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DIRECT REDUCTION OF IRON ORE

a bibliographical survey

The Metals Society, London

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I

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FOREWORD

Although there have been great developments in blast furnace technology during the last twenty years, in the same period considerable research and development work has been carried out throughout the world on producing iron from ore without the blast furnace. Because there are so many possible process and plant parameters a large number of proposals have been put forward.

The former High Authority of the European Coal and Steel Community (ECSC) and the present European Communities' Commission has from the start regarded ensuring the raw material and energy base for iron and steel production as one of its most important tasks. In this connection the Commission has attached great importance to the development and application of methods of direct reduction of iron ore and has actively shared in the development of both new and existing direct reduction processes by providing considerable assistance for research in accordance with article 55 of the ECSC Treaty.

In order to give all interested parties an overall view of the present state of process development and of the results of ECSC research, and to stimulate further research in this field, the then High Authority of the European Coal and Steel Community, as early as 1958, commissioned a group of experts from the countries of the Community to produce a bibliographical review of the methods of direct reduction of iron ores. This first edition was followed in December 1960 by a second, revised and extended edition; due to the rapid development in the field of direct reduction of iron ore it was superseded in May 1967 by a third.

Since then the development, and particularly the industrial application, of some of these processes has made rapid progress, so that it appeared urgently necessary to produce such a bibliographical review again, also taking into account the results of the research projects supported by the European Communities' Commission.

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In the meantime it has also become possible, because of the application of direct reduction on an industrial scale, to obtain more precise data on the process parameters and on the economics and costs. Thus the purely bibliographical part has been complemented by a large section, namely an investigation into possible supplies of sponge iron in the European Coal and Steel Community, in which both the general development and the availability of iron ore and reducing agents for direct reduction are discussed in relation to various typical locations.

The Commission wishes to express its thanks to the experts who have made themselves available for this work. The bibliographical review was produced in the main by the following:

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The European Communities' Commission also wishes to thank the developers of the processes, who have helped by providing data to complete the study.

The investigation into the possible supplies of sponge iron in the European Coal and Steel Community was carried out jointly by members of the Institut de Recherches de la Sidérurgie Française and the Verein Deutscher Eisenhüttenleute.

Brussels, September 1976

European Communities'
Commission

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Possible sources of supply of sponge iron in the European Coal and Steel
Community

1. The scrap market situation

A study of the future scrap requirements is of particular value in estimating the probable development of direct reduction of iron ore since sponge iron is in direct competition with scrap for steel-making. The scrap requirement depends on the total production of crude steel, on the proportions produced by the various methods and, finally, on the specific quantities of scrap consumed in the various steelmaking processes.

The estimates are based on the study on scrap by the International Iron and Steel Institute⁽¹⁾, on findings of the scrap commission of the Club des Sidérurgistes⁽²⁾ and on data from the European Communities' Commission. The estimates are made for the countries of the European Community assuming first an average level of steel production and then boom conditions until 1985.

Table 1 shows the estimated probable scrap requirement in the countries of the ECSC for an average level of production. According to the forecasts of the European Communities' Commission the total crude steel production will be 170 million tonnes in 1980 and 188 million tonnes in 1985. The proportions of the crude steel produced by the individual processes are based on data provided by the Club des Sidérurgistes and on published investment figures from the ECSC.

Table 2 shows the estimated scrap requirement for boom conditions. the total crude steel production will, according to the same source, be 183 million tonnes in 1980 and 200 million tonnes in 1985.

(1) World Prospects for Steel Scrap up to 1980, IISI, July 1974.

(2) Supply and Demand of Reduced Pellets in the European Community by 1980, April 1975.

Table 1. Estimated probable scrap requirement up until 1985 (average economic activity, according to estimates of the European Communities' Commission) (in 1000 t)

Steel production	1973	1980	1985
Oxygen - blown converters	87,267	123,170	138,800
Open hearth furnaces	24,747	12,100	4,200
Electric arc furnaces	22,631	34,730	45,000
Other	14,860	-	-
Total	149,505	170,000	188,000
Scrap consumption, Steelworks			
Oxygen - blown converters	21,978	32,110	37,767
Open hearth furnaces	16,620	8,415	3,373
Electric arc furnaces	22,846	35,814	46,908
Other	2,395	-	-
Total	63,839	76,339	88,048
Scrap requirement, Steelworks			
Internally recycled scrap	32,443	33,150	34,780
Bought-in scrap	31,396	43,189	53,268
Scrap requirement, BF	1,352	1,900	2,100
Scrap requirement, foundries etc.	7,484	8,500	10,000
Total scrap requirement, bought scrap	40,232	53,589	65,368
Availability of scrap in the ECSC	38,289	51,400	61,300
Deficit/Excess	- 1,943	- 2,189	- 4,068

The amount of internally recycled scrap was taken into account in both cases; it was put at 21.7% for 1973, 19.5% for 1980 and 18.5% of the crude steel production in 1985 because of the increased use of continuous casting. The specific amounts of scrap charged in the individual processes were determined as means of the estimates of the ECSC countries⁽²⁾, as in Table 3.

The trend which emerges from these estimates indicates that with an average level of steel production we may expect in the ECSC a scrap deficit of about 2 million tonnes in 1980 and about 3 million tonnes in 1985. Under boom conditions the scrap shortage would be considerably greater, about 3.5 million tonnes for 1980 and about 5.5 million tonnes for 1985. The scrap deficit would of course vary according to region. A clear example of this is provided by Italy, where a particularly great deficit may be expected because of a much more than proportional increase in electric steelmaking. At the same time the availability of scrap and so the possibility of buying-in from countries outside the Community is declining because of the fairly high increase in steel production in these countries. The discrepancies between supply and demand for scrap are due to the drop in the amount of internally recycled scrap because of the increase in continuous casting and the fact that electric steelmaking has grown much more than steel production as a whole.

It can also be seen from Tables 1 and 2 that apart from the shortage of scrap the ratio of internally recycled to bought-in scrap is becoming worse and worse. This means that the quality of the scrap is deteriorating overall, since bought-in scrap is generally of poorer quality than recycled scrap because it contains undesirable elements.

Both results indicate the need to make up the shortage of scrap with sponge iron, which from the point of view of both its chemical composition and metallisation is suitable for use in electric arc furnaces and in oxygen-blown converters.

Table 2. Estimated probable scrap requirement up until 1985 (boom conditions, according to estimates of the European Communities' Commission) (in 1000 t)

Steel production	1973	1980	1985
Oxygen - blown converters		132,810	147,580
Open hearth furnaces		13,890	5,200
Electric arc furnaces		34,730	47,220
Other		1,570	-
Total		183,000	200,000
=====			
Scrap consumption, Steelworks			
Oxygen - blown converters		36,624	40,157
Open hearth furnaces		9,660	4,176
Electric arc furnaces		35,814	49,222
Other		149	-
Total		80,247	93,555
=====			
Scrap requirement, Steelworks			
Internally recycled scrap		35,685	37,000
Bought-in scrap		44,662	56,555
Scrap requirement, BF		1,900	2,100
Scrap requirement, foundries etc.		8,500	10,000
Total scrap requirement, bought scrap		55,062	68,655
=====			
Availability of scrap in the ECSC		51,400	63,000
Deficit/Excess		- 3,662	- 5,655

Table 3. Specific amounts of scrap charged according to method of steelmaking (according to an investigation by the scrap commission of the Club des Sidérurgistes)

Process	Specific amount of scrap ¹⁾ in kg/t crude steel		
	1973	1980	1985
Oxygen-blown converter	251.8	260.7	272.1
O H furnace	671.6	695.5	803.1
Electric arc furnace	1009.5	1031.2	1042.4
1) Average values for the countries of the ECSC			

2. Estimated development of direct reduction

2.1. Critical appraisal of the published plans for building installations

Table 4 lists all the direct reduction plants in operation, under construction or at the planning stage throughout the world as at December 1975. The nominal capacity of all the plants in operation is about 6 million tonnes per year. A further 8.3 million t.p.y. are under construction and 8.8 million t.p.y. are planned. This gives altogether about 23 million tonnes nominal capacity per year (gas reduction about 90%, solid reducing agent 10%) of which about 55% is produced by the shaft furnace processes, 33% by the retort process (HyL), 8.5% using a rotary kiln and 3.5% a fluidised bed. It should be noted that these figures only cover plants in operation which produce sponge iron with a degree of metallisation of > 75%.

An earlier study⁽³⁾ shows that at present the actual production of all the plants in operation is about 80% of the nominal capacity available. This ratio varies for the different groups of processes. It is best in the retort process (HyL) and declines in the following order: shaft furnace, rotary kiln and fluidised bed.

Table 5 is based on a comprehensive review of the literature and shows all the direct reduction projects which are at an experimental stage as at December 1975. It must be emphasised that this list was produced without critical assessment, that is, in many cases it is uncertain whether these projects will come to fruition. The plans listed in this Table indicate planned direct reduction installations with a total capacity of about 56 million tonnes per year - about 17 million tonnes

(3) Kalla, U. and R Steffen: Stahl u. Eisen 94 (1974) pp 1109-14.

in the Middle East, about 3 million tonnes in the rest of Africa and about 6.3 million tonnes overall capacity in Asia, Australia and Oceania, 7.2 million tonnes in North America, about 9.0 million tonnes in Central and South America and finally about 16.5 million tonnes in Europe as a whole. It is not clear with several of these projects how suitable the proposed ores are for direct reduction or for processing the sponge iron to steel at a favourable cost.

There are three main criteria for choosing the location of direct reduction plants:

1. the availability of suitable reducing agents,
2. the availability of suitable ores and
3. the location of the sponge iron consumer.

Fig.1 shows the geographical location of the plants listed in Tables 4 and 5. It is interesting to note that the direct reduction plants in question are located primarily in relation to the availability of suitable reducing agents. This applies particularly to those in Table 5 planned in North Africa and the Middle East. Suitable ore supplies are of only secondary importance in choosing the location. However, the proximity to the consumer can be of prime importance and transporting the ores and reducing agents over longer distances can be acceptable, as is shown by the projects for Europe, with a planned capacity of about 16.5 million tonnes.

Table 4. List of direct reduction plants throughout the world +)

Process	Company	Location	In operation	Nominal capacity in 10 ³ tpy			
				year	planned	year	
<u>Gaseous reducing agent</u>							
Wiberg	Sandvik	Sandviken		(1952)			
	SFK	Hofors	25	(1960)			
	Hitachi Metals	Yasugi	10	(1964)			
HyL	Hylsa	Monterrey I	100	(1957)			
	Hylsa	Monterrey II	270	(1960)			
	Tamsa	Vera Cruz	235	(1967)			
	Hylsa	Pueblo I	300	(1969)			
	Usiba	Bahia	325	(1974)			
	Hylsa	Monterrey III	450	(1974)			
	Hylsa	Pueblo II		(Mexico)	700	(1977)	
	Sidor	Matanzas		(Venezuela)	400	(1976)	
	NISIC	Ahwaz		(Iran)	1200	(1977)	
	Krakatau-Ferrosteel	Kota Baja		(Indonesia)	500	(1977)	
	Tika	Solvezi		(Zambia)	300	(1977)	
	State Company	Khor El Zubeir		(Iraq)	600	(1977)	
	Midrex	Oregon Steel	Portland	400	(USA)		
Georgetown Ferreduction		Georgetown	400	(USA)	(1969)		
Hamburger Stahlwerke		Hamburg	400	(Germany)	(1971)		
Sidbec-Dosco		Contrecoeur I	400	(Canada)	(1972)		
Sidbec-Dosco		Contrecoeur II		(Canada)	(1973)		
Dalmine Siderca		Compana		(Argentina)		(1976)	
Sidor		Matanzas		(Venezuela)		(1976)	
Acindar		Villa Constitucion		(Argentina)		(1976)	
NISIC		Ahwaz		(Iran)	1275	(1978)	
Qatar Steel Company		Umm Said		(Qatar)	420	(1977)	
FI-EL Korf		Sao Jose dos Campos		(Brazil)			
		Kursk		(USSR)	400	(1977)	
State Company		Hunterston		(UK)	2500	(1979)	
British Steel Corp.		Ipatinga		(Brazil)	800	(1978)	
Usiminas					800	(1978)	
Purofer		Thyssen Niederrhein	Oberhausen	100	(Germany)		
		NISIC	Ahwaz		(Iran)	(1971)	
	Cosiuma	Santa Cruz		(Brazil)	360	(1976)	
					360	(1976)	

Armco	Armco	Houston I	(USA)	330	(1972)				
	Armco	Houston II	(USA)					380	(1977)
ICEM	Inst.Cerc.Metalurcice	Bucharest	(Rumania)	8	(1972)				
H-D	Sider.Dedini	Piracicaba	(Brazil)					200	(1977)
HIB	Orinoco Mining	Puerto Ordaz	(Venezuela)	350	(1972)				
Fior	Fior de Venezuela	Matanzas	(Venezuela)			400	(1976)		
Allis Chalmers	Niagara Metals	Niagara Falls	(Canada)	30	(1973)				
	Sudbury Metals	Falconbridge	(Canada)			200	(1976)		
NSC	Nippon Steel	Hirohata	(Japan)			150	(1976)		
<u>Solid reducing agent</u>									
Höganäs	Höganäs	Höganäs	(Sweden)	162	(1963 & 74)				
	Höganäs	Riverton	(USA)	70	(1954 & 60)				
	Gränges	Oxelösund	(Sweden)	35	(1954)				
SL/RN	New Zealand Steel	Glenbrook	(New Zealand)	120	(1970)				
	Acos Finos Piratini	Porto Alegre	(Brazil)	65	(1973)				
	Nippon Kokan	Fukuyama	(Japan)	350	(1974)				
	Hecia Mining	Casa Grande	(USA)	65	(1975)				
	Stelco	Red Lake	(Canada)	350	(1975)				
	Siderperu	Chimbote	(Peru)			100	(1977)		
Krupp Sponge Iron	Dunswart Iron & Steel	Benoni	(South Africa)	150	(1973)				
Kawasaki	Kawasaki	Chiba	(Japan)	60	(1968)				
	Kawasaki	Mizushima	(Japan)	180	(1974)	180	(1977)		
Koho	Nippon Steel	Muroran	(Japan)	36	(1971)				
SPM	Sumitomo Metal Ind.	Kashima	(Japan)	100	(1975)				
SDR	Sumitomo Metal Ind.	Wakayama	(Japan)	160	(1975)				
Kinglor Metor	Danieli	Buttrio	(Italy)	5	(1973)				
	Ferriere Arvedi	Cremona	(Italy)					40	
				6.06	million tpy	8.33	million tpy	8.81	million tpy

+) Only installations with a degree of metallisation of more than 75% (as at March 1976)

Table 5. Uncritical list of direct reduction projects at the pilot stage (as at March 1976)

Country	Location	Operator	Capacity 10 ³ tpy	Process	Year
<u>I. Middle East</u>					
Abu Dhabi		State-owned company-Kawasaki	1 000	Midrex	1979
Algeria	El Hadjar	S N S	1 200		
Egypt	Dakhella Heluan	State Co-Korf-Itoch-CVRD National Metals Ind.	1 600 800	Midrex	1977 1977
Iran	Isfahan Bandar Abbas Busheher-Kangan Masshad	British Steel Corp-NISIC Finsider-NISIC Krupp-Korf-Klckner-Demag-NISIC NISIC	1 000 2 000 3 000 400		1979 1979 1980 1982
Kuwait	Shuaiba	Kuwait Iron Steel	400		
Libya	Misurata	State Company	400		
Morocco	Nador		400		
Mauretania			600		
Saudi-Arabia	Al-Jebail	Petromar	1 000 2 000		1978
Tunisia	Gabes	Korf-Itoch-CVRD	1 000	Midrex	
approx. 17 million tpy			16 800		
<u>II. Rest of Africa</u>					
Nigeria		State Company	2 000		
South Africa	Pretoria Richardsbay	ISCOR Itesa Corp.	800 200	Rotary kiln process	
approx. 3 million tpy			3 000		

III. Asia, Australia, Oceania

Australia		Broken Hill Prop.	500	
Western Australia			1 500	
Bangladesh		State	400	
India	Palvancha	Andhra Pradesh Ind.Dev.Corp.	50	SL/RN
	Hospet	Chowgule Co	150	SL/RN
	Hissai	Haryana Ind.Dev.Corp.	100	
	Ahmedabad		200	
	Salem		400	
Vijayanagar		Steel Authority of India	100	Rotary kiln process
Indonesia		P.T.Works Ltd.	350	
New Zealand		Glenbrook II	120	SL/RN
Pakistan		Punjab	400	
Philippines		Marcelo Steel Corp./State	400	SL/RN
Singapore		National Iron & Steel Mills (NISH)	400	SL/RN
South Korea		Inchon Iron Works	250	Allis Chalmers
Taiwan		Kaohsiung	400	
Thailand		Bangkok	150	SL/RN
Vietnam (North)		State	400	

approx. 6.3 million tpy

6 270

IV. North America

Canada	Contrecoeur III	Sidbec-Bosco	1 000	Midrex
	Lake St.Joseph	Canad.Bechtel Steep Rock Mines	1 200	
	Whitby	Leasco	300	
	Saskatchewan	Ipsco	400	Midrex
USA	Gulf Coast	Timken, National Steel etc.	1 500	Flor
	Georgetown II	Georgetown Ferreduction	400	Midrex
	Beaumont	Gulf Ferreduction	1 200	Midrex
		Sovereign Ind. Ariz.	400	Krupp
	Gadsden, Ala.	Republic Steel		SL/RN
	Canton, Ohio	Republic Steel	350	SL/RN
		Northw.Steel and Wire.	400	SL/RN
	Duluth	Allis Chalmers		

approx. 7.2 million tpy

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V. Central and South America			
Argentina	San Nicolas Rio Negro Avellaneda Silla Conetlic- urion II	Somisa Nipasa Gutierrez Acindar Sideras Cosigna Usiba II Constr. Jose Mendez Jr. Sider. Mannesmann Acos Finos Piratini Cosipa Dedini CAP Acer. Paz del Rio Kawasaki-Hitauf-Estel State Company Ylor de Venezuela Acelcar Sivenas	600 400 400 400 400 360 270 300 200 200 400 400 500 350 1 200 100 1 500 500 300
Bolivia	Matun		
Brazil	Santa Cruz II S. Salvador Juiz de Fora Belo Horizonte Porto Alegre II Betim(Minas Gerais) Magallanes		1979 1979 1980 1978
Chile			
Colombia			
Trinidad-Tobago	Point Lisas		
Uruguay	Valentines		
Venezuela	C. Guayana Matanzas C. Guayana		1980 1978 1978

approx. 9 million tpy 8 780

VI. Europe			
West Germany	Hamburg II Emden	Hamburger Stahlwerke Norddeutsche Ferroverke Rohrenwerke Bors/Mannesmann	500 1 200 350 400 400 400 800 1 000 280 400 800 400 350 200 200 70 1 300 300 400 400 300 200
France	Lorraine Fos-sur-Mer Dunkirk	Solmer Usinor Hainaut-Sambre	Midrex Midrex Midrex
UK	Llwynpham	Ferrofeed Ltd. (Bispa)	Midrex
Greece		Sider. Hellas	Midrex
Italy	Pollonica	Egna (Solimine) Montalcone Siderndria	Midrex Midrex Midrex
Ireland			
Norway	Karmøy	Sydvaranger	Midrex
Rumania		State Company	Midrex
Sweden	Hälsan	Hälsan	Midrex
Spain	Bilbao, Barcelona Barcelona Alarcitas	Prenasa Predeasa Sider. Gibraltar	Midrex Midrex Midrex
Turkey	Izmir	Edas	Midrex
USSR	Revol Kurek (2nd stage)	State Company State Company	Midrex Midrex

approx. 15.5 million tpy 15 650

Fig.2 shows that the development of sponge iron metallurgy is at present being considerably speeded up. The reasons for this are on the one hand the advanced state of technical development of the direct reduction processes and on the other, particularly in the developing countries, the realisation that cheaper, available energy reserves can be used for producing iron by direct reduction (see Section 2.2). Another reason for this speeding up is the fact that there is not sufficient scrap available to meet the requirements of the present and planned smelting capacities worldwide (see Section 1). The present trend in the production of crude steel, pig iron and sponge iron throughout the world is shown in Fig.3.

The sponge iron from the direct reduction plants at present in operation is used exclusively for internal consumption, so that none is freely available on the market. However, from the planned capacities in Table 5 it can be seen that already several large projects have been planned with a view to producing sponge iron for export to industrialised countries using native energy and/or ore reserves.

