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DIRECT REDUCTION OF IRON ORE

by

R. R. ROGERS

MINERAL DRESSING AND PROCESS METALLURGY DIVISION

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ABSTRACT

The present status of direct reduction processes for the production of iron from iron ore, is outlined. A number of typical processes of this type are briefly reviewed.

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^{**} Head, Chemical Metallurgy Section, Mineral Dressing and Process Metallurgy Division, Mines Branch, Department of Mines and Technical Surveys, Ottawa, Canada.

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INTRODUCTION

Between 1947 and 1957 the annual production of iron ore in Canada increased from less than 2,000,000 tons to about ten times that amount. By 1980 it is expected to be not far short of 100,000,000 tons. So little is known about the extent of the iron ore deposits in this country that any estimate would be meaningless. It is known that one of the ore formations extends in an almost continuous belt from 150 miles north of the Gulf of St. Lawrence to the northerly tip of the west coast of Ungava Bay. It seems obvious that Canada's future will depend to a considerable extent on the successful development of her iron industry.

At the present time the ferrous metallurgists in every country are carefully scrutinizing the methods of producing iron which are now in use, and comparing them with the numerous new methods which are being developed. They are aware that the country which possesses the strongest iron industry will have a great advantage in the years to come.

The blast furnace method of producing iron is by far the most important at the present time because it enables large

quantities of the metal to be produced at a comparatively low price. However, it is becoming increasingly evident that this method has certain important disadvantages. The cost of a modern blast furnace and its auxiliaries is tremendous (in the order of \$40,000,000 without accompanying coke ovens, or \$70,000,000 with the ovens) and there is comparatively little flexibility in its operation. In many large and important areas good coking coal is not available at a reasonable price, and in other areas the market is not great enough to utilize the large amount of iron produced in a single blast furnace. Some authorities believe that, in spite of these handicaps, the blast furnace will overcome all serious competition for many years to come because of the important advances which are being made in production methods. They point out that a blast furnace built to-day can produce about 67% more iron than a similar-sized furnace could have produced in 1940. Other authorities believe that few more blast furnaces will be built and that additional requirements of iron will be met by the use of other methods of reducing the ore.

Perhaps the most realistic prediction is that the number of blast furnaces will be increased at a sufficiently high rate to take care of the needs of those large industrial areas which are located within a reasonable distance of adequate coal supplies. The exact nature of the reduction process used in each of the remaining areas will be determined by the conditions existing in that area, such as its location, the type of ore most readily available, and the presence of oil, gas, coal or electric power in the vicinity.

Among the processes under serious consideration at present are those that take place by direct reduction. These will be

discussed briefly in this present article. Unfortunately it will be impossible to refer to every one of the existing processes of this type, in view of the lack of space and the fact that in some cases the details have not been disclosed. Another type of process of particular interest in Canada, where power is unusually cheap, is that in which the iron ore is fed directly into an electric smelting furnace and the product is pig iron.

Direct reduction is said to take place when iron ore in the solid state is converted to metallic iron in the same state. The great majority of the direct reduction processes belong to one of the following types:

fluidized bed

rotary kiln

vertical shaft or retort

Some of them are continuously operated, others are batch processes. Some have been operated for a considerable length of time in commercial-size equipment, others have only reached the pilot plant stage. It should be emphasized that the optimum operating conditions for each process may vary considerably, depending on the nature of the ore.

H-IRON PROCESS (fluidized bed, batch)

This is an American process for the reduction of dry, finely-divided, high grade iron ore by hydrogen of at least 95% purity. The comparatively low operating temperature of about 482°C (900°F) ensures that the particles of charge and product do not adhere together, and that the reduction takes place in the steps $\text{Fe}_2\text{O}_3 \rightarrow \text{Fe}_3\text{O}_4 \rightarrow \text{Fe}$, avoiding the slow reaction $\text{FeO} \rightarrow \text{Fe}$. It also

makes possible the use of a comparatively inexpensive material of construction. The fairly high pressure of 400 psi is advantageous because of the higher rate of reaction, the lower capital cost due to the smaller equipment required, and other reasons.

Hematite and magnetite ores can be used equally well in this process. An ore of high iron content should be used since none of the gangue is removed during the reduction. Comparatively few natural ores are sufficiently pure for use without beneficiation. Possibly the mineral dressing concentrates of a number of other ores would be satisfactory. By-product iron oxide from the leaching or roasting of nonferrous ores or concentrates would be suitable feed material for this process if it were not for the comparatively high content of residual elements such as copper or nickel. Only part of the iron oxide in the ore is reduced, and the product may be made to contain more than 90% Fe or as little as 65% Fe, each grade having its own special uses.

