

April 5, 1960

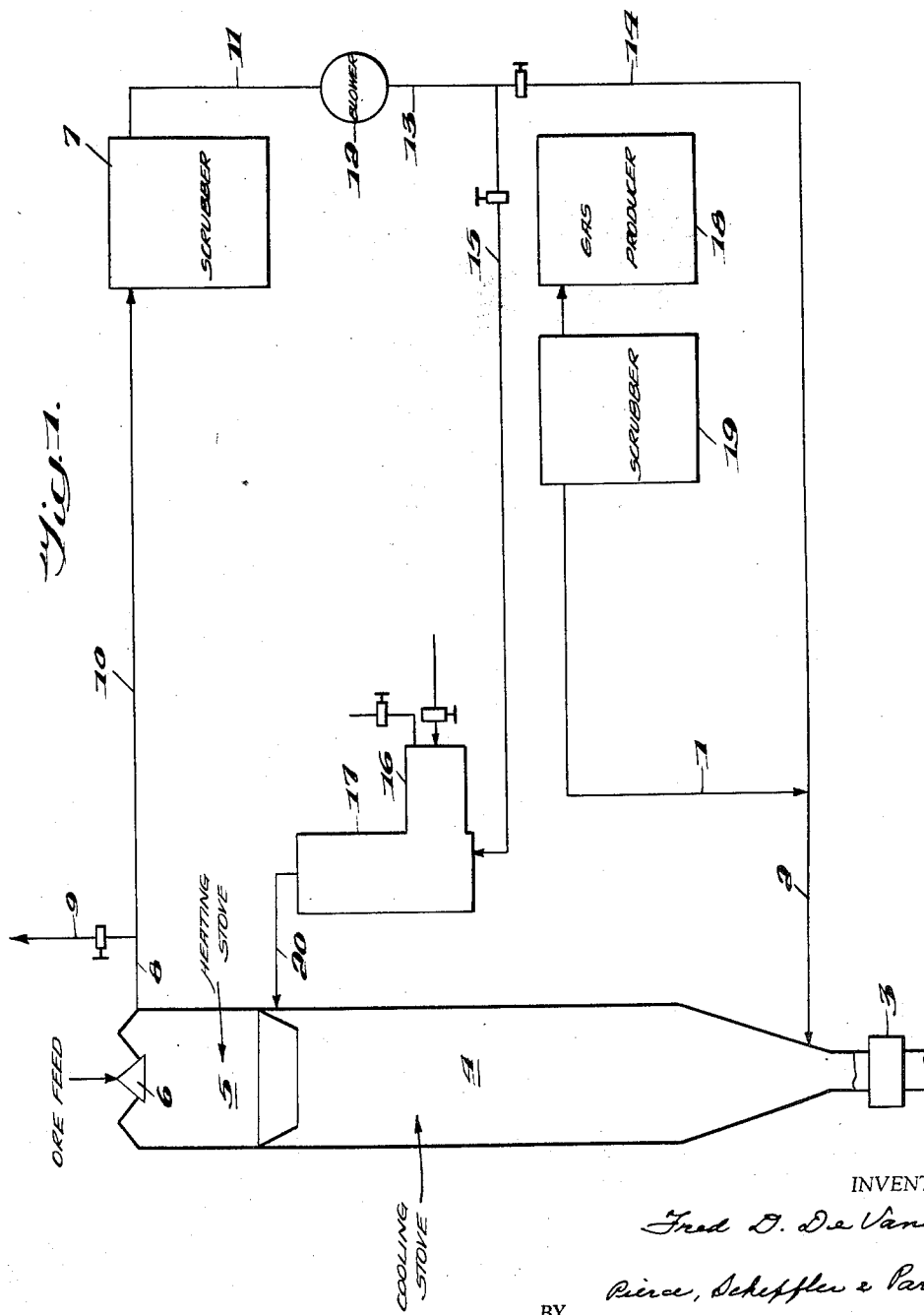
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2,931,720