If we consider the potential availability of sponge iron in the countries of the European Coal and Steel Community either produced within the ECSC countries or imported from countries outside, after a critical evaluation of the projects listed in Table 5, we arrive at a theoretical figure of about 7 million tonnes per year in the period 1980-85 (Table 6). Of this about 4 million tonnes of sponge iron a year could be produced in the ECSC while it would seem possible to import about 3 million tonnes a year from other countries. One limiting factor is that about 2 million tonnes will only be available temporarily, since the capacity of these plants, which is planned for internal processing of the sponge iron in steelworks, will

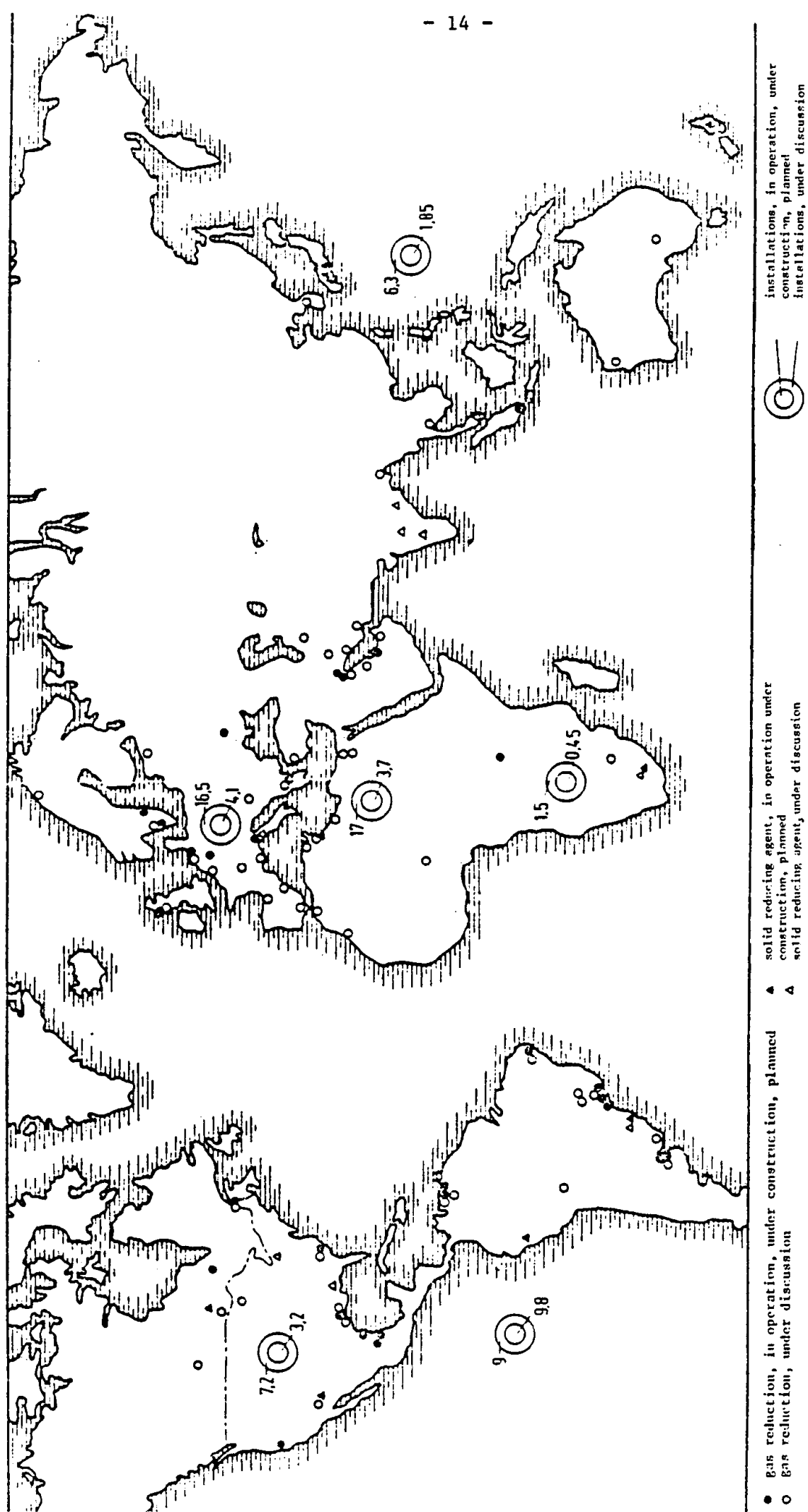


Fig.1. Geographical location of the known direct reduction plants (as in Tables 4 and 5)

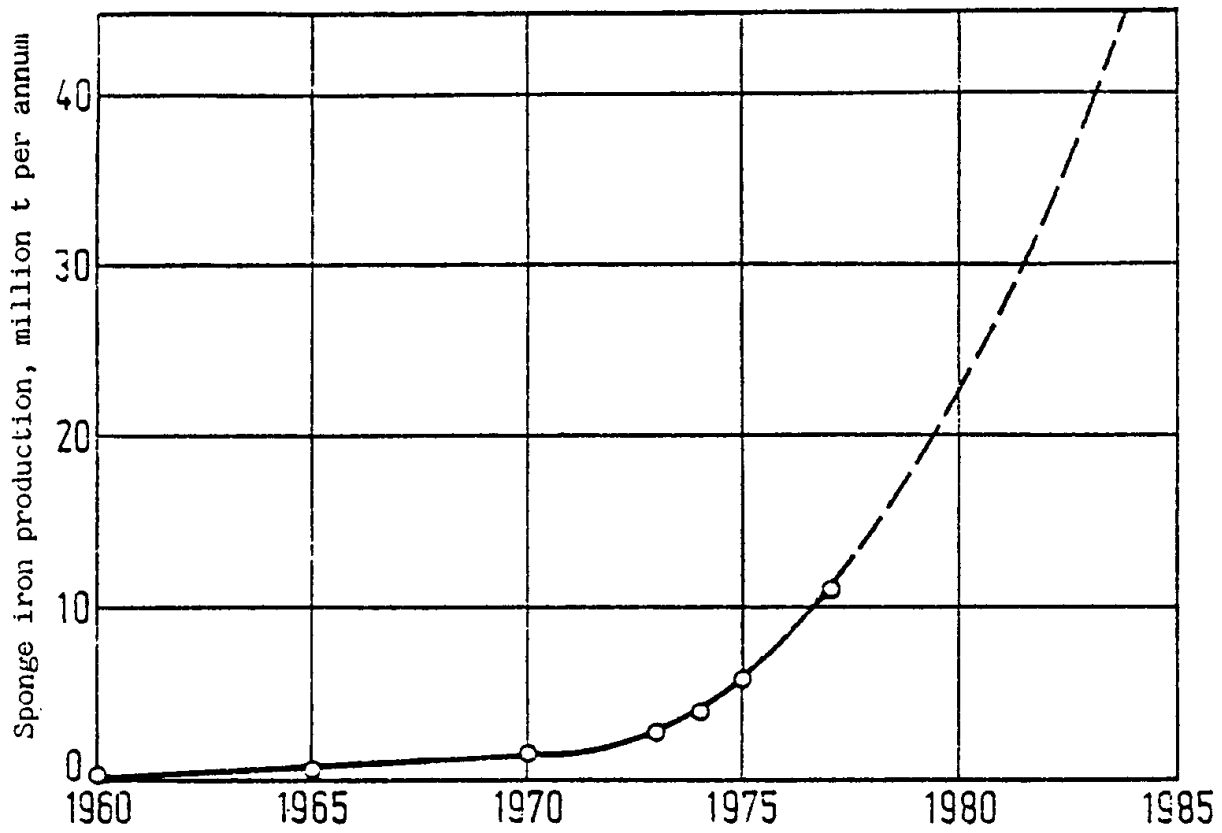


Fig. 2 Development of sponge iron metallurgy

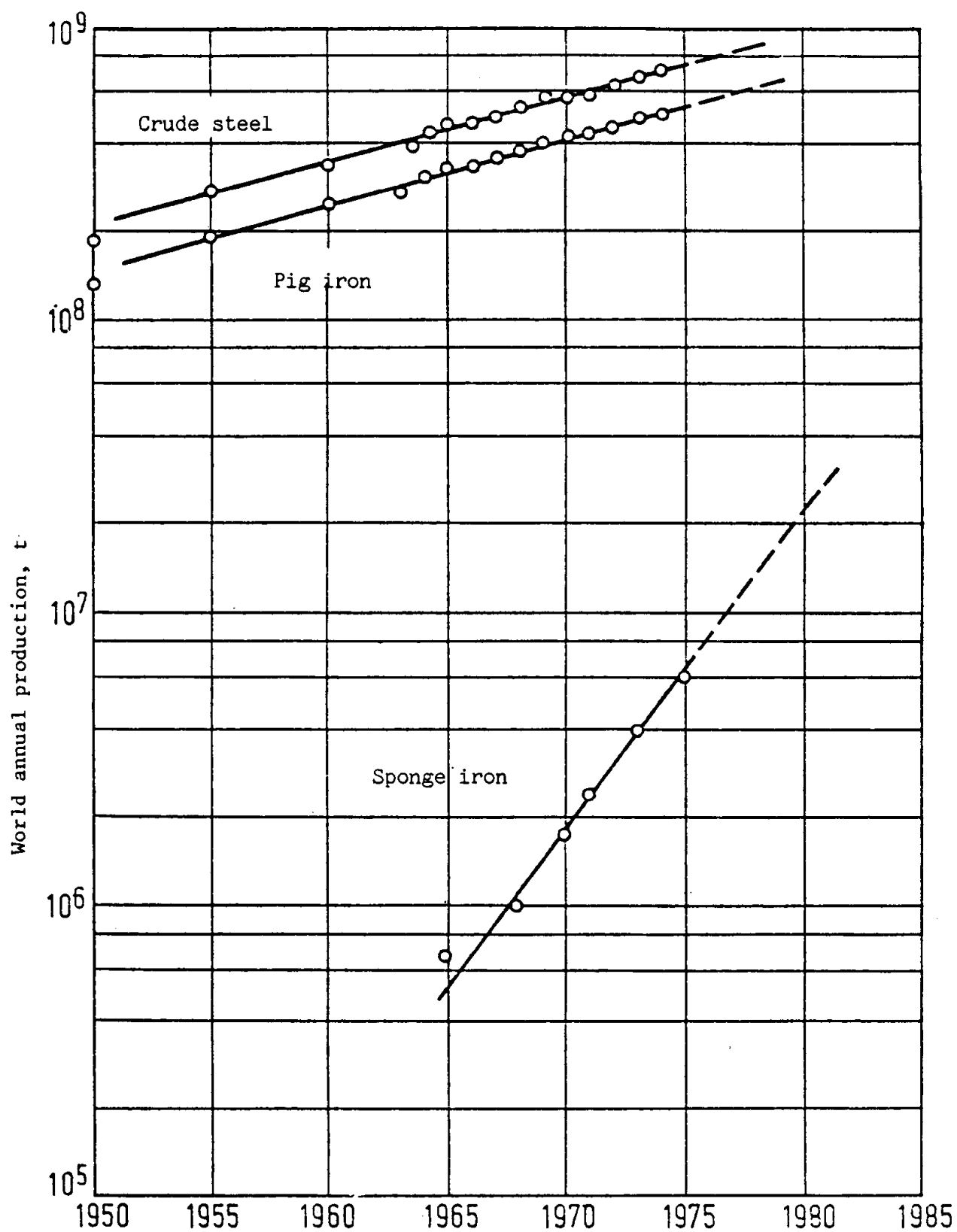


Fig.3. Trends in the production of crude steel, pig iron and sponge iron throughout the world.

only be available for a limited period until the steelworks are completed. However, as in the meantime other projects for direct reduction plants for export purposes should be forthcoming, this estimate does not appear to be unsubstantiated. It must be emphasised, however, that considerable capital will be required for investment to bring these projects into being and also - particularly under European conditions - considerable quantities of high grade ores with gangue contents which give acceptable processing costs when used in steelworks.

However, it seems possible in principle to make up the shortage of scrap already estimated in the countries of the Community with sponge iron.

2.2. Survey of the natural gas and coal reserves throughout the world

The details given in Table 5 on the construction of direct reduction plants give a rough indication of how direct reduction processes will develop up until 1980. In order to assess the situation beyond this date it would appear necessary to investigate the already known deposits of reducing agents (natural gas and coal) as well as ores suitable for direct reduction and to discuss the possibilities of producing sponge iron, taking into consideration the geographical distribution of the deposits as well as possible ways of transporting reducing agents, ores and sponge iron. We have not considered the production of reducing gas from oil since the good transport facilities mean that location has largely ceased to matter.

Table 6. Expected availability of sponge iron for the ECSC
in the period from 1980 to 1985

<u>Production in the ECSC</u>		
Germany	HSW 1	400,000
	Emden	1,000,000
France	Lorraine	400,000
	FOS	400,000
Great Britain	BSC	800,000
	Ferrofeed	400,000
Italy	Solmine	400,000
Ireland		350,000
	Total	4,150,000
		=====
<u>Imports from outside</u>		3,000,000 ¹⁾

1) of which approximately 2 million t available only for a limited period.

Following more recent and accurate studies of world natural gas reserves the estimates have increased from year to year. For instance, according to the figures given by Esso AG, Hamburg, they were 18,6000 billion Nm^{3*} in 1960, 44,978 billion Nm³ in 1970 and 57,910 billion Nm³ in 1973.

Between 1972 and 1973 alone the estimates of natural gas reserves were increased by about 8% or about 4200 billion Nm³. Of these about 2000 billion Nm³ or 48% were in Russia and a further 2000 billion Nm³ in the Middle East, whereas the figures for natural gas reserves in Africa, North America and Central and South America dropped. The estimates for Western Europe were increased by about 440 billion Nm³ or just 9%. The world natural gas reserves at January 1975 were 67,580 billion Nm³. Particularly large finds were reported in Algeria, the USSR and Iran.

Table 7 provides a survey of the known natural gas reserves as at 15th March, 1976 and the nett output in 1975, that is without the amounts pressed back and burned off and without internal consumption and losses. According to these figures the net output of natural gas and petroleum gas in the world constitutes about 2.1% of the reserves known in 1975. Of the world's natural gas reserves Western Europe has 8.7% of which 7.1% is in the EEC countries, the countries of Eastern Europe and the USSR have 37.5%, Africa 9.2%, the Middle East 23.8%, North America 11.9%, Central and South America 4.0% and the countries of the Far East and Australia 4.9%.

Table 8 provides an interesting survey of the possible future use of natural gas and petroleum gas for direct reduction in 11 OPEC countries. According to statistical data published by OPEC approx. 213.5 billion Nm³ of natural

*billion = thousand million

Table 7. Natural gas and petroleum gas reserves and nett world requirement in 1975 (according to information from ESSO AG, Hamburg 1976)

Country	Reserves billion Nm ³	Nett requirement ¹⁾ billion Nm ³
Western Europe	5 550	168.1
W Germany	305	18.2
Denmark	15	-
France	150	7.1
UK	1 415	34.0
Italy	280	15.2
Netherlands	2 400	91.3
Norway	710	-
Austria	20	2.3
Other	255	-
Africa	5 870	11.6
Algeria	3 570	6.0
Libya	745	5.0
Nigeria	1 255	0.1
Other	300	0.5
Middle East	15 260	36.0
Iraq	770	1.0
Iran	9 330	22.0
Kuwait	900	5.0
Saudi-Arabia	2 920	3.0
Other	1 340	5.0
North America	7 600	667.6
Canada	1 510	97.5
USA	6 090	570.1
Central and South America	2 560	45.1
Argentina	205	7.4
Chile	65	2.4
Mexico	340	17.4
Venezuela	1 190	11.5
Other	760	6.4

1)

Without gas pressed back and burnt off and internal consumption and losses.

Table 7. (Continued)

Country	Reserves billion Nm ³	Nett requirement ¹⁾ billion Nm ³
Far East/Australia	3 160	20.8
Afghanistan	100	3.0
Australia	920	4.0
Indonesia	425	4.5
Japan	50	2.8
Pakistan	465	4.0
Other	1 200	2.5
Eastern Europe/USSR/China	24 000	363.5
China	710	23.0
Poland	140	8.0
Rumania	300	27.0
USSR	22 650	288.5
Hungary	80	5.0
Other	120	12.0
World	64 000	1 312.7

Table 8. Destination of the natural gas and petroleum gas produced in the OPEC countries in 1972 (from OPEC Statistical Bulletin 1973)

Country	Nature	Quantity billion Nm ³
Abu Dhabi	gross output nett consumption burnt off	11.215 1.038 4.904
Algeria	gross output pressed back nett consumption burnt off	15.529 1.877 5.027 8.625
Indonesia	nett consumption	4.147
Iran	gross output nett consumption burnt off	41.685 17.757 23.928
Iraq	gross output nett consumption burnt off	7.420 0.935 6.485
Kuwait	gross output pressed back nett consumption burnt off	18.343 1.866 5.126 11.351
Libya	gross output nett consumption burnt off	14.047 7.813 6.234
Qatar	gross output nett consumption burnt off	5.380 1.103 4.277
Saudi Arabia	gross output pressed back nett consumption burnt off	32.568 2.703 2.806 27.059
Venezuela	gross output pressed back nett consumption burnt off	46.020 20.514 10.979 14.527
Nigeria	gross output nett consumption burnt off	17.122 0.273 16.849

gas were recovered in 1972. Of this amount, however, about 61% or 129.5 billion Nm^3 were burned off. With this amount of gas the fantastic volume of 280 million tonnes of sponge iron could have been produced in the OPEC countries alone.

As a complement to Table 7 Fig.4 shows the geographical distribution of the most important natural gas reserves in the world. Comparison of this Figure with Fig.1 shows that natural gas is already being considered for production of sponge iron in North, Central and South America as well as in Europe. Africa, the Middle East and the Far East and Australia. On the other hand, apart from the project in Kursk, no plans have yet been published for using natural gas for direct reduction on a large scale in Eastern Europe, the USSR or China.

The offshore natural gas reserves are becoming increasingly important throughout the world. Proven, recoverable reserves are at present put at about 13,200 billion Nm^3 . Of these about 6,640 billion Nm^3 are in the Persian Gulf, 2,300 billion Nm^3 in the North Sea, 1,061 billion Nm^3 in South-East Asia, 700 billion Nm^3 in the Maracaibo Sea, 540 billion Nm^3 in Canada, 540 billion Nm^3 in New Zealand and 150 billion Nm^3 in the Mediterranean and Adriatic region. In contrast, the total output of offshore natural gas throughout the world in 1974 was 166 billion Nm^3 or approx. 1.3% of the recoverable offshore reserves. Of this gas in offshore areas 70 billion Nm^3 were extracted in the United States, 33 billion Nm^3 in Iran, 29 billion Nm^3 in Italy and 7 billion Nm^3 in Saudi Arabia. The offshore deposits of oil and natural gas are shown in Fig.5. These supplies of natural gas should be of particular importance in realising the plans for direct reduction within the Community.

