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De Castejon

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[54] **METHOD FOR THE PRODUCTION OF IRON AND STEEL**

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June 5, 1970 Spain..... 380451

Related U.S. Application Data

[63] Continuation-in-part of Ser. No. 142,255, May 11, 1971, abandoned, and Ser. No. 146,981, May 26, 1971, abandoned, and Ser. No. 148,856, June 1, 1971, abandoned.

[52] **U.S. Cl.**..... **75/11;** 75/38
[51] **Int. Cl.**[2]..... **C21C 5/52**
[58] **Field of Search**..... 75/38, 46, 55, 92, 52, 75/33, 34, 36, 38, 6, 7, 46, 11

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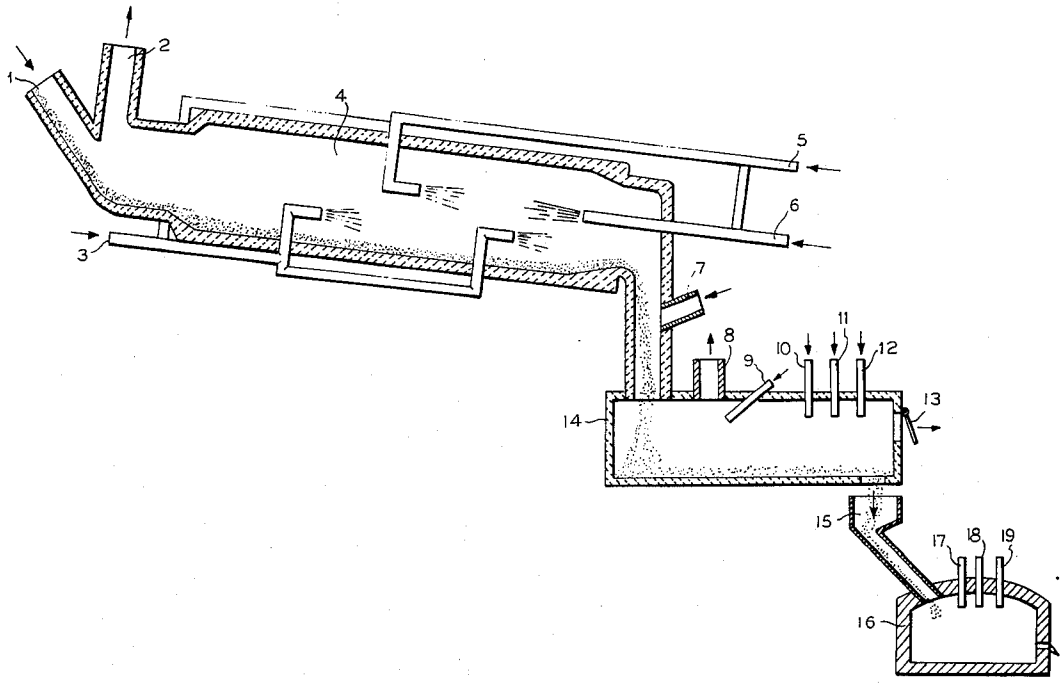
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Primary Examiner—M. J. Andrews
Attorney, Agent, or Firm—Robert R. Priddy

[57] **ABSTRACT**

Process for production of iron and steel. Although initial reduction is performed in rotary kiln or similar apparatus, products of excellent quality can be produced at lower cost than in processes based on blast furnace, irrespective of whether ore charged to kiln is of high or low gangue content and high or low in phosphorous and/or sulfur content. Moreover, improvements in economy of production are obtained even when ore is of low quality and final adjustment of composition to produce steel is carried out in direct arc electric furnace.

10 Claims, 8 Drawing Figures



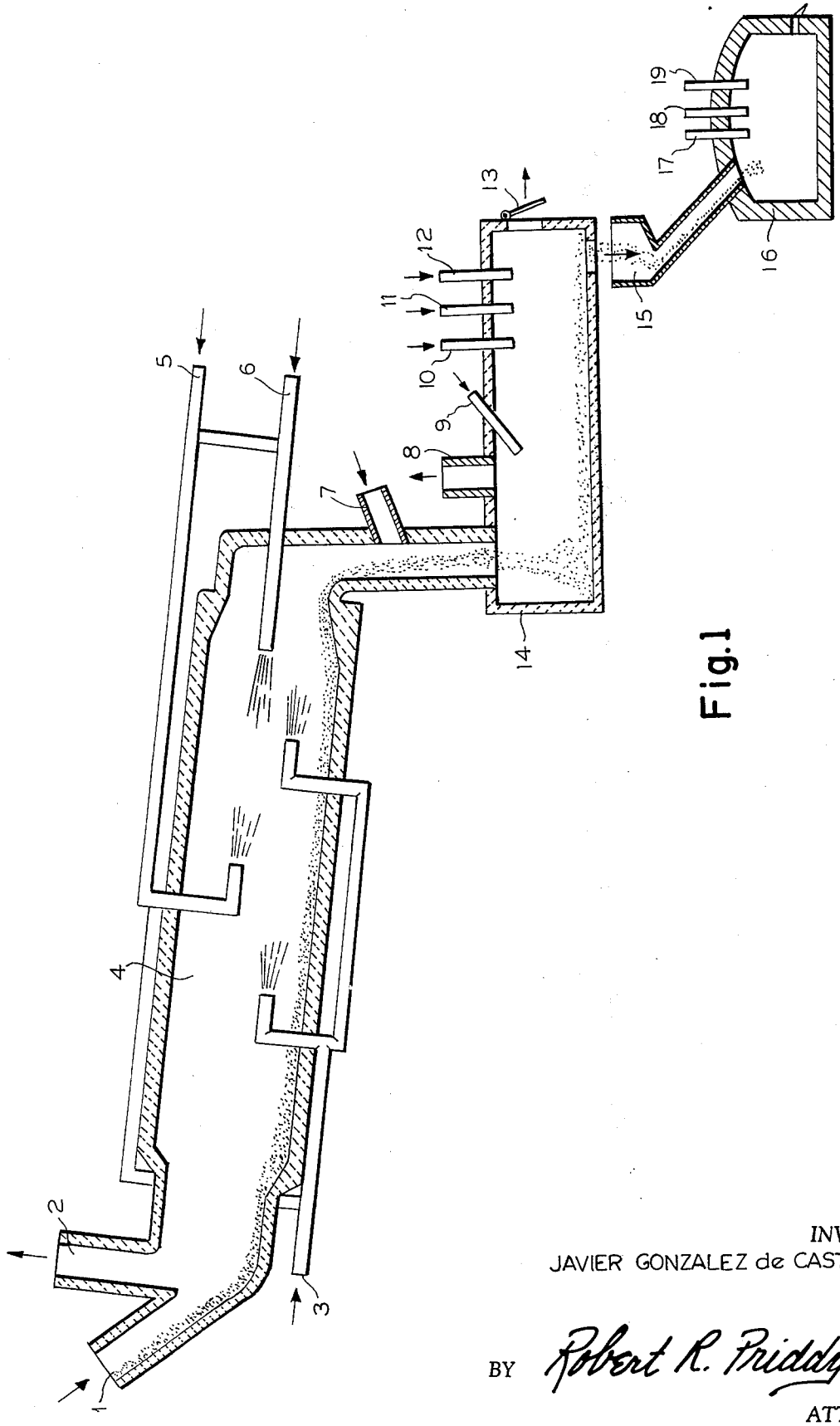


Fig.1

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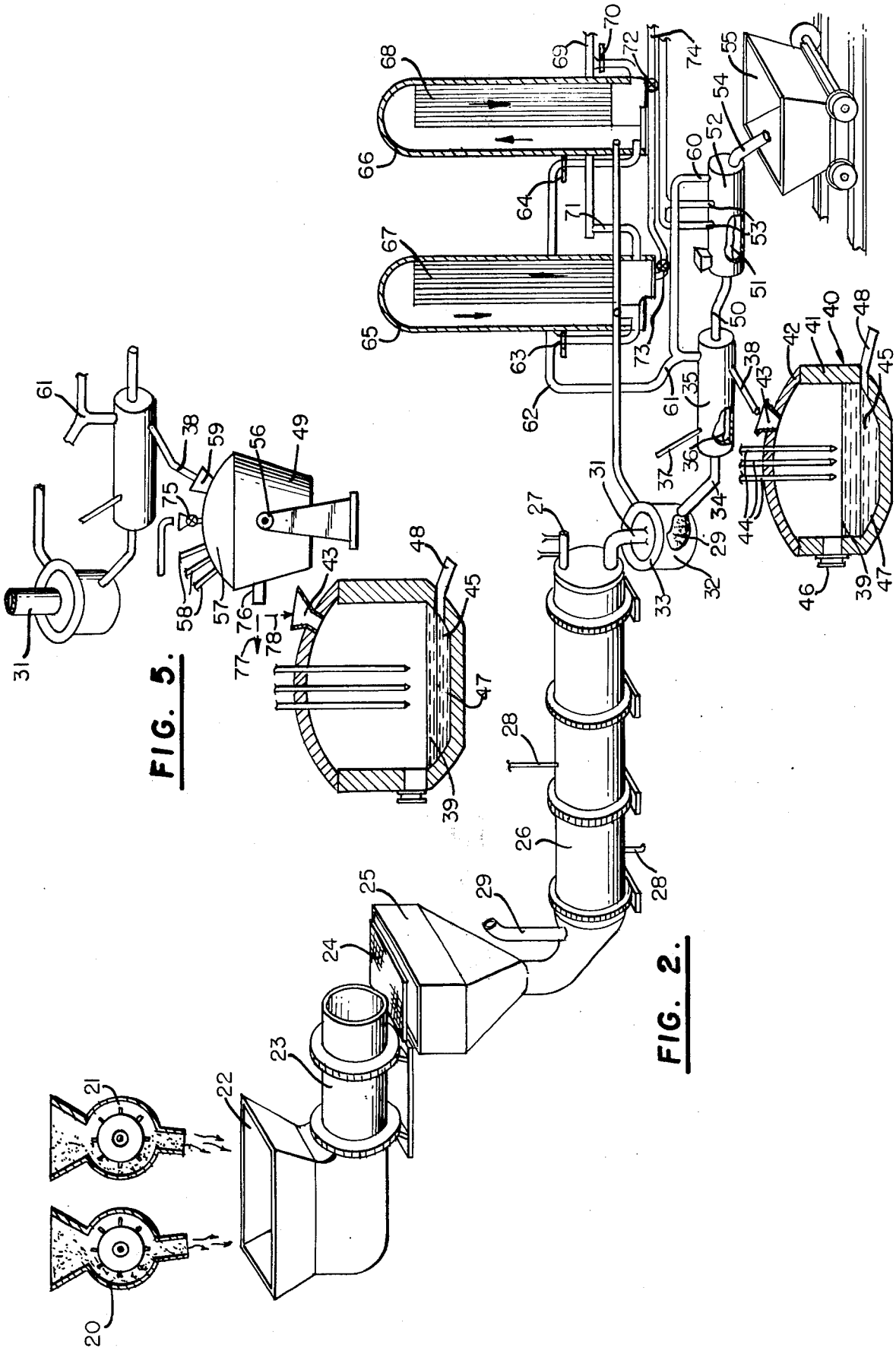

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FIG. 2.

FIG. 5.

