Oct. 24, 1950 E. S. HARMAN 2,526,659

CONTINUOUS SMELTING PROCESS

Filed March 9, 1945 5 Sheets-Sheet 1

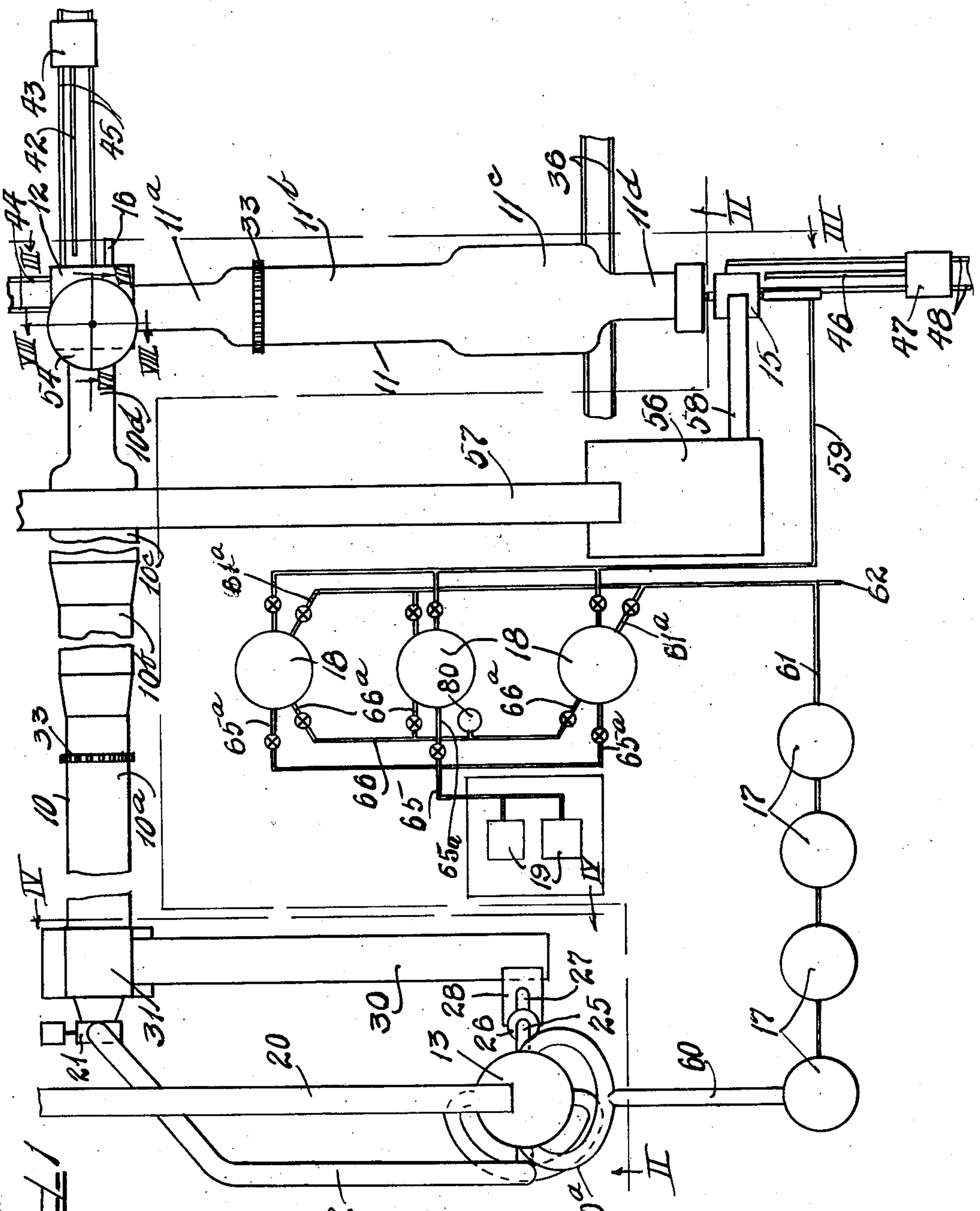

Fig. 1

Inventor
Eugene S. Harman.
by Charles Hills
Attys.

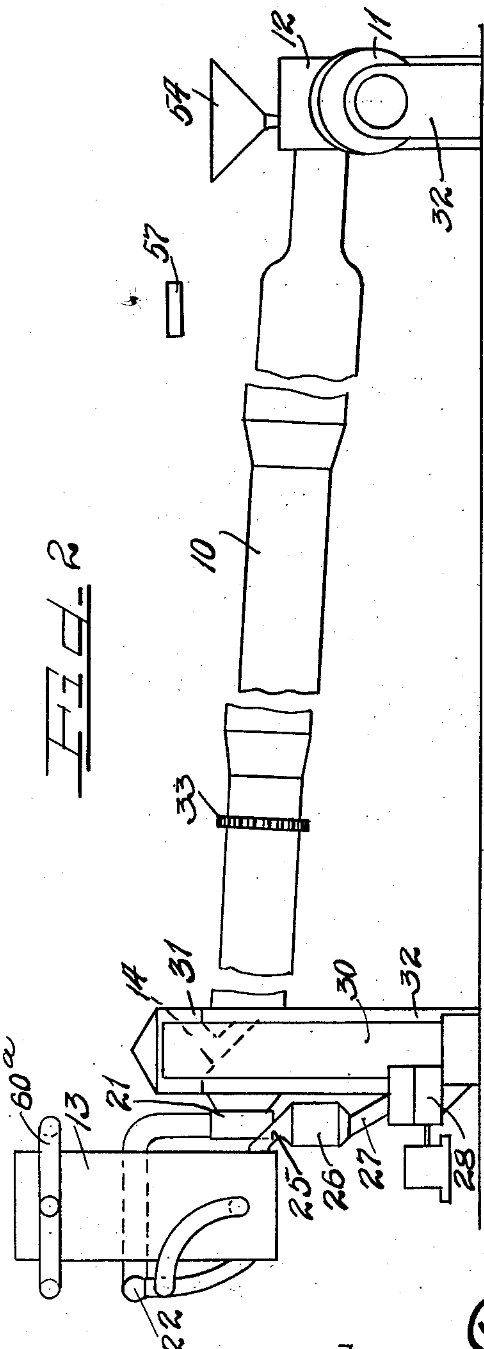

Fig. 2

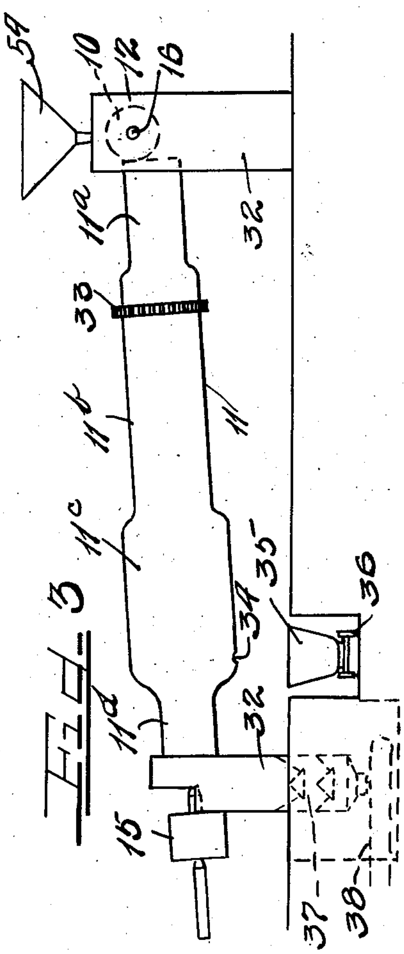

Fig. 3

Inventor
Eugene S. Harman
by Charles Hills
Attys.