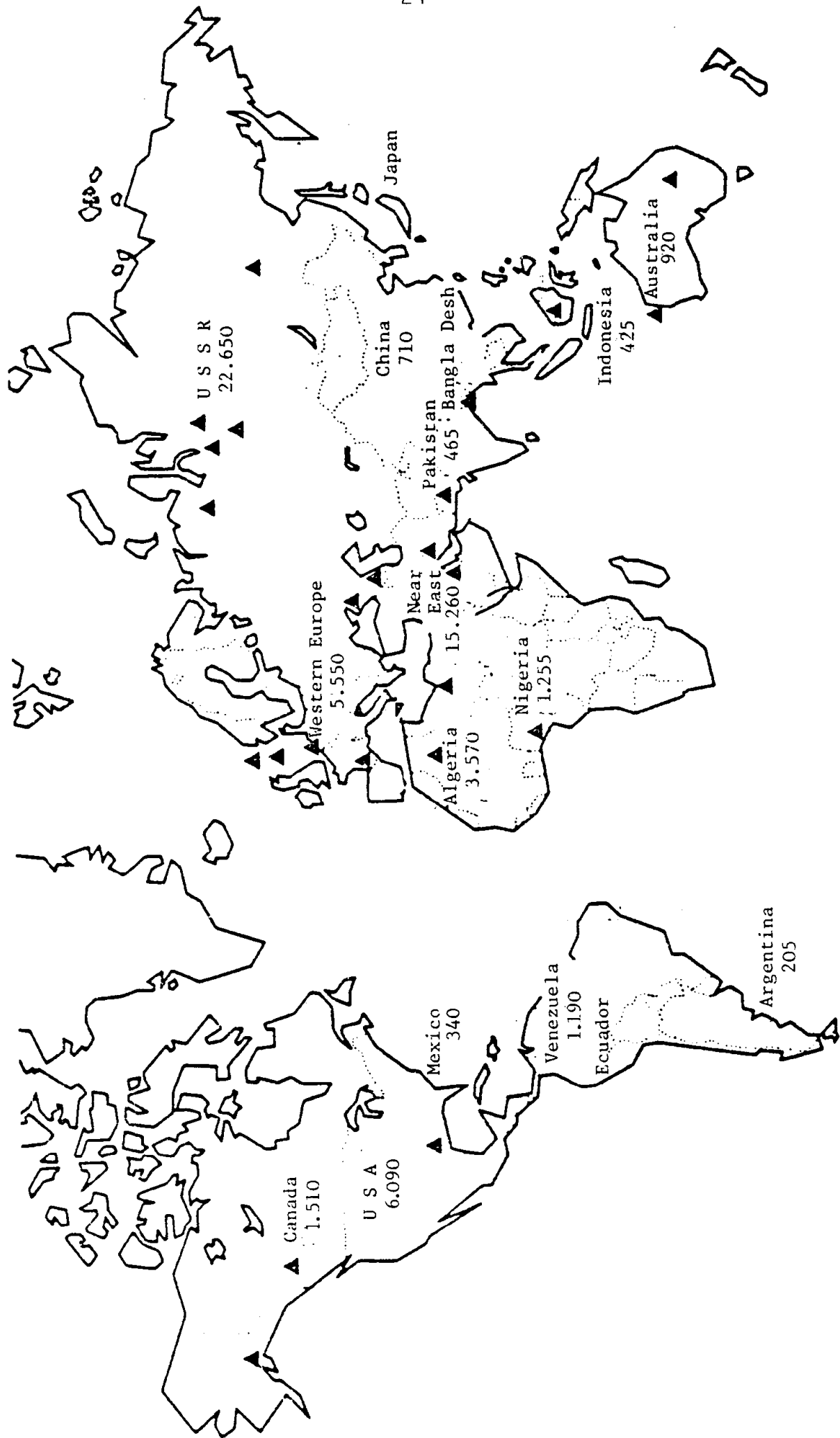


Fig.4. Geographical distribution of the most important natural gas reserves throughout the world (according to information from ESSO AG, 1976) (Figures in billion Nm³)

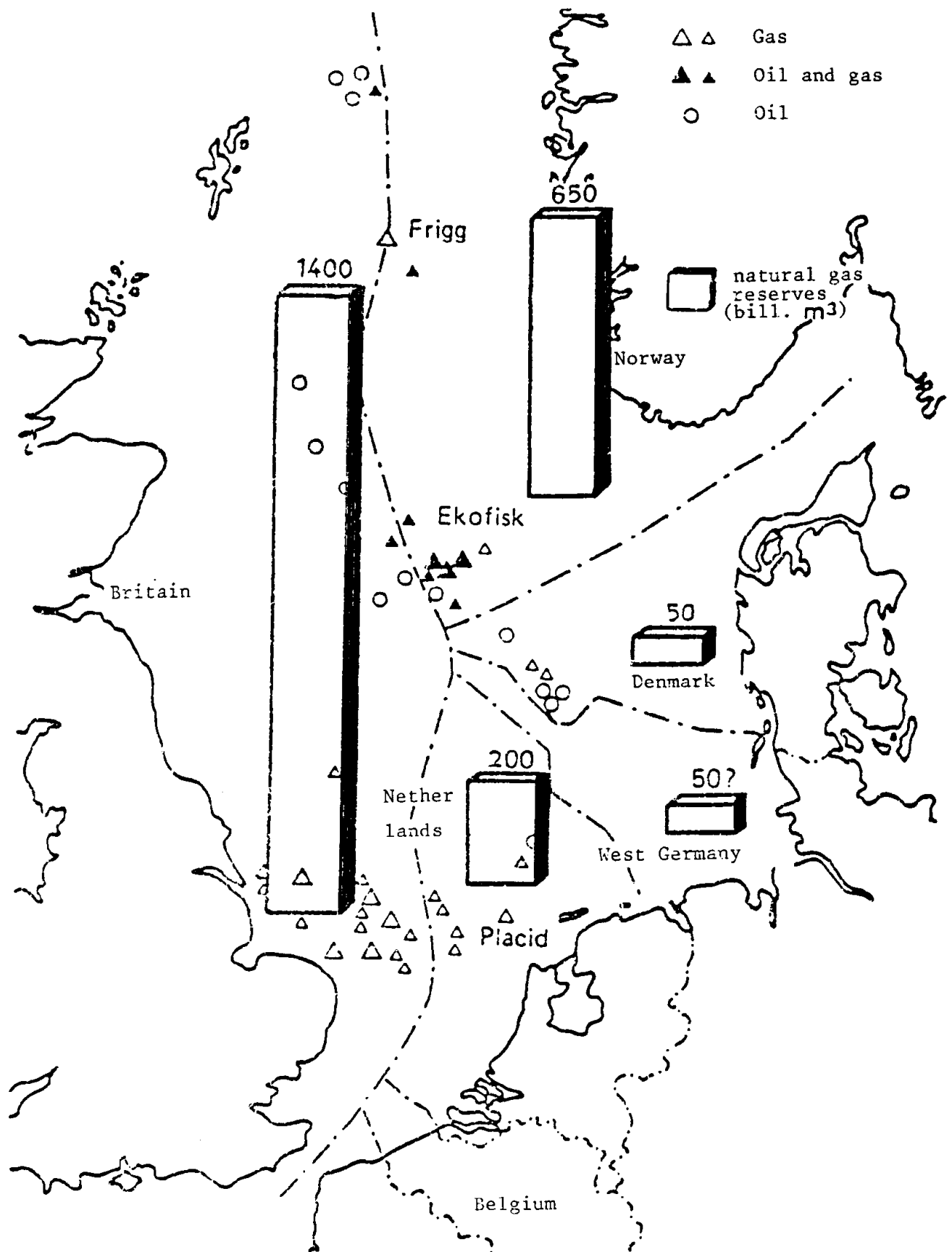


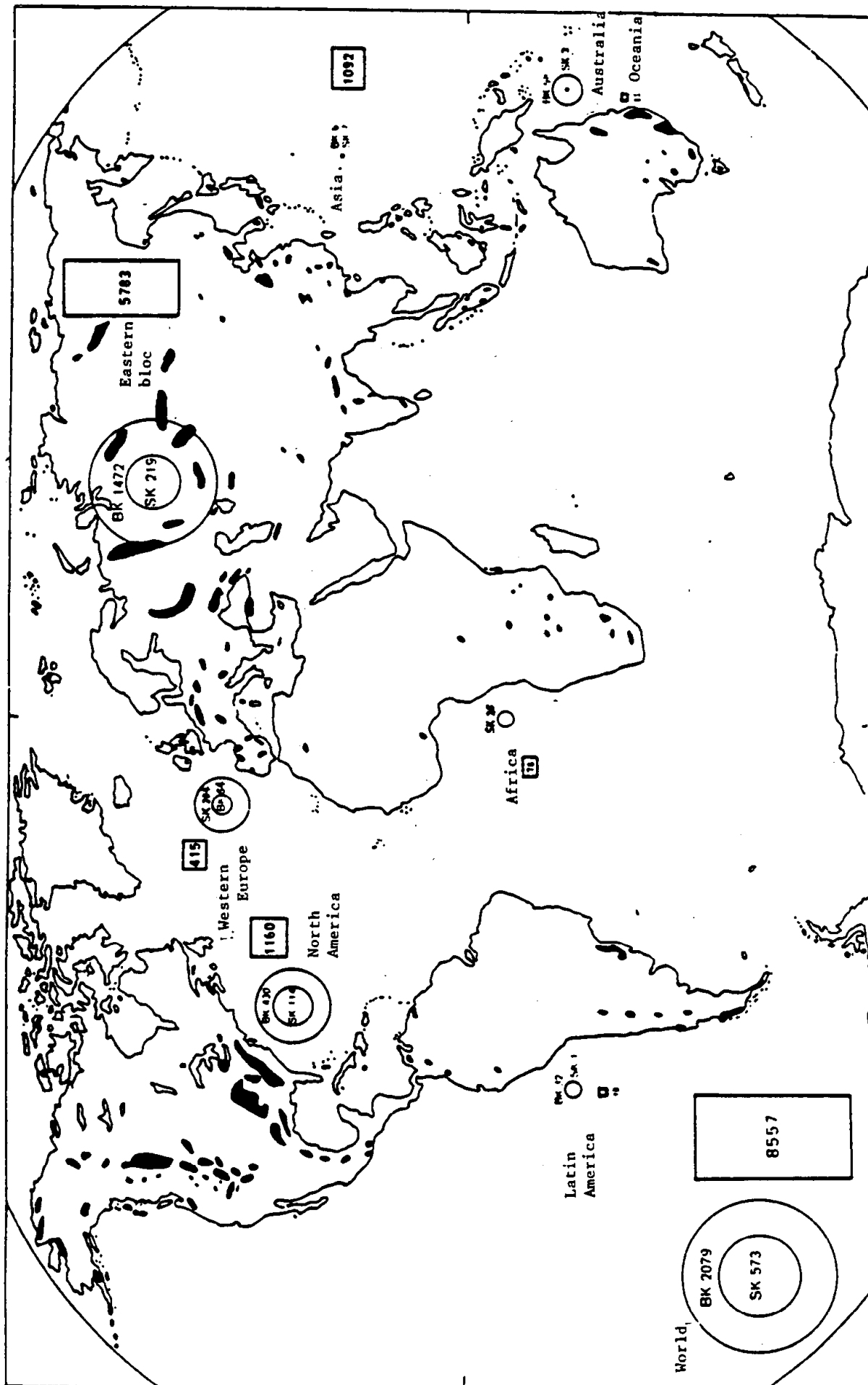
Fig.5. Oil and natural gas fields in the North Sea (according to information from Ruhrgas AG, 1974)

It should be mentioned that the natural gas deposits referred to are of variable quality. Before it is used hydrogen sulphide in particular has to be removed. For direct reduction the nitrogen content has to be taken into account in the processes using recycling of the gas.

Solid carbonaceous fuels are of great importance as reducing agents in rotary kiln processes for the direct reduction of iron ores because world reserves of coking coal are limited. Whereas when development of rotary kiln furnaces started they mainly used coals with a low gas content, such as anthracite and coking duff, it is possible nowadays, at the stage now reached in process technology, to use practically all types of coal the ash of which has a sufficiently high softening and fusion temperature. Because of their high reactivity the types of coal with high contents of volatile matter are of particular interest.

The largest organic energy reserves are undoubtedly those of bituminous coal. At present the total reserves are estimated at about 8550 billion tonnes. It has to be taken into account, however, that some of these coal seams are situated at depths which cannot yet be reached by present-day mining techniques. A depth of 1,200 m is at present regarded as the limit for technically controllable and economic mining operations.

Fig.6 shows the geographic distribution of the world's coal reserves. There are at present proven deposits of approx. 573 billion tonnes of bituminous coal and 2102 billion tonnes of lignite throughout the world. These reserves are divided between the continents as follows:



Estimated total reserves of bituminous coal: [8557] in billion tonnes

Coalfields (Proven reserves of bituminous coal (SK) in billion t.

(Proven reserves of lignite (BK)

Fig. 6. Geographical distribution of the coal deposits throughout the world (after Bischoff, Gluckauf 110 1974, No. 14, pp.588)

	Bituminous coal (billion t)	Lignite (billion t)
Western Europe	204	64
Eastern bloc	219	1472
North America	114	430
Central and South America	1	32
Africa	26	-
Asia	7	6
Australia and Oceania	2	98

These figures should be set against a world output in 1974 of approx. 2.23 billion tonnes of bituminous coal and approx. 838 million tonnes of lignite. The workable reserves in the European Coal and Steel Community amount to about 32 billion tonnes of bituminous coal and about 60 billion tonnes of lignite.

As far as direct reduction is concerned we have the bituminous coal and lignite fields of the Rocky Mountains region, in both the USA and Canada, and also the deposits on the east coast of Virginia and Kentucky. The coal resources of South Africa, including Rhodesia, which are estimated at 79 million tonnes, are suitable for direct reduction, as are in particular the lignite fields of Australia and New Zealand. In South America the coal deposits often have a higher ash content and are not very extensive, but those of Colombia and also Brazil, in the state of Rio Grande do Sul, are important for use in direct reduction (see Fig.1).

Seeing that because of its high reactivity and its basic ash lignite is particularly suitable for direct reduction, selected data on lignite output and reserves in 1974 are given in Table 9. As lignite can largely be mined by open-cast methods, specific costs are usually lower than for bituminous coal.

Table 9. Reserves and output of lignite throughout the world in 1974 (after E Grtner, Braukohle 28 1976, No.5, p.182)

	Lignite reserves billion t	Output billion t
World	approx. 2,625	837.6
Western Europe	80	183.5
West Germany	55	126
Yugoslavia	22	33
Greece	2	14.1
North America	650	17.5
USA	639	14.1
Australia	87	26.3
Eastern Bloc	1,787.4	591.7
East Germany	30	243.4
USSR	1,720	160
CSSR	10	82.2
Poland	15	39.8
Bulgaria	5	24
Hungary	6	22.6
Rumania	1.4	19.7
Africa	0.6	0.4
Others	20.2	18.6
Albania	2	0.9
India	2	3.0
Japan	1.2	0.1
North Korea	5	5.0
Turkey	6	5.7
New Zealand	0.4	2.2
Other Countries in Asia	3.6	1.7
Indonesia	2	0.1
Philippines	0.1	0.5
Thailand	0.2	0.5
Burma	0.3	-

When used for direct reduction in rotary kilns the coal has to meet certain requirements as regards the size of the lumps and dimensional stability during heating in addition to the chemical analysis. While bituminous coal usually fulfils these requirements, lignite may have to be specially prepared by briquetting. Rhine lignite in briquette form has been used successfully in a large scale test⁽⁴⁾.

Apart from the direct use of coal for direct reduction, gasification of coal and use of the gas so produced for reduction processes is of great interest. Suitable processes for the gasification of coal by conventional methods or even by using process heat from high temperature reactors are being developed.

(4) Liestmann, W.D: Stahl u. Eisen 96 (1976) No.3,
pp 97-105

2.3. Availability of iron ores for direct reduction

The purpose of this section is to give an overall view of the ore supply prospects in the short and medium term. Tables and reports are also given on the other principal iron ore deposits in the world.

2.3.1. Specification of ores for direct reduction

The first question to be clarified before going into details concerns the definition of ores suitable for direct reduction. It depends primarily on the method of reduction used: rotary furnace, shaft furnace, retort process or fluidised bed and, consequently, the chemical and physical properties of the corresponding ores. This is a very complex problem which we shall not examine in detail.

On the other hand, the problem becomes much simplified if we consider not the suitability of the ores for direct reduction but the demands imposed by the processing of the sponge iron produced. This can in fact be used in the blast furnace or as a cooling addition in place of scrap in oxygen steelworks and also for melting in electric arc furnaces, likewise as a replacement for scrap. This latter use involves the most severe demands as regards the chemical quality.

It is generally accepted that the amount of slag produced during electric arc furnace steelmaking should not exceed 150 kg per tonne of steel and that the

$$\frac{\text{CaO} + \text{MgO}}{\text{SiO}_2 + \text{Al}_2\text{O}_3}$$

index of this slag should be greater than or equal to 1.5.

In practice, operating with 100% sponge iron, the only scrap being that produced inside the works, these conditions require an

$$\frac{\text{SiO}_2 + \text{Al}_2\text{O}_3}{\text{Fe}}$$

ratio <5%. CaO and MgO are generally not present in sufficient quantities to matter. On the other hand, some of the accompanying elements have to meet relatively strict conditions: it is generally accepted that the phosphorus content of the ores should be less than 0.05%.

They should contain no sulphur or residual metals (Pb, Cu, Zn, Cr, Co), but generally only sulphur is a problem. However, this arises only with some rare ores and anyway the sulphur can be easily removed during pelletising of the ore.

All the ores fulfilling these chemical conditions are called first class ores. The conditions may be less strict if it is intended to use the sponge iron in a smaller proportion in an electric steelworks.

$$\frac{\text{SiO}_2 + \text{Al}_2\text{O}_3}{\text{Fe}}$$

ratios between 5% and 9% may still be considered acceptable if the proportion of sponge iron is as low as 50%. We shall call second class ores those fulfilling the less severe chemical conditions. These chemical conditions are independent of the direct reduction process used. On the other hand the physical form and particle size of the ore may mean that it has to be used in a particular direct-reduction process.

Let us consider the various types of iron ore on the market:

a) Oxide Pellets

This is the product favoured for the continuous shaft furnace processes (Midrex, Armco, Purofer) or batch (HyL) and the rotary kiln processes (SL/RN, Krupp). Moreover, they can only be used in these processes. In addition to their chemical composition, the physical properties of the pellets may be very important in assessing their value. However, no criteria have yet been laid down and for the moment the properties accepted in the blast furnace are generally used.

b) Pellet-Feed

This is generally a commercial designation for a product of 0-100 μm size which can be converted into pellets without crushing or after very little. Its applications are therefore similar to those of pellets, but may be preferred for direct reduction plants having an integrated pelletising plant up-stream. Some designers even envisage using "green", unfired pellets for direct reduction. Pellet feed would then be the preferred feedstock.

c) Sinter Feed

This too is a commercial designation for products smaller than 10 mm, or more generally 6 mm. Normally this product is intended for sinter strands for blast furnaces.

In the field of direct reduction, it is the preferred feedstock for fluidised-bed installations. A maximum proportion of superfines smaller than 50 μm (less than 30%) then seems to be necessary.

Sinter feed can, moreover, be converted by crushing into pellet feed. For direct reduction plants having a pelletising plant preceded by a crusher, sinter feed may be a worthwhile feed, having in particular transport problems which are easier to resolve than pellet feed.

d) Graded Ores

These are products larger than 10 mm or 6 mm for which the upper size limit is very variable. It may be as low as 15 mm (often called natural pellets) and as high as 50 or 70 mm. The most usual upper limit is 30 mm.

After screening to remove the fines formed during transport and handling, these products can be used under the same conditions as pellets. However, their physical properties, in particular their strength, are generally inferior to those of pellets, which causes the production of a high proportion of fines in the reduction furnaces. This fact, combined with the problem of the fines formed during handling and transport, make the feeding of reduction furnaces exclusively with graded ores unlikely at present. Nevertheless, graded ores can, alongside pellets, constitute a certain proportion of the feed for reduction furnaces (shaft and rotary furnaces). Certain technical innovations suggest the possibility of feeding entirely with graded ores.

It is not in the least impossible for graded ores, after crushing, to be converted into either sinter feed (for reduction in a fluidised bed) or even pellet feed. Such a use could appear economically of little value because of the higher price of graded ores. However, as we shall see later, it is

graded ores which, of the natural products without concentration, most often show the best chemical characteristics for direct reduction. This fact, together with the difficulties in using them directly as mentioned above, could militate in some cases in favour of their conversion into sinter or pellet feed.

2.3.2. Technical availability in mines in production

First class ores

Table 10 gives an idea of the technical availability in existing mines corresponding to a

$$\frac{\text{SiO}_2 + \text{Al}_2\text{O}_3}{\text{Fe}}$$

ratio of less than 5% and a phosphorus content below 0.05%.

We have not included in this table the U.S. and U.S.S.R. production which is integrated in steelworks. Moreover it hardly ever meets the criteria quoted.

A comment should be made at the start regarding the analyses given in the table. They relate to typical analyses corresponding to means actually found. On the other hand they are not, in most cases, analyses guaranteed by contract. Thus fluctuations around these mean values, to varying degrees, have to be allowed for, which could be unacceptable in a direct reduction plant. These fluctuations can be greater where they relate to products not obtained by concentration, such as graded ores. Yet, in some still very limited cases supplies are sent to existing direct reduction plants and the contents correspond to effective guarantees. This applies to Swedish and Brazilian pellets and pellet feed from Peru.

Table 10 Mines in production supplying ores with a $\frac{\text{SiO}_2 + \text{Al}_2\text{O}_3}{\text{Fe}}$ index $\leq 5\%$ and a phosphorus content $\leq 0.050\%$

Country	Company	Mines	Reserves 10 ⁶ t	Qualities produced ¹⁾	Production 1973 10 ⁶ t	Fe	SiO ₂	Al ₂ O ₃	P	$\frac{\text{SiO}_2 + \text{Al}_2\text{O}_3}{\text{Fe}}$	Fe content of reserves %
						%	%	%	%	%	%
Australia	BHP	Koolan	45	graded ore	1.9	69.0	0.4	0.2	0.020	0.9	58-66
		Cockatoo	15	graded ore	1.0	67.0	1.5	1.2	0.020	4.0	45-64
	Savage River Mines	Savage River	215	pellets	3.4	67.5	1.3	0.35	0.015	2.4	34-52
Brazil	C V R D	Hematite	850	graded ore	8.0	67.5	1.5	1.00	0.050	3.7	67-68
				graded ore	4.0	67.5	1.2	1.00	0.050	3.3	
				pellet- feed(C)	2.3	67.5	2.0	0.5	0.030	3.7	
				pellets	4.5	67.2	1.7	0.7	0.020	3.6	
				sinter- feed B	13.5	66.0	2.0	1.0	0.040	4.5	50
				pellet- feed(C)	4.7	67.5	2.0	0.5	0.030	3.7	
	M B R		700	graded ore	1.5	68.6	0.33	0.67	0.048	1.4	64-68
				graded ore	2.3	68.4	0.37	0.73	0.048	1.6	
				pellet-feed	3.1	67.6	0.58	0.88	0.071	2.2	
				sinter-feed	3.1	67.5	1.00	1.75	0.080	2.8	
Canada	Wabush Mines	Pointe Noire	>1000	pellets	5.4	65.9	2.83	0.29	0.008	4.7	36
	Hilton Mines	Shawville	40	pellets	0.9	67.2	2.02	0.28	0.007	3.4	18-20
	Wesfrob Mines	Tasu	18.5	pellet- feed(C)	0.5	63.5	1.30	0.43	0.009	2.7	38
Peru	Marcona		967	pellet- feed(C)	1.2	70.0	1.5	0.4	0.050	2.7	45-60
Sweden	L K A B	Malmberget	600	A concentrate	3.2	71.2	0.45	0.25	0.008	0.98	60
				pellets MPR	0.8	67.8	1.45	0.46	0.015	2.81	
				pellets MPS	0.8	68.1	1.12	0.38	0.022	2.20	
South Africa	ISCOR	Sishen	4220	graded ore	3.0	67.0	2.34	0.89	0.038	4.8	66-68

1) (C) = concentrate

1. The total production involved was only 69 million tonnes in 1973, that is about 8% of the total world production of iron ores and 13% of world production outside the U.S.S.R. and the U.S.A. Nevertheless, if this production is relatively low, the main reason is the paucity of outlets.
2. The distribution of production between the various categories of products is as follows:
 - pellets : 15.8 million tonnes
 - graded ores : 21.7 million tonnes
 - pellet feed : 12.2 million tonnes
 - sinter feed : 19.4 million tonnes

It will be seen that pellets and pellet feed represent less than half the total production. As we have said, these are the only products for which the contents can be guaranteed. The other products, sinter feed and graded ores, may be more subject to variation.