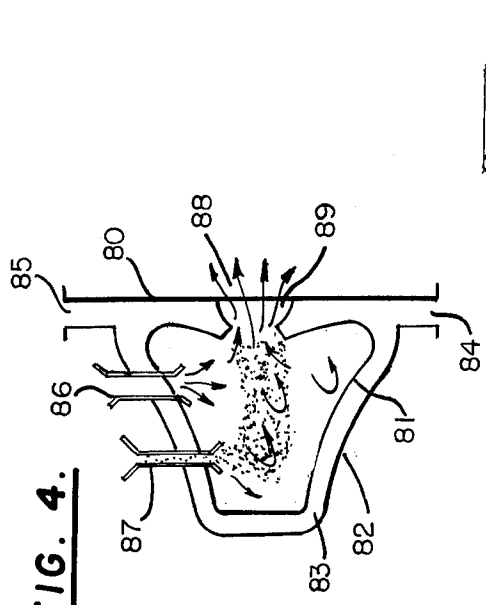

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FIG. 4.

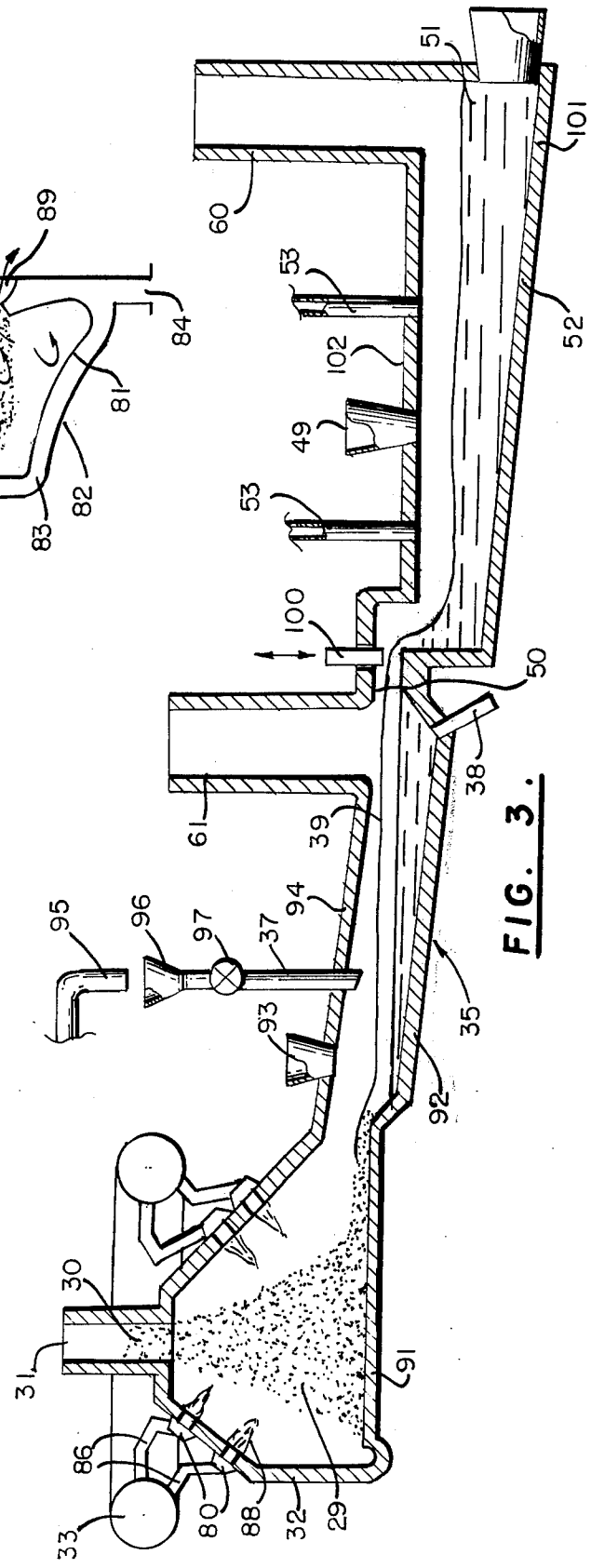

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FIG. 3.

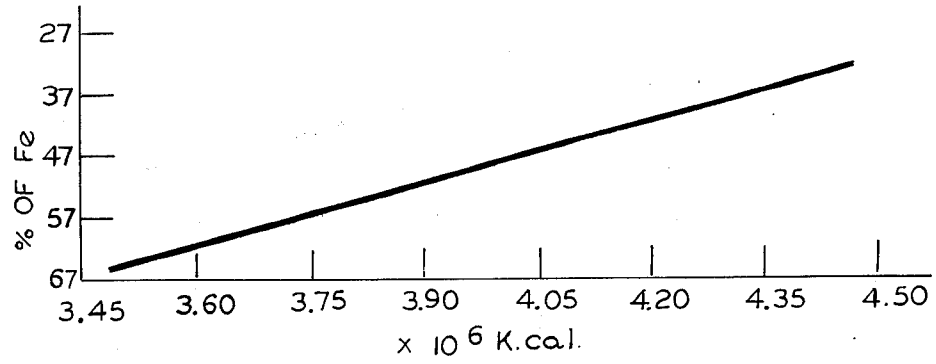


FIG. 6

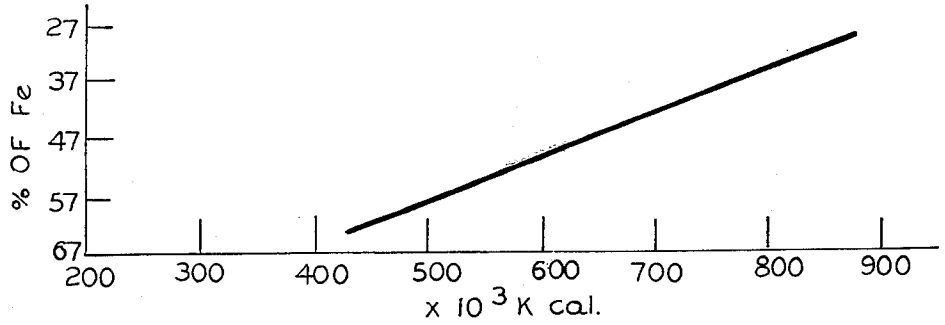


FIG. 7

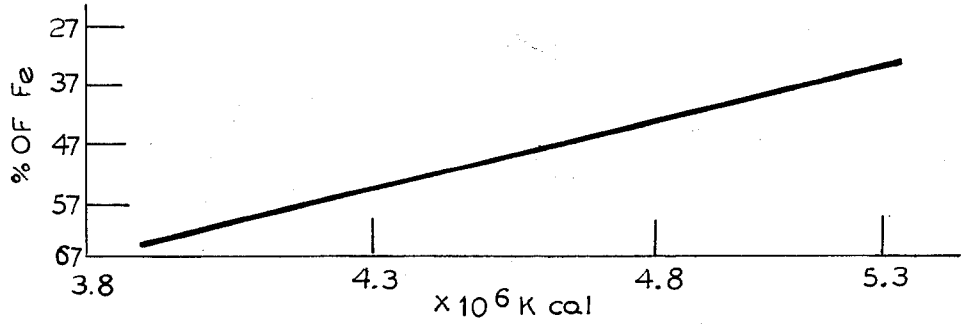


FIG. 8

METHOD FOR THE PRODUCTION OF IRON AND STEEL

This application is a continuation-in-part of prior copending applications Ser. No. 142,255, filed May 11, 1971, Ser. No. 146,981, filed May 26, 1971 and Ser. No. 148,856, filed June 1, 1971, which prior applications are now abandoned.

BACKGROUND

More than 50 years have been consumed in the search for a direct reduction process which would have the versatility, ease of operation, economy and quality of product necessary to justify its adoption on a widespread commercial basis. A complete review of the thousands of past proposals would fill many volumes.

Considering the facts that many of the past proposals differ from one another only slightly, and that the many details of such processes have been subjected to close scrutiny over the past 50 years, one might expect that the problems are now thoroughly understood and the solution obvious. Nevertheless, a number of the commercial direct reduction plants built in the recent past have either failed to operate properly from the beginning or had to be shut down after a time due to their inability to cope with various problems, including the elimination of impurities in the ore and reductant, notably phosphorous and sulfur. Thus, the percentage of overall iron and steel production by direct processes is still quite small.

A vast and bewildering array of different factors is emphasized in varying degree in prior direct production proposals. For instance, some processes emphasize the use of gaseous reducing agents in preference to solid reducing agents. Some processes beneficiate ore prior to reduction; others reduce the whole ore and seek to purify the reduced product before melting same. In some processes, it is necessary to form fine ore into pellets or grind lump ore to pieces of relatively large size compared to carbonaceous reductant and desulfurizer used in the process. The reduction portion of some processes is conducted in fluidized beds or fixed beds or steeply inclined hearths, while others employ rotary kilns. In some processes, for quite specific reasons, reduction and melting take place in the same chamber or in different zones of the same apparatus. In other processes, these functions are performed in separate apparatus. In some processes, the order of operation is substantially reversed, in that the ore is melted and the bulk of the reduction is conducted with the ore in a molten state.

The lessons which are learned from a comparison of these many processes are these: each method of reduction has its particular limitations in respect to its versatility, economy, ease of operation, and control over the ferrous and nonferrous contents of its reduced product; the available steel making processes have their limitations in respect to the composition of the reduced product which they can successfully and economically convert to steel of acceptable quality; and it is abundantly clear therefore that one may not indiscriminately combine individual factors from existing technology to develop a worthwhile process. Rather, the process must be considered as an organic whole. Let us consider some of the past proposals.

More than 50 years ago, Allingham suggested (see U.S. Pat. No. 1,512,262) preheating of ore in a rotary furnace with some reduction of the ore to lower oxides,

followed by melting the ore and carrying out the major portion of its reduction while it was flowing downwardly in a thin layer in a steeply inclined shaft furnace.

One of Greene's U.S. Pats. No. (1,920,379) indicates that sponge iron was produced prior to 1925 by heating a mixture of crushed iron ore and solid carbonaceous material to a reducing temperature below the sintering temperature of the ore. This was done in a rotary or other form of furnace, using an excess of the carbonaceous material. The hot product was then cooled and the remaining carbonaceous material subsequently separated.

Greene stated that the foregoing process had not been a commercial success. By way of explanation he cited the problem of reoxidation of the fine iron produced thereby, the difficulty of separating the excess carbon, and the inability to liquefy the sponge iron thus produced. As a solution to these problems, the patentee proposed passing the hot partially reduced effluent from the rotary kiln into a hearth furnace where it was floated on a bath of molten slag and melted.

An essential feature of the last-mentioned process was the maintenance of iron oxide in the slag in the hearth furnace in order to counteract or remove any carbon taken up by the reduced iron in the previous treatment. The iron oxide served to decarbonize the reduced iron and form a low carbon iron beneath the slag. Re-carburization of the iron was prevented by keeping the excess carbon floating on the top of the slag layer where it could react either with the iron oxide of the slag or with air which was admitted through an opening in the top of the melting chamber.

In the apparent belief that direct impingement of turbulent gases on the slag bath would interfere with proper flotation of the carbon and foster re-carburization of the iron, the admission of air to the hearth was "controlled" and no facilities were provided for injecting the air. Moreover, the burner and electrodes for heating the bath were located in a separate chamber where they were shielded from the pile of kiln effluent at the hearth inlet by an intervening refractory wall.

Another Greene Patent (U.S. Pat. No. 1,920,377) was concerned with the knotty problem of treatment of raw materials containing both phosphorous and sulfur. In the attempt to adapt the rotary-kiln/hearth furnace combination to this problem, it was recommended that reduction in the rotary kiln should be limited, that the hearth furnace should be elongated, that the temperature at the upstream end of the hearth furnace should be low enough so that it would not melt the reduced iron, and that the subsequent melting of the charge should be completed under the influence of electric arcs at the tapping end of the hearth. At least 10% and preferably from 15 to 25% of unreduced iron oxide was maintained at all times in the slag.

