

Direct Iron Ore Reduction for Asia

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THE increasing shortage of high grade basic materials for, and the high investment cost of, the conventional blast furnace are responsible for the trend in iron ore reduction technology towards direct processes.

In Western Europe the low or short shaft furnace has attracted a considerable amount of development effort while very little research work was devoted there to the fluidised bed technique. On the other hand, the American situation is characterised by a multitude of fluidised bed processes to the virtual exclusion of shaft furnace techniques.

Apart from these main trends of development, numerous other processes and promising ideas are tested in pilot or laboratory plants and some have attained almost the stage of commercial utilisation.

The technical situation is thus characterised by a great variety and uncertainty. Long range planning is thereby rendered exceedingly complex especially under the pressure of economic conditions which enforce, as is the case in Asia, the strictest economy of investment and operation and should be based on a realistic assessment, as is attempted here, of processes which are well proved in commercial application for a considerable time.

Along with these principal processes which are appraised technically and economically and which should command first consideration by planning authorities, significant associated processes are mentioned in accordance with their technical features. Active development projects conclude the listing.

Technical process assessment

Commercial processes: Apart from the conventional blast furnace, four main groups of processes are used commercially for the reduction of iron ore: Electric Processes, Moving Bed, Rotary Kiln and Fixed Bed Process: the order in which they are mentioned is a measure of their general readiness for reliable commercial operation.

A common feature of all these processes is that their unit production is relatively small compared with the blast furnace. This has the technical consequence that multi-unit installations are necessary to achieve large scale production. On the other

hand, there is a case for smaller scale production equipment, especially where home consumption is small, or where the export of a higher quality material is economically and sociologically interesting and more attractive than the export of the unbeneficiated basic material, i.e., the ore.

The direct reduction processes, therefore, offer to Asian countries the advantages, firstly, of a smaller investment since the smaller unit production adjusts itself readily to the often lower level of demand, and, secondly, of a generally wider range of acceptable raw materials, compared with the blast furnace-coke oven combination. These processes thus overcome the supply shortage of high grade coking coal as it prevails in most Asian countries.

Electric Reduction process: This process¹ is well established and works reliably with a wide variety of materials. The heat necessary to achieve reduction is derived from an electric current while the reducing agent is still carbon. The electric current finds its way through the charge, so that for heating, its resistance is decisive rather than radiation, the heat being generated by resistance heating in the ore particles and arc-heating between them. Off-gases are mostly rich. Electric power ranges up to 17,000 kVA, iron output up to 250 t/day which can be increased to 5,000 t/day.

The classical representative is the *Tysland-Hole furnace*, represented e.g. by Lectromelt Furnace, U.S.A., Demag Electrometallurgie GmbH, Duisburg, West Germany, and Elektro-Chemisk, Oslo, Norway, in which lump ores or sinters are reduced to molten pig iron with high or low grade carbonaceous fuel such as coke breeze, lignite, anthracite, etc.² If coke is used, 50 per cent of the fuel can be coke breeze. High ash content presents no great difficulty.

The iron ore should not contain too many fines. Ordinary lump ores or sintered concentrates with, say, 55-56 per cent Fe, are preferred raw materials. The ore, together with limestone, is admitted to the melting chamber through holes in the roof through which three electrodes protrude downwards.

The end product is molten iron suitable for the open hearth or the electric furnace. The iron content is 95 per cent, with C normally around 4 per cent. Sulphur can be removed but 90 per cent of the phosphorus remains with the iron. Thus, P should be low in the raw materials. Slag is drawn off separately.

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In this conventional form the off-gases contact the burden for so short a time that very little pre-reduction is achieved and it is only recently that earlier proposals have been realised which allow a better utilisation of the off-gases by passing them over the charge in a special pre-processing step.

The Electro-Chemisk³ and the more recently introduced *Strategic-Udy*⁴ processes meet exactly these shortcomings by combining the electric reduction furnace with a rotary kiln which accepts the ore with fluxing material and carbonaceous reducing materials such as coke, anthracite or even bituminous coal. As the charge passes through the kiln it is, in the Strategic-Udy design, heated to about 900 to 1,200°C to ensure free flowing without ring formation. The combustion of the furnace off-gas can assist in the heat generation from the admixed primary solid fuel. The hot charge enters the reducing zone of the furnace where a short arc is maintained essentially in the slag layer so that a controlled reduction is achieved. The molten slag overflows, and the liquid metal can be cast into pigs or, if necessary, processed further for refining or alloying in another electric furnace. Removal of selected material should be possible at this stage of the process as well as in the earlier pre-reduction stage.

No specified preparation of the ore is required for this process except, as is the case also for the Krupp-Renn process, fine granulation of the ore which can be of low grade quality. Naturally, both heat economy and capacity suffer as the Fe content of the charge becomes smaller. A control of the carbon content of the iron appears feasible so that the steel process can be shortened and simplified. The unit production appears to be of the order of 50 t/day.

The last mentioned process is represented by the Strategic Materials Corporation in Buffalo, New York, U.S.A. and the Koppers Company. A pilot plant is in operation at Niagara Falls, Ontario, producing 25-30 t/day of liquid iron.

Other designs achieve added advantages such as Electro-Chemisk by rotating the melting vessel under the electrode³ and Demag extends the operation to non-ferrous materials⁵ and utilises the off-gas still further⁶. A further development of the earlier Lubatti furnace⁷ is constituted by a duplex method producing directly steel⁸.

With its newer modifications, especially with a preheating and pre-reduction stage, electric reduction processes are technically reliable, accept a wide range of raw materials with moderately low grade characteristics but they depend entirely on the availability and price of electric power. In case of excess electric power, say in the early stages of electrification where the load is low and the network comparatively restricted, they can be used with advantage in connection with larger hydro-electric schemes. Their cooperation with thermal power stations is advisable if low grade fuels with

suitable characteristics are available but investment cost for these schemes would inevitably be high.