3. The most striking point in Table 10 is probably its geographical aspect. Almost all the production concerned is concentrated in two countries, Brazil and Sweden. Countries as important in the world iron ore market as Canada and particularly Australia are hardly mentioned. As regards Australia, the production figures quoted relate to relatively small mines with little reserves and not to the immense deposits of Western Australia.

Second class ores

Table 11 gives an idea of the technical availability in existing mines corresponding to a

$$\frac{\text{SiO}_2 + \text{Al}_2\text{O}_3}{\text{Fe}}$$

ratio between 5% and 9% and a phosphorus content below 0.05%.

Table 11 Mines in production supplying ores with a $\text{SiO}_2 + \text{Al}_2\text{O}_3$ index between 5% and 9% and a phosphorus content < 0.050%
Fe

Country	Company	Mines	Reserves	Qualities, ¹⁾ produced	Production 1973 10 ⁶ t	Fe	SiO ₂	Al ₂ O ₃	P	SiO ₂ + Al ₂ O ₃	Fe content of reserves %
						%	%	%	%	Fe %	
Australia	BHP	Iron Monarch- Iron Prince	113	graded ore	6.7	63.5	3.0	2.0	0.045	7.9	47-66
		Whyalla		pellets	2.0	65.5	3.34	2.10	0.027	8.3	
	Mount Goldsworthy Mining	Mount Golds- worthy	250	graded ore		64.7	4.39	1.10	0.038	8.5	62.8
	Mount Newman Joint Venture	Mount Whaleback	800	graded ore	17.6	65.1	4.37	1.34	0.032	8.8	64
	Hamersley Iron Pty.Ltd.	Tom Price- Paraburdoo	1100	graded ore	17 (1974)	65.75	2.74	1.55	0.038	6.5	63.5
Canada	Iron Ore of Canada	Carol Lake	400 proven	pellets	9.35	65.19	5.47	0.40	0.009	9.0	38
			2000 probable	pellet- feed(C)	3.22	65.52	5.00	0.31	0.009	8.1	
	Quebec Cartier Mining	Cagnon	1050	pellet- feed(C)	9.11	66.00	5.00	0.75	0.025	8.7	33
	Adams Mine	Kirkland Lake	100	pellets	1.19	65.71	4.39	0.12	0.013	7.6	19
	Griffith Mine	Bruce Lake	360	pellets	1.54	67.19	3.11	0.39	0.020	5.2	24
	Marmoraton Mining Co.	Marmora	17	pellets	0.49	65.4	3.61	0.67	0.007	6.5	36
	Sherman Mine	Temagami	90	pellets	1.05	65.55	5.54	0.25	0.018	8.8	21
	Texada Mines Ltd.	Texada	35	pellet- feed(C)	0.51	65.01	4.26	0.76	0.100	7.7	35
Liberia	Lamco	Mont Nimba	264	sinter-feed	5.6	65.4	3.64	1.05	0.050	7.2	
	Bong Mining Co.	Bong Range	500	concentrate	4.66	66.0	5.50	0.30	0.040	8.8	
		Bong Range		pellets	2.35						
Peru	Marcona		967	sinter-feed	4.5	66.0	3.0	0.7	0.030	5.6	45-60
				pellets	3.9	65.2	4.1	0.9	0.020	7.7	
Sweden	L K A B	Kiruna	1800	pellets	2.0	65.96	3.99	0.55	0.043	6.9	60-70
		Svappavaara	300	pellets	2.1	65.60	4.17	0.70	0.023	7.4	35-63
		Malmberget	600	pellet-A)	0.5	65.54	4.88	0.90	0.039	8.8	50
				feed B)		68.12	2.83	0.72	0.055	5.2	
				pellets MPB		66.08	3.96	0.58	0.014	6.9	
	Granges AB	Strassa	100	pellets	0.5	66.12	3.80	0.56	0.003	6.6	31
Mauretania	Miferma	Tazadit	worked out by 1983	ungraded ore 500-200 mm	10.33 for the Miferma complex as a whole	66.2	3.20	0.95	0.020	6.3	-65
				ungraded ore 10-75 mm		65.7	3.50	1.10	0.020	7.0	
		F'Derik		FDA graded ore		66.7	2.50	1.10	0.39	5.4	
South Africa	ISCOR	Thabazimbi	100	graded ore	2.3	66.65	2.45	1.13	0.036	5.4	35-67

1) (C) = concentrate

The total production concerned is much greater than shown in Table 10 and amounts to 109 million tonnes. In addition, the major part of the production of the U.S.A. and U.S.S.R., not included in this table because it is integrated, would probably come into this category. Thus it will be seen that if sponge iron has to represent only part of the feed to an electric furnace, the rest being scrap, the problem of supply will probably not arise.

As regards quantity, this category covers mainly pellets and graded ores, in almost equal proportions; sinter feed is relatively little represented, less than 20 million tonnes. It still remains true that the richest products are either among the graded ores because of a natural selectivity of the ores or among the superfines because of concentration. It is very rare to find sinter feed which is naturally rich, and enrichment is often difficult at this size.

Geographically, two important producing countries which were absent from Table 10 are strongly represented, that is Australia and Canada. However, the ores concerned are very different. In this category Australia produces mainly naturally rich graded ores, in which the fluctuations in contents can be considerable. Canada produces pellets or sinter feed after concentration and the contents could be better guaranteed or even improved if necessary. Canada could even prove better able to supply relatively rich sinter feeds suitable for use in fluidised beds.

Africa is relatively little represented in this class, as in the first. This is due to the importance of non-concentrated sinter feed in African ore production. Some countries such as Mexico and Venezuela do not appear in either Table 10 or Table 11, although they already

Table 12 Mines in production supplying ores with a $\text{SiO}_2 + \text{Al}_2\text{O}_3$ index > 9% and a phosphorus content above 0.050% (including integrated mines, USA & USSR)

Country	Company	Mines	Reserves 10 ⁶ t	Qualities produced	Production 1973 10 ⁶ t	Fe	SiO ₂	Al ₂ O ₃	P	SiO ₂ + Al ₂ O ₃	Fe content of reserves
						%	%	%	%	%	%
Australia	Mount Newman Joint Venture	Mount Whaleback	800	sinter-feed 0-6 mm	9.1	62.0	6.3	2.6	0.038	14.3	64
	Hammersley Iron Pty. Ltd.	Mount Tom Price	500	pellets	2.0	63.1	5.35	3.3	0.050	13.7	64
Canada	Calan Ore Co. Ltd.	Atikokan	190	pellets graded ore	1.22 0.81	63.4 58.7	6.14 5.15	0.98 0.54	0.010 0.022	11.2 9.7	55
	National Steel Corporation		80	pellets	0.71	63.4	8.03	0.59	0.009	13.6	33
	Steel Rock Iron Mines		350	pellets	1.48	62.9	7.37	1.68	0.015	14.4	51
	Wesfrob Mines Ltd.	Tasu	34	sinter-feed (C)	0.40	61.3	6.16	1.37	0.020	12.3	37
Liberia	Liberian Mining	Somi Hills	Closing in 1975	concentrate	0.60	64.5	6.00	0.75	0.042	10.5	45
USA	Pickands Mather	Erie-Aurora	several billion	pellets	13.1	62.7	5.80	0.29	0.015	9.7	20-60
	The Cleve Cliffs	Empire-Mine-Palmer		pellets	3.5	64.4	6.19	0.24	0.013	10.0	
		Republic Marquette		pellets	2.7	64.24	4.61	0.75	0.051	8.3	
	Eveleth Taconite Co.	Eveleth		pellets	2.3	64.4	5.35	0.26	0.017	8.7	
	The Hanna Mining Co.	Burlier-Itasca		pellets	2.65	64.3	4.50	0.20	0.013	7.3	
	National Steel Corp.	National		pellets	2.9	64.2	4.88	0.21	0.009	7.9	
	Jackson County Iron Co.	Black River Falls		pellets	0.75	64.8	5.75	0.13	0.015	9.0	
	Bethlehem Mines Corp.	Cornwall Division		pellets	0.80	63.3	4.30	1.30	0.024	8.8	
		Morgantown Division		pellets	1.50	65.4	3.00	0.71	0.008	5.7	
	Kaiser Steel Corp.	Eagle Mountain		pellets	2.80	64.3	4.05	0.67	0.020	7.3	
	US Steel Corp.	Mountain Iron-Minnetac		pellets	12.6	65.0	5.85	0.16	0.012	8.8	
Brazil	C V R D	See Table 10		sinter-feed	5.5	64.3	4.4	1.7	0.060	9.5	
Mauretania		Tazadit	Worked out by 1983	ungraded ore A	10.33	64.4	4.90	1.25	0.023	9.55	~65
				ungraded ore B	for the Miferma complex as a whole	64.0	5.50	1.74	0.024	10.8	
				TZF graded ore 0-10 mm		63.2	6.10	1.70	0.020	12.3	
		F'Derik		FDB graded ore		64.2	5.80	1.30	0.040	11.1	
Venezuela		Cerro Bolivar	353	graded ore	19.3	63.0	1.50	1.50	0.100	4.6	63
		El Pao	97	for BF	2.5	61.7	1.95	4.60	0.050	10.6	61
Angola	Companhia Mineira do Lobito	Cassinga	150	sinter-feed	2.9 2.9	63.5	5.3	2.02	0.059	11.5	40-61
USSR		Krivoi-Rog Basin Karelia, etc.	> 50000	pellets sinter-feed (C)	200 approx	55-65		2.14	0.02-0.12		
India			5400		35.4	62-65		5.6	0.050		

1) (C) = concentrate

produce ores for direct reduction. In Mexico the output is entirely used locally by existing direct reduction installations. In view of the relatively small ore reserves in Mexico, not only will this situation continue but Mexico will become an importer in the medium term.

In Venezuela the output is also used locally in integrated plants. In addition, the chemical characteristics of the corresponding ore are not those which we have defined for use in the electric furnace, and the sponge iron obtained is used in the blast furnace instead.

2.3.3. Commercial availability - mines in production

With regard to commercial availability, we shall concentrate mainly on the first class ores

$$\left(\frac{\text{SiO}_2 + \text{Al}_2\text{O}_3}{\text{Fe}} < 5\% \text{ and phosphorus content } < 0.05\% \right),$$
 as the other ores are much more plentiful and their commercial problems become similar to the general one of supplying a steelworks based on the blast furnace.

It is in any case more difficult to discover commercial than technical availability. However, the following may be said:

1. Australia

In Australia two mines capable of supplying suitable graded ores from the technical point of view (Koolan and Cockatoo) belong to an Australian steelmaking company (BHP) and the tonnages not used by this company are taken by Japan under contract. Besides, the reserves are small. Nevertheless, there could be some available from 1977 onwards. Moreover, the contents at present guaranteed by contract are much lower than the values given in the Table (iron content 64%, silica 4%).

The Savage River pellets are of very acceptable quality for direct reduction and it is probable that this quality could be guaranteed. On the other hand, it seems that all the output is reserved for Japanese steelmaking companies under longterm contract. The reserves make it difficult to envisage any increase in production to satisfy other needs. Thus the commercial possibilities, at least short term, are poor.

2. Canada

The commercial possibilities are also poor in Canada. The pellets of Wabush Mines are reserved for the shareholders, American and Canadian steelmakers, with a small European holding. Only an increase in production not planned at present could enable other needs to be met. Besides, the chemical quality is a little limited; the pellets have in addition to silica and alumina relatively high manganese contents which could make them unsuitable for some applications.

The other Canadian ores, which are technically acceptable, are of very little commercial interest. The mine at Shawville will close in 1975. The mine at Tasu, in British Columbia, would find it difficult to send its ore elsewhere than Japan. Besides, production and reserves are very limited.

3. South Africa

South African ore is still in fact very little exported. In addition the guaranteed contents are likely to be very far from the mean contents given in Table 10. We shall return to South Africa and the Sishen mine when discussing projects.

4. Brazil, Peru, Sweden

We have grouped these three countries together since they seem in the immediate future to be of the greatest commercial interest. As we have seen, Brazil is by far the largest producer of iron ore of the class concerned. A large quantity of pellets and partially graded ores is already charged to direct reduction installations; in view of the nature of the Brazilian deposits it is probable that the variation between the contents quoted and the guaranteed contents is slight. On the other hand, the production quoted is already sold under contract to various Japanese, European or American customers, mainly for conventional steelmaking. Nevertheless, the size of the reserves and the plans of CVRD, to which we shall return, mean that this country certainly remains the most capable in the short term from the commercial point of view of supplying products of suitable quality for direct reduction. This appears true for all categories of ores (graded ores, pellets, pellet feed and sinter feed). For graded ores Brazil is one of the few countries, perhaps the only one in the world, to be able to guarantee such high contents in a non-concentrated product. For sinter feed, pellet feed and pellets, the existence of high contents is due to the possible techniques for enriching Brazilian ores. As far as individual companies are concerned it seems that MBR is better able to conclude immediate agreements than CVRD.

Sweden is also a large supplier of very rich feedstock for direct reduction. It is obtained by concentration after fine crushing and so can cover only pellet feed or pellets. Present production is still limited and certainly already reserved by existing installations, in particular in Europe and Canada. It seems, however,

that if need be the additional commercial possibilities would be very easy to develop. As an example, some pellet feeds obtained by concentration are considered too rich for making conventional pellets for blast furnaces and are artificially made leaner by adding sand. Thus the production of rich pellets for direct reduction could easily be increased.

Peru likewise already supplies concentrates for direct reduction. The total production is intended for Oregon Steel in the USA where the concentrate is pelletised. Thus there seem no immediate commercial possibilities. It is possible, however, that a simple modification in the treatment schemes could provide new commercial possibilities.

To sum up, it seems that the commercial possibilities in the short term are limited to Brazil, Sweden and Peru. Peru could only supply pellet feed, Sweden pellet feed and pellets. Only Brazil would be able to supply, in addition to the pellet feed and pellets, very rich graded ores and perhaps sinter feed. The exact quantities available could only be less than those given in Table 10 and could only be discovered more precisely by direct enquiry to the companies concerned.

If we limit the quantities commercially available to pellets, which would be the case for direct reduction installations without pelletising plants, commercial availability becomes very low and in any case much less than the 15.8 million tonnes of pellets in Table 10.

2.3.4. Technical and commercial availability - projects in hand and under consideration

The lack of precision which is difficult to avoid in statistics relating to mines in operation becomes still greater in projects. This imprecision can be at several levels:

- degree of certainty as to whether the project will in fact be carried out,
- date it is put into effect,
- degree of knowledge of the analyses which can only be estimated and not known from commercial production figures.

Nevertheless, Table 13 gives the list of mining projects which we estimate will be able to produce first class ores for direct reduction. Capacities of the order of 90 to 150 million tonnes have been announced. However, these figures should not delude us; at least part of the production, particularly the sinter feed, will probably not meet the content requirements laid down.

This being so, we find that the major part of the production envisaged consists of pellets. Moreover, the largest, and also the most definite, of the projects are still confined to Brazil and concern numerous Brazilian companies. Outside the Carajas project, for which the date for effective fruition is still very debatable, the other projects seem almost certain.

Another very large project outside Brazil is that of Mifergui in Guinea. This involves an ore which, exceptionally, is said to be naturally very rich. This means that the products sold could be graded ores or sinter feed.

Table 13. Projects: qualities produced with $P < 0.05\%$ and $\frac{\text{SiO}_2 + \text{Al}_2\text{O}_3}{\text{Fe}} < 50\%$

Country	Company	Mines	Reserves 10^6 t	Quality produced	Production 10^6 t	Remarks	Fe content of reserves %
Canada	Quebec Cartier	Fira Lake	400	pellets 67% Fe	6	Start of operation planned for 1977. 50% of the output is for Contracoer. Feasibility study in progress. Starting date planned for 1979.	35
	Steep Rock Iron Mines	Lac St. Joseph	700	oxide pellets	3		23
				sponge iron pellets 92% Fe	1		
Guinea	Miferqui	Nimba	1450	graded ore sinter-feed	15	Feasibility study in progress by L K A B - production planned for 1978-79.	68.9
Brazil	C V R D	Tubarao	2700	pellets	20-25	Expansion of existing mines, construction of 5 pelletising plants with various partners - start of production between 1976 and 1979. Future expansion planned after 1979.	66.7
	Amazonia Mineracao	Carajas	15750	sinter-feed graded ore	< 50 10-36 2-9	Start of production planned for 1979 with 12 million tpy rising to 45 million tpy around 1985. It is possible a pelletising plant may be built at the port of Itaqui.	
	Samarco Mineracao	Alegria	16000	pellets pellet-feed	5 5	Start of production 1977	50-64
	M B R	Agua Claras	375	graded ore sinter-feed	 12-25	12 million tpy in 1975 17 million tpy in 1977 25 million tpy thereafter Studying a pelletising plant of 3-9 million tpy.	64-68
	Ferteco Mineracao	Fabrica	62	pellets	2.5	Start of production 1976	
Venezuela	Orinoco Mining	San Isidor	677	sinter-feed pellets	3 2.5	Expansion planned in the future to 10 million tpy ore + pellets	55-68
Alaska	Mitsubishi US Steel	Klukwan	1000	pellets	3.5	Deliveries will be made 5 years after the start of building, which is not in progress.	15-50
	Marcona Corp.	Snettisham	1300	pellets	7		
Norway		Sydvaranger	100	pellets	3	Increase in production and improvement of the iron content of the pellets.	38

The Canadian projects are much smaller, but one of them, that at Fire Lake, is very certain. In contrast some projects such as those in Alaska still remain very debatable.

Commercial availability is, of course, still more difficult to assess precisely than technical. It can nevertheless be said that the large projects for producing pellets in Brazil are practically integrated, large steelmaking companies being generally both shareholders and purchasers. Most of the large mining projects (Mifergui, Alaska) are in a similar position, but it is very probable that as these projects progress some tonnages will really become available on the world market.

2.3.5. Ore requirements for direct reduction

Faced by the situation in which, as we have seen, the availability is sometimes very difficult to assess precisely, from both the technical and commercial points of view, it would be useful to set a level for present and future needs of direct reduction plants. In December 1974 the production of sponge iron was estimated at about 6 million tonnes. If we deduct the figures corresponding to very special products (by-products, titaniferous ores) the production of sponge iron from iron ores is only 4.3 million tonnes which requires about 6.2 million tonnes of iron ore. This figure is fairly low if we compare it with the possible production of close to 69 million tonnes of first class ore, of which 15.8 million tonnes are pellets. Again, in considering availability we have not taken account of either Mexico or Venezuela, where production is used locally. Thus it would seem that, for the immediate future, there is no supply problem.

In the very near future the production of sponge iron will rise from 6 million tonnes to more than 20 million tonnes. If we again exclude production based on special ores (South Africa, New Zealand) or by-products from steelworks, the expected production drops to about 17 million tonnes up until about 1979.

The production of sponge iron from iron ores in the free world will only be 14.5 million tonnes, which would require about 21 million tonnes of ore. This figure becomes large compared with the 69 million tonnes possible present production of very rich ore. If we also take into account the fact that in most cases the requirements will not be for graded ores but primarily for pellet feed or pellets, the production of which is only 28 million tonnes, the balance of supply and demand no longer seems very sure. (We have, however, excluded Mexico and Venezuela which could probably supply their own direct reduction plants, which would improve the statistical situation a little).

In the longer term (by about 1984) the production of sponge iron is estimated at about 50 million tonnes outside the USSR. The corresponding ore requirements would be about 80 million tonnes. If all the mining projects quoted materialise, as regards quantity and quality, there would probably be no supply problem. Otherwise this problem looks serious in view of the present capacity. That is why we shall deal in the next section with the technical possibilities of producing iron ores with a very high content, emphasising those which have not at present materialised in projects of varying precision.