The Krupp-Renn process went into commercial use in Germany and elsewhere in 1939. This was a pyrometallurgical beneficiation process for low-grade siliceous ores not economically beneficiated by other methods. Reduction was performed in a rotary kiln with inferior coal as the reducing agent and fuel, and with the burning of coal as fuel for heating the kiln. Temperature was gradually increased along the length of the kiln. Reduction began at about 600°C. When 1100°C. was reached, additional heat was introduced to give a final reduction temperature of 1250°C. which was maintained long enough to produce solid iron nod-

ules dispersed in viscous slag. The kiln effluent was cooled with a water spray, crushed and magnetically separated.

According to Dennis, in "Metallurgy of the Ferrous Metals", Pitman and Sons, London, 1963, most of the sulfur and phosphorous in the Krupp-Renn kiln charge went into the reduced iron, and the iron could not therefore be used directly to make steel; the concentrate was usually charged to blast furnaces. In Hamilton and Holland's review, "Direct Reduction of Iron Ores — 1962–1967", Broken Hill Propietary Company, Australia, 1967, the Krupp-Renn process was described as one in which the nodulized product of the kiln must be crushed and separated magnetically. In the "Proceedings, Symposium on Iron and Steel Making", Base Metals Division, South African Institute of Mining and Metallurgy, Johannesburg, September 1965, in a discussion of a more modern version of the Krupp-Renn process (Krupp-Sponge Iron Process), it was pointed out that in order to get a sponge suitable for direct use for steel production in preference to scrap, it was necessary to use ore with more than 62% iron. Moreover, the product was of interest for steel melting only if the sulfur content did not exceed 0.1%, or better 0.05%.

At approximately the time that the Krupp-Renn process was being used in Germany, Harman and Loftus invented a process somewhat similar to that in the first-mentioned Greene Patent (see U.S. Pat. No. 2,526,658). However, where Greene employed excess carbon and regulated the temperature in his kiln so as not to form clinker or fuse the charge material, Harman and Loftus did exactly the opposite. They operated their kiln in the absence of excess carbon and recommended strong heating in the downstream end of the kiln in order to sinter or nodulize the charge as was done in the Krupp-Renn process. These procedures were deliberate and not mere matters of choice. The carbon limitation was to control reduction of silica. Sintering or nodulizing the rotary kiln burden before its discharge required modification of the kiln to provide a boring bar mounted on a carriage to dislodge the nodulized and pasty iron which tended to accumulate and stick upon the kiln wall in the sintering zone. The sintered or nodulized product was dropped into a combustion zone containing a bath of slag floating on molten iron. The nodulized kiln effluent formed a pile on the hearth of this furnace, the pile being maintained at such height as to prevent flotation. Thus, the pile of kiln effluent, while in the process of melting, was immersed in the slag formed by previously charged kiln effluent.

While Harman and Loftus played the flames of their melting burners directly upon the exposed upper portion of the aforementioned pile, they were careful to screen the bath from the effect of the oxidizing flames from the burners. This was accomplished by forming a layer of carbon monoxide flames above the bath and between that the bath and the flames which melted the pile.

This patent claimed to produce a superior charging stock for open hearth and electric furnaces for direct refinement into high grades of both alloy and carbon steel. However, this was accomplished only by charging a relatively high quantity of limestone and employing either pulverized coke that was lower in sulfur content and less in amount than that charged in the normal blast furnace, or by employing coal that was exceptionally low in sulfur content.

In his U.S. Pat. No. 2,750,277, Marshall prophetically warned of the increasing difficulty of obtaining coal which would make a strong enough coke for the blast furnace, and the necessity of maintaining high coke quality as the availability of high quality lump ores diminished. Perhaps in recognition of the economic penalty of conducting rotary kiln processes with the high quality coke and unusually low sulfur coals required by Harman and Loftus, it was Marshall's recommendation to reduce iron in the rotary kiln with a gaseous reducing agent.

However, in U.S. Pat. No. 2,919,983, Halley criticized the processes of Harman et al. and Marshall on the grounds that they must inevitably suffer either from a deficiency of heat and reducing capacity or from an excessive fuel consumption and an excessive gas production. His solution was to reduce ore with gases in a fluidized bed or vertical column, melt the effluent in a hearth zone with combustion gases carefully controlled both as to composition and balance with reducing gas requirements, reform those gases and employ them as the reducing agent in the reducing chamber. The process was demonstrated by an example wherein the iron content of the ore was 63%, and the sulfur and phosphorous contents of the ore and iron product were not stated.

In U.S. Pat. No. 2,805,930, the Udy process is disclosed. That process includes the steps of mixing and preheating a charge of ore, limestone, and coal in the rotary kiln, followed by partial reduction in a stationary vertical furnace. Melting and completion of the reduction took place in an electric furnace, the carbon monoxide off-gas from which was used to preheat the burden in the rotary kiln. In a subsequent modification of this process, the partial reduction was conducted in the kiln and the partially reduced kiln effluent was then passed direct to the electric furnace. It was suggested that phosphorous-bearing ores could be handled in this process, but only in conjunction with less than 80% reduction of the rotary kiln charge and a special two-stage melting procedure in the electric furnace. A commercial plant based on the Udy process was constructed in Venezuela, but was unsuccessful and had to be closed down.

U.S. Pat. No. 3,206,299, assigned to the Independence Foundation, discloses suggested improvements in the Udy process. As in the Udy process, there is a partial reduction of ore in a kiln, followed by melting and completion of the reduction in an electric steelmaking furnace. However, in recognition of the problems engendered by the presence of substantial amounts of sulfur in certain ores and coals, this patent proposes oxidizing conditions be maintained in the first half of the rotary kiln in order to burn out sulfur in the kiln charge. However, as more air was injected into the first half of the kiln to remove more sulfur, the reduction effiency of the kiln was reduced considerably. In another Independence Foundation patent (U.S. Pat. No. 3,171,878) it was suggested that the rotary kiln and electric furnace be combined.

In accordance with a procedure suggested in Rüter's and Bayer's U.S. Pat. No. 3,295,958, one might sinter and prereduce ore in a rotary kiln charged with carefully sized particles of ore, flux, coal and crushed coke. The ore particle size must be small enough to permit

fluidization, while the coke size must be such as to resist fluidization. The kiln effluent is dropped into a vertical shaft over a red-hot bed of coke supplied with air under pressure. The reduced ore particles are fluidized while being melted and the coke particles replenish the burning bed of coke. Success of the process is said to hinge upon proper coordination of a number of interrelated factors within narrow limits. These are air velocity, air temperature, coke layer height, the particle sizes of the particles charged to the kiln and the extent of sintering therein.

In Sherwood's U.S. Pat. No. 3,503,736, a process is described in which prepelletized ore or concentrated ore of at least 65% iron content, finely divided coal and desulferizing agent are charged to a rotating kiln having reducing, screening and melting sections. Key steps in the process are the feeding of ore to the kiln in the form of relatively large pellets and the screening of the excess coal and desulfurizer from the reduced pellets prior to the melting of same. Alloying and fluxing materials may be introduced into the molten metal in the melting zone. Although Sherwood recognizes the possibility of conducting the process in an apparatus having separate reduction, screening and melting zones, he indicates that such action is probably not justified; and in any event, the melting zone would be a rotary drum.

Notwithstanding the efforts of Udy, Marshall, Harman, Loftus and the others, and the early criticism which Greene directed against the process of rotary kiln reduction followed by cooling of the kiln effluent and cold separation of carbon and iron before charging the iron to a steel-making furnace, the latter type of process presently is the "high water mark" of the prior art. Today, those processes are the ones most thoroughly developed, and most widely recognized as workable and economic. Ways have been found to eliminate or sufficiently avoid the problems which Greene noted, and there are at this time three commercial plants in operation. See Janke and Garbe, "Sponge Iron-Production by the SL/RN Process and Further Treatment to Obtain Steel", Metallgesellschaft AG, Review of the Activities, No. 12/**1969.**

In the SL/RN process, ore, fresh reduction coal, return char and limestone or dolomite are charged into a rotary kiln and preheated to the reduction temperature of approximately 1100° C. The kiln product passes through a gas-tight chamber into a water-cooled drum which cools it down to a temperature below 100° C. to avoid reoxidation of the sponge iron. The cooler discharge consisting of coarse grains and fine grained sponge iron, return char, coal ash and desulfurizing agent is separated by screening and magnetic separation into the various constituents. The grain size of the kiln feed is so adjusted that the major part of the sponge iron is removed by low intensity magnetic separators. The return char can also be separated from the coal ash and the desulfurizing agent by screening to remove the −1 millimeter fraction. This fraction comprises the desulfurizing agent which is fed in a grain size of −1 millimeter and the major part of the coal ash. The iron recovered from the screening is then converted to steel in a separate steel-making furnace.

Experts on the SL/RN process have accepted the proposition that selection of ore type is limited by the method chosen to convert the screened iron product to steel. If the sponge iron is to be used in an electric arc furnace, the gangue content of the ore should not exceed 5%. Moreover, the substitution of this process for the blast furnance — open hearth method has been justified on the basis of an ore having an iron content of 65% or higher. Also, unless one is willing to accept the special charging procedures, processing techniques and extra cost which they entail, one would not normally choose a rotary kiln charge containing high levels of phosphorous or of both phosphorous and sulfur if the rotary kiln effluent were intended for conversion to steel in an electric arc furnace. Limitations on the rotary kiln charge would also be dictated by the phosphorous- and sulfur-handling capabilities of other steel-making processes in which these impurities played a role in process versatility, economy, ease of operation and product quality.

At the present time, the production of steel in electric arc furnaces is on the increase. In practice, the principal and often the only ferrous raw material used in such furnaces is steel scrap. As the annual tonnage of production in electric arc furnaces has grown, a shortening of the supply of scrap has been noted, especially in localities or countries which do not produce large quantities of scrap. In such places, a rapid increase in the price of scrap is anticipated or has already taken place, it being estimated that the cost of scrap could in certain localities approach or equal that of pig iron.

In view of the foregoing, it is apparent that a need remains for improvements in iron and steel-making processes which would permit economic and technically feasible use from both low and high grade ores and non-coking coals (or those which give low quality coke) without undue problems in the elimination of sulfur and phosphorous contamination. Also, a need remains for improvements which lessen the dependency of the electric arc furnace on scrap as a necessary charge material. It is the object of the present invention to fulfill these and other needs.

PROCESS SUMMARY

Iron oxide and solid carbonaceous reductant are charged into a rotary kiln. The iron oxide may be, for instance, virtually any kind of iron ore. Included are the highest grade natural ores and concentrates and ores which are extremely poor in iron and contain large quantities of gangue. And this will be true even where the iron is intended for conversion to steel in an electric arc furnace. Very substantial quantities of sulfur and/or phosphorous may also be present. Similarly, the reductant may be virtually any solid carbonaceous material whose use is economically feasible. This includes for instance noncoking coals and even those with very substantial sulfur and ash contents.

The oxide or ore and reductant may be charged in the particle sizes which are normal for rotary kiln reduction. There is no particular requirement for pelletizing the ore and/or reductant. Conventional charging ratios of oxide and reductant may be employed. However, the amount of reductant can be and preferably is such as to provide in the kiln an excess of carbon relative to the oxygen to be removed from the oxide by reduction.

If desired, one may add with the ore and reductant conventional fluxes such as limestone in slag-forming proportions. However, such fluxes are generally basic materials. Moreover, they are not always essential to proper operation of the kiln, and this is particularly

true in the context of the present invention. When the kiln is operated in the substantial absence of such added slag-forming materials as limestone and/or dolomite, or with a limited amount of such materials, thereby maintaining an acid condition in the charge as it reduces, varying amounts up to 50% of the sulfur can be removed from the charge as SO_2 in the rotary kiln off gases.