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CONTINUOUS SMELTING PROCESS

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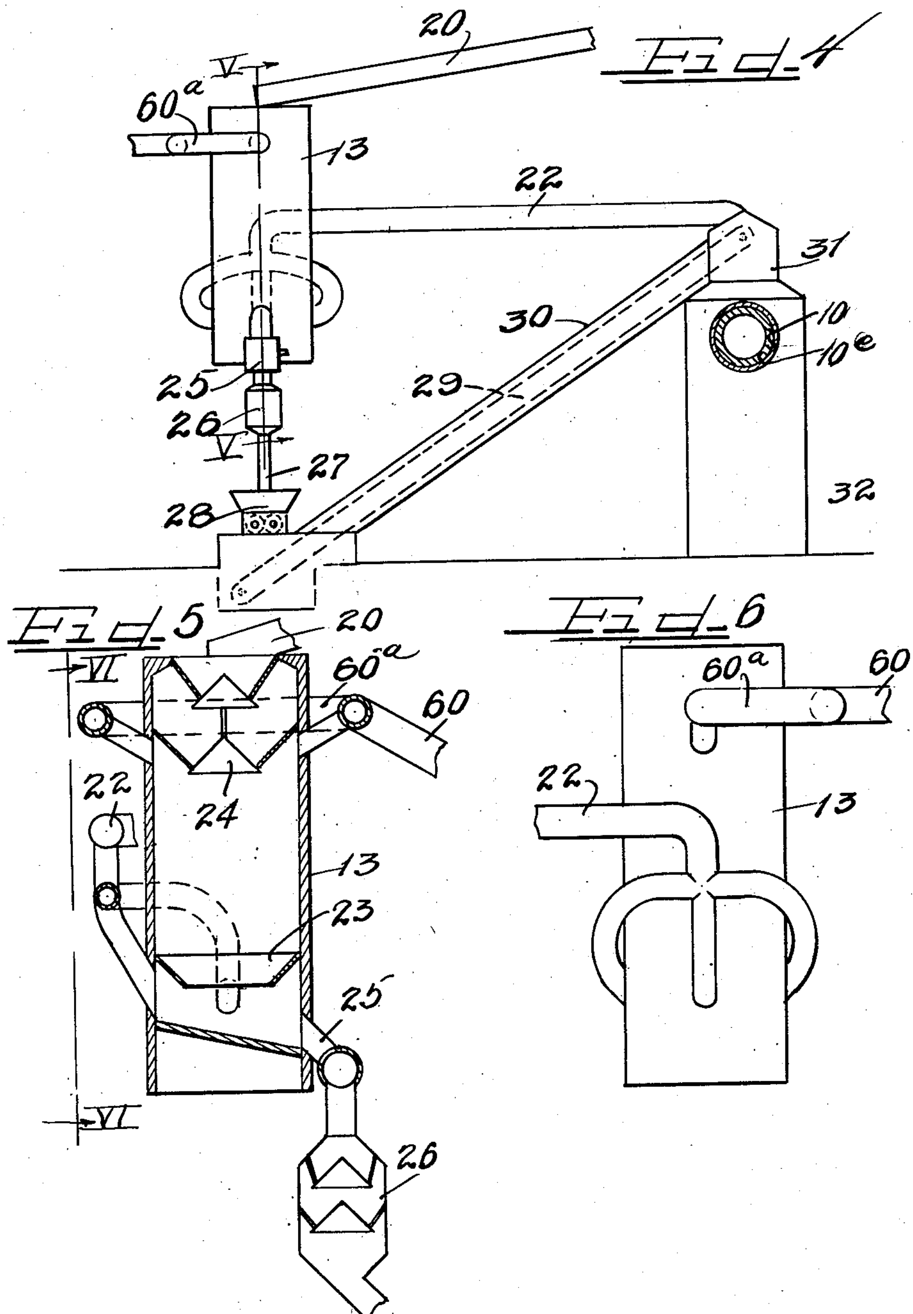

Inventor
Eugene S. Harman
by Charles Hill
Attys.

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CONTINUOUS SMELTING PROCESS

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Fig. 7

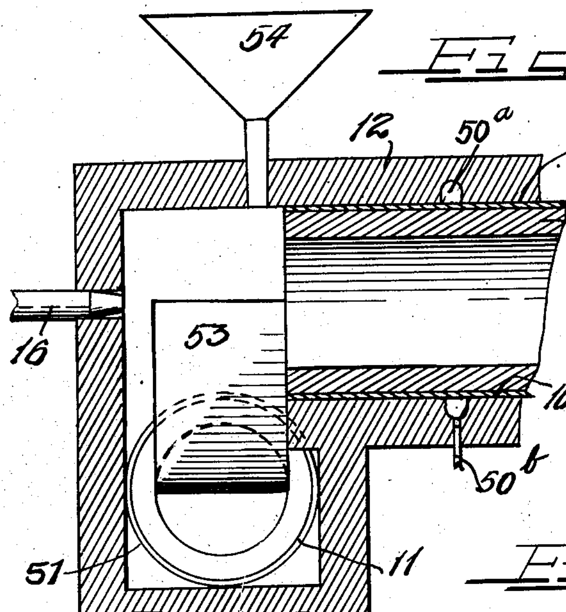


Fig. 9

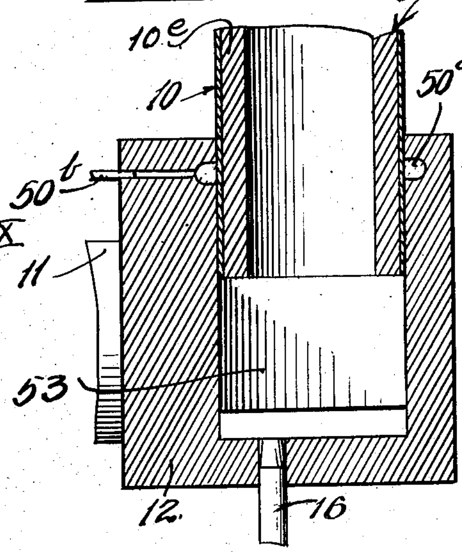
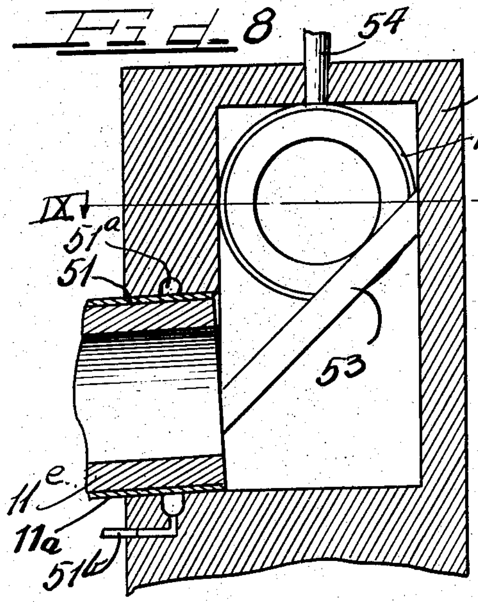


Fig. 8



Inventor
Eugene S. Harman.
by Charles Hill Attys.

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CONTINUOUS SMELTING PROCESS

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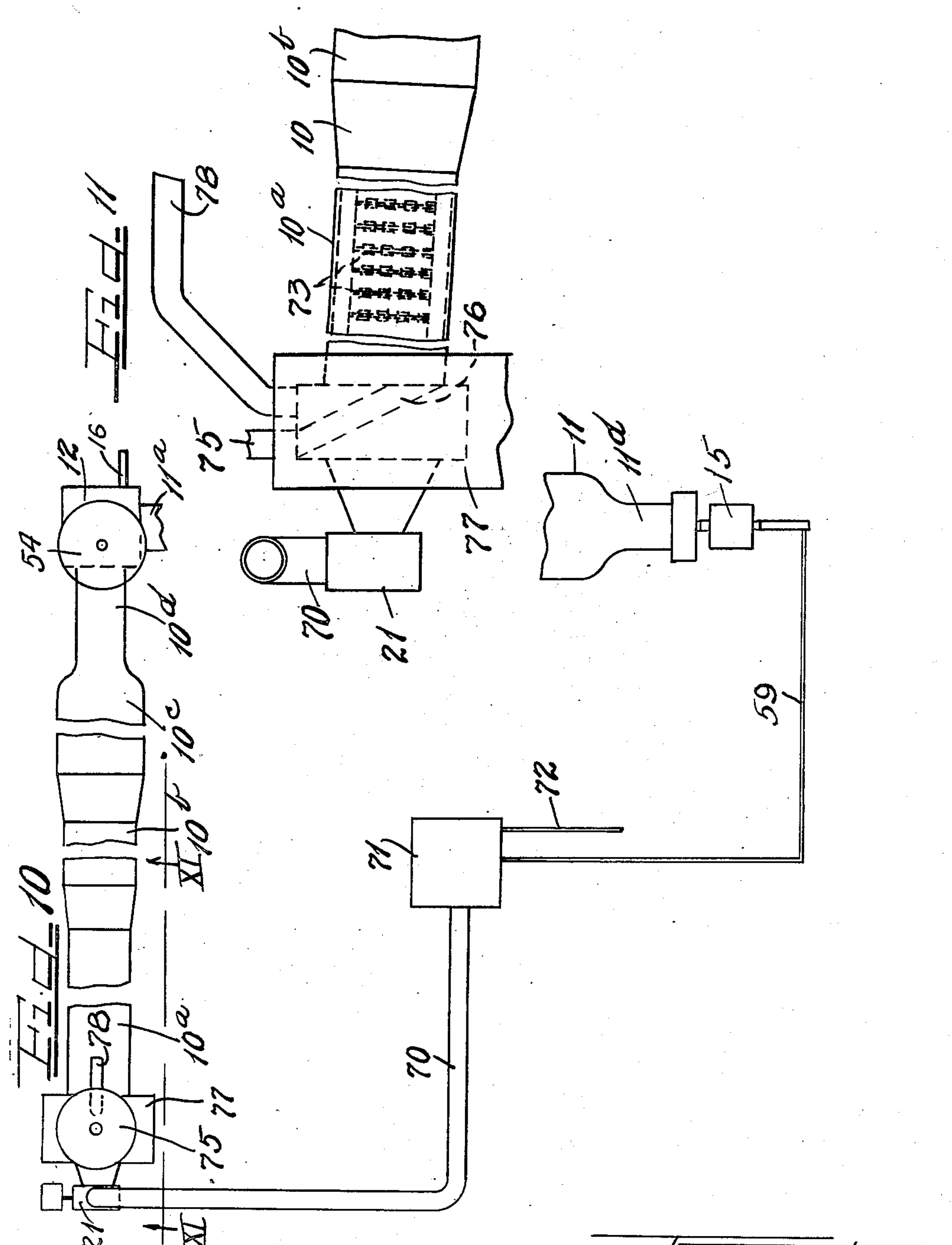

