03	0.009
1.30	1.32			0.10	0.030
1.32	1.34			0.43	0.128
1.34	1.36			0.93	0.277
1.36	1.38	2.24	1.573	98.51	29.356
1.38	1.40	0.64	0.449		
1.40	1.42	0.02	0.014		
1.42	1.44	0.04	0.028		
1.44	1.46	0.01	0.007		
1.46		0.01	0.007		
	TOTAL	100.00	70.200	100.00	29.800
Sink 1.36 S.G. in cleaned coal		2.96	2.078		
Float 1.36 S.G. in sinks		—	0.444	1.49	
Total misplaced material			2.522		

Table 6

Specific gravity tests on products from Tromp primary vessels cleaning 50 mm.d. to 8 mm. sq. coal, Mine no. 22 washing plant, Old Ben Coal Corp. Average for period, 2 to 21 April, 1952 (4)

		Cleaned coal		*Primary sinks*	
Sink	*Float*	*Per cent weight*	*Per cent weight of feed*	*Per cent weight*	*Per cent weight of feed*
	1.33	96.66	77.618	1.77	0.349
1.33	1.35			0.45	0.089
1.35	1.37			0.58	0.114
1.37	1.39			0.66	0.130
1.39	1.41			0.85	0.168
1.41	1.43			1.62	0.319
1.43	1.45	1.17	0.940	94.07	18.531
1.45	1.47	1.11	0.891		
1.47	1.49	0.52	0.418		
1.49	1.51	0.29	0.233		
1.51	1.53	0.12	0.096		
1.53		0.13	0.104		
	Total	100.00	80.300	100.00	19.700
Sink 1.43 S.G. in cleaned coal		3.34	2.682		
Float 1.43 S.G. in sinks		—	1.169	5.93	
Total misplaced material			3.851		

Table 6 indicates that the density of separation of the 50 to 8 mm. coal was at 1.43 S.G. or very slightly above that point. The data in Table 6 shows that the feed contained 78.787% of coal floating at 1.43 S.G. of which 77.618% was recovered in the cleaned coal or a recovery of 98.52% of the float 1.43 S.G. in the feed. Table 6 also shows that 87.35% of sink 1.43 S.G. was removed in the sinks product. As the feed to the 50 to 8 mm. Tromp vessels contained about 10% of crushed middlings, its specific gravity distribution must have been quite different from the data for this size shown in Table 4. Unfortunately, the information available does not enable one to compare the misplaced material with the amount of material in the feed between 1.41 and 1.45 S.G., as this latter figure is unknown. It is evident that the separation in the 50 to 8 mm. vessels is not so sharp as in those cleaning the 150 to 50 mm. coal. However, the following is of interest: "A summarized comparison is available with respect to earlier separation of similar raw coal from the same source of supply in a Baum jig washer. Heavy media separation in the small coal bath units has provided a cleaned product with around 1.5% less ash, together with a small increase in recovery" (4).

At the time this paper was prepared detailed performance was not available on the operation of the vessel separating the middlings from the 150 to 25 mm. primary sinks.

This installation is still being adjusted and its performance improved. "Initial operation entailed an excessive magnetite loss due to accidents and other difficulties". Beginning in March 1952, the feed was changed from 75 to 150 mm. to 0. "Magnetite loss during the month was reduced to an average of 0.444 kg. per ton of raw coal delivered to the bath units. Allowance for the estimated volume of recirculated middlings would indicate an average magnetite loss of 0.404 kg. per ton of input to the baths" (4).

The second Tromp installation in the United States of America is being made for the Eastern Coal Corporation, Stone, Kentucky, and will clean 320 tons per hour of 125 mm. round to 6 mm. square coal from the Pond Creek and Freeburn seams. The cleaned coal will be largely shipped as a high quality coking coal. The raw coal will come from two mines and contains quite low ash pure coal, but also has a large amount of intermediate gravity middlings with a small quantity of high ash refuse. Separation will be made at about 1.375 S.G., but can be varied above or below this density as experience may dictate. Test data on the blend of 125 to 6 mm. feed coal from the two mines shows the following:

Table 7

		Per cent
Float 1.375 S.G.	Weight	79.80
Float 1.375 S.G.	Ash	4.06
Sink 1.325: Float 1.425 S.G.	Weight	17.80
Float 1.45 S.G.	Ash	5.06
Raw coal	Ash	10.35

With about 18% of the feed lying in the range plus or minus 0.05 S.G. from the separating density an extremely precise and sharp separation must be made to produce a clean coal that will closely approximate the theoretical recovery and ash content of the coal floating at 1.375 S.G. With the ash content of the float coal increasing 1.00% from 1.375 to 1.45 S.G., it is apparent how important is a quite precise separation at or very close to 1.375 S.G.

The 320 tons per hour of 125 to 6 mm. feed will be cleaned in two Tromp two-product vessels each 2.743 m. wide and when separating at 1.375 S.G. the washed coal recovered should equal the amount of theoretical float at that gravity and its ash content should be less than 0.1% greater than the theoretical. Thus, the washed coal should not exceed 4.2% ash with a feed like that in Table 7.

The third Tromp installation being made in the U.S.A. is to treat Mary Lee seam coal in Alabama for the production of coking coal. This seam is noted for its high ash coal due to finely laminated refuse inclusions. The feed of 300 tons per hour will be crushed minus 28 mm. and the plus 6 mm. square raw coal will be cleaned in two Tromp two-product vessels, each 1.829 m. wide. It is planned that separation will be made at a specific gravity of 1.40 or slightly lower where the amount of near gravity material is considerable. The refuse rejected will be approximately 40%. The feed to the two vessels will amount to about 165 tons per hour.

REFERENCES

(1) Test data from private files.

(2) K. F. Tromp, Private communication.

(3) John Griffen, "The Tromp Heavy Media Washing Process." Blast Furnace, Coke Oven and Raw Materials Conference, American Institute of Mining and Metallurgical Engineers, Pittsburgh, Pa., U.S.A., 1 April, 1952.

(4) J. W. MacDonald, Chief Engineer, Old Ben Coal Corp., "Dense Media Separations by Tromp Process", American Mining Congress, Cincinnati Coal Convention, 7 May, 1952.

The Coal Used in the Mexican Iron and Steel Industry

SALVADOR CORTÉS OBREGÓN

The map of Mexico (Figure 1) shows the location of the Sabinas and Saltillito coalfields, from which all metallurgical coal used in Mexico is derived. The Instituto Nacional para la Investigación de los Recursos Minerales de Mexico (National Institute for Research into Mexico's Mineral Resources), with the co-operation of the United States Geological Survey, is carrying out detailed geological surveys of the coal formations. These surveys are progressing satisfactorily.

STRATOGRAPHY

Three distinct formations have been classified in the Upper Cretaceous which are, mentioning the oldest first: San Miguel, Olmos and Escondido.

The Olmos formation in the Sabinas basin has many fine seams of coal but all the prospecting, drillings and studies show that only the lower double seam can be mined economically.

During mining operations, it has been found that locally the coal seam adopts the horizontal stratigraphic characteristic of more or less horizontal lenses. The seam varies in thickness from 0.3 m. to 5 m. The best mines have been developed where the double seam loses its central partition and becomes a single seam 2 m. thick. This deep vein is found in lenses which may be as much as 2 km. in diameter. The presence of these lenses, together with other details such as thinning or impoverishment through excess ash, can be determined only by drilling.

There are also poor lenses, but fortunately these occur to a much lesser extent. They are probably due to intrusions during the formation of the coal, resulting in a very thin or very dirty coal seam. As an example there is the curious case of No. 5 mine in Palau, which was blocked by a belt 1 km. wide, in the form of a horseshoe, of very poor, thin coal; when this belt was reached, working ceased.

STRUCTURE OF THE SABINAS COALFIELD

The Sabinas coalfield is situated in a desert plain; its basin is a synclinal with slight folds which take the shape of a spoon; the horizontal section has the form of an ellipsoid, its longer axis being 50 km. and the shorter 20 km. At the deepest point the seams are some 350 m. below the surface. Modern limestone gravels cover the coal so that its outcrops can be seen only in streams or other places where the banks of gravel have been removed.

The seams are most conveniently worked when they are almost horizontal and near the surface. This type of working has been most developed in the Sabinas and Saltillito coalfields. Even with gradients of 10% or less, a system of faults has been found which has affected mine work. In most cases drillings made before developing the mine did not indicate the faulted zones, and important changes have had to be made after the faults were encountered. Faults have a dip reaching in some cases as much as 40 m.

Through partial knowledge of the fields, continuity of the seams and outcrops, and geological inference, estimated reserves are set at some 2,000 million tons of coal, most of which is suitable for coking. More conservative figures set the reserves at 1,250 million tons.

MINING

The topographical and structural features of the Sabinas and Saltillito coalfields give rise to three differing types of mining: the system known as "pocitos" (little pits) which works the seam at the outcrop and consists in perforating the overburden with pits reaching to a depth of 20 m.; once the seam is reached, it is worked for distances up to 50 m., communications being established with other pits to ensure natural ventilation; at times it is facilitated by a small wooden chimney, 5 m. high, placed above one of the pits. This type of working is naturally very uncertain and is conducted by small concerns. The costs of this system are relatively low as there is no heavy investment; cheap labour is employed and there is no machinery as the coal is broken down by pick and shovel; hauling is effected by means of trucks to the shafts where it is hoisted to the surface. On the rare occasions when water is found in the seam, it is extracted by small pumps. This method accounts currently for only 2.7% of national production, its cost being 73, taking the average cost of total production as 100.[1]

Table 1

COAL PRODUCTION

Years	*Kilogrammes*
1940	815,907,000
1941	855,696,787
1942	914,268,500
1943	1,025,325,893
1944	904,198,293
1945	914,614,115
1946	977,874,100
1947	1,040,359,856
1948	1,057,225,624
1949	1,074,707,483

Source: Boletín de Minas y Petróleo.

Note: Annual averages in: March 1946, July 1948, May 1949, June 1950.

Another system is based on mining outcrops. Here coal is extracted from the vicinity of outcrops or even from protective barriers left between pits up to a certain depth and with dips in the seam ranging usually from 10 to 4%. This type of mine gives a heavy yield at quite low unit cost, and even though it re-

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[1] Author's estimate.

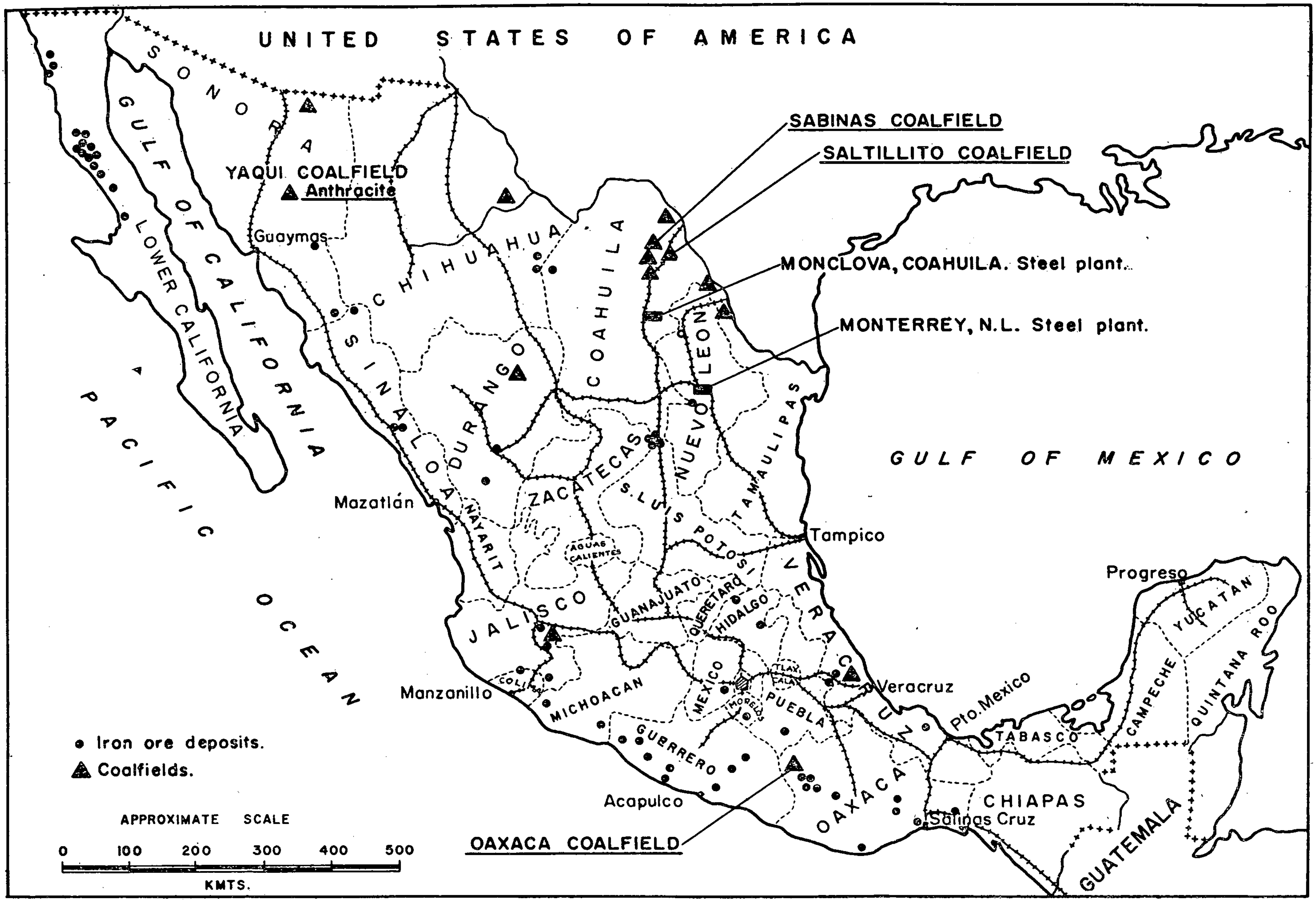
UNITED STATES OF AMERICA
SONORA
GULF OF CALIFORNIA
LOWER CALIFORNIA
YAQUI COALFIELD
Anthracite
Guaymas
CHIHUAHUA
SINALOA
DURANGO
COAHUILA
NUEVO LEON
TAMAULIPAS
ZACATECAS
S. LUIS POTOSI
AGUAS CALIENTES
NAYARIT
JALISCO
COLIMA
MICHOACAN
GUANAJUATO
QUERETARO
HIDALGO
MEXICO
MORELOS
TLAXCALA
PUEBLA
VERACRUZ
GUERRERO
OAXACA
CHIAPAS
TABASCO
CAMPECHE
YUCATAN
QUINTANA ROO
GUATEMALA
SABINAS COALFIELD
SALTILLITO COALFIELD
MONCLOVA, COAHUILA. Steel plant.
MONTERREY, N.L. Steel plant.
GULF OF MEXICO
PACIFIC OCEAN
Mazatlán
Manzanillo
Acapulco
Tampico
Veracruz
Pto. Mexico
Progreso
Salinas Cruz
OAXACA COALFIELD
• Iron ore deposits.
▲ Coalfields.
APPROXIMATE SCALE
0
100
200
300
400
500
KMTS.

Fig. 1.

presents larger investments, as it requires a more integrated mine, it does not have to face the problems of deep mining. Some 53.3% of national coal production is extracted from this type of mine, different methods being used to remove the coal, as for instance: use of explosive and pneumatic drills, extraction with pick and shovel, mechanical loading devices for cars and even conveyors; internal transport by means of mule, electric trains, cars hauled by a cable, or conveyor belts. In these cases forced ventilation by fans is used. Costs in this type of mine represent 93.2, on the same basis.

Finally, deep mining presents much greater problems: the heavy load on the roof requiring more elaborate timbering, greater need of ventilation, as well as longer vertical lift. Working of these mines is similar to that in the preceding case, but a vertical extraction shaft is used to take the coal to the surface. This type of mine accounts for 44% of total production and has a cost of 110.1 taking the same base as in the previous instances.

In 1950, production amounted to 1,070,500 metric tons. In view of the development plans at the mines, it is probable that by 1953 annual production will reach 1,200,000 tons. About 86% is produced by the Sabinas coalfield, 1% from the Fuente field which is non-coking coal, and the rest from the Saltillito field which also produces coking coal.

The Mexican mines tend only to partial mechanization, on account of technical difficulties, mainly a roof which is not conducive to the use of heavy machinery. In almost all the mines the roof is composed of clay which tends to fall in, and therefore requires too close timbering to permit use of machines. The method currently followed is that of closed timbering, quick working and later allowing the mine to fall in. The abundance of cheap labour is another factor working against mechanization.

The general method of extraction in all the mines is that of "room and pillar", consisting in carrying out advance work until a zone is "blocked out" and later allowed to fall in upon retreat, leaving only supporting pillars in order to maintain gangways for extraction, and for general transport.

ANALYSIS OF THE COALS

The analysis of representative coals from the Sabinas and Saltillito fields is given below:

PROXIMATE ANALYSIS

Saltillito coalfield

	Moisture free				
Mine	*Volatile matter*	*Fixed carbon*	*Ash*	*Sulfur*	*Free swelling index*
Saltillito	21.6	51.9	26.5	—	7.5
Barroterán	22.5	49.4	28.1	—	—
	Sabinas coalfield				
Rosita	24.4	47.1	28.5	1.10	9
Cloete	22.6	52.3	25.1	1.10	9
Palau	21.2	57.2	21.6	1.30	8
Sauceda	21.1	55.4	22.5	1.30	8
Sabinas	22.0	52.6	25.4		6
San Felipe	19.2	60.9	19.9	1.25	6
El Coyote	20.8	54.1	25.1	1.25	7

SOFTENING TEMPERATURE FOR THE ASHES

(Rosita coal)

	Reduction zone	*Oxidation zone*
Initial deformation, °C	1538	1532
Softening temperature above	1593	1566
Melting temperature above	1593	above 1593

SCREENING ANALYSIS

Barroterán coal, run of mine (Saltillito coalfield)

Sizes		*Per cent weight*	*Per cent fractions*			*Per cent cumulative*		
Screened	*Retained*	*Original*	*Weight*	*Ash*	*S*	*Weight*	*Ash*	*S*
	1″	31.0	—	50.7	2.3			
1″	¼″	31.7	45.9	32.3	2.1	45.9	32.3	2.1
¼″	⅛″	13.3	19.3	32.5	2.5	65.2	32.4	2.2
⅛″	½″	10.5	15.2	25.4	2.5	80.4	31.0	2.1
½″	0	13.5	19.6	19.2	2.2	100.0	28.7	2.3
	TOTALS	100.0	100.0	28.7	2.3			
Composite 1″ – ½ mm.		100.0	31.0	2.3				

Palau coal (Sabinas coalfield)

		Per cent sample (1)				*Per cent sample (2)*			
Sizes		*Fractions*		*Cumulative*		*Fractions*		*Cumulative*	
Screened	*Retained*	*Weight*	*Ash*	*Weight*	*Ash*	*Weight*	*Ash*	*Weight*	*Ash*
	1¼″	53.5	23.0	53.5	23.0	11.6	51.9	11.6	51.9
1¼″	¾″	—	—	—	—	12.3	48.4	23.9	50.1
1¼″	½″	16.5	15.5	70.0	21.2	—	—	—	—
¾″	¼″	—	—	—	—	22.8	44.6	46.7	47.2
½″	0	30.0	14.5	100.0	19.2				
¼″	20 m.	—	—	—	—	47.3	38.5	94.0	42.9
20 m.	0	—	—	—	—	6.0	33.5	100.0	42.4

Sample (1) is run-of-mine coal.

Sample (2) was taken specially to study the dirtiest coal which could be obtained from the mine in the event of complete mechanization and full seam mining. Sample (2) gave very good results in washability and coking tests.

As may be seen from the screening analyses, both the Sabinas and Saltillito coals are cleaner in their smallest size fractions.

COAL PREPARATION

Three main coal washing plants are situated at the coking plants in the vicinity of the Sabinas coalfield. The Rosita plant receives coal with 32 to 28% of ash and 1% of sulfur. With a 70% yield it reduces the ash content to 15% and 0.8% sulfur.

ANALYSIS OF COAL AND PRODUCTS OF THE ROSITA WASHING PLANT

(Percentages)

	Volatile matter	*Fixed carbon*	*Ash*
Run-of-mine coal	22.3	45.5	32.2
Fine coal for the coking ovens	25.8	59.2	15.0
Washed pea coal	25.1	58.1	16.8
Waste	13.5	17.5	69.0

The Palau plant receives coal with 26% of ash and 1.25 sulfur and reduces it to 17% and 1%, respectively, with a 70% yield. A modern plant of greater capacity is being built to replace it, and will be ready by January 1953. It will treat 3,000 tons of raw coal in 16 hours. It is also expected to have a yield of about 70% and it will have the following equipment:

The coal will first pass through a 6 in. grid, whence the oversize will go to a hand picking table and later, the 6 in. — ¾ coal passes through a screen and the diaphragm. The washed coal will be dewatered and stored for shipment, the middlings from the jig will be crushed with the raw fines and all together will pass through a battery of 12 Deister tables, which will give fine washed coal for coking purposes and the waste which will be added to that from the jig. Water used in washing will be recirculated through a settlement tank. The plant will deliver 120 tons of washed coal per hour.

Finally the third existing plant is the Agujita washer used for washing coal mainly derived from the Barroterán mines in the Saltillito coalfield. This washer with a 70% efficiency reduces the ash from 28.0% to 15.68% and has a capacity of 60 tons of raw coal per hour.

THE OAXACA COALFIELD

This coalfield is some 350 km. to the south of Mexico City, on the northern slope of the Mixteca hills. It is presently being investigated, especially with a view to finding coking coal sources in this district.

The coal formation has been sub-divided into three:

1. The lower coal formation (present only in the Tezoatlân zone), with a thickness of 300 m., composed of lutites and slates with layers of coal and carbonaceous shales, sandstones and a dark brown conglomerate of limestone rubble in clayey cement.

2. The intermediate conglomerate (present in the Tezoatlân and Mixtepec zones) with a thickness of 50 to 100 m., composed of quartzite and quartz conglomerates with very hard siliceous cement.

3. The upper coal formation (present in the zones of Tezoatlân and Mixtepec), over 500 m. thick, composed of sandstones, lutites and sandy shales with layers of coal and of carbonaceous sandstone and slate, and thin layers of argillaceous limestone.

Three workable seams have been found: one in the lower formation with a thickness of 1.5 to 2 m. and two in the upper formation, also from 1.5 to 2 m. thick, appearing in some places as a single seam of 3 or more m. thick; elsewhere only one of the seams is present, the second being absent or very thin.

From the general information obtained from small prospecting mines it is found that the coal is usually dirty with highly dispersed ash and with one or two bands of bone of varying thickness which can be easily separated. The roof is usually strong; inclination of the seams is variable, the dips usually varying from 10 to 30° with a tendency to a lesser inclination as depth increases.

In the Mixtepec zone, which is the least explored, reserves of 3 million tons of workable coal have been proved; they will be good for coking coals (this reserve will be increased when more exploratory work is completed).

In the Tezoatlân zone 90 million tons have been proved. Two drilling rigs, one for 350 m. of depth and the other down to 680 m. have completed the first three drillings at depths ranging from 250 to 350 m. These drillings confirm geological inferences and convert "possible" into "proved" reserves. In the Tezoatlân zone no caking coal has been found, but it is hoped that conditions may improve at larger distances from the igneous rocks.

ANALYSIS OF SAMPLES TAKEN IN EXPLORATION OF OUTCROPS

Sample No.	*Thickness sampled (metres)*	*Dry Base*				
		Volatile matter %	*Fixed carbon %*	*Ash %*	*Calorific value BTU*	*Sulfur %*
66	1.00	19.50	26.26	54.24	4,509	0.26
111	1.60	23.75	40.42	35.83	—	—
113	0.70	19.97	45.35	34.68	7,694	0.49
77	0.48	21.80	49.34	28.86	—	—
3	2.20	17.82	48.44	33.74	8,213	0.40
4	1.47	17.92	53.28	28.80	9,120	0.43
213	1.48	13.78	47.57	38.65	7,909	0.41
229	1.19	12.43	55.49	32.08	9,311	0.47
14	1.42	7.63	43.25	49.11	7,146	0.77
27	1.52	16.76	44.41	38.83	8,523	0.64
42	3.70	18.45	46.40	35.15	8,747	0.64

The melting temperature of the ash in these coals varies, since initial deformation takes place at from 2,600 to 2,850°F. (most of the samples at around 2,800°F.) and melting at 2,800°F. to 2,900°F. (most of them at 2,900°F.).

ANALYSIS OF SAMPLES TAKEN IN SMALL MINES OVER 20 METRES OUTCROP

	Lower formation Tezoatlán Zone		*Upper formation Mixtepec Zone*	
	As received	*Free from moisture and ash*	*As received*	*Free from moisture and ash*
Moisture	.9	—	1.0	—
Volatile matter	9.1	12.0	17.4	25.4
Fixed carbon	66.6	88.0	51.1	74.6
Ash	23.4	—	30.5	—
Hydrogen	3.4	4.3	3.8	5.3
Carbon	68.3	90.2	59.0	86.0
Nitrogen	.7	.9	1.0	1.4
Oxygen	3.6	3.8	4.9	6.2
Sulfur as:	.6	.8	.8	1.1
Sulphates	.02	.02	0.03	0.05
Pyritic	.13	.17	.23	.33
Organic	.43	.57	.51	.74
BTU	11,560	15,270	10,430	15,220
	Free swelling: non-coking		Swelling index: Free: 5	

The ash content is high, and analyses showed no appreciable difference in the ash content for different sizes. Float-and-sink tests showed that in small sizes between 6 and 60 mesh per square inch, it was possible to wash the coal to ash contents below 20% with recoveries ranging from 60 to 75%. Most of the samples were of weathered coal as they came from the outcrops; later samples of deeper coal (over 20 m.) have given better results.

The reserves proved so far, added to their proximity to Mexico City and considering the necessity of grinding them to fines in order to wash them, suggest the following uses for the Oaxaca coals:

Combustion in pulverized form, manufacture of briquettes for domestic fuel as a substitute for charcoal (briquetting tests in the laboratory have been very satisfactory and it is planned to continue them on a pilot plant scale). As regards uses, in view of their reduced binding possibilities, it has been thought that they could be utilized for iron ore reduction by methods other than the blast furnace.

Laboratory tests have been made for the manufacture of metallurgical coke, by carbonizing in a small oven at 1,000°C., blends of coals from the Mixtepec zones (with greater binding power) with those from the Tezoatlán (with few binding properties); it has been found that coke samples can be obtained with mixtures of up to 65% of Tezoatlán coal with 35% of Mixtepec coal.

THE YAQUI COAL DEPOSIT

This is located in the northeast of Mexico in the state of Sonora.

In 1945 complete surveys were made of the superficial geology of this deposit (2) and (3), and a study for its utilization in an electric plant (1).

Possible reserves, estimated in one of the surveys (3) which covered an area of 9 sq. km., are 2 million tons; according to the other survey (2), which covers a larger area (about 50 sq. km.), the zone has a reserve of 22 million tons of workable coal.

In general the zone is much folded and faulted, with large areas covered by igneous rocks and rocks that are younger than the formation. Many superficial areas are also barren because they are covered by intrusive masses. There are several dikes and sills as well, with occasional sills of latite and quartziferous latite in the vicinity of the seams and adjacent to them, giving rise to the formation of natural coke.

With the structure described above, it has naturally been impossible to establish whether there is one deposit communicating below or whether there are several, since the coal-bearing zones appear in the form of patches, and it is difficult to determine their size and continuity.