In common with some other rotary kiln processes, the ore is reduced with the solid carbonaceous material as the primary reductant at a temperature in the range of about 600° to about 1300°C. The time, temperature and atmosphere in the kiln are controlled to produce an effluent which is a mixture of particles of iron, gangue, desulfurizing materials (if such have been added), ashes and possibly unburned reductant which are of solid to pasty consistency and in which the reduction of the available iron has been completed to the extent of at least about 80% of theoretical completion.

In a particularly preferred form of the invention, the kiln conditions are controlled to produce an effluent in which the particles are in a solid condition, that is, the charge is held below the fusion temperature of its ingredients throughout its passage through the kiln. Conditions in the kiln and/or the charge to the kiln are selected and controlled for inhibiting the formation of sinter or nodules in the kiln effluent. Thus, at least about 75%, more preferably about 90% and most preferably at least about 95% by weight of the rotary kiln effluent is unsintered and unnodulized.

Contrary to prior practice, the rotary kiln effluent is not charged directly to a blast furnace, electric melting or smelting furnace, fluidized bed or rotary melting unit; nor is it quenched prior to separation of the slag as has been done in the predominant rotary kiln processes. Upon removal from the rotating chamber of the kiln at a temperature in the above range, and while being maintained in said temperature range, the rotary kiln effluent is introduced into a melting zone which is separate from the kiln and has normally stationary walls. In this melting zone, the iron in the rotary kiln effluent is melted for the first time.

The major portion, and preferably substantially all of the heat applied to the kiln effluent to raise its temperature in the melting zone from kiln outlet temperature to the range of about 1500°–1700°C. is supplied by burning solid carbonaceous fuel and/or fluid fuel in burner means disposed in position to direct the resultant flames and combustion gases directly onto the unmelted and melting effluent, or by directing a blast of oxygen, air or other oxygen-bearing gas at solid carbonaceous fuel (including unburned reductant) floating on the slag which forms in the melting zone, or by a combination of the foregoing. During the melting, the unmelted and melting effluent is supported on a solid support, such as the floor of the melting zone, and/or on a bath of molten effluent, rather than being suspended in, for instance, a fluidized bed. Melting of the effluent provides molten iron, on which will float the slag containing the gangue, barren material, ashes and so forth. When the rotary kiln charge has a ratio of (CaO/SiO_2) which is about 0.5 and preferably less, the reactions $SFe + SiO_2 + 2C \rightarrow SSi + 2CO + Fe$ (1150°C+) and $S_2Fe + SiO_2 + 2C \rightarrow SiS_2 + 2CO + Fe$ (1650°C+) will be encouraged in the effluent as its temperature progresses upwardly in the melting zone. Thus, when working with ores containing appreciable amounts of both combined sulfur and SiO_2, varying amounts ranging up to as much as 30 or 40% of the sulfur originally present in the rotary kiln charge can be removed as SSi and SiS_2.

Advantageously, the molten iron can be further refined in the melting zone by injecting into the iron suitable powdery solids and/or gases for removing sulfur, phosphorous or other impurities. The powdery solids may be injected with inert carrier gases, such as nitrogen, or may be entrained in active gases, such as oxygen or air. The resultant slag and the impurities entrapped therein are separated from the iron by density; that is, the iron and slag are caused to stratify and are then separated, e.g., by decantation or by separately tapping the slag and molten iron layers.

In one particularly advantageous form of the invention, the melting zone is provided with two hearths, the first being situated at a higher level than the second and communicating with the latter through suitable troughs or conduits. As the rotary kiln effluent is delivered to the first hearth, it forms a pile which is attacked by flames and/or air or oxygen blasts as described above, and the iron and slag drain, as they melt, through the troughs or conduits to the second hearth. There the injection of the above-mentioned refining agents and necessary adjustments in slag composition may be effected. By operating in this manner, the effluent need not be deeply immersed in molten slag while the effluent is melting, thus reducing the tendency for the iron to absorb impurities from the slag.

When the quantity of phosphorous in the rotary kiln charge is not too high (e.g., less than about 1%), it is convenient to carry out dephosphorizing in the melting zone by injection into the molten iron of for instance lime and oxygen (e.g. oxygen gas or iron oxides) through injection means extending through the walls of the melting zone. This same treatment can eliminate varying amounts ranging up to approximately 60% of the remaining sulfur contained in the molten iron in the melting zone.

When the rotary kiln charge contains a relatively high level of phosphorous, e.g., about 1% or more, a somewhat different technique is often convenient. In this case, desulfurization is carried out in the melting zone by injecting basic flux into the molten iron while limiting the oxygen potential of the slag. In this way, the sulfur content of the molten iron can be reduced to varying levels as low as 0.06%. When desulfurization is thus achieved, the molten iron may be passed from the hearth of the melting zone to a tilting holding furnace in which by basic and oxidizing fluxes a complete dephosphorization may be achieved and a commercial Thomas slag produced.

The molten iron may be passed from the melting zone or holding furnace to any steel producing furnace without further treatment and be converted expeditiously to steel. The iron is preferably charged into the steel producing furnace in a molten state. Even when the iron is produced from such low-grade ores in the foregoing manner, it may represent about 60–100% by weight of the total iron and steel charged to the electric furnace.

APPARATUS SUMMARY

The apparatus of the invention is useful in the above-described processes and in other melting processes. The apparatus takes the form of at least two chambers

which are in open communication with one another, either directly through openings in their respective chamber walls if they are immediately adjacent, or through suitable conduit means or trough means or the like. The first chamber, referred to as the initial heater chamber, has a floor which is elevated relative to the floor of the second or hearth chamber, the initial heater chamber floor being disposed for delivery of material by gravity flow to the floor of the hearth chamber.

Through the walls of the initial heater chamber, at an elevated position relative to the chamber floor, is an inlet of reduced cross-section relative to the area of said floor. This inlet is adapted to deliver solid material to said floor and to replenish the supply thereof during operation of the apparatus. Also extending through the walls of the initial heater chamber, at positions elevated in respect to the floor, are a plurality of flame ports in position for directing the flames of suitable burner means directly upon the material supported on the floor. The necessary burner means are preferably mounted directly upon the exterior surfaces of the chamber with their outlets opening directly into said flame ports, but may be spaced outwardly from the chamber walls and connected with the chamber by lengths of conduit extending from the burners' outlets through the burner ports of the chamber.

In the preferred form of the apparatus, the burner means and flame ports are so arranged as to direct the flames from the burners against the pile of material with an angle of incidence, measured between the flame port axes and a line perpendicular to the adjoining surface of the pile, in the range of 0° to about 45° and preferably 0° to about 30°. The preferred form of burner is a cyclonic burner having suitable inlets for solid carbonaceous fuel and combustion supporting gas for generating a whirling vortex of burning solid fuel. However, it will be appreciated that other types of burners may be employed.

The hearth chamber is provided with a roof which is at a lower elevation than the roof of the initial heater chamber. Also, the average height of the hearth chamber roof, relative to its floor, is smaller than the average height of the initial heater chamber, relative to its floor. Thereby, combustion gases flowing from the initial heater chamber are held in close proximity to the contents of the hearth chamber.

The levels of the contents of the hearth chamber are controlled by a molten metal tap communicating with the floor of the hearth chamber, and by a slag outlet extending through the walls of the chamber at a higher level than the molten metal tap. In the most preferred embodiments of the invention, the slag tap is at least partly below the level of the floor of the initial heater chamber, and the burner means are distributed laterally at a plurality of points completely surrounding the material supported on the initial heating chamber floor.

With the aid of the accompanying drawings, and the text which follows, some examples of how to practice the invention will be given. It should be understood that these examples are illustrative in nature and that many variations, fully within the scope of the present invention, will occur to those skilled in the art.

BRIEF DESCRIPTION OF THE DRAWINGS

FIG. 1 is a schematic diagram of a first embodiment of the invention;

FIG. 2 is a schematic diagram partly in section, of a second embodiment of the present invention;

FIG. 3 is a sectional view of a melting zone apparatus useful in either of the foregoing embodiments;

FIG. 4 is a detailed view, in section, of a portion of FIG. 3;

FIG. 5 is a schematic diagram of a modification of the invention which may be applied to either the first or second or other embodiments of the invention; and

FIGS. 6–8 are graphs of heat balances for the various phases of the process with varying grades of iron ore being charged to the rotary kiln.

DESCRIPTION OF A FIRST EMBODIMENT

In FIG. 1, the numeral 4 represents a rotary reduction furnace of a conventional type equipped with three downstream-directed burners, an upstream-directed blast tube 6 at its outlet end, inlet pipes 3 and 6 for combustion air and blast air respectively, and fuel conduit 5 which supplies both the aforementioned burners and the blast tube 6. At the upstream end of the furnace is an outlet 2 for combustion gases. The solid raw materials are fed to the kiln through a chute 1 located in said outlet.

The iron ore may be of any grade, containing for instance about 20 or 30% to about 69% or higher of iron.

The major reductant, that is, the material employed to reduce the major weight proportion of the total reducible furnace charge, is any suitable carbonaceous solid. It is a common practice in this art to refer to the carbonaceous solid as reacting directly with the iron oxide. However, there is also respectable support for the hypothesis that the carbonaceous solid is first converted to carbon monoxide by combustion with oxygen bearing gases available in the kiln, and that it is this carbon monoxide (and possibly also hydrogen if high volatile coal is used) which reacts with the iron oxide in the ore. Irrespective of which theory is correct, the solid carbonaceous reductant is nevertheless considered to act directly or indirectly as the major reductant. Even if it does first convert to CO, it still furnishes the major portion of the carbon from which the CO is formed. This is not to say, however, that reduction with solid carbonaceous reductant present in the bed of material in the rotary kiln is equivalent to reduction which is carried out with carbon monoxide which is predominantly generated from sources of carbon outside the bed. The presence of carbon affects both physical and chemical changes which occur in the bed during rotary kiln reduction. Although high grade materials could be used, the invention makes it possible to use low grade carbon materials such as non-coking coal, lignite, coke breeze, anthracite fines, bituminous coal, high sulfur coal, coal char and the like, even in the same charge with low grade ores.

The solid raw materials are charged in the usual particle sizes for rotary kiln reduction and at a carbon to iron weight ratio of about 0.4 to about 0.9 and preferably about 0.45 to about 0.8. The materials gradually travel down the incline of the kiln, mixing with intimate contact, and steadily increase in temperature. At about 600°C., reduction of the ore commences. The temperature and residence time of the charge in the kiln are controlled to complete the reduction of the iron ore to the extent of at least about 80% of theoretical completion and to impart a final or outlet temperature to the charge which is about 900°to about 1300°C. and pref-

erably about 1100°to about 1300°C. The resultant pasty particles of iron sponge, gangue and other materials fall down a chute connected to the outlet end of the kiln. Desulfurizing materials such as lime may be added to the effluent through chute **7** when desired.

If desired, the rotary kiln effluent may be stored for a time in suitable storage vessels, (not shown) but preferably it is introduced into the melting zone of melting furnace **14** directly from the kiln, preferably without extraction of heat or unburned reductant (if any). Preferably the charge to the melting zone is the entire rotary kiln effluent containing all the reduced iron and gangue from the kiln.

The iron and gangue melting zone has a substantially imperforate supporting wall or floor but may of course have a few tap holes or other appropriate channels therein for removal of molten material. It is preferred that the side walls of the melting zone be "normally stationary", meaning that they are usually at rest during the melting operation, but such terminology includes the possiblitity that they may move from time to time such as to assist in dumping the contents of the zone, as in a tilting type of furnace.