Inventor
Eugene S. Harman.
by Charles Hills
Attys.

Patented Oct. 24, 1950 2,526,659

UNITED STATES PATENT OFFICE

2,526,659

CONTINUOUS SMELTING PROCESS

Eugene S. Harman, Bronxville, N. Y.

Application March 9, 1945, Serial No. 581,857

6 Claims. (Cl. 75—40)

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This invention relates to the reduction of iron ore, and consists in improvements both in method and in apparatus.

My analysis of the problem of producing kiln iron indicates that, in order to secure high tonnages of molten iron with a predetermined carbon content, three conditions must be observed. First, the hearth must be maintained at a high temperature. Second, a reducing atmosphere must be maintained in the reducing zone without interfering with the intensity of the flame heating the hearth and without limiting the length of the hearth zone. Third, the amount of carbon added to the reduced iron must be subject to independent control.

Another difficulty heretofore encountered in the production of iron in kilns is due to the fact that the iron before fusing or melting goes through an intermediate pasty state, at a temperature of from 1900° to 2000° F. At such temperature the pasty iron adheres to the kiln walls, accumulating to the point of choking off the kiln. The use of a boring bar has been suggested for the removal of the accumulated pasty iron. However, if the iron is melted, as it normally is, in the kiln below the zone in which the accumulation occurs, the so-called pasty zone will be located quite a distance from the kiln end, whereby the boring bar must traverse the hearth zone in order to reach the pasty iron. As there is a practical limit to the length of the boring bar that may be used, the size and length of the kiln must be restricted severely. No large capacity kiln can depend on a bar inserted through a discharge end of the kiln for removal of pasty iron.

Still another source of trouble in large capacity kilns for the reduction of iron ore is the extraordinary kiln length required. Such kilns, being lined with refractive material, have considerable weight, and tend to bow and buckle despite all attempts to support the kilns at points distributed throughout the length of the kilns. Attempts have been made to operate kilns in sections that are superimposed one upon another, and in sections that are arranged in stepped tandem formation, but such attempts have failed because of difficulties in providing adequate foundations and in maintaining alignment of the sections. Also, trouble is encountered in obtaining the desired flow of material from one kiln to another. In sectioned kilns, it is also extremely difficult to prevent leakage of hot gases through the housed joints between the kiln sections. These hot gases represent not only thermal and metallurgical losses, but tend to damage the kiln parts exposed thereto.

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These are only a few of the difficulties heretofore encountered in attempts to replace blast furnaces with rotary kilns. The difficulties have been so serious that at the present time no significant amount of iron is produced in rotary kilns, this in spite of the many advantages as to cost and quality of metal obtainable in kiln operation.

The object of my invention is to provide improved apparatus and methods for the commercial production of relatively pure iron particularly an iron that is superior to blast furnace or duplex iron for the charging of open hearth and electric furnaces. In keeping with such object, I solve the problems and eliminate difficulties hitherto confronting the art, with the effect that such production of iron becomes economically feasible, not only with the usual ores of high iron content, but with the inferior ores of low iron content.

Other objects and features of invention will become apparent in the ensuing specification.

In general, my process consists in advancing iron ore and cement-forming materials, together with a limited amount of carbon, first through a low temperature kiln zone wherein the ore is reduced, and then through a high temperature hearth zone wherein the iron is melted. Additional carbon in regulated quantity is normally introduced to the metallic iron in course of advance from the kiln zone to the hearth zone. The flame in the hearth zone is preferably oxidizing in character.

The reduction of the iron ore to iron is effected largely by the carbon charged in limited quantity with the ore, but partly by a reducing flame obtained by the conversion of the oxidizing flame streaming from the hearth zone into the reducing kiln zone.

The conversion of the flame from oxidizing to reducing characteristics is effected by the injection of pre-conditioned or activated carbon into the flame. Such carbon preferably comprises pulverized coke breeze or coal, preheated to incandescence, is blown into the flaming gases leaving the hearth zone, in such manner as to bring about intimate contact between the coke breeze and the flaming gases, without decreasing the temperature of the flame in the hearth zone. The amount of carbon injected, to convert carbon dioxide into carbon monoxide, is sufficient to establish a $CO:CO_2$ ratio adequate to produce an atmosphere that is at least neutral and preferably reducing. While in electric furnace smelting the ratio may be 50:50, in kiln practice I consider a ratio of 65:35 to be safely reducing. As already noted, the reaction between carbon and carbon dioxide is endothermic, with the consequence that the injection of carbon lowers the temperature of the gases at the point of injection.

The pre-conditioned or activated carbon for conversion of the flame may consist of the carbon component of carbon monoxide, or of a hydrocarbon cracked by preheating.

In further accordance with my invention the reducing and hearth zones are, to the extent hereinafter explained, structurally independent. The employment of a relatively short hearth kiln more readily permits the obtaining of the high temperatures, favoring rapid melting of the low carbon iron. The injection of carbon for the conversion of the oxidizing flame to a reducing flame is advantageously effected in the course of flow of the products of combustion from the melting zone or hearth kiln to the reducing zone of the shaft kiln. Carbon should also be added to the reduced iron at the delivery end of the shaft kiln, to protect the reduced iron against reoxidation, as will more fully appear below. The point of separation of the hearth and shaft kilns should be located immediately after the sintering zone in which the iron turns pasty, thus rendering this zone readily accessible to boring bars. The boring bars need not traverse the length of the hearth kiln to reach the pasty iron accumulations on the kiln walls.

I have found that if the two kilns are set at an angle to each other, adequate foundations can be provided for both kilns, even though the kilns are disposed at different levels. The material moving from the lower end of the upper kiln cascades into the upper end of the lower kiln. In order to permit operation of the kilns under slight internal pressure, whereby better control of the combustion gases and maximum temperature may be secured with economy, air seals are provided at the kiln joints. At these seals, cool air is urged into the kilns, rather than permitting hot gases to leak out of the kilns.

My apparatus may, as already noted, further include independently fired hot air stoves, or recuperators, for preheating the air for combustion. Waste stack gases may be burned for such purpose. A vertical kiln shaft may also be included as a part of the kiln train, raw materials being charged into the kiln shaft and there heated by waste kiln gases blown through the shaft.

In the accompanying drawings, I diagrammatically illustrate apparatus in which and in the operation of which the invention is realized:

Figure 1 is a fragmentary plan view of apparatus embodying the invention;

Figure 2 is a fragmentary view of the apparatus, partly in side elevation and partly in section, on the plane II—II of Figure 1;

Figure 3 is a fragmentary end elevation of the apparatus, as seen on the plane III—III of Figure 1;

Figure 4 is a view of the apparatus, partly in elevation and partly in section, on the plane IV—IV of Figure 1;

Figure 5 is an enlarged cross-sectional view, taken on the plane V—V of Figure 4;

Figure 6 is a view in end elevation as seen on the plane VI—VI of Figure 5;

Figure 7 shows to larger scale a transverse sectional view on the plane VII—VII of Figure 1, but with certain parts shown in elevation;

Figure 8 shows to larger scale a transverse sectional view, taken on the plane VIII—VIII of Figure 1;

Figure 9 is a horizontal sectional view, taken on the plane IX—IX of Figure 8;

Figure 10 is a fragmentary plan view of a modified form of apparatus embodying the invention; and

Figure 11 is a larger view, with parts shown in section, taken along the line XI—XI of Figure 10.