2.3.6. Technical possibilities of extending the production of high quality iron ores outside the projects quoted

A distinction will be drawn between two groups - those where production is possible from natural ores without concentration and those requiring concentration techniques of varying degrees of complexity.

1. Production from natural ores without concentration
(or with very limited concentration)

In practice it is very difficult to find in this category any really new ores which have not been considered already. Primarily we have the existing production of Brazilian ores and that planned at Carajas. The ores corresponding to the qualities sought are above all possible as graded ores. Some sinter feeds obtained by size grading, removing the finer products, may also be possible. We have also put in the category of natural ores suitable for direct reduction the ore of Mifergui in Guinea.

There are two large producer countries for naturally rich ores the position of which seems weaker in the field of direct reduction possibilities - South Africa for the Sishen deposits and Western Australia. We have already included South Africa in the countries now supplying ores of the desired quality. It is possible that this production can be increased in the future, but only as regards graded ores. Western Australia seems even less able than South Africa to supply in future graded ores chemically suitable for direct reduction.

Venezuela would be able to supply graded ores and sinter feed from the San Isidro deposits but available supplies will be absorbed by the Venezuelan direct reduction projects.

The possibilities of production from natural ores without concentration thus seem considerable and the reserves immense. However, they have the following serious drawbacks:

The content of the ores can only be uniform and guaranteed if particular care is exercised from the mines onwards. This is the more serious since, for a long time, direct reduction plants will still be of relatively modest capacity and any irregularity in content cannot be compensated by mixing and stockpiling.

A particularly troublesome feature of natural ores is the phosphorus content. It often exceeds 0.05%. All these reasons lead to poorer possibilities of producing ores for direct reduction than the gross figures would suggest.

A second drawback in using natural ores for direct reduction is that they are generally graded ores. We have already indicated that using these ores direct could raise serious problems. In contrast, conversion of graded ores easily usable in the blast furnace into pellets would not most of the time be very attractive economically. That is why we shall examine the technical possibilities of producing ores for direct reduction by concentration.

2. Possibilities of producing ores for direct reduction by concentration

The concentrates used for direct reduction are often called superconcentrates. A distinction should be drawn between two very different cases. In the first the ore is too lean or contains an inhibiting impurity such that it has in any case to be enriched even for use in the blast furnace. A superconcentrate can even

sometimes be obtained directly in this way (some Swedish ores). In the second the ore is usable as-mined for the blast furnace and the enriching operation is only necessary for direct reduction. Of course, the economic value of concentration is then more debatable.

Ores already enriched at present

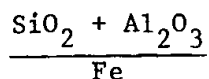
In the first category of concentrates easily adaptable to direct reduction we have the Swedish and Norwegian concentrates enriched by wet magnetic separation at low intensity which already constitute industrially a source of ores for direct reduction. These concentrates could only be supplied in the form of pellet feed or pellets. The reserves are several hundred million tonnes and the possibilities of production by adapting concentrates at present produced for the blast furnace are easy.

Peru seems in a very similar situation. The ore is naturally fairly rich, but the high sulphur contents make enrichment essential and will do so more and more. The reserves are considerable, several hundred million tonnes. The products will likewise be in the form of pellet feed or pellets.

Among the concentrates which can be easily adapted to superconcentration are a number of projects based on magnetite deposits:

- Man project on the Ivory Coast
- Wologosi project in Liberia
- Kudremukh project in India

These three projects appear elsewhere in the list of mining projects supplying ores with a



ratio between 5% and 9% (Table 14) but a drop in this ratio seems possible.

Table 14 Projects: Qualities produced with $P < 0,050\%$ and $SiO_2 + Al_2O_3$ between 5 and 9%
Fe

Country	Company	Mines	Reserves $10^6 t$	Quality produced	Production $10^6 t$	Remarks	Fe content of reserves %
Canada	Quebec Cartier	Mt Wright	3500	sinter- feed(C)	16	Production should start during 1975	31.4
South Africa	I S C O R	Sishen	4200	graded ore	6-30	Start middle of 1976 with 6 million tpy 10 million tpy in 1977-78 20 million tpy from 1980 onwards. 50-60 million tpy subsequently	66-68
	Consolidated Af. Mines	Gamagana Ste Croix	170	graded ore	7	First deliveries planned for 1977	
Ivory Coast	Intern. Consortium	Mt Klahoyo Man	1037	pellets	6-9 or 12	Building will start in 1976. Deliveries in 1980.	36.3
Liberia	L I S C O	Wologisi	1269	pellets	10	Production should start 1979-80	35.5
Angola	Cie do Mang- anese de Angola	Cassala	90	pellets	3	Production should start 1977	33-35
India	N M D C	Kudremukh	600	pellet-feed	7.5	Start of deliveries 1978 Start of working planned for 1978.	39
			155	graded ore	1.6		
		Donimalai		sinter feed pellets	1.8		
	Chowgule Metal Ind.	Chowgule	42	pellets	2	Start of deliveries 1977 Project in progress production planned around 1980	64-67
		Bailadila	316	pellets	4		
				graded ore	8		
Brazil	Ferteco Mineracao	Fabrica	62	graded ore	0.8	Starting mid-1976	
Australia	Goldworthy Mining	Marra Mamba	750	pellets	3	Start of deliveries 1977-78	57-62
				graded ore	8.2		
	Mt Newman Mining	Mt Whaleback	800	sinter-feed graded ore	6.8 12-14	The extension project allows for a production of 41 million tpy in 1976	64
				sinter-feed	23-28		
	Northern Mining	Weld Range	97		5-10	A production of 5 million tpy is planned in 1975 and 10 million tpy later.	58.6
Venezuela	Orinoco Mining	San Isidor	677	graded ore	~ 3		

1) (C) = concentrate

Table 15 Projects: qualities produced with $P < 0.050\%$ or $SiO_2 + Al_2O_3 > 9\%$ (including integrated mines, USA & USSR)
Fe

Country	Company	Mines	Reserves $10^6 t$	Quality	Production $10^6 t$	Remarks	Fe content of reserves %
Canada	Canadian	Julian und Jubilee	738	pellets	12	Project under a study	
South Africa	I S C O R	Sishen	4200	sinter-feed	4-20	Start mid 1976-10 million tpy in 1977-78, 20 million tpy from 1980 50-60 million tpy later	66-68
	Consolidated Af. Mines	Gamagara Ste Croix	170	sinter-feed	5	First deliveries planned for 1977.	
	Palabora	Palabora	70	pellets or pellets pellet-feed	6.6 4.4 2.2	Feasibility studies are in progress and it may be built in 1976	
Liberia	Bong Mining	Putu	> 100	pellets	8	Production around 1982	39
	LMC-NIOC	Bee Mountain	-1000	pellets	8	To replace Bomi Hills which will cease operation at the end of 1975	45
Gabon	Somifer	Mekambo	860	sinter-feed pellet-feed	15	Production around 1982-83	62-65
Angola	Cia Mineraria do Lobito	Cassinga	2000	pellets	4	Start of production has not been determined	
Mauretania	Miferma SMIN	Guelbs	estimated 2000 certain 300	sinter-feed pellet-feed	2 4 8 12 24	in 1980 and 1981 in 1982 and 1983 from 1984-86 from 1987-89 from 1990	30-45
Brazil	Ferteco Mineracao	Fabrica	62	sinter-feed	0.8	Start mid 1976	
	Minas de Sera Geral	Campanema	233	graded ore sinter-feed	2 4	Start of production in 1977	56-64
Australia	Marandoo Mining	Marandoo	221	graded ore sinter-feed	12.5	Production planned for 1979	
	Mc Cameys Iron Ass.	Mc Cameys Monster	260	graded ore sinter-feed	5-10	Held over	61-62
USA	Cleveland Cliffs	Empire				Extension of the production of pellets from 3.5 to 5.3 million tpy in 1975	36
		Tilden	900	pellets	4	The capacity is 10 million tpy. Started in 1974. Production of concentrates or oxide or sponge iron pellets.	
		Carter Creek	30				
	Eveleth Taconite	Eveleth				Extension of the production of pellets from 3.6 to 6 million tpy in 1976.	
	Hibbing Taconite	Hibbing				Building of a new pelletising plant which will produce 5.4 million tpy from 1976 on and 8.1 million tpy in 1978.	
USSR	Inland Steel	Minorca				Building of a pelletising plant of 2.5 million tpy. Start April 1977.	
	National Steel	Keewatin				Extension of the production of pellets from 2.5 to 6 million tpy 1977.	
	US Steel	Virginia				Extension of the production of pellets from 12 to 18 million tpy 1978.	
USSR		Kostamus (Karelia)	2000	pellets	8	Starting working 1978 or 1980. Part of the production will come on to the world market.	31
		Kursk (Krivoi-Rog)		pellets		A large pellet capacity may be expected. Part will be offered on the world market. Part of the production will be sponge iron.	

The situation in Canada is a little different. In this connection we will only consider the large deposits of the Labrador basin. (The Steep Rock deposits in Ontario, where the reserves are relatively large, already appear among the projects capable of supplying ore for direct reduction in Table 13). The existing workings (Quebec Cartier, Wabush) or projects (Mt Wright, Fire Lake, Lake Julian) are characterised by the production by gravity of relatively coarse concentrates 0-1 mm which can be used either direct as sinter feed or after re-crushing as pellet feed and pellets. In the present situation these concentrates or pellets have 5-6% SiO_2 and their use as-supplied is only possible in limited proportions (thus the Carol Lake pellets are already used in the direct reduction plant at Sidbec). However, it seems possible to increase their iron content.

One way already in use at Wabush is to re-treat them without re-crushing by electrostatic separation. The silica content is thus reduced to 2%. This way has the advantage not only of supplying a pellet feed without re-crushing, but also of supplying directly a feedstock for fluidised bed direct reduction installations.

Another possible way of improving the content of the concentrate would be re-crushing followed by flotation. This way can only be adapted to pellet feed production and has not yet been studied in detail. It would probably have the advantage of being more efficient and reliable than electrostatic separation.

The situation in Brazil is also similar. The major part of the concentrates obtained in the wet magnetic separation at high intensity by treating itabarites already meet

the desired specifications. The particle size of these concentrates (0.020-0.6 mm) allows them to be used either as pellet feed after re-crushing or for feeding fluidised bed installations.

Some African deposits also seem of a type which can be easily adapted to produce concentrates which meet the requirements of superconcentration. The following may be quoted in this connection:

- the Guelbs deposits in Mauretania which could, by adapting the treatment arrangements, supply part of the production in the form of superconcentrates;
- the Marampa deposits in Sierra Leone (the reserves seem small).

Finally, among the prospects which are possible but not demonstrated, two countries could adapt their present production of concentrates for producing pellets for the blast furnace to richer products for direct reduction. These are the United States with the magnetic taconites of Minnesota and the U.S.S.R. with the similar ores of Krivoi-Rog and Kursk.

In both cases the reserves are immense, particularly in the Soviet Union. The concentrates are at present obtained by wet magnetic separation at low intensity after fine grinding. They still contain 5-8% silica. Superconcentration is possible and has been studied on a laboratory and pilot scale for the American and also, it would seem, for the Soviet concentrates. It requires the relatively expensive flotation method. In the United States no industrial plant is planned. In the U.S.S.R. it is possible that this type of method is planned for supplying the future direct reduction complex at Kursk

(2.5 million tonnes per year). It seems, however, from certain information, that obtaining superconcentrates directly by simple magnetic separation is possible in the U.S.S.R. from ores from the Kursk magnetic anomaly, where the reserves are immense.

Of course, the commercial possibilities on the world market are debatable, but the possibilities of large internal supplies which result are worth mentioning.

Possibilities of producing ores for direct reduction by concentration from natural ores marketable as-mined

Some deposits produce ores which naturally have 5-8% $\text{SiO}_2 + \text{Al}_2\text{O}_3$ and are sold directly, although after more or less complex treatment this can be brought down to a much lower level.

Of this type, we have the Brazilian ores of Minas Gerais. In particular the Brazilian sinter feeds sold by CVRD under the designation of "common fines" and which have 6.1% $\text{SiO}_2 + \text{Al}_2\text{O}_3$ could probably be brought down to a lower level, even after a relatively moderate crushing to about 1 mm and treatments such as size grading and high intensity magnetic separation.

However, the technical possibilities of superconcentration should not be generalised. Many natural ores and concentrates cannot be brought to a content higher than that at present obtained, at least not without recourse to chemical or metallurgical treatment, not likely in practice for economic reasons, even if the technical possibilities were demonstrated. By way of an example, such large producers of iron ores as Western Australia and India cannot envisage superconcentration of most of their ores for technical reasons.

General Conclusions

Iron ore resources suitable for use in direct reduction followed by steelmaking in an electric furnace are plentiful in the world and there should be no supply problems in the long term. These reserves are distributed as follows:

The naturally rich ores without treatment are very localised geographically:

The most important deposits of this type are found in Brazil both in the area at present exploited by various companies in the Minas Gerais and in the planned operation of Carajas in Amazonia. In both cases, the reserves are several billion tonnes and the marketable ores could be either graded ores, pellet feed or pellets.

Another naturally rich ore deposit could be that of Mifergui in Guinea, the exploitation of which is planned. Its reserves are likewise very large. The possible suppliers also include Venezuela.

South Africa could also supply ores suitable for direct reduction, but the technical possibilities seem more debatable.

The ores which could be made suitable for direct reduction after concentration are still more numerous. This category includes Brazil with the large itabarite deposits of Minas Gerais which are, moreover, already enriched at present and can provide rich sinter feeds or pellet feeds.

Sweden has magnetite-based deposits which can easily be enriched to give very rich concentrates and is already a large supplier of pellets for direct reduction. Peru is in the same position.

Other countries could likewise supply rich concentrates by simple magnetic separation:

- the Ivory Coast for the Man deposits
- Liberia for the Wologosi deposits
- India for the Kudremukh deposits
- Mauretania (in part) for the Guelbs deposits

which are all deposits of several hundred million tonnes.

Canada is able to adapt its production of concentrates for the blast furnace to the requirements of direct reduction by means of simple modifications.

It could be the same for the United States, with the magnetic taconite deposits of Minnesota, and the U.S.S.R. with the similar deposits of Krivoi-Rog and Kursk. However, in both these cases the adaptations seem more difficult and would have to be developed on an industrial scale.

Although the longterm statistical situation for ores for direct reduction appears good, the same is not necessarily true for the medium and short term, for several reasons:

- 1) The producing countries concerned are much less well distributed geographically than the iron ore producers as a whole, a number of large producers like Australia, India, and most of the African countries being unable to supply suitable products, in view of the nature of their deposits.
- 2) Some ores are suitable for direct reduction on average, but the guarantees required make them difficult to use and may necessitate selection of the ores at the mine. The total production available for direct reduction would thus be much less than the total production. Besides,

the adaptation of these naturally rich ores to stricter contractual requirements would certainly necessitate a transition period of varying length according to the particular case.

- 3) For deposits which can supply ores suitable for direct reduction, after concentration or superconcentration, the adaptation period is still longer. Besides, the investments required probably imply participation or longterm contracts from the prospective customers.
- 4) In the short term, it is probably the problems of adapting quality to requirements which appear the most urgent. In fact most of the direct reduction projects are based on shaft furnaces. The feed for these may be composed of graded ores but these pose problems of metallurgical quality (even for a suitable chemical quality). The basis of the planned feed should therefore be pellets. In the short term the needs of the free world could be around 21 million tonnes. The market could harden if the building plans for pellets are not carried out and if a large part of the rich pellets produced continued to be used in the blast furnace. It seems necessary for the producers of pre-reduced products to envisage their supply in the form of participation or longterm contracts.

To sum up, although the technical possibilities for ore for direct reduction are great, the statistical situation as regards supply is still rather confused since a real market for such ores does not yet exist.

2.4. Description of the transport situation

Because of the considerable fluctuations in freight rates for ores, for the purpose of this study costs were calculated using the following formula:

$$K = G [TK + m P_m + KD] + R n P_n$$

K = cost of a round trip, including loading, unloading and other turnaround time.

G = total time required for a journey, expressed in days, including loading, unloading and other turnaround time.

R = travelling time in days, without loading, unloading and other turnaround time.

TK = costs per day, in UA/day

m = consumption of diesel oil, in tpd

P_m = price of diesel oil, in UA/t

KD = service of capital, in UA/t

n = consumption of bunker fuel, in t.p.d.

P_n = price of bunker fuel, in UA/t

Table 16 contains the consumption, price and cost rates used as a basis for determining the sea freight.

The loading capacity for pellets and sponge iron in a harbour overseas was generally taken as 5,000 t/hr; the unloading capacity in a European harbour was found to be approximately 2,000 t/hr. For the transport of oxide pellets a vessel of 100,000 tdw was generally assumed. On the basis of these data the loading time for pellets is 1 day, the unloading time 2.1 d and the remaining turnaround time, 1 day.

Table 16. Consumption, price and cost rates for determining the sea freight for a distance to Europe of 10,000 nautical miles

		Unit	Size of vessel	
1		2	3	4
Capacity		tdw	100,000	20,000
Period of operation per year		dpy	345	345
Speed		knots	15	15
Distance, round trip		nautical miles	10,000	10,000
Costs per day	(TK)	UA/d	4,000	2,000
Consumption of diesel oil	(m)	tpd	3	1
Price for diesel oil	(P _m)	UA/t	83	83
Service of capital	(KD)	UA/d	7,700	1,830
Consumption of bunker fuel	(n)	tpd	80	27
Price for bunker fuel	(P _n)	UA/t	55	55
Investment costs	(I)	10 ⁶ UA	25	5.9
Depreciation on I		%	6.67	6.67
Interest on I		%	4	4

For the transport of sponge iron, because of the danger of reoxidation, the vessel must not exceed a certain maximum capacity. In order to restrict the exposure time and thus the risk of reoxidation of the sponge iron to reasonable proportions, a capacity of 20.000 tdw was assumed. Taking the loading and unloading capacities already used, we have a loading time of 0.2 days, an unloading time of 0.5 days and a remaining turnaround time of 1 day.

The transport costs for sponge iron were also determined using the formula given on page 60 and the consumption, price and cost rates given in Table 16, column 4. The results obtained from the calculation of transport costs for oxide pellets and sponge iron are given in Table 17. According to this, the freight costs are 5 UA/t* for oxide pellets and 7.75 UA/t for sponge iron or, related to one tonne Fe in the sponge iron, assuming 1,539 kg pellets per tonne Fe in the sponge iron, 7.70 UA for oxide pellets; assuming 1,104 kg of sponge iron per t Fe in the sponge iron the cost is 8.55 UA for sponge iron. The sponge iron produced has a degree of metallisation of 95% and a metallic iron content of 92%.

Fig. 7 shows the transport costs for oxide pellets and sponge iron. The curves, in each case, refer to 1 tonne of material and, with reference to the two ship sizes given, can also be applied to the transport of other bulk materials such as coal or lignite briquettes. The relations given here can easily be applied to the transport per tonne of Fe in the sponge iron, if the required quantities per tonne of Fe in the sponge iron are known.

*) 1 UA = 46,06 FB; 3,05 DM; 3,13 hfl; £0.58;
7,20 Dkr; 5,22 FF; 797,77 Lit; 1,17 US \$
(31.12.1975).