Melting zone **14** is also provided with various means for heating the rotary kiln effluent to and maintaining it at the desired temperature, which is about 1500° to about 1700°C. and preferably about 1500° to about 1650°C. These means are arranged to direct heat downwardly from a position above the melting iron and gangue. One such means may be an air blast tube **9** for burning any residual carbon floating on the slag. Other such means are the burners **10** and **12,** which may burn coal or a "fluid" fuel, meaning a liquid, vapor or gaseous fuel, such as for example, fuel oil, or steam-atomized residual tars, or methane, or carbon monoxide or any combination of these and other fuels. The air and/or oxygen and fuel flow rates of the burners are controlled to maintain the desired temperature in the melting zone. The major portion of the heat applied to the kiln effluent to raise its temperature from its kiln outlet temperature to the range of 1500° to 1700°C. is supplied by burning of the aforementioned coal and/or fluid fuel (including residual coal in the kiln effluent) as distinguished from the burning of coke in a blast furnace and by chemical combustion as distinguished from electric heating. The balance of the applied heat may be generated in any desired manner, but in the preferred form of the invention, substantially the entire heat requirements for raising the kiln effluent to and maintaining it at 1500°–1700°C. in the melting zone are produced with the aforementioned means by combustion of coal and/or fluid fuel in the substantial absence of added blast furnace coke and without consumption of electrical energy for direct heating of the effluent. Gaseous effluent departs through chimney **8.**

The molten iron may be drained from the melting zone as quickly as it is produced, possibly necessitating some auxiliary separation of slag in a unit separate from the melting zone, downstream thereof. However, in this embodiment of the invention, there is some hold-up of molten iron in the melting zone, so that a pool of molten iron is maintained therein and the auxiliary slag separation unit may then be dispensed with. In such case, the major portion but preferably substantially all of the freshly charged rotary kiln effluent enters the pool of molten iron in the solid to pasty condition in which it leaves the rotary kiln, so that the slag is caused to float upwardly in the pool from the melting solid to pasty effluent as the latter melts, while the freshly melting iron settles downwardly (relatively) from the slag. Operation of the melting zone may be on a batch or continuous basis with the kiln effluent having a preferred average residence time of about 5 minutes to 5 hours in the melting zone. In any case, the hold-up time of the molten iron in the melting zone will normally be at least sufficient to effect stratification of the molten iron and slag and may be extended for a longer period, depending for example on the extent of chemical refining procedures, if any, which are practiced in the melting zone. Refining agents, when used, may be injected in the form of powders and/or gases through inlet **11.**

The melting furnace **14** is also provided wiith a side door **13** for the decanting off of the slag. This door connects to chutes and slag disposal means which are not shown. The molten iron can be withdrawn from the melting zone via an outlet in the bottom of said zone.

If one is willing to sacrifice the economic benefits of using the resultant iron in molten form, it may be solidified in the form of nodules for shipment to an electric furnace or other steel furnaces located at a remote point. However, in accordance with a preferred embodiment of the invention, the rotary kiln, the melting furnace **14** and electric arc furnace **16** are co-located, and iron is fed from the melting furnace to the electric arc furnace in molten form through chute **15.** In the electric arc furnace **16,** electric arcs generated by electrodes **17, 18,** and **19** and conventional steel making additives convert the iron from the melting furnace, or mixtures of such iron with scrap, into steel.

It will of course be appreciated that to the extent necessary all of the foregoing equipment is lined with refractory brick or other high temperature resistant material suitable for the conditions which prevail therein. Moreover, the various burners and gas tubes have been shown in schematic form since the provision of proper burners is well within the skill of those active in this art.

The content of impurities (phosphorous and sulfur) in the liquid iron transferred from the melting furnace **14** to electric arc furnace **16** depends upon the quality of the raw materials charged to the rotary reduction furnace **4.** In this connection, it should be pointed out that the nature of the present process makes it quite simple to economically eliminate substantial proportions of these impurities from the iron prior to its charging to the electric furnace, thus, in turn making it possible to drastically reduce or eliminate the requirement for charging scrap to the electric furnace. To the liquid iron in the electric furnace **14** may be added various refining materials such as lime and the like.

DESCRIPTION OF A SECOND EMBODIMENT

In FIG. **2,** there is a schematic diagram, partly in section, of a second and preferred embodiment of the invention. This embodiment includes an optional raw materials preparation train. This can, of course, be omitted where the raw materials are received in the desired particle sizes.

It is an advantage of the invention that it does not require pelletization of the iron oxide and reductant, either individually or in combination. Thus, it is possible, in accordance with the present invention to charge lump ore to ore grinder **20** and lump coal to coal grinder **21.** The particles of reduced size which are discharged from these mixers enter the hopper **22** of drum

mixer **23**, in which they are agitated in order to bring them into a more homogeneous admixture. Mixing of the oxide and reductant may be accomplished without the addition of binder and without conversion of the whole mixture to pellets. Because the invention does not require the charging of pellets of ore and/or reductant to the kiln, it is possible to subject both the oxide and reductant simultaneously to screening in the same set of screens **24** which receive the free-flowing discharge from drum mixer **23**, separate fine and coarse fractions of predetermined particle sizes and deliver the remaining rotary kiln charge to receiving hopper **25** of rotary kiln **26.**

The solid carbon charged to the rotary kiln is normally at least equal to but is preferably in excess of that required to reduce all the iron in the charge. The excess carbon may be present in the initial charge to the kiln. However, when reducing with highly volatile reductants such as lignite for instance, it is conventional practice in the art to hold a portion of the reductant out of the initial charge and add it to the bed in the kiln at some point or points intermediate the ends thereof. In such case, the excess might not be present until the charge had been partially reduced. Also, volatile reductants are usually employed with the object of burning the volatile matter and possibly part of the solid carbon therein to provide heat for the kiln. When a portion of the carbon in the reductant is burned, the total amount of reductant charged will of course be more than the stoichiometric amount for the reduction reaction. In the case of reductants which are not substantially pure carbon, the amount of carbon or any excess will of course be calculated on the basis of the amount of carbon actually available in the reductant.

Excess carbon in the rotary kiln charge not only assists in inhibiting sintering and/or nodulization of the iron in the rotary kiln effluent, but also is available for burning to generate heat and for reduction of unreduced iron oxide in the melting zone. While extremely large excesses of carbon can be tolerated in the rotary kiln charge, the inclusion of amounts which are substantially in excess of that which is consumed by burning and reduction in the melting zone will normally constitute an unnecessary waste of fuel.

Because it is not necessary to nodulize the iron in the kiln, nor form a filamentary slag such as is used in the Krupp-Renn process, nor cool and magnetically separate the rotary kiln effluent, the composition of the rotary kiln charge may be varied with far less difficulty than is the case in the above-mentioned existing processes. Thus, in the present embodiment, basic fluxes, (such as limestone, dolomite and the like) are either omitted entirely from the rotary kiln charge or are restricted in amount so as to maintain as acid condition in the slag, thereby enhancing the percentage of sulfur in the charge which is converted to SO and SO_2 in the kiln. Up to 50% of the sulfur present in the rotary kiln charge can thus be eliminated during the reduction. Also, the productivity of the kiln is thereby increased.

Rotary kiln **26** should be of such length and be fitted with the necessary heating means to achieve a reduction of the iron oxide in the charge to the extent of 80 to 95% of theoretical. The preferred kilns are those employed in the SL/RN process, and for further information on such kilns, attention is directed to the above-mentioned review by Janke and Garbe in Metallgesellschaft AG. Thus, for instance, such kiln may be provided with a gas or oil fired burner **27** at its downstream end and chimney **29** (for off-gases) at its upstream end. At various locations along the length of the kiln, one may provide shell burners **28** which are of assistance in maintaining the desired temperature profile in the kiln. When the downstream burner **27** is replaced by a powdered coal injector and/or the reductant includes sufficient volatile matter to provide the majority of the heat required in the rotary kiln, the shell burners may be replaced with pipes for injecting oxygen-bearing, combustion supporting gases such as air, oxygen and/or carbon monoxide into the kiln.

In accordance with the preferred mode of operation, the rotary kiln is operated throughout at working temperatures below the melting point of the ingredients of the moving bed therein, and preferably substantially below the melting temperature of all the ingredients of the effluent. Still more preferably, the maximum working temperature and/or ash fusion temperature of any coals which might be used as carbonaceous reductants are selected so that the working temperature is at least about 50°C. less and better about 100°C. less than the ash fusion temperature of the coal. When operating with the foregoing temperature conditions, employing excess carbonaceous reductant and eliminating or restricting the basic fluxes as above described, the 80 to 95% reduced product of the rotary kiln is a free-flowing mass of particles of iron, unreduced iron oxide (FeO), gangue, unburned carbon or coal, ashes and flux or sulfur absorbent (if any). This effluent may contain as much as 80% of the phosphorous present in the materials charges to the rotary kiln.

The effluent, whatever its final form (solid to pasty) drops down a chute **31** into the first portion **32** of the melting zone, where it is attacked by flames from the combustion of fluid and/or solid carbonaceous fuel, as described in further detail below. In most instances, the unburned carbon in the effluent (that C which has not been consumed in the kiln either by combustion or reduction) will range from about 5 to about 25% by weight of the carbon charged to the kiln, with about 5 to 15% or 10% being most typical.

The upper end of the first or initial heater portion of the melting zone is surrounded by air distribution ring **33** which is adapted to receive preheated air through conduit **34.** Air from distribution ring **33** and fuel are supplied to burners (not shown) which are oriented to play flames and combustion gases upon a pile **29** of rotary kiln effluent in initial heater portion **32.** Initial heater portion **32** communicates through a conduit **34** with a hearth **35,** which is at a lower elevation than the floor of the initial heater. The burners in the initial heater portion are operated at a sufficiently high flame temperature, e.g., about 1600°–2000°C., to at least commence the melting of the rotary kiln effluent and/or bring about a sufficient change in viscosity to cause it to flow by gravity through conduit **34** into the hearth **35** while in a semi-molten or molten condition.

As the pile **29** of unmelted and melting effluent in initial heater section **32** diminishes, it is continually replenished with fresh effluent from kiln **26.** Preferably, the kiln and initial heater section are operated in such a manner as to keep the connecting chute **31** at least partially filled with unmelted effluent. In this manner, the passage of gases through the chute **31** is impeded, and the major portion of the combustion gases generated in initial heater portion **32** are discharged into the

hearth 35 for further contact with the semi-molten and molten effluent, thus possibly providing assistance in completing the melting.

If there has been a high removal of sulfur in the rotary kiln, the melting of the kiln effluent in the melting zone with the slag in an acid condition can almost totally eliminate the remaining sulfur. This may occur through reactions described above, and/or by formation of sulfates and/or by gasification, and sulfur may be recovered if desired, by solvent extraction of the off-gases with polyvinyl pyrrolidone or by other methods. In this manner, up to about 30to 40% of the S of the rotary kiln charge can be removed from the iron.

It is an advantage of the present embodiment that the base of the pile 29 of rotary kiln effluent is at or above the normal level of the bath 36 which is present in the hearth. Since the unmelted and melting effluent in initial heater portion 32 is not deeply immersed in the slag layer of the bath, and the slag and iron are permitted to flow to their respective layers in the bath 36 as soon as their viscosities permit, the effluent pile 29 is not subjected to unnecessarily long contact with the slag and therefore is less prone to absorb impurities therefrom. Also, non-immersion of the pile 29 in the bath 36 can improve the transfer of heat, especially to the lower portion of the pile.

To the extent necessary, additional burners (not shown) and/or means for injecting oxygen-bearing gases into the effluent (thereby burning residual coal) may be employed in the hearth 35 to insure that melting of the iron is completed before it leaves the melting zone. Thus, a bath 36 of molten iron with semi-molten or fully molten slag floating on the top is present in the hearth portion 35.

Stratification of the molten iron and slag in the melting zone enables separate recovery of the iron and slag through molten metal tap 38 and conduit 50. Generally, the replenishment of pile 29 with fresh effluent from the kiln is kept substantially in balance with the transfer of molten and semi-molten effluent to hearth 35, and the tapping of iron and slag are kept substantially in balance with such transfer to maintain reasonably stable inventories of materials in the various zones when operating the process on a continuous basis. The molten iron, with or without prior refinement as described below, may then be charged to any steel-making furnace or converter for final adjustment of its composition in respect to carbon and other constituents.