Associated processes

In an early stage of development is the Jensen process⁹, also called Norwegian H-Iron process, which shows some of the characteristics of the electric furnace in combination with a shaft furnace. The ore is reduced by a gas mixture of H₂ and CO passing through the shaft upwards while the burden, sliding down, is heated to about 900°C by virtue of its electrical resistance. The current is transferred to the burden with the help of one ring shaped electrode cooperating with two grid-type electrodes traversing the shaft above and below the ring electrode. The off-gas is liberated from its CO₂ and H₂O content and is recirculated to the shaft. Admixing of preheated—900°C—fresh reducing gas facilitates the heating action by the electric current.

The process produces sponge iron of which a part has to be returned to the shaft in order to equalise the electric conductivity of the burden so that overheating and presumably sticking is avoided.

The process has not been applied to commercial operation. The mode of heating the burden appears to be problematical.

The *Lurgi-Galluser process*¹⁰ which provides for electric induction heating of the burden while moving down through a shaft furnace and being reduced by a gas predominantly composed of H₂ and CO flowing upwards, is also in an early stage of development. Ore pellets are charged from the top, the reducing gas enters the shaft cold and is heated by the reduced pellets. In a region of high burden temperature methane is injected which dissociates on impinging upon the hot charge. The off-gas is dried and scrubbed of CO₂ before it is recycled.

It is claimed that methane can be replaced by solid fuels which would be mixed to the ore. The reducing gas would thus be assisted by the reaction H₂O with carbon in the solid fuel forming CO and H₂.

The firm of Charmilles in Geneva, Switzerland, sponsors this development but no commercial use of the process has become known.

In the order of industrial importance, moving shaft processes are to be mentioned next, the first among them being:

The Wiberg Process: This process¹¹ is based on a patent by Martin Wiberg (1918) and has since been developed and perfected.

Its basic technical features are the following: Iron ore is reduced by a gas mixture of CO and H₂ in a ratio of 3:1 to sponge iron at temperatures below its melting point, mostly in the form of pellets which move downward through a shaft furnace. The gas entering the moving bed at the bottom with a temperature of close to 1,000°C is to a large extent withdrawn in the upper part, and after replenishing and reforming with electricity in a carburetor and after sulphur removal, is recirculated. The rest of the gas which was not withdrawn, burns in the uppermost section of the

shaft, preheating the ore there and perhaps burning the fresh pellets. The process accepts also lump ore, but the ore has to be crush resistant and rich—either naturally or beneficiated since the gaseous reduction leaves the gangue in the end product. At present, magnetic concentration of the ore—up to 66-69 per cent is used. The end product could be further upgraded.

The advantages of the process are the clean product which can be fed e.g. to an electric furnace. The operational temperature is relatively low (900°C) so that generally sticking is avoided. The process could be adopted to work with a gaseous material or liquid fuel gasification. Sulphur is not a problem here.

On the other hand, reduction is slow and incomplete (90 per cent) which could be a drawback if used as artificial scrap. The process is coal (or coke) sensitive, and methods of concentration or pelletising of the ore must be tried before operation since different types of ores behave quite differently in this respect. Also, the shaft type operation limits the height, while sliding and discharge considerations limit the diameter of the furnace. Commercially, this appears to hold the size of a Wiberg furnace to a limit of about 50-100 t/day capacity.

The oldest plant—10,000 t/year—has been operating at Soderfors in Sweden for fifteen years. Three bigger furnaces of 20-25,000 capacity are in regular operation. The largest plant, which recently commenced production, is said to produce 30,000 tons per year.

The rights in the process are held by Stora Kopparbergs Falun, Sweden.

For any adoption of the process electrical power must be available and should be economical, and the raw materials must be tested in an actual Wiberg shaft furnace before an installation could be considered in greater detail.

Associated process: Within the group of moving bed processes the example of the blast furnace has inspired attempts to reduce investment by scaling down its dimensions and thus to devise a small, low shaft furnace with otherwise conventional features. The best known installation of this type¹² is located at Calbe in East Germany which comprises ten furnaces utilising extremely low grade ore and local lignite.

Each furnace has an average yield of 70 to 80 t/day pig iron derived from an ore with sometimes only 19-22 per cent Fe. The coke is made from brown coal, i.e., lignite, and is not porous but has a highly volatile content. It is processed in a special briquetting plant and its performance there as well as in the furnace depends critically on the quality of the original coal seam. Since this changes considerably, difficulties are often encountered varying from a severe variation of the pig iron analysis to the gradual subsiding of the reduction process. Air is preheated to 700°C.

Due to the narrow limits of operation the process does not allow wide application although it represents no mean technical achievement. Its example has

spurred the recognition of the low shaft furnace as a tool for speedy and cheap industrialisation of countries in the earlier stages of technical development, such as India and particularly China. In India, the most recent installation of an experimental low shaft furnace in Jamshedpur will undoubtedly add to the advancement of knowledge of iron ore reduction in a scientific and unbiased manner.

A furnace of apparently similar conception works in *Amberg*, West Germany. It can be called a compromise between a hot blown low shaft and a conventional blast furnace, and has a shaft of about 20-25' height with 9-10' diameter. It produces 100 t/day of pig iron in regular operation from a charge of ore of about 10 mm to 20, even to 40 mm dia. A large variety of fuels are acceptable varying from coke to lignite, but it should not contain too many fines, though 5 mm sizes are still possible.

The installation is conventional: off-gas is cooled and cleaned, and drives gas engined air compressors. Air is preheated in recuperators.

The furnace—the only one of its type—is built by Dinger Werke, Zweibrueken, and owned by the *Luitpold Huette*.

The experimental low shaft project¹³ at *Ougree* in Belgium which merits mentioning, is an undertaking of the OEEC—an organisation in Western Europe. It has greatly added to the knowledge of low shaft furnace operation. One small scale industrial installation is said to be built in the Philippines.

The main attraction of the low shaft furnace, i.e., its reduced demand for physical resistance of the burden, becomes particularly evident if augmented by oxygenated air blast which reduces the amount of the chemically inactive nitrogen in the air and allows high temperatures to be obtained.

The protagonist of the basic conception is Prof. Dr. Durrer of the *Gesellschaft der Ludw. von Roll Eisenwerk A.G.* in Switzerland. One low shaft furnace is installed in *Gerlafingen*¹⁴ and operates satisfactorily. Other experimental installations have, however, not attained industrial operation sustained over extended periods.