BENEFICIATION OF LOW-GRADE HEMATITIC ORE MATERIALS

Filed Sept. 25, 1958

4 Sheets-Sheet 1



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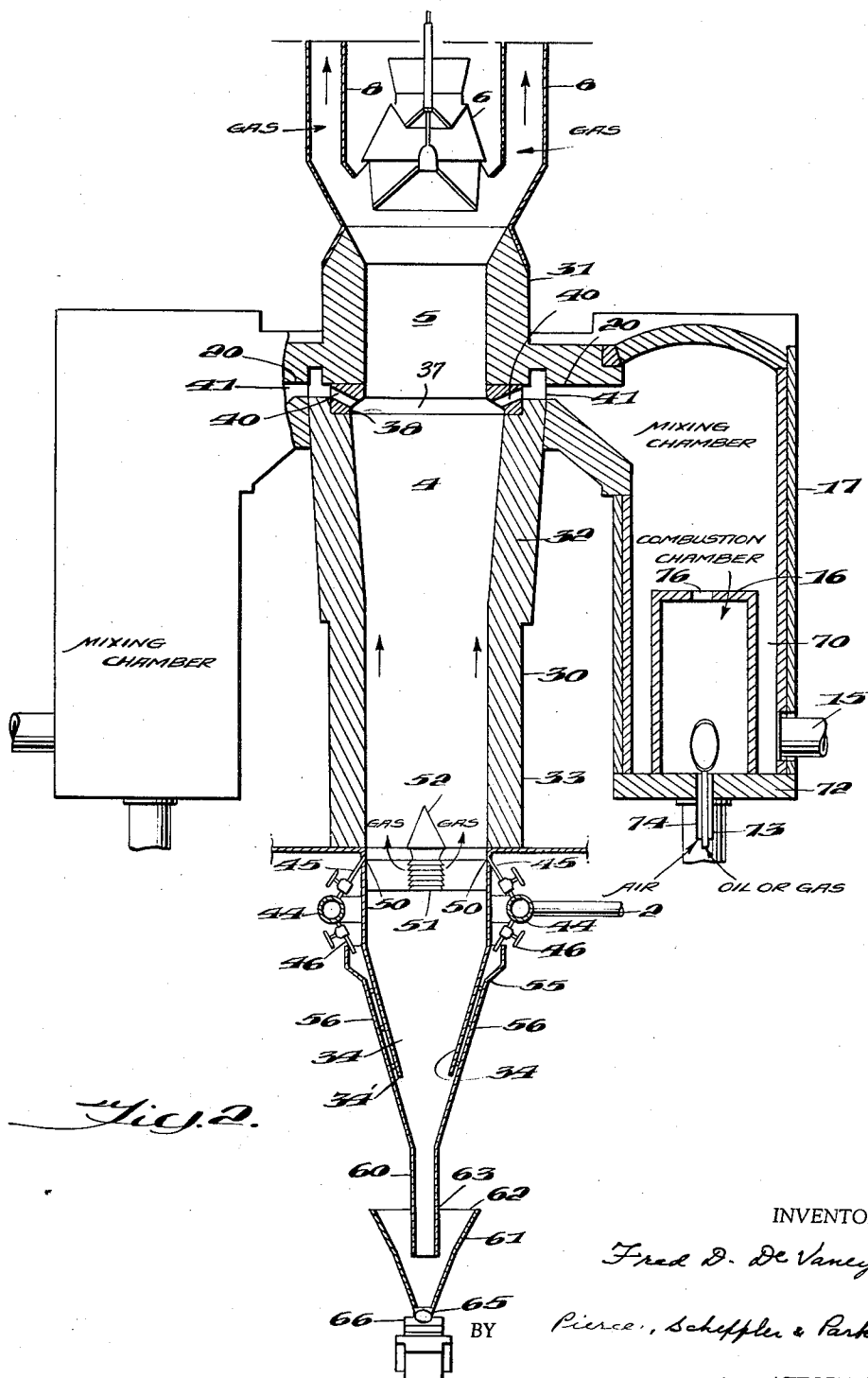
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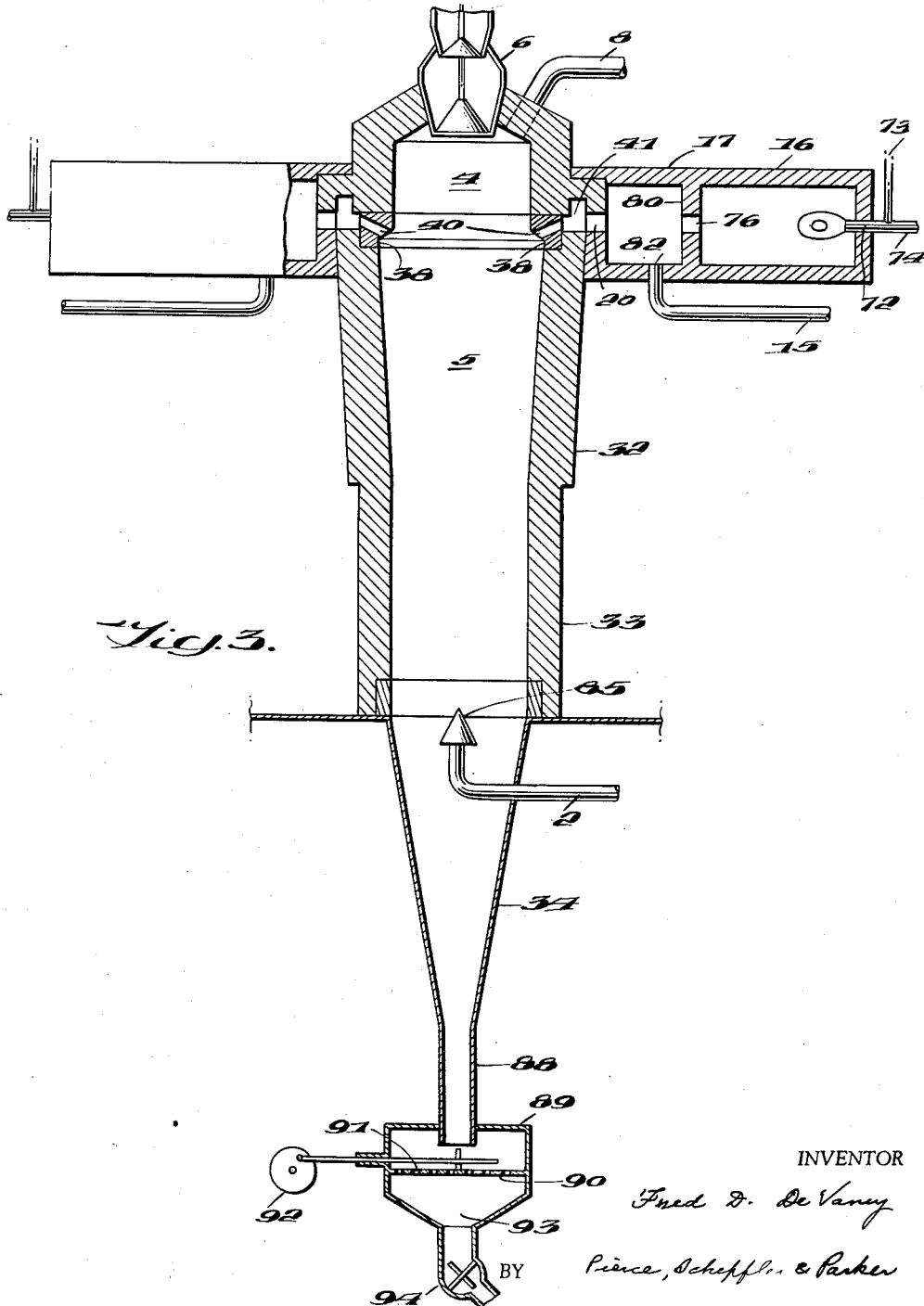
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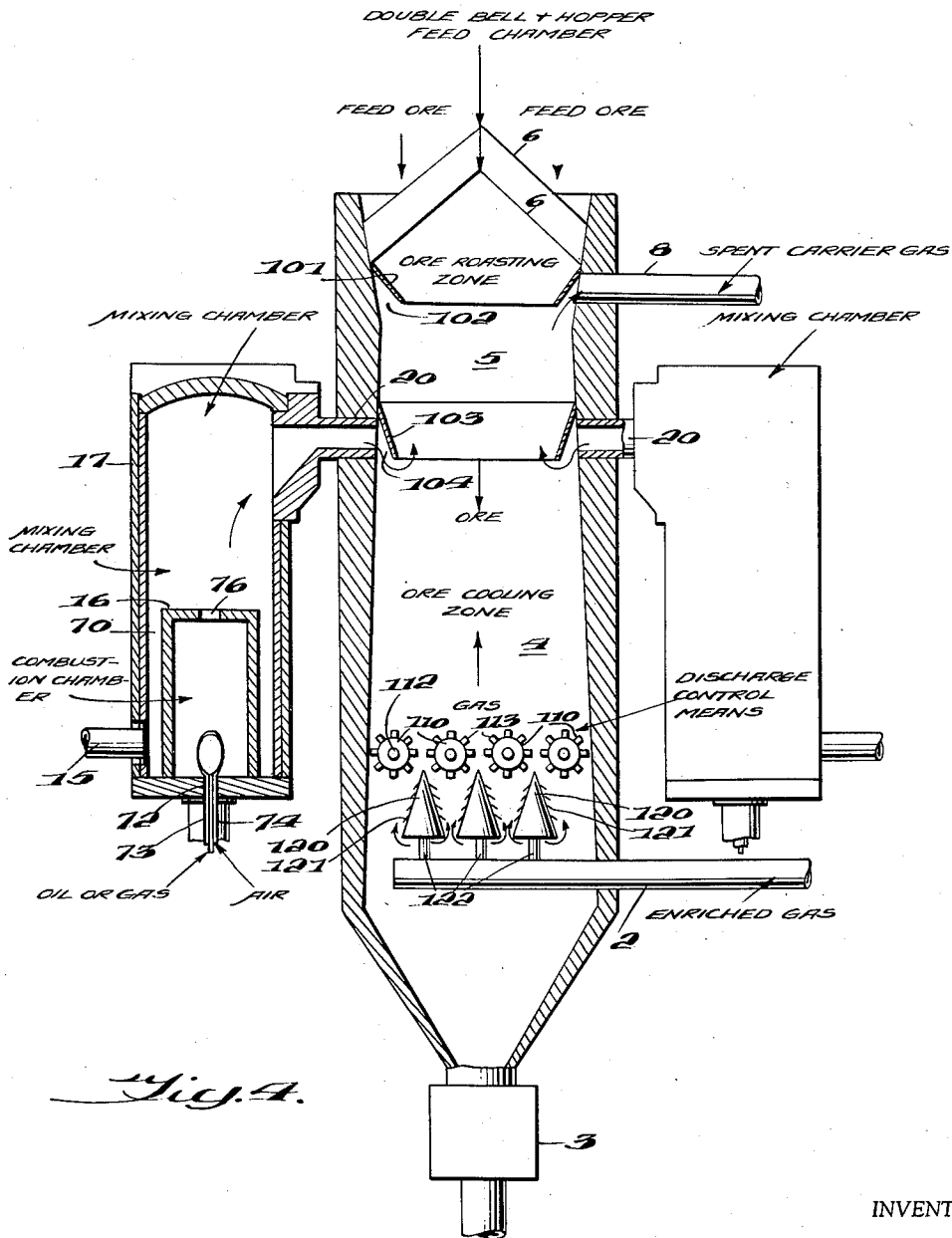
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BENEFICIATION OF LOW-GRADE HEMATITIC ORE MATERIALS