As many as nine different seams have been found, with a thickness exceeding one metre, and there are numerous prospects and small mines which enabled representative samples to be taken of the quality of the coal. The dip of the layers varies according to the zone, most of the outcrops showing angles of 10 to 30°. There is no large mine, however. In order to confirm geological hypotheses, study the continuity of the seams and confirm the reserves, drilling has been planned but has not been carried out so far.

The Yaqui anthracites have the following general characteristics:

Moisture	*Volatile matter*	*Ash*	*Fixed carbon*	*BTU/lb.*
3 to 10%	2 to 8%	6 to 19%	72 to 85%	10,000 to 13,000

FUSIBILITY OF THE ASH

Sample No.[a]	*Temperature initial deformation °C.*	*Temperature softening °C.*	*Temperature melting °C.*
15	1,119	1,232	1,338
4	1,195	1,232	1,339
7	1,260	1,365	1,438
9	1,280	1,383	1,458
10	1,277	1,377	1,449
13	1,193	1,316	1,438
13	1,110	1,171	1,266
16	1,210	1,280	1,379
21	1,230	1,343	1,415
22	1,225	1,340	1,420
27	1,178	1,231	1,360

[a] The number of the sample refers to the survey (2), from which the analyses on page 46 were also taken.

obsolete or new, but are often inefficient (because of increased net production, or a higher ash content of the run-of-mine with the progress of mechanization); or to improve the utilization of the raw coal with a view to saving fuel by an adjustment of the type of fractions to be selected, of the sales conditions and of the apparatus utilizing coal.

GENERAL CONSIDERATIONS ON TERMINOLOGY

The problem to be examined at this meeting is necessarily sub-divided into:

(*a*) The comparison of processes already in use or being tested;

(*b*) The comparison of raw coals treated in a given installation with other raw coals to be treated;

(*c*) The final comparison of the results of preparation, that is, of an operation in which both the coal and the product intervene.

Before drawing conclusions, it is essential to indicate the characteristics selected, and the reason for the choice made.

Comparison of processes

The researches made by CERCHAR (Centre d'études et recherches des charbonnages de France) prove conclusively that each process is characterized by a coefficient which is independent of the coal and of the cut considered. This coefficient is imperfection grade I, which is related to the standard deviation E by the equation $I = \frac{E}{dp-1}$, dp being the separation density. For all apparatus of the same type, in good working condition, the real values of I represent a slight dispersion only. There is a definite industrial value in studying the results to be anticipated from a coal-preparation operation based on the average value adopted.

Discussion of the results of a cleaning operation

Here it is important to compare the actual result with the result of a perfect separation, in order to determine the organic efficiency. Errors can be made by those who consider the percentages of "tramps", which would be too variable with the proportion of middlings, for example, to have any meaning; these percentages are useful only for controlling the operation of a given installation.

Comparison between the washabilities of various coals

Accurate information on the results of any separation is obtainable from the washability curve, but attention has to be paid to the following:

(*a*) A glance at the three curve diagrams is sufficient to provide full knowledge of the products to be expected from a perfect cut;

(*b*) On the other hand, these curves give no indication as to the relative facility of the separation process.

Even the best-trained experts may make mistakes in seeking to obtain an idea of the organic yield to be expected from the treatment of a given coal in a given apparatus, by comparing its curve with that of another coal previously treated in the same apparatus, and particularly by comparing the aspects of these two curves near the separation density envisaged. It would thus be misleading to rely on the slope of the washability curve near the separation point; there are examples of quite different organic yields obtained at points where the slope of the curve was the same.

It is therefore necessary to draw a complete network of curves in order to learn the washing possibilities of a given coal in apparatus with different imperfections. These calculations can be made quickly with an integrating machine which CERCHAR built. Such a network of curves may be necessary for any discussion; a coefficient k[1] can be deduced from it which gives a general idea of the washability of a given coal. This coefficient could, however, be misleading in the event of the lightest elements of the raw coal not having the very low ash content generally prevailing in French coals.

The French terms used in coal washing are as follows:

(*a*) *Grains* are the particles $>$ 10 mm. (large size) (singles in Great Britain);

(*b*) *Fines* are the particles $<$ 10 mm. (small size) (peas in Great Britain);

(*c*) *Très fin* or *pulvérulent* are the fractions from 0 to 0.3 or 0.5 mm., or 1 mm. (fines or duff in English);

(*d*) *Fines dépoussiérées* are the $<$ 10 mm. freed from the fines (*pulvérulent*), that is, sizes ranging from 0.3, 0.5 or 1 mm. to 10 mm.

COAL WASHING IN FRANCE—PRELIMINARY REMARKS

The frame within which we operate—and which may be extended to any given country—may be traced by considering:

(*a*) The raw coals to be treated;

(*b*) The probable purchasers;

(*c*) The main economic features of the country.

The raw coals to be treated

France has insufficient coal and has to resort to imports, which are scarce and expensive. Competition, even prior to the existence of the European Coal and Steel Pool, has imposed the need for controlling the extraction cost and reducing it, but has little influence on any decision as to whether or not to work out some unfavourable raw coals.

French raw coals usually have an average ash content of 31%; the run of removed products (dumped shales) represent 27% of the mine and the range of washing difficulties extends from k = 3 to k = 30. Consequently coals are still being treated which could be brought down only to 30% ash (average percentage for the marketed coal) if a 90% organic yield were required in a jig having an imperfection grade I = 0.20. In fact, these values of k apply to the average having a given calibre and treated in a cleaning plant. Some of the seams which are combined there are certainly outside that range of figures.

Probable purchasers

It is very important to remember that French mines normally produce marketable coals and middlings.

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[1] The definition of that coefficient is: with every apparatus the organic yield increases the higher the fraction, that is, gives a dirtier washed coal. Given an apparatus with an imperfection of 0.20, the washed coal should have a minimum ash content k if the organic yield is to be equal to 90% or higher.

A selection of typical analyses is given below:

FREE FROM MOISTURE AND ASH

Sample No. A.S.T.M. classification	*Moisture %*	*Ash dry %*	*Volatile matter %*	*Fixed carbon %*	*BTU*	*C %*	*H %*	*N %*	*O %*	*S %*	*Specific gravity*	*Porosity* [a]
Anthracite......15	11.37	6.38	7.24	92.76	14,455	96.91	0.55	0.58	1.72	0.24	1,957	
Semi-anthracite.. 4	6.52	5.51	8.78	91.22	13,969	93.50	0.89	0.43	4.82	0.36	1,885	
Anthracite...... 7	5.52	23.25	7.03	92.97	13,951	93.88	0.64	0.62	4.31	0.55	1,931	
Natural coke.... 9	1.82	14.50	9.34	90.66	13,842	93.39	0.54	0.48	4.54	1.05	2,104	0.2609
Semi-anthracite..10	6.80	12.14	8.42	91.58	13,944	93.91	0.48	0.53	3.82	1.26	1,908	
Anthracite......13	5.08	9.72	5.51	94.49	14,511	94.77	1.31	0.35	3.22	0.35	1,828	
Natural coke....13	3.00	19.36	6.65	93.35	14,715	95.78	0.98	0.77	1.70	0.77	1,987	0.297
Semi-anthracite..16	3.47	13.18	13.34	86.66	13,048	88.79	0.91	0.41	9.46	0.43	1,989	
Anthracite......21	5.62	10.52	7.52	92.48	14,052	94.27	0.73	0.36	4.19	0.45	1,963	
Semi-anthracite..22	3.71	14.55	9.72	90.28	13,822	92.28	0.84	0.73	5.71	0.44	1,959	
Anthracite......23	7.10	14.67	6.94	93.06	14,434	95.60	0.85	0.39	2.49	0.67	1,899	

[a] Porosity = real specific gravity — apparent specific gravity.

It can be tentatively stated (1) that the Yaqui coal can be used as fuel for boilers equipped with suitable grates and hearths. In order to reach a definite conclusion, a test using about six railway cars of coal was planned in a steam plant, equipped with adequate grates; unfortunately control of the test was deficient.

Successful laboratory tests have been made with a view to producing briquettes with anthracite fines from Sonora, to be used as domestic fuel. The briquettes had sufficient physical strength but were difficult to ignite.

Some Yaqui coal shipped to Guadalajara has been used to make calcium carbide, the low-volatile content making it excellent for this purpose. An attempt was also made at the beginning of the century to use this coal in locomotives of the South Pacific network, but it was found unsuitable for the type of fire boxes, which had been designed for bituminous coal.

Some years ago Yaqui coals were used in the copper smelter in Boleo, Lower California. The coal used, with 87% of fixed carbon and 10% ash, a heating value of 14,000 BTU (dry) with 4 to 5% graphitic material, gave no difficulty in reverberatory furnaces. This coal has also been used mixed with fuel oil in a cement furnace in the city of Hermosillo.

Sonora anthracites have also been thought of as a blast furnace fuel. Use of these fuels for iron ore reduction could be encouraged as there are excellent iron ore deposits, with large reserves, in the state of Sonora.

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Description of Chilean Coals Used in the Steel Industry

ALFREDO GONZÁLEZ

The coal used in the Chilean steel industry is derived from the Lota and Schwager mines, located in the Gulf of Arauco, some 43 km. to the south of the Steel Plant of San Vicente (Huachipato) which uses more than 200,000 tons a year of this coal for coke production.

DESCRIPTION OF THE MINES

The coal beds worked at Lota and Schwager correspond to synchronic deposits. There are no less than nine coal veins in the deposits, of which only four are workable and these not uncovered in all faces so far.

In Lota the workable seams are, from the top to the bottom: the Manto Arriba of 1.20m.; the Manto Chico of 1 m. and the Manto Alto of 2 m. Four independent mines are worked here at present, each being isolated from the others by a barrier of from 50 to 100 m. thick. All the coal faces are under the sea, below the Gulf of Arauco. The uppermost exploited seam is approximately 400 metres below sea level, and is 1.20 to 1.40 m. thick. Some 40 m. below it is the Manto Chico with a thickness of from 1 to 1.10 m. Manto Alto, the third one, is 10 metres below Chico and almost 2 metres thick. In none of the four Lota mines are more than two of these three seams workable.

These same veins are worked in the Schwager Company's concession, which lies on the northern side adjacent to Lota. In this case, Lota's Arriba and Alto seams are called No. 3 and No. 5 respectively; the Manto Chico is too thin to be worked in Schwager; on the other hand, No. 2, which in Lota is rarely found on a commercial scale, is occasionally worked in Schwager and called the Manto Nuevo.

According to United States coal classifications, made by the Bureau of Mines,[1] this is "High Volatile

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[1] Report of Investigation 3296.

A bituminous coal" because it contains less than 69% of fixed carbon, over 31% of volatile matter and a heating value exceeding 14,000 BTU. This coal is fragile, has a bright aspect, and there is no cleavage in the seams.

The roof of these veins is soft and sometimes sandy shale, requiring extensive timbering. Bottoms are fireclay, subject to rapid swelling in the moist atmosphere.

The average dip ranges from 18° in Schwager to 15° in Lota, in a westerly direction.

Gas is liberated at the working faces and ventilation is adjusted to maintain about 1% of methane on the main returns.

RESERVES

According to surveys of the Lota and Schwager reserves made by the mining engineer, Mr. J. W. Woomer, the following proved and probable coal reserves have been established at the different concessions of these companies:

	Tons
Lota: Present workings	38,750,000
Future workings	38,250,000
	77,000,000

Of these reserves, 80% are coking coals and correspond to Manto Alto or No. 5. The tonnage included under "present workings reserves" comprises all the coal located at less than 500 m. below sea level. The tonnage for future workings includes that part of the deposits comprised between 500 and 1,000 m. below sea level, to be worked by means of projected new shafts.

	Tons
Schwager: Present workings	6,100,000
Future workings	39,000,000
Buen Retiro [a]	22,900,000
	68,000,000

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[a] The Buen Retiro concession belongs to the Lota Company, but negotiations for its transfer are currently taking place.

Almost all this tonnage is derived from beds Nos. 3 and 5 in equal proportions, so that the coking coal represents 50%.

The 6,100,000 tons comprise the reserves down to 750 m. below sea level; the future workings refer to the coal found from 750 to 1,200 m. below sea level, for the exploitation of which a new shaft is being sunk.

Over 60 sq. km. of submarine area have been worked in the Bay of Arauco since the opening of these mines, one hundred years ago.

As regards additional possible coal reserves, Mr. Woomer gives the following figures:

	Tons
Compañía de Lota	10,000,000
Compañía de Acero del Pacífico	150,000,000
Lebu Company	100,000,000
TOTAL	260,000,000

COAL PRODUCTION

Coal production in Chile during the last ten years, expressed in thousands of tons, is as follows:

Year	*Lota*	*Schwager*	*Other mines*	*Total*
1942	901	627	619	2,147
1943	947	584	734	2,265
1944	985	668	627	2,280
1945	881	771	428	2,080
1946	825	655	484	1,964
1947	849	609	508	2,066
1948	965	756	546	2,267
1949	969	709	460	2,138
1950	1,023	752	442	2,217
1951	1,007	714	489	2,210

A plan is being developed for modernizing the Lota collieries, tending to combine exploitation in one single unit, thus simplifying and reducing the cost of internal transport, ventilation, etc. Output would be expanded to 5,000 tons a day in three shifts, or 4,000 tons a day with two shifts, as against the present production of 3,500 tons a day. As part of this programme, two shafts have been driven to a depth of over 500 m. and the main and return entries have been advanced to a length of 5,400 m. over sterile ground. The main gangways in the Alto and Chico Mantos have also been developed.

The Schwager Company is also modernizing its mines. Eventual production is to be 4,000 tons a day, as compared with the present 2,600 tons.

The two new shafts contemplated in this plan will reach a depth of 915 m. From these, three entries over 3 km. in length are to be drilled in rock.

COAL FOR STEELMAKING

The Chilean steel industry uses in its Huachipato plant only the coals from the Manto Alto at Lota and the No. 5 at Schwager, which currently represent respectively, in round figures, 60% and 50% of total production.

Production in 1950, in metric tons per seam, was:

	Arriba or No. 3	*Chico or No. 4*	*Alto or No. 5*	*Total*
Lota	310,000	92,000	621,000	1,023,000
Schwager	387,000		365,000	752,000

Coal suitable for the national steel plant...986,000
Huachipato uses less than a quarter of this amount at present.

The sulfur content of the other seams (over 3%) is considered too high for this industry.

According to samples taken by the United States Bureau of Mines, the proportion of ash and sulfur in samples of different sizes, is as follows:

Size	*Weight %*	*Ash %*	*Sulfur %*
Manto Alto, Lota			
1½" - ⅜"	71.24	2.72	0.73
⅜" - 48 mesh	27.13	3.56	0.77
- 48 mesh	1.63	12.62	1.03
	100.00		
No. 5, Schwager			
1½" - ⅜"	59.07	4.67	0.94
⅜" - 48 mesh	38.60	5.49	0.80
- 48 mesh	2.33	12.93	1.08
	100.00		

CHARACTERISTICS OF THE COAL

The results obtained from float-and-sink tests for the Schwager and Lota coals, using 135 and 166 S.G. liquids, are as follows:

Schwager No. 5 bed

	Analysis of dry sample				*Cumulative*		
Sample	*Ash %*	*Sulfur %*	*Weight Kg.*	*Weight %*	*Weight %*	*Ash %*	*Sulfur %*
Arauco							
¾″ - ¼″	14.9	1.30	12.99				
F 1.35	4.0	1.04	10.33	35.94	35.94	4.0	1.04
1.35 - 1.6	18.3	1.62	1.24	4.31	40.25	5.5	1.42
S 1.6	73.2	2.12	1.42	4.94	45.19	12.9	1.50
¼″ - 28 mesh			12.78				
F 1.35	3.1	0.83	9.94	34.59	79.78	8.7	1.21
1.35 - 1.6	19.3	3.06	0.85	2.96	82.74	9.0	1.28
S 1.6	72.6	1.59	1.99	6.92	89.66	14.0	1.30
28 - 48 mesh			1.37				
F 1.35	2.3	0.71	.95	3.31	92.97	13.5	1.28
1.35 - 1.6	18.6	1.81	.07	.24	93.21	13.6	1.28
S 1.6	71.8	1.42	.35	1.22	94.43	14.3	1.28
— 48 mesh	27.6	1.07	1.60	5.57		15.0	1.27
TOTAL			28.74		100.0		
Envidia							
¾″ - ¼″	11.5	0.99	13.48				
F 1.35	2.9	0.86	11.84	44.39	44.39	2.9	0.86
1.35 - 1.6	9.9	1.42	0.39	1.46	45.85	3.1	0.88
S 1.6	78.0	1.68	1.25	4.69	50.54	10.1	0.95
¼″ - 28 mesh			10.69				
F 1.35	2.0	0.76	9.14	34.27	93.81	10.0	0.92
1.35 - 1.6	17.8	1.43	.05	.19	94.00	10.0	0.92
S 1.6	70.6	2.08	.16	.60	94.60	10.4	0.92
— 48 mesh	19.1	1.12	1.44	5.40	100.0	10.8	0.94
TOTAL			26.67				
Manto Alto, Lota							
New shaft							
1″ - ¼″	16.1	0.70	32.25				
F 1.35	2.4	0.70	27.59	47.41	47.41	2.4	0.70
1.35 - 1.6	21.2	1.37	.63	1.08	48.49	2.8	0.72
S 1.6	75.3	1.73	4.03	8.92	55.41	11.9	0.84
¼″ - 28 mesh			21.57				
F 1.35	2.2	0.65	15.42	26.49	81.90	8.7	0.78
1.35 - 1.6	18.4	1.38	0.71	1.22	83.12	8.9	0.79
S 1.6	74.8	0.34	5.44	9.35	92.47	15.5	0.74
— 48 mesh	44.2	0.34	4.38	7.53		17.7	0.74
TOTAL		58.20			100.0		

WASHING PLANTS

LOTA: In Lota, coal preparation takes place in two sections:

(a) *Screening plant.* Coal is sent from the mines to a central screening plant. This consists of three conventional screens working in parallel and followed by hand-picking tables plus the corresponding bins for loading railway cars. Hourly capacity of each screen is 200 tons when screening over 100 mm., and 50 tons when screening over 23 mm. One of these screens is used exclusively for treating the coal previously selected for metallurgical purposes.

At present screening is done at 23 mm. round.

(b) *Washing of under 23 mm. coal.* This plant uses a Baum jig, with an hourly capacity of 25 to 30 tons under 23 mm. feed. Ash content is reduced from 20 to 5% without substantial losses because this coal has few middlings (bone).

The modernization plan includes a washing plant for treating run-of-mine coal in two sections, one of which will handle exclusively the coal selected for coking.

Run-of-mine coal of 0-350 mm. and about 20% ash, will be produced at a rate of 400 tons per hour and separated in a Baum screen with 178 mm. mesh. The oversize (178-350 mm.) to the amount of 24 TPH will pass to the picking table. The balance of 376 TPH of 0-178 mm. will be separated at 3 mm. size. The fraction of 3-178 mm. size, in the amount of 308 TPH will pass to the Baum jig where its ash content will be reduced from 20 to 6%. This clean fuel is submitted to its final classification through a 23 mm. dewatering screen, which will separate the lump coal from fines under 23 mm. and dewater the material until there is only 4% of surface moisture left.

The fine 0.3 mm., which will amount to 68 TPH, will not be washed, but will be sold with 20% ash or added to washed 3-23 mm. size, yielding a blend having 11% ash altogether.

SCHWAGER. Coal is here classified in vibrating screens. Run-of-mine coal is first screened at 100 mm., the oversize is then hand picked. The—100 m. product is classified successively in sizes of 100 x 50 mm., 50 x 19 mm. and 19 x 0 mm. This last size is not washed. The other two fractions are cleaned in a dry-cleaning plant using plate-glass sheets to separate rock intrusions from the coal.

This equipment makes use of the difference in elasticity between coal and rock. The coal and the rock particles which fall from the shaker screens are collected by slightly curved sheets of glass placed 0.10 m. below the bottom of these screens. When this material touches the glass, it bounces and jumps outside. The coal is thrown farther away than the rock on account of its greater elasticity. The conveyor which collects coal is some 0.10 m. from the edge of the glass. The rock is collected in another conveyor placed immediately beyond the glass. The glass plate is kept dustfree by a pressurized air current.

For the new development it is planned to replace the present preparation plant by a more suitable and more economical one. Its capacity is planned at 480 tons of crude coal per hour. Here, as in Lota, the slack under 3 mm., will be separated and sold without cleaning.

REQUIREMENTS OF THE COMPAÑÍA DE ACERO DEL PACÍFICO

The requirements for quality of coal are:

Specifications	*Lota*	*Schwager*
Moisture	4.0% max.	4.0% max.
Ash	7.0% "	7.0% "
Sulfur	0.9% "	1.4% "

Using the Geiseles plastometer, values vary as follows:

Initial melting in °C	370/380	330/350
0.10 divisions/minute °C	390/400	370/380
Maximum fluidity °C	425/435	430/440
Solidification point °C	450/456	450/465
Maximum fluidity div. minute	3/7	8/20

SAMPLING

Only the coal of certain pre-selected faces is used for coking, and during operation mine cars produced from these faces are sampled as follows:

Lota: Sampling is done from railway cars, three lots of 5 kg. each being taken per car (40 to 45 tons).

Schwager: Samples are taken off the conveyor-belt which feeds the cars, and are composed of two lots of 10 kg. each per car.

Slack from Lota and Schwager: Samples are taken from the cars while discharging at the steel plant, taking three lots of 2 kg. per car.

Problems Related to Colombian Coal Used for Steelmaking

JOAQUÍN PRIETO ISAZA, JAIME LÓPEZ T, BENJAMÍN ALVARADO AND VICENTE SUÁREZ HOYOS

INTRODUCTION

The Technical Department of the Empresa Siderúrgica Nacional de Paz de Río S.A., has engaged in extensive research on the properties of the coals in the Paz de Río-Sogamoso region (1, 2, 3), with a view to using them in the Belencito steel mill which is currently under construction. The coking properties of these coals have been discussed in another paper, so that this one will be confined to washability properties. (4)

LOCATION OF THE COALS AND RESERVES

The location of the coal deposits in the Paz de Río-Sogamoso region is shown on the map (Figure 1). Some 1,920 million tons of probable reserves have been found within a 40 km. radius of the Belencito plant (1, 2, 3). This is therefore an extensive coal-bearing region within the formation of Colombia's eastern cordillera, which extends from 60 km. south of Bogotá right to the frontier with Venezuela—some 600 km.—in which some 20,000 million tons of coal have been estimated so far. (5) This coal field thus contains Colombia's greatest reserves, since the estimated total for the country is 40,000 million tons. (5)

Within the area, there were naturally a few deposits offering greater accessibility, better working conditions and higher grade coals for steelmaking purposes; the one finally selected for working was La Chapa, some 36 km. by railroad from Belencito. This deposit, which is a synclinal opening out towards the west, is estimated to have probable reserves of 200 million tons, only some 15 million of which are currently being prepared for mining by the Empresa Siderúrgica.

ORIGIN AND QUALITY OF THE COALS

Colombian coals are usually drift coals originating from vegetable matter which did not grow in the places where the deposits are found, and therefore vary considerably in ash and sulfur content. Furthermore, they, like most South American coals, are of recent origin, so that their coking properties may vary, (6) according to the geological pressure and heat conditions to which they were subjected, (6) added to the fact that there was probably greater diversity in the vegetable matter forming the coals than in other regions such as the Appalachian Mountains in the United States. The vegetable matter grew on the slopes of the hills and was dragged into the beds of ancient rivers or lakes. Most of the Colombian coals investigated do not have partitions of refractory clay, as in the case of all coals formed *in situ.* Certain amounts of sand and mud were carried off with the coal, so that relatively high and variable ash contents are found, as well as sulfur, derived from organic matter, from sulfides dissolved in the waters where the material was deposited, or from pyrites and marcasites carried off simultaneously.

La Chapa coals are of the Upper Cretaceous age, distributed in nine seams ranging from 0.50 to 4 m.

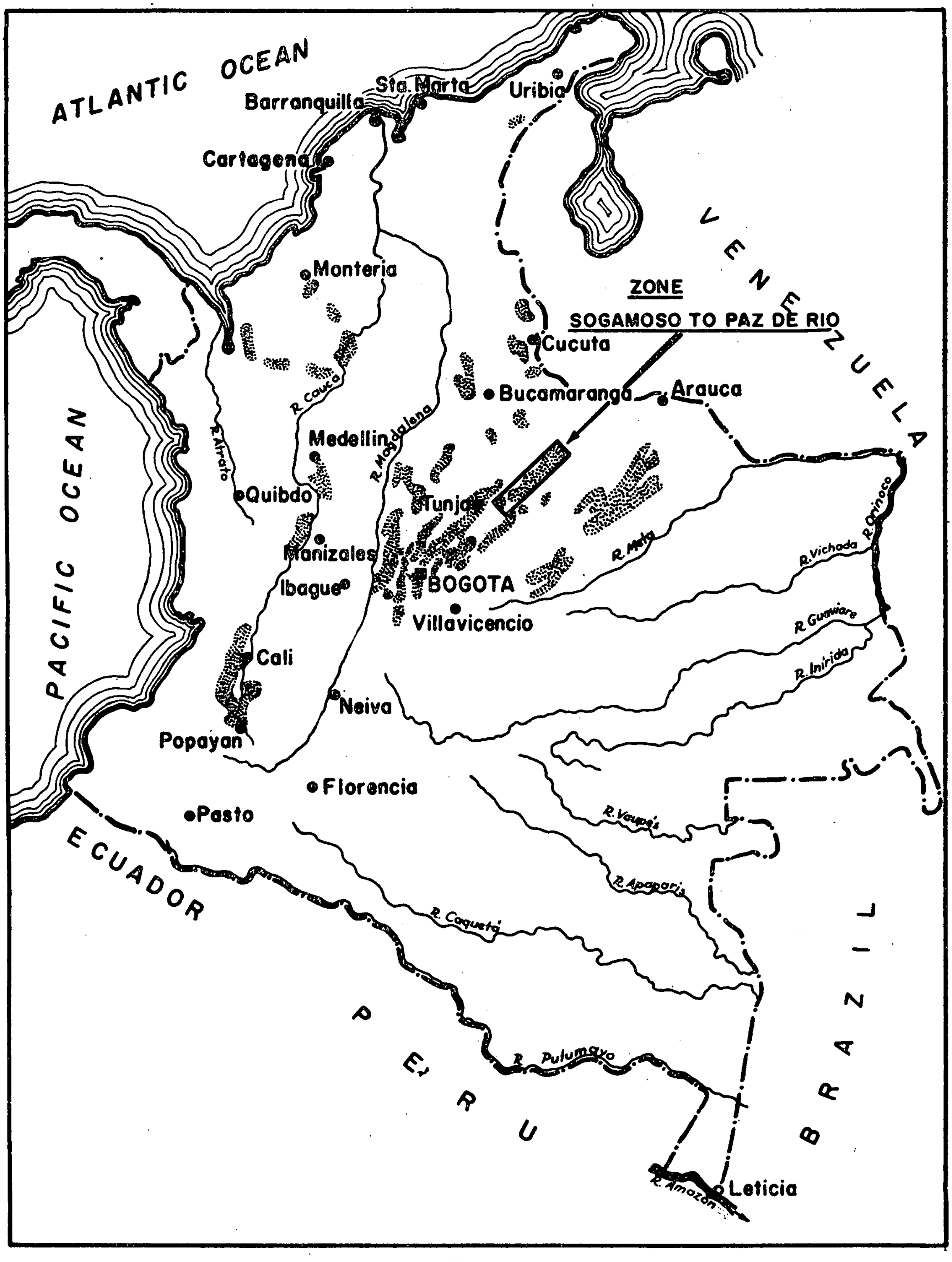


Figure 1

PRINCIPAL COALBEARING ZONES OF COLOMBIA

Total reserves:	40,000 million tons
Reserves in the eastern cordillera:	20,000 million tons
Reserves in the Sogamoso-Paz de Río zone	1,920 million tons

thick, within a geological stratum of 250 m. and separated by partitions of slatey clays and some banks of sandstone. The coal is bituminous, bright, clean-looking, but with an ash content which may be as high as 15% or even 19% in some exceptional cases, although the average is 12-13%. The sulfur content, which averages 1.5%, is mainly due to iron sulfides, and can therefore be considerably reduced by washing, as described later on. According to A.S.T.M. classification, these are High Volatile A coals. (7) Volatile matter averages 35%, which is slightly lower than in the coal formation continuing to the south, and higher than the coals in the northern region.