The conventional blast furnace consumes per ton of coke 3.85 t of air of which 900 kg of oxygen are utilised while 2,950 kg of nitrogen pass without active participation. Oxygen added to the blast reduces the amount of heat transported away from the reduction zone, and of gas needed per unit of output. Process time can be reduced further by charging the solid materials in smaller sizes, or more intimately in contact with each other, say in the form of briquettes. The process produces liquid pig iron.

A consequence of the smaller amount of sensible heat transferred by the smaller quantities of the existing gases to the burden allows to shorten the oxygenated shaft furnace and thus to fully utilise the capabilities of the low shaft conception.

Per ton of pig iron 2,060 kg of ore with 44 per cent Fe are charged together with 2,900 kg of

dry coal. The oxygen content in the blast is about 40 per cent. Daily production came to 8 t of pig iron with 4 to 4.3 per cent C, 3 per cent Mn, 1.5 per cent Si and 0.02 per cent S (S in the coal was 0.25 per cent). The off-gas is rich: 51 per cent CO, 7 per cent CO₂, 2.4 per cent CH₄, 8 per cent H₂. The experiments proved that the furnace allowed the reduction of iron with coal from low grade fine ores.

The installation requires the manufacture of oxygen and is economically bound to its investment and production cost.

The effects of oxygen enrichment can, to some extent, be secured by highly preheating the blast. 1000°C would be equivalent to 30 per cent O₂—so that the oxygen equipment would be unnecessary though more elaborate means for air preheating would have to be provided instead.

In spite of numerous efforts and considerable sums spent on pilot plants, there is no process sufficiently advanced for large scale commercial and industrial use.

In a still earlier stage of development is the *Finsider process*¹⁵ which provides for the reduction of sintered or pelletised ore in a moving bed with a gas derived from the gasification of coal with oxygen. Rich basic materials are required and the end product is sponge iron. The process is sponsored by the Istituto Siderurgico Finsider at Genova, Italy. No large scale pilot plant operation has become known.

Other processes worth mention are:

The Krupp-Renn process: This is a rotary kiln process¹⁶ which became known before the last war, and has a number of installations to its credit.

In this process the ore and the solid reducing agent passes slowly through a slightly inclined, rotating kiln towards the open end where a highly radiating flame produces the heat necessary for reduction. The slag is pasty and has to be present in large amounts in order to secure the formation of iron "luppen". The ore has to be fine and acid, and comparatively poor. Coke or anthracite can be used as fuel but sulphur has to be low since it is largely absorbed by the iron. The final product has to be separated magnetically from the slag and contains 90-97 per cent Fe. It is ready to go to the electric furnace, the open hearth or the blast furnace but is generally more suited for the latter than as melting stock.

The principal attraction of the process is its capability to reduce ore with an iron content of as low as 20 per cent.

In the largest possible installation, a daily production of 3-400 t is expected.

Depending on the raw material the end product may have a high sulphur content. However, the critical point of the operation is the refractory lining of the furnace which, at present, has a life expectancy of about 7.71½ months. Investment costs depend critically on the type of ore and tend to be high for low grade ores. Correspondingly

higher grade fuel has to be used in such a case.

This process is marketed by the Fried-Krupp Industriebau, Abt. C in Essen, West Germany.

Since 1950 the firm has installed one furnace each in Spain and in Greece. In Germany, Watenstedt has three furnaces in operation, and a six-furnace plant with 40,000 t/year output is under construction in Borbeck. There is one more small kiln completed which was integrated with the Sturzelberg installation.

Indian ores are rich and as such do not seem to be suitable for this process which requires, besides, a highly siliceous ore.

Associated process: In isolated industrial use are, in Western Europe, the Basset¹⁷ and Sturzelberg¹⁸ rotary furnaces while other processes such as Kalling Avesta¹⁹ and Kalling Domnarfvet²⁰ reached the pilot plant stage some time ago but have not reached commercial operation. Laboratory or pilot plant investigations are reported in France according to the Azincourt process¹⁵ and from Italy according to Scortecci process²¹. The first mentioned process produces liquid iron in a conventionally designed rotary kiln with heating the charge while the latter process provides for cracking of solid fuel reduction and a burner/methane at contact with high temperature surfaces under formation of carbon producing sponge iron. From Sweden a new development is reported, the Tanner process²² which uses a hydrocarbon gas which is catalytically broken down to CO and H₂ with which the ore is partially prereduced. A final reduction step is undertaken with the carbon deposited during the catalysis step. A rotating catalysis and pre-reduction furnace is used for the laboratory experiments.

In the U.S.A., the outstanding representative of this group of processes is the *R-N process*²³. It is developed by the Republic Steel Corporation and the National Lead Company.

The charge to the kiln consists of finely divided iron ore, coke breeze and limestone. Natural gas is burned in a combustion chamber at the discharge-end of the kiln. The process is designed to work on both low or high grades ores. The percentage of iron in the ore has been varied from less than 28-70 per cent. The reducing medium is fine coke breeze, less than 1/8" in size, which is mixed with the iron ore and recycle coke extracted from the product, before it is fed to the kiln. Dolomite or limestone is added to the kiln feed to control the sulphur in the product.

The incoming feed absorbs heat from the combustion gases and the heated walls of the brick-lined rotating kiln. The ore reduction proceeds from the outside of the ore particle toward the centre. This produces a fresh metallic surface which, devoid of iron oxides and smeared with dust, does not stick.

The reduction is carried out at about 1,070°C over a major length of the kiln, tapering off to a final temperature of 330°C for the exit gas. Special air inlet ports are located along with the kiln to control the oxygen over the entire length of the reducing chamber.

Under proper operating conditions, the combustion reaction products are largely carbon monoxide, hydrogen and nitrogen and minimum quantities of carbon dioxide. Water and finely divided carbon are also present. The reducing character of these gases emerging from the combustion chamber provides available fuel for combustion heating farther along the kiln. This permits the development of the long, high average contour in the reduction zone, thus providing for the fullest possible reduction of the ore charge by the coke in the bed.