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4 Sheets-Sheet 4



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2,931,720

BENEFICIATION OF LOW-GRADE HEMATITIC ORE MATERIALS

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Application September 25, 1958, Serial No. 763,348

5 Claims. (Cl. 75—35)

The present invention relates to the beneficiation of low-grade iron ore materials in which the iron values largely are present as non-magnetic oxides (and/or hydroxides), e.g. hematite. The invention is particularly concerned with the beneficiation of such iron ore materials in which the non-magnetic iron mineral is too fine grained to be concentrated by gravity methods of concentration such as sink-float, tabling, jigging, cyclone or the like.

Heretofore, a few of the cleanest fine-grained, low-grade, essentially non-magnetic iron ore materials in which the predominant iron mineral was crystalline specularite have been concentrated by a procedure involving froth flotation. With such materials the flotation process has been relatively successful. However, the majority of lean iron ores are non-crystalline and cannot be effectively concentrated by the flotation process.

It has been found that the non-magnetic iron oxide contents of such ore materials can advantageously be concentrated by magnetically roasting the crushed ore under conditions to convert substantially all—or at least, the greater part—of the non-magnetic oxidic iron content thereof to magnetite, grinding to fine particle size, and magnetically separating the magnetic portion from the non-magnetic tailing.

In accordance with the general principles of the present invention, the magnetic roasting step is carried out in a generally vertical, shaft-type furnace through which the coarsely crushed ore material gravitationally descends—continuously, or substantially continuously—as a continuous column in counter flow to a current of a gas mixture having a net reducing effect and maintained at a controlled temperature. Said gas mixture comprises a major amount of carbon dioxide, nitrogen (and other non-oxidizing gaseous components of atmospheric air), and a relatively very small amount of an active reducing gas of the group consisting of hydrogen, carbon monoxide and a mixture of hydrogen and carbon monoxide.

This general process and apparatus for use in carrying it out have been disclosed in U.S. Patents Nos. 2,528,552, 2,528,553, and 2,670,946 to Percy H. Royster.

According to the last two mentioned of these patents, substantially all of the gas (i.e., gas mixture) passed in contact with the column of ore was gas which had been, in unheated state, introduced adjacent the bottom of the latter. After a certain amount of counter-current flow through the ore column—during which it became heated to some extent by heat transfer from the ore—a part of this gas stream was diverted from the column to a spatially separate mixing chamber wherein the diverted gas was mixed with highly heated, essentially neutral, gaseous combustion products whereby its temperature was materially elevated, and the resulting hot mixture was reintroduced into the ore column—at a level substantially above the level of diversion—to mingle with the undiverted part of the gas stream and with the latter to traverse the remainder of the ore column. This procedure, and, more particularly, the disclosed apparatus for

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affecting it, has acquired the designation of “three-high process” and “three-high furnace,” respectively, in that the furnace included three rather well-defined parts serially arranged from top to bottom, namely, an uppermost part 1—called the “heating stove”—in which the ore was heated to the selected high temperature and at least to some extent reduced; an intermediate part 2—called the “middle stove”—in which the hot ore from part 1 was traversed only by the aforesaid undiverted portion of the gas stream and in which reduction was completed (in the event reduction had not already been completed in part 1); and a lowermost part 3—called the “cooling stove”—in which the reduced ore was cooled in contact with the freshly introduced, unheated gas stream.

This “three-high” procedure was open to two main disadvantages. Firstly, the gas diverted at the top of part 3 of the furnace to the aforesaid mixing chamber was inherently dust-laden and the entrained dust created serious problems. Thus, dust carried by the diverted gas tended to settle out in the flues between furnace part 1 and the mixing chamber, in the mixing chamber itself, and in the flues and ports between the mixing chamber and the bottom of part 1 of the reducing furnace, and the build-up of such settled-out dust resulted in uneven flow of gases from and to the furnace. Moreover, dust carried into the mixing chamber tended to clinker when subjected to the highly heated gaseous combustion products, and clinker-formation was distinctly disadvantageous if not actually dangerous. Experience proved that removal of settled-out dust from mixing chambers was a very troublesome procedure. Measures for counteracting these dust problems were only partially successful.

Secondly, the “three-high” procedure had the inescapable disadvantage that the distribution of the gases which traveled through the furnace proper and the amount which was diverted to the mixing chamber varied widely, an exact split being impossible to attain. Such variation in split was due to changing pressures (e.g., changing pressures in the intermediate part 2 of the furnace) and to build-up of settled-out dust in the flues to and from the mixing chamber. For these reasons, the furnace operation was incapable of close control. For example, while the temperature of the gas leaving the mixing chamber could be maintained constant, its volume was not subject to exact control, and hence the total heat input to part 1 of the furnace varied. Accordingly, the temperature of the gas stream passing through part 1 of the furnace fluctuated because of varying percentages of relatively cold undiverted gas mixing with the relatively hot diverted gas.

The process and apparatus of the present invention avoid the above mentioned disadvantages and provide an improved procedure (styled a “two-high” procedure) for magnetically roasting the hereinbefore described ore materials. From the standpoint of structure, in the vertical shaft-type furnace per se of the present invention the above referred-to “middle stove” has been omitted entirely, the “heating stove” being directly followed by the “cooling stove,” and likewise there has been omitted any and all flues for diverting partially heated gas from the top of the “cooling stove” to the mixing chamber. From the standpoint of process the spent gas, after having been cleaned and cooled and otherwise conditioned, is split into two fractional streams one of which is conducted to the mixing chamber (wherein it is heated to a suitable elevated temperature without significant change in composition, for introduction into the reducing furnace at the level of the bottom of the “heating stove”), and the other of which is—after suitable “fortification” with active reducing gas—conducted to the bottom of the “cooling stove.”

Specifically, the procedure according to the present in-

vention is carried out in the following manner: intermediate the top and the bottom of the ore column there is introduced into the column and caused to flow upwardly through the upper part of the column a current of non-oxidizing heating gas mixture different in chemical composition from the spent gas mixture which exits from the top of the ore column but differing from such spent gas in that, as introduced, it is at an elevated temperature of the order of from 800 to 1700° F. Simultaneously, there is introduced into the ore column—at a level adjacent the bottom of the latter—a current of a “cold” (i.e., unheated) reducing gas mixture comprising the gaseous constituents of said spent gas “fortified” with a small addition of active reducing gas (CO, H₂ or a CO—H₂ mixture). This cold gas mixture, in being forced upwardly through the ore column, reduces any non-magnetic iron oxide (not already reduced) to Fe₃O₄, and simultaneously abstracts heat from the ore thereby cooling the latter, and eventually mingles with the current of heating gas mixture (introduced—as stated above—at a level intermediate the top and the bottom of the column) and in admixture with the latter completes the countercurrent passage through the remainder of the ore column and exits at the top as “spent gas.”