However, a particularly advantageous embodiment of this invention involves the injection of solid and/or gaseous phosphorous and/or sulfur removing agents into the molten iron in the melting zone. This is facilitated by the free space above the bath 36, through which various injection means 37, including chutes, conduits, probes and lances, may communicate with the bath from above.

With or without the injection of refining agents for phosphorous and/or sulfur, the molten iron from molten metal tap 38 of hearth 35 is preferably charged to a direct electric arc steel-making furnace 40 having a basic lining 41, a roof 42, a hearth 47, a receiving port 43 in the roof, and electrodes 44 extending through the roof. These electrodes are connected to conventional power sources (which may be of lower capacity than are normally required when operating with cold charges) and are supported by automatic lowering gear which maintains them either slightly above or in contact with the molten bath of slag 39 and metal 45. A door 46 is provided for introduction of additives, sampling, manipulation of the slag and so forth. A spout 48 is provided for tapping molten metal and slag. This furnace may be operated in a conventional manner, employing the usual fluxes, oxidizers, and alloying agents according to double slag or single slag practice.

In this preferred embodiment of the invention, provision is made for the separate removal of molten slag from hearth 35 through conduit 50 (separate from the molten iron). The molten slag forms a bath 51 in the slag hearth 52. There it is blasted with oxygen-bearing gas to burn residual carbon floating on the slag to recover its potential heat, and may be mixed with any desired additives to render the slag, after solidification, useful as fertilizer, aggregate, abrasives and the like. Burners (not shown) may also be provided in this zone if needed to keep the slag molten or assist in incorporating the additives. The thus treated slag is discharged in a molten condition through spout 54 into a car 55 in which it may be transported to another location for conversion to the desired physical form.

Hot combustion gases are collected from the hearth 35 and slag hearth 52 through chimneys 60 and 61 and conduit 62. These gases are fed alternatively through valves 63 and 64 to recuperative heat exchangers 65 and 66 containing brick checkerworks 67 and 68 respectively. When receiving the hot combustion gases, the checkerworks heat up. Upon reaching a predetermined temperature, the hot combustion gases are switched to the other heat exchanger, and the heat exchanger with the hot checkerwork is then used to heat incoming air. After heating the checkerworks, the gases depart via valves 72 and 73 through conduit 74. The air to be preheated enters the checkerworks through conduit 69 and is supplied to the appropriate heat exchanger through valves 70 and 71. Upon being heated by passage through the heated checkerworks 67 or 68, the air is discharged from the heat exchanger into conduit 34 and air distribution ring 33 as described above.

In a particularly preferred form of the melting zone, which is shown in FIGS. 3 and 4, the initial heater portion of the melting zone has an upper end in the form of a truncated cone, through which opens conduit 30 connected to the rotary kiln. Air distribution ring 33, fed by conduit 34 surrounds the upper end of the initial heater portion, on which are mounted cyclone burners 80. The burners 80 have inner and outer walls 81 and 82, which between them define a space 83 for the circulation of coolant, such as water, which enters through inlet 84 and departs outlet 85. High velocity combustion supporting gas enters burners 80 via distribution ring 33 and tangential inlets 86, forming a vortex in which is burned low ash, high emissivity coal admitted through conduit 87. The resultant intense radiant flame 88 (e.g., 1600°–2000°C.) projects through flame port 89 and away from the chamber wall at an angle which can for instance range from a few degrees and preferably at least 30° (and more preferably at least 45°) to the perpendicular (as shown) thus focussing the hottest part of the flame on the pile rather than the refractory walls. Refractory wear can thus be reduced while using flames which are as hot as or possibly hotter than in open hearth furnaces.

Initial heater section floor **91**, at a higher level than hearth floor **92**, is connected by troughs or other means for gravity flow of material from the upper floor to the lower. Molten metal tap **38** is in floor **92**. Initial heater section **32** is in open communication with the hearth portion **35** of the melting zone to provide for free passage of gases from said section to said hearth portion. A vault or roof **94** is provided above the normal level for the bath of molten metal and slag **39** which forms in this melting zone. The roof **94** is placed in sufficient proximity to the level of the bath in order that it may guide over the surface of said bath the combustion gases discharged from the initial heater section.

Intermediate the ends of hearth **35** are a charging chute **93** (which may be provided with a door to close it when not in use) and at least one lance **37** having a downwardly directed outlet. Lance **37** may be connected to a supply **95** of oxygen-bearing and/or inert gas, and may be provided with means **96** for entraining solid refining agents. A valve **97** controls the flow of gas through lance **37**. It will be understood that the melting zone will be provided with a sufficient number of said lances, suitably spaced, to complete the necessary refining within the desired residence time and with an acceptable level of uniformity.

The level of the slag lever in hearth section **35** may be controlled by a movable gate or wier **100** in the conduit **50**. The gate or wier means may be provided with means (not shown) for raising and lower it, thereby controlling the flow of slag into optional hearth **52**, where the slag forms a bath **51**. This slag hearth has a floor **101** which is at a lower level than the floor of both the initial heater section **312** and hearth section **35**. Like hearth section **35**, the slag hearth **52** has a roof or vault **102** which is in sufficient proximity to the normal level of the slag bath **51**, for maintaining a close contact between the surface of said bath and the gases which pass through the space above the bath. Intermediate the ends of the slag hearth, blast tubes **53** pass through the roof **102** for forcibly injecting oxygen-bearing gas into the slag bath **51**. Also a hatch **49** (with cover, not shown) is provided for introduction of solid materials as required to adjust the slag to the desired composition. Combustion gases depart the hearth section and slag hearth through chimneys **61** and **60** respectively.

The phorphorous content of rotary kiln effluents normally comes mainly from the iron ore and generally constitutes at least the majority and typically 80% by weight of the phosphorous which was present in the charge to the kiln. The sulfur content of such effluents may emanate from the iron ore or the coal (usually for the most part from the latter) or from both. The sulfur content of the effluent may vary widely but can be for instance about 50–80% by weight of that present in the kiln charge.

Phosphorous forms iron phosphide, which dissolves in the iron. The resultant coarsening of grain size and formation of undesirable massive aggregates, embrittles the final product. Most steel products contain less than 0.05% of sulfur. Although up to about 0.5% of sulfur may be included in carbon and alloy steels to improve machinability, the resultant product is prone to corrosion and lacks ductility. Also, excessive sulfur interferes with welding, except with special electrodes which are useful only on certain steels. Thus, the phosphorous and sulfur content of steel products is carefully controlled.

Economical production of a wide range of commercially acceptable steel products has heretofore been a challenge when the iron was produced in a rotary kiln from a charge containing relatively high amounts of phosphorous and/or sulfur. And this challenge has been particularly formidable where it was desired to convert the resultant iron to steel in an electric steel-making furnace. In response to this challenge, various special techniques have been introduced into direct reduction processes aimed at control of these impurities. A few examples include elongation of the rotary kiln and oxidation of the charge in the first portion to control sulfur; treatment of rotary kiln effluent in the solid state to remove sulfur prior to introduction into the steel-making furnace; and special melting techniques carried out in the electric furnace for control of phosphorous and sulfur. The foregoing suggestions do not appear to have met with wide acceptance, possibly due to questions concerning the ease of operation and/or the economic feasibility of direct reduction processes employing these special techniques, as compared to the more conventional blast furnace-open hearth route to the production of steel.

Viewed against the background, it is significant to note that the present invention provides a rotary kiln direct reduction process which, despite the use of ores and coals of widely varying quality, can economically deliver to the ultimate steel-making furnace a charge very easily convertible to steel of the highest quality; and when that steelmaking furnace is an electric furnace, the conversion to steel can be accomplished with very favorable consumptions of electricity, electrodes and furnace operating time.

Refining of the iron in the melting zone can be carried out in various ways. For example, one can dephosphorize in the hearth **35** using basic powders such as lime and calcite, and in some cases oxygen-bearing solids such as iron oxide (e.g., ore). These basic powders and other additives may, for instance, be entrained in a high velocity blast (e.g., subsonic or supersonic) of oxidizing gas and blown down through the slag layer (suitably controlled as to thickness) through probes **37** (see FIG. **3**) and thereby injected into the molten iron as in the LDAC and OLP processes. The blast locally displaces the slab and forms FeO where the blast strikes the molten metal. The FeO is immediately available to flux the basic refining agent, thereby hastening the removal of P.

The removal of phosphorous is favored by such factors as moderate temperatures, a high content of iron oxide in the slag and a high content of oxygen in the molten metal. However, as pointed out in "The Making, Shaping and Treating of Steel", McGannon, U.S. Steel 1964, sulfur removal is favored by exactly the opposite conditions. Thus, removal of sulfur depends upon such factors as high temperatures, low iron oxide in the slag, e.g., less than 10%, and a low oxygen potential in the metal. It is therefore of interest that when most of the phosphorous is removed from the molten iron in accordance with the above-described refining technique, up to 60% of the S present in the molten iron prior to the refining procedure can also be removed Thus, by operating the kiln and melting zone with an acid charge in such a way as to maximize sulfur removal, and by dephosphorizing in the above-described manner in the melting zone, one can often produce an excellent quantity of iron which can be

charged directly to the steelmaking furnace and easily converted therein to high quality steel, especially when the molten iron contains less than about 1% of phosphorous.

On the other hand, when the phosphorous content of the rotary kiln charge is low enough so that dephosphorization in the melting zone is not required, and the sulfur content is relatively high, one may merely desulfurize in the hearth **35**. This may be done for example by injecting a basic refining powder such as lime through probes **37** in the above-described manner, but using inert or reducing injecting gases, limiting the iron oxide content of the slag, and maintaining higher operating temperatures. Operating in this manner, one can reduce the sulfur content of the iron to as low as 0.06%.

Refining treatments of the above type may be facilitated by suitable management of the slag in the melting zone. For instance, when the effluent is such as to provide a slag of essentially acid composition, and it is desired to perform a refining procedure involving the use of basic slag in hearth **35**, means may be provided for withdrawing at least a portion of the acid slag prior to such treatment. This may be done, for instance, by use of a slag draw-off assembly positioned upstream of the place where the basic slag is to be formed. Such arrangements may also be useful when it is desired to perform successive dephosphorization and desulfurization treatments (in either order) in different portions of the same hearth. The performance of such successive treatments can also be facilitated by dividing the melting zone into additional zones or chambers so configured that the molten metal (but none or only limited amounts of slag) from a preceding chamber may gain entry to the next one downstream. These techniques therefore would make possible in the melting zone a procedure wherein the load to the initial heating chamber is essentially acid and heating is limited initially to the amount required for melting, a portion of the acid slag is separated from the molten iron, the remaining slag is rendered basic and phosphorous removed by treatment in the hearth section with injections of oxygen containing entrained lime (with eventual increase in the bath temperature to a level more suitable for sulfur removal), the phosphorous-bearing slag with incidentally removed sulfur is separated and the iron transferred to a separate downstream chamber in the melting zone connected with the hearth and desulfurization is carried out by blasting with oxygen containing gases containing entrained lime. Although lime and oxygen have been repeatedly used as examples of refining agents in the foregoing discussion, those skilled in the art will readily recognize that the invention is not limited to particular refining agents, and any of the known refining agents can be used.

When the phosphorous and sulfur contents of the molten iron are sufficient, one may desulfurize in the hearth **35** in the manner described above and dephosphorize in a separate holding furnace, where commercially valuable slags can be produced. This technique is particularly convenient when the molten iron contains 1% or more of phosphorous and it is desired to produce a steel containing about 0.01% or less of phosphorous. This technique will be explained in greater detail with FIG. **5** which discloses suitable apparatus.