The coal seams, within this local structure, have inclinations ranging from 38 to 15°, but usually they are inclined seams with dips of 20 to 30°, so that working conditions may be compared with those frequently found in the states of West Virginia and Washington, in the United States.

COAL PREPARATION

The incentive for cleaning metallurgical coals is the lower cost per ton of pig iron, due to the greater uniformity of the coke and the lower ash and sulfur contents. Furthermore, bearing in mind the nature and origin of Colombian coals and the considerable variations which may arise in the contents of ash and sulfur, washing is obviously indicated from every point of view.

SAMPLES

In each seam three of the more advanced coal faces offering the best sampling conditions were selected for taking samples; no appreciable meteorization was considered to have taken place therein. The galleries were advanced some ten metres in order to take the samples on new faces; one ton was taken from each, or a total sample of three tons for each seam. The samples were then reduced to one ton and packed in sealed metal drums for despatch to the laboratory.

Samples were also taken by making 2 in. by 2 in. pillars in the working face, with a view to determining the different components and their thicknesses in the seam. Each sample was subjected to a flotation test in liquids with densities ranging from 1.30-1.60, and the fractions which floated on each liquid were combined and an immediate analysis was made.

EXPERIMENTAL WORK

The float-and-sink test has been frequently described in the literature, (8) and is currently the standard method used in the coal industry for determining the washing properties of the material, both to examine the products of plants in operation and for assessing their effects.

It consists in submerging the coal in a series of dense liquids, with a specific scale of densities, so that the two separate fractions can be examined.

1: *Flotation analysis with pillar samples*

As an example of the tests made with samples taken in a channel the length of a seam in 2 in. by 2 in. pillars, the results of the tests made on Seam No. 1 of La Chapa are given in Tables 1 and 2. The first indicates the stratigraphic pillar of the seam, with the various components found therein, their thicknesses, the classification and densities at which they float or sink. The total thickness of the seam in this case was 3.30 m., and throughout its length only a 5 cm. belt of slatey clay was found, so that the coal is clean. It has a bright aspect and the density of 1.35, below which most of the material floats, is another proof of the cleanness of this coal.

Table 2 shows the immediate analyses of the various columnar fractions which floated at a specific density,

Table 1

PILLAR SAMPLE—LA CHAPA, SEAM I

Thickness metres	*Cumulative thickness*	*No.*	*Classification*	*Densities*	
				Float	*Sink*
0.1397	—	1	Splint coal	1.45	—
0.0381	0.1778	2	Bone coal		1.6
0.1397	0.3175	3	Dull coal	1.4	—
0.0508	0.3683	4	Bright coal	1.35	—
0.1270	0.4953	5	Splint coal	1.35	—
0.2921	0.7875	6	Bright coal	1.35	—
0.0508	0.8382	7	Slate	—	1.6
0.2794	1.1176	8	Banded	1.3	—
0.2286	1.3462	9	Bright coal	1.35	—
0.3810	1.7272	10	Bright coal	1.3	—
0.3937	2.1209	11	Banded	1.3	—
0.3556	2.4765	12	Bright coal	1.3	—
0.1524	2.6289	13	Splint coal	1.3	—
0.2794	2.9083	14	Bright coal	1.35	—
0.3667	3.1750	15	Bright coal	1.3	—
0.6350	3.8100	16	Banded	1.35	—

Table 2

PROXIMATE ANALYSIS FOR ALL FRACTIONS FLOATED

Density of flotation	*Per cent*					*Button*
	H_2O	*Vol. matter*	*Fixed carbon*	*Ash*	*Sulfur*	
Fractions floated at:						
1.30	0.96	37.18	58.54	3.32	0.90	Silver grey, very swollen, spongy
1.35	0.84	33.30	58.14	7.72	0.87	Grey, swollen, spongy
1.40	0.83	31.41	49.18	18.58	1.97	Grey, hard, somewhat swollen, average size
1.45	0.76	29.50	46.00	23.74	1.35	Grey, somewhat swollen, brittle, spongy
1.50	0.80	28.76	40.70	29.74	0.90	Grey, cracked, hard, average size
1.60	1.80	25.80	38.18	34.22	0.90	Black, weak, fine grain
Fractions which sank at:						
1.30	0.65	29.53	54.44	15.38	0.73	Grey, weak, spongy
1.35	0.84	28.50	53.80	16.86	0.64	Grey, somewhat hard, fine grain
1.50	1.08	20.98	27.76	50.18	0.50	Agglomerated dust
1.60	1.23	16.25	22.68	59.84	0.53	Agglomerated dust

once combined in one single sample; this provides information on the ash and sulfur contents of each component.

From the various float-and-sink tests conducted on pillar samples in the different seams, it was concluded that the La Chapa coals, as a whole, are mainly coking, their petrographic components ranging from vitrain to durain, with ash contents reaching 15%, largely due to fine veins of slatey shales which break the continuity of the seam.

These clay veins can be eliminated quite easily by washing, as shown by the results given in Table 3, also referring to Seam No. 1, illustrating what happens with these coals which are, therefore, relatively clean.

2. *Washability tests and curves*

(1) Coal classification

Normal flotation (float-and-sink) tests to determine the washability curves were made with the one-ton samples received in sealed drums by the laboratory, and derived from the three-ton samples taken from each seam.

The samples were classified as indicated in Tables 1, 2 and 3, in fractions of: over 4 in.; 4 in.–3 in.; 3 in.–2 in.; 2 in.–1½ in.; 1½ in.–1 in.; 1 in.–½ in.; ½ in.–¼ in.; and ¼ in.–0 in.

Flotation tests at a density of 1.45 were made with each one, and at the same time the content of volatile matter and ashes was determined to discover the size of the greatest proportion thereof. The same determinations were made in the blend left after combining the various "floats" and in the cumulative waste. These data are presented in Tables 3 and 4 for seams I and VI of La Chapa, as representative examples.

Table 3

Screening and cleaning of coal

La Chapa—Seam No. 1

Size	*Screening*			*Washing density = 1.45*				*Analysis*					
	Raw coal			*Coal*		*Refuse*		*Raw coal*		*Washed coal*		*Refuse*	
	Weight	*% C>¼"*	*% Total*	*Weight*	*% Screened*	*Weight*	*% Screened*	*V.M.*	*Ash*	*V.M.*	*Ash*	*V.M.*	*Ash*
	Kg.			Kg.		Kg.							
Over 4"	38,440	6.31	4.48					37.22	2.34				
4"–3"	38,410	6.30	4.47					37.74	2.64				
3"–2"	79,915	13.12	9.31					38.04	6.66				
2"–1½"	71,525	11.74	8.33					35.60	6.96				
1½"–1"	97,040	15.93	11.30					34.92	9.00				
1"–½"	148,395	24.35	17.28					35.50	8.84				
½"–¼"	135,550	22.25	15.79					35.76	8.32				
Over ¼"	609,275	100	70.96	569,265	93.43	40,010	6.57	36.06	7.44	34.84	6.32	25.82	41.46
¼"–0"	249,335		29.04	235,024	94.25	14,331	5.75	34.54	7.94	33.20	5.58	23.62	46.06
Total	858,630		100	804,289	93.67	54,341	6.33	35.62	7.59	34.35	6.10	25.24	42.67

Table 4

Screening and cleaning of coal

La Chapa—Seam No. VI

Size	*Screening*			*Washing density = 1.45*				*Analysis*					
	Raw coal			*Coal*		*Refuse*		*Raw coal*		*Washed coal*		*Refuse*	
	Weight	*% C>¼"*	*% Total*	*Weight*	*% Screened*	*Weight*	*% Screened*	*V.M.*	*Ash*	*V.M.*	*Ash*	*V.M.*	*Ash*
	Kg.			Kg.		Kg.							
Over 4"	57,285	7.76	6.42					33.64	8.36				
4"–3"	106,185	14.38	11.89					30.46	15.06				
3"–2"	143,105	19.38	16.03					32.76	19.94				
2"–1½"	98,135	13.29	10.99					32.14	20.46				
1½"–1"	99,940	13.54	11.19					34.88	10.10				
1"–½"	133,610	18.09	14.97					34.74	10.90				
½"–¼"	100,160	13.56	11.22					36.02	9.48				
Over ¼"	738,420	100	82.71	680,035	92.09	58,385	7.91	33.50	14.02	37.28	8.95	26.32	41.50
¼"–0"	154,335		17.29	136,140	88.21	18,195	11.79	34.28	14.20	37.24	7.34	22.00	49.54
Total	892,755		100	816,175		76,580		33.63	14.05	37.29	8.68	25.45	43.41

(2) Washability curves

Using the one-ton samples, the fraction exceeding 3 in. was crushed until all the coal was in the 3 in.–0 in. size, and the sample was classified once more to obtain fractions of 3 in.–1½ in.; 1½ in.–½ in.; ½ in.–¼ in.; and ¼ in.–48 mesh. Following standard procedure, representative samples of these fractions were taken at once, weighing 50 kg. each, and with this amount the corresponding float-and-sink tests were made in liquids of the following densities: 1.30, 1.40, 1.45, 1.50, 1.60, 1.80 and 2.00.

The results are shown in the respective washability curves, the extreme variations of which appear in Figures 2 to 4.

If a metallurgical coke is required with an ash content of about 12%, these coals will have to be washed down to 8-9% ash. From the washability curves, this seems possible in washing them to a density of 1.45, which case the yields exceed 85%.

It also appears from the washability curves that these coals can be washed satisfactorily by crushing them to 3 in. and ¼ in., separating the ¼ in.–0 in. fraction, which is advisable from the washing properties of those fines. In order to verify this, approximately one ton of samples was crushed to sizes between 3 in. and ¼ in. and subjected to the float-and-sink test at a density of 1.45, while the ¼ in.–0 in. fraction was similarly tested. The content of volatile matter, ash and sulfur was also determined for the raw coal as it left the mine, for the washed coal and the corresponding refuse. The results thereof are shown in Table 5, from which it may be concluded that it is feasible to obtain an ash content below 9%, on the basis of blends of the coals from the different seams in the deposit, classified to 3 in.–¼ in. Table 6 shows washability yields for the coal above ¼ in. or below ¼ in. for the different seams. These results show that according to the washability curves, it is possible to obtain yields exceeding 85% in the wash, with satisfactory ash and sulfur contents.

Table 5

LA CHAPA — ANALYSIS

(*Percentage*)

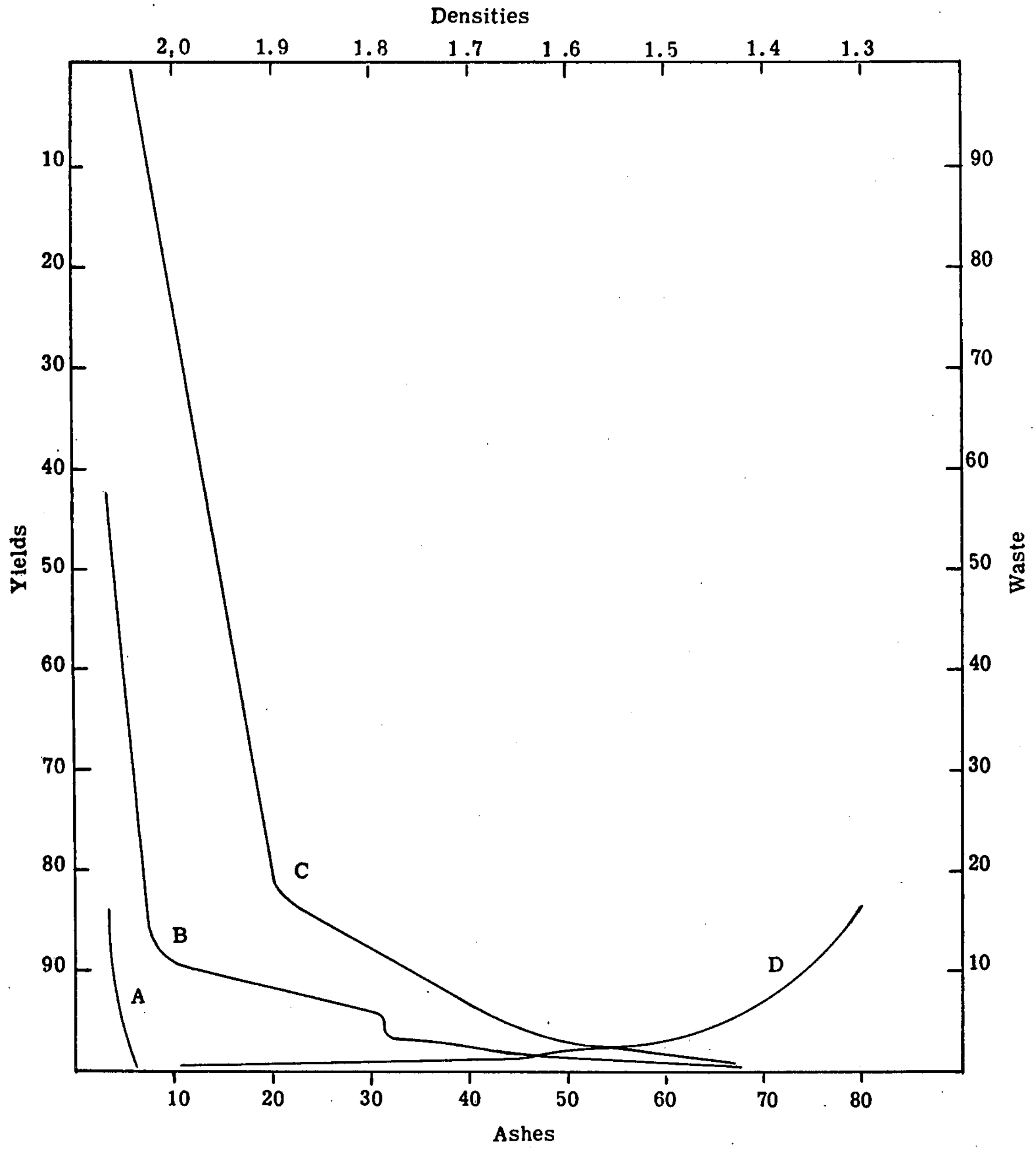
Samples	*Raw coal*			*Washed coal*			*Refuse*		
	Volatile matter	*Ash*	*Sulfur*	*Volatile matter*	*Ash*	*Sulfur*	*Volatile matter*	*Ash*	*Sulfur*
Seam I									
Over ¼″..........	32.63	14.97	1.43	34.88	6.27	1.00	21.42	58.35	3.58
Under ¼″.........	31.16	14.62	1.44	33.78	7.48	1.02	15.84	56.40	3.92
Seam IV									
Over ¼″..........	33.67	13.62	1.80	36.20	8.76	1.43	22.21	50.17	6.14
Under ¼″.........	33.68	16.14	2.32	35.47	9.08	1.29	21.65	48.18	4.76
Seam V									
Over ¼″..........	35.67	15.97	2.57	38.18	10.08	1.50	27.22	35.78	5.37
Under ¼″.........	35.89	15.58	1.82	37.91	10.28	1.41	27.81	36.80	3.47
Seam VI									
Over ¼″..........	33.92	14.08	1.00	35.53	9.54	0.89	21.61	48.89	2.58
Under ¼″.........	29.81	13.79	1.04	31.64	8.53	0.87	20.15	43.44	2.23
Seam VII									
Over ¼″..........	35.58	14.59	1.20	37.09	10.72	1.12	23.67	45.10	1.87
Under ¼″.........	35.06	19.08	1.25	37.02	13.98	1.18	23.34	47.00	1.66

Table 6

WASHABILITY YIELD

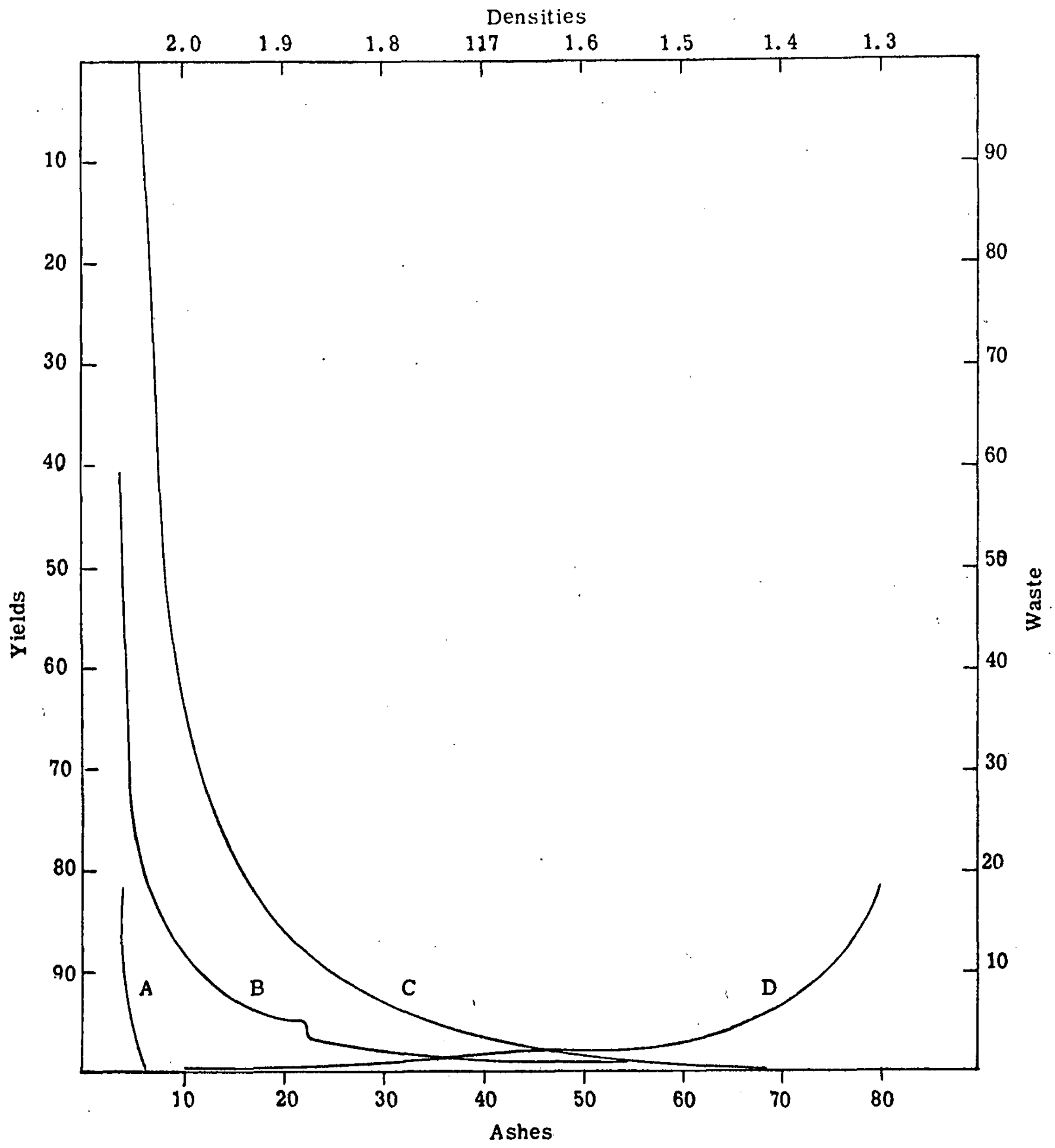
La Chapa — Seams Nos. I, IV, V, VI and VII

Samples	*Raw coal*		*Washed coal*		*Refuse*	
	Weight	*Per cent*	*Weight*	*Per cent*	*Weight*	*Per cent*
Seam I: 2,750 kg.						
Over ¼″........................	1,712,225	62.26	1,437,242	83.94	274,983	16.06
Under ¼″......................	1,038,055	37.74	886,499	85.40	151,556	14.60
Seam IV: 2,255 kg.						
Over ¼″........................	1,803,460	79.94	1,587,765	88.04	215,695	11.96
Under ¼″......................	452,475	20.06	393,925	87.06	58,450	12.94
Seam V: 837 kg.						
Over ¼″........................	623,345	74.41	480,474	77.08	142,871	22.92
Under ¼″......................	214,365	25.59	171,492	80.00	42,873	20.00
Seam VI: 2,626 kg.						
Over ¼″........................	1,805,275	68.74	1,603,630	88.83	201,645	11.17
Under ¼″......................	820,930	31.26	697,216	84.93	123,714	15.07
Seam VII: 1,884 kg.						
Over ¼″........................	1,473,880	78.22	1,307,774	88.73	166,106	11.27
Under ¼″......................	410,510	21.78	347,045	84.54	63,465	15.46



Density	Primary Data		Cumulative Data			
	Percentage		Percentage that floats		Percentage that sinks	
	Weight	Ash	Weight	Ash	Weight	Ash
Over 1.30	81.47	3.88	81.47	3.88	100.00	6.32
" 1.40	12.23	9.38	93.70	4.60	18.53	17.10
" 1.45	2.26	22.12	95.96	5.01	6.30	32.08
" 1.50	1.31	22.84	97.27	5.25	4.04	37.65
" 1.60	0.82	30.80	98.09	5.46	2.73	44.76
" 1.80	1.24	45.00	99.33	5.95	1.91	50.76
" 2.00	0.57	60.16	99.90	6.26	0.67	61.43
Below 2.00	0.10	68.68	100.00	6.32	0.10	68.68

Figure 2: DENSITIES
WASHABILITY CURVES OF COAL FROM SEAM I, "LA CHAPA"
Size between 3"–1½"
A = Washability curve C = Refuse curve
B = Characteristic curve D = Densities curve



Density	Primary Data		Cumulative Data			
	Percentage		Percentage that floats		Percentage that sinks	
	Weight	Ash	Weight	Ash	Weight	Ash
Over 1.30	83.72	3.32	83.72	3.32	100.00	6.39
" 1.40	9.34	9.62	93.06	3.95	16.28	22.14
" 1.45	2.82	31.06	95.88	4.75	6.94	38.99
" 1.50	1.60	31.72	97.48	5.19	4.12	44.42
" 1.60	0.60	39.76	98.08	5.40	2.52	52.48
" 1.80	0.64	42.38	98.72	5.64	1.92	56.45
" 2.00	0.50	56.26	99.22	5.90	1.28	63.48
Below 2.00	0.78	68.10	100.00	6.39	0.78	68.10

Figure 3: DENSITIES
WASHABILITY CURVES OF COAL FROM SEAM I, "LA CHAPA"
Size between ¼"—48 mesh
A = Washability curve C = Refuse curve
B = Characteristic curve D = Densities curve

(*a*) The mines are systematically equipped with power plants capable of burning middlings, average ash content of which exceeds 40% and sometimes reaches 50%, yet giving an excellent yield (for all modern plants about 800 gr. of these low products or 3,500 calories are consumed per net kWh at the power station);

(*b*) The consumer of marketable products may generally burn fuel relatively high in ash:

  (i) Domestic hearths accept a higher ash percentage, if constant; i.e. the same throughout, and not due to stones which cannot burn. Therefore, besides screened products with 7 to 8% ashes, there is a heavy consumption of ovoids at 18 to 20% ash obtained from pitch-bound, very unclean coals;

  (ii) Commercial hearths consume products at 8 to 18% according to use. A heavy tonnage is made up of semi-washed fines, which usually bear 13% ash;

  (iii) French metallurgical coke, on the other hand, should be sufficiently pure owing to the nature of French ores. The coking fines must not exceed 8% ash content if possible.

The economic features of the country

The production cost of a net ton of French coal is about 4,750 francs, of which about 300 francs represent cost of preparation, while 2,320 francs correspond to manpower. It corresponds to a production of 900 kg. per shift and a manpower cost (including wages and social security costs) of 2,090 francs per shift.

Comparing manpower costs and those of industrial products in France, it may be said that:

(*a*) A ton of steel girders costs about 40,000 francs;

(*b*) A cubic metre of reinforced concrete in place costs about 35,000 francs;

(*c*) The cost (wages plus social charges) of a highly skilled worker is about 350 francs per hour.

Finally, when comparing the heavy-media processes with others, it should be remembered that magnetite is relatively scarce in France, and costs about 11,000 francs a ton, crushed and delivered to the cleaning plant.

Evolution of these factors

Efforts are being made to mitigate some defects in these peculiarities of French mines.

An endeavour is being made to simplify the range of marketable products. The investments for French cleaning plants are still far too much overburdened by the multiplication of the separate devices, recycling, mixing apparatus, and so forth, which of course allow for a great variety of sales, but are not generally beneficial from the point of view of a well-understood utilization.

TECHNICAL VIEWPOINT FOR A NEW CLEANING PLANT IN FRANCE

The precision of the fractionation for any given process is defined, as already stated, by the imperfection I of that process. The present classification of the main processes used in France is the following (average figures):

Rheowashers	0.20 to 0.30
Pneumatic tables	about 0.20
Plunger jigs (old) [2, 3]	0.15 to 0.25
Modern jigs with pneumatic controls [2, 3]	0.12 to 0.15
Heavy-media apparatus	0.05

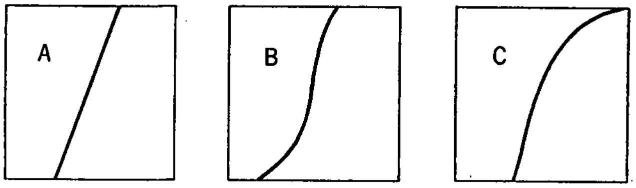
Treatment of large sizes (i.e. > 10 mm.)

The heavy-media washers being slightly less expensive in so far as installation costs are concerned, all the new French plants in construction or planned are of that type.

From the recent initial achievements it has been found that the heavy-media washer, once the medium circuits are adjusted, is easier to operate than other types. On the other hand, the question of the operation cost has not yet been definitely proved.

From the point of view of the sharpness of the fractionation, it may be said, roughly speaking, that all the heavy-media processes are equivalent. The choice is determined by the simplicity or the delivery delay, and therefore various solutions have been adopted in French collieries according to circumstances.

Anomalies have sometimes been observed in the partition curve: if the transformed (amorphosed) curve corresponding to a regular operation presents the shape shown in A, there are at times shapes like B or C.



In the B form, some coal goes off with the shales. This defect may be due to an excess of turbulence; the granulometric study of the densimetric fractions shows that the tramps are mainly composed of fine products.

The form C is frequently observed; there is a "tail" for heavy products; some shales are going with the coal. That form may be due to a "raft" effect: fine shales being carried away on large lightweight coal pieces. In order to avoid it, it is necessary to feed under the surface and agitate the bath slightly.

The methodical study of the separation curves drawn for the various granulometric categories (if, for example, the machine treats a 10-120 type, one would trace the partition curves for the 10-20, 20-40, 40-120 types) reveals the working anomalies and offers a clue for correcting them.

French experience with classical heavy-media machines built abroad in great series showing such defects, has been that they may easily be avoided by modifying the cross sections and divisions.

Treatment of "small sizes" (peas in Great Britain) (that is < 10 mm.)