Because this “spent gas” still contains some residual active reducing gas values, the process is made essentially cyclic in character, in the following manner: the spent gas, after having been diverted from the top of the furnace, and after having been slightly diminished in volume by venting a few percent to atmosphere—the amount of gas so vented being equivalent to the gain in volume of the gases in the closed system due (a) to formation of water vapor from the moisture contained in the feed ore and (b) to the addition of gaseous products of combustion of fuel in the mixing chamber—is cooled substantially to room temperature or thereabouts and simultaneously cleaned in a scrubber and then is split into two streams of unequal volume. The stream of smaller volume is forced through a mixing chamber wherein it is mixed with a stream of substantially inert, non-oxidizing, high-temperature gas to form the aforesaid “heating gas mixture” which latter is introduced into the ore column adjacent the top and the bottom of the latter. The other, larger-volume, stream is “fortified” by admixing with it a small amount of make-up gas rich in active reducing agent (CO, H₂ or mixture thereof) which small amount is sufficient (1) to compensate in volume for the wasted spent gas and (2) to re-formulate the aforesaid current of “reducing gas mixture” which is introduced—at about room temperature—into the ore column at a level adjacent the bottom of the latter for countercurrent flow therethrough.

The temperature and volume of the aforesaid “heating gas mixture” are so adjusted, and the ratio of said heating gas mixture to a unit volume of crushed ore material is so maintained, that at the level of introduction of said mixture the ore material is heated to from 800 to 1700° F. which temperature level slopes downwardly to an exit temperature of the “spent gas” of about 200° F. (or somewhat higher) by reason of heat exchange between the hot gases and the ore material particles contacted by said gases (and which had been charged to the column at room temperature).

In the process just described, essentially all of the non-magnetic oxidic iron of the charge column is reduced to Fe₃O₄ without, however, further reduction to FeO (or, to metallic iron). Prevention of reduction beyond the magnetite stage, in spite of the high temperature conditions obtaining in the zone of major reduction, is assured by the presence in the gases of a relatively very large amount of carbon dioxide gas or water vapor or a combination of the two. It heretofore had been considered necessary, when reducing with a gas rich in hydrogen, to maintain a relatively low temperature in order to avoid over-reduction beyond the Fe₃O₄ level.

It will be appreciated, from the foregoing description, that the present invention avoids some or all of the above-discussed disadvantages inherent in the operation of a “three-high” reducing furnace of the vertical shaft type. By not diverting partially heated gas from the furnace per se to the mixing chamber one eliminates dust from the mixing chamber and from the flues and ports leading from the latter to the “heating stove” part of the reducing furnace; likewise, one entirely eliminates any and all flues leading from the reducing furnace and, hence, entirely avoids material settling problems relative to such flues. Equally important, by the present invention one attains the ability exactly to control the distribution of the gases to the bottom of the “cooling stove” and to the mixing chamber in any desired proportion: thereby, the optimum amount of gas can be distributed to each point. The resulting exact distribution of gas makes it possible much more uniformly to control the temperatures obtaining in the “heating stove,” both gas temperature and gas volume being subject to close control, and hence to secure efficient roasting.

As will be appreciated, this procedure is best suited to processing a relatively coarse material—e.g., an ore material which has been crushed to 1 inch—in order to maintain good gas flow and to minimize back pressure. Consequently, for best operation the charge material should contain a minimum of minus 10–14 mesh material. If the crusher product is relatively coarse and does not contain appreciably more than 10% of fines (i.e., particles finer than 10–14 mesh), the entire crusher product can be charged to the ore column. If, however, the crusher product contains larger amounts of the fines, it is expedient either (1) to screen out the fines and to magnetically roast them by a different process (e.g. by a fluo-solids procedures) or (2) to agglomerate them by the known techniques such as balling, briquetting or extruding, and to associate the resulting pellets or balled-up masses of fines with the plus 14 mesh fraction being charged to the shaft-type reducing furnace. In this latter connection, I have found that most low-grade starting materials contain an appreciable amount (a few percent) of a clay-like plastic component, and that such component provides a very satisfactory “binder” for the fines, thereby making possible a very much simplified procedure for avoiding an unduly high content of fines in the charge column, as follows. Where the starting material (when crushed) inherently produces a substantial amount of the fines but does not contain an appreciable amount of said clay-like plastic component, I crush the ore to about 1 inch, pass the entire crusher product through a balling drum (e.g. a balling drum such as that described in my U.S. Patent No. 2,831,210) wherein the finer particles agglomerate into small balls or pellets—which pellets give the same net effect as do the coarser pieces of the crusher product—and charge the entire product of the balling drum to the ore column of the reducing furnace. This special procedure practically avoids the presence of fines in the column, making for uniform gas flow with a minimum of back pressure.

On some ores which are essentially rock-like and have little plasticity and, therefore, cannot be balled or extruded, I have found the fines can be mixed with a small amount of cement—in the order of 5% by weight—and then briquetted and allowed to set for approximately 48 hours. At the end of this period these briquetted fines have acquired sufficient strength so that they can be charged along with the natural coarse material and little breakage will take place in the passage of these briquets through the furnace.

It should be appreciated that it is possible to use, as a source of reducing gas, almost any of the common manufactured gases in which CO or H₂ are the principal reducing agents. Natural gas which contains a high percentage of CH₄ must be reformed into CO and H₂ before it can be effectively used. It is preferable, however, to

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use a gas containing some CO rather than all H₂ in order to maintain a favorable CO₂ to CO+H₂ ratio to prevent formation of FeO in the roasting operation. There is also some advantage in having some CO in the reducing gas since the reaction of CO+3Fe₂O₃→2Fe₃O₄+CO₂ is exothermic which tends to maintain furnace temperatures. If only hydrogen gas is available the tendency to over-roast to FeO can be largely minimized by introducing water vapor into the entrant gas to the furnace.

From the standpoint of the apparatus aspect of the present invention, it already has been mentioned that the reducing furnace is of the shaft type. This "shaft" may be circular in cross-section, or it may be square or rectangular in cross-section. For reasons to be discussed hereinafter, it is expedient in many cases so to design the shaft furnace that its upper part is generally circular, while its lower part is rectangular, in cross-section.

The invention will now be described in greater particularity in the following and in connection with the appended drawing, in which

Fig. 1 is a schematic representation, in flow sheet form, of apparatus operable for use in the cyclical reductive roasting process of the invention;

Fig. 2 is a somewhat enlarged vertical sectional view of a reducing roasting furnace according to the invention, showing a particular form of apparatus for use in contributing heat to the reduction process;

Fig. 3 is similar to Fig. 2, and shows a modified form of reducing furnace; and

Fig. 4 is a vertical sectional view of a form of reducing furnace construction embodying principles for insuring uniform descent of the ore column and for uniform distribution of cooling gas across the cross-sectional area of an ore column resident in the lower part of the cooling stove.