Table 17. Total and specific costs of sea freight for pellets or sponge iron

	Unit	Pellets	Sponge Iron	Pellets and sponge iron
1	2	3	4	5
Size of vessel	tdw	100,000	20,000	100,000
Pellets	t	100,000	-	80,000
Sponge iron	t	-	20,000	20,000
Total costs	UA	500,100	155,150	500,100
Specific costs	UA/t	5	7.75	5
Specific costs for pellets	UA/t Fe	7.70	-	7.70
Specific costs for sponge iron	UA/t Fe in sponge	-	8.55	5.50

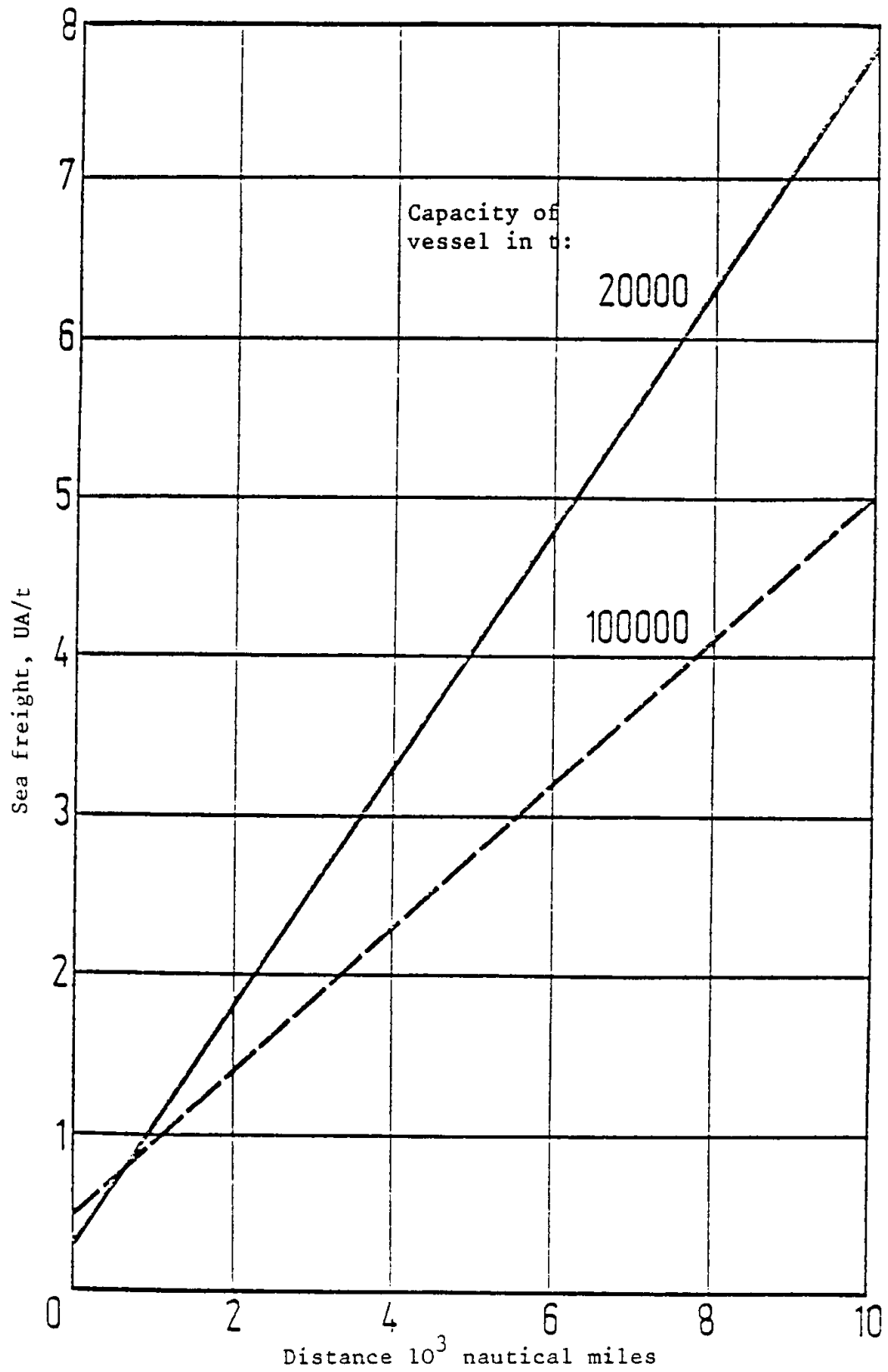


Fig. 7. Sea freight costs against distance (round trip)

The risk of reoxidation during transport, loading, unloading and storage of sponge iron can be avoided if sponge iron is briquetted. Hot briquetting, in particular, has the following advantages:

- cooling and cooling equipment no longer required,
- no storage facilities have to be provided by the producer and consumer for protecting the sponge iron against reoxidation,
- less abrasion and dust emission during transport, loading, unloading and handling,
- higher bulk density and thus higher loading and unloading capacities
- better melting down of the briquetted sponge iron.

The greatest advantage, however, is that because of the reduced risk of reoxidation sponge iron can be transported as a part cargo in large vessels. Taking as an example a vessel of 100,000 tdw travelling 10,000 nautical miles the difference between columns 4 and 5 in Table 17 denotes the freight savings per tonne Fe in the sponge iron as against the 20,000t freight for sponge iron, if 20% of the capacity of the 100,000 tdw vessel is used for carrying sponge iron. The difference in freight costs between the two possibilities amounts to 3 UA per tonne Fe in the sponge iron.

The technical possibilities for hot briquetting are still being tested; it may be safely assumed, however, that the costs for hot-briquetting do not exceed the savings calculated on the freight charges.

For transshipment, inland transport and internal handling of the sponge iron the same costs per tonne were assumed as for the transport of pellets.

Natural gas can be conveyed overland through pipelines. For transport overseas it can be liquified and transported in tankers. It can also be transported in the form of methanol.

Fig. 8 contains unpublished information on the transport of LNG and methanol, based on the simple distance between the place of liquefaction and the port of arrival, expressed in nautical miles. The figures are based on a natural gas price at source of 1 UA/Gcal. It was assumed that the losses occurring during transport as far as the liquefying plant and during the liquefying and re-evaporating process amount to approx. 20%. As far as the production of methanol is concerned losses occur both during transport from the source to the methanol plant and during the methanol production process, both during the heat consuming production of synthesis gas and during the highly exothermic methanol synthesis. These losses may be as much as 45%.

The price for natural gas at source makes itself felt to a considerable extent in the production of methanol because of the high losses but not so much in respect of the LNG transport chain. On the other hand, because of the high sea freight rates, the transport costs for LNG over longer distances increase much more quickly than for methanol. The intersection of the two curves for equal costs occurs after a single distance of approximately 7,000 nautical miles, which means that the transport of methanol is of no interest unless longer distances are covered. According to recent American calculations equal costs for the transport of LNG and methanol do not occur until after 12,000 nautical miles. The retransformation of the methanol into synthesis gas is not taken into account in these calculations.

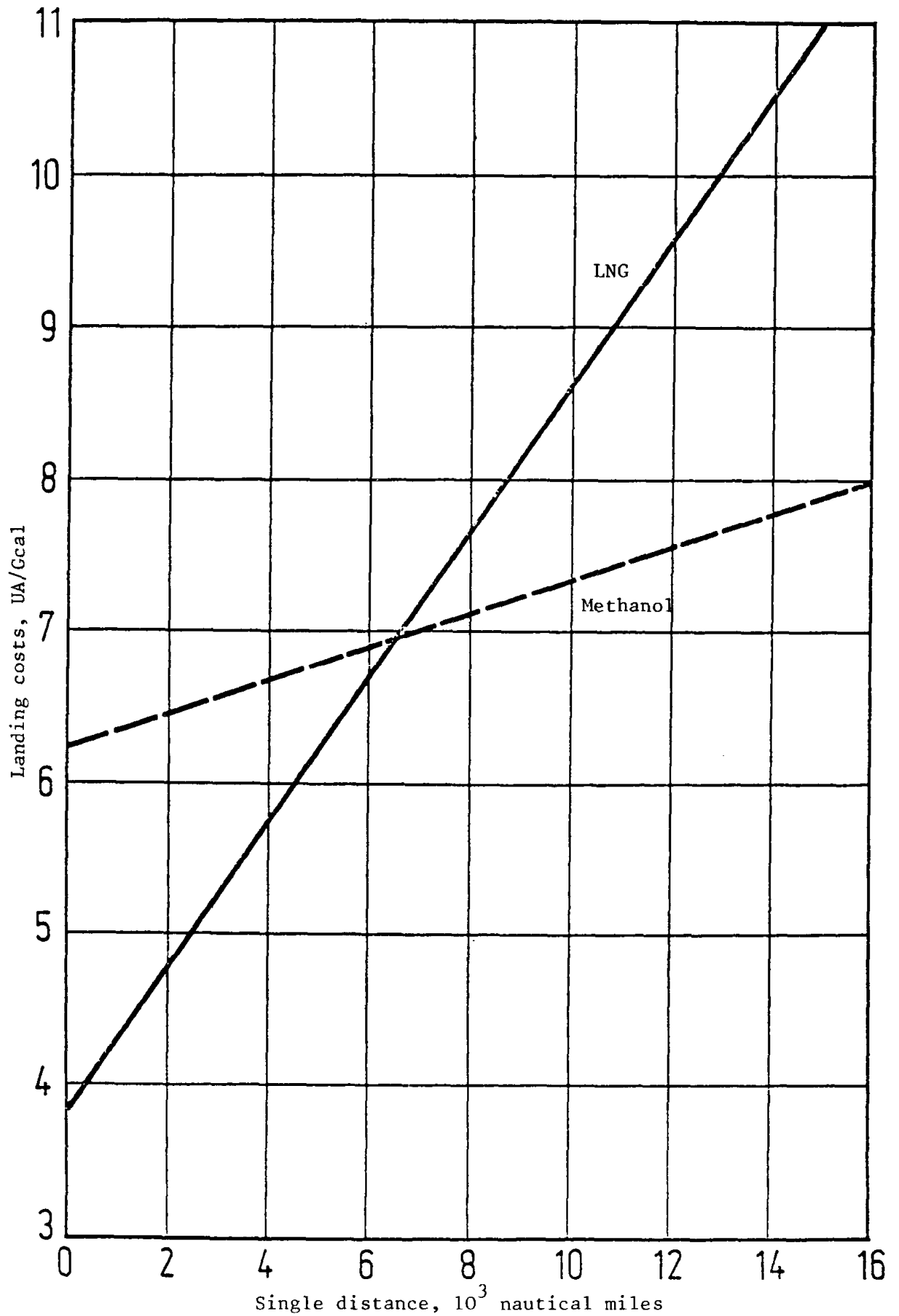


Fig. 8. Effect of distance transported on the landing costs (including costs of return journey) for LNG and methanol, assuming a natural gas price at source of 1UA/Gcal.

This study shows that because of higher losses and increasing basic prices of natural gas, the transformation into methanol is of no interest for direct reduction. An exception may be the transport of methanol for direct use in the chemical industry.

So far, LNG has only been purchased by gas supply companies; after re-evaporating it is fed into the general supply network for natural gas. There is no difference in price between normal natural gas and LNG, so that the consumer cannot hope for any advantages in respect of price from this type of transport. Whether more favourable prices might result if much greater quantities of natural gas were imported in the form of LNG and if the direct reduction plant acted as an individual large-scale customer, seems more than doubtful in view of a number of press reports. A direct reduction plant with a capacity of, say, 1 million t Fe per year requires relatively small quantities of natural gas. With a heat content H_u of 10,000 kcal/Nm³, the gross heat consumption would be about 0.33 billion Nm³, which is about one fiftieth of the annual supply of LNG recently discussed between the nationalised Algerian company SONATRAC and the European gas companies amalgamated into the Société d'Achat de Gaz pour l'Europe (SAGAPE).

In conclusion it may be said that in principle, because of its high heat content, natural gas has favourable transport properties. However, the low requirements of direct reduction plants mean that they cannot have their own supply system, so generally the usual prices for natural gas have to be paid.

2.5. Discussion of the possibilities for producing
sponge iron

Taking into account the comments contained in sections 2.2 - 2.4, the following locations seem to be particularly suitable for the construction and operation of direct reduction plants.

Canada and South Africa can be regarded as particularly favoured places for operating with solid reducing agents because they have both reducing agents suitable for direct reduction and suitable ores. Preferred locations for gas reduction processes are Venezuela and, with certain limitations, Australia if we consider the processing of Hamersley lump ores with natural gas from off-shore deposits on the west coast.

The type of ore deposit required for direct reduction is found in Brazil, the Ivory Coast, Guinea, Liberia, Peru and Sweden.

Areas with sufficient supplies of suitable reducing agents are Western Europe, the USA and Australia for coal, and for gaseous reducing agent mainly North Africa, the near East and North America.

In the next chapter, the possibilities and advantages of the various locations will be discussed from an economic point of view. Social structures and political situations will be largely ignored.

3. Estimating the production costs for sponge iron

For European sites close to the consumers the probable production costs for sponge iron can be determined with relative certainty, because in this area the required price rates for raw materials, transport, fuels, processing and investment are largely known. This basic calculation will be followed by an estimate of the production costs for other possible sites.

Quantities and prices for determining production costs are given in Tables 18 and 19. All calculations are based on the use of pellets containing 67% Fe, although the use of lump ores has also been proved. Since losses, assumed to be about 3%, occur during transport of iron ore pellets and during the reduction process 1,539 kg pellets have to be supplied per t Fe in the sponge iron.

By the time the consignment reaches the consumer, further loading, unloading and transport costs have had to be added, depending on the site of the direct reduction plant. The respective costs are also given in Table 19.

Solid and gaseous reducing agents can be used for direct reduction depending on the process. For the rotary kiln processes lignite briquettes from the Rhine region can be used in the ECSC. The heat content H_u of lignite is taken as 4,700 kcal/kg and the gross heat requirement as 5.2 Gcal/t Fe. Of the gross heat requirement of 5.2 Gcal, 1.7 Gcal is excess heat in the off gas. As under European conditions the utilisation of this off gas is a pre-condition for using the rotary tubular kiln process, an off gas credit of 2.60 UA/Gcal was calculated.

For direct reduction in a shaft furnace a gross heat requirement of 3.3 Gcal/t Fe was assumed. The heat content H_u of the natural gas used was 7,550 kcal/Nm³.

Table 18. Table for determining the production costs for sponge iron in Europe

Term	Unit	Value	
Quantity of iron ore pellets supplied	kg/t Fe sp ¹⁾	1,539	
Fe content	%	67	
Losses	%	3	
Nett input of iron ore pellets	kg/t Fe sp ¹⁾	1,493	
Amount of sponge iron supplied	kg/t Fe sp ¹⁾	1,104	
Fe content	%	92	
Losses	%	1.5	
Nett input of sponge iron	kg/t Fe sp ¹⁾	1,087	
Loading capacity	t/h	5,000	
Unloading capacity Europe	t/h	2,000	
Speed of vessel	knots	15	
Turnround time on a round trip	d	1	
		reduction process	
		Gas	Coal
Fuel consumption, gross	Gcal/t Fe sp ¹⁾	3.3	5.2
Off gas credit	Gcal/t Fe sp ¹⁾		1.7
Fuel consumption, nett	Gcal/t Fe sp ¹⁾	3.3	3.5
Salaries	h/t Fe sp ¹⁾		0.044
Wages	h/t Fe sp ¹⁾		0.351
Electricity	kWh/t Fe sp ¹⁾	135	50
Water	m ³ /t Fe sp ¹⁾	4	5
Maintenance costs	% of amount invested	4	4
Service of capital	% of amount invested	15	15

1) "sp" means "in the sponge iron"

Table 19. Prices for determining the production costs for sponge iron in Europe

	Material UA/t	Fe in sponge iron UA/t Fe
<u>Input costs</u>		
Price for iron ore pellets, f.o.b.	24.60	37.85
<u>Sea freight</u>		
Sea freight for pellets from overseas to Europe in 100,000 tdw vessels	5.--	7.70
Sea freight for sponge iron to Europe in 20,000 tdw vessels	7.75	8.55
Sea freight for sponge iron to Europe as part cargo on 100,000 tdw vessels	5.--	5.50
<u>Costs for transshipment and inland transport for pellets (1.539 kg/t Fe in sponge iron)</u>		
Transshipment at sea port	1.05	1.61
Inland transport	1.30	2.02
Transshipment at inland port	0.80	1.26
Internal transport	<u>0.55</u>	<u>0.86</u>
Total	3.70	5.75
<u>Cost of transshipment and inland transport for sponge iron (1.104 kg/t Fe in sponge iron)</u>		
Transshipment at sea port	1.05	1.16
Inland transport	1.30	1.45
Transshipment at inland port	0.80	0.90
Internal transport	<u>0.55</u>	<u>0.62</u>
Total	3.70	4.13
<u>Fuel costs</u>	Gross UA/Gcal	Nett UA/Gcal
Lignite	5.--	6.15
Natural gas	7.20	7.20
Off-gas credit	2.60	-

Table 19. (Continued)

		UA/unit
<u>Processing costs</u>		
Salaries	UA/h	7.80
Wages	UA/h	5.45
Electricity	UA/kWh	0.05
Water	UA/m ³	0.12
Transport costs	UA/t Fe	0.33
<u>Maintenance</u>		
for reduction with gas	UA/t Fe	3.55
for reduction with coal	UA/t Fe	3.80
<u>Service of capital</u>		
for reduction with gas	UA/t Fe	13.30
for reduction with coal	UA/t Fe	14.25
<u>Specific investment costs</u> per t annual production		
for reduction with gas	UA/t Fe	65.60
and for infra structure	UA/t Fe	22.95
for reduction with coal	UA/t Fe	75.40
and for infra structure	UA/t Fe	22.95

For determining the processing costs, the data contained in Tables 18 and 19 are used. The specific costs per hour for workers and clerical staff have been derived from the employment plan given in Table 20. As the rates for reduction processes with solid and gaseous reducing agents are not always identical, separate calculations were made for each process.

Tables 21 and 22 contain the production costs per tonne Fe in the sponge iron for both direct reduction processes. The production costs for the gas reduction process exceed those for the rotary tubular kiln process with solid reducing agents by 2.85 UA/t Fe. This is due to higher reducing agent costs of 2.25 UA/t Fe and higher processing costs of 0.60 UA/t Fe. As far as the processing costs for the gas reduction process are concerned, in the gaseous reduction process the lower costs (1.25 UA/t Fe lower) for maintenance and service of capital, in view of the 6.55 UA lower investment per tonne annual production, are set against the higher energy costs.

The assumptions on which these calculations are based apply to the erection of a direct reduction plant not connected with an existing iron and steel works. If the direct reduction plant were incorporated into an iron and steel works, the investments for infrastructure would not arise and for the rotary kiln process a somewhat higher off gas credit could be assumed. These advantages might mean a reduction in the production costs for sponge iron of approximately 4-5 UA. If suitable lump ores are used, the production costs for sponge iron in the basic calculation are reduced by approximately 11.50 UA/t Fe.

This basic calculation will be followed by a comparison with various selected overseas sites for the production of sponge iron in order to find out whether the European market can be supplied with sponge iron from overseas at reasonable cost.

Table 20. Employment Plan

	Workers				Clerical		
	Total	Production	Maintenance	Total	Production	Maintenance	
Sponge iron plant							
Workers (incl. clerical)	102	75	27	13	10	3	
+ 25% for illness and holidays							
= workers (incl. clerical)	128	94	34	16	12	4	

Table 21. Production costs in Europe in UA/t Fe in the sponge iron when using lignite briquettes as reducing agent

Cost group	kg/t Fe sp.or Gcal/t Fe sp.	UA/t or UA/Gcal	Costs in UA/t Fe sp.
1.	2	3	4
Pellets fob	1,539	24.60	37.85
Sea freight	1,539	5.--	7.70
Transshipment & transport costs	1,539	3.70	5.75
Input cost at works	1,539	33.30	51.30
Fuel costs	3.5	6.15	21.54
Processing costs			22.37
Production costs			95.21

Table 22. Production costs in Europe in UA/t Fe in the sponge iron using natural gas as reducing agent

Cost group	kg/t Fe sp.or Gcal/t Fe sp.	UA/t or UA/Gcal	Costs in UA/t Fe sp.
1	2	3	4
Pellets fob	1,539	24.60	37.85
Sea freight	1,539	5.--	7.70
Transshipment & transport costs	1,539	3.70	5.75
Input costs at works	1,539	33.30	51.30
Fuel costs	3.3	7.20	23.80
Processing costs			22.96
Production costs			98.06

It is difficult to estimate the production costs for sponge iron on these sites because data about consumption and prices, of which there is a fairly definite knowledge in Europe, are not available. Therefore, a number of assumptions have to be made and, for some types of costs and prices, calculations will have to be based on rough estimates.

The first example of a location with resources of ore and reduction agents will be Venezuela. Table 8 shows that the net consumption of natural gas in Venezuela makes up only 23% of the gross output, whereas 45% are pressed back and 34% or approximately 14.5 billion Nm³ natural gas per year are burned off.

The requirements as compared with conditions in Europe are slightly different. For Europe it had been assumed that up to the production of sponge iron stage 3% losses occur during the direct reduction process and transport.

As in the present case the sponge iron is produced overseas and shipped from there to the European works, it is assumed that 1.5% losses occur during the reduction process and 1.5% during the transport of the sponge iron. This means that for producing 1 tonne of Fe in sponge iron only 1,515 kg of oxide pellets need to be used. The quantity of sponge iron supplied with an Fe content of 92% is then 1,104 kg/t Fe in the sponge iron.

The price for shipping pellets to Europe through a suitable ore supplier was assumed to be 24.60 UA per tonne of material f.o.b.; the same price will be applied to the direct reduction process overseas. It is presumed that in Venezuela the mining of the ore, upgrading to concentrate with the required Fe content and production of pellets takes place under similar conditions. The loading and unloading costs contained in this price are equated with the transport costs as far as the direct reduction plant.

The calculation of the costs for the reduction agents are based on the same calorific value of the natural gas and the same consumption. In view of the amount of gas which, previously pressed back or burned off, can now be put to use, a price of 1.95 - 2.60/Gcal should be adequate. Also, differing from European conditions, electric energy was priced at 0.0065 UA/kWh.

Maintenance costs and service of capital were varied too. They are calculated by means of the same percentages, but they are derived from a 20% higher specific investment per tonne of annual output. The higher specific investment is said to be due to more difficult conditions of delivery and greater engineering services. Wages per tonne Fe are assumed to be the same. Transport costs, too, are based on European quantities and prices. Under such conditions, the processing costs for the gas reduction process in Venezuela amount to 24.10 UA per tonne Fe in the sponge iron.