Referring to Figures 1 to 9 of the drawings, the apparatus in general includes a shaft kiln **10** and a hearth kiln **11**, set at right angles to one another, and connected at the adjacent ends by means of a housing **12**. The apparatus includes a shaft **13** for preheating raw material, a hopper **14** for charging the preheated raw material into the shaft kiln **10**, a burner **15** for injecting a high-temperature oxidizing flame into the hearth kiln **11**, and an injector **16** for introducing carbon to the flame and gases streaming from the hearth kiln **11** (through the flue formed by the interior of the housing **12**) into the shaft kiln **10**, to convert such flame and gases from oxidizing to reducing characteristics. Cleaners **17** are provided, to prepare the waste kiln gas for service as fuel in blast stoves **18** that heat the combustion air for the hearth kiln. The air is forced through the stoves by blowers **19**.

The devices for preheating and charging the material are shown in Figures 1, 2 and 4 to 6. Ore, limestone, and coal or coke are fed into the preheater shaft **13** by a skip-hoist or conveyor **20**. The material in the shaft **13** is heated by hot waste kiln gas forced by a blower **21** through a branched conduit **22** that opens into the lower end of the shaft **13**. The shaft **13** includes in its lower end a centrally open cone **23** (Figure 5), below which the several terminal branches of duct **22** open. The chamber or space below the cone **23** is effective to promote more uniform gas flow and better material distribution within the shaft. Since the shaft **13** is under internal gas pressure, a bell-valve **24** is provided at the inlet of the shaft, while the duct **25** leading from the outlet of the shaft terminates in a bell-valve hopper **26**. The heated material discharged through the bell-valve hopper **26** moves through a duct **27** into a grinder **28**, and thence over a conveyor **29** (Figure 4) to the kiln hopper **14**. In order to prevent the loss of any of the fine material, the conveyor **29** and the kiln hopper **14** are enclosed within air tight housings **30** and **31**, respectively.

Scales (not shown) are provided for weighing material charged into the preheater shaft **13**.

The shaft and hearth kilns **10** and **11** are supported upon foundations **32**. The two kilns are inclined to the horizontal and set at different levels, and, as viewed in plan, they extend at right angles to one another. The kilns are adapted to be rotated by means of powerfully rotated pinions (not shown) meshing with gears **33**.

The shaft kiln **10** includes a zone **10*a*** for the further heating of the preheated and ground material delivered by the hopper **14**, and a reduction zone **10*b***, enlarged at **10*c*** to form a reservoir or holding zone just ahead of sintering zone **10*d***. The sintering zone is reduced in diameter, to increase the efficacy of the heat within the kiln in rendering the reduced iron pasty.

The hearth kiln **11** is set below the lower end of the shaft kiln **10**, to facilitate the flow of material from the latter kiln into the former. As sintering (the transition of the reduced iron to pasty condition) may not be entirely complete by the time the advancing material enters the hearth kiln, the upper end of the latter kiln is constricted, as at 11a, providing a zone in which the sintering action is further accelerated, and completed. The melting or fusion of the reduced iron is effected in hearth zone 11b, while a wide hearth zone 11c serves to receive molten iron for discharge, either continuously or intermittently, through a tap hole 34 into a ladle 35 mounted for travel on tracks 36. The cement clinker, advancing through the zone 11c, continues through a zone 11d whence it is discharged through a hopper 37 to a conveyor 38. The conveyor carries the cement clinker to a cooler (not shown), and from the cooler the clinker is conveyed to storage bins or cars, to await pulverization, to form cement. The hopper 37 is provided at its discharge opening with an automatic bell-valve that is in this case water cooled.

While the drawing shows the separation of the hearth kiln and shaft kiln to be at the sintering zone, it is to be understood that this division may be provided immediately beyond the end of the sintering zone, the important feature being that the zone where pastiness occurs shall be accessible to a boring bar, and the unsintered ore not exposed to the oxidizing flame from the melting chamber.

The high hearth temperatures make for a short life of the refractories with which the hearth kiln is lined. Provision is made for replacement of a worn hearth kiln by a spare.

The hearth kiln may be supported on bearings, and the kiln housing may be supported on tracks. The tracks, as well as permanently located hydraulic jacks, may be provided under the kiln. The jacks may be equipped with rollers arranged to engage the kiln shell when the jacks are elevated. With such an arrangement the hearth kiln may be moved out of the kiln train, and through the use of the jacks the kiln may be raised from its normal bearings and portable bearings mounted on track buggies may be inserted under the kiln. The kiln may then be moved out of the line of production for relining, and a spare kiln previously heated may be moved into the line. This makes possible the relining of a worn hearth kiln, without undue interruption of production.

A boring bar 42, mounted on a carriage 43 movable on tracks 44 and 45, is provided for removing accumulated pasty iron from the walls of the kiln zones 10d and 11a. Another boring bar 42, mounted on a carriage 47 movable on a track 48, is provided for removing any slag or iron carry-over that may accumulate on the wall of the kiln zone 11d.

The housing 12 and the devices associated therewith are best shown in Figures 7 to 9. The kiln ends 10d and 11a extend into apertures 50 and 51 in the walls of the housing. The apertures are provided with circumferential grooves 50a and 51a, into which air under pressure is admitted through ducts 50b and 51b, for preventing leakage of hot gases between the kiln ends and the housing. An apron or chute 53 serves to guide material discharged from the kiln 10 into the kiln 11, and this chute may be water-cooled. Carbon or other material may, through a funnel-shaped hopper 54, be added to the pasty iron passing from the shaft kiln 10 to the hearth kiln 11. As shown, the kilns 10 and 11 are interiorly coated with refractory linings 10e and 11e.

Coal or other fuel is conveyed to coal preparation equipment 56 by a conveyor 57. The coal preparation equipment 56 will include coal pulverizers, and coal preheaters. In the event that coke breeze is used, it will be heated to a temperature approximating 1800° F., by burning the distilled gases with sufficient combustion air in contact with the coke particles. The treated fuel is conveyed through a duct 58 to the powdered incandescent coke burner 15. Preheated combustion air is fed through a line 59 extending from the blast stoves 18 to the burner 15.

As mentioned above, the hot waste gases of the kiln system are blown through the conduit 22 and through the raw material in the preheater shaft 13. After having yielded most of their heat to the raw material, the kiln gases conducted through a pipe 60 (branched at 60a to form a plurality of openings near the top of the shaft 13) to the gas purifiers 17, which may consist of dust precipitators and gas washers, and then through a conduit 61 to the hot blast stoves 18 for combustion therein. Kiln gases in excess of the requirements of stoves 18 are led away through a conduit 62, and conserved for combustion elsewhere.

The air blowers 19 are connected to the hot blast stoves by a conduit 65 that has three valved branches 65a entering the hot blast stoves 18, and the waste gas conduit 61 has three valved branches 61a entering the stoves. A conduit 66, having three valved branches 66a, connects the stoves to a stack 80 for carrying off the stack gases. All of these conduits and connections make possible the operation of any one of the hot blast stoves, or the joint operation of any two or more of the stoves.

Figures 10 and 11 show an apparatus generally similar to that shown in Figures 1 to 9, with respect to kiln and burner arrangements, but provided with simpler and less expensive devices charging the raw materials and preheating the combustion air. Parts similar to the parts of the apparatus of Figures 1 to 9 are indicated with like numerals in Figures 10 and 11. The dissimilar parts include a conduit 70 for leading waste kiln gas directly to a recuperator 71, wherein waste gas is burned for preheating the air supplied by a cold blast line 72.

The kiln zone 10a is provided with slack chains 73 suspended from the inside of the kiln at spaced-apart points and serving to settle the dust that otherwise would be carried over to the recuperator.

The coke, or coal, and limestone are charged into the kiln end zone 10a through a hopper 75 discharging upon an apron 76 in a housing 77 for the kiln end. Iron ore in slurry form is charged through a conduit 78 to the apron 76, whence it flows into the kiln end zone 10a.

In the operation of the apparatus of Figures 1 to 9 ore, limestone and coal are charged into the shaft 13 and are there heated to about 400° F. by the waste kiln gases whose temperature is between 650° F. and 700° F. The temperature of the gases leaving the preheater shaft will be about 350° F. With low velocity of the gases flowing through the material, and with a relatively shallow depth of material, the required blower pressure will be about 1 to 1½ pounds per square inch.