In Fig. 1, the reducing furnace per se is a substantially vertical shaft composed of a lower cooling stove 4 and an upper heating stove 5. Heating stove 5 is provided at its top with a double bell-and-hopper feeding means 6 for introducing feed ore into the furnace without loss of gas from the system, while cooling stove 4 is provided at its bottom with a reduced ore product discharge means 3 for positively removing reduced and cooled ore from the reducing furnace, at controllably variable rates, without loss of gas from the system. Discharge means 3 may—as shown—include a conventional star gate, or it may comprise any other equivalent discharging device. At 7 there is schematically represented a wet dust collector-scrubber for cleaning and cooling spent carrier gas exiting from the top of stove 5 by way of spent gas conduit 8. Conduit 8 is provided with a valved vent means 9 for wasting to atmosphere a small (variable) fractional part of the total spent carrier gas, and conduit 10 conducts the residual spent carrier gas to scrubber 7. In scrubber 7 the carrier gas is cooled (e.g., to room temperature, 60° F.) and the dust removed and excess moisture condensed out and expelled together with excess CO₂. The so-conditioned carrier gas is withdrawn from scrubber 7 through conduit 11 by means of blower 12, and by the latter is forced through cold carrier gas conduit 13. Conduit 13 delivers to two branch valved conduits 14 and 15 which split the stream of cold clean carrier gas for delivery to inlet conduit 2 and mixing chamber 17, respectively. A conduit 1 delivers active reducing gas, produced in gas producer 18 and cleaned in scrubber 19, to inlet conduit 2 for commingling therein with carrier gas to provide an enriched gas. At 16 is indicated a burner means operable for burning fuel oil in a controlled amount of air to produce hot gaseous combustion products devoid of free oxygen, for commingling in mixing chamber 17 with clean cold carrier gas delivered to mixing chamber 17 through valved conduit 15. The hot carrier gas-neutral gaseous combustion products mixture produced in chamber 17 is, through conduit 20, introduced into the reducing furnace, at a level adjacent the

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bottom of heating stove 5, and passes upwardly—in association with ascending enriched gas (introduced at the bottom of cooling stove 4) with which latter it commingles—through that part of the total column of ore resident in heating stove 5. In such passage said hot carrier gas-neutral gaseous combustion products mixture gives up a major part of its heat to the ore thereby heating the latter to desired reduction temperature. The active reducing agent component of the enriched gas, for its part, becomes oxidized (to CO₂ or/and H₂O, as the case may be) by reaction with the Fe₂O₃ of the ore, and loses heat (acquired in passage through the ore in stove 4) to the ore in stove 5. The commingled gases exit from the top of the furnace—through conduit 8—as the aforesaid "spent carrier gas" thus completing the gas cycle of the process.

Fig. 2 more specifically illustrates one form of furnace adapted for use in carrying out the process and including particular means for producing the heating gas used for heating the ore to desired temperature for effecting reduction of Fe₂O₃ to Fe₃O₄.

According to this embodiment, the furnace shaft, generally designated 30, is composed of, in series, a generally cylindrical uppermost part 31; an elongated middle part of which the upper portion 32 has the form of an inverted frustum of a cone and of which the lower portion 33 is generally cylindrical and has substantially the same cross-sectional area as that of uppermost part 31 and of the apex end of portion 32; and a generally conical lowermost part 34. The "base" end of the frusto-conical portion 32 has a cross-sectional area larger than that of uppermost part 31 and the junction wall 37 joining the open bottom of part 31 with the base end of portion 32 provides an annular free space 38 between the furnace wall and the periphery of a column of ore resident in the furnace. Parts 31, 32 and 33 are constructed of firebrick backed by heat-insulating material for conserving heat within the space enclosed by them: part 34 suitably is constructed of sheet metal.

Junction wall 37 is provided with a plurality of downwardly directed ports 40, 40 spaced as closely as practicable about the periphery of 37, which ports communicate with an annular chamber 41. Into chamber 41 there discharge a pair of conduits 20, 20 from the pair of mixing chambers 17, 17 to be described below.

Conical lowermost part 34 is provided with particular means for introducing cold enriched gas into the furnace and distributing the same through the cross section of an ore column resident in the furnace. As shown, inlet conduit 2 discharges into an annular bustle pipe 44 from which latter lead a plurality of valved branch conduits 45, 45, 46, 46. Branch conduits 45, 45 communicate between bustle pipe 44 and the peripheral terminal branches 50, 50 of a gas-distributing means 50, 51, 52 centrally disposed within and adjacent the top of lowermost part 34, there being as many branch conduits 45, 45 as there are terminal branches 50, 50 (two each being shown in Fig. 2). In said gas-distributing means, branches 50, 50 extend radially outwardly from a centrally (i.e., axially) disposed multi-louved gas distributor 51 the louvers of which are so arranged as to tend to direct gas under pressure radially outwardly therefrom and to distribute the same with substantial uniformity across the cross section of a column of ore particles resident in the lower part of cooling stove 4. A conical cap piece 52 atop of distributor 51 permits the column of ore to pass smoothly over the distributor. Branches 50, 50 may, if desired, be suitably slotted to permit the discharge of part of the supplied gas therethrough, the residual part passing to the centrally disposed distributor.

In order to ensure the cooling of the walls of the conical part 34 of stove 4, a part of the total gas supplied through conduit 2 may, as shown, be diverted through valved branched conduits 46, 46, to an annular chamber 55 surrounding said conical part adjacent the "base" of

the cone. At its lower part annular chamber 55 merges into a conical vessel 56 the wall of which is spaced from and generally parallel to the wall of conical part 34 thereby providing a gas space therebetween for downward passage of cool gas (from annular chamber 55) over the surface of part 34. Conical vessel 56 extends substantially beneath the lower edge 34' of conical part 34 and terminates in a generally cylindrical discharge tube 60 for delivery of cooled reduced ore from the furnace shaft to a gas-locked product discharge means in communication with the open lower end of said tube. Gas after passing through the space between parts 34 and 56 discharges at the lower edge 34' into the ore column about the periphery of the latter.

Said gas-locked product discharge means includes a hopper 61 provided with a gas-tight cover 62 through a central orifice 63 in which cover tube 60 extends into the interior of the hopper. A slide valve means 65 closes the bottom of the hopper and functions to deliver ore particles, from a constantly maintained supply thereof in the hopper, to belt conveyor means 66 for the forwarding of product to a point of use.

In this embodiment of the invention, the heating of the current of cold carrier gas, provided through valved conduit 15, is effected in an efficient manner. For this purpose, the generally cylindrical combustion chamber 16 has a diameter smaller than, and is axially disposed within, the generally cylindrical mixing chamber 17, the relative sizes of the two chambers being such that an annular space 70 is provided between them, into the lower part of which annular space gas is delivered from 15 to circulate about chamber 16 in passing into mixing chamber 17. A burner 72 is axially disposed in the base of combustion chamber 16, said burner being supplied with metered (or otherwise controlled) amounts of fuel oil and air through valved oil pipe 73 and valved primary air pipe 74, respectively, for producing a supply of highly heated neutral gaseous combustion products. These latter stream through central opening 76 in the top of the combustion chamber and into the main space within mixing chamber 17 for thorough mixing with the carrier gas preheated in passage through annular space 70.

The mixed gases pass through conduit 20 into annular chamber 41.

Only one heating unit has been described above. However, it is preferred that a pair of identical heating units be employed, the same delivering hot mixed gas into annular chamber 41 at opposite sides of the furnace shaft.