Table 23 gives details on the costs. If the price of the reducing agent is 1.95 UA/Gcal, the production costs in Venezuela amount to 67.8 UA; if the price for the reducing agent is 2.60 UA/Gcal, the production costs amount to 70 UA. The cost price at the consumer's works of the sponge iron received is calculated by means of the quantity supplied, sea freight, the cost for transshipment in the harbour, inland transport and internal transport.

If the price of the reducing agent is 1.95 UA/Gcal the cost is 81.65 UA, at 2.60 UA/Gcal it is 83.95 UA. So the difference between the sponge iron imported from Venezuela and the sponge iron produced in Europe amounts to approximately 14 UA and 16.50 UA/t Fe respectively.

Table 23. Production costs for sponge iron overseas and cost price for sponge iron in UA/t Fe in sponge iron for the link Venezuela-Europe

Cost group	kg/t Fe sp.or Gcal/t Fe sp.	UA/t or UA/Gcal	Costs in UA/t Fe sp.
1	2	3	4
Input pellets	1,515	24.60	37.25
Fuel costs	3.3	1.95 (2.60)	6.50 (8.65)
Processing costs			24.10
Production costs			67.85 (70.--)
Amount of sponge iron supplied	1,105	62.35 (64.35)	68.90 (71.10)
Sea freight			8.55
Transshipment, inland transport			
Internal transport costs			4.15
Cost price at plant			81.60 (83.80)

Another example of availability of solid fuel in addition to suitable ore is the South African Republic (Saldana). The production costs have been determined by means of the same formula as used in the case of Venezuela. The costs for the reducing agent, in this case coal from the Transvaal, were assumed to be 2.60 to 3.30 UA/Gcal at the plant. With a simplified heat requirement of 4 Gcal/t of Fe and a reducing agent price of 2.60 UA/Gcal, the production costs for sponge iron in South Africa amount to 72.15 UA/t Fe; if the price of the reducing agent is 3.30 UA/Gcal, they amount to 74.75 UA per tonne of Fe. The cost prices at the consumer's works in Europe were calculated to be 88.50 UA/t Fe and 91.15 UA/t Fe respectively. So in this case the differences between the production costs in Europe and those for imports from South Africa amount to 6.50-7 UA/t Fe.

Where only ore or reducing agent is available the production costs are determined qualitatively, by means of a diagram (Fig. 9) because the prices for ore and reducing agents, which influence the end result considerably, fluctuate and vary a great deal from place to place. The diagram was worked out on the basis of the cases of Europe and Venezuela described above and shows the production costs for sponge iron, against the prices of reducing agents and ores. In particular, the diagram invites discussion of 4 extreme cases:

- I High reducing agent prices and high ore prices (Europe)
- II High reducing agent prices and low ore prices (Brazil)
- III Low reducing agent prices and high ore prices (Saudi Arabia)
- IV Low reduction agent prices and low ore prices (Venezuela).

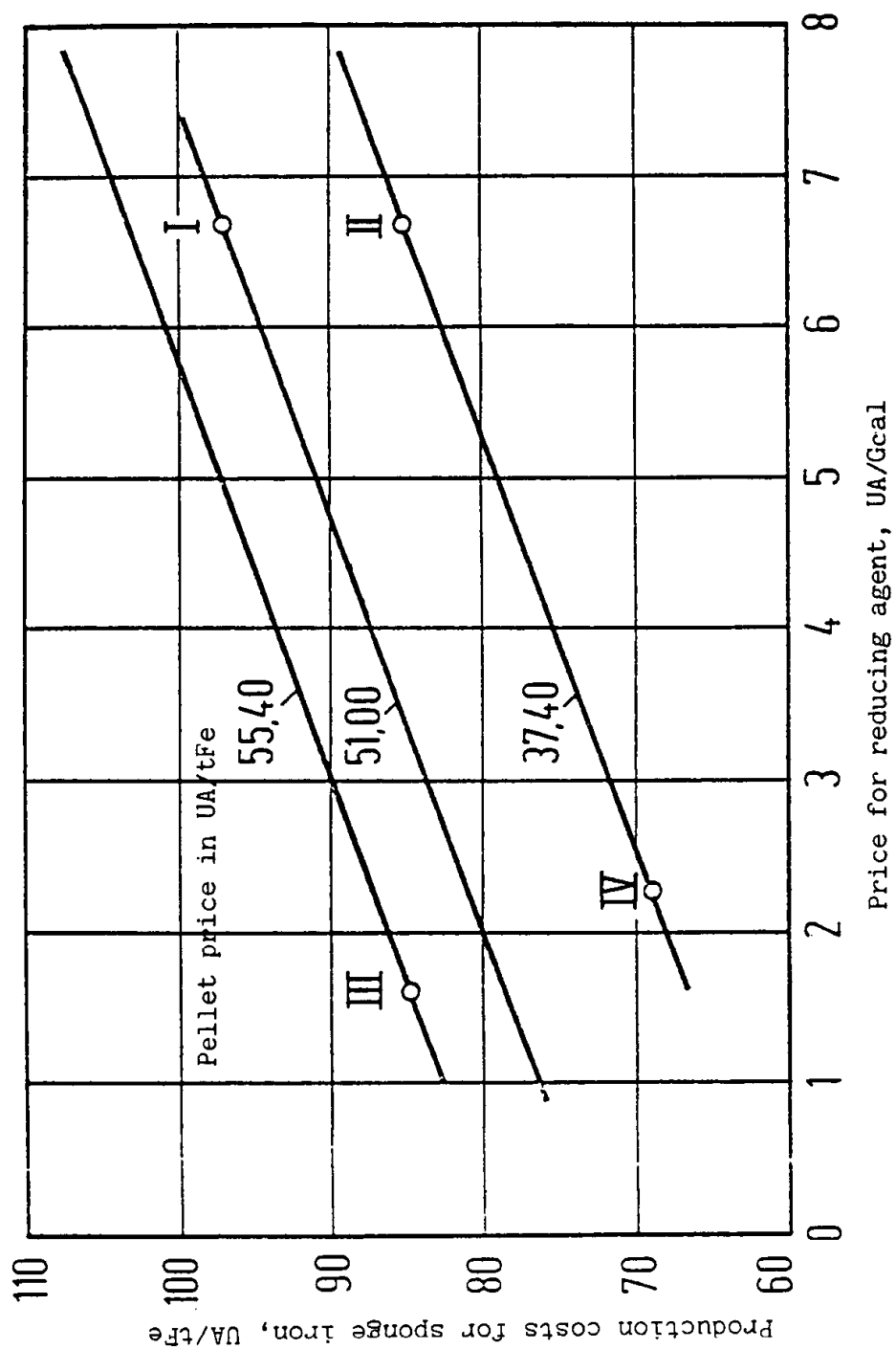


Fig. 9 Production costs for sponge iron at various places against price of reducing agent and costs for oxide pellets.

For case II with suitable ore reserves, pellet prices at the plant of 37.40 UA/+Fe and reducing agent prices of 6.25 - 7.20 UA/Gcal were assumed. Fig. 9 gives production costs of 83.30 - 86.90 UA/t Fe. After adding the costs calculated for transport and loading and unloading of sponge iron, we obtain cost prices at the consumer's works in Europe of 97.50 - 100 UA/t Fe.

For case III, i.e. for conditions such as prevail in Saudi Arabia for instance, the pellet price at the reduction plant was determined as being 55.4 UA/t Fe; it includes the freight costs for pellets from Brazil to the Persian Gulf. The reducing agent prices (natural gas) were estimated to be between 1.30 and 2 UA/Gcal. With the same processing costs as in case II we obtain production costs for sponge iron of 83.90 - 86.20 UA/t Fe and cost prices in Europe of 97 - 100 UA/t Fe.

Table 24 contains a summary of cases I to IV, separating manufacturing costs at the place of sponge iron production and cost prices at the consumer's works after importation into Europe. After having taken into account all the uncertainties inherent in the assumptions made, we can see that the cost prices to the consumer in Europe for cases I + II do not differ widely. We may conclude from this that the production of sponge iron for exportation to Europe at a location with suitable ore reserves but relatively high prices for reducing agents has no particular advantages in comparison with the production of sponge iron in Europe.

Case III clearly demonstrates to what extent production costs and thus cost prices to the consumer in Europe are dependent on the transport route for ore pellets or sponge iron. For the relatively unfavourable case assumed here with very long transport routes for both ore pellets and sponge iron we obtain cost prices which are as high as those of Case I. These conditions,

Table 24. Production costs for sponge iron in UA/t Fe at various locations and cost price of sponge iron produced internally and imported at consumers' works in Europe.

	Case I	Case II	Case III	Case IV
Pellet price at reduction plant	51.--	37.40	55.40	37.40
Price of reducing agent	6.25 to 7.20	6.25 to 7.20	1.30 to 2.--	2.-- to 2.60
Processing costs	23.--	24.10	24.10	24.10
Production costs	95.-- to 98.50	83.30 to 86.90	83.90 to 86.20	67.90 to 70.20
Transport & transshipment costs	-	12.80	11.80	12.80
Losses	-	1.30	1.30	1.--
Cost price	95.-- to 98.--	97.50 to 100.-	97.-- to 100.-	81.60 to 84.--

for instance, prevail in the case of Saudi Arabia: the ore is obtained from Brazil and the sponge iron is shipped to Rotterdam via the Suez Canal. Fig. 7 shows that if it is possible to shorten the transport route (e.g. by obtaining the ore from South Africa or Australia), the cost price as in Table 24 also decreases.

It should be possible to obtain equally favourable cost prices for sponge iron in the case of locations where the transport route for both pellets and sponge iron is short, i.e. if the site is "on the way" as it were from the port where the ore is loaded to Europe. For instance, for an assumed site in Algeria, with reducing agent prices of 2.60 UA/Gcal and with the ore being supplied by Brazil, the cost price for sponge iron to the consumer in Europe was calculated to be approximately 85.25 UA/t Fe. These prices are similar to those of case IV, i.e. to those calculated for producing sponge iron at a site where there are ore and reducing agent reserves. The difference in price to the consumer in Europe between such a site and sponge iron production in Europe amounts to approximately 16.50 UA even if, for the transport of sponge iron, one takes into account the relatively unfavourable aspect of one-way freight in 20,000 tdw vessels.

With a price difference of that magnitude the possibility of making up for the lack of scrap in the ECSC by importing sponge iron is worth while considering. In order to meet the sponge iron requirements of the European iron and steel industry on a long-term basis, it is necessary to set up further direct reduction plants on suitable sites with European capital participation, because the plans known at present, from a long-term point of view, will mainly serve the purpose of supplying their operators' own needs.

ALLIS-CHALMERS PROCESS

1. Characteristics

1.1. Name and history of process

Allis-Chalmers Ported Kiln Process
(New designation: ACCAR process)

1.2. Principle of process

A rotary kiln containing iron ore undergoing reduction by natural gas introduced underneath the ore. Heating is by combustion of surplus gas leaving the ore bed by air introduced into the upper part of the kiln.

Ports in the wall of the kiln are supplied with natural gas during immersion in the reacting bed and with air during the period when they are exposed (see Fig.1)

1.3. Inventors

Allis-Chalmers Ltd.

1.4. Companies involved in development

Allis-Chalmers Ltd.

2. General description (schematic drawing)

See attached drawings.

3. Plant required

Not known

4. State of development

4.1. Name of laboratories where the process is situated

- (1) Allis-Chalmers Reduction Systems Division, Milwaukee.
- (2) Niagara Metals plant, Niagara Falls, Ontario.

4.2. Pilot plant

4.2.1. Location

Niagara Falls (Canada)

4.2.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

Kiln 2.5 m o.d. x 45 m.

4.2.3. Capacity in tonnes per 24 h of final product

100 tonnes per day

4.3. Industrial installations

Sudbury Metals, Falconbridge (Canada) with a capacity of 260,000 tpy. Commissioned in 1976 after conversion of the closed SL/RN plant.

5. Reducing agents

5.1. Raw materials on which the reducing agent is based

Natural gas, or liquid hydrocarbons. Some coal may be used (up to 20% of charge)

5.2. Process for conversion of above raw materials into a form for use in reduction

Internal to process.

- 5.3. Condition and analysis of reducing agent at moment of use
Hydrocarbon is cracked in bed of hot ore within the system.

6. Ores and ferruginous materials

6.1. Source

Lump ore or agglomerate

6.2. Analysis

60% iron: dependent on purity of product required

6.3. Size

Lump or sinter

6.4. Preparation

See 6.1.

7. Additions

Not known

8. Temperature zones in reduction
in °C in the reduction apparatus

Bed temperature dependent on ore characteristics - typically
1050°C - 1100°C.

9. Pressure zones in reduction
in kg/cm² in the reduction apparatus

Atmospheric pressure.

10. Product
 of the reduction apparatus

10.1. Form

As raw material charge

10.2. Analysis

Metallic Fe 85-90%, carbon 0-1%.

10.3. Degree of reduction

10.3.1. Proportion of oxygen removed from iron oxides as a percentage
 of the total oxygen combined with iron in the ores

Up to 95 %

10.3.2. Metallic iron as a percentage of the total iron content
 of the product

92 - 95 %

11. Further treatment of the product (10) of the reduction apparatus

11.1. Method of cooling

Indirect water spray

12. Final product obtained after treatment according to 11

See 10

13. Iron yield

Not known

14. Utilisation of final product

Destination of final product (blast furnace, electric or other furnace). Technical and economic differences as compared with using pig iron and scrap

Electric arc furnace melting

15. Consumption data per tonne of final product (12)

15.1. Raw materials for reduction (5.1)

Not known

15.2. Fuels and thermal values required for the entire process (fuel characteristics) (Thermal balance)

2.75 - 3.25 Gcal/t using oil/coal

3.00 - 3.25 Gcal/t using oil

3.50 - 3.75 Gcal/t using natural gas

15.3. Electrical energy

35 kWh

15.4

to Not known

15.9

16. Overall Capital Investment

Not known

17. Opinions expressed on the process in relation to:

17.1. State of development

Large pilot plant had completed 6 months campaign June 1973.
Extensive melting trials have been done using the product.
The first industrial scale plant was commissioned in 1976
at Falconbridge under the name ACCAR process.

18. Bibliography

18.1. Articles and publications

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- 2) Briber, F.E., Ablonczy, G.S., and Henrichsen, J.D.: The Controlled Atmosphere Furnace - Its Design and Application. Journal of Metals, 19, June 1967, pp. 90-92.
- 3) Canadian Plant Demonstrates Direct Reduction Process - Allis-Chalmers Engineering Review 1973.

18.2. Patents

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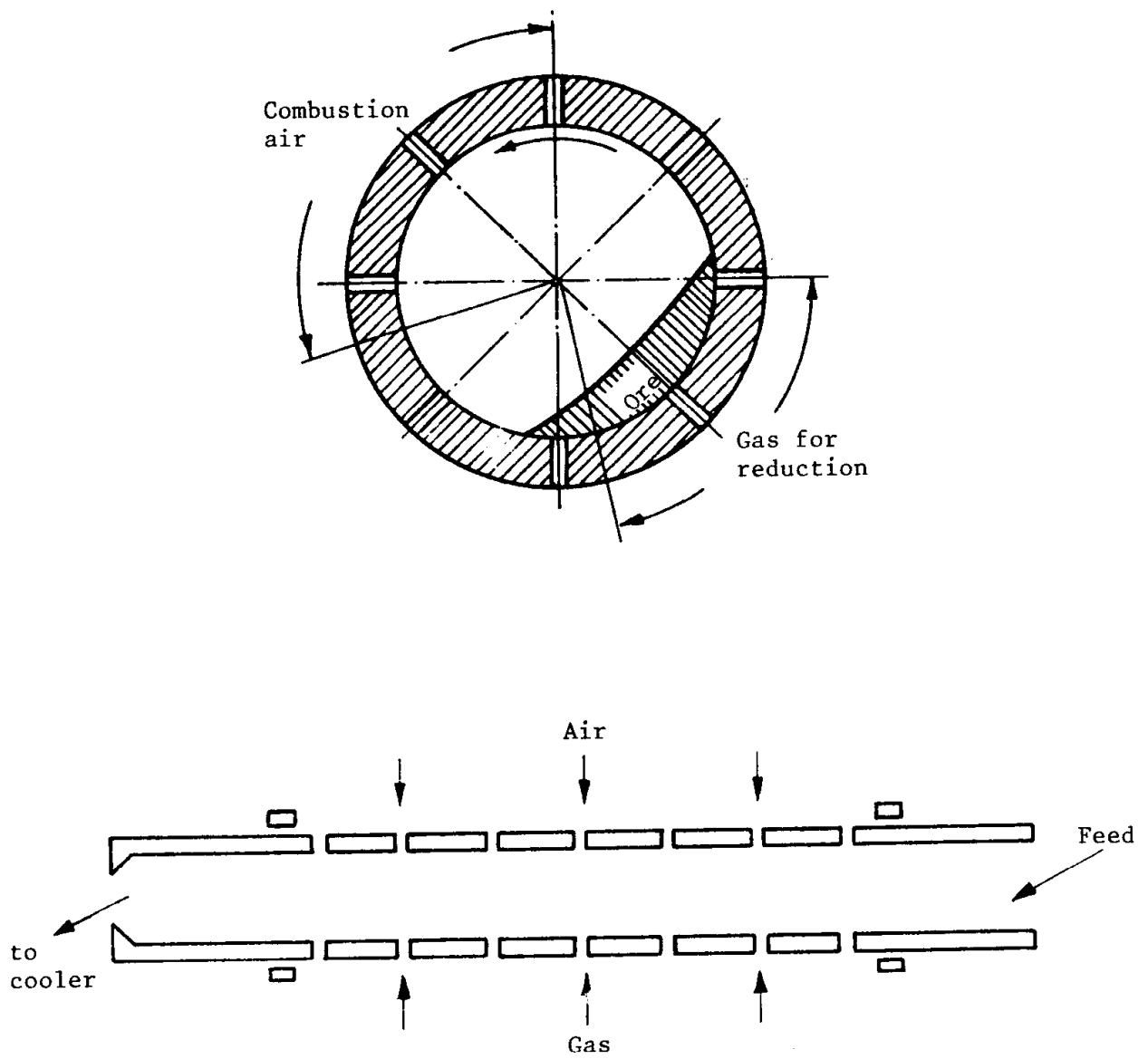


Fig.1. Allis-Chalmers Ported Kiln

ARMCO PROCESS

1. Characteristics

1.1. Name and history of process

Armco direct reduction process

- 1963: Experimental furnace, 1 t.p.d.
- 1964: Pilot furnace, 50 t.p.d., based on the partial oxidation of natural gas by air.
- 1965: Modification of pilot furnace for reforming of natural gas by top gas in "Pebble stoves".
- 1972: Industrial-scale unit at Houston, Texas, with a capacity of 300,00 t Fe per year, based on the reforming of natural gas by steam.

1.2. Principle of process

Counterflow reduction of iron ores by gases obtained by reforming hydrocarbons with steam, in a continuous shaft furnace.

1.3. Inventors

Armco Steel Corporation

1.4. Companies involved in development

Armco Steel Corporation and Foster Wheeler. Fried. Krupp GmbH Industrie- und Stahlbau, VÖEST (Austria), Italimpianti (Italy) and IHI (Japan) have construction licences.

2. General description (schematic drawing)

See Fig. 1

The shaft furnace is charged with a mixture of oxidised pellets and graded ores. The reducing gas, obtained by reforming natural gas with a slight excess of steam, contains approximately 70% hydrogen, 20% carbon monoxide, 7% steam and 3% carbon dioxide and methane. It reaches the furnace at a temperature of around 760 - 780°C. The oxidised gas from the reduction is cooled by scrubbing and used to heat the reforming stove and to cool the sponge iron in the lower part of the shaft furnace.

3. Plant required

Shaft furnace, reforming stove, cleaning plant for spent gas, ore store and storage of the metallised product.

4. State of development

4.1. Name of laboratories where the process is situated

Armco Steel Corporation
P O Box 700
Middletown
Ohio 45042, U.S.A.

4.2. Pilot plant

4.2.1. Location

Kansas City, U.S.A.

4.2.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

Shaft furnace with bell top, 1 m in diameter. N.B. This furnace at first used reducing gas prepared by partial oxidation (Fig. 2), then cyclic reforming by top gas in stoves filled with alumina pebbles (Fig. 3) (system similar to that in the Purofer process), but none of these solutions was used for the industrial plant.

4.2.3. Capacity in tonnes per 24 h of final product

The nominal capacity of this plant was 50 t Fe per day, that is approximately 55 t product per day.

4.3. Industrial installations

4.3.1. Location

A single industrial unit is in operation at present at Houston, Texas, in the iron and steelworks of the Armco Steel Corporation.

4.3.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

Diameter of shaft furnace	5.0 metres
Height of reduction zone	14.3 metres
Height of cooling zone	13.1 metres

4.3.3. Capacity in tonnes per 24 h of final product - per unit of production

950 t.p.d. 92% metallised sponge containing 92% Fe.