The H-Iron process is at present in the pilot plant stage.

MADARAS PROCESS (vertical retort, batch)

This is an American process in which the iron ore, in small lumps or pellets, is reduced by a 3:1 mixture of hydrogen and carbon monoxide, which is produced from natural gas. The ore is preheated for about 5 hours, and then the gas is introduced at a temperature of about 982°C (1800°F). The reduction requires two or more hours. The gas enters the retort at 30 psi and is pulsed 21 times per minute by opening and closing a valve in the discharge pipe.

The process is still in the pilot plant stage.

WIBERG-SODERFORS PROCESS
(vertical shaft, continuous)

This was first operated on an industrial scale in Sweden in 1932, but was not run continuously until 1941. Hematites such as the Canadian Steep Rock ore, and certain magnetites, may be used. At Soderfors a magnetite sinter containing about 62% Fe is charged. Rich concentrates in the form of pellets also have given satisfactory results.

The reducing gas consists of carbon monoxide and hydrogen in the proportions of about 75:21, although this ratio is not particularly critical. The reduction with carbon monoxide is exothermal and that with hydrogen is endothermal, and the proportions of these gases are chosen so that the heat produced in the first reaction just compensates for that absorbed in the second reaction plus the heat losses.

A portion of the spent gas, together with some water, is passed through a carburettor containing coke, which has been heated electrically to about 1100°C (2012°F), thereby producing an additional quantity of gas suitable as a reducing agent. This gas is passed through a filter consisting of dolomite lumps to remove the sulphur, and then enters the reaction chamber at about 945°C (1733°F).

According to the most recent information available the product contains 80% total Fe and 70% metallic Fe, corresponding to a reduction of 87%. Each unit has a daily output of 27 tons.

STELLING PROCESS
(fluidized bed, continuous)

This has been developed to the pilot plant stage in Sweden. The reduction takes place by means of carbon monoxide only. Hematite concentrates are prereduced to ferrous oxide (FeO) at about 750°C (1382°F). The latter is introduced into a fluidized bed having cementite (FeC_3) as the main constituent and a temperature of about 600°C (1112°F). Reduction here takes place by means of the carbon monoxide which passes through the bed. When the ratio between the cementite and ferrous oxide has reached the desired value the mixture of materials is removed. On heating at 750°C (1382°F) or higher the cementite and ferrous oxide react to form metallic iron. The spent reducing gas is passed through a carburettor to increase its carbon monoxide content, and then is re-used. If the original ore is magnetite a preliminary oxidation to hematite is highly desirable.

The following advantages are claimed for this process:

- (1) the capital cost is comparatively low
- (2) no sticking takes place during the operation
- (3) cementite is resistant to air oxidation and is not pyrophoric
- (4) the operation takes place at atmospheric pressure
- (5) the production of carbon monoxide is less complicated than that of hydrogen
- (6) the heat economy is good, being at least equal to that in the Wiberg-Soderfors method.

R-N PROCESS
(rotary kiln, continuous)

This has been operated in the U.S.A. on a pilot plant scale only. The ore, solid carbonaceous material (coke breeze,

anthracite fines, etc.) and limestone (or dolomitic limestone) are fed into the rotary kiln. The amount of carbonaceous material fed is several times the actual requirement, resulting in the recycling of 75% of this material. The kiln may be fired with gas or oil.

As the ore travels through the kiln, counter-current to the heat flow, it is dried, preheated and reduced. The reduction always takes place at a temperature below the melting point of substantially all of the constituents of the charge, including the iron. The temperature range in the kiln varies according to the nature of the ore, a typical range being 982 to 1093°C (1800 to 2000°F).

The materials discharged from the kiln are cooled and separated by screening. Then they are separated further by gravity or magnetic means. The excess carbon is cleaned, dewatered, and returned to the kiln for re-use. The magnetic portion is freed from gangue by grinding and further magnetic and gravity procedures. Two different products are obtained:

	(1)	(2)
Total iron %	> 95	85
Metallic iron %	> 90	> 70
Silica %		< 8

These are briquetted to provide feed for the open hearth or electric furnace in the first case, and for the blast furnace in the second case. Sometimes it is desirable to combine these two products prior to briquetting.