The use of the preheater shaft avoids the undesirably high stack temperatures of prior kilns. Additionally, the cost of drying and crushing the ore and limestone preparatory to charging is eliminated.

The upper section of an old blast furnace may be used as the shaft for drying and preheating the raw materials prior to grinding and feeding into the rotary kiln for smelting. Also, old blast furnaces of low capacity may be rehabilitated, as all parts of the plant except the hearth and bosh section of the furnace may be utilized. The blowing equipment may be low pressure blowers, as the blast pressure need only be high enough to overcome the resistance through the stoves. But whatever the structure, the ore is roasted prior to smelting, thus making it possible to use high sulphur ores or pyrites of low cost.

In cases where fuel is cheap and the drying of ore and limestone prior to grinding is not desired, the preheater 13 and the blower 21 may be eliminated, and the material may be discharged into the rotary kiln, in which case stack gas losses will be somewhat higher. Where a small and less expensive installation is desired, the apparatus of Figures 10 and 11 may be employed. Such installations may be used to produce liquid metal having desired carbon and silicon contents to replace or supplement either steel scrap or pig iron. A suitable starting material is the fine ore or flue dust always obtained in and detrimental to the operation of blast furnaces. Such fine ore is charged in the form of slurry into the kiln. Lump ore under ½ in. may be charged, if desired. Raw limestone, preferably though not necessarily ground, but at least crushed, is charged in quantities as required into the kiln. The slag may be fused and tapped with the iron, if it is not so viscous as to adhere to the kiln walls to an objectionable extent. A better method is to maintain a kiln temperature below the melting point of the slag, or to maintain a calcium content in the slag sufficient to raise the slag melting point above the kiln temperature, it being noted that the reaction between the lime and the impurities in the liquid iron is just as effective when the slag is solid as when the slag is fused.

The charging of the ore in the form of a sludge or slurry and the provision of the dust-catcher chains in the high end of the shaft kiln reduces the flue dust content in the waste kiln gases to a minimum, so that the use of the most elaborate elements 17 of the apparatus of Figures 1 to 9 may be avoided. The recuperators used in place of stoves will operate with burning gases that contain more dust than gas treated in gas washers and dust catchers, and they are less expensive than blast stoves.

In the event that liquid slag is discharged continuously, a slag tap hole is provided in the clinkering zone 11*d*, and the lower end of the kiln 11 is dammed slightly to act as a retaining wall, preventing slag discharge through the kiln end.

The plant of Figures 1 to 9 is capable of producing three hundred fifty tons, or more, per day. In the event that an output in excess of that obtainable with a single kiln unit is desired, two or more kiln units may be installed. They may be arranged parallel to each other, and a single set of auxiliaries, such as recuperators, blowers and material or slag handling equipment, or the stoves and preheater shaft of the more elaborate plant of Figures 1 to 9, will serve for the several kiln units.

The plant of Figures 1 to 9 may be adapted for the selective charging of lime and ore of different analyses. A vertical partition wall may be used to divide the preheater shaft 13, and two mixes of charging materials may be selectively withdrawn from the shaft through a divided hopper.

In the practice of the present invention, the minimum carbon addition required is 114 pounds per ton of iron (which yields carbon free iron) if a $CO:CO_2$ ratio of 70:30 is to be maintained in the shaft kiln atmosphere. This minimal carbon consumption is dependent upon a partial reduction of the ore by the carbon monoxide content of the kiln gases. About 450 pounds of carbon per ton of iron (or about 25% to 30% of the total fuel consumed) is required for complete reduction of the iron ore by solid carbon. The amount of carbon charged with the ore can be limited to that required for reduction (about one-quarter of that charged in a blast furnace), when carbon is added, as advantageously it is, at the entrance to the melting zone. The carbon content of the iron may be varied anywhere between zero and saturation (a little over 4%) by adding appropriate amounts of carbon to the reduced iron. Thus, to produce an iron having a 1.5% carbon content, at least 30 lbs. of carbon per ton of iron must be added as the iron reaches the hearth kiln. When the waste gas temperature is 350° F., 876 lbs. of carbon per ton of iron are required for combustion purposes. Hence, to produce iron containing 1.5% carbon, the minimum total carbon requirement will be 114+876+30 or 1020 lbs. per ton of iron. This quantity corresponds to 1457 lbs. of coal having a 70% carbon content. The solid carbon used may be charcoal, or oil coke, if coal or coke is not available.

The source of the sulphur in iron made from Lake Superior ore is the coke charged with the ore. Since my process consumes only one-fourth the quantity of coke consumed in blast furnace operation, the sulphur content of my kiln iron will be only one-fourth that of blast furnace iron. Ores from certain other sources have a high sulphur content, and thus increase the sulphur content of the charge. In these cases, as well as in cases where it is desired to produce cement clinker from the slag, a high lime content in the charge is required. The slag should be clinkered rather than molten, since a clinkered slag will not attack the kiln lining (which can be made with a cement, clinker base), and since melting will adversely affect the cement-forming properties of the slag. When low carbon iron is made and cement clinker is not produced, a low lime charge may be made, in which case the slag will be liquid. The most serious objection to liquid slag is its tendency to attack the kiln lining. The most satisfactory practice is to maintain the slag in clinker form, even though this practice requires crushing the slag-forming materials. Ordinarily cement-forming material is charged in amounts to make about 9.8 tons of cement clinker per ton of iron, and in such case about twice as much limestone is charged as in blast furnace practice.

A flame containing preferably at least 70% carbon monoxide and having a temperature of 2300° F. enters the lower end of the shaft kiln 10.

Carbon, or coke, or coal is fed through the hopper 54 into the material delivered from the reduction kiln, in amounts varied as required to protect the iron from reoxidation in the hearth kiln, and to remove any FeO present in the burden of the hearth kiln, as well as to supply whatever carbon content may be desired in the finished kiln iron. Thirty pounds of carbon per ton of iron is normally a suitable amount.

Nodulized iron at approximately 2000° F. is deposited into the hearth kiln along with carbon and cement clinker. The reoxidation loss is minimized, because of the carbon applied to the nodules, and since the time required to melt the iron at this temperature is materially less than that required to melt a relatively cold sponge iron in an open hearth. The molten iron runs along the sloping kiln bottom to the tap hole, and is discharged as soon as the iron is melted. Alternatively, if the iron is tapped periodically from the hearth kiln, the depth of bath will be so determined as to minimize reoxidation.

The hearth and shaft kilns are operated at a balanced draft or a slight pressure, to avoid uncontrolled air infiltration.

The hot flame injected into the hearth kiln may be generated from any type fuel. The fuel preheater, referred to above as part of the fuel preparation equipment **56**, may be used to secure more rapid combustion and higher temperature where coke breeze or anthracite culm is used. Such fuels may be heated approximately to 2000° F., by admitting air into the heater or container in sufficient quantities partially to burn the gas distilled from the fuel. The gases so distilled may be recirculated through the fuel, and the surplus gas may be bled off into a gas main to be used as fuel for other purposes. Fuel-air ratio proportioners are preferably installed in the burner **15**, so that a proper percentage of carbon dioxide is maintained in the melting zone of the hearth kiln. The flame should not be supplied with excess air, as this would necessitate additional fuel for converting the oxidizing flame to a reducing flame, as explained below. A small carbon monoxide content is preferable from the standpoint of fuel economy, low oxidation loss, and generation of high temperatures. If desired, the burner **15** may include upper and lower flame injectors, and the lower injectors may be adjusted to give a reducing flame, thus forming over the bath a relatively thin layer of reducing atmosphere, through which radiates heat from the oxidizing flame of the larger injectors.

Where low initial construction cost is desired, blast stoves may be provided only for the air for the reducing flame injectors, the air for the oxidizing flame injectors being preheated in recuperators.

The oxidizing gases leaving the hearth kiln are converted to reducing gases before entering the heating and reducing kiln **10**. This conversion can be effected simply by adding carbon monoxide to the oxidizing flame through the injector **16**. Carbon or other fuel may similarly be injected into the oxidizing flame. In atmospheres in which the $CO:CO_2$ ratio is 50:50, reaction between iron oxide and solid carbon occurs according to the equation:

$$Fe_2O_3+2C=2Fe+CO_2+CO$$

However, reactions according to the formula

$$Fe_2O_3+C=2FeO+CO$$

occur in atmospheres of considerably smaller carbon monoxide content. These reactions proceed at velocities dependent on the partial pressures of the carbon monoxide and carbon dioxide gases and follow the phase rule. For the purpose of the present invention it is sufficient if the $CO:CO_2$ ratio at the entrance to the shaft kiln be maintained at 65:35.