The centrifuge processes give nearly perfect separation for small sizes but they lack the test of large-scale commercial operation.

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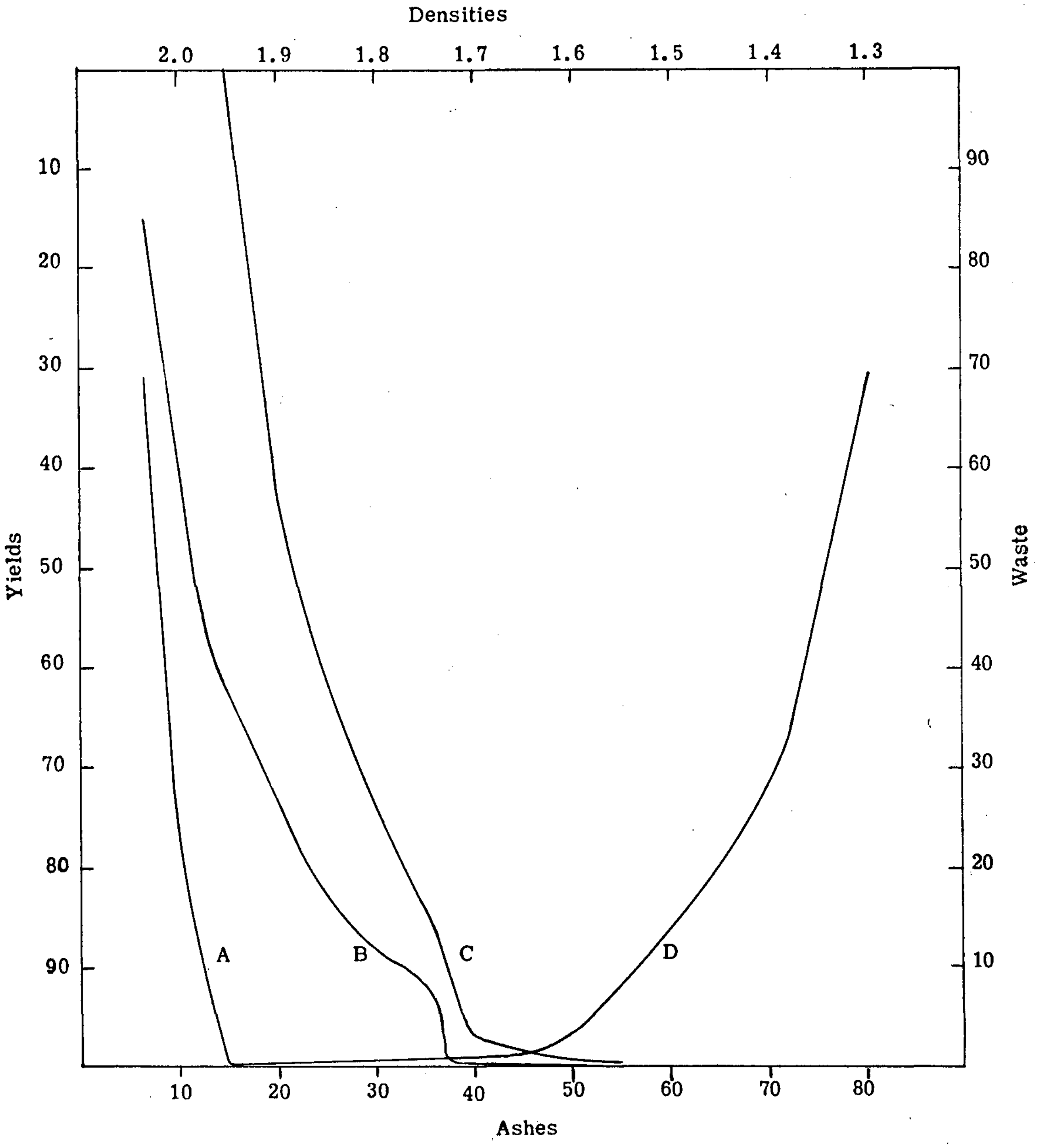
[2] The French term *bac à piston*, like the English term "jig" may be applied equally to apparatus which really has a piston and to those which have a pistoning effect through a pneumatic device.

[3] These figures refer to the jigs for "fines"; for "grains", the figures are better; old types of jigs 0.12 to 0.20; modern jigs 0.60 to 0.15 according to calibre.



Density	Primary Data		Cumulative Data			
	Percentage		Percentage that floats		Percentage that sinks	
	Weight	Ash	Weight	Ash	Weight	Ash
Over 1.30	30.84	6.62	30.84	6.62	100.00	15.06
" 1.40	41.31	11.66	72.15	9.51	69.16	18.82
" 1.45	6.86	20.96	79.01	10.50	27.85	29.43
" 1.50	6.75	24.04	85.76	11.57	20.99	32.20
" 1.60	10.99	35.08	96.75	14.24	14.24	36.07
" 1.80	2.52	36.90	99.27	14.82	3.25	39.41
" 2.00	0.35	40.14	99.62	14.91	0.73	48.06
Below 2.00	0.38	55.36	100.00	15.06	0.38	55.36

Figure 4: DENSITIES
WASHABILITY CURVES OF COAL FROM SEAM V, "LA CHAPA"
Size between 3"–1½"
A = Washability curve C = Refuse curve
B = Characteristic curve D = Densities curve

It is equally possible to wash the fraction ¼ in. –0 in. with good results, and the qualities of the coke obtained when including that fraction are satisfactory. It has been decided, however, not to wash these fines initially in the commercial plants being built, but to make the necessary provisions for subsequent installations when better information is available on the nature of that fraction in large-scale mining. The washing plant being installed is of the Drewboy system, using magnetite as the agent for producing the heavy medium; its capacity is 125 tons of washed coal per hour.

The results of the research ensure that a clean coal will be obtained, satisfactory for the industrial uses envisaged.

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Summary of Discussion

The Tromp Heavy Media Coal Washing Process, ***presented by Mr. Woodhead in the absence of the author***

Mr. Cheradame opened the general discussion of the paper, and agreed with Mr. Woodhead's observation that the three-product vessel had been adopted by most of the European countries. However, he pointed out that in a few installations, the two-product vessel was used. A variant had been developed to counteract the hard-coal problem peculiar to the Saar region. However, the process was only workable for small units.

Mr. Woodhead observed that the two-product system was used in the United States and it was his view that it afforded a great deal of accuracy.

Mr. Turpin referred to the possibility of using coarser medium solids by utilizing the three-product vessel and inquired whether coarse medium solids could also be used in the two-product bath.

Mr. Woodhead replied that the Tromp process would work equally well in both cases.

Mr. Albala stated that occasional very large lumps of coal were peculiar to Chilean mines and wondered whether there was any limit to the size of coal utilizable in the Tromp system.

Mr. Woodhead replied that the Tromp process was capable of handling from 5 to 500 mm. However, a specially designed bath would be necessary. In the United States the maximum size thus far utilized had been 300 mm. Due to the high costs of labour in the United States, the trend was to avoid lump coal as much as possible, and there had been no requests to produce a density medium bath to wash that type of coal. Although it was quite possible that a suitable bath could be prepared, no experiments had yet been carried out.

The Chairman requested information on the quantity of water per ton consumed in the Tromp system.

Mr. Woodhead stated that the volume of water consumed depended on several factors. In some plants, for instance, the medium became viscous, requiring an overflow from the plant of about 600 tons per hour.

The Coal Used in the Mexican Iron and Steel Industry, ***presented by the author***

Mr. Albala wondered whether washing to obtain coals with a lower ash content would be too difficult and whether such improved washing were economically feasible. Chilean coals were usually prepared to an ash content of 7%. He wished to know the reason for washing coals to 15% ash content in Mexico.

Mr. Cortés Obregón replied that he was not at all sure why the coal was not washed to a lower ash content. Mr. Price was, however, presenting a paper dealing with the over-all efficiency of coal washing in Mexico, and could, perhaps, give an explanation.

Mr. Price, in answer to Mr. Albala's question, said that the paper he would be presenting later on dealt with the washability curve of typical Mexican coals. It had been established that 15% ash content was the lower limit if costs were to be kept proportionate with savings in the blast furnace process.

Mr. Belugou requested further information on the reserves of Mexican coal.

Mr. Cortés Obregón stated that, in his personal opinion, the coal reserves of Mexico were of the order of 1,000 to 2,000 million tons. The figure was, of course, subject to variations as in certain zones the deposits might be unprofitable to exploit. However, new deposits had been discovered in areas where it was thought that the supply was exhausted, and that phenomenon was not without favourable implications.

Description of Chilean Coals Used in the Steel Industry, ***presented by Mr. Leuschner in the absence of the author.*** There were no questions.

At the request of the CHAIRMAN, Mr. PYLES LOZANO then took the chair, to enable Mr. PRIETO to present the paper of which he was co-author.

Problems Related to Colombian Coal Used for Steelmaking, ***presented by Mr. Prieto, co-author.***

Mr. OSPINA HERNANDEZ remarked that the deposits of the Amagá region had not been mentioned in the paper, and he wondered whether the resources of that area had been taken into account.

Mr. PRIETO replied that they had, and that a special paper dealing with that region would be forthcoming.

Mr. CORTÉS OBREGÓN requested permission to make an observation, not related to the paper under review, but on an entirely different matter. He strongly supported the movement towards establishing uniform mining terminology throughout the continent, as he was sure that it would greatly facilitate the interchange of ideas among the various Latin-American countries.

Mr. PRIETO agreed wholeheartedly with the previous speaker, and wished to recommend the inclusion of metallurgical terms as well.

Mr. CHERADAME remarked that there appeared to be a great similarity between the terms used in French and in Spanish so that their etymology could be easily traced. He thought, therefore, that a correlation between the terms used in the two languages could easily be effected.

Mr. PRIETO stated that the Spanish Institute of Coal had published a text which contained a fair correlation between the English and Spanish terms of the industry.

The CHAIRMAN observed that in Brazil, a very good dictionary of English, French, German and Portuguese terms had been published.

FUELS

Reduction of coal impurities (*concluded*)

15 October 1953—Afternoon

Chairman:

Eduardo PYLES LOZANO, Companhia Mineração Geral do Brasil, Gão Paulo, Brazil

Contributed Papers:

The Washing of Peruvian Anthracite Fines
Kurt BAUM, Consulting Engineer, Paneuropéenne d'installation d'equipements industriels, Essen, Germany
Notes on the Production of Metallurgical Coal in Brazil
Alvaro DE PAIVA ABREU, Chief, Laboratorio da Produçao Mineral, Departamento Nacional da Produçao, Rio de Janeiro, Brazil
New Processes for Extracting Primary Ash from Coal
Walter VOGEL, Consulting Engineer, Santiago, Chile

Summary of Discussion:

Participants: Messrs. BAUM, CORTÉS OBREGÓN, ALLARD, LEUSCHNER, CHERADAME, ABREU

The Washing of Peruvian Anthracite Fines

KURT BAUM

Low-volatile anthracite is one of the main fuel resources in the Santa district near Chimbote, Peru. The mining in this area is rapidly developing, due especially to the increasing market for this high-grade fuel and favourable exchange freight facilities, for instance with Argentina.

One of the main problems in the economy of these mines is the relatively high percentage of lower valued fines resulting from mining and preparation activities. The use of such fines 0–3 mm., common in anthracite mining, represents a difficult problem in so far as their high ash content of 25–30%, in addition to their fine grain size is adverse to long distance shipping, whereas local requirements are very limited due to the lack of industry in this area. Even briquetting of such fines would not much improve their market prospects in their present condition. Since anthractie is in general a rather hard coal all friable constituents are essentially concentrated in this fraction resulting in a relatively high ash content. The fraction of 0–0.5 mm., for instance, represents nearly equal proportions of mineral matter and anthracite.

So far the only means of cleaning such fines has been through flotation. Experiments with this method, however, have not been very successful so far, due to the special characteristics of this anthracite and the special nature of the free mineral impurities entrained. Even if successful flotation could be obtained, such anthracite fines after passing a de-watering screen show a relatively high moisture content (20–25%), the ballast of ashes hereby being replaced by an almost equal amount of water.

Intensive tests with about 100 tons of such anthracite fines of the Pallasca and Ancos mines using the recently developed Convertol process of Bergwerksverband, Essen, Germany, have shown that the difficult problems could be solved very satisfactorily. The fundamental difference between this and the flotation process is shown by the table on page 60.

Table 1

Original sludge composition		*Flotation yields*		*Convertol yields*	
Coal	50% (H_2O 0%) (Ash 8%)	Coal	60 tons (H_2O 23%) (Ash 8%)	Coal	80 tons (H_2O 8–10%) (Ash 8%)
Free mineral matter	20%	Refuse	40 tons (Coal 50%) (Ash 50%)	Refuse	20 tons (Coal 6–10%) (Ash 90%)
Water	30%				
TOTAL	100%		100 tons		100 tons

Whereas for flotation a very small amount of special flotation oil is added, the coal fines being floated by means of air percolation, while the heavy refuse is sinking, the Convertol process uses a large amount of oil (any kind of low-grade fuel oil may be used) and this oil is brought into intimate reaction with the coal with a "Pallmann mill" after adding a certain amount of water. In this mill, in addition to a certain grinding, a very fine dispersion is created by the action of two steerers rotating at high speed in opposite directions. Through this action, the coal particles—being hydrophobe—are wetted by the oil while the free ash particles, especially colloidal clay, remain in the watery solution. By the addition of special chemicals it is possible to increase this effect as has been proved in the case of the elimination of fine pyrites. The separation of the coal particles covered by a fine film of oil is afterwards performed by means of a high-speed screen centrifuge, in which water and free ash are eliminated through centrifugal force and practically dry coal is extracted under the conditions mentioned above. The screens used in the centrifuge are of special form with round perforations of 0.2–0.3 mm. diameter. In spite of the fact that a considerable percentage of the fines is below 60 microns, the oil-coal remains on the screen. It is clear, on the other hand, that only such mineral matter below 0.25 mm. will be eliminated with the water through these openings and that larger particles of mineral matter will remain on the screen with the coal. Where the coal feed is not all below 0.3 mm., a previous separation should be performed in a settling cone before the sludge enters the centrifuge. The consumption of oil in this process varies between 3 and 7%, depending on the coal to be treated, and remains with the final product.

Notes on the Production of Metallurgical Coal in Brazil

ALVARO DE PAIVA ABREU

None of the coal mined in Brazil is of a quality such that, by ordinary preparation techniques, an entirely satisfactory raw material can be obtained for the production of metallurgical coke.

Moreover, within the deposits, veins are relatively thin, separated by inclusions of shale and clay, and this disadvantage is rendered even more serious by the high content in pyrites or other forms of sulfur.

With few exceptions, Brazilian coal is difficult to wash owing to the fine dispersion of ash, whatever the granulation considered. The problem of washing is in fact one of the most difficult known throughout the world. This explains partly why Brazilian coal products are mediocre in quality and high in price. Natural circumstances are not propitious either for mining or processing them.

The coalfields currently being worked in Brazil are located in the three southern states, namely, Paraná, Santa Catarina and Rio Grande do Sul. A few deposits are also found in the state of São Paulo, but they are of little importance both from the standpoint of quality and quantity.

The coal reserves of Paraná, known to be fairly small—little more than 20 million tons—are mostly bituminous and coking, but their sulfur content is extremely high and is difficult, if not impossible, to reduce. However, certain deposits in Paraná show extremely advantageous washing characteristic with regard to reduction of the ash content. In the lighter parts, a great deal of organic sulfur is found and frequently organic sulfur content is higher than ash content. In some cases as much as 80% of the sulfur occurs in organic form. This makes the coal from Paraná useless as a raw material for the production of metallurgical coke.

The coal of Rio Grande is bituminous; its quality is poor and moisture high. It cannot, therefore, be used as coking coal and will not be considered in the present paper. In order to attain some idea of the difficulty of processing these coals, used chiefly as steam coals on the railways, it is sufficient to state that a current type, mechanically prepared and known as "washed 290", has a 29% ash content.

The coal deposits of Santa Catarina are located in the southern part of the state in a narrow belt lying between the ocean and the Serra do Mar, which produces all the metallurgical coal in the country. Though there are at least five coal-bearing seams in the "Grupo Bonito" in the Santa Catarina system,[1] only two—those of Barro Branco and Irapuá—are of any commercial importance. This formation is assumed to belong to the Permo-carboniferous age.

Most of the mines work the Barro Branco deposits; the Irapuá layer is being exploited only in two or three places.

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[1] I. C. White, 1908.

The Barro Branco seam features a clay bed varying in thickness from 20–40 cm., above and below which occur the workable veins. Just above the Barro Branco is a shale seam in which certain amounts of coal occur. This seam is called "Quadração" and its total thickness varies between 35 and 60 cm. No consistent attempts have been made to mine the coal contained in the Quadração, since the maximum thickness of the thicker veins thereof rarely exceeds 6 cm. Above the Quadração is the "carvão de forro" (lining coal), the thickness of which ranges from 30 to 40 cm. Below the Barro Branco is the "banco" coal, a series of thin coal veins separated by two to four inclusions of shale. The total thickness varies between 70 and 50 cm., of which about 40 cm. are coal.

The Barro Branco layer, therefore, has a total thickness of between 1½ and 2 m. and the coal is mined from the "forro" and the "banco".

Most of the mines are operated manually. There are few pits or slopes; galleries following the seams and starting from the surface are the most frequent systems of entrance; extraction generally follows the "room and pillar" method. There is only one open pit mining operation. Successful tests have been made for mechanizing and transport, but no enterprise has yet employed these methods systematically.

The mining of coal in Santa Catarina was formerly characterized by the dispersal of mining in small units. At one time (1940–47) there was a total of 80 mines producing some 750,000 tons of coal yearly.

Before the Companhia Siderúrgica Nacional plant began working, there was no demand in Brazil for coking coal, as blast furnaces used only charcoal. Although under normal conditions it is not economical to use exclusively Brazilian coal for steelmaking, the Companhia Siderúrgica Nacional has an interest in using the highest possible proportion of domestic coal.

When this company was established, it was impossible to obtain metallurgical coal with less than 18% ash content, and it was obvious that existing coal preparation facilities (1940–42) in Santa Catarina were inadequate to handle the volume required by that company.

In 1942, the existing equipment consisted of two preparation plants absorbing most of the run-of-mine and two small plants for preparation of fines obtained after manual separation in different establishments. Of the two preparation plants mentioned, one, belonging to the Companhia Carbonifera Urussanga, was equipped with mechanical vibration jigs of German origin; fines below 4 mm. were washed in feldspar-bed jigs. The nominal capacity of this plant was 500 tons daily. The other plant, belonging to the Companhia Nacional de Mineração de Carvão do Barro Branco with an hourly capacity of 60 tons, was equipped with two Elmore jigs and three Deister Overstrom tables for washing the fines.

There are two more Deister tables, near Cresciuma, washing fines produced by the Sociedade Carbonífera Prospera and the Companhia de Araranguá.

The product of these tables contained about 16% ash, and was used by city gas plants in Rio de Janeiro, and São Paulo. These plants also consumed coal washed in small manual jigs improvized alongside several mines in the state of Santa Catarina. It was obvious that the metallurgical coal required for the Companhia Siderúrgica Nacional could not be produced by these small units.

The general principles determining the washing programme of the Companhia Siderúrgica Nacional were based on the assumption that it would be impossible to produce a metallurgical coal with a reasonable recovery rate without the concomitant separation of intermediary products, and that it would be desirable to prepare steam-raising coal, principally for the railways, which then accounted for half the coal consumption in Brazil.

It was thus concluded that the washing unit should produce metallurgical coal, two types of steam coal—fine and coarse—and, providing it were economically possible, a high-ash type of fuel for use in thermoelectric plants located near the washery.

The washer was supplied by the McNally Pittsburg Manufacturing Corporation. It was installed on the river, near the town of Tubarão, and handles coal from all the southern part of the state of Santa Catarina, that is Cresciuma, Siderópolis, Lauro Muller and Urussanga.

The coal received by the plant is subjected to a previous selection at the mines, the ash limit being set at 34%.

The capacity of the preparation plant is 400 tons per hour. Crude-coal wagons are unloaded by gravity on to a hopper built on the railway line, whence the coal is carried by belt conveyor to the screening and crushing unit. The feed is separated on oscillating screens into three grades; (*a*) above 4 in.; (*b*) between 4 in. and 1½ in.; and (*c*) below 1½ in. Coal exceeding 4 in. size is tranferred to a table for hand picking; it is then crushed to less than 4 in. and returned to the raw coal screens. The fraction from 4 in.–1½ in. is crushed down to 1½ in., added to the −1½ in. fraction, and then transferred to six 100-ton silos where the coal from each district is deposited separately after having been automatically sampled.

Conveyor belts of varying speeds enable a total of from 50–200 tons per hour to be removed from each of the silos, thus facilitating the blending of coals from different areas in the proportions required by the operator.

This blend is transported by conveyor belt to the washer, where another conveyor distributes it uniformly to two parallel Baum jigs with an individual capacity of 400 tons hourly.

The washed coal is passed on an oscillating screen which divides it into three parts: (1) between 1½ in. and ½ in., (2) between ½ in. and 5/16 in. and (3) under 5/16 in. (1) The fraction of 1½ in.—½ in. constitutes the steam raising coal. (2) The ½ in.–5/16 in. fraction is crushed to less than 5/16 in. and carried to two rheowashers of four laundries producing metallurgical coal, steam raising fines and waste. (3) Similarly the fines, screened to −5/16 in., are processed in one or two 4-laundry rheowashers which also produce metallurgical, steam coal, and waste.

The plant has equipment for recrushing the waste from the jigs to less than 5/16 in. and washing in a rheowasher with a single laundry. This operation is not normally carried out, since the additional recovery of coal would not compensate for the wear and tear on the crushers fed with material containing large proportions of pyrites. Metallurgical coal of from 5/16 in.–0 in. prepared in the two groups of rheowashers mentioned previously, is sent to a de-watering tank, five centrifuges, and thence to the railway cars.

The 5/16 in.–0 in. steam coal passes through de-water screens and is carried on conveyors to the railway wagons. During loading these fines can be blended with the coarse steam coal.

The coal dust from the recrushed coal lifts, as well as the fine metallurgical coal leaving the rheowashers, is carried to a thickener, whence the solid material is recovered by means of an 8-disc vacuum filter. Coal thus recovered can be taken to the metallurgical coal bins or to those of fine steam coal.

In spite of the large amount of water recirculated, the exceptionally high pyrite content of the raw coal makes it necessary to use a very high volume of clean water.

Originally, all the waste from the plant was eliminated by pumping. However, since recrushing of the waste was abandoned, it has been removed in wooden railway cars.

A 15,000 kW thermo-electric plant was installed at the same time as the preparation plant.

The run-of-mine has a high shale content. As explained when describing the characteristics of the Barro Branco layer, there are always a certain number of shale inclusions (from two to four, though generally four).

Current practice in these unmechanized mines is to make a summary selection at the working face itself, where the cars are being loaded. The ash content of the coal thus obtained is not uniform, and therefore most of the producers carry out a second cleaning or picking above ground.

Production of Santa Catarina coal is partly sold to the Companhia Siderúrgica Nacional for preparation in its Tubarão plant. The remainder is screened to eliminate fines (less than 1/2 in.). The resultant coarse coal is hand picked, so that steam coal with about 30% ash content can be sold, this being almost entirely used by the railways. The fines below 1/4 in. are washed on tables and blended with imported coal for the manufacture of gas in Rio de Janeiro and São Paulo.

Raw coal delivered to the Companhia Siderúrgica Nacional must conform to certain specifications established by law in September 1946. The legislation also fixed the price, which is still in force. Legal specifications for "washery-type" coal establish the following limits: Size: below 500 mm.; moisture 3%; ash content (dry basis) 34%; sulfur 7%, and heating power 5,500 calories. The established price for such coal is 150 cruzeiros per ton. The law establishes a 6% premium or penalty per unit of ash below or above 34% to a minimum limit of 30.6% below which no premium is paid, and a maximum of 37.4%, above which it can be rejected.

The price does not include transport to the washer, which is charged at a uniform rate of 8.40 cruzeiros. The price is not considered satisfactory under present economic conditions and readjustment is now under consideration.

In order to meet the aforementioned requirements, producers must carry out certain cleaning operations, which, however, are not complex, nor does the law establish the methods to be used. Certain plants remove the more visible fragments of shale, others screen part of the product and treat the fines on tables or manual jigs, remixing the cleaned portion with the crude coal. Such freedom of operations does not tend to improve uniformity of the feed to the washer.

Total raw coal production in Santa Catarina in 1951 amounted to 1,017,143 tons, of which 700,543 were sold to the Companhia Siderúrgica Nacional as "washery-type coal". From this supply, the processing plant produced 141,359 tons of metallurgical coal and 339,809 tons of steam coal. The metallurgical coal has an ash content of 18%, a sulfur content of 1.75% and 28 to 30% of volatile matter.

The Companhia Siderúrgica Nacional has established a maximum of 13% ash content for the coke used at Volta Redonda. The proportion of domestic coal used in the manufacture of coke is thus limited to 30%. The blend consists of the above-mentioned domestic coal plus American coal with an ash content of from 5 to 6%, and stringent specifications regarding volatile matter.

At present, the Companhia Siderúrgica Nacional consumes some 180,000 tons of domestic metallurgical coal, and it is estimated that 410,000 tons will be required in 1953, after completion of the projected expansion.

Table 1 shows the granulometric distribution of the raw mixed coal. Tables 2–4 show the distribution according to densities of each granulation indicated on Table 1. It should be noted that all these data refer to one given day of operation. These data, though indicating the difficulties of washing this coal, are not sufficient for an adequate discussion of the problem, despite the aid of washability curves, or recovery characteristics of different preparation methods. The density scale adopted does not give an adequate picture of the extreme difficulties shown by the washability curves.

Table 1

GRANULOMETRIC DISTRIBUTION OF RAW BLENDED COAL[a]

Size	*Weight %*	*Ash %*	*No. of the sample*
+3/4"................	34.9	36.6	110
+5/16"................	30.7	31.7	120
+14 mesh............	21.5	31.4	130
+28 mesh............	4.7	30.2	140
+100 mesh...........	6.2	34.5	150
−100 mesh...........	2.0	41.1	160

[a] Companhia Siderúrgica Nacional.

Following recommendations made by Mr. Thomas Fraser, the Company has sampled large lots and subsequently screened and studied the distribution of the product over a wide scale of densities, with a view to ascertaining in detail the densimetric features of the coal from different origins inside the Santa Catarina coalfield.

The Brazilian Government is at present studying measures for placing the national coal industry on a rational basis. The programme covers all coalfields being exploited and, as regards those in Santa Catarina, seeks to establish conditions which will facilitate the concentration of the work in a smaller number of mining enterprises. This would, in turn, make mechanization possible, as well as a saving in labour and an appreciable reduction in production costs.