The heating units may be constructed somewhat more simply according to the modification illustrated in Fig. 3. According to this modification, in each of the pair of heating units the combustion chamber 16 and the mixing chamber 17 are series portions of a single horizontally disposed chamber separated one from the other by a partition wall 80 provided with a central opening 76 for passage of highly heated gaseous combustion products from combustion space into mixing space. Carrier gas is led into the latter at an opening 82 in the side wall thereof, for thorough mixing with the heating gas.

In this modified form, cold enriched gas, delivered by conduit 2, is discharged into the ore column through a hooded discharge member 85.

Ore gravitating through conical lower part 34 passes into the tubular extension 88 and thence into closed vessel 89. This latter is divided into upper and lower portions by a generally horizontal apertured partition 90 through the apertures of which ore particles pass by the aid of a reciprocatory "pusher" device 91, 92. A supply of the ore particles is maintained in lower part 93, the bottom of which latter is in communication with a star gate 94 of conventional form for discharge of solids.

Fig. 4 illustrates a type of furnace peculiarly well adapted to handle ores in which there may be some diffi-

culty in material flow. In the roasting process the temperatures employed are well below the fusion point and thus no clinkering occurs. However, with certain ores, because of their physical nature and because of the amount of moisture or fines present, it is sometimes desirable to incorporate in the design a series of rotating control shafts whose purpose it is to regulate the descent of the charge. These control shafts serve to break up any consolidated masses that have been formed—through the packing of the material rather than through clinkering of the material—and to insure a uniform descent of the charge through the furnace which at the same time tends to insure a uniform flow of gas up through the shaft of the furnace.

In Fig. 4, as in Fig. 2, but one of the pair of identical heating units has been illustrated. Also as in Fig. 2, the external gas circuit has been omitted as having already been shown in Fig. 1 and described in connection with the latter.

According to this embodiment, the upper portion of the furnace shaft is made circular in cross-section—thus making it possible to utilize the structural advantages inherent in a circular design, and permitting the use of a simple double bell-and-hopper device for feeding the ore—whilst in the middle portion of the furnace shaft a conversion is made from a circular to a rectangular cross-section in order to make possible the inclusion of a series of horizontally disposed, parallel, rotating breaker shafts in the lower part of the cooling chamber or stove 4. Preferably, the conversion is a gradual one, starting from just below the level at which the heating gas is introduced into the furnace and extending to a level above the bank of breaker shafts above mentioned.

The breaker shafts extend across the rectangular part of the shaft and are journaled in bearings which are or may be incorporated into the masonry (brickwork) wall of the shaft, at least one end of each shaft extending exteriorly of the wall of the shaft and being provided at the exposed end with conventional means (not shown), e.g., a connection with a drive rod and crank arm therefor, actuated by a hydraulic cylinder and piston, for oscillating the shaft. In lieu of such oscillating means, there may be used a rotating means including a drive shaft provided with a plurality of driving gears cooperating with driven gears keyed to the exposed ends of the shafts, said drive shaft being rotated by a variable speed motor, preferably of the reversible type. Other conventional means for oscillating or rotating the breaker shafts may be used, it being essential only that said means be adapted to being actuated at a controlled variable rate of speed. Preferably, the actuation of all of the breaker shafts is effected at one side of the furnace and by a single actuating means. The breaker shafts are provided, about that portion of the periphery of each of them which is disposed between the opposite walls of the shaft, with spaced "teeth"—arranged either in rows longitudinally and radially of the shaft or helically about the shaft—or equivalent protuberances for positively augmenting the downward movement of the ore column.

In this embodiment, the cold (or, cool) enriched gas is introduced, by way of conduit 2, into the ore column by means of a plurality of spaced, parallel, louvered inverted trough-like gas distributors disposed adjacent to but below the bank of breaker shafts and between each pair of adjoining breaker shafts. Thereby, the enriched gas serves to cool the breaker shafts, and is distributed uniformly over the cross-section of the ore column.

As is suggested above, a substantially dry ore of relatively coarse size, and containing little "fines," may not require the assistance of the above-described breaker shafts in descending substantially evenly through the furnace, in which event the breaker shafts may be dispensed with: regardless of the exclusion or inclusion of breaker shafts in the organization, the above-described means of

distributing cool enriched gas throughout the cross-section of the ore column constitutes an important feature of this embodiment of the invention.

In Fig. 4, an annular sloping baffle member 101 is incorporated in the masonry wall of the upper stove at a level adjacent to and above conduit 8, which baffle member extends inwardly and downwardly to form, with the shaft wall, an annular gas-collecting space 102 with which the inner end of conduit 8 communicates. Similarly, annular baffle 103 is provided at a level adjacent to and above conduit 20, which baffle extends inwardly and downwardly to form, with the shaft wall, an annular plenum space 104 from which heating gas—supplied through conduit 20—is forced into the ore column.

A plurality (four illustrated in the drawing) of horizontally disposed, spaced parallel breaker shafts 110, 110 extends across the rectangular part of stove 4, to the exposed ends 112, 112 of which are attached conventional means (not shown) for oscillating the breaker shafts. About the periphery of each breaker shaft is disposed an array of spaced "teeth" 113, 113 which—when the breaker shafts are moved—engage the particles of the ore column, loosen the ore and (their primary function) break up any agglomerated masses or chunks of material which may have formed. The spacing between the toothed shafts is such that all particles other than said chunks freely pass between them regardless of whether or not they are being rotated.

Disposed closely beneath breaker shafts 110, 110 and parallel with the latter, are spaced, parallel, inverted trough-like gas distributors (three shown) 120, 120 the sloping sides of which are louvered (as shown at 121) to provide an array of gas inlet means making for extensive distribution of gas across the cross-section of the rectangular part of stove 4. Gas from conduit 2 enters the trough-like gas distributors by way of branch pipes 122, 122 communicating between conduit 2 and the interior of members 120, 120. As is illustrated in Fig. 4, the number of gas distributors 120, 120 is one less than the number of breaker shafts 110, 110, and each gas distributor is positioned between (and just beneath) each pair of adjacent breaker shafts, whereby cool gas is directed onto the breaker shafts and into the loosened ore particles moving past said shafts and is uniformly distributed throughout the cross-section of the ore column.

The process will now be described in further detail and exemplified by the following specific examples.

Example 1

The starting material was a low-grade, siliceous taconite-like ore from the western Mesabi district, Minnesota. The iron mineral was largely hematite, but was too fine grained to be concentrated by float-sink, jigging or cyclone methods of concentration. The ore had the following analysis on a dry basis:

	Percent
Fe ₂ O ₃	55.0
SiO ₂	41.5
Al ₂ O ₃	1.2
As received, moisture content 6%.	

Batch tests indicated that this ore could be satisfactorily reductively roasted after crushing to pass a 1 inch square opening.

The ore was crushed to this size and, without further sizing, was fed to the reductive roasting furnace schematically represented in Fig. 1.

The apparatus had the following dimensions: upper cylindrical stove 5 had an inside diameter of 9 feet. Ore was charged, at 6, at the rate of 25.44 gross (long) tons per hour, at 60° F.