The finely divided primary kiln discharge has a temperature of around 950°C and contains 78 per cent total and 68 per cent metallic iron. This product is further processed—magnetically separated to form a high grade product which will contain about 96.5 per cent total iron, 91.3 per cent metallic iron, 7 per cent moisture, 0.25 per cent C, 0.5 per cent Si and 0.05 per cent P. After being compressed to form large briquettes, the products constitute a suitable charge for the open hearth.

These values are secured for an iron ore with about 63 to 66 per cent total Fe. For proper ores the high grade product will contain about 90 per cent total Fe and the standard grade about 84 per cent total Fe after concentration. It became known recently that only one high grade product can also be obtained.

The secondary sponge iron is briquetted to an eggshaped form and sintered in a neutral or reducing atmosphere in a rotating furnace before being fed to a blast furnace. It is claimed that the coke consumption is materially reduced if these briquettes are mixed with the blast furnace charge.

A pilot plant in Birmingham, Alabama, has been in operation since 1954 producing about 50 t of briquettes per day. No commercial large scale installation has become known.

The process appears to have its application in conjunction with existing further processing stages such as a blast furnace which then would attain higher production. The operation should also suffer from the danger typical of all rotary kiln processes, of sticking and balling of the burden either together or to the walls. This can be aggravated by unsuitable gangue materials which necessitate large scale tests. On the other hand, the uniform temperature distribution in the R-N kiln reduces this danger to a reasonable level. Phosphorus and sulphur removal are effective in the R-N process which is, perhaps, due to the solid contact inside the burden.

Heating at the feed end as practised at times by the R-N method but not by the European plants, secures the advantage of rapidly heating the ore in an oxidising stage thus eliminating sulphur and sticking and discharging it in a reducing atmosphere. The R-N method, at least for medium ores, depends on copious after-treatment by magnetic separation, concentration and compacting.

Other isolated industrial installations in the U.S.A. producing mostly sponge iron are at the National Radiator Company at Johnstown, Pennsylvania and at the Anaconda Company, the latter

using the sponge produced for copper cementation.

The U.S. Bureau of Mines has undertaken pilot plant experiments with the *Larmie kiln*²⁴ which has a preheating zone of smaller diameter, and a reduction zone of larger diameter producing sponge iron which is crushed, magnetically separated and concentrated and then briquetted. No further developments are reported.

Recently, the *Aspegren process*²² became known which is characterised by the addition of heat to the charge passing the rotary kiln with the help of hot metal balls which circulate through the furnace in opposition to the charge movement. The principle has been applied to the pyrolysis of oil shales and could be used also for the gaseous reduction of iron ore. No experimental work seems to have been undertaken.

Next in industrial importance are the "fixed bed" processes.

This group of processes which has attained industrial application has, as a common feature, a burden which during reduction does not move in its container forming thus a "fixed bed" which is permeated by the reducing gas produced either externally or derived from solids surrounding the ore and charge. The most recent advent in the realm of industrial production is:

The HyL process: The development of this process²⁵ commenced with a pilot plant with 30 t/day production. It was installed by the Hojalata y Lamina S.A. of Mexico.

The design of the pilot plant was essentially determined by the raw material situation which at the plant site in Monterrey in Mexico is characterised by a rich ore and by cheap and abundantly available natural gas. It provides for a reactor vessel in which the lump ore rests. The hot reduction gas which is derived from the natural gas by steam reforming²⁶ passes downwards through the bed.

Experiences derived from the operation of this pilot plant which is still in commercial operation, were utilised for the construction of a larger plant designed for the production of sponge iron from hematite ore (Fe_2O_3). The plant handles 200 metric tons per day of total iron delivering 85 per cent as sponge iron and the remainder as ferrous oxide (FeO) representing removal of 90 per cent of the oxygen in the ore. The spent reduction gas is used as fuel in furnaces of the plant thereby effecting a considerable economy.

Natural gas and steam are preheated and then reformed in two identical reforming furnaces. Before mixing in the furnace, the natural gas is desulphurised in the desulphurisation reactors which contain alternate beds of bauxite and iron oxide. The reformed gases pass through quench boilers before entering the reformer quench tower for removal of water vapour. The quenched reformer products, called primary gas, contain 89 per cent carbon monoxide and hydrogen.

Steam is generated in the reformer furnace stacks and effluent boilers. All of the steam is superheated and a part is used to meet the reforming requirements

and the excess may be used for driving turbines or is exported.

Iron ore with 66 per cent average content of Fe, crushed to 1/4" to 1 1/2" is disposed in five reduction reactors each holding approximately 30,000 lb of ore. After preheating, the reducing gas passes through the reactor and the exit gas is piped to a primary quench tower where the water vapour formed during the ore reduction is removed. This gas designated as "secondary gas" is again heated before entering another reactor. After a further removal a water vapour in a "secondary" quench tower the tall gas contains 56.3 per cent CO and H₂.

At any given time two reactors are in the primary position, two are in the secondary position, and one is being loaded or unloaded. The cycle time for a single reactor is five hours. Remotely operated valves and a cycle control system with safety interlocks are employed to minimise operating hazards. Each reactor is directly connected to its own reduction gas preheater and its own quench zone to eliminate the necessity of hot valving. The preheaters are, therefore, of special design to quickly reach operating temperatures since the cooler gases during heating-up must pass through the reactors.

Before opening a reactor for unloading, the charge is treated with natural gas to carburise the sponge iron for control during the subsequent melt. The reduced product retains the approximate size of the charged ore but it has become porous.

After the desired degree of reduction is reached the sponge is dumped out of the reactors and moved to the electric steel furnace where it constitutes up to 50 per cent of the charge.

This plant, designed by the M.W. Kellogg Company, has been in regular operation for well over a year. Its reliable functioning and commercial success have induced Hojalata y Lamina to install another plant of even larger capacity. This plant, also designed by M. W. Kellogg, is now under construction. It will have a capacity of 500 t/day and incorporate some advanced design features predominantly aimed at the reduction of heat requirements. It is anticipated that about 20 per cent of the total heat requirements will be saved.

The gas flow between the reactors is directed by hydraulically and pneumatically operated valves actuated from a control room. The reactors remain in a fixed position, ore being charged from a hopper through the removable top head of the vessel.