The simplest and preferred method of converting the oxidizing flame to a reducing flame is to blow incandescent pulverized coke breeze into the chamber space, so as to bring about intimate contact between the particles of coke and the gases leaving the hearth kiln. The amount of coke breeze injected must, of course, be adequate to convert a sufficient amount of carbon dioxide into carbon monoxide to render the heating kiln atmosphere neutral or reducing. Sufficient fuel may be burned in the injector **16** to heat the particles of coke to incandescence, or the coke may be heated before entering the injector by burning gases distilled from the coal in direct contact with the coke. The use of oil or gas fuel to convert the oxidizing flame to a reducing flame is also held in contemplation, even though the high hydrocarbon content of such fuel produces a high water vapor content in the combustion gases. As already mentioned, water vapor is oxidizing, and, in fact, water vapor seems to increase the oxidizing properties of combustion gases. However, as in the case of the carbon dioxide in the combustion products, the oxidizing tendencies of the water vapor can be counteracted by the provision of a corresponding excess of hydrogen or carbon monoxide.

Of the heat work performed in smelting iron ore, approximately 25% is done in the hearth kiln and the remaining 75% is done in the heating and reducing kiln. The relation between the heat consumed and supplied, and the relation between the fuel consumption and the temperatures obtained at various points, are made evident in the following calculations, in which the flame conversion effected by addition of carbon monoxide is considered first:

It is assumed that the stack gas will have a $CO:CO_2$ ratio of 70:30 and that the fuel used is coke breeze or anthracite culm burned in multiple burners injecting a lower reducing flame and an upper oxidizing flame in such ratio that the average analysis of the combustion gases in the hearth zone shows a $CO:CO_2$ ratio of 10:90. The combustion air is preheated to 1500° F. The temperatures and products of combustion may be calculated as follows:

PRODUCTS OF COMBUSTION OF 1 LB. OF CARBON TO 10% CO AND 90% CO_2

Air required for CO $\frac{0.1\times1.33}{0.23}=0.578$ lb.

Carbon, 0.1 lb.

Products of combustion to CO, 0.678 lb.

Air required for CO_2 $\frac{0.9\times2.66}{0.23}=0.578$ lb.

Carbon, 0.9 lb.

Products of combustion to CO_2, 11.3 lb.

Total products of combustion $=0.678+11.3=11.987$ lb. per lb. carbon

Heat of combustion

$CO=0.1\times4375=437$ B. t. u.
$CO_2=0.9\times14{,}750$, 13,280 B. t. u.
Heat supplied by air at 1500° F. $=(11.3-1)\times1440\times0.247)=3{,}660$ B. t. u.
Total heat supplied, 17,377 B. t. u. per lb. of carbon

Theoretical flame temperature

$$\text{Flame temperature}=\frac{17377}{11.978\times0.257}=5650°\text{ F.}$$

Carbon to be added for conversion

If, in order to create a reducing atmosphere having a $CO:CO_2$ ratio of 70:30, a CO flame is added at the entrance to the shaft kiln, the amount of carbon required as CO can be calculated as follows:

Carbon present in gas as CO_2, 0.9 lb.
which must equal 30%, hence
Carbon 100%=0.0:0.3=3.0 lb.
and
Carbon as CO=3.0−0.9=2.1 lb.

The initial CO content of the flame is 0.1 carbon. Hence carbon to be added=2.1−0.1=2.0 lb.

Temperature of added CO flame

$$\text{Air required}=\frac{2\times 1.33}{0.23}=11.55\text{ lb.}$$

Carbon added, 2.0 lb.
Total products of combustion, 13.55 lb.
Heat supplied by air at 1500° F.=11.55×1440×0.247=4100 B. t. u.
Heat supplied by combustion to CO=2×4375=8750 B. t. u.
Total heat supplied, 12850 B. t. u.

$$\text{Flame temperature}=\frac{12850}{13.55\times 0.247}=3690^\circ\text{ F.}$$

Total net heat supplied per lb. of carbon fired in hearth

By combustion of one lb. of carbon to CO_2 in hearth, 17377 B.t.u.
By combustion of two lb. of carbon to CO at hearth exit, 12850 B.t.u.
Total heat supplied, 30227 B.t.u.

$$\text{Heat supplied per lb. of total carbon}=\frac{30227}{3}=10076\text{ B. t. u.}$$

Stack loss per lb. carbon at 350° F. and $CO:CO_2$ ratio of 70:30, 588 B.t.u.
Net heat available per lb. carbon=10076−588=9488 B.t.u.

Heat required per lb. of iron

For heating in reduction kiln, 1330 B.t.u.
For reaction in reduction kiln, 1580 B.t.u.
For radiation in reduction kiln, 300 B.t.u.
Total heat required in reduction kiln, 3210 B.t.u.
Total heat required in hearth kiln, 972 B.t.u.
Total heat required (above stack gas), 4564 B.t.u.
Less heat recovered from CO_2 and CO released, 394 B.t.u.
Net heat required (above stack gas), 4170 B.t.u.

Carbon required per lb. iron

$$\text{Carbon required}=\frac{4170}{9488}=0.438\text{ lb.}$$

⅔ of this carbon, or 0.292 lb., is fired at the shaft kiln.
⅓ of this carbon, or 0.146 lb., is fired at hearth kiln.

Temperature of gases leaving hearth

Heat supplied in hearth per lb. of iron=0.146×17377=2535 B.t.u.
Heat required in hearth per lb. iron, 972 B.t.u.
Heat in gases leaving hearth=2535−972=1563 B.t.u.
Weight of gases per lb. or iron leaving hearth=0.146×11.978=1.75 lb.

$$\text{Temperature gases leaving hearth}=\frac{1563}{1.75\times 0.257}=3480^\circ\text{ F.}$$

Average temperature at sintering zone

Temperature of gases leaving hearth, 3580° F.
Temperature of CO flame added at exit, 3690° F.
Average temperature of gases entering shaft kiln

3480×⅓=1160
3690×⅔=2460
Average temperature—3620° F.=3620° F.

The theoretical temperature at the bosh of a blast furnace is 3090° F. Hence, the theoretical temperature of 3620° F. at the kiln sintering zone is ample for raw material heating and reduction purposes.

Turning now to the case in which incandescent pulverized carbon is added at the entrance to the shaft kiln for the purpose of converting the $CO:CO_2$ ratio from 10:90 to 70:30, the reaction and consequent heat relationships will be as follows, it being assumed that the carbon thus added is in the form of a low volatile coal heated to 200° F. by partial combustion to CO:

Amount of coal to be burned to CO to heat the coal to 2000° F.

Heat required per lb. of carbon 0.42×(2000−60)×1=815 B.t.u.
Heat of combustion of 1 lb. of carbon to CO, 4375 B.t.u.

$$\text{Amount carbon to be burned}=\frac{815}{4375}=0.186\text{ lb. carbon per lb. carbon}$$

Hence, for every pound of carbon added, approximately 0.8 lb. will be solid and about 0.2 lb. will be in the form of CO.

The reaction is expressed by the equation

$$CO_2+C=2CO$$

The gases approaching the outlet of the hearth kiln comprise $9CO_2+1CO$. The carbon added at the entrance to shaft kiln is in the form of $8C+2CO$, and the desired atmosphere is $7CO+3CO_2$. For every pound of carbon burned in the hearth, 0.545 lb. of carbon is added at the entrance to shaft and the mixture there will be:

$$9CO_2+1CO+0.545(8C+2CO)=9CO_2+1CO+4.36C+1.09CO$$

The 4.36 parts of carbon will unite with 4.36 parts of the oxygen in the $9CO_2$ to form $4.36CO+4.36CO+4.64CO_2$. The total carbon gases will be 10.81+4.64=15.45. The CO content will be 10.81:15.45=70%, and the CO_2 content will be 4.64:15.45=30%, so that the $CO:CO_2$ ratio will be 70:30.

The heat required for this reaction is 10400 B.t.u. per lb. of carbon converted from CO_2 to CO or

$$\frac{4.36\times 10{,}400}{15.45}=2940\text{ B. t. u. per lb. of carbon fired}$$

The heat supplied by burning 1.09 lb. of carbon to CO, together with the heat of reaction of 4.36 lb. of carbon burned to CO, will be

$$\frac{(1.09+4.36)\times 4375}{15.5}=1438\text{ B. t. u. per lb. of carbon fired}$$

The oxygen required to fire 20% of the carbon added to CO for heating the carbon to incandescence is 1.33×0.109=0.145 lb., and the air brought in is

$$\frac{0.145}{0.23}=0.631\text{ lb. per lb. of carbon fired}$$

With an air temperature of 1500° F., the heat brought in by the air will be 0.631×(1500−60)×0.247=224 B.t.u. per lb. of carbon fired. Hence, the heat for the reaction will be 1538+224−2940=1178 B. t. u.