The blower 12 forced 818.0#/minute of cold, clean, spent carrier gas, delivered to it by conduit 11 from the

scrubber 7, through the cold gas conduit 13, this gas having the following composition:

	#/min.	Percent
CO ₂	70.4	8.6
CO	5.7	0.7
H ₂	0.0	0.0
H ₂ O	14.8	1.8
N ₂	727.1	88.9
Total	818.0	

Reducing gas used in this example was made from coke in a slagging type gas producer 18, and, after being scrubbed at 19, was forced through conduit 1, this gas having the following volume and composition:

	#/min.	Percent
CO ₂	2.8	2.9
CO	31.3	32.4
H ₂	0.0	0.0
H ₂ O	1.7	1.8
N ₂	60.8	62.9
Total	96.6	

The re-circulating carrier gas was split into two portions. One portion was diverted into conduit 15 and the other portion into conduit 14. The gas in conduit 14 and conduit 15 had the following composition and volumes:

	Percent	Conduit 14, #/min.	Conduit 15, #/min.
CO ₂	8.6	49.0	21.4
CO	0.7	4.0	1.7
H ₂	0.0	0.0	0.0
H ₂ O	1.8	10.3	4.5
N ₂	88.9	506.3	220.9
Total	100.0	569.6	248.5

The carrier gas from conduit 14 and the reducing gas from conduit 1 commingled in conduit 2 to produce an enriched carrier gas of the following composition and volume:

	#/min.	Percent
CO ₂	51.8	7.8
CO	35.3	5.3
H ₂	0.0	0.0
H ₂ O	12.0	1.8
N ₂	567.0	85.1
Total	666.1	

This "enriched" carrier gas entered the ore column in lower stove 4, where it recovered heat from the ore, and ascended into the upper ore column in upper stove 5. As it entered the latter, it commingled with the gas from mixing chamber 17.

In mixing chamber 17, 248.5#/min. of gas from conduit 15 was mixed with the hot (1000° F.) gaseous combustion products, analyzing

	#/min.	Percent
CO ₂	12.7	21
H ₂ O	4.3	7
N ₂	43.5	72
Total	60.5	

issuing from the combustion chamber 16 appurtenant to the mixing chamber, in which combustion chamber fuel oil had been burned with a carefully controlled amount of air so as to produce a heating gas containing no free oxygen.

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The resulting mixture of gases at the bottom of the ore column in stove 5 had the following volume and composition:

	#/min.	Percent
CO ₂	85.8	8.8
CO.....	37.1	3.8
H ₂	0.0	0.0
N ₂	831.4	85.3
H ₂ O.....	20.8	2.1
	975.1	

In order to keep the system from increasing in pressure, a vent means 9 was provided to keep the system in balance. The minimum top gas wasted to atmosphere was 12.5%, based on a nitrogen balance. This percentage varied somewhat with temperature, moisture content, and CO content.

The remainder of the spent gases passed into a wet dust collector-scrubber 7 where the same was cooled and the excess moisture condensed out and expelled together with the excess CO₂ over and beyond that accumulating in the system for equilibrium while maintaining a nitrogen balance. The volume and composition of these top gases were as follows:

	#/min.	Percent
CO ₂	132.1	12.6
CO.....	6.8	0.6
H ₂	0.0	0.0
N ₂	831.4	79.2
H ₂ O.....	79.8	7.6
	1,050.1	

Of these gases 12.5% were wasted to atmosphere by the vent means (conduit 9) and 87.5% passed to the scrubber (conduit 10).

The heat requirements of the above system were such that 201,000 B.t.u./min. were required to heat the ore from 60° F. to 1,000° F.; 67,000 B.t.u./min. were required to evaporate the moisture to dehydrate the ore; and 10,000 B.t.u./min. were necessary to compensate for radiation losses, for a total of 278,000 B.t.u./min.

The heat sources were: 44,000 B.t.u./min. from the exothermic reaction; 160,000 B.t.u./min. were recovered by the ascending gases; and 74,000 B.t.u./min. were supplied by combustion of fuel oil at the combustion chamber.

The heat losses from the reducing furnace were:

	B.t.u./min.
Radiation.....	10,000
Heat in rejected ore.....	41,000
Heat in top gases of which 67,000 was for dehydration of ore.....	227,000
Total.....	278,000

The ore as fed to the furnace had a temperature of 60° F. In the upper stove 5 of the furnace it was heated to a maximum temperature of 1000°–1600° F., and was cooled to approximately 260° F. in passage through the lower stove 4 of the furnace. Ore was discharged, at 3, at the rate of 23.82 lt./hr. and at the discharge temperature of 260° F.

It is of interest that of the total gas blown into the reducing roasting furnace approximately 65% was introduced, into the ore column, adjacent the reduced ore product discharge 3 to cool the descending (hot) ore down to an exit temperature of approximately 260° F. Even at this relatively low temperature there was some danger of reoxidizing the magnetite formed from the hematite in the roasting operation. Consequently, the

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rate of discharge through 3 was controlled by means of a star gate and the material was discharged under water in a spiral type classifier.

All of the heat to the system was supplied by burning 5 fuel oil in the "dutch oven" combustion chambers 16 within the mixing chambers (one, only, of which has been indicated in the flow sheet). The procedure in burning the fuel oil with no excess of oxygen made it possible to burn the oil completely and effectively and to reduce radiation losses.

The quenched discharged ore from 3 was ground, in a 9 ft. by 12 ft. ball mill to all minus 100 mesh, and then was concentrated magnetically on three, three-drum rotary wet magnetic separators.

Product	Results of Concentration Tests— Example 1			
	Percent Wt.	Percent Iron	Percent Silica	Percent Total Iron
Crude ore.....	100.00	38.72	41.50	100.00
Roasted ore.....	95.45	40.56	43.47	100.00
Magnetic Concentrates.....	56.51	63.44	7.86	92.61
Non-Magnetic Tailing.....	38.94	7.35	-----	7.39

Example 2

In this example, the starting material was ore identical in analysis, and structure, to that used in Example 1.

The apparatus was essentially the same as that used in Example 1, and the ore was charged to it at the same rate of 25.44 gross (long) tons per hour.

The blower 12 forced 869.8#/minute of cold, clean, 35 spent carrier gas through the cold gas conduit 13. This gas had the following amounts and composition:

	#/min.	Percent
CO ₂	74.8	8.6
CO.....	4.3	0.5
H ₂	1.7	0.2
H ₂ O.....	15.7	1.8
N ₂	773.3	88.9
	869.8	

Reducing gas, made in a Wellman-Galusha type of gas producer using coal as a fuel, was forced through conduit 1. This gas had the following composition:

	#/min.	Percent
CO ₂	0.62	3.9
CO.....	4.87	30.4
H ₂	1.89	11.8
H ₂ O.....	0.29	1.8
N ₂	8.35	52.1
	16.02	

The re-circulating carrier gas was split into two parts. One part was diverted to conduit 15 and the residual part into conduit 14. The gas in conduits 14 and 15 had the following composition and volumes:

	Percent	Conduit 14, #/min.	Conduit 15, #/min.
CO ₂	8.6	55.9	25.1
CO.....	0.5	3.3	1.5
H ₂	0.2	1.2	0.6
N ₂	88.9	577.8	259.6
H ₂ O.....	1.8	11.7	5.3
		650.0	292.1

The carrier gas from conduit 14 and the reducing gas 75 from conduit 1 commingled in conduit 2 to produce an

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enriched carrier gas of the following amount and composition:

	#/min.	Percent
CO ₂	56.5	8.5
CO.....	8.2	1.2
H ₂	3.1	0.5
N ₂	586.1	88.0
H ₂ O.....	12.0	1.8
	666.0	

This "enriched" carrier gas entered the ore column in lower stove 4 where it recovered heat from the ore and ascended into the upper ore column (stove 5). As it entered, it commingled with the gas from mixing chamber 17. This mixture of gases had the following amount and composition:

	#/min.	Percent
CO ₂	96.5	9.4
CO.....	9.7	0.9
H ₂	3.7	0.4
N ₂	896.8	87.1
H ₂ O.....	22.3	2.2
	1,029.0	

In order to keep the system from increasing in pressure, the vent 9 was used to keep the system in balance. The minimum top gas waste to atmosphere was 5.8% based on a nitrogen balance. This percentage would vary with temperature, moisture content, and CO content.