It is claimed that both magnetic and non-magnetic ores, having a wide range of iron content, may be treated by this process. Sulphur in the final briquette can be expected to be in the order of

0.02 to 0.05% regardless of the sulphur content of the ore or fuel used. Phosphorus rejection from the ore of greater than 90% has been experienced in the case of many ores.

KALLING PROCESS
(rotary kiln, continuous)

This has been operated on a pilot plant scale in Sweden, with a view to producing cheap iron. Purity of product was a secondary consideration. It was believed that former direct reduction experiments in rotary kilns had met with only limited success primarily because of difficulties in preventing the ore from balling in the furnace, or from forming rings on the walls in the hottest zone. The latter seemed to be the most difficult to avoid.

There is a very narrow margin between the temperature required for a sufficiently rapid and complete reduction of the ore, and that at which sticking of the ore occurs. In the Kalling process the charge usually has consisted of finely crushed sinter and coke breeze, the latter being considerably in excess of that required for reduction of the ore. The air required for the combustion of the coke, and of the carbon monoxide formed during the reduction, is introduced through a central pipe inserted in the furnace, and passes through openings in the pipe in the reduction zone. Because of this arrangement the temperature of the charge is kept fairly close to that of the furnace wall and the risk of sticking is greatly decreased. The reduction temperature usually has been in the range 1050 to 1100°C (1922 to 2012°F). Magnetite ores are in general preferred to hematite ores as they do not disintegrate so much during reduction. Good results are more difficult to obtain if the melting point of the gangue is too low. Reduction of 85 to 95% has been obtained, the

product containing an average of about one per cent of carbon. A large proportion of this carbon may be removed by fine grinding and magnetic separation. The phosphorus content of the product is generally somewhat higher than that of the ore, and the sulphur absorption from the coke is considerable. Important advantages of the process are the low consumption of fuel and the fact that cheap grades of reductant can be used. The method is not suitable for the reduction of finely divided ores.

STRATEGIC-JUDY PROCESS
(rotary kiln, continuous)

This has been operated on a pilot plant scale in Canada. It is claimed that fines, off-grade ores, and ores contaminated with titania or similar deleterious materials, as well as standard iron ores, may be treated. The ore with essential fluxes and carbonaceous reductant is fed into a direct-fired rotary kiln. Emphasis is placed on the speed of through-put of a free-flowing system of solid particles with a discharge temperature of about 1100-1300°C (2012-2372°F), rather than on complete reduction. The maximum temperature of the kiln operation is determined by the point at which a free-flowing product is produced without ring formation in the kiln. The product from the kiln is transported to an electric smelting furnace in which reduction of the ore is completed.

It is claimed that agglomeration, briquetting, sintering, pelletizing and nodulizing of the ore are unnecessary; also that low grade ores can be used without primary beneficiation. Ores having an iron content as low as 25% can be economically treated. Less power is required for the electric smelting when the furnace feed has

been partially prereduced in the kiln.

KRUPP-RENN PROCESS
(rotary kiln, continuous)

This has been in operation on a commercial scale in Germany and elsewhere since 1939. It has been particularly successful in treating low grade ores, even though they may contain a considerable amount of silica. The final slag usually contains 55-65% SiO_2 . Ores containing higher than 50% Fe are diluted with slag from the process. Even fairly low grade fuels can be used successfully for the heating and reduction.

As the mixture of ore and solid carbonaceous material proceeds down the kiln, its temperature gradually rises until reduction commences at about 600°C (1112°F). When a temperature of about 1100°C (2012°F) has been reached, additional heat is introduced to achieve a final temperature as high as 1250°C (2282°F). The charge remains in this zone for several hours while the iron particles are welded together to form "lupen", which remain distributed in the pasty slag and are discharged with it. The "lupen" lumps are separated from the slag magnetically.

Unless ore and fuel of low phosphorus and sulphur contents are used, the resulting iron will be undesirably high in these elements. The manganese content of the ore goes mainly into the slag. The titanium and silicon in the ore are not reduced.

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It should be pointed out that the products of all these direct reduction processes must be given further treatment, as in open hearth or electric smelting furnaces or even the blast furnace.

The number of direct reduction processes proposed, and the caution with which the industry is viewing them, are evidence of the efficiency of a properly operated blast furnace for iron making. However, its high capital cost and its need for metallurgical grade coke are disadvantages which are going to increase rather than decrease in the future. It seems certain that processes designed to use iron concentrates or fines, and reductants other than coke, will become of increasing importance with time, and that the process used in any particular case will depend more on local conditions than on any overall metallurgical advantage enjoyed by the process itself.

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