The HyL process, as do all "fixed bed" processes, offers *a priori* the advantage of the burden being at rest so that sticking represents a very much smaller risk. The absence of the need for further concentration allows the hot metal to be processed in a final electric stage of melting and steel processing. The inherent simplicity promotes reliability and this has been demonstrated in long range economic operation. Inherent limitation to enlarging the plant capacity of the HyL process seems to depend on a reasonable distribution of reducing gas flow through the bed. These, however, do not come into play

up to a foreseeable plant size of 3,000 t/day capacity.

The development of this process has, so far, been based on the locally available raw material sources and, at present, depends on a gaseous basic source for the reducing gas. For either liquid or solid fuels suitable gasification stages could be made to precede the principal process, and this extension of the applicability of the process is, at present, under very active consideration.

Associated processes: The *Hoganas process*²⁷ was developed in Sweden where the most modern plant in Oxelesund produces 50 t/day of sponge iron. Its features are duplicated in an installation at Riverton, U.S.A., where in two tunnel furnaces 15-20,000 t/year of sponge iron are produced. These furnaces are very long, since the charge consisting of columns of finely ground ore, lime and coke enclosed in saggars, must be exposed to a peak temperature of 1,150°C for 12 to 36 hours. Higher temperatures are possible if ceramic saggars are used.

Loading, moving and unloading of the containers is completely mechanised. They remain standing on low cars which travel to the different points of the cycle of the process.

The process principles are simple; the heat is supplied by hot furnace gases, the reducing gas is formed by the solid carbon; and this simplicity is the reason for its reliability. The technical and economic disadvantages are low rate of production and the breakage of the ceramic saggars. Metallic saggars would be advantageous but allow temperatures of only up to 1,090°C approximately.

The ore is packed in the saggars in an annular layer surrounded by the powdered reductant so that mechanical cleaning of the sponge cylinder from excess coke and ash is possible. Normally, highly enriched ore is used, and the end product has about 97 per cent Fe, 0.25 per cent C, 0.01 per cent P, 0.01 per cent S.

It should be noted that the saggars are highly endangered by iron oxide corrosion, by furnace and reducing gas attack, by thermal shock, and lastly, by handling.

Another method using a fixed bed is the *Norsk-Staal*²⁸ process which found brief application in West Germany.

Development projects

These processes are still being investigated and none has, so far, been used as the basis for industrial production.

A large sector among them is characterised by a fluidised bed in which powdered iron oxides are held in suspension in an upward flowing reducing gas until reduction is achieved. Pressure and temperature in the reduction vessel vary widely and so cause process material flows.

This technique has been widely adopted in the U.S.A. and to a very small extent only in Europe, where only two methods apply it in laboratory experiment scale.

The first is the *Stelling process*²⁹ developed in Sweden which uses CO as reducing gas. The

reduction process is subdivided into three stages not unlike the Wiberg process, the first preheating the powdered concentrate, while at the same time magnetite is oxidised to hematite at about 900°C, the second pre-reducing it at about 750°C, the third finally reducing it at 600°C whereby all stages are fluidised. The important promise of the process is that at the temperatures chosen, sticking is avoided due to the formation of cementite FeC—which occurs rapidly and out-paces the decomposition of CO into C³ and CO₂. Thereby, an end product is gained which is not pyrophoric and allows many types of further treatment.

Only laboratory tests have been done along the lines proposed by the inventor, Prof. O. Stelling, from Stockholm.

The second European process working with a fluidised bed, is being tested in France at the ONIA³⁰, the *Office National Industriel de l'Azote* in Toulouse. It applies a gas mixture of H₂ and CO at 650 to 700°C derived by cracking of methane for the reduction of the powdered concentrate. An experimental installation of a production of 10 t day is planned.

In the U.S.A., the *H-Iron process*³¹ is sponsored by the Hydrocarbon Research, Inc., in connection with Bethlehem Steel, and the Allan Wood Company.

The H-Iron process is a fluid bed system operating at about 400 psig and 900°F using 99 per cent purity hydrogen as the reducing agent and producing a fine mesh high grade sponge iron. The process is best suited for operation with high grade ore having less than 3-4 per cent gangue. Since many beneficiated processes produce high purity fines as their product these concentrates can be used directly.

High purity hydrogen (in excess of 99 per cent H₂) is fed to the plant. It can be produced by any number of methods such as steam reforming of hydrocarbons, partial oxidation or coal gasification. The high purity is required since any CO is quantitatively converted to CH₄ under the conditions used in the reactor.

Wet hydrogen leaving the reactor is cooled first by heat exchange with the dry recycle hydrogen and then further cooled to 100°F in a direct contact cooling tower. In this way all the water of reaction is condensed from the recirculating stream. A portion of the recycle stream is vented to control the concentration of inerts. The recycle is then combined with fresh hydrogen. All CO and CO₂ are removed from the feed by passing it through a shift converter and then through a CO₂ scrubber. The mixture is boosted to 400 psig, heated in the furnace to 1,000°F and sent to the reducer.

Iron ore is first fed to a rotary drier which dries the ore and preheats it to 900°F. It is then transferred pneumatically to a vertical charge hopper and then charged under hydrogen pressure to the reducing vessels. Dry hydrogen preheated to 100°F under 400 pounds pressure is passed slowly through the charge. The heat of reaction is

provided by the sensible heat of the hydrogen above 900°F.

The bed is subdivided into numerous wall "tubes" having a high L-D ratio. Intimate solids-gas contact is obtained by passing the dry preheated H₂ in slug-flow fashion upwards in the bed, the ore particles showering through the rising bubbles.

Because the reduction is batchwise with respect to the iron ore, conversion varies from the beginning to the end of the reaction. In order to avoid either having to waste hydrogen by increasing the purge as less hydrogen is consumed or varying the supply over wide ranges two or three beds are reduced in series. Each bed is in a different state of reduction thus keeping the degree of H₂ consumption relatively constant. When the batch in the bottom bed is finished it is dumped and the contents of the upper bed are dropped to a lower bed. A new batch of ore is then charged to the top bed.

The process produces a pyrophoric iron powder. It can produce iron suitable for processing into steel in the electric furnace or open hearth (say 85-90 per cent reduction), or a highly reduced material (95 per cent reduction) suitable for powder metallurgy. A fuel gas by-product is produced by purging.