The remainder of the spent gases passed into wet dust collector-scrubber 7 where they were cooled and the excess moisture was condensed out and expelled together with the excess CO₂ over and beyond that accumulating in the system for equilibrium while maintaining a nitrogen balance.

The amount and composition of these top gases, prior to venting, were as follows:

	#/min.	Percent
CO ₂	103.9	8.4
CO.....	4.7	0.4
H ₂	1.8	0.1
N ₂	1,029.0	83.2
H ₂ O.....	97.4	7.9
	1,237.2	

Of these gases 7.5% exited through the vent (conduit 9) and 88.4% passed to the scrubber 7 (conduit 10).

The heat requirements of the above system were such that 201,000 B.t.u./min. were required to heat the ore from 60° F. to 1,000° F.; 67,000 B.t.u./min. were required to evaporate the moisture and to dehydrate the ore; and 10,000 B.t.u./min. were needed to meet radiation losses for a total of 278,000 B.t.u./min.

The heat sources were: 31,000 B.t.u./min. from the exothermic reaction; 160,000 B.t.u./min. were recovered by the ascending gases; and 87,000 B.t.u./min. were supplied by combustion of fuel oil at the combustion chamber.

The heat losses from the reducing furnace were:

	B.t.u./min.
Radiation	10,000
Heat in rejected ore	41,000
Heat in top gases of which 67,000 is for dehydration of ore	227,000
	278,000

In this example the "thermal history" of the ore during the treatment was essentially identical with that of the ore in Example 1 above.

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The quenched, discharged ore from 3—23.82 l.t./hr.—was ground in a ball mill to all minus 100 mesh, and was then concentrated magnetically on a series of three three-drum rotary wet magnetic separators.

Product	Results of Concentration Tests— Example 2			
	Percent Wt.	Percent Iron	Percent Silica	Percent Total Iron
Crude Ore.....	100.00	38.46	41.55	100.00
Roasted Ore.....	95.32	40.35	43.59	100.00
Magnetic Concentrates.....	55.29	64.62	6.46	92.90
Non-Magnetic Tailing.....	40.03	6.84		7.10

I claim:

1. Cyclical process of reductively roasting a ferruginous ore material to convert the iron content thereof to magnetite, which comprises establishing a gravitationally descending column of the initially substantially unheated ore material in particulate form, removing roasted ore material from the bottom of the column and adding fresh ore material to be roasted onto the top of the column to maintain the latter's height, introducing into and counter-currently passing through the column from substantially the bottom to the top thereof a reactive gas mixture, initially at about room temperature, containing carbon dioxide, inert gases and a gaseous active reducing agent selected from the group consisting of hydrogen, carbon monoxide and a mixture of hydrogen and carbon monoxide, there being present substantially more carbon dioxide than gaseous active reducing agent, introducing into the column, at a level intermediate the top and the bottom of the same and substantially above the level of introduction of the reactive gas mixture, and countercurrently passing through the remainder of the column a substantially inert, non-oxidizing, heating gas mixture comprising hot gaseous products of the combustion of a fluid fuel in air, said heating gas mixture being introduced at such an elevated temperature and in such a volume as to maintain the ore in a maximum temperature zone in the upper part of the column at a temperature of from about 800° F. to about 1,700° F., cleaning and cooling a substantial part of the spent gas mixture issuing from the top of the charge column, adding to a portion of the same a gas rich in said gaseous active reducing agent in an amount to reconstitute the aforesaid reactive gas mixture in composition and in volume, using the latter as the reactive gas mixture in continuation of the process, burning a fluid carbonaceous fuel in an amount of air just sufficient to convert the combustible content of the fuel to CO₂, mixing with the resulting hot gaseous combustion products a further portion of the cleaned and cooled spent gas mixture in an amount to reconstitute said heating gas mixture, using the resulting hot gas mixture as the substantially inert, non-oxidizing heating gas mixture in continuation of the process, and maintaining the volume of circulating gases substantially constant by diverting a minor part of said spent gas mixture from the cycle.

2. In a cyclical process of reductively roasting an initially substantially non-magnetic iron ore material, whereby to convert the oxidic iron content thereof to magnetite, in a substantially gas-tight shaft type furnace, by counter-current circulation, through at least a part of a gravitationally descending column of the ore material, of a current of a gas mixture containing no free oxygen and consisting essentially of a mixture of relatively inert gases including carbon dioxide and a relatively active reducing gas selected from the group consisting of carbon dioxide, hydrogen and a mixture of carbon dioxide and hydrogen, in which gas mixture the content of carbon dioxide is several times the content of active reducing gas, which current of gas mixture after each passage through the ore material is deficient in active reducing gas and contains moisture vapor and an enhanced content of carbon diox-

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ide and which after each such passage is scrubbed and then is reconstituted as to original volume and composition and is reintroduced into the ore material column in a repetition of the cycle, the improvement which consists in splitting the scrubbed gas mixture into two unequal portions, to the relatively cold larger portion there being added an amount of relatively cold active reducing gas equivalent to that which in the countercurrent passage through the ore column had been oxidized, and to the smaller portion there being added a hot, substantially neutral non-oxidizing gas mixture at a temperature sufficiently elevated to raise the temperature of the resulting heating gas mixture of scrubbed recirculated gas and hot gas mixture to within the range 800-1700° F., and introducing said relatively cold enriched gas portion into the column of ore material adjacent the bottom of the latter for countercurrent passage through said column and introducing said heating gas mixture into said column of ore material at a level intermediate the bottom and the top of said column for countercurrent passage through the upper part only of said column.

3. The improved cyclical process defined in claim 2, in which the active reducing gas is a mixture of hydrogen and carbon dioxide.

4. The improved cyclical process defined in claim 2, in which said enriched gas mixture has the approximate composition:

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	Percent
N ₂ -----	88.0
CO ₂ -----	8.5
CO -----	1.2
H ₂ -----	0.5
H ₂ O -----	1.8

5. The improved cyclical process defined in claim 2, in which the mixture of gases countercurrently passing through the upper part of such ore column has the approximate composition:

	Percent
N ₂ -----	87.1
CO ₂ -----	9.4
CO -----	0.9
H ₂ -----	0.4
H ₂ O -----	2.2

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UNITED STATES PATENT OFFICE
CERTIFICATE OF CORRECTION

Patent No. 2,931,720

April 5, 1960

Fred D. De Vaney

It is hereby certified that error appears in the printed specification of the above numbered patent requiring correction and that the said Letters Patent should read as corrected below.

Column 14, lines 37 and 56, for "non-oxidizing", each occurrence, read -- non-oxidizing --; same column 14, lines 69 and 70, and column 15, line 24, for "dioxide", each occurrence read -- monoxide --.

Signed and sealed this 27th day of September 1960.

(SEAL)

Attest:

KARL H. AXLINE

Attesting Officer

ROBERT C. WATSON
Commissioner of Patents