The stack loss per lb. of carbon in this case will be the same as in the previous case, in which the $CO:CO_2$ ratio in the exit or waste gas is 70:30. The stack gas temperature being 350° F., the stack loss will be 588 B.t.u. per lb. of total carbon.

The total carbon used will be 1.545 lb. per lb. of carbon fired in the hearth kiln, and the total net heat supplied per lb. of total carbon fired will be the same as in the previous case, the $CO:CO_2$ ratio being the same. Hence, the total carbon fired per lb. of iron will be the same, i. e., 0.438 lb. However, the amount of high temperature heat supplied to the hearth kiln will be considerably greater, since approximately two-thirds of the total carbon fired is burned to CO_2 in the hearth, as compared with approximately only one-third, when CO gas instead of incandescent carbon is added at the entrance to the shaft kiln. The theoretical flame temperature of combustion in the hearth kiln will be the same in both cases, but in the latter case, in which two-thirds of the total carbon is burned to CO_2, the temperature of the gases at the outgo end of the hearth kiln will be materially higher, since the heat given up in the hearth (being the same in both cases) comprises a smaller percentage of the heat available to the hearth. However, the heat required in said latter case for the endothermic reaction between the CO_2 and the incandescent carbon at the entrance to the shaft kiln takes up heat from the gases, so that the gases entering the shaft kiln are at the same temperature in both cases.

From the foregoing calculations it will be understood that, in the case in which CO is added to the flaming gases leaving the hearth kiln, the total heat supplied to the hearth is 0.146×17,377=2535 B. t. u. per lb. of iron. The hearth absorbs 972 B. t. u. per lb. of iron from the gases, leaving 1563 B t. u. per lb. of iron in the gases. In the case in which incandescent carbon is added to the flaming gases at the outgo end of the hearth, the amount of carbon fired in the hearth is

$$\frac{1\times100}{1.545}=64.75\%$$

of the total fuel or 0/6475×0.438=0.284 lb. carbon per lb. of iron, wherefore the heat supplied to the hearth is 0.284×17,377=4925 B. t. u. per lb. of iron. Upon giving up 972 B. t. u. in the hearth, the gases still contain 3953 B. t. u. per lb. of iron. In other words, with one and the same over-all fuel consumption in both cases, there is a variation of approximately 100% in the amount of high temperature heat which is supplied to the hearth kiln, and manifestly the operator can enjoy wide control over the amount of high temperature heat supplied to the hearth without substantial variation in fuel consumption.

From the foregoing specification, it will be understood that either liquid or gaseous fuel may be used in the hearth kiln; that under proper conditions liquid or gaseous fuel may be used for converting the oxidizing gases of the hearth to reducing gases for the shaft kiln; and that the methane (and like neutral gases included in such fuel), as it breaks down into hydrogen and carbon, provides reducing agents that are effective in the conversion of the oxidizing gases. The water vapor, formed by combustion of the hydrogen, must be neutralized by hydrogen or carbon monoxide, at least to the same extent as compensation is made for carbon dioxide.

If oil or gas is burned in the hearth, the exit gases will contain water and carbon dioxide, and if incandescent carbon is added at the entrance of the hearth, the reactions will be somewhat as follows:

$$2H_2O+2CO_2+2C=H_2O+H_2+CO+2CO+CO_2=H_2O+H_2+3CO+CO_2$$

The character of the resulting gas will be slightly reducing, for while the $H_2:H_2O$ ratio is 50:50, or neutral, the $CO:CO_2$ ratio is 75:25, which is highly reducing.

If oil or gas is also injected at the shaft kiln, the methane or other hydrocarbon will break down into carbon and hydrogen in the presence of the hot gases flowing from the hearth kiln. Alternatively, the methane or other hydrocarbon may be cracked in the injector. The reactions occurring will be somewhat as follows:

$$2H_2O+2CO_2+2CH_4=2H_2O+2CO_2+2C+4H_2=H_2O+H_2+CO+CO_2+2CO+4H_2=H_2O+5H_2+3CO+CO_2$$

The resulting atmosphere is highly reducing, as the $CO:CO_2$ ratio is 75:25, and the $H_2:H_2O$ ratio is 84:16. The economy of such practice is dependent upon the analysis and cost of the fuel, and also upon the value of the waste kiln gases as fuel. When such gas or liquid fuel is burned in the hearth kiln, and is added to the gas issuing from the hearth kiln, the only solid carbon required is that fed with the ore for reduction, together with that added at the entrance to the hearth kiln to obtain the desired iron analysis and to insure against FeO loss in the clinker. As stated hereinabove, this solid carbon will amount to from 144 to 458 lb. per ton of iron.

If desired, the carbon-monoxide burners for the hearth kiln may be fed with oxygen in amounts so regulated as to furnish a hot carbon monoxide flame. The following calculations will indicate the amount of oxygen required to be added:

Heat supplied

(W=lbs. of O_2 enriched air required per lb. of carbon for CO flame)

By O_2 enriched air at 1500° F.=(1500−60)×0.247×W=356 W B. t. u.

By combustion of 1 lb. of C to CO, 4375 B. t. u.

Total heat supplied per lb. carbon (4375+356 W) B. t. u.

Pounds (W) of oxygen enriched air per lb. carbon

Products of combustion per lb. carbon (1+W) lb.

Flame temperature desired with O_2 enriched flame 5650° F.

$$5650=(4375+356\ W):(1+W)\times0.257$$
$$W=2.67$$

In other words, 2.67 lbs. of oxygen enriched air per lb. of carbon must be supplied to raise the temperature of the carbon monoxide flame to 5650° F.

The amount of oxygen required for burning carbon to carbon monoxide is (16:12) or 1.33 lb. per lb. of carbon. Therefore, the oxygen percentage in the oxygen enriched air is

$$(1.33\times100:2.67)\text{ or }49.8\%$$

and the nitrogen content of the air is 50.2%. The original nitrogen content of the air is 0.77 lb. per lb. of air. Since 0.77 lb. of nitrogen comprises 50.2% of each pound of oxygen enriched air, it follows that each pound of original air will correspond to 0.77:0.502 or 1.535 lb. of enriched air which contains 0.498×1.535 or 0.764 lb. of oxygen. The original oxygen content of the air being 0.23 lb. per pound of air, 0.534 lb. of oxygen must be added, which represents 0.534×100:1.535 or 34.8% of the enriched air.

The amount of carbon burned at the hearth kiln, when CO is added at the hearth kiln exit, is 0.438:3 or 0.146 lb. per pound of iron. For each 10% of this carbon burned to CO the amount of oxygen required is 0.0146×1.35 or 0.0194 lb. of oxygen. The amount of enriched air required is 0.0194:0.498 or 0.039 lb., of which 0.348×0.039 or 0.01358 lb. is added oxygen. If such working is resorted to where incandescent solid carbon is added at the hearth kiln exit, the cost of oxygen would be twice as great, because ⅔ instead of ⅓ of the total fuel is burned at the hearth. Carbon monoxide generated at 5650° F. with oxygen enriched air may be injected either from the lower burners of the hearth or from all of the burners, to provide a protective blanket over the hearth metal.

The use of oxygen for obtaining high hearth temperatures with a carbon monoxide flame is particularly suitable to the reduction of ferro-manganese or ferro-silicon, both of which are more susceptible than iron to oxidation.

The use of an oxygen-fed carbon monoxide flame to produce high temperature heat for the hearth section is but one of several permissible methods of obtaining the essential high temperature heat required to utilize fully the capacity of the hearth zone. And as already described, another method of obtaining a flame sufficiently hot to supply the heat required by the hearth zone is to preheat both the fuel and the combustion air. It is to be understood, therefore, that my invention is not limited to any particular method of providing the essential high flame temperature, not to the specific procedural or structural features disclosed.