The plant using this process is now under construction at Conshohocken, Pennsylvania, for the Allan Wood Steel Company. The product will be used for powder metallurgy. The production rate is said to be 50 t/day.

The *Esso A. D. Little process*³⁵ is sponsored by the Esso Research and Engineering Company, and Arthur D. Little, Cambridge, Massachusetts.

It uses a fluidised bed reactor, operates at essentially atmospheric pressure and 815-870°C and produces a powdered sponge iron. The reducing medium is hydrogen and carbon monoxide generated by partial combustion of natural gas with air in the bottom of the reactor.

The air is preheated to about 840°C and the natural gas to 340°C and the two gases are introduced into the bottom of the reducer reactor.

Unlike either the H-Iron or Nu-Iron process, the carbon dioxide and water of combustion are not removed from the reducing gas since the reducing gas is used once through so that large quantities of by-product fuel gas for use throughout the plant are available.

Ore which has been preheated to about 840°C by burning a portion of the gas leaving the reactor is fed to the top tray of the reactor and there reduced to FeO. In the bottom bed the reduction is completed to give a product containing from 5-30 per cent FeO. Temperatures in the bed are maintained at about the level given. The gas leaving the reactor is only partially converted to H₂O and CO₂ and still retains sufficient calorific value (1,200 BTU/ft³) to be used as a fuel.

High grade powdered ores or concentrates are used. Lower grade ores can probably be handled also, but at a loss of both operating and thermal efficiency. The product will, of course, have to be concentrated

since all the gangue remains with the product.

No industrial installation has, so far, become known.

The *U. S. Steel (Nu-Iron) process*³³ is developed by the U. S. Steel Company in connection with Mr. Shipley.

The Nu-Iron process uses a mixture of hydrogen and carbon monoxide as the reducing agent. Temperature and pressure at which the reduction is conducted are about 700°C and 60 psig.

The process is best suited for operation with high grade ore although it can probably handle low grade ores also. In the latter case, the product can be concentrated by passing it through a magnetic concentrator.

High grade iron ore fines (minus 1/4") are preheated first to 370°C with the sensible heat in the off-gas from the reducing reactor. This preheated ore then passes through an ore heater where, by combustion of natural gas or any suitable fuel, it is heated to about 925°C.

The hot ore passes into the top bed of the reducer where at 700°C it is reduced essentially to FeO. The sensible heat in the ore supplies a part of the heat needed for the endothermic reducing reactions of this primary reactor. The partially reduced ore passes to the second bed where again at about 700°C the FeO is reduced to metallic iron.

The off-gas from the reactor contains CO, H₂, CO₂ and H₂O and some N₂. This gas passes through a cyclone and electrostatic precipitator to remove dust and waste heat boiler to recover heat. The gas is then cooled to about 38°C to remove water vapour. After mixing with the make-up gas from the reforming unit, the total gas is compressed to 60 psig and scrubbed with monoethanolamine to remove CO₂. The resulting gas is then preheated to 870°C. To maintain inerts down to an acceptable level, some of the gas is purged from the system.

No industrial installation has, so far, become known.

The *Inland Steel Company* in Chicago, Illinois, is also known to have a fluidised process under development.

With regard to fluidised processes, it can be said that the principles of the technique are far from general acceptance. Since particles are of different sizes their reduction will be substantially complete at different times. Besides, residence time requirements are generally high at temperatures sufficiently low to avoid sticking and defluidisation of the bed. Lower operating temperatures, on the other hand, can cause a substantial decrease of a reducing reactor. The highest permissible temperature depends strongly on the type of ore. The fluidised bed technique is also problematic with respect to bubble formation in the bed and the danger that bursting bubbles can carry over large amounts of solids.

All fluidised bed processes depend to a decisive degree on the fluidisation characteristics of the ore: Certain ores, e.g. from Venezuela I prove unsuitable, others such as Labrador ores, can be used without difficulties. The phenomenon of "slumping" is still unexplained and can, so far, be avoided by low

temperatures, or by leading the process to a different product, i.e., iron carbide, as does the Stelling process. The particle size of the ore must meet the demands of fluidisation—the ore must be rich, or a concentrate must be used.

The processes are not yet developed to such a level that they could be considered for future industrial planning of guaranteed production.

Some other development projects may be mentioned.

In England, the *Cyclo-Steel process*³⁴ has been investigated. Powdered ore is reduced to iron in a gas swirl prior to a melting stage where the iron is molten by powdered coal which burns under addition of oxygen. From this zone gases rise to the reduction zone causing preheating and pre-reduction. The gases move in a free vortex due to their tangential entry into the vessel in which the powdered solids are in suspension. Liquid metal is the end product which can have the characteristics of steel by preventing C-pick-up in accordance with the amount of O₂ added.

A major difficulty appears to be the high rate of heat losses through the walls although an operation in analogy to slagging gas turbine combustion chambers should be feasible.

The Ontario Research Foundation investigated in Canada the *O.R.F. process*²⁶ which provides for a travelling bed of magnetite concentrate moving through a furnace where it is first preheated and burned to hematite thus binding the particles together and raising the temperature. Preheated reducing gas with 3:1 CO:H₂ passes the bed and reduction to a sintered cake is complete within one hour. The cake temperature allows immediate rolling into a steel slab. The process depends obviously on a very pure concentrate with small gangue content.

The *Jet Smelting process*²⁶ similarly to the Cyclo-Steel process uses the fast reduction of powdered oxides at extremely high temperature as it exists in a methane oxygen flame whereby magnetite is first burnt. The hot hematite is then reduced by the carbon monoxide and hydrogen derived from the cracking of methane. Final reduction takes place in the bath of molten metal where a considerable carbon content is maintained. The process needs high grade ore and requires an oxygen plant.

None of these development projects appears to be sufficiently promising to allow consideration for large scale installation under financially stringent conditions.

Economic process assessment

For a valid economical assessment of different processes they should be compared when working under identical operating conditions or, if this is not feasible, the effect of deviations from a normalised operation should be known and taken into account.

This ideal is not attainable due, mainly, to differences of the external circumstances under which the different processes operate, and the vastly different degrees of development they have reached.