In FIG. **5** are shown a melting zone and a steel production furnace which are the same as and bear the same part numbers as those in FIG. **2**. However, in the present embodiment, the molten metal tap **38** (with optional additional chutes and conduits as required) is disposed to deliver molten metal to the inlet **59** of a tilting holding furnace **49**, journalled in bearings **56** on a suitable support. The interior of furnace **49** has a basic lining and is covered by a domed roof **57**. Through roof **57**, into the interior of the vessel, project electrodes **58** for heating its contents (although any heating means can be used). There is also an injector **75** which extends through roof **57** and is adapted to direct a blast of gas and entrained solids at the surface of the vessel contents. In the upper portion of the vessel side wall is a spout **76** through which slag and molten metal may be tapped. As the furnace is tilted, streams of slag **77** and molten metal **78** respectively may be poured (with such additional chutes and conduits as required) to a slag receiving vessel (not shown) and to the receiving port **43** of electric furnace **40**. If and when the holding furnace is operated as a batch furnace while the melting apparatus operates continuously, two furnaces **49** may if desired be charged alternately by one melting apparatus.

Forming the oxidizing slag for dephosphorization, by injections of lime and/or calcite with oxygen carrier gas produces exothermic reactions. Therefore the power requirements for furnace **49** are exceedingly low.

The reason for having a tilting furnace is to facilitate rapid removal of the dephosphorizing slag. As the slag contains a high concentration of phosphorous, the partition coefficient is very sensitive to small changes in temperature, which makes it adviseable to remove the slag as soon as it has formed and been used. With 1% or more of P in the molten iron charged to furnace **49**, it is possible to produce Thomas type slags of commercial value, thus offsetting much of the cost of dephosphorization.

As already indicated, the foregoing procedure makes it possible to reduce the percentage of phosphorous in the molten iron from a value of 1% or more to less than 0.01%. This is very difficult to achieve when one attempts to carry out in the same furnace all the steps forming and working the oxidizing slag, removing that slag, forming and working the reducing slag, removing that slag, adjusting the carbon content and adding ferroalloys for the production of steel. When attempting to operate in the last-mentioned way, one finds that a ring forms in the place where the dephosphorizing slag was in contact with the refractory lining of the furnace. This can often occur regardless of how fast the slag is removed and how thoroughly the interior of the furnace is rabbled. This ring includes oxidized phosphorous that will pass into the molten metal when contacted by the subsequent reducing slag. When according to this invention, the metal is subjected to oxidizing slag in the holding furnace **49** and not treated with reducing slag until it has been transferred to another vessel, e.g., electric furnace **40**, this problem is overcome.

The dephosphorization step is very fast when the powders are injected in the above-described manner. The slag is formed almost instantaneously and acts in a few minutes. It has been shown that during the dephosphorization, one can simultaneously eliminate, through the formation gases, part of the sulfur in the molten iron. This can be as much as 60% of the sulfur. Thus, when operating in accordance with this embodiment of the invention, dephosphorization does not present technical or economical problems, no matter

how high the phosphorous content of the molten iron may be.

ECONOMIC ADVANTAGES

Those skilled in the art will appreciate that different forms of the invention have varying combinations of technical and economic advantages over the prior art in varying measure. When practising the preferred forms of the process, the economic advantages can be striking. By way of introduction to this topic, let us consider the heat balance and fuel requirements of the melting zone.

The basis of this discussion is a plant of the type disclosed in FIG. **2**, employing the melting zone of FIGS. **3** and **4**. The ore is a siliceous iron ore of 40% Fe and 60% gangue. The gangue composition is SiO_2 = 60/**80%**, Al_2O_3 = 10/**20%**, and CaO = 10/**20%** and has a melting temperature of 1400°–1500°C. The rotary kiln effluent is received in the melting zone at 1100°C and is composed of the following relative quantities of materials per unit of time: reduced iron 800 kilos, barren material 1200 kilos, FeO 257 kilos (which includes 200 kilos Fe) and 125 kilos coal of 80% active C content.

In the melting unit, heat must be provided to reduce the 257 kilos of FeO with the carbon available in the effluent, melt 1000 kilos of iron, and — in respect to the barren material — either melt it and/or at least obtain the proper viscosity in the bath to successfully separate it. Based on the reaction FeO + C = Fe + CO, the reduction would require −64 + 26 = −38 Kcal/mol or 132,000 Kcal, producing 80M³ of CO and consuming 42 kilos of the available C.

Two sets of heat requirements will now be considered: those based on melting by burning the C to neutral combustion gases ($CO/CO_2 = 1$) with a flame temperature of 1700° and heating the melting zone contents to 1500°C; and those based on melting by burning the C to CO with a flame temperature of 1900°C and heating the melting zone contents to 1600°C.

In the first case, the heat requirements for the melting would be (for the iron) 1000 kilos × 0.2 Kcal × 400°C. plus (for the barren material and ashes) 1200 kilos × 0.3 × 400°C = 224,000 Kcal. With heat losses of 40,000 Kcal by radiation to the melting zone walls, the reduction and melting requirements are 132,000 + 224,000 + 40,000 for a total of 400,000 Kcal.

Burning C to neutral combustion gases with air preheated to 650°C. releases 3,300 Kcal per kilo of C., so (400,000/**3,300** or 120 kilos of C are needed. The preheating of the required combustion air, 120 kilos C × 7.5 M^3 = 900 M^3, consumes 900M^3 × 0.3 × 650°C. = 175,000 Kcal which can be provided by burning an additional 22 kilos of C to CO_2. This coal requires an additional 22 kilos × 10 M^3 = 220 M^3 of combustion air which in process of being heated to 650°C. consumes another 5 kilos of C. Thus, the fuel requirements for the melting zone are 120 + 22 + 5 = 147 kilos of C. Based on coal with 80%C, or 6000 Kcal/kilo, it appears necessary to burn 184 kilos of coal.

However, the 1500°C off-gases have a sensible heat of 1,250 M^3 × 0.3 × 1500°C. = 562,000 Kcal and potential heat of 467,000 Kcal for a total of 1,000,000 Kcal. At a recuperation yield of 60%, there is a make-up of 600,000 Kcal, thus reducing the coal requirements by 100 kilos to 84 kilos.

In the second case, the heat requirements for the melting would be (for the iron) 1000 kilos × 0.2 Kcal × 500°C. plus (for the barren material and ashes) 1200 kilos × **0.3** × **500**°C. = 280,000 Kcal. With heat loses of 41,000 Kcal by radiation to the melting zone walls, the reduction and melting requirements are 132,000 + 280,000 + 41,000 for a total of 452,000 Kcal.

Burning C to CO with air preheated to 650°C. releases 1,300 Kcal per kilo of C., so (452,000/**1,300**) or 347 kilos of C are needed. The preheating of the required combustion air, 347 kilos C × 5 M^3 = 1735 M^3, consumes 1735 M^3 × 0.3 × 650°C. = 338,000 Kcal, which can be provided by burning an additional 41 kilos (338,000/**8,000**) of C to CO_2. This coal requires an additional 41 kilos × 10 M^3 = 410 M^3 of combustion air which in process of being heated to 650°C. consumes another 10 kilos of C. Thus, the fuel requirements for the melting zone are 347 + 41 + 10 = 398 kilos of C. Based on coal with 80% C, or 6000 Kcal/kilo, it appears necessary to burn 500 kilos of coal.

However, the 1860 M^3 of 1600°C. off-gases attributable to burning have a sensible heat of 1860 M^3 × 500 plus a potential heat of 1860 M^3 × 0.347 × 3,120 for a total of 2,942,000 Kcal. Sensible + potential heat of reduction byproduct CO = 80M^3× 500+80 M^3 × 0.347 × 3120 for a total of 127,000 Kcal. Total recuperable heat is 2,942,000 + 127,000 for a total of 3,069,000 Kcal. At 70% recuperation yield, there is a make-up of 2,148,000 Kcal, thus reducing the coal requirements by 356 to 144 kilos per 1000 kilos of molten Fe. From this, it may be seen that the heat balance of the melting zone does not constitute an obstacle to the successful use of this process. Also, it should be apparent that, as compared to the blast furnace off-gases of the present invention have a high heating value.

An analysis of the heat requirements of the process produced the graphical data in FIGS. **6–8**. FIG. **6** illustrates the requirements for the rotary reduction furnace with ores of varying Fe content, assuming a charge containing 1000 Kgs Fe in iron oxides and a metallization of 80 to 95%. FIG. **7** shows the requirements of the melting zone, assuming the recovery of 900 to 1000 Kgs of molten iron. FIG. **8** gives the total heat consumption for the rotary furnace and melting zone.

A definite economic advantage over the blast furnace can be obtained. Based on the use of the same rich iron ore in both processes, the projected costs per ton of molten Fe are:

Blast Furnace	
Ore - (with necessary preparation for blast furnace operations)	$20
Coke	15
Operating Cost	7
Amortization	10
Total, molten Fe in the blast furnace	$52
Invention	
Ore - (with the scant preparation required the rotary kiln)	$17
Non-coking coal	10
Operating cost	7
Amortization	4
Total, Molten Fe in the melting zone	$38

A cost saving of about 27% is projected.

The economics of the process are also quite attractive when compared to an electric steelmaking process

wherein the ferrous charge to the furnace is prepared by reduction of high grade ore in a rotary kiln and cold separation of the effluent as is practised in the SL/RN process. Such a process involves the following costs for items within plant battery limits.

First, there is a substantial capital investment and operating cost involved in the equipment downstream of the rotary furnace, including the cooling unit to cool the effluent from the rotary furnace, and the means for separating the unburned fuel, ashes, barren material and desulfurizing lime from the resultant iron sponge. For a plant with an annual production capacity of about **500,000** metric tons of iron sponge, it is estimated that the aforesaid downstream cooling and separating equipment would add approximately four dollars to the cost of each ton of sponge iron product. The use of such sponge iron in replacement of scrap steel in an electric arc furnace makes it necessary to raise the temperature of the sponge in the furnace from ambient temperature to about 1650°C. so that in modern electric furnaces of high capacity, on the order of 450 Kilowatt hours plus 4.5 or 5 kilograms of graphite electrodes are consumed per ton of molten sponge. Also, the refractory lining of the furnace is worn down, resulting in labor and materials costs for the necessary repairs. Thus, the cost of melting the iron sponge in the electric furnace is estimated at about $12 dollars per ton of sponge.

On the other hand, assuming that substantially all of the sensible heat and potential heat (from any unburned coal therein) of the rotary furnace effluent is preserved, the melting and separation operations of the present invention, including amortization, are estimated to have a cost of about $4 per ton of liquid iron produced. Based on the use of the same high quality raw materials in both the processes, the savings are about $12 per ton.

However, even greater savings may be anticipated when employing lower grade ores and coals. Thus, although the method of the invention is readily applicable to the best ores and coals, it also successfully handles rotary kiln charges which comprise less than 55% or less than 50% and down to about 30% or even about 20% of iron, and about 5 to about 65% by weight of gangue, and which include total sulfur and/or phosphorous contamination, based on the weight of iron in the charge, of: (a) more than about 0.1% or more than about 0.3% and up to about 2.7% of sulfur; or (b) more than about 0.1% or more than about 0.2% and up to about 3% of phosphorous; or (c) both phosphorous and sulfur with at least one of them being in the respective range indicated. It should be understood that the upper values in the aforementioned ranges of impurities do not represent absolute technical limits, but rather indicate levels of impurities above which the economic advantages of the invention are diminished by the additional measures required to remove the impurities. The ability to profitably dispose of the slag produced will influence the economics of using the poorer ores.