Table 2

WASHABILITY DATA ON RAW BLENDED COAL

Sample No. 110. Size: 1½″ - ¾″. Percentage of the total: 34.9%

Specific gravity	Values found			Cumulative data					
				Floating fractions			Sinking fractions		
	Weight %	Ash %	Sulfur %	Weight %	Ash %	Sulfur %	Weight %	Ash %	Sulfur %
Float 1.55	50.2	21.0	1.93	50.2	21.0	1.93	100.0	35.9	12.27
1.55 - 1.65	9.6	33.8	3.01	59.8	23.1	2.10	49.8	50.8	22.69
1.65 - 1.75	5.0	40.7	3.90	64.8	24.4	2.24	40.2	54.9	27.39
1.75 - 1.85	3.2	48.3	4.42	68.0	25.5	2.34	35.2	56.9	30.73
Sink 1.85	32.0	57.8	33.36	100.0	35.9	12.27	32.0	57.8	33.36

Table 3

WASHABILITY DATA ON RAW BLENDED COAL

Sample No. 120. Size: ¾″ - 1/16″. Percentage of the total: 30.7%

Specific gravity	Values found			Cumulative data					
				Floating fractions			Sinking fractions		
	Weight %	Ash %	Sulfur %	Weight %	Ash %	Sulfur %	Weight %	Ash %	Sulfur %
Float 1.55	60.6	19.7	1.79	60.6	19.7	1.79	100.0	31.9	8.05
1.55 - 1.65	9.2	34.4	2.70	69.8	21.6	1.91	39.4	50.6	17.69
1.65 - 1.75	4.7	41.4	3.55	74.5	22.9	2.01	30.2	55.5	22.25
1.75 - 1.85	3.0	47.3	5.09	77.5	23.8	2.13	25.5	58.2	25.70
Sink 1.85	22.5	59.6	28.45	100.0	31.9	8.05	22.5	59.6	28.45

Table 4

WASHABILITY DATA ON RAW BLENDED COAL

Sample No. 130. Size: 5/16″ - 14 mesh. Percentage of the total: 21.5%

Specific gravity	Values found			Estimated cumulative data					
				Floating fractions			Sinking fractions		
	Weight %	Ash %	Sulfur %	Weight %	Ash %	Sulfur %	Weight %	Ash %	Sulfur %
Float 1.55	64.4	16.2	1.51	64.4	16.2	1.51	100.0	29.7	5.48
1.55 - 1.65	6.7	34.9	1.97	71.1	18.0	1.55	35.6	54.0	12.66
1.65 - 1.75	4.1	40.8	2.49	75.2	19.2	1.60	28.9	58.5	15.14
1.75 - 1.85	2.8	46.7	3.23	78.0	20.2	1.66	24.8	61.4	17.24
1.85 - 2.00	4.0	52.8	4.35	82.0	21.8	1.79	22.0	63.3	19.02
Sink 2.00	18.0	65.6	22.28	100.0	29.7	5.48	18.0	65.6	22.28

Table 5

WASHABILITY DATA ON RAW BLENDED COAL

Sample No. 140. Size: 14 - 28 mesh. Percentage of the total: 4.7%

Specific gravity	Values found			Estimated cumulative data					
				Floating fractions			Sinking fractions		
	Weight %	Ash %	Sulfur %	Weight %	Ash %	Sulfur %	Weight %	Ash %	Sulfur %
Float 1.55	55.6	13.3	1.52	55.6	13.3	1.52	100.0	30.9	5.10
1.55 - 1.65	8.5	30.8	1.82	64.1	15.6	1.56	44.4	52.9	9.58
1.65 - 1.75	6.3	37.2	1.94	70.4	17.6	1.59	35.9	58.1	11.42
1.75 - 1.85	4.8	46.2	2.28	75.2	19.4	1.64	29.6	62.6	13.43
1.85 - 2.00	4.5	53.7	2.88	79.7	21.3	1.71	24.8	65.7	15.59
Sink 2.00	20.3	68.4	18.41	100.0	30.9	5.10	20.3	68.4	18.41

Table 6

WASHABILITY DATA ON RAW BLENDED COAL

Sample No. 150. Size: 28 – 100 mesh. Percentage of the total: 6.2%

Specific gravity	*Values found*			*Estimated cumulative data*					
				Floating fractions			*Sinking fractions*		
	Weight %	*Ash* %	*Sulfur* %	*Weight* %	*Ash* %	*Sulfur* %	*Weight* %	*Ash* %	*Sulfur* %
Float 1.55..........	45.6	14.2	1.48	45.6	14.2	1.48	100.0	34.3	5.24
1.55 – 1.65..........	11.1	31.1	1.83	56.7	17.5	1.55	54.4	51.2	8.39
1.65 – 1.75..........	9.2	33.2	1.87	65.9	19.7	1.59	43.3	56.3	10.08
1.75 – 1.85..........	6.5	44.5	2.33	72.4	21.9	1.66	34.1	62.6	12.29
1.85 – 2.00..........	5.6	54.0	2.87	78.0	24.2	1.75	27.6	66.8	14.63
Sink 2.00..........	22.0	70.1	17.63	100.0	34.3	5.24	22.0	70.1	17.63

New Processes for Extracting Primary Ash from Coal

WALTER VOGEL

IMPORTANCE OF THE PROBLEM FOR SOUTH AMERICA

Coal preparation during recent decades has been directed towards separating the fractions according to their ash content and on the basis of commercial usage, and of obtaining the most perfect separation possible.

The South American continent is not well endowed with high-grade coal deposits. There are a number of coalfields where the primary ash content is so high that even the most precise heavy-media processes will not result in commercial products that can compete with coal imported from other continents. The sub-bituminous formation of Magallanes is a typical example of such a coalfield.

In view of the shortage of better-grade coal, washing processes which permit the removal of even a part of this primary ash acquire particular interest.

This paper deals with a new process which, although in its infancy, offers prospects for solving the problem. Its findings are based on laboratory research conducted by the Chilean Corporación de Fomento de la Producción on the applicability of such processes to Magallanes coal, and on data supplied by the Deutsche Kohlenbergbauleitung concerning use of one such process on an industrial scale in Germany. Although many of the conclusions refer specifically to Magallanes' sub-bituminous coals, they may eventually be adapted to other coals presenting similar problems, such as those of Rio Turbio in Argentina, Barro Branco in Brazil and also, to a certain extent, to the anthracite fields in northern Peru.

DIFFICULTIES INHERENT IN THE PROBLEM

The difficulty in separating primary ash from coal increases in direct ratio to the extent of the dispersion, or, in other words, in proportion to the fineness of the ash particles in the coal. Figures 1 and 2 are microphotographs of Magallanes coal cuts enlarged 250 times; they show that the ash is found, on the one

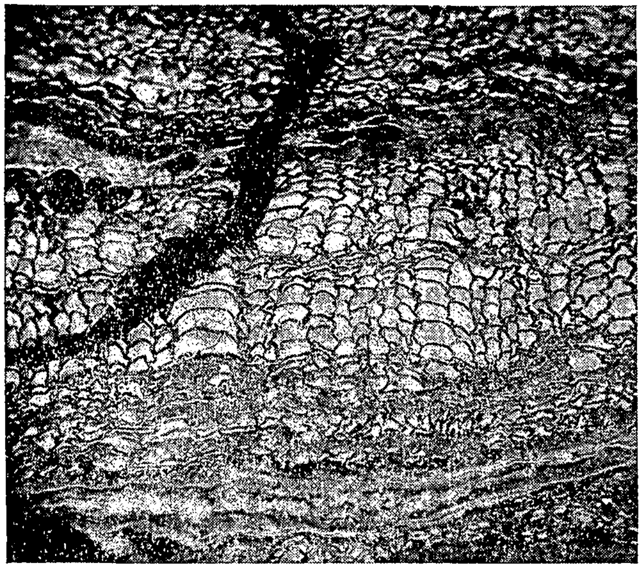

Figure 1

Figure 2

hand, in the form of thin clay veins of only 10 to 50 microns, and on the other, in the form of dispersed particles which are apparently part clay, part carbonates, and are only about 1 micron in size.

These photographs indicate that in order to separate the clay layers grinding below 200-mesh is required, that is, to below 0.072 mm. To separate the particles of carbonates and small clay nodules, even finer crushing is required. This approaches the range of colloids, and provides a clear indication of the difficulty of separating this last ash, which will be called here "dispersed primary ash".

The experiments were made with coal ground to below 200-mesh, but they only occasionally reached the field of dispersed ash; this was for practical reasons in order to remain within familiar grain sizes.

With such a fine grain, of all known washing processes only flotation and electrostatics are applicable. There are serious difficulties in using the second process, however, in the case of the finer particles, and it is out of the question with younger coals such as the sub-bituminous types.

Although flotation has been used successfully in bituminous coal-washing plants, it cannot be applied to sub-bituminous coals because of the content of humic acids or their rapid appearance upon contact with air. Furthermore, flotation is increasingly difficult in ratio to the fineness of the material, and the greater the clay content; these two factors are typical of the problem entailed in cleaning coal with a high content of primary ash.

In view of the foregoing, the CORFO studies concentrated on the phase-separation processes. The encouraging results obtained in the laboratory tests led to a decision to install a pilot plant, which should be in operation by the end of 1955.

THE PHASE-SEPARATION PROCESSES

Phase-separation processes, like flotation, are based on the different surface tensions shown by the solid particles of two different substances in relation to two non-combinable liquids (water and oil, for instance), or in relation to a liquid and gas (water and air, for example).

In flotation, separation is obtained by the adherence of the particles of one of the substances to drops of one of the liquids previously emulsified in the other (as was the case in the early stages), or to gas bubbles (as in later stages). The phase-separation processes differ in that they require each particle of one of the substances to be totally covered by a film of viscous liquid (usually a heavy oil), completely isolating it from the other (usually water) and subsequent adherence of such covered particles to each other and to the excess viscous liquid. This phase thus consists of particles akin to the oil plus this oil separated from the water phase, plus the remainder of the particles not akin to oil. Because of this complete separation, these have been known as phase-separation processes.

The very nature of these processes is such that larger amounts of oil are used than in flotation, but on the other hand a concentrate with much less water is obtained, so that there are fewer impurities in the product, some such as clay being extracted with the water.

Since the particles of the concentrate have to be entirely covered with an oil film, they have to be kept in motion in relation to each other either by slipping or by rotation, to ensure complete contact with the oil. Consequently these processes can be applied to smaller particles than those used in the ordinary flotation process.

Processes requiring a large addition of oil can naturally be applied only on a commercial basis in those cases where there is subsequent thermic treatment to recover the oil. Removal of primary ash presupposes fine grinding and inevitably leads to subsequent treatment, either in the form of briquetting, carbonization or coking, in order to obtain a consistent product. Such a subsequent process will either lead to recovery of the oil or else to its direct use as a binding agent, in which case a careful selection of the oil is indicated.

Undoubtedly, in the case of sub-bituminous coals similar to those of Magallanes, some thermic treatment would be indispensable in any case, as apart from its high ash, its moisture content limits the radius of its utilization as well as its high percentage of volatile matter, causing propensity to weathering and spontaneous combustion. Addition of a larger amount of oil during preparation is not an unfavourable factor.

To date, three coal preparation processes based on phase-separation have been developed:

(*a*) The Trent process;

(*b*) The kneading process;

(*c*) The Convertol process.

(a) *The Trent process*[1] is only of historic significance to-day. It consists essentially in adding to an aerated pulp of finely crushed coal, 30 to 40% of viscous oils with a density below 1 (pitch or petroleum residues mixed with lighter liquid hydrocarbides), then stirring the mixture to achieve intimate contact, until a pulp has been obtained which floats on the surface while the extracted ash remains in the water. The pulp is removed and dried in presses. The process was used with anthracite and bituminous coals, up to 80% of their ash content being removed.

This process was not propagated because of the high consumption of blending oils which were partially lost in the tailings and also because at that time no market for the concentrate was found. For this reason and because of the lack of precise knowledge of the process, the idea did not prosper. With present-day understanding of the phenomena of phase-separation, it appears that the fault lay in the fact that it did not usefully solve the problem of the kinematics of the coal particles in order to ensure their being covered with oil, resorting, in the absence of this to the use of an excess of oil. On the other hand, the need to give the paste a density below 1 (so that it would float), requires the use of lighter oils which, partly through emulsioning and partly through absorption of the particles of the tailings are lost with these. Furthermore, difficulties arise in de-watering the paste in filters.

(b) *The kneading process* was specially perfected by I. G. Farben engineers during the war to treat lignites

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[1] British patent 151236 of 1919; United States patents 1420165 and 1421862 of 1920, and German patents 357388 of 1920 and 42382 of 1922.

There are other processes such as jigs, pneumatic tables, rheowashers of all types, flotation, which give an imperfect separation.

In France, the only ones used in new installations are the jigs, with pneumatic pulsations and filtering layer, pneumatic tables and flotation. Each one of these has a well-defined field of application.

A preliminary remark must be made concerning very fine coal:

(*a*) The very fine suspended coal cannot be allowed to flow out with the effluent. In France, this is both a water problem (to prevent river pollution) and an economic one (to recover the "slimes" which will later be incorporated into the coking paste, or burned in the power stations with the middlings);

(*b*) Recovery of the very fine fractions, in the liquid cleaning processes, is therefore tantamount to the treatment of the slurries, which is always considered as a nuisance in France. By contrast with Great Britain, where the flotation process is willingly adopted, the Charbonnages de France try to avoid it or to reduce its importance. For the processes treating de-dusted $x-10$ mm. fines, therefore, particular importance is attached to a value of x as low as possible.

Jigs with automatic pistoning and filtering layer

The tests made by CERCHAR, both at the laboratory and at the commercial scale, have shown the undoubted superiority of dissymmetrical over symmetrical pistoning. Pneumatic control permits any degree of dissymmetry and is therefore more suitable than control by piston and eccentric.

According to French practice, the removing of shales through a filtering layer is the best process.

It is usually economically advantageous to use a slightly over-size and therefore more expensive jig, but this depends on the nature of the coal; a study of "washing possibilities" will show it beforehand.

The imperfection of such a jig is near to 0.15. It may fall to 0.13 if the jig is over-sized, and rise to 0.17 if the jig is overcharged.[4]

In either case, it is very important to feed the jig regularly, for example by introducing a small feed-regulating hopper; it is better to have a steadily and reasonably overcharged jig than one which is irregularly underfed, since the jig tends to empty itself, despite the presence of the "self-deshaler" and the composition of the washing bed changes.

What is the value of x? Down to 0.5 mm. the imperfection does not increase noticeably; there is therefore no reason to cut the fines above 0.5.

In practice, the modern feldspar jigs in France treat from 0.3-10 mm.; some mines even treat non-dedusted raw coal from 0 to 10 mm.

Baum jigs

There are none of these in France, i.e. no jig treating large and small sizes simultaneously, although they are common in Anglo-Saxon countries. According to French experience this type can be used only when the coal is very easy to prepare and when the nature of the raw coal and market conditions are such that all the size categories are saleable. The study of such a single fractionation shows that the separation density obtained for the different categories varies within very broad limits. Generally the large material is cut at a low density (1.4 for example) and the fines at a high density (1.9). In that case the fines have a high ash content and often require re-washing in a second jig.

The imperfection is of the order of 0.20 for the larger sizes and 0.15 for the small sizes (peas in Great Britain).

Pneumatic tables

These machines give satisfactory results ($I=0.20$) provided that the coal is dry (less than 4 to 5% moisture), and that the ratio of screening is low (2 to 2.5); the misplaced elements must be eliminated after passing over a table, and treated again on a table handling the lower granulometry. Under these conditions, it has been observed that the separation curves are always close to the theoretical form ("normal" curves when logarithms of the density are taken as abscissae).

The process is of interest for limited capacity plants when, for example, dry products for subsequent agglomeration are required.

This process, suitable for raw coal between 2 and 50 mm. might possibly be advantageous for Latin America.

Flotation

This well-known process is reserved for products < 0.75 mm. In France, it is considered expensive, and it is used solely for the treatment of coking coals.

Cyclone

The oldest centrifugal process is the Dutch cyclone. It has been studied closely in a French experimental station, and the results may be summarized as follows:

1. It is possible to treat 0.5 mm. dust-free fines at the rate of over 50 tons per hour in a cyclone 500 mm. in diameter. If the separation density is below 1.75, a shale medium may be used; above 1.75, either barytes (in which case the coal particles must be floated) or magnetite must be used.

   It should certainly be possible to use a medium made of pyrite particles common in some Latin-American coals. Under certain conditions it should be possible to develop a circuit in which the pyrite losses would be compensated by fresh pyrite derived from the raw coal. The adjustment of such a circuit would clearly require some laboratory and semi-industrial tests (which also applies to the heavy-media processes for treatment of large sizes —singles in Great Britain).

2. For treating un-de-slimed raw coal, a magnetite medium must be used. Separation results are as follows: for the 0.1-0.2 mm. fraction, poor; for the 0.2-0.3 mm. acceptable;[5] for higher fractions very

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[4] The studies made by CERCHAR during the winter 1951–52 have shown that these figures may be lowered to about 0.02 by replacing feldspar, of density 2.7, by a lighter bed, such as earthenware cubes ($D=2.2$).

[5] But not worse than separation by flotation.

from central Germany. Unfortunately, post-war conditions in Germany precluded continuance of this work.

Experiments made with Magallanes coal after 1948, following a similar road, proved the possibilities of this process for treating the sub-bituminous Magallanes coals with their clayey ashes.

By way of contrast with the Trent process, this one does not operate in a pulpy state, but in a pasty state, also with an addition of 30 to 40% of oil. The phase-separation is obtained by subjecting the paste to intense kneading which produces in the mass innumerable areas of slipping and shearing, thus causing complete coverage of the particles with oil. The phase-separation occurs at the moment of completing this oil coverage, and a water pulp begins to curdle from the mass, bringing away with it the hydrophile components of the ash. Once the phase- separation has occurred, the surface of the paste only has to be washed with water during kneading in order to eliminate the ash. For this, the process requires a theoretical addition of oil to fill the interstices between the coal particles in order to effect complete removal of the ash-charged water from the pulp. This explains why this process, unlike the Trent, does not require low densities of the oils; heavy oils, which emulsify as little as possible, are in fact more desirable.

Only a pilot plant at Hoechst, Germany, has worked on a commercial scale, using this process. It was useful for testing industrially the purification of central German lignites having a quartzite ash. There are no details available of these tests. From private sources, it appears that aided by the absence of clay it proved possible to reduce the ash content to very low figures. An acid medium was used. A kneading pump was used, developed for treating plastics in the chemical industry, described by Kieskalt.[2]

For the reasons given above, the development of the process was not completed. In our experiments it was thus possible to ascertain that there were many possibilities for improving the work of the kneading pumps in the sense of removing part of the ash in the pump itself. Using kneading pumps as currently constructed, it would be necessary to pass the mass through three or four consecutive pumps, washing it as it leaves each pump, in order to obtain complete elimination of the ash separated at the phase-conversion stage.

Notwithstanding the high addition of oil required by the process, it deserves consideration because it delivers almost coal-free tailings, particularly in the case of very fine coal. However, at the present stage it has been surpassed by the Convertol process.

(c) *The Convertol process* was only developed in 1950 by engineers of the *Deutsche Kohlenbergbauleitung*, Essen, with an immediate view to using slurries from the conventional washing plants for coking purposes.

As in the case of the Trent process and ordinary flotation, the Convertol process works with a coal pulp from 200 to 500 gr. of coal per litre. It differs fundamentally, however, from the other two methods, because it achieves the conversion of phases by means of simultaneous dispersion of the combination of coal, ash, oil and water, subjecting it to violent and turbulent agitation and/or to continuous changes in pressure. In this way it achieves phase-conversion with a much lower addition of oil—from 5 to 12% according to the type of coal. This is due to the sharp turbulence and to the dispersion of the oil in the pulp. Through the turbulence, the coal particles rotate around themselves and the frequent shocks cause them not only to fracture, producing fresh and more active surfaces, but also to enter into better contact with the dispersed oil drops.

This covering of the particles in the midst of dispersion and with a minimum addition of oil results in their only being flocculated to a certain degree, thus facilitating the separation of mud from the ash by simple screening in a high-speed centrifuge. It has in fact proved sufficient to use in these centrifuges screens with holes having a diameter of 0.15 to 0.25 mm. and high centrifugal force (1,000 to 1,500 times gravity) in order to remove the water content from the concentrate to only 5–10% with no appreciable coal loss. This means that the oil-covered coal particles agglomerate among themselves, forming coagulates of over 0.25 mm. This considerably simplifies the process. Naturally it is necessary to feed the purification plant with coal inferior to the size of the screen-hole because otherwise grains of rock and slate would remain on the screen and, therefore, in the concentrate.

Where it is desirable to have a larger size in the feed, the difficulty can be overcome by interposing a flotation cell between the converter and the centrifuge. Particles covered with an oil film float very easily in ordinary flotation cells so that the larger-sized ash can easily be removed therefrom. A flotation cell can also be added to the centrifuge to control tailings with regard to coal losses, a small number of cells being sufficient. Incidentally, this reduces the amount of oil required for conversion. In interposing flotation between the two machines, the water removed in the centrifuge can of course be returned to flotation in order to control eventual losses. This would mean an increase in the number of flotation cells, but would ensure absolute control of losses. This control of losses is necessary because of the wear on the screens increasing the size of the mesh. Figure 3 shows a flow-sheet for the Convertol process.

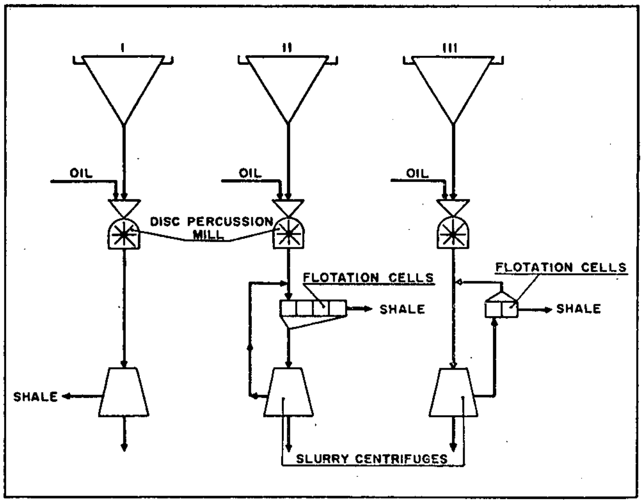

Figure 3. Flow sheet

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[2] Review of the V.O.I. 1942, page 752.

In accordance with the requirements of the Convertol process, the builders of screens have perfected the construction of the sides of the screen for centrifuges with conical bodies. Using a new stamping process and stainless metals, highly corrosion-resistant, calibrated screens with very fine orifices can be obtained.

The essential technical element in this process is the dispersion mill for the conversion of phases. The Pallmann mill has been very successfully used for this purpose. It is a percussion mill with a rotary disc, and consists of a plate which revolves at high speed within a fixed shell and in front of a second fixed plate, the edge of which is very close to that of the first, leaving only a narrow groove between both. In the concavity between the two plates, an agitator of four arms also rotates at great speed counterwise to the moving plate, thus causing considerable turbulence and continuous shocks to the material. This is fed jointly with water and oil by a central aperture in the fixed plate.

Petroleum fuel oil known as "Bunker-C" has usually given best results.

The first plant to use this process is installed at the Hanover mine in the Ruhr. (See Figure 4). Fine refuse (slurry) from the washing plant of the bituminous coal mine is being treated in it. It has the following granulometric composition and ash distribution:

Screened fraction	*Per cent in weight*	*Per cent in ash*
Above 1 mm	10.7	6.4
1.0–0.5	7.9	7.2
0.5–0.3	5.6	9.2
0.3–0.2	21.9	17.9
0.2–0.1	31.2	26.8
0.1–0.075	6.6	39.7
0.075–0.06	3.7	43.2
Below 0.06	12.4	32.6
TOTAL AND AVERAGE	100.00	22.3

No difficulty was encountered in obtaining phase conversion of these fines using a Pallmann mill with a capacity of four tons of concentrate per hour. Ash content was reduced to 7% with a moisture of 10 to 12%. It should be noted that the most perfected centrifuges were not yet available nor were flotation cells used for control purposes. The concentrate was granulated and could be poured without difficulty into the storage bins.

A second experimental plant is in operation on an industrial scale at the Königen Elizabeth mine, where it treats middlings discharged by a flotation plant used for obtaining products with a very low ash content. Thus the process permits obtaining fines suitable for coking from a product which could previously only be used in boilers. Several other plants are in course of construction.

For laboratory studies, conversion conditions in a disperse- and turbulent-medium can easily be obtained by using cocktail shakers with high-blending revolutions. If selected with a glass container, there is the additional advantage of being able to observe the moment at which phase-separation occurs.

SOME OBSERVATIONS DERIVED FROM EXPERIMENTS WITH SUB-BITUMINOUS MAGALLANES COAL

Application to coal preparation of processes based on differences in surface tension will naturally depend to a large extent on the chemical composition and surface properties both of the carbonaceous substance and the ash, on the dispersion of the ash in the former and, finally, also on the quality of the water and oils available. Considerable variations will thus exist from one case to another and, as with mineral flotation, the exact conditions will have to be determined in each case by laboratory experiments.

It is therefore worth summarizing here some interesting observations made in the course of the laboratory experiments carried out by the Corporación de Fomento de la Producción in Chile with the sub-bituminous coals from Magallanes and the central zone of Chile. It is perfectly possible to obtain a phase separation with these coals but it is much more difficult than in the case of bituminous and anthracite coals. Certain deviations in the working methods have to be followed and certain reactives have to be added.

The large amount of oxygen in these coals which appears in the form of non-saturated hydrocarbons makes them have a closer affinity with water and, what is even more important, increases their affinity with oxygen. The surface of the coal fractions therefore becomes covered with a film of humic acids and other oxidation products of these coals, making them oil-repellent.

The following primary analysis of a sample of the coals illustrates these conditions:

	Per cent	*Related to coal-bearing substance—per cent*
C	54.30	74.38
H	4.15	5.68
N	0.92	1.27
S	0.88	1.20
O	12.75	17.47
Moisture	16.50	—
Ash	10.50	—
TOTAL	100.00	100.00

Naturally, it was found during the experiments that any sample of this coal, even if kept below water, gradually loses its aptitude for oil-water phase-separation. Moreover, time required for phase conversion increases from one to four, six and more minutes after the sample has been kept three, four and more months under water, until there is complete failure after six to eight months of immersion. This is undoubtedly due to the slow oxidation of the coal by oxygen in solution in the water. It was also found that coals from mines of lesser depth having been in contact with underground water were much more difficult to convert than coals from deeper mines.