When collecting economic data it should also be

realised that the price structure for materials and services vary widely. Indications in literature are, in addition, often clouded for competitive reasons.

In Table I operational data of the processes enumerated before, are given starting from the ore feed characteristics (L=lump, G=granulated ore, P=pelletised ore, B=beneficiation before reduction indicated) and the iron content of the ore. The primary sources of the reducing agent (S=solid, l=liquid, f=fuel) and of the flexibility of the process are given and the form of (m=molten, S=sponge, \angle u=luppen) iron content and likely further treatment (B=beneficiation, St=steel) of the end product are shown. Current plant sizes and those feasible in the future conclude the table.

It should be noted that processes which do not rely on the separation of the slag from the pig iron in liquid form, have to rely on pre-or-post beneficiation with pre-processing of the ore being

applied predominantly by fluid bed and fixed bed processes, while rotary kiln processes prefer post-reduction up-grading. The first process group is very inflexible as to its sources of energy, while the Wiberg and the rotary furnace groups are much less demanding. The fixed bed process group calls for a gaseous medium, which can be derived from any form of fuel, and this holds for fluidised bed processes, too. The blast furnace imposes important restrictions on the basic materials it accepts. If plant sizes as they are under construction at present are compared, the figure of the HyL process becomes very significant.

In Table II average basic material requirements are listed. These figures depend very much on the type of the ore and its characteristics and, thus, tend to penalise processes operating with low grade ores or fuels. With increasing flexibility of a process consumption figures are less significant since

TABLE I
Operational data of iron ore reduction processes

Designation	Ore-Feed	% Fe	Slag Flux	Reducing material		End product			Plant size t/day	
				Source	Flexibil.	Form	% Fe	Treatment	Current	Future
1. <i>Tysland-Hole</i>	L	56	Yes	s.f.	none	m	94	—	70	250
Elektro-Chemisk	L	53-63	Yes	s.f.	none	m	—	—	—	250
Strategic-Udy	L	58	Yes	s.f.	none	m	—	—	30	150
Lubatti	L	50-55	Yes	s.f.	none	m	—	—	40-70	—
Jensen	P, L, B	>33	No	s.f. gasif., gas	small	s	93.8	B	—	—
Lurgi-Galluser	P, L, G	—	No	s.f. and gas	medium	s	—	—	—	—
2. <i>Wiberg</i>	1-3, P	66	No	s.f. gasif.	small	s	90	B	100	200
Calbe	1/8-1/4	20-25	Yes	s.f.	none	m	94	—	70	150
Ougree	to 1, P	25-55	Yes	s.f.	small	m	93.5	—	20-40	—
Durrer (Oxygen.)	to 1	44	Yes	s.f.	small	m	94	—	8	—
Finsider	L, P	62	No	s.f. gasif., gas	small	s	90	—	—	—
3. <i>Krupp-Renn</i>	1/3, G	60(30)	Yes	s.f.	none	\angle u	96	B	220	300
Basset	G	67	Yes	s.f.	none	m	—	—	60	—
Sturzelberg	G	60	Yes	s.f.	none	m	—	—	80	—
Kalling-Domn.	G	55	Yes	s.f.	none	s	—	B	50	—
Azincourt	G	40-55	Yes	s.f.	none	m	—	—	5-15	—
Scortecchi	G	59-63	Yes	gas from f.	large	s	—	B	2	—
R-N	G	30-60	Yes	s.f.	none	s	96.5	B	50-75	200
4. <i>HyL</i>	1/4-1-1/2	58-66	No	gas from f.	large	s	90	B	550	2,000
Hoganas	G	70	No	s.f.	none	s	96	B	50	100
Norsk-Staal	L	68	No	s.f. gasif.	small	s	92	B	18-38	—
Stelling	B, P	62.5	No	s.f. gasif.	small	Fe ₃ C	—	—	0.1	—
ONIA	B, P	60-68	No	L. f. or gas	medium	p	—	—	0.5	—
H-Iron	B, P	66	No	H ₂ from f.	medium	p	95	B	50	2,000
Esso-Little	B, P	64.5	No	nat. gas	small	p	95	—	1	—
US-Steel (Nu)	B, P	—	No	gas from f.	large	p	95	—	5	1,000
ORF	B	—	No	gas from f.	large	—	—	st	—	—
Blast Furnace	1/4-2-1/4	50	Yes	s.f.	none	m	94	B	2,800	3,500

TABLE II
Basic material data of iron ore reduction processes.

Designation	Coke (t)	Consumption/ton Coal (t)	Total Iron in Oil	Final product (Others) ¹	Power (kWh)	Credits for
1. <i>Tysland-Hole</i>	0.310	—	—	—	2,020	Gas
Elektro-chemisk	0.390	—	—	—	1,400	Gas
Strategic-Udy	—	0.430	—	—	1,030	—
Lubatti	—	—	—	—	2,200-2,400	Gas
Jensen	—	{ 0.16 C or 200 Nm ³ n.g. }	—	{ 750 Nm ³ gas 41 Nm ³ n.g. + 151 kg coke }	1,130-2,300	—
Lurgi-Galluser	—		{ 63 kg+ 111 kg }		850-1,000	—
2. <i>Wiberg</i>	0.18-0.23	or	{ oil }	or	870-1,000	—
Calbe	1.7-2.2	—	—	—	—	Gas
Ougree	1.2	—	—	—	—	Gas
Durrer	—	2.9	—	350 Nm ³ O ₂	—	Gas
Finsider	0.376	—	—	110 Nm ³ n.g.	152 for O ₂	Gas
3. <i>Krupp-Renn</i>	0.770-1.0	0.19-2.50	—	—	62-110	—
Basset	0.650	—	350 kg	—	—	—
Sturzelberg	0.7-0.90+	0.600	—	—	—	—
Kalling-Domn.	0.650	—	—	—	—	—
Azincourt	—	0.800+	200 kg	—	100	Gas
Scortecci	—	—	—	1,500-3,500 Nm ³ n.g.	—	Gas
R-N	0.550	—	—	74.5 Nm ³ n.g.	146	—
4. <i>HyL</i>	—	—	—	4,200 Nm ³ n.g.	10.5	Stm. & Gas
Hoganas	0.65-0.80	—	or 236 ltr.	—	100-120	—
Norsk-Staal	0.22	+ coke oven gas	—	115 Nm ³	15-1,600	—
H-Iron	—	—	500 ltr. or	465 Nm ³ n.g.	100-165	—
Esso-Little	—	—	—	435 Nm ³ n.g.	—	Gas
Blast Furnace	0.905	—	—	—	—	Gas