It should be noted also that the invention has a very definite advantage over ore-melting processes in that it can operate at temperatures similar to or in some cases lower than those conventionally employed in the production of pig iron. Another important advantage of the process is that it can economically produce liquid iron which has less carbon and silicon dioxide than pig iron from the blast furnace. Thus, the yield of iron can be greater and the costs of eliminating these impurities during conversion of the iron to steel reduced accordingly.

The economics of the process must in each case be developed on an individual basis, taking into account the raw materials to be used. Although use of the lower grade ores reduces yield and productivity, thus increasing the production cost per ton of molten iron, this increase can be partly or completely offset by the lower cost of the ore and the greater value of the slag produced. This is the basic explanation for the great flexibility of this process.

EXAMPLE 1

Into a rotary kiln of the kind shown in the drawings are charged on a basis of 2.4 metric tons of silicious iron ore per ton of iron produced having an assay of 42% iron and 58% gangue, 1 ton of non-coking coal with a carbon content of 79.6% and 90 Kg of limestone ($CaO:SiO_2 = 16$). The furnace is operated in such a manner as to reduce the iron mainly at 900°– 1000°C. At an outlet temperature of 1100°C. the resultant effluent is transferred to a fusion furnace as shown in which the entire effluent is heated at 1650°C. for about 120 minutes whereby the iron in the effluent is melted for the first time during the process. The resultant slag separates readily from the iron and contains a portion of the sulfur and/or phosphorous present in the rotary kiln effluent. The iron and slag are separately removed from the melting zone. 1080 Kg of the resultant molten iron is transferred from the melting zone to an electric furnace along with 60Kg of metallurgical lime, 40 Kg granulated lime, 22 Kg mill scale, 5 Kg coke and 4 Kg ferro alloys. With about 150 Kwhr applied through electrodes with a consumption of 1.5 Kg of electrodes, 1.8 m^3 of oxygen and 1 m^3 of nitrogen as a carrier for injection of the refining ingredients, the molten iron is converted to steel of good quality.

EXAMPLE 2

A silicious iron ore of about 40% iron and 60% gangue is used. The gangue itself is made up of 60 to 80% SiO_2, 10 to 20% Al_2O_3 and 10 to 20% CaO, includes 2 % S and 1.6 %P and has a melting point of approximately 1400°C. The ore is ground just to small pieces ranging in size of about 0.3 to 0.7 inches in their maximum dimension. A noncoking coal containing 80% carbon is ground to particles ranging in size from about 3/32nd to 3/64th inches in their maximum dimension. The ore and coal are premixed in a rotary mixer in a ratio of 2,000 kilograms of ore to 625 kilograms of coal. This mixture is reduced in a rotary kiln and is discharged with an outlet temperature of 1100°C. The resultant rotary kiln effluent is composed of 800 kilograms of reduced iron, 200 kilograms of iron in 257 kilograms of FeO, 1200 kilograms of gangue and ash, 125 kilograms of coal. This effluent is continuously charged to a melting furnace as disclosed in FIGS. **3** and **4** in which the temperature of the mass is raised to about 1600°C by the burning of coal to CO. The cyclone burners are fed with preheated air at 650°C and 6,000 Kcal/kilo coal in the amount of 144 kilos (with recuperation yield of 70% from the melting zone). Burning of the coal with the preheated air produces a flame temperature of about 1900°C. The bath is desulfurized in the hearth of the melting zone to a sulfur content of 0.06% in the molten iron by the injection of 25

Kg of lime entrained in 0.5 Kg of N_2 discharged at a velocity of 60–80 Kg/min. against the surface of the slag layer. The desulfurized molten iron is separated from the resultant slat and dephosphorized in a separate holding furnace as shown in FIG. 5, injecting 20 Kg of lime and 10 Kg of Fe_2O_3 entrained in 2.5 Kg of oxygen at a velocity of 70 Kg/min. to obtain a final P content of 0.01%. The resultant Thomas slag is separated while the iron is transferred directly in the molten state to a colocated direct electric arc furnace and converted to steel as in example 1.

The invention is of course not intended to be limited to the foregoing examples, which are given for purposes of illustration only. It will be apparent that many changes can be made without departing from the spirit of the invention. Thus, some of the advantages of the invention can be obtained by using an open hearth furnace or other steel furnaces instead of the electrical furnace. Although the economic advantages of the invention are most evident when working with both low-grade iron ore and coal, it will be appreciated that the invention has utility with any iron ore and with any carbonaceous reductant. Although the rotary kiln treatment, melting and hot separation of the slag from the molten iron removes a portion of sulfur and phosphorous impurities, it will be evident that the construction of the melting furnace and the nature of this process make it feasible to remove phosphorous and/or sulfur during the melting step by injection of refining powders into the molten iron and/or by other means. It should be evident also that the conditions in the rotary kiln and the melting furnace may be manipulated so that there is some further reduction of the rotary kiln effluent in the melting furnace. Such technique, will of course involve the charging of sufficient fuel, including carbonaceous fuel, into the rotary kiln and/or into the melting furnace for this purpose. And it should be understood that it is an advantage of the present invention that when the rotary kiln effluent contains some unburned coal that the heating value in this coal may be employed to advantage for generating heat for the melting by injecting air or oxygen into the effluent for burning the residual coal. The melted iron need not be charged directly to the electric or other steel making furnace, but may be temporarily stored, if needed, in the molten form in holding furnaces at temperatures sufficiently high, e.g., 1600°C., to retain the iron in molten form. Heretofore, when rotary reduction furnaces have been operated to impart a final temperature of about 1300°C. to the furnace effluent, the composition of the charge to the rotary furnace has been regarded as relatively critical in order to obtain a filamentary slag at the outlet of the furnace. The invention provides the advantage of significantly reducing the criticality of the rotary kiln charge. Accordingly, it is apparent that the present invention is a broad one and is not to be limited except as required by the scope of the claims granted herein.

What is claimed is:

1. Method for producing iron comprising forming a rotary kiln charge which includes appreciable amounts of sulfur contamination and contains iron ore and excess solid carbonaceous reductant; reducing the ore in said charge with said carbonaceous reductant in a rotating kiln; operating said kiln in the substantial absence of added slag-forming materials or with a limited amount of such materials for maintaining an acid condition in the charge as the ore is reduced and removing sulfur from the charge in the form of at least one gaseous oxide of sulfur including SO and SO_2 in the off gases discharged from said kiln; selecting the charge to the kiln, and controlling kiln operating conditions, for inhibiting the formation of sinter or nodules in the kiln effluent for producing, at temperatures in the range of about 600° to about 1300° C and less than the fusion temperature of the ingredients of the rotary kiln charge, a kiln effluent wherein the major solid ingredients are unsintered and unnodulized reduced iron and gangue; maintaining the kiln effluent in the aforesaid temperature range until it has been transferred to a separate melting zone which is connected with the rotary kiln; melting said effluent by contact with gaseous products of combustion of solid carbonaceous fuel and/or fluid fuel; causing the resultant slag to flow upwardly in said melting zone from the melting effluent, while causing the resultant molten iron to settle downwardly therefrom; and recovering the iron from said melting zone in molten form.

2. Method in accordance with claim **1** wherein a connecting chute is provided for transferring kiln effluent from said rotary kiln to said melting zone and said chute is maintained at least partially filled with unmelted effluent for impeding the passage of gases through said chute.

3. A method in accordance with claim **1** wherein the hot rotary kiln effluent is transferred to a separate melting zone having two hearths, the first hearth being situated at a higher level than the second and communicating with the latter through conduit means; a pile of said effluent is formed in the first hearth; the flames and gaseous products of combustion of forced draft burners burning solid carbonaceous fuel and/or fluid fuel are impinged directly upon said pile of said effluent in said first hearth; the resultant combustion gases and melting or molten effluent are conveyed through said conduit means to said second hearth; and a bath of molten iron with an overlying layer of slag is formed in said second hearth.

4. Method in accordance with claim **3** wherein the base of the pile of effluent formed in said first hearth is maintained at or above the normal level of the bath of molten iron and slag which is formed in said second hearth.

5. Method in accordance with claim **3** wherein the combustion gases conveyed through said conduit means to said second hearth are guided over the surface of the bath of molten iron and slag formed in said second hearth for heating said bath.

6. A method in accordance with claim **1** wherein said steel production furnace is an electric furnace and the molten iron is heated during its conversion to steel by electric power therein, whereby savings in electric power may be obtained.

7. A method in accordance with claim **6** wherein said rotary kiln, melting zone and electric furnace are colocated.

8. Method for producing iron comprising forming a rotary kiln charge which includes appreciable amounts of sulfur contamination and contains iron ore and excess solid carbonaceous reductant; reducing the ore in said charge with said carbonaceous reductant in a rotating kiln; operating said kiln in the substantial absence of added slag-forming materials or with a limited amount of such materials for maintaining an acid con-

dition in the charge as it reduces and removing sulfur from the charge in the form of at least one gaseous oxide of sulfur including SO and SO_2 in the off-gases discharged from said kiln; selecting the charge to the kiln, and controlling kiln operating conditions, for inhibiting the formation of sinter or nodules in the kiln effluent for producing, at temperatures in the range of about 600° to about 1300° C and less than the fusion temperature of the ingredients of the rotary kiln charge, a kiln effluent wherein the major solid ingredients are unsintered and unnodulized reduced iron and gangue; maintaining the kiln effluent in the aforesaid temperature range until it has been transferred to a separate melting zone which is connected with the rotary kiln; providing a connecting chute for transferring kiln effluent from said rotary kiln to said melting zone and maintaining said chute at least partially filled with unmelted effluent for impeding the passage of gases through said chute; transferring said kiln effluent through said chute to a separate melting zone having two hearths, the first hearth being situated at a higher level than the second and communicating with the latter through conduit means; forming a pile of effluent in the first hearth; impinging directly upon said pile of effluent in said first hearth the flames and gaseous products of combustion of forced draft burners burning solid carbonaceous fuel and/or fluid fuel; conveying the resultant combustion gases and melting or molten effluent through said conduit means through said second hearth; forming a bath of molten iron with an overlying layer of slag in said second hearth; and recovering the iron from said melting zone in molten form.

9. Method for producing iron, comprising forming a rotary kiln charge containing iron ore, solid carbonaceous reductant and appreciable amount of SiO_2, and which may also include CaO, wherein the carbon to iron weight ratio is in the range of about 0.4 to about 0.9, wherein the ratio of CaO to SiO_2 is in the range of about 0.5 or less, and wherein said charge includes sulfur and/or phosphorus contamination, based on the weight of iron in the charge, of: (a) more than about 0.1% to about 2.7% of sulfur; or (b) more than about 0.1% to about 3% of phosphorus; or (c) both phosphorus and sulfur with at least one of them being in the respective range indicated, said charge further comprising less than 55% of iron and about 5 to about 65% by weight of gangue; reducing the ore in said charge with said carbonaceous reductant in a rotating kiln while (A) operating said kiln in the substantial absence of added slag-forming materials or with a limited amount of such materials for maintaining an acid condition in the charge and promoting removal of any sulfur from the charge as sulfur oxide and sulfur dioxide in the off-gases of the kiln, and (B) controlling kiln conditions to inhibit formation of sinter and nodules for producing a solid kiln effluent containing reduced iron, ashes, gangue and unburned carbonaceous reductant, maintaining the kiln effluent in the aforesaid temperature range until it has been transferred to a separate melting zone which is connected with the rotary kiln; melting said effluent by contact with gaseous products of combustion of solid carbonaceous fuel and/or fluid fuel; causing the resultant slag to flow upwardly in said melting zone from the melting effluent, while causing the resultant molten iron to settle downwardly therefrom; and recovering the iron from said melting zone in molten form.

10. A method of producing steel wherein molten iron produced in accordance with claim **9** is transferred to an arc type electric furnace for final adjustment in composition.

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