The shaft kiln may be adapted to be axially oscillated rather than rotated, or it may be both oscillated and rotated. The melting chamber or hearth zone may consist in the rotary kiln shown, or in a stationary furnace, or a tilting furnace, the primary purpose in separating the hearth zone or melting chamber from the reducing zone being to remove the heating and metallurgical reactions from the melting process, whereby the best conditions for each may be established independently of the other, or substantially so. The reduction of the ore proceeds continuously, and in the course of advance of the ore towards the discharge end of the shaft kiln the ore requires progressively a more intensive reducing atmosphere, as it is exposed to progressively higher temperatures. Perfect accommodation to this circumstance of operation is gained by the separation of the shaft kiln from the hearth.

The coal used in the practice of invention is pulverized, and may be coked by heating it with hot gases in cyclone apparatus, thus driving off the volatiles as by-product gas. The coke so produced may be heated to incandescence by passing it through a recuperator. The apparatus disclosed in Letters Patent Nos. 1,954,350, 1,954,351 and 1,954,352 granted in the year 1934 to F. L. Dornbrook et al. may be used for this purpose.

The gas generated in this coke-producing operation will consist largely of hydrogen and as such, is not the most desirable for firing the kilns.

If there is an auxiliary field of use for by-product gas and end products such as tar, the means used for carbonizing the coal for injection into the carbon dioxide flame may profitably be employed for carbonizing the coal to be burned in the melting kiln. A larger quantity of by-products is then obtained, and may be credited against the cost of reducing the iron ore. With the useful employment of by-products, all of the benefits now available to the iron and steel industry through the employment of coke ovens in connection with blast furnaces will be realized in the practice of my kiln process, and at a much lower cost. There is an additional advantage in such practice; that is, carbonized coal need not be quenched and then reheated to incandescence, as it is in a blast furnace. The gas yield is higher in my process, because I do not employ regenerators in my coking apparatus. In the by-product coke industry about 40% of the total gas generated is used in heating regenerator checkers.

The iron produced by my process is relatively pure. The carbon content is low, since the iron contacts only relatively small amounts of coal or coke. The iron contains but a trace of sulphur due to the high lime content of the kiln charge. The silicon content is low, since the kiln charge does not contain the excess carbon found in the blast furnace charge, and the silica in the ore is not reduced to silicon.

Because of its comparatively high degree of purity, kiln iron made according to my process may directly be refined into steel in open hearths or in electric furnaces. There is no need to dilute a relatively pure iron with scrap iron, or other iron that is relatively free from carbon, sulphur, silicon and phosphorus, as must be done with blast furnace iron. Steel production at a reasonable price is thus rendered independent of scrap iron supply. The production of any given open hearth or electric furnace is much greater when operated with my kiln iron, particularly when the iron is charged in molten condition, with no cold scrap or the like added. The fuel or power consumption of any open hearth or electric furnace is lowered accordingly, as is also the consumption of slag-forming materials, smaller amounts of impurities being present in the furnace charge and requiring removal. Since the slag of furnaces using my charging metal contains smaller amounts of impurities, the ability of the slag to remove impurities is encumbered to less degree.

The use of my hot kiln iron in an open hearth charge serves to preserve the usual manganese content of the iron ore (which is retained by the kiln iron), for in the relatively short time required to refine the kiln iron in an open hearth smaller quantities of the manganese are lost by oxidation than when blast furnace iron is refined.

From the standpoint of increasing steel capacity, it should be noted that the installation of my combustion iron and cement clinker plant affords increased steel capacity without requiring additions to existing coke plant or open hearth equipment. Hence, the total capital and material expenditures required per ton of steel capacity are reduced to a minimum. The limit, placed upon steel production by the available supply of scrap, is eliminated.

From a geographical standpoint, my kiln process makes iron production possible in localities lacking resources of coking coals. Since fuel oil or natural gas may be used in the process, iron may now be produced in localities remote from the coal fields, but economically accessible to oil and gas fields.

It may be remarked that my process may utilize cheap coal or culm in place of expensive coke. This saving will amount to about 10% of the cost of the charging material, depending, of course, upon the cost differential between the coke and the coal, gas, or oil used in the kiln. If the value of the cement clinker is credited to the cost of production, the cost of my kiln iron will be only about 65% of that of blast furnace iron. The finished kiln iron, if molten, is equivalent to liquid scrap having the desired carbon content, and low sulphur and silicon contents, and is therefore worth at least $2.00 per ton more than blast furnace iron.

The cost of scrap in normal markets is usually greater than the cost of blast furnace hot metal, and hence the cost of steel ingots produced from a 100% charge of molten kiln iron will be materially less than the cost of steel produced by conventional methods. The reduction in the cost of melting, and the higher tonnages secured in the open hearth charged with kiln iron, further reduce the cost of steel ingots. The net effect is that the cost of open hearth ingots is reduced by 30 to 40%, taking into consideration the credit accruing from the production of cement clinker.

The above figures pertain to conventional open hearth practice, where the number of tons of steel produced may vary from 4 to 17 tons per furnace melting hour, depending largely on the furnace size. In duplex plants the corresponding figure ranges from 20 to 50 tons per hour. However, duplex steel is not acceptable for many purposes, such as deep drawing sheet steel, due to the presence in duplex steel of undesirably high amounts of objectionable iron-nitrogen compounds originating from the Bessemer steel used in the production of duplex steel. My kiln iron, on the other hand, when charged hot into an open hearth increases the operating capacity of the hearth by 100 to 300%, thus reaching a capacity comparable to duplex plants, while yielding steel comparable if not superior, to open hearth steel, made from blast furnace iron and scrap.

Within the terms and intent of the appended claims many refinements and variations in the procedure and structure described may be made without departing from the principles of the invention. This application is a continuation-in-part of patent application Serial No. 436,849, filed March 30, 1942, now abandoned.

I claim as my invention:

1. In the method which comprises reducing iron ore in a reducing zone and delivering the reduced metal into a melting zone, the invention herein described which includes firing the melting zone with high temperature oxidizing flames to melt the metal in such zone, leading the products of combustion from the melting zone through the reducing zone, and reducing a substantial part of the carbon dioxide in said products of combustion flowing into said reducing zone by injecting into such products carbon preheated to a temperature for immediate reaction with the carbon dioxide.

2. In the method which comprises reducing iron ore in a reducing zone and delivering the reduced metal into a melting zone, the invention

herein described which includes firing the melting zone with high temperature oxidizing flames to melt the metal in such zone, leading the products of combustion from the melting zone through the reducing zone, and reducing a substantial part of the carbon dioxide in said products of combustion flowing into said reducing zone by injecting into such products carbon particles preheated to incandescence for immediate reaction with the carbon dioxide.

3. In the method which comprises reducing iron ore in a reducing zone and delivering the reduced metal into a melting zone, the invention herein described which includes sintering the reduced metal before delivery into said melting zone and impregnating and coating the sinter with carbon, firing the melting zone with high-temperature oxidizing flames to melt the sintered metal while protected by the carbon against re-oxidation, leading the products of combustion from the melting zone into the reducing zone, and reducing a substantial part of the carbon dioxide in said products of combustion flowing into said reducing zone by injecting into such products carbon particles preheated to incandescence for immediate reaction with the carbon dioxide.

4. In the method which comprises reducing iron ore in a reducing zone and delivering the reduced metal into a melting zone, the invention herein described which includes sintering the reduced metal and spraying carbon thereon to form carbon impregnated and coated nodules before delivery into said melting zone, firing the melting zone with high-temperature flames to melt the metal under the protection of said carbon against re-oxidation, leading products of combustion from the melting zone into the reducing zone, and reducing a substantial part of the carbon dioxide in said products of combustion flowing into said reducing zone by injecting into such products carbon particles preheated to incandescence for immediate reaction with the carbon dioxide.

5. In the method which comprises reducing iron ore in a reducing zone and delivering the reduced metal into a melting zone, the invention herein described which includes sintering the reduced metal and spraying carbon thereon to form carbon impregnated and coated nodules before delivery into said melting zone, firing the melting zone with high-temperature flames to melt the metal under the protection of said carbon against re-oxidation, leading products of combustion from the melting zone into the reducing zone, and reducing a substantial part of the carbon dioxide in said products of combustion flowing into said reducing zone by injecting into such products a hydrocarbon preheated and cracked for the immediate reaction of the hydrogen and carbon contents thereof with the carbon dioxide.

6. The method which comprises reducing iron ore in a reducing zone and delivering the reduced metal into a melting zone, firing the melting zone with high temperature oxidizing flames to melt the metal in such zone, leading the products of combustion from said melting zone through said reducing zone, and reducing a substantial part of the carbon dioxide in said products of combustion flowing into said reducing zone by injecting into such products a hydrocarbon preheated and cracked for the immediate reaction of the hydrogen and carbon contents thereof with said carbon dioxide.

EUGENE S. HARMAN.

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