TABLE III
Economics of iron ore reduction processes

Designation	Type	Manhours h/t	Int. Oper. Cost \$/t	Total Invest. \$	Plant Capacity t/year	Spec. Inv. Cost \$/t year
1. <i>Tysland-Hole</i>	E	2.45	70-75	94
Strategic-Udy	E+R	1	44	...	200,000	40-50
Jensen	E+S	10·10 ⁶	190,000	55.5
Lurgi-Gallusen	E+S	...	35
2. <i>Wiberg</i>	S	1.1-2.0	45	20·10 ⁶	35,000	50-60
Calbe	S	0.85	38	1.8·10 ⁶	30,000	52
3. <i>Krupp-Renn</i>	R	3.0-4.0	44	21.4·10 ⁶	420,000	51
Basset	R	...	41	2.6·10 ⁶	28,000	94
Sturzelberg	R	3.5·10 ⁶	85,000	41
R-N	R	0.42	39	14·10 ⁶	350,000	40
4. <i>HyL</i>	Fi	0.16	29-35	22·10 ⁶	725,000	30
Hoganas	Fi	0.10	40	6·10 ⁶	30,000	200
Norsk-Staal	Fi
H-Iron	FL	0.22	35-43	24·10 ⁶	660,000	36
Esso-Little	FL	14·10 ⁶	330,000	42.5
Blast Furnace	S	0.85	35-50*	80-100

* Depending upon degree of integration, age and size of blast furnace.

the greater adaptability of a process to a range of conditions tends to outweigh low consumption. The faculty of a process to use available raw materials will often predominantly determine its application.

A main consideration is the prices of the raw materials, their transport cost, etc., so the choice has to be undertaken on the ground of actual costs rather than consumption figures. Where, as is the case in India, lignite and coal are plentiful, processes using these materials will be favoured. For this reason processes which work with a gaseous medium, i.e. fixed bed processes, should receive first consideration since they provide flexibility with respect to the material from which the gas is manufactured.

Several processes produce usable by-products which can be utilised by integration with other production lines.

In Table III, finally, integrated operational costs in \$/ton of the final product, total investment cost in \$ for a plant whose yearly capacity is given, and the specific investment costs per yearly ton are listed. These figures have been derived from actual raw material costs and operating data for some principal locations in the U.S.A. for fully integrated conditions and include maintenance and capital services.

The table shows that the integrated blast furnace is practically—since Hoganas is a very specialised process—the most expensive scheme with respect to investment, and is the second costliest with respect to operational costs. This latter fact reflects that purely electrical processes are economically not feasible in that region and thus underlines that such comparisons, to be valid, must be undertaken for a given locality.

Comparing the relative figures of Table III, it is revealed that other processes have investment figures 40–65 \$/yearly ton with operating cost of 40–50 \$/ton. These figures are confirmed in the literature²⁶. A considerably cheaper process is the HyL process, with costs close to \$ 30 for both values. Its manhour requirements are also low.

In Table III the type of the process is also indicated E—electric, R—rotating kiln, S—shaft furnace, Fi—fixed bed, FL—fluidised bed.

In the last table, Table IV, average costs are given which are valid for a cross-section of different locations

in the U.S.A.

For any economic assessment, say in Asia, all relevant factors such as equipment and total installation cost at site, raw material requirements, operating cost, maintenance, labour and capital charges have to be taken into account for investment and production expenditures for technically possible processes for a given location, a task which cannot be undertaken in general but only in specific terms.

Conclusion

Resources of rich iron ore in Asia, particularly in India and the Philippines, are extremely plentiful but high grade fuels often are scarce. Economic conditions enforce the strictest financial limitations which together with restricted levels of demand, determine the frame within which technological solutions for the production of iron and steel have to be found.

Modern developments of ore reduction technology answer the quest for an indigenous iron and steel industry for Asian countries by providing for lower unit production, for lower operating and investment cost while accepting a much wide range of fuels and ores, especially also fines than is required by the conventional blast furnace. They often discard the necessity for pre-sintering.

From among the processes considered in the more recent technical literature four principal process schemes evolve as proved in regular large scale economic and commercial operation. They are: the Tysland-Hole, the Wiberg, the Krupp-Renn and the HyL processes which, therefore, demand first consideration for planning in Asia. Their individual operational features allow for flexibility of their raw material requirements in varying degrees.

The HyL process offers the best chance of suitability for areas with cheap raw materials, for the reducing gas it employs, namely, primarily natural gas, other fuels, liquid, and particularly solid can be used in combination with conventional gas producers.

Economic comparisons of these principal processes reveal that the HyL process is the cheapest with respect to operation and investment. The fact that it can be constructed with local labour is further reason for its Asian conditions.

TABLE IV
Representative raw material prices used in economic comparison

Raw material	Coke, \$/t	Lime-Stone, \$/t	Elect-rodes, \$/t	Pwr, $\frac{\text{mils}}{\text{kWh}}$	Coal, $\frac{\text{\$}}{\text{t}}$	Coke breeze, $\frac{\text{\$}}{\text{t}}$	Lignite, $\frac{\text{\$}}{\text{t}}$	Nat. Gas, MCF $\frac{\text{\$}}{\text{t}}$	Fuel Oil, $\frac{\text{\$}}{\text{Bl.}}$	H ₂ 99.5%, $\frac{\text{\$}}{\text{MCF}}$	Scm, $\frac{\text{\$}}{\text{Mlb}}$	Labour, $\frac{\text{\$}}{\text{hr}}$
Unit Price	13.00	16.50	260	8.0	5.50	6.50	2.70	0.20	3.00	0.35	0.60	3.30

Iron ore assumed to cost \$11.60/t at Fe content of 51.5%, and \$0.35 for each additional % Fe.

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