Guided by these observations, laboratory tests were made grinding the dry coal sample in an inert atmosphere, using N, CO_2, NH_3 and town-gas. Crushing with CO_2 brought no notable improvement; the coal could only be moistened with difficulty, and phase-separation took longer, indicating that a reaction took place covering the surface of the coal with a film more difficult to cover with oil. The same was true using town-gas. On the other hand, an evident improvement was obtained by crushing in atmospheres of N and NH_3, but whereas the experiments on the basis of N resulted in clean pulps with the ash expelled, on using NH_3, atmosphere in the crushing, the tailings were

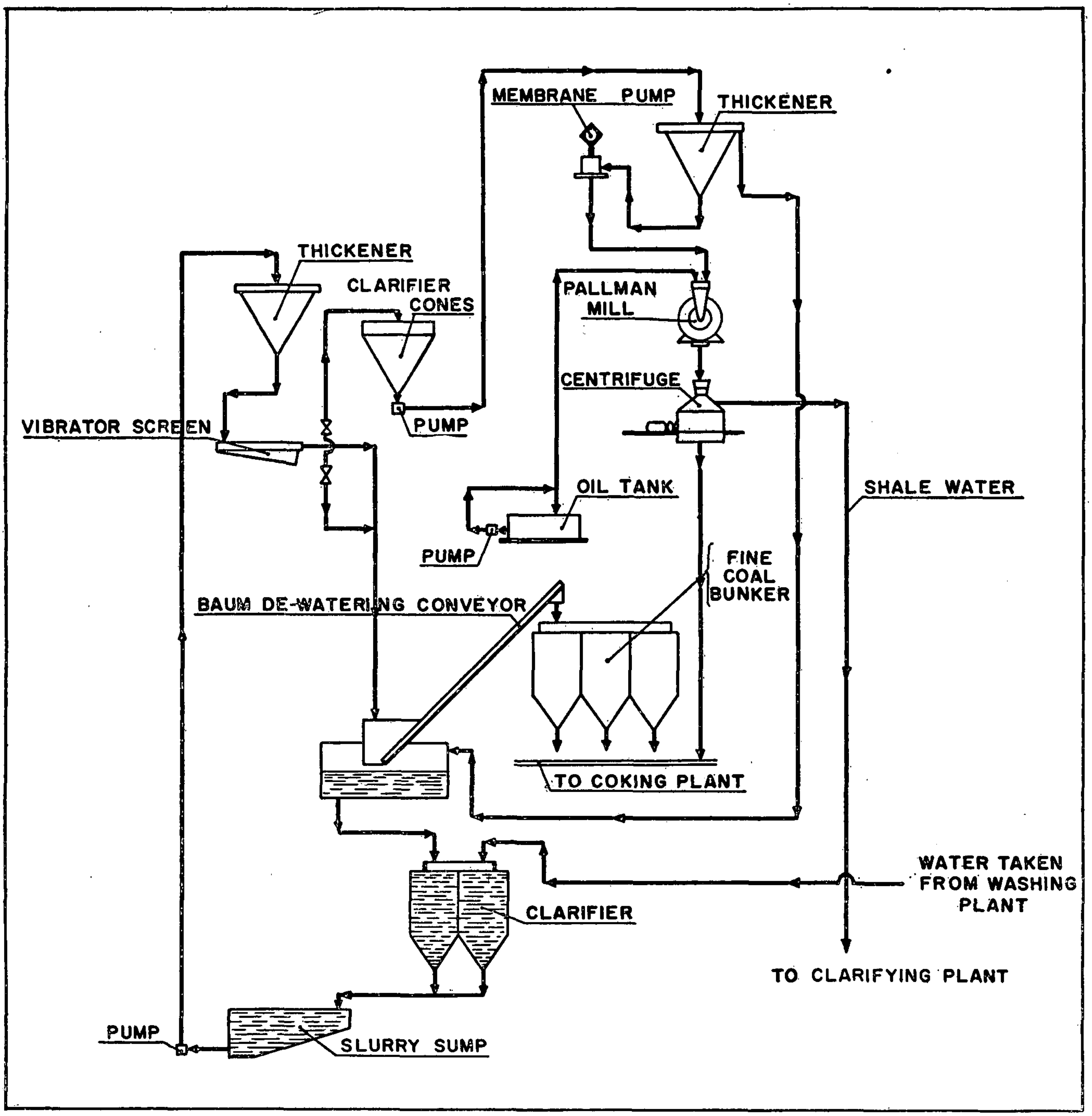

Figure 4
Preparation of coal slurries at Hanover IIII Pits.

contaminated with organic substances, probably humates. In addition, ash extraction was much better in the first case.

The conditions of the whole experiment were so visibly improved that all subsequent tests were made on this basis. Moist crushing was thereafter abandoned, but at this point other factors are present, reference to which will be made later.

It should also be noted here that presence of air bubbles in the dispersing apparatus also gave more unfavourable results. In fact, all experiments in which, as a precaution, agitation took place in an N atmosphere were considerably better. This effect of N is naturally not evident in coals which have been much exposed to oxidation.

An attempt to influence the oxidized surfaces with an addition of high-grade alcohol, such as Laurol, which had given good results in Germany with bituminous coals, was only effective with the Chilean coals when considerable amounts, of the order of 4 to 5%, were used, which economically is not possible.

These remarks pave the way for a discussion of the possibilities of influencing separation conditions by reactives just as in the case of flotation. Two possibilities have to be considered: either to try and activate the coal surface, or to depress it, that is, to make the ash components more repellent to oil or increase their affinity to water.

Naturally it is known from flotation experience that the value of the pH in which work is carried out has important effects on separation and on the behaviour of any reactive. In the course of experiments it was found that among the sub-bituminous coals there were two types with different behaviour distinguished by the pH at which water dispersion did occur. The first type which had certainly been less exposed to oxidizing effects within the deposit showed a slightly alkaline reaction with pH values between 7.2 and 7.6. It gave the best results when alkalinized with lime. It is worth noting that any lime added remains entirely in the concentrate, perhaps due to reaction with the humic acids. If by contrast these types of coals are acidulated, the separation deteriorates considerably, improving again when the pH value is below 5. The second group shows an acid reaction with pH between 6.2 and 6.6. When attempting to alkalinize it, its buffer effect can be noted, as large amounts of alkali are required to modify the pH value. It cannot be converted once it is alkalinized, but rather a phase separation is obtained with the pH remaining acid, or even increasing the degree of acidity.

Successful activating agents for the first type of coals are not so with the second, and vice versa. Among the main activating agents for the first type of coals, there is rodamina, and for the second certain reactives which are used in flotation as cationic collectors or similar chemical compounds. Thus good results have been obtained with paraminophenol, although rodamina and cresilic acid also act favourably.

With regard to the oils to be used for phase-separation, it is important to use heavy residues which are in a liquid state at an ordinary or slightly raised temperature (40 to 50° C.) and containing a minimum of light fractions. The best results are obtained with petroleum residues of the asphalt type. Oil or parafinous pitch residues are more difficult to use for phase-separation. In any case they give better results with acid types of coal than with the basic types. To a certain extent conversion can be activated with small additions of pine oil, but in that case special care has to be taken to prevent froth formation within the dispersing apparatus. Less efficient results were obtained through use of high-temperature pitches in the separation of sub-bituminous coals. Results improved using heavy fractions of low-temperature pitch obtained from the carbonization of the same coals.

A lengthy series of experiments tended to influence the surface of the clayey refuse. Here it could be proved that in the first place a breakdown of the clay to colloidal-size particles before the separation is obtained should be prevented, because its more active surface absorbs lighter fractions of hydrocarbides, thus tending to pass into the concentrate. For these reasons moist crushing gives poorer results than dry crushing. In many cases it could be observed that the extraction of clay was much more perfect when dry crushing took place and when the dispersion had been left to settle immediately after the phase-separation began, completing the dispersion half an hour later. This can be explained by the fact that the absorption of water by the clay and the resultant colloidal disintegration facilitates the separation process without being harmful because the activating reactives had acted beforehand on the coal.

With the alkaline types of coal the best result as depressants of the clay were obtained by some dyes, in particular rodamina, as it acted simultaneously as an activating agent on the coal surface. Methylene blue is also effective, though not so much as rodamina. Best results were obtained with a rodamina addition of 150 gm. per ton.

Two inorganic reactives, waterglass and sodium metaphosphate, act favourably in an alkaline medium, although they do not give as good results as rodamina. Other refuse-depressing reactives known from flotation, such as cyanides, tannic acid, quebracho, starch, etc., did not give any results. In any case there is a considerable field for experiment; conditions may vary greatly with coals having other ash composition.

Finally, two separation results obtained by the process, one with acid coals and the other with basic coals, are given below:

Pecket coal (Magallanes)		*Pupunahue coal (Valdivia)*	
Alkaline type:	pH = 7.4	Acid type:	pH = 6.4
Original sample:		Original sample:	
	H_2O = 22.5		H_2O = 16.2
	Ash = 14.4		Ash = 15
Volatile matter....	32.1	Volatile matter....	33.3
Fixed carbon......	31.0	Fixed carbon......	35.5
TOTAL	100.0%	TOTAL	100.0%

With Magallanes coal, using the kneading process with an addition of 30% of lime-saturated water, 150 mg. of rodamina per kilo, and 30% of fuel oil from Peru, 44.5% of the ash could be extracted. The composition of this ash and the residue remaining in the concentrate may be seen from the following figures:

	Extracted ash	*Residual ash*
SiO_2........................	55.08%	24.67%
Al_2O_3........................	38.68	15.81
Fe_2O_3........................	1.99	9.63
CaO........................	0.97	23.13
MgO........................	—	3.49
Alkalis........................	2.38	20.38
SO_3........................	—	—

It can be deduced from these analyses that an ash has been extracted composed almost exclusively of SiO_2 and Al_2O_3 with a relationship of 2.5 to 1 molecules. It is probably a mixture of kaolin with beidellite. It is interesting to note that the compounds containing Fe, Ca and Mg have practically not been expelled. These are mainly carbonates, gypsum and pyrites which can be observed in the micro-photographs dispersed in very fine particles.

In the Valdivia coal, with an addition of 100 mg. per kilo of paraminophenol and 10% of Magallanes fuel oil (parafinic), a 51% ash expulsion was obtained, using the Convertol principle. The analysis of the ash is as follows:

	Extracted ash	*Residual ash*
SiO_2........................	58.95%	38.50%
Al_2O_3........................	35.68	28.22
Fe_2O_3........................	1.35	10.92
CaO........................	0.61	11.79
MgO........................	—	1.23
TO_2........................	0.18	0.50
Alkalis........................	3.19	0.79
SO_3........................	—	7.28

Here again, an almost pure clay has been expelled in which the relationship of SiO_2 to Al_2O_3 is 2.8 molecules. It is therefore fairly close to the composition of beidellite. The suspension properties of the highly colloidal muds and the great water absorption of these clays makes their use as bentonites quite probable.

In this case, also, it may be noted that the chemical composition of the residual ash corresponds precisely to those ash components which appear finely dispersed, such as carbonates of Ca, Mg and Fe, sulphate of lime and iron pyrites.

SUMMARY

Although these processes are still in their initial stages, it may be seen that they offer many possibilities for purifying difficult coals with high-primary ash, even in the case of clayey impurities. The work already begun also promises to solve the question of expulsion of finely dispersed ash.

Summary of Discussion

The Washing of Peruvian Anthracite Fines, *presented by the author*

Mr. BAUM explained that the Convertol process had been developed to extract coal fines below 60 microns which flowed into the waste water of coal preparation plants. The normal Convertol procedure was the following: the coal fines floating in water were mixed with 3 to 6% of heavy oil in a special disintegrating machine, the "Pallmann mill", previously used for grinding. The ratio was 1.5 : 1; a change of phase took place so that the water-repelling coal was covered with an oil film, and the ash, more easily wettable, remained in the water in suspension. The material was then passed through a high-speed centrifuge with very fine perforations through which the water escaped, carrying the ash, while the oily film with the coal particles was retained and extracted. As the water did not stick on the oil-covered coal particles, the water content of the remaining product was only about 8%.

He then stated that the difference between the Convertol and the froth flotation processes lay in the fact that the former used a great deal more oil, which was not lost, however, and that there was not even one per cent of coal remaining in the water. The colloidal clay could also be eliminated. With froth flotation the separation was not so clear, and the product contained 20 to 25% of water after filtering, which made it difficult to dry and add to coking blends.

The Convertol process was useful where ash was very finely dispersed.

Referring to a question put by Mr. Prieto at an earlier meeting, on the problems of coal with 15% ash, he said that while no promises could be made, nevertheless if the coal were ground very fine, it would at least be possible to separate a substantial part of the ash from the coal.

In the case of non-coking coal intended for use as a raw material for making coke, the hot Convertol method could be used, in which case work was conducted at a temperature of about 70°C. to 80°C. in hot water, using liquid pitch instead of oil as the agent.

The data thus far brought forward during the meeting had indicated the serious nature of coal-cleaning problems in Latin America; but in view of the fact that oil and asphalts existed in the region, it might be possible to improve coking coals by using the Convertol method.

Mr. CORTÉS OBREGÓN, in opening the general discussion on the paper, asked whether the reference was to coal tar and pitches or petroleum tars and pitches.

Mr. BAUM explained that Talara fuel oil and Talara pitch had been used in Peru.

Mr. CORTÉS OBREGÓN said that he understood from some experiments that coal had more affinity with coal tars and pitches than with petroleum.

Mr. BAUM replied that the same thing had been observed in their work and that it was difficult to forecast whether the oil would dissolve the pitch, especially when using it as a coking agent. The Talara fuel oils, however, contained more light fractions than those in Germany, which facilitated solution of the pitch.

Mr. ALLARD asked whether it were possible to give any idea of the expense involved in the process.

Mr. BAUM replied that it was difficult to give an accurate figure on power consumption, but that the actual cost of the Convertol process depended on the life of the screen, the cost in Germany being about 2.50 marks per ton.

Mr. LEUSCHNER referred to a paper on the subject being prepared Mr. Vogel,[1] who was working on experiments with the Convertol process, using Magallanes coals from the southern region of Chile, ranging from lignites to high volatiles. For unforeseen reasons, the equipment had not been completed in time, and therefore the experiments had not been finished, so that it had proved impossible to present the paper at the meeting. It would, however, be circulated later and included in any printed version of the proceedings.

The Magallanes lignites had a very high ash content, so dispersed that the usual coal-preparation processes based on differences of specific gravity were not applicable. Conversely, anthracites could in most cases be froth flotated. He wondered, therefore, how the cost of the Convertol process would compare with that of flotation. Froth flotation was very difficult with high-bituminous coals, as in the case of the Magallanes coals, but much easier with any anthracite. He inquired whether any comparison of the costs of the two processes were available.

Mr. BAUM said that he was unable to give any cost comparison, as the process had never been applied in commercial-scale operations, as yet. Furthermore, it had never yet been used on clean coal with finely dispersed ash.

Mr. CHERADAME stated that it had been found in France, with certain procedures in which mixtures of

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[1] New processes for extracting primary ash from coal.

coal and oil were used for briquetting, that an increase of oil from 1 to 2% was an economic advantage. When 4% of oil was used, it corresponded exactly to the findings of different studies.

Mr. BAUM explained that voids in the fines to be briquetted represented from 20 to 25%, and that the total surface of the grains was much larger than was the case with ordinary briquettes for fuel, and that it was therefore very difficult to distribute the more viscous pitch evenly on the increased particle surface.

Notes on the Production of Metallurgical Coal in Brazil, *presented by the author*

Mr. CHERADAME opened the general discussion on the paper, and raised the question of the rates for the premiums and penalties in relation to the ash content of the raw coal delivered by the small miners to the washing plant as described in the paper, suggesting that some of the theories and methods of calculation mentioned on the previous day might be applied in order to establish new rules for premiums and penalties, with a view to improving the over-all efficiency of mining and washing operations. That might perhaps be advantageous from the national economic point of view.

Mr. ABREU explained that the penalties or premiums established for that coal were imposed at a time when there was not such a perfect gauge available as existed at the present time. When there were seams of 20 and 40 cm. separated by an intermediate layer of more than 60 cm., the washing process required the removal of the top layer and of the rock at the bottom, before reaching the washing plant and there was a tendency on the part of the miners underground to place everything on the car and on that of the entrepreneurs to deliver it. To prevent that practice it became necessary to devise the system of differential payment. The measure had been successful and a certain uniformity had resulted. It was not exactly fair to the operators, but it was intended to make the necessary changes which would improve the quality of the coal delivered and benefit the different parties, so that the situation described would change. It had not been possible, however, to make such changes as yet.

Mr. BAUM requested information concerning the nature of the finely dispersed ash in Brazilian coals and as to whether gypsum was also found therein.

Mr. ABREU replied that he could not describe it exactly, but that it should be of a clayey nature, judging by its silica and alumina content. It was easy to separate the ash of Santa Catarina coals, but not of Paraná coals. There was no gypsum in the coal.

FUELS

Production of coke from poorly coking coals and substitute fuels

16 October 1952—Morning

Chairman:

Eduardo Pyles Lozano, Companhia Mineração Geral do Brasil, São Paulo, Brazil

Contributed Papers:

Utilization of High-Volatile French and Saar Coals for the Production of Metallurgical Coke—Application of the Conclusions to Latin-American Coals
R. Cheradame, Directeur général adjoint, Centre d'études et recherches des charbonnages de France (CERCHAR)

The Blending of Western Coals for the Production of Metallurgical Coke
John D. Price, Superintendent, Coke Plant, The Colorado Fuel and Iron Corporation, Pueblo, Colorado, U.S.A.

Low-Temperature Char as a Substitute for Low-Volatile Coal in the Production of Metallurgical Coke
John D. Price, Superintendent, Coke Plant, The Colorado Fuel and Iron Corporation, Pueblo, Colorado, U.S.A.

The Manufacture of Metallurgical Smelting Carbon from Non-Coking Coals
Kurt Baum, Consulting Engineer, Paneuropéenne d'installation d'equipements industriels, Essen, Germany

Summary of Discussion:

Participants: Messrs. Leuschner, Cheradame, Albala, Prieto, Baum, Lopez, Price, Powell, Allard

Utilization of High-Volatile French and Saar Coals for the Production of Metallurgical Coke—Application of the Conclusions to Latin-American Coals

R. CHERADAME

I. Introduction

The process of the transformation of coal into coke is unknown. Bibliography and CERCHAR'S own work have provided only fragmentary information, such as measures of quantity or indices selected arbitrarily, from which no comprehensive explanation has as yet been derived.

When it is known why this or that coal or blend of coals swells or why this or that coke cracks, simple measures will suffice to deduce the possibility of carbonizing coal either alone or in any mixture.

That is why the CERCHAR laboratories have undertaken this work. Very interesting observations have been made, notably with its new techniques of X-raying the coking; new and numerous measurements bearing on physical and mechanical properties (contraction, density, resistance to extension, etc.) have also been made. It is hoped that many obscure details may soon be eliminated, but the hoped-for solution has not yet been obtained.

Under such conditions, it is possible to:

(*a*) Give some exact data concerning the results obtained with different French coals; but the application to other coals is always doubtful. Many identical measurements are necessary before it can be asserted that, in two regions as far distant as the Saar and Lorraine coalfields are from those in Latin America, two types of coal are identical. Even then, an experimental test, such as can be conducted in existing experimental coking plants as, for instance, that of Marienau in Lorraine, is an indispensable precaution.

(*b*) Furthermore, the data received up to now on Latin-American coals give no information whatever about their characteristics. The percentage of volatile matter is very insufficient information. The Gieseler index given for two Colombian coals obtained with a plastometer is not standard in Europe and does not furnish the author with information. The measurements depend very much on the instrument and on the experimental procedure. It is not possible therefore to foresee in this paper an application of CERCHAR'S results to Latin America.

Knowledge of the ash content is no guide to whether it can readily be brought down to the figures required for use in the blast furnace, nor whether it is possible to eliminate the sulfur when its percentage is too high —a problem which does not exist in France—nor the nature of the ash, a factor which probably has an appreciable bearing on the formation of coke and its properties.

It is indispensable also to know the type of coke to be produced.

The qualities of French cokes are measured by a test with the "Micum" drum, which consists of drawing a 50 kg. sample of the size category over 50 mm. and submitting it to a 4-minute rotation in a standard drum, after which the product is screened through 10 and 40 mm. mesh. The percentages under 10 mm. and over 40, called respectively the M 10 and M 40 indices, are satisfactory empirical measures of the tendency to abrasion and of the cohesion of the coke.[1]

French metallurgists use cokes for which M 10 is in principle under 8 and M 40 higher than 75 and often 80. However, although the Ruhr cokes may for instance be much better than the above figures, some countries use cokes which would not meet these conditions. Most French furnaces would probably get used to a poorer coke than the above if they could get no other type, and provided that its quality were constant. On the other hand, the burdens for French blast furnaces are essentially composed of a very peculiar ore, siliceous or calcareous ore from Lorraine, averaging only 30 to 35% iron.

Consequently a blend deemed unsuitable for producing coke for a French furnace might be considered good in Latin America. This is all the more important inasmuch as several tests have given rather inadequate results. It will be seen later that the experimental conclusions are considerably improved if coke can be made with 68 or even 70 M 40 index.

Similarly other conditions (ash, sulfur, etc.) should be indicated which may affect the use of coke in Latin America.

Finally, two economic considerations are determining factors in any study of this kind, distinguishing mixtures to be studied from those which will not be considered.

Such questions arise as the maximum production cost admissible for the coke; the feasibility of adopting expensive transformation techniques, and so forth, as well as which foreign coals can be considered as normal and constant sources of supply for incorporation in the blends.

French tests are obviously limited to solutions which would be economically possible for France. Other solutions may apply in other countries, some of which may be technically more favourable.

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[1] J. Sabatier, according to the work of the Institut de recherches de la sidérurgie, points out that the following ratios exist between these two indices and the indices ST 2 in. (oversize with the 2 in. screen) and ST ½ in. (oversize with the ½ in. screen) of the British shatter test:

M 40 is, on an average, lower by 12 to 15 points than ST 2 in.

M 10 is, on an average higher by 4 to 6 points than ST ½ in. (Communications to the 4th Congress of B.C.R.A., 25 October 1950).

Table 1

CHARACTERISTICS OF THE COALS MENTIONED

		Swelling index	*M.V. on pure organic matter*	*Gray King index*	*Elementary composition*			*Dry coal ash excluded*	
					C	*H*	*N*	*S*	*O*
I. Main categories of the Saar-Lorraine district	Gras A	7–9	31–36	G_7					
	Gras B	5 to 7	34 to 40						
	"Flambant gras"	2 to 5	35–42						
	"Flambant sec"	2	37–44						
II. Selected coal from this district	Saint Charles (Gras B)	6 to 7	40	G to G_1	82.2	5.7	1.1	1.0	7 [b]
	Sainte Fontaine (Gras B)	6 to 7	35	G_2	79	5.5	1.2	0.7	9 [b]
	Wendel (Flambant gras)	5	35		77.8	5.1	0.9	0.7	15.5 [a]
	Simon (Flambant gras)	2.5	35 to 40	B to C	72.6	4.9	1.0	0.7	20.8 [a]
	Camphausen (Gras A)	8–8.5	36	G_7	80	5.4	1.1	0.8	12.7 [a]
III. Other French coals mentioned	Courrieres (Coking)	9	28	G_{10}	86.3	5.3	1.3	0.7	6.4 [a]
	Dourges (Coking)	9	25	G_9	86.3	4.9	1.4	0.6	5.3 [b]
	Drocourt	9	25	G_9	84.0	5.2	1.4	0.6	8.8 [a]
	Fuveau	0 to 1	45 to 30		71.2	5.3	1.8	4.9	15 [b]
IV. German coals (various origins)	Ruhr (Coking)	8.5	25		86.5	4.8	1.6	1.2	5.9 [a]
	Aix-la-Chapelle	3	12 to 18						
	Carolus Magnus	3	12 to 18						

Bibliographic references.

Swelling Index: Norme AFNOR NF-M 11 001.

British Standards Institution No. 1016–1942.

"British standard methods for the analysis and testing of coal and coke", pages 64 to 70.

[a] Oxygen determined by difference.

[b] Oxygen determined directly.

Definitions of the Gray King Test. The coking power is defined by comparison of the bottoms obtained when cooking the coal in a retort with circumferential heating (50./minute from 300 to 600). The classification is made by reference to a scale of samples, from anthracites to the low-swelling coals, each one referring to an index A, B, . . . G, in the case of the highly swelling coals the characteristic is the quantity of inert matter (electrode carbon standard) just necessary to avoid increase of volume during coking: G_1, G_2 if 1/20 or 2/20 are enough.

II. FRENCH INDUSTRIAL RESULTS

This section deals only with the studies relating to the use of large quantities of coals from the Saar and Lorraine. These coals all have a high percentage of volatile matter, but swelling and binding properties differ widely. Table 1 shows this diversity and attempts to define their characteristics as well as those of all the coals mentioned in this paper.[2]

The addition of small quantities of meagre coals or high-volatile coals to coals approaching self-coking grades will therefore be omitted.

The following four series of French studies may be considered:

(*a*) Studies made at Thionville with a view to increasing the percentage of Lorraine coal in the blend, without using any special technique;

(*b*) Studies made at Carling on blends containing a very high proportion of Lorraine coal—technique including addition of coke-dust and stamping;

(*c*) Studies made at Marienau on blends with a very high proportion of Lorraine coal, the process including addition of semi-coke dust and the dry feeding of the ovens;

(*d*) Studies made at Thionville on the Burstlein process, which consists of a selective crushing of the various fractions of Lorraine coal to be introduced in the blend.

The results of series (*a*), (*b*) and (*d*) have been obtained on an industrial scale. Large quantities of coke have been and are still being produced and used in French blast furnaces.

Series (*c*) was tried on an industrial scale for one month. The result is somewhat unsuitable for the Lorraine iron works, but may be considered as satisfactory under economic conditions differing from those in France.

First Thionville studies

The Thionville coking plant normally used blends made up of the following coals:

Courrieres	Classical coking coal
Carolus Magnus	"De-fatting" coal (*amaigrissant*)
Saint Charles	Lorraine—"gras B"

Attempts were made to increase the proportion of the last-named as much as possible.

By using, instead of the 0–6 mm. the 7–15 mm. size crushed at the time of use, the following limit could be reached:

Courrieres	21%
Carolus Magnus	12%
Saint Charles	67%

The resulting coke presents the following characteristics:

M 10 = 10
M 40 = 57 to 61

It cannot be used in the blast furnace alone, but it has been used without difficulty when mixed with other cokes.

With 50 to 60% of St. Charles, M 40 is about 60 to 66, a value probably acceptable in other countries; and with 40 to 50% of the same "gras B", the coke can be used in France directly, without blending.

It has been pointed out that the use of "peas" was necessary to achieve that result. This may be due to the fact that the Lorraine coal in the fine state is readily oxidized and loses its binding properties during transport from the mine to the plant, and to the fact that the proportion of *fusain* (hard coal) is lower in the 7–15 mm. grade than in the 0.6 mm.

Carling technique

Three blends are used at will in the Carling coking plant.

The composition is as follows:

	A	*B*	*C*
Lorraine "gras"[3]	75	48	–
Lorraine "flambant gras"	–	24	62–65
Coke dust	16	12	7
Coking coal (Pas de Calais or Ruhr)	9	16	28–31

The coke obtained is homogeneous, with a very regular structure. Microscopic examination shows that the walls surrounding the pores are perforated by very fine channels.

The coke pieces are roughly cube-shaped.

The Micum test results are as follows:

M 40 = 79 to 81.
M 10 = 7 to 8.

The density in bulk is 450 kg. of dry coke for 1 cubic metre.

Porosity varies between 45 and 50%.

The coke is highly combustible.

From these mixtures, the Carling coking plant has made metallurgical coke since the war. In order to develop the use of "flambant gras" coal it is now preferred to use the B and especially the C formulae. The coke in either case is of the same quality.

Since 1948, 500,000 tons of coke have been thus produced and consumed by the 500 tons per day blast furnaces of the Lorraine iron region. The furnaces of Rombas have consumed four fifths of the total.

The process consists in: the preparation and crushing of these ternary mixtures; stamping of the oven charge to a dry coal density of about 1.0; carbonization at the highest possible speed for the ovens.

The equipment may be described briefly as follows:

The preparation plant, like all of its kind, is equipped with storage bunkers, weighing and crushing devices. It is recommended that the latter should produce the least possible quantity of duffs (under 0.5 mm.) and small peas (over 3 mm.) The only special feature consists, in the case of the use of coke-dust, in the presence of a subsidiary drying and pulverizing plant for that dust. Ball mills are used to ensure the

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[2] In the table referred to, Fuveau coal is also mentioned. It is a tertiary coal resembling sub-bituminous American coals, but which has not been submitted to CERCHAR's tests.

[3] The various coal categories of the Saar-Lorraine district—Gras A, Gras B flambants gras and flambants secs—are defined at the end of this paper.

crushed product does not contain material over 0.3 mm.

The crushed blend is stored in a tower. The charge to be put in the oven is prepared in a special machine fed from the tower, running in front of the ovens, and consisting of the following parts:

(*a*) Reserve bunkers with sufficient capacity for loading several ovens;

(*b*) A mould, the bottom (shovel) and the ends (shields) of which are mobile. The briquette is stamped in this apparatus;

(*c*) A stamping device comprising two mobile cars each with three stamps;

(*d*) A charging device for loading the oven with the compressed briquette, which is carried by the shovel and held up at the ends by the shields;

(*e*) A discharging device (similar to the usual coke oven type);

(*f*) A door-lifting device.

A small subsidiary machine, running over the ovens, handles the front shields. It takes them out of the ovens and returns them to the main machine.

The ovens are built of silica brick. The only special feature distinguishing them from ordinary coke ovens is that the proportion of the height to the width of the briquette must remain below a given limit. The stability of the briquette depends on this ratio for given stamping conditions. The limit increases with the progress made in the technique and is now about 8.5.

It should be noted that this limitation of the height of the oven for a given width in no way entails a decrease in the daily load as compared with a normal coke oven. In fact, the useful volume is decreased, but the charge is increased owing to the high density, whereas the carbonizing time is practically the same.

The other equipment (coke screening, by-product plants, etc.) is just the same as in a normal coking plant.

The Marienau technique

The industrial test has been made on about 12,000 tons of coke. This coke was produced at Thionville, by charging a dry blend of the following composition:

"Gras B" (Sainte Fontaine or St. Charles)	60%
"Gras A" (Camphausen)	25%
Semi-coke	15%

As the Thionville coking plant is not equipped either for drying the coal or for the manufacture of semi-coke, the coking blend was prepared in Marienau and sent to Thionville. The production of Marienau being much less than the consumption of the blast furnace (about 12,000 tons a month), preparation and coking of the mass was carried out at intervals between July 1950 and February 1951. The semi-coke used was obtained in a rotary kiln by treating a high-volatile, scarcely coking ("flambant gras") coal (Simon), or a non-coking, flaming coal ("flambant sec"). It is characterized by extreme fineness of the particles: 45 to 75% < 0.5 mm.

The coke produced is distinguished chiefly by its low density and its rather fine size distribution.

The figures are as follows:

Sizes	>80	48%
	60 - 80	30%
	40 - 60	18%
	0 - 40	4%
Micum test	M 40	70
	M 10	6.5
Density	0.450	

Although the M 40 may be inferior to the usual French values mentioned above, the blast furnace test was conducted without incident. The consumption per ton of pig iron is slightly more than with the usual cokes. Furthermore, the composition of the pig iron was somewhat irregular.

The coke made at Thionville, however, was not as good as the coke produced at the experimental Marienau plant when developing the process.

In these tests, using the same mixture, the M 40 index reached 75. The reason for the difference is doubtless the alteration of the coking blend between its preparation at Marienau and its treatment at Thionville (average delay: 8 to 10 days). It is to be assumed, therefore, that normal manufacture would yield a coke suitable for our plants.

To recapitulate, the process consists in:

(*a*) Drying the coal when de-dusting;

(*b*) Preparing a ternary mixture made of:
  (i) Lorraine "gras" or "flambant gras".
  (ii) Addition of coals rich in bitumens.
  (iii) Semi-coke.

(*c*) Carbonizing this mixture at the highest possible speed of heating.

Equipment

The main features are the following:

The drying of the coals used must necessarily be effected with a contact time between coal and hot gases and a rise of temperature as low as possible to avoid oxidation. For that purpose, an apparatus is used in which the coal falls freely against a stream of hot gases circulating the other way.

The coals are separately dried, cooled, crushed and stored in bunkers.

The semi-coke is next produced in a revolving kiln with indirect heating. The upper part is arranged to permit a preliminary oxidation if the flaming coal used is caking. The semi-coke obtained is cooled and stored in bunkers.

The preparation of the mixture, the blending in a revolving drum, the storing into a tower, the charging, present no difficulties when compared with a normal coking plant, except inasmuch as handling of a fine, dry product is concerned. A few easy precautions have, however, to be taken against the formation of dust clouds.

The coking ovens, built of silica brick, are of a classical type, like the rest of the equipment.

The SOVACO technique

The SOVACO cell at Thionville was put in service at the end of September 1951. It has worked since that date and treats about 1,400 tons a day of blend good, the results being practically the same as those of a perfect separation. Magnetite consumption is about 1.5 kg. per ton.

More careful supervision of the circuit is required than in the previous case, because any abnormal increase in the viscosity of the medium determines a loss in the "duff" (whereas the cyclone can accept high-viscosity media while treating the 0.5–10 category).

Other centrifugal machines

The PIC company offers an apparatus consisting of a juxtaposition of washing cylinders. The results on the semi-commercial scale seem satisfactory. A commercial-scale installation is being built in France.

CERCHAR is now studying a machine already described at the Congress on Coal Preparation (Paper E 5)—the mechanical cylinder. Built since that date with a 275 mm. diameter, it gave the same results as the cyclone for de-slimed products. Research will be continued using larger diameters.

Dense-media cones

This intermediate process may in some special cases constitute a good solution, as it permits treatment of products considerably below 10 mm. France has had no great experience with these processes (only one recent installation). Separation has proved excellent down to 5 mm. and not as good below 5 mm. It may be quite good, however, if the coal is easy to wash. Prior study of the coal's washability is necessary before selecting this process.

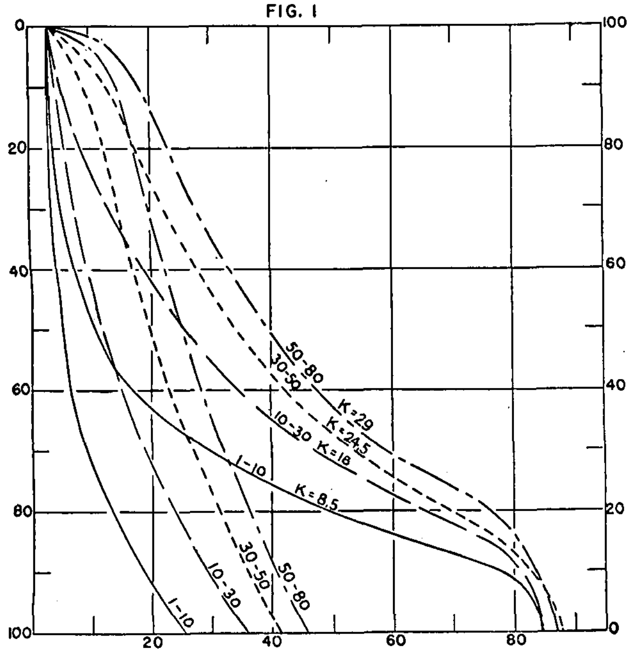
Examples of solutions adopted in the case of coals difficult to wash

(a) *Le Bousquet d'Orb colliery*

Figure 1 shows the washability curves for the various grades. Calculation of the washability indices k gives the following results:

50–80................................k=29
30–50................................k=24.5
10–30................................k=16
1–10................................k= 8.5

FIG. 1



The small sizes are relatively easy to wash, and will be treated in a jig. The > 10 are exceptionally difficult to wash and will be treated by heavy media.

(b) *Graissessac colliery*

Figure 2 shows the washability curves of the various grades. The k indices are:

50–80................................k=25
30–50................................k=24.5
10–30................................k=16
1–10................................k=11

The desired ash content—12%—for the 1–10 "fine size" (peas in Great Britain) is obtained under good conditions with a jig. The larger categories will be cleaned with heavy media.

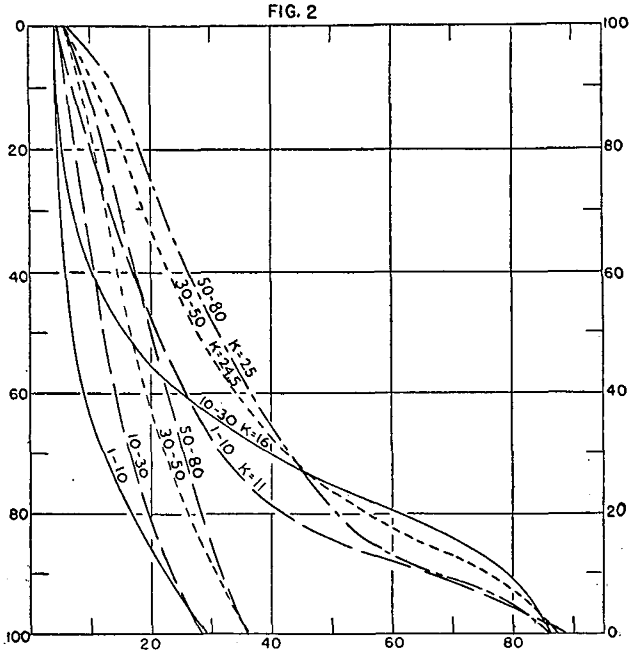
Conclusion

The new washer, which can treat coal from these two neighbouring mines, will consist of:

(i) A modern jig using a feldspar bed and compressed air ($I = 0.15$) for the small sizes (peas in Great Britain);

(ii) A heavy media "drewboy" made by the PIC company for the *grains*.

FIG. 2



(c) *Champagnac colliery*

Figure 3 shows some washability curves for the different classes of raw coal.

Group 0.3–10 would appear to have an index k = 10.5, and could be treated in a jig. But the 10–60, prepared so as to obtain a saleable coal with 16% ash, gives a large quantity of middlings.

It has finally been decided that the 10–60 fraction of the middlings would be crushed down to 0.3–10, added to the raw "small size" coal (they represent in weight 18% of the raw "small size" coal and bring k corresponding to a daily production of about 800 tons of metallurgical coke. That volume is consumed entirely by the blast furnaces of the Thionville steel plant.

At present, the blend for which that technique seems to be best suited and which has been used for several months, has the following composition:

Wendel (Lorraine "flambant gras") coal	40
Lorraine "gras B"	20
"De-fatting" (Aix-la-Chapelle basin)	10
Coking coal (Ruhr basin)	30

The coke obtained is of regular texture—the results of the Micum test are the following:

M 40	77 to 80
M 10	7 to 8

Bulk density of the dry coke: 0.410 to 0.420 kg. cu.m.

The process is based on the well-known fact that a partial crushing of some coals followed by screening, gives fractions of varying compositions (in petrographic constituents and in ash contents) and that it is worthwhile to eliminate some of these fractions and to re-crush, more or less finely, each of the others.

The Sovaco-Burstlein process applies this technique with various special procedures. More particularly, in the present form of application, a first partial crushing followed by a screening, gives: an intermediate fraction, which is a concentrate of "vitrain" (bright coal) and is withdrawn from any subsequent crushing, remaining in the form of small grains; a coarser fraction which is a concentrate of "durain" (hard coal) and above all of "clarain" (soft coal) and is re-cycled with the finest fraction for further crushing to a very fine size, while avoiding the formation of fine dust; finally, the optimum charging density is obtained by granulometrical control of the final mixture and by addition of oil.

Equipment

It consists of:

Electric-heated, non-clogging screens, which permit an exact granulometric classification, even with wet coals.

Disintegrators working by percussion in order to avoid the formation of fine dust.

Double cage mixers.

Figure 1 shows an example of application.

The remaining equipment of the coking plant is identical with that of other coking plants. More particularly, there are no dimensional restrictions for the coking ovens.

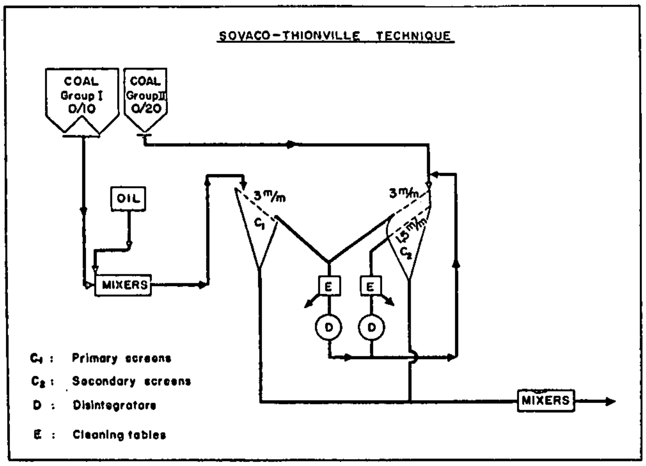


Figure 1.

III. Economic information about these processes

Investments[4]

These have a common basis for each process, with complements or modifications.

For example, in France, the common basis is 20 million francs for each oven having a daily capacity of 16 to 17 tons of total coke. That includes the oven itself together with a quota for the subsidiary devices: oven machinery and fittings, quenching tower, coke wharf, coal tower, electric equipment and civil engineering work.

The particular features of the various processes are the following:

A. *Carling.*

Installations for drying and for pulverizing the coke-dust

For a complete equipment (with buildings and other works) capable of treating 10 tons per hour of wet dust, corresponding to a coking plant of 1,400 tons per day capacity, the total investment would be about 100 million francs.

Oven machinery

The set of machines, in working order (charging, discharging), coke-cake guide, "shield car" for a battery where charge is stamped, costs approximately 150 million francs. It can serve, at the most, a battery with a total output of 900 tons of coke per day.

For comparison, the set of similar machines (charging, discharging, etc.) for ovens with overhead charging would cost 125 millions, but it could serve an installation with total daily output of 1,300 tons of coke.

To sum up: for an industrial ensemble producing 2,600 to 2,800 tons of coke per day, the supplementary investment would be 200+450−250=400 million francs—that is, 2.5 millions per oven with a 16-to 17-ton daily capacity.

B. *Marienau*

Coal-drying installation

An installation with four dryers of 35 tons per hour, with one more in reserve would cost approximately 225 million francs. It would serve a coking plant producing 1,300 tons a day of total coke.

Semi-coking installation

The complete installation with its coal preparation, its Bruay-Carbolux type ovens, and the by-products plant would cost about 300 million francs. It could produce the 300 tons of semi-coke per day required for a coking plant of that size.

Thus 525 million French francs are required for 1,300 tons a day, or a little over 6 million francs per oven.

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[4] For easier comparison, figures are repeated here from the author's earlier paper at this meeting:
One ton of steel girders (beams, joists) costs about 40,000 francs.
One cubic metre of reinforced concrete, erected, costs 35,000 francs.
The cost (wages and social charges) of a highly skilled workman is about 350 francs per hour.

C. *Sovaco-Thionville*

Installation for petrographic preparation

A complete installation would cost about 250 million more than the normal preparation. It could supply a coking plant producing 1,100 tons per day of total coke.

The supplementary expenditure would thus be 4 million francs per oven.

Working costs

They cannot be estimated for the Marienau process, which has not been put on a permanent industrial basis, and no figures are available from the Société Sovaco. Those corresponding to Carling can be given, however, and it is better, perhaps, to present them in physical quantities rather than francs.

Table 2 gives these results per ton of total coke produced, for a coking plant producing 350,000 tons of coke per annum.

Table 2

RESULTS OBTAINED AT CARLING

Fuels charged at 10% water	
"Flambant gras"	0.860 t
Additional coal	0.425 t
Coke-dust	0.135 t
Manpower	
Operation: staff	0.12 man/hour
Operation: workers	1.06 man/hour
Maintenance: staff	0.015 man/hour
Maintenance: workers	0.20 man/hour
Heating gas:	
Coke ovens (blast-furnace gas)	890 th. lower calorific value
Dust drier	22 th. lower calorific value
Supplies:	
Maintenance equivalent to	0.75 man/hour
Internal transport equivalent to	0.50 man/hour
Steam	245 kg.
Power	36.6 kWh.
Water	3.3 cu. m.
Sulphuric acid 60° B	10.2 kg.
Sulphuric acid 66° B	1.1 kg.
Washing oil	0.9 kg.
Miscellaneous, equivalent to	0.125 man/hour
Corresponding production would be:	
Coke:	
Metallurgical grade	0.842 t
20–40	0.068 t
10–20	0.330 t
Dust	0.060 t
Gas at 4.2 th. lower calorific value/m3	
By-products:	
Tar	41 kg.
Sulphate of ammonia	10.2 kg.
Washed benzols	14.8 kg.
Naphthaline and used oil	1.1 kg.

IV. RECENT LABORATORY RESEARCH

It is mainly at Marienau that search for satisfactory formulae is carried out, while at the same time studies continue relating to the manufacture of semi-coke.

After trial in the blast furnace, of a formula containing only Saar-Lorraine coal, blends have been tested containing a certain proportion of Ruhr coal (Blumenthal):

1. Ternary mixture: "Gras B", Ruhr, semi-coke.

Eighty tests, bearing on 17 different mixtures, have led to the following formulae, which are almost the same:

"Gras B"	65	65	60
Ruhr	25	30	35
Semi-coke	10	5	5
	100	100	100

The resulting coke has an M 40 of about 75.

2. Ternary mixture: "Flambant gras" Ruhr, semi-coke.

A series of systematic trials in an experimental oven has indicated the formula:

"Flambant gras" (Wendel)	57
Ruhr	35
Semi-coke	8
	100

Of this mixture, 3,800 tons, prepared at Marienau, were coked at Thionville between 8 July and 13 August 1951. The characteristics of the coke obtained were as follows:

	Thionville	*Marienau*
M 40	73	73.5
M 10	9.4	9.1
Density	0.4	

Thus, several formulae have been found giving a coke the M 40 of which is certainly higher than 70.

The Marienau station is also conducting studies on the production of semi-coke. The rotary kiln brought into existence in Bruay twenty years ago is the only industrial process. It is satisfactory, but the product is not perfectly homogeneous. It should be possible today to find a more economical process giving a better product.

Trials with the Metalcoke oven have failed up to the present. Transfer of good laboratory results to the pilot plant raised engineering difficulties which have not yet been overcome. Nevertheless the tests continue.

Several chute ovens are being tried, but on a small scale only.

Finally, good results have been obtained at Verneuil with an oven producing semi-coke by the fluid method on the scale of 100 kg. per hour, and a trial unit for this process with a capacity of one ton per hour is being built at Marienau.

CONCLUSIONS

The above results appear to lead to the following conclusions:

1. The use of large quantities of non-caking "flambants secs" has not become regular practice in France, but it has been noticed that the Marienau blends yield a coke with an M 40 index comprised between 70 and 75 and contain 15% of semi-coke which may be obtained from 1.3 times its weight of such coal.

2. Coke-dust does not have exactly the same effect as semi-coke duff. According to the technique and doubtless to the blend used, one or the other is more efficient. Normally, the required percentages of coke-dust would have the economic disadvantage of requiring a certain amount of calibrated coke to be crushed. At Carling, there are both domestic coke and metallurgical coke batteries, so that it is not difficult to obtain there the required quantities of coke-dust. The operation is profitable, that dust having no commercial value.

3. The Carling and Sovaco processes consume the same kinds of Lorraine coal, that is to say, those with a swelling index higher than about 4.

The Carling process has at present the advantage of consuming more of that coal (see section on Carling technique); the B formula corresponds to $\frac{72}{72+16}=82\%$ of raw coal "gras+flambant gras", as against 60% in the Sovaco blend. The C formula gives $\frac{65}{65+28}=70\%$ of "flambant gras". It is less expensive to erect and doubtless operating costs are lower. It is, however, difficult to draw definite conclusions, as the Sovaco test is too recent.

4. It should be remembered that, without any complementary installations (first Thionville studies), it is still possible to incorporate 40% of "gras B" in the blend, which may suffice to solve some particular cases.

There can be no doubt that among these results, some may be applicable in Latin America. But, here again, more information is required concerning the coals and the problems of these countries. Emphasis must be laid here on the usefulness of the efforts to standardize the tests defining the coking properties of coals, and in particular the work of the International Coal Commission in Geneva, in which CERCHAR laboratories have taken an active part.

The Blending of Western Coals for the Production of Metallurgical Coke

JOHN D. PRICE

INTRODUCTION

This paper aims at exploring the various phases of coal blending and at presenting information which may make it possible to correlate the work being done elsewhere with a view to improving the coke made from Latin-American coals. It is of particular interest to consider the use of the coals from the western part of the United States in this regard, for these coals are largely of the same geological age and exhibit many of the same tendencies towards the production of inferior quality coke as do the Latin-American coals.

The data and information contained in this paper may in many cases apply only to a limited number of coals and the benefits of the various blending processes as shown will apply equally well to all coals.

PREVALENCE OF BLENDING

A survey (1) of the 86 coke plants which were in active operation in the United States during the year 1949 indicates that only 10.5% of the total charged one single rank of coal into their ovens, while the remaining 89.5% made use of blending in some form. This report indicated that of these total plants:

5 or 5.8% used straight high-volatile coal.
4 or 4.7% used straight medium-volatile coal.
47 or 54.6% used blends of high- and low-volatile coals.
25 or 29.1% used blends of high-, medium- and low-volatile coals.
2 or 2.3% used blends of high- and medium-volatile coals.
3 or 3.5% used blends of medium- and low-volatile coals.

If the term "coal blending" is to be applied in its broadest sense, even those plants which showed the use of a straight type of coal practised some blending, for they invariably used several coals of the same type or rank and paid careful attention to the proper mixing of these in order to secure uniformity of their coal charge and the quality of the resultant coke.

PURPOSES OF BLENDING

1. The general term of coal blending covers two fields; the first of these is the mechanical mixing of a number of coals to secure uniformity. Quite often it is found necessary to secure coal for coke production from a number of different mines; these coals, while of the same general type or rank, may differ in their chemical composition or in the physical qualities which they may impart to coke made from them. Again, it is not unusual to find that coal from different sections of the same mine may show similar variations. Under such conditions, it is necessary to introduce a system of blending in order to secure a uniform mixture.

2. The second type of blending, while also for the purpose of coke improvement, deals more particularly with the use of a blending agent differing in character from the base coal; it is this form of blending that will be discussed here, it may be utilized for the following reasons:

A. To improve the chemical and physical properties of the coke.

(1) Where a single coal or mixture of coals of the same rank does not produce a satisfactory coke. A high-volatile coal when used alone is likely to contract when coked so that a comparatively weak coke is formed.

(2) Where necessary operating practices are not conducive to the production of a suitable product. This condition is frequently encountered where a plant is operated to produce gas for sale as a public utility and at the same time to make coke suitable for foundry cupola use.

(3) Where a product of some special quality is required as, for example, where a plant ordinarily producing blast-furnace coke must at times produce a high-grade foundry coke.

B. To reduce objectionable impurities in the coke or to prevent damage to oven brickwork which would result from the use of a single coal.

(1) Certain coals may contain impurities, such as sulfur or phosphorus, which may have an objectionable effect in the ultimate use of the coke.

(2) Other coals, particularly of the medium- or low-volatile types, show excessive expansion pressures when coked and may cause distortion of oven walls.

C. To secure increased yields of coke or its co-products. Depending upon the economic values of the products it may be found desirable to increase the yield of one or the other.

D. To utilize a supply of some "particular" coal.

(1) To protect the reserves of high-quality coking coal. Blending for this purpose is fairly common in European countries.

(2) To utilize a surplus or inferior product not otherwise usable.

IV. Blending agents

Many materials have been used for blending purposes, the exact agent to be used depending both upon the condition to be corrected and the nature of the base coal. No universal blending agent, which can be used to improve the coke made from all coals, is available. The nearest approach to this will be found in the use of a low- or medium- volatile coal as practised at 75% of the plants in the United States. But even in this event each plant must work out the proper type and quality of coal to use and must carefully consider the economics of the practice. In effect, coal blending is an individual plant problem and must be worked out by each plant to secure best results. Among the blending agents which have been used are the following:

1. Coking coals of different volatile content or of different rank;
2. Coking coals of different expansion properties (very closely related to 1);
3. Low-temperature char, or partially devolatilized high-volatile coal;
4. Non-coking high-volatile coals;
5. Coke breeze;
6. Anthracite coal;
7. Petroleum-coke breeze;
8. Coal-tar pitch.

V. Substitute materials and procedures

1. The blending agent which appears to be most desirable to correct a certain condition is frequently not available, and can be secured only at excessive cost. In such event a substitute material can often be secured or produced. Thus, while anthracite fines are desirable for use in increasing the size of coke pieces, fine coke breeze can be used in its place with nearly as good results. Again, low-volatile coking coals are not always economically available, and in such localities low-temperature char produced from local high-volatile coals presents a very acceptable alternate. The use of such materials will be touched upon later.

2. While in most cases not as effective as the use of proper blending agents, certain alternate procedures have been practised at many plants with gratifying results. A few of these changed methods are described briefly below:

A. Control of carbonizing temperatures has an effect on the yields of coke and co-products and on the physical properties of the coke. For example, as the temperature of carbonization is decreased, the average size of the coke and its resistance to shatter are both increased, while the tumbler test index shows little change (2a). The yield of coke and of tar increase as the carbonizing temperature is lowered, while the yield of gas and of benzol products decreases. Ammonia recovery shows a peak at a carbonizing temperature of 800°C., decreasing with both higher and lower temperatures (3a).

B. Pulverization of coal to 70% minus ⅛ in. shows highest percentages of large coke and resistance to shatter, these values decreasing as the pulverization is changed either way from 70%. Tumbler index, on the other hand, increases constantly with finer pulverization (2b). The finer the degree of pulverization, the lower will be the bulk density of the coal charge and the higher the porosity of the coke (4).

C. Control of the moisture in the charge will have an effect on the bulk density of the coal since lowest bulk densities will be found within the limits of 6 to 8% moisture, the bulk density increasing as the moisture content is either reduced or increased from these limits (2c).

D. Washing of the coal prior to coking results not only in a reduction in the ash and sulfur content but also in an improvement in the shatter and tumbler indices of the coke (5a).

E. A change in the method of charging the ovens may prove of benefit. Where coal of an expanding nature must be used, an automatic charging device on the larry car may be an advantage (5b). A study of the distribution of bulk density zones in the oven will show that these are dependent upon a number of factors and greater uniformity throughout the oven may be secured by a modification in the method of filling the ovens (6a).

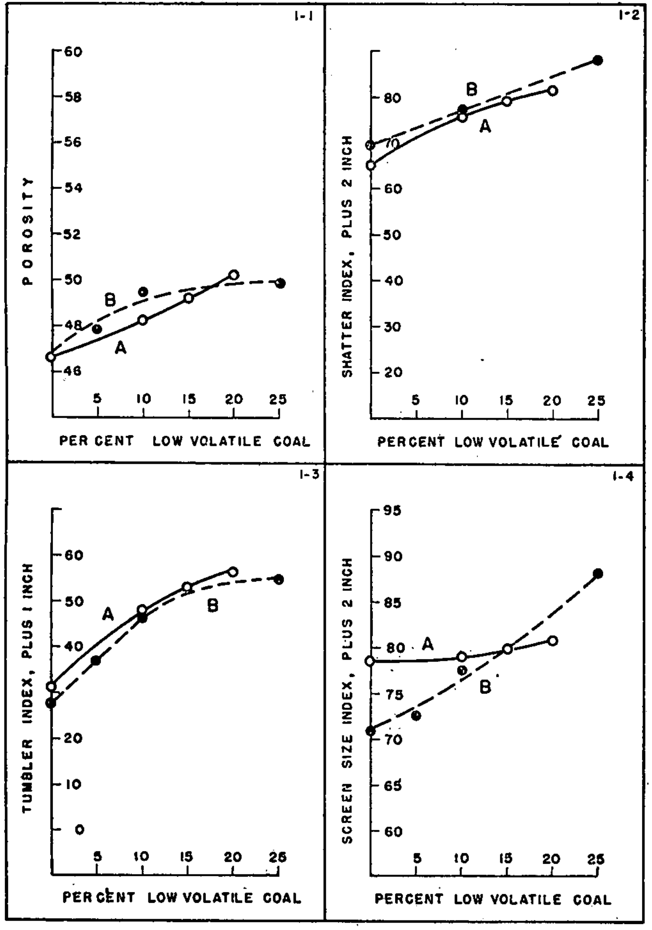
Use of blending agents to improve coke quality

It will be impossible to do more than touch upon the use of the many possible blending agents and on the results secured from their use. The examples will be limited to problems which have been encountered in the western portion of the United States.

1. Problem: To correct a deficiency in physical strength of coke made from a single rank of high-volatile coking coal. This is probably the most common problem in blending encountered anywhere in the United States and is of particular importance in the Rocky Mountain states, as only coals of this type are found here. Although some plants have been able to make satisfactory coke from straight high-volatile coal, these are few in number. The coals of the states of Colorado and Utah, when used alone, generally produce coke of inferior quality and the use of low-volatile coking coals, in amounts ranging from 10 to 12.5%, has become a common practice. The analysis of six Colorado and New Mexico coals are shown in Appendices A and D. These coals are used in varying proportions. A–1, Frederick, is a high-grade coking coal of heavy coking properties. It shows so high a fluidity that it is impossible to obtain accurate readings in a Gieseler plastometer test. A–2, Morley, and A–3, Koehler, are coals of lower coking properties. B–1, Bear Canon, is a coal of somewhat better coking properties than Koehler. B–2, Crested Butte, and B–3, Kebler, are semi-coking and non-coking coals respectively. All of these coals are regularly used in the high-volatile portion of the oven charge at the Colorado Fuel and Iron Corporation,

B-2 and B-3 being limited to 10% due to their coking deficiencies. The quality of coke produced from such mixture leaves much to be desired and has been improved by the addition of small amounts of low-volatile coking coal. Blending with Pocahontas coal, analysis given under C-1, Appendix C, gave considerable improvement (6b) and the results are shown as Curve A on Chart No. 1. The high freight on Pocahontas coal makes its use too expensive and a similar grade of coal from Arkansas (see C-2, Appendix C) has been used in its place with satisfactory results (8) and as shown by Curve B on Chart No. 1.

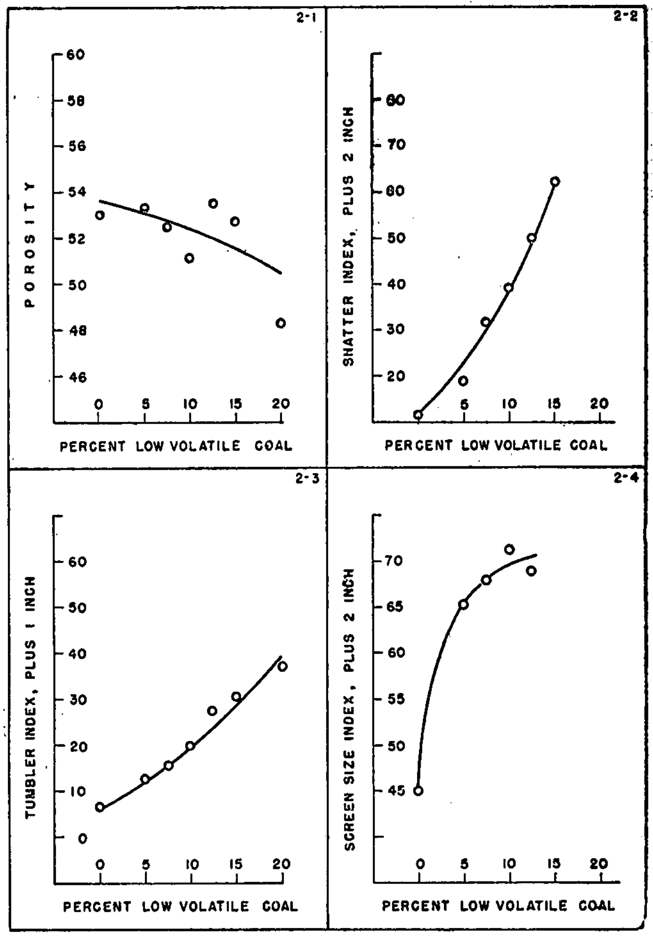
CHART NO. 1



2. Problem: To correct a deficiency in coke quality similar to that discussed in Problem 1, but where the poorer coking coals as found in Utah must be used as the base coals. Analyses of these Utah coals are shown on Appendix D. The blending with Sunnyside coal (D-2, Appendix D) of increasing proportions of Bokoshe coal from Oklahoma (for analysis see coal C-3 on Appendix C) gave very pronounced improvement (11a) and with results as shown by the curve on Chart No. 2.

3. Problem: To correct the same condition as expressed in Problem 2, but using some material other than low-volatile coal to supply the coke-forming bitumens lacking in the low-rank Utah coals. The use of petroleum-coke breeze and of coal-tar pitch have been suggested for this purpose. Petroleum-coke breeze of about 8.5% volatile content has been used (10a) with Frederick coal (Coal A-1, Appendix A) and with results as shown on Chart No. 3. Coal-tar pitch has been used (11b) with success with Utah coals (see Coal D-2, Appendix D) and the results of this addition are also shown on Chart No. 3.

CHART NO. 2



4. Problem: To correct the same condition as above, but where low-volatile coal, coal-tar pitch or petroleum-coke breeze are not available or their use may be undesirable. In cases such as this, it has been demonstrated, both by pilot and commercial-plant work, that the use of a low-temperature char gives a very satisfactory product. The story of such low-temperature char, its preparation and its use, would require a complete paper in itself. Its use, as do all other blending problems, requires individual study of the base coals to be used, both in the final oven mixture and the production of the char. The char, if best results are to be obtained, must be specially prepared for each case. The quantity of char used, its temperature of production, its volatile matter content, the degree of pulverization, all are critical and optimum values and must be determined in each individual case. The results secured from the experimental use of char, of the nature as shown as C-6, Appendix C., at the Colorado Fuel and Iron Corporation plant, using Frederick coal (10a) (A-1, Appendix A) is shown as Curve A on Chart No. 4. The results from the use of char of 18.0% volatile content made from Sunnyside, Utah, coal (D-2, Appendix D) with this same base coal are shown as Curve B on this same chart.

5. Problem: To produce a coke of special properties, as for example a coke which will meet the requirements of foundry cupola use. Such a coke is usually

CHART NO. 3

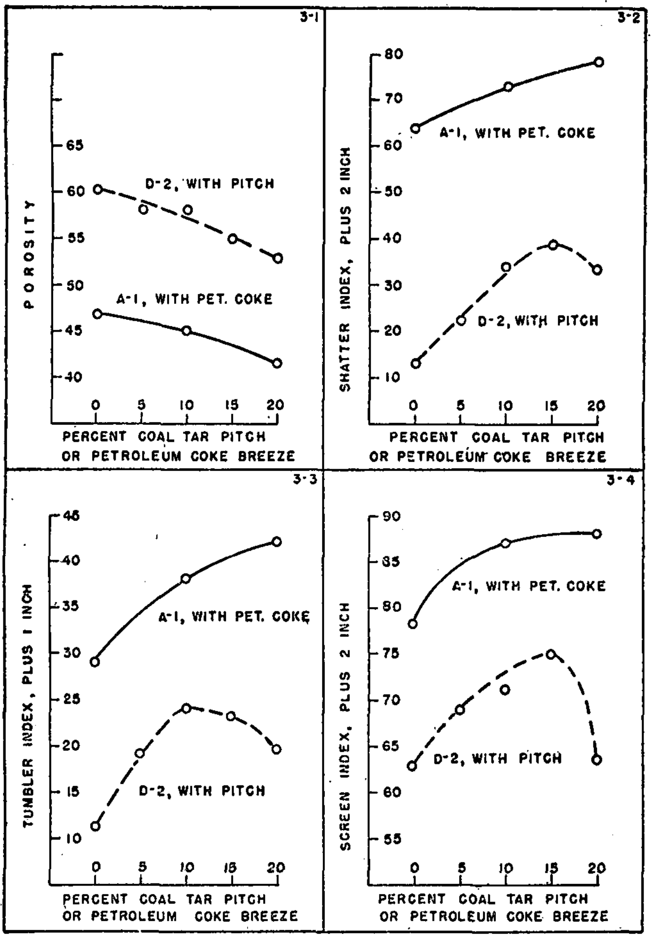
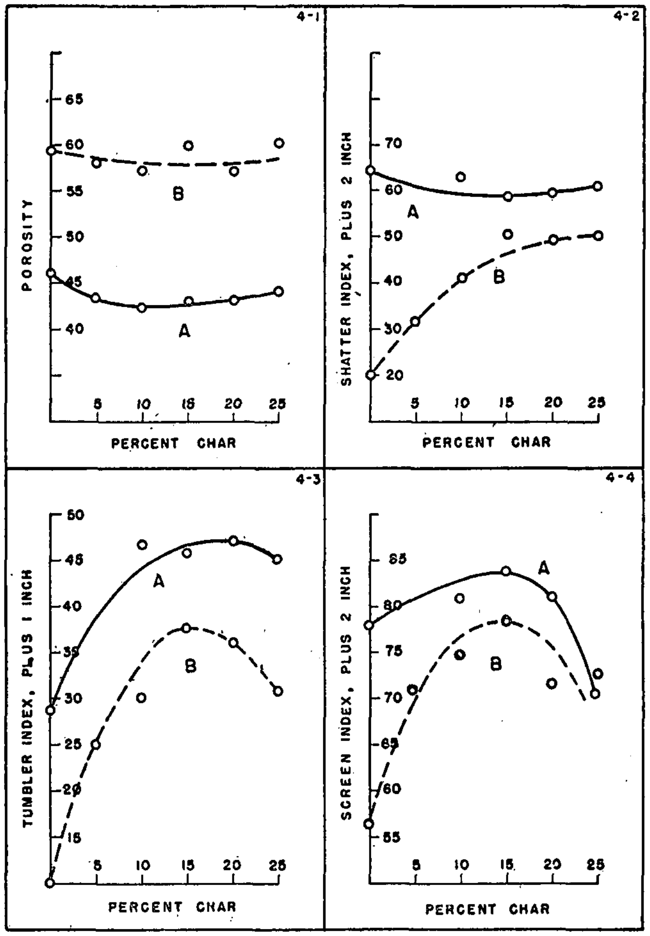


CHART NO. 4



required to be larger in size, of greater physical strength and lower in porosity than is coke for blast-furnace use. It has already been shown that increased proportions of low-volatile coking coal in the mixture will increase the shatter and tumbler indices of the coke, and an extension of coking time will increase the size of the pieces. Results (8) of this combination blending plus temperature control are shown on Chart No. 5, the following specific changes having been made:

	Base coal	*Furnace coke mix*	*Foundry coke mix*
Low-volatile coal, per cent[a]	0	10	25
Coking time, hours	22	22[b]	34
Oven temperature, deg. Cent.	1,100	1,100	1,000

6. Problem: To correct excessive contracting or expanding properties of the base coal (6a). Excessive contraction during coking is almost a universal feature of high-volatile coals when coked alone and is probably the main cause of the weak coke so formed. It appears that the high rate of contraction causes incipient fractures in the coke structure which in turn tend to reduce its resistance to shatter and to abrasion. The effects of expanding low-volatile coal and of neutral char have already been shown, and these are the more common means of correction (7). It might be expected that the use of small quantities of coke breeze would tend to correct the excessive contraction, but in general the use of this material for this purpose has a detrimental effect on coke strength (8). But coke breeze does have some beneficial effect when used in blends containing low-volatile coals as will de discussed later.

6b. Excessive expansion is an inherent characteristic of most, if not all, of the medium and low-volatile coking coals, such as are shown in Appendix C. While these coals, when coked alone, may yield a coke of fairly good strength, their expansion properties may be so great as to cause damage to the oven walls. Expansion pressures are usually found to be greater with the lower-volatile coals; it will be noted that as shown in the second section of this paper, no plant in the United States is using straight low-volatile coal, although four plants are using straight medium-volatile and three plants a blend of these two. Coke in Mexico is produced from a medium-volatile coal. This appears to be a middle-of-the-road coal; it has been coked without blending for many years in by-product ovens yet laboratory tests indicate it to exert dangerous pressures (5c).

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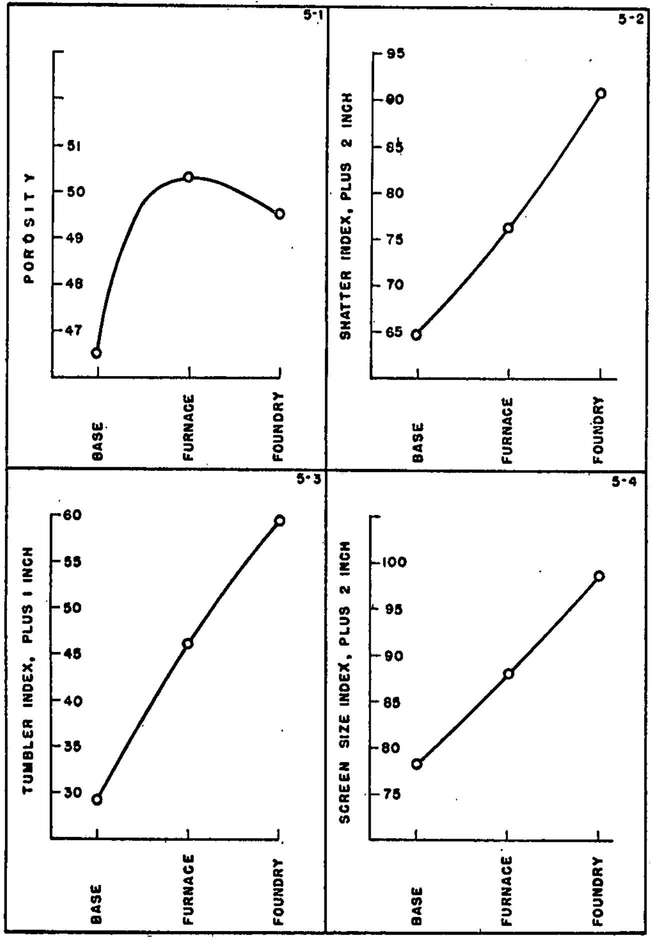
[a] The base coals used in these blends are listed in Appendices A and B, and the low-volatile coal used is that shown as coal C-2 in Appendix C. While the base coal (high-volatile) mixtures are *not* exactly the same when producing furnace and foundry cokes, this difference is not sufficient to have any effect on the physical properties of the two grades of coke.

[b] A coking time of 22 hours for the production of furnace coke may appear high. The plant from which this data was secured is operating ovens of 18.25 in. in width and, while they have frequently operated at coking times approaching 18 hours, the data available at 22 hours' coking time appeared to be most authentic. This is equivalent to 1,205 hours per inch of oven width. At this same rate, 17.15 hours' coking time would be used in an oven of 14.25 in. average width.

CHART NO. 5



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Appendix A

ANALYSES OF COLORADO AND NEW MEXICO HIGH-VOLATILE COALS

Mine	*Frederick*		*Morley*		*Koehler*	
Identification code	A-1		A-2		A-3	
Classification	Coking		Coking		Coking	
Kind	*Raw*	*Washed*	*Raw*	*Washed*	*Raw*	*Washed*
Moisture[a]	2.6	—	3.8	—	3.9	—
Volatile matter[b]	29.2	31.0	28.9	30.0	33.6	37.3
Fixed carbon	52.3	57.5	53.2	55.7	47.8	48.4
Ash	18.5	11.5	17.9	14.3	18.6	14.3
Phosphorous	.041	.039	.001	.001	.001	.001
Carbon[c]		84.61		83.72		82.09
Hydrogen		5.74		5.80		6.00
Oxygen		7.32		7.86		9.44
Nitrogen		1.80		1.80		1.67
Sulfur		.53		.82		.80
Free swelling index		8.5		7.0		3.5
Agglut. index 10/1		12.0		10.1		7.4
Agglut. index 15/1		—		—		—
Plastic range		—		392–473		388–456
Ash—SiO_2		53.10		70.31		61.42
Fe_2O_3		7.63		3.65		4.12
Al_2O_3		30.40		24.55		31.68
CaO		5.98		1.01		3.84
MgO		1.45		.96		.65
Source of information		(14)		(14)		(14)

[a] As received.
[b] Moisture free.
[c] Moisture and ash free.

Appendix B

ANALYSES OF COLORADO HIGH-VOLATILE COALS (*continued*)

Mine	*Bear Canon*		*Crested Butte*		*Kebler*	
Identification code	B-1	B-1	B-2	B-2	B-3	B-3
Classification	Coking		Semi-coking		Non-coking	
Kind	*Raw*	*Washed*	*Raw*	*Washed*	*Raw*	*Washed*
Moisture[a]	2.7	—	7.3	—	6.2	—
Volatile matter[b]	33.6	35.7	38.2	39.6	35.0	37.6
Fixed carbon	52.1	52.5	51.5	55.4	42.1	51.7
Ash	14.3	11.8	10.3	5.0	12.9	10.7
Phosphorus	.128	.132	.015	.014	.001	.001
Carbon[c]		83.23		79.78		77.62
Hydrogen		5.94		5.63		5.41
Oxygen		8.32		12.46		14.83
Nitrogen		1.71		1.68		1.48
Sulfur		.80		.45		.66
Free swelling index		6.5		1.5		—
Agglut. index 10/1		10.1		—		—
Agglut. index 15/1		—		—		—
Plastic range		388–475		—		—
Ash—SiO_2		50.58		48.67		48.87
Fe_2O_3		6.29		11.41		—
Al_2O_3		37.15		27.70		—
CaO		4.78		9.05		
MgO		1.33		2.34		
Source of information		(14)		(14)		(14)

[a] As received.
[b] Moisture free.
[c] Moisture and ash free.

Appendix C

ANALYSES OF LOW- AND MEDIUM-VOLATILE COALS

Mine *State*	*Carswell* *West Virginia*	*Philpott* *Arkansas*	*Bokoshe* *Oklahoma*	*Stigler* *Oklahoma*	*Char* *Colorado*
Identification code	C-1	C-2	C-3	C-4	C-6
Used in blend	A	B	A		
Shown on chart	1	1	2		
Moisture[a]	—	5.3	2.8	—	—
Volatile matter[b]	17.1	15.9	18.3	24.6	17.4
Fixed carbon	78.0	77.4	66.8	71.1	72.1
Ash	4.9	6.7	14.9	4.3	10.5
Phosphorus	—	.014	—	.007	—
Carbon[c]	91.2	89.33	90.9	87.43	
Hydrogen	4.6	4.58	4.4	5.32	
Oxygen	2.1	2.39	2.0	4.38	
Nitrogen	1.3	2.11	1.8	2.13	
Sulfur	.8	.93	.7	.74	.6
Free swelling index	9.+	9.+		9.+	
Agglut. index 10/1	20.+	20.+		17.7	
Agglut. index 15/1	—				
Plastic range	—	452–507			
Ash—SiO_2		43.13			
Fe_2O_3		11.53			
Al_2O_3		30.59			
CaO		8.81			
MgO		1.65			
Source of information	(6)	(14)	(8)&(11)	(8)	(10)

[a] As received.
[b] Moisture free.
[c] Moisture and ash free.

Appendix D

ANALYSES OF UTAH HIGH-VOLATILE COALS

Mine	*Castlegate*	*Sunnyside*	*Book Cliffs*	*Columbia*
Identification code	D-1	D-2	D-3	D-4
Classification	Lt.-coking	Lt.-coking	Lt.-coking	Lt.-coking
Kind	Raw	Raw	Raw	Raw
Moisture[a]	4.1	5.8	9.0	5.3
Volatile matter[b]	39.9	40.8	35.7	40.1
Fixed carbon	50.2	53.4	50.2	54.4
Ash	5.8	5.8	5.1	5.5
Phosphorus	.012		.019	
Carbon[c]	79.66		79.13	
Hydrogen	5.76		5.74	
Oxygen	12.68	8.7	12.68	10.0
Nitrogen	1.52		1.59	
Sulfur	.38	1.2	.86	.8
Free swelling index	2.0	4.8	3.0	4.5
Agglut. index 10/1	2.6	—	3.3	—
Agglut. index 15/1	—	3.9	—	3.0
Plastic range	—		—	—
Ash—SiO_2				
Fe_2O_3				
Al_2O_3				
CaO				
MgO				
Source of information	(14)	(9)	(14)	(9)

[a] As received.
[b] Moisture free.
[c] Moisture and ash free.

Appendix E

ANALYSES OF COALS FROM BRAZIL

	Brazilian coal				*United States coal*	
Mine or bed	*Santa Catarina*		*Barro-Branco*		*Frederick*	
Kind of coal	*Raw*	*Washed*	*Raw*	*Washed*	*Raw*	*Washed*
Moisture	1.4	1.4	—	—	2.6	—
Volatile matter	23.0	30.4	29.4	34.5	29.2	31.0
Fixed carbon	44.0	51.0	36.7	49.7	52.3	57.5
Ash	33.0	17.6	33.9	15.8	18.5	11.5
BTU per pound	9,200	11,700	9,600	—		
Carbon		86.6[a]				84.61
Hydrogen		5.9				5.74
Oxygen		4.2				7.32
Nitrogen		1.5				1.80
Sulfur		1.8	11.4	1.4		.53
Analysis of coke						
Volatile matter						
Fixed carbon						
Ash						
Sulfur						
Size, per cent +2"				80.7		74.7
Shatter index +2"				68.5		67.0
Tumbler +1"				52.4		29.5
+¼"				67.8		—
Porosity				42.0		46.6
Source of information	(13)	(13)		(12)		(14–8)

[a] Ultimate analysis on a different sample from proximate.

Appendix F

ANALYSES OF COAL FROM CHILE

	Chilean coal		*Corresponding coals from USA*
Mine or bed	*Schwager*	*Lota*	*Sunnyside Utah*
Kind of coal	*Raw*	*Raw*	
Moisture	2.5	—	5.8
Volatile matter	43.9	43.9	40.8
Fixed carbon	47.3	52.7	53.4
Ash	8.8	3.4	5.8
BTU per pound	13,300		
Carbon[a]	82.6	82.8	
Hydrogen	5.9	5.8	
Oxygen	9.0	9.3	8.7
Nitrogen	1.4	1.4	
Sulfur	1.5	.7	1.2
Analysis of coke			
Volatile matter			
Fixed carbon			
Ash			
Sulfur			
Size, per cent +2"	53.5	38.0	45.0
Shatter index +2"	22.4	14.3	11.5
Tumbler 1"	.4	.3	6.5
+¼"	56.4	58.6	—
Porosity	—	—	53.0
Source of information	(13–12)	(12)	(9)

[a] Ultimate analysis made on a different sample from proximate.

Appendix G

ANALYSES OF MEXICAN COKING COALS

	Mexican coals			*Corresponding coals of USA*	
Mine or bed	*Palau*		*Rosita*	*McCurtain Oklahoma*	*Stigler Oklahoma*
Kind of coal	*Raw*	*Washed*	*Washed*		
Moisture	—	—	—	.3	—
Volatile matter	19.8	21.6	24.5	23.6	24.6
Fixed carbon	54.5	61.2	60.5	69.8	71.1
Ash	25.7	17.2	15.0	6.6	4.3
BTU per pound					
Carbon			86.4		87.43
Hydrogen			5.5		5.32
Oxygen			5.1		4.38
Nitrogen			1.5		2.13
Sulfur	1.25	1.00	.8	.7	.7
Analysis of coke					
Volatile matter	1.0	1.0	1.0		
Fixed carbon	64.7	76.0	80.2		
Ash	34.3	23.0	18.8		
Sulfur	—	—			
Size, per cent +2"	—	—			68.9
Shatter index +2"	77.7	83.9	88.0		54.8
Tumbler +1"	33.5	46.8	48.0		50.0
+¼"	45.7	55.4	70.6		58.5
Porosity					58.4
Source of information	(5)	(5)	(5)		(8)

Low-Temperature Char as a Substitute for Low-Volatile Coal in the Production of Metallurgical Coke

JOHN D. PRICE

I. INTRODUCTION

In connexion with the blending of coals for the production or improvement of metallurgical coke it has been pointed out that the most common practice is the blending of low-volatile with high-volatile coking coals. This practice seems to be universally successful, regardless of whether the high-volatile coals possess high coking properties or are deficient in this regard. But the low-volatile coking coals are not always economically available, and in such cases it has repeatedly been demonstrated that char, made by the partial devolatilization of high-volatile coal at low temperatures, offers a very satisfactory substitute.

II. HISTORICAL

There is nothing particularly new in this idea. Early in this century Kotaro Shimomura received a Japanese patent for "the method of making a non-fingery coke out of bituminous coals without using natural coals of low-volatile matter". (1). About the same time Adolph Thau in Germany (2) and James G. West in the United States (3) described methods which they had developed for this same purpose. Practically all the early writers on this subject have agreed on one point; they have claimed that char, for optimum results in improvement of the physical properties of coke, should be made from the same coking coal or mixture which is later to be used in the preparing of the final mixture. This reasoning appears sound, for both char (provided its plastic properties are not completely destroyed) and the base coal will possess the same fusing temperatures and range and thus form a coke of truly homogeneous structure. This, however, has been disproved by tests made both in France (4) and in Colorado (5) where char made from a non-coking coal has given very satisfactory results.

III. LOW-TEMPERATURE RETORTS

The early days of this century brought about the development of hundreds of types of low-temperature retorts. Most of these are now entirely a thing of the past, and time will not permit even the listing of their names. A few have persisted in commercial, pilot plant or laboratory scale units and a list of such plants in the United States and Canada would include the following:

(1) Rotary drum retorts, externally heated
   (*a*) Disco, Pittsburgh, Penna., U.S.A.
   (*b*) Hayes, Pueblo, Colorado, U.S.A.
   (*c*) Caunt, Windsor, Ontario, Canada.
(2) Shaft type retorts, internally heated
   (*a*) Lurgi, Dickenson, North Dakota, U.S.A.
   (*b*) National Fuels Corp., New Haven, Conn., U.S.A.
   (*c*) Stansfield, Edmonton, Alberta, Canada.
(3) Fluidization type retorts, internally heated
   (*a*) U.S. Bureau of Mines, Denver, Colorado, U.S.A.
   (*b*) Singh Company, Chicago, Illinois, U.S.A.
   (*c*) Pittsburgh Consolidation Coal Co., Pittsburgh, Penna., U.S.A.
(4) Pan conveyor type, externally heated
   (*a*) Chicago, Wilmington and Franklin Coal Co., West Frankfort, Illinois, U.S.A.
(5) Vibrating conveyor type, externally heated
   (*a*) Storrs International Chemical Co., Salt Lake City, Utah; Urbana, Illinois; Chicago, Illinois, U.S.A.
(6) Vibrating deflector shaft type, externally heated
   (*a*) Petit Metalcoke Oven (France). To be installed at Pueblo, Colorado.
(7) Inclined cylinder retort, internally heated by super-heated steam
   (a) Records, Spokane, Washington; Wellington, Utah, U.S.A.

IV. SELECTION OF A PROCESS

To be fully acceptable from operating and economic standpoints, char-making equipment must first of all be demonstrated as capable of producing a char which will serve to produce a high-quality coke. Certain inherent principles of some of the processes may be found undesirable from this viewpoint. In addition, the process should meet as many as possible of the following requirements:

(1) Brisk agitation is necessary to prevent the coal from caking in large pieces (this is particularly true when the retort charge is composed wholly or in large part of coking coal);

(2) The apparatus is to be provided with the fewest interior projections or appendages possible, consistent with its proper working, in order to prevent semi-coke from accumulating and hanging on;

(3) For evident economic purposes, the operations of feeding and discharging must be continuous;

(4) Easy access to the inside of all parts of the retort throughout its entire length without stopping the working is required for inspecting and cleaning when necessary;

(5) Ease of regulation to suit different kinds of coal;

(6) It should not require that the coal be closely sized nor that all fine coal be removed for its satisfactory operation;

(7) Relatively high rate of daily throughput, in order to reduce the number of separate units required;

(8) Ability to operate at fairly high temperatures—1,400° to 1,500° F.—in order to secure the proper quality of char;

(9) Sufficiently rigid construction to permit it to withstand deformation in event of emergency shutdown. No auxiliary power source should be needed;

(10) Of simple and rugged design, thus making possible low repair and maintenance cost.