4.3.4. Method of operation (continuous or batch, number of operating hours per day)

Continuous furnace.

4.3.5. Coefficient of utilisation in time (number of working days per annum)

The annual rate of utilisation is 330 d.p.y.

4.3.6. Maximum rated production

The maximum capacity is 1100 t.p.d.

5. Reducing agents

5.1. Raw materials on which the reducing agent is based

The reducing gas is produced by reforming of natural gas.

5.2. Process for conversion of above raw materials into a form for use in reduction

Methane is converted into hydrogen and carbon monoxide by catalytic reforming with a small excess of steam.

5.3. Condition and analysis of reducing agent at moment of use

For a steam/carbon ratio of 1.5 in reforming, the reducing gas has the analysis:

H ₂	H ₂ O	CO	CO ₂	CH ₄
68.3%	8.4%	20.2%	2.0%	1.1%

5.4. Preheat temperature on entering reduction apparatus

The reducing gas leaves the reforming stove at 870°C and should be cooled by the addition of recycled top gas before being injected into the furnace at a temperature between 760° and 800°C.

5.5. Gas velocity in reduction apparatus

At the tuyeres : 1 m/sec.

5.6. Analysis and temperature of gas after passing through reduction apparatus

Analysis of top gas:

H ₂	H ₂ O	CO	CO ₂	CH ₄
51%	23%	14%	11%	1%

Temperature of top gas: approximately 400°C.

6. Ores and ferruginous materials

6.1. Source

Oxidised pellets and graded ores of various origins

6.2. Analysis

A typical charge is composed of 80% Pea Ridge pellets and 20% graded Brazilian ore the analyses of which are:

	Fe _{tot}	SiO ₂	CaO	S	P
Pellets	67.1%	2.46%	0.34%	0.001%	0.09%
Ore	69.5%	0.56%	0.08%	0.004%	0.036%

6.3. Size

5 - 20 mm

6.4. Preparation

Wet screening to 5 mm of the pellets and ores before the charging hopper.

7. Additions

None

8. Temperature zones in reduction
in °C in the reduction apparatus

The injection temperature of the gas depends on the possibility of softening and sticking of the material used; it varies between 760° and 800°C.

9. Pressure zones in reduction
in kg/cm² in the reduction apparatus

The reducing gas is injected into the furnace at a gauge pressure of 1.2 kg/cm²; at the throat the gauge pressure is 0.4 kg/cm².

10. Product
of the reduction apparatus

10.1. Form

Same as the product charged. The sponge iron contains 5% fines below 5 mm which are not separated.

10.2. Analysis

Example of analysis for normal charge:

Fe _{tot}	Fe _{met}	C	O ₂ residual
91.5%	84.2%	2.4%	2.3%

10.3. Degree of reduction

10.3.1. Proportion of oxygen removed from iron oxides as a percentage of the total oxygen combined with iron in the ores

94%

10.3.2. Metallic iron as a percentage of the total iron content of the product

92%

10.4. Utilisation of by-products

The top gas is cooled by injecting water and then scrubbed. Part of the cold gas is used directly to heat the reforming stove, the rest is compressed and used to regulate the temperature of the reducing gas injected at the tuyeres and to cool the sponge iron.

11. Further treatment of the product (10) of the reduction apparatus

11.1. Method of cooling

The reduced product is cooled in the shaft furnace by cold top gas injected at the bottom. This gas then passes to the reduction zone.

11.2. Crushing, compacting or briquetting

None.

11.3. Details of final size grading

In the case of Pea Ridge pellets it differs little from that of the initial product; there are 5% of fines below 5 mm which are not separated.

11.4. Method of concentration (e.g. magnetic separation or sintering)

None.

12. Final product obtained after treatment according to 11

Product identical to that described under 11.

13. Iron yield

13.1. Iron contained in product (10) as a percentage of iron in the charge

The iron yield is almost 100%; the losses through the carrying out of dust in the gases are small.

13.2. Iron contained in final product (12) as a percentage of iron in the charge

Same as yield quoted under 13.1.

14. Utilisation of final product

Destination of final product (blast furnace, electric or other furnace). Technical and economic differences as compared with using pig iron and scrap.

The sponge iron may be used in the blast furnace, in oxygen or electric steelmaking. At Houston the metallised pellets are used in the electric arc furnaces, where they represent approximately 25% of the charge.

15. Consumption data per tonne of final product (12)

15.1. Raw materials for reduction (5.1)

Consumption of natural gas : $390 \text{ Nm}^3/\text{t}$ or $420 \text{ Nm}^3/\text{t Fe}$.

15.2. Fuels and thermal values required for the entire process (fuel characteristics) (thermal balance)

The natural gas should be desulphurised to less than 0.5 ppm before reaching the reforming stove. The overall heat consumption at Houston is 3.3 Gcal per tonne (i.e. 3.6 Gcal per tonne Fe). For a 1200 t.p.d. plant the guarantees would be $3.2 \text{ Gcal/t} \pm 5\%$ for a product 92% metallised $\pm 2\%$. Moreover, Armco expects to be able to reach 2.9 Gcal per tonne.

15.3. Electrical energy

15.3.1. For heating

None

15.3.2. For power

25 kWh/t Fe or 22 kWh/t product

15.4. Steam used

15.4.1. For reducing agent

515 kg/t product or 560 kg/t Fe

15.4.2. As energy

990 kg/t product or 1070 kg/t Fe.

15.5. Oxygen

None.

15.6. Cooling water

1.6 m³/t product or 1.8 m³/t Fe.

15.7. Refractories

Not known.

15.8. Man hours of productive labour

0.14 h/t product or 0.16 h/t Fe.

15.9. Labour and materials for routine maintenance

Labour: 0.12 h/t product or 0.13 h/t Fe

Materials: 9 FF/t product or 10 FF/t Fe.

16. Overall capital investment

Capital expenditure per annual tonne excluding construction is at present estimated as 450 FF/t for a 1200 t.p.d. unit.

17. Opinions expressed on the process

It is essential to desulphurise the natural gas to less than 0.5 ppm (if possible 0.2 ppm) to avoid poisoning the catalyst.

18. Bibliography

18.1. Articles and publications

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18.2. Patents

Not known.

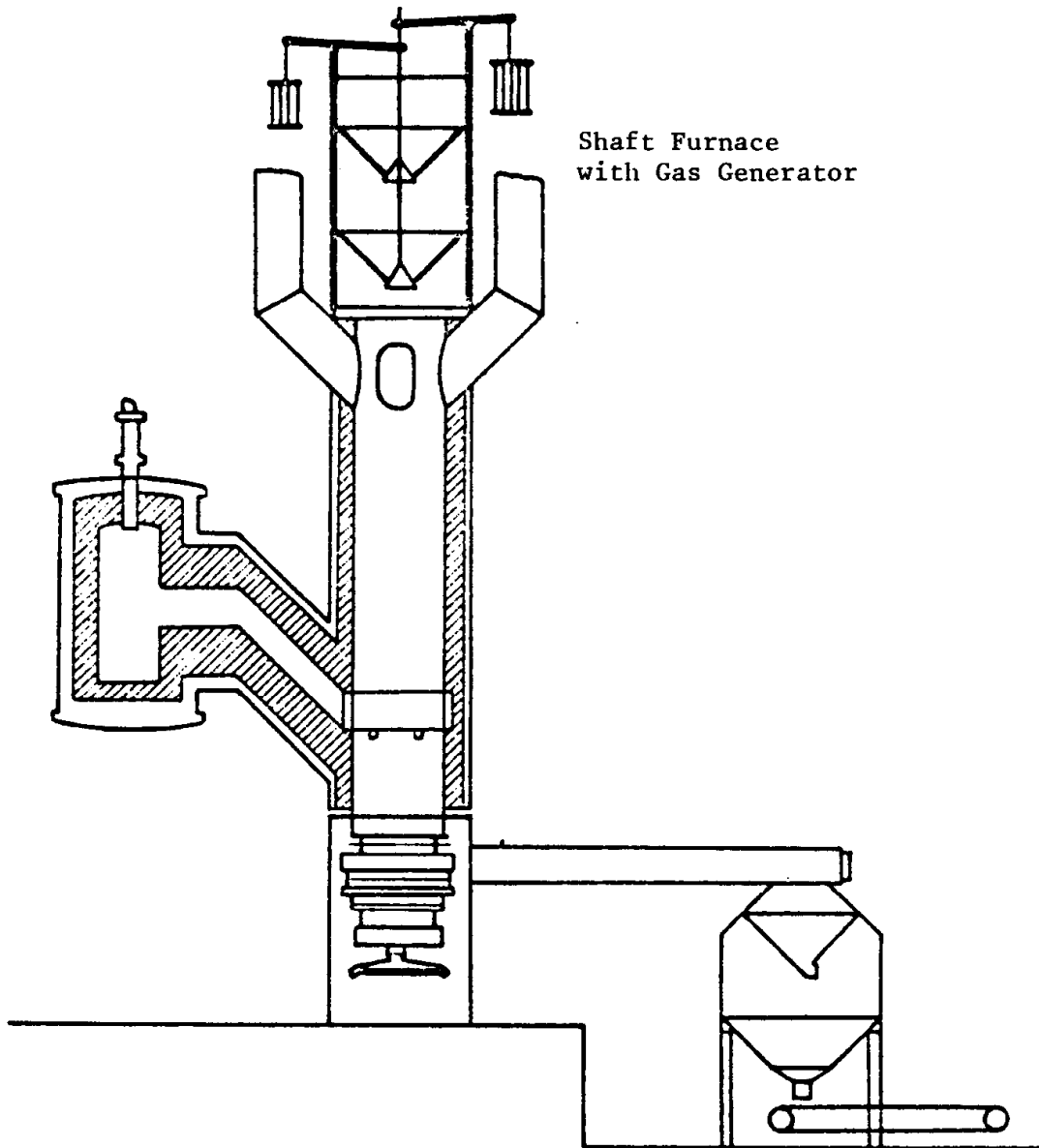


Fig. 2. Kansas City pilot plant
Partial oxidation of natural gas

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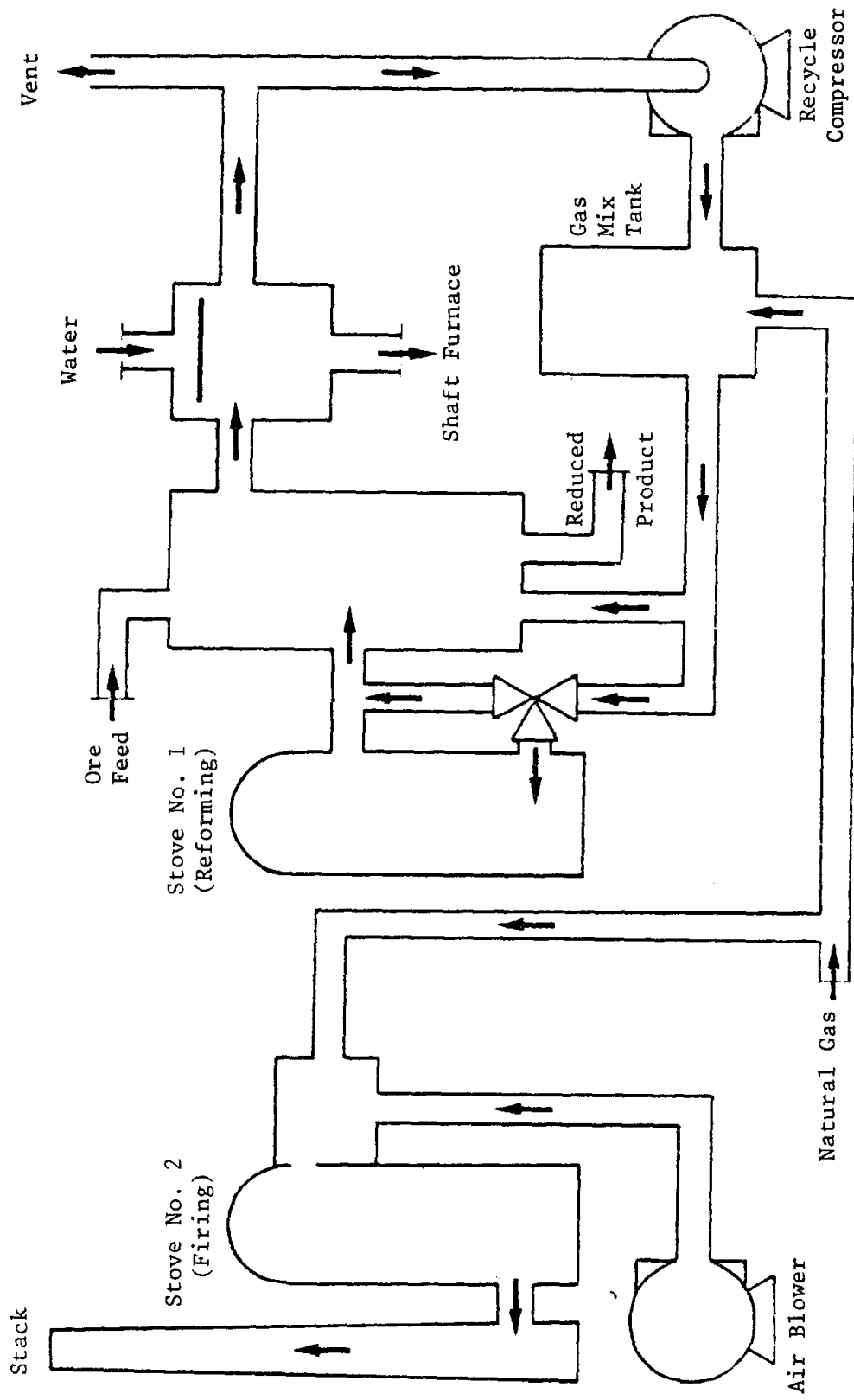


Fig. 3. Diagram of Armco Process - Reforming of top gas.

ESSO FIOR PROCESS

1. Characteristics

1.1. Name and history of process

Esso Fior process

In 1961 Esso Research and Engineering Company decided to build a pilot plant (production capacity 5 tonnes per day) at the Esso Research Laboratories at Baton Rouge, Louisiana, for direct reduction of iron ores in a fluidised bed. The pilot plant was in operation from April 1962 to March 1966, and made 25 test runs. After the feasibility of the process had been demonstrated using this pilot plant, a demonstration plant on a semi-industrial scale with a production capacity of 300 t.p.d. was built at Imperial Oil Enterprises Ltd., Dartmouth, Nova Scotia. This plant was commissioned in October 1965. The demonstration plant ran until November 1969, having achieved its target.

1.2. Principle of process

Reduction in a fluidised bed reactor in several stages by a counterflow of reformed gas.

1.3. Inventors

Esso Research and Engineering Company

1.4. Companies involved in development

Esso Research and Engineering Company, Baton Rouge, Louisiana
and Imperial Oil Enterprises Ltd., Halifax, Nova Scotia.

2. General description (schematic drawing)

See Fig. 1.

The iron ore fines are fed to a system of fluid bed reactors in counterflow to gases at above atmospheric pressure. In the first stage drying of the ore takes place, together with a certain degree of desulphurisation by contact with combustion products. In the subsequent stages the reduction is achieved by reducing gases generated by catalytic "steam reforming" or partial combustion of natural gas, LPG or fuel oil. The iron powder is hot briquetted and suitably cooled. The product is not pyrophoric and is resistant to oxidation. The gas leaving the reduction reactor is cooled, cleaned of H_2O and CO_2 and dust and then recycled.

3. Plant required

3.1. Ore preparation plant

3.2. Plant for reforming using steam or partial oxidation

3.3. Fluid bed reactor with several stages

3.4. Briquetting plant

4. State of development

4.1. Name of laboratories where the process is situated

Esso Research Laboratories

4.2. Pilot plant

Esso Research Laboratories

4.2.1. Location

Baton Rouge, Louisiana

4.2.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

Not known

4.2.3. Capacity in tonnes per 24 h of final product

5 t.p.d.

4.3. Industrial installations

- 1) Imperial Oil Enterprises Ltd.
- 2) Fior de Venezuela

4.3.1. Location

- 1) Dartmouth, Nova Scotia, 1965 - 1969
- 2) Matanzas, Venezuela, commissioned 1976

4.3.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

Not known

4.3.3. Capacity in tonnes per 24 h of final product - per unit of production

- 1) 300 t.p.d.
- 2) 1200 t.p.d.

4.3.4. Method of operation (continuous or batch, number of operating hours per day)

Continuous

5. Reducing agents

- 5.1. Raw materials on which the reducing agent is based
Any hydrocarbons, from natural gas to the heavy oils
- 5.2. Process for conversion of above raw materials into a form for use in reduction
Reforming using steam or partial combustion
- 5.3. Condition and analysis of reducing agent at moment of use
Not known
- 5.4. Preheat temperature on entering reduction apparatus
540°C max.

6. Ores and ferruginous materials

- 6.1. Source
Iron ores from various sources
- 6.2. Analysis
Rich iron ores with a gangue content less than 3%; preferably hematites (not specular), of low moisture content
- 6.3. Size
<4 mesh with less than 20% <325 mesh
- 6.4. Preparation
Drying, crushing and screening

7. Additions

None

8. Temperature zones in reduction
in °C in the reduction apparatus

Not known

9. Pressure zones in reduction
in kg/cm² in the reduction apparatus

Pressure greater than atmospheric

10. Product
of the reduction apparatus

10.1. Form

Reduced, powdered ore (24 - 45% <325 mesh)

10.2. Analysis

Example: 92.4 % Fe_{tot}, 85.0 % Fe_{met}, 0.7 % C, 0.07 % P,
0.02 % S, 0.7 % Al₂O₃, 2.35 % SiO₂, 1.1 % (TiO₂
+ MgO + CaO), 0.1 % MnO

The carbon content of the sponge iron can vary from 0 to 2 %
and more.

10.3. Degree of reduction

Degree of metallisation 92 %

11. Further treatment of the product (10) of the reduction apparatus

11.1. Method of cooling

None

11.2. Crushing, compacting or briquetting

Hot briquetting

11.3. Details of final size grading

Not known

11.4. Method of concentration (e.g. magnetic separation or sintering)

None

12. Final product obtained after treatment according to 11

12.1. Form

Briquettes of various shapes, preferably pillow-shape,
11 x 6 x 4 cm. Apparent density of each briquette: 5.7 gm/cm^3 .
Bulk density: 3.1 gm/cm^2

12.2. Analysis

See 10.2.

13. Iron yield

Not known

14. Utilisation of final product

Destination of final product (blast furnace, electric or other furnace). Technical and economic differences as compared with using pig iron and scrap.

As solid charge in steelworks and foundry. So far 45,000 tonnes have been produced, in various countries, by various companies, preferably for making steel in an electric furnace. The percentage of briquettes in the charge has reached 100%. It is most often 85%. The product may be stored without special protection. In a stock of briquettes of low to medium porosity metallisation losses occur, under the surface exposed to the atmosphere, of the order of 3%.

15. Consumption data per tonne of final product (12)

(Estimated figures for a plant of 1,500,000 t.p.a.)

15.1. Raw materials for reduction (5.1)

Ore: approximately 1,500 kg (depending on iron content);

Reducing agent: altogether approximately 3.65 Gcal/t sponge iron.

15.2. Fuels and thermal values required for the entire process
(fuel characteristics) (thermal balance)

Fuel: 3.65 Gcal per tonne sponge iron

15.3. Electrical energy

45 kWh per tonne sponge iron

15.4. Steam used

With "steam reforming" of natural gas there is a steam surplus of 240 kg/t at 20 atm.

15.5. Oxygen

None

15.6. Cooling water

Make-up water: 44 m³ per tonne sponge iron

15.7. Refractories

Not known

15.8. Man hours of productive labour

0.35 h per tonne sponge iron (including supervision)

15.9. Labour and materials for routine maintenance

4% p.a. of capital expenditure

16. Overall capital investment

16.1. For the plant described in (3) and related to one tonne of capacity for the final product (12)

Capacity expressed in tonnes per day or per year

\$ 125/(tpy) for a plant with a capacity of 400,000 tpy,

\$ 88/(tpy) for a plant with a capacity of 2 million tpy

(1976 figures from 7) under 18.1.)

16.2. Construction costs of ancillary plant not covered in (3)

Not known

17. Opinions expressed on the process in relation to:

17.1. State of development

The process is consolidated by 8 years of research and by operation on a semi-industrial scale.

17.2. Practical operating difficulties

Not known

17.3. Possibilities of exploitation

For 5 years Lukens Steel has taken from Fior de Venezuela up to 150,000 tpy sponge iron briquettes for steelmaking.

18. Bibliography

18.1. Articles and publications

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Journal of Metals 8 (1966) 2, pp 237-242.
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Reduction of Iron Ore, ECE Seminar on direct reduction
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- 5) U.Kalla, R. Steffen - Present position and development
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- 7) Oehlberg, R.J. and D.C. Violetta: The role of the Fior process in the expanding worldwide direct reduction industry. ECE Seminar on the use of sponge iron in steelmaking, Bucharest, Rumania, 1976.

18.2. Patents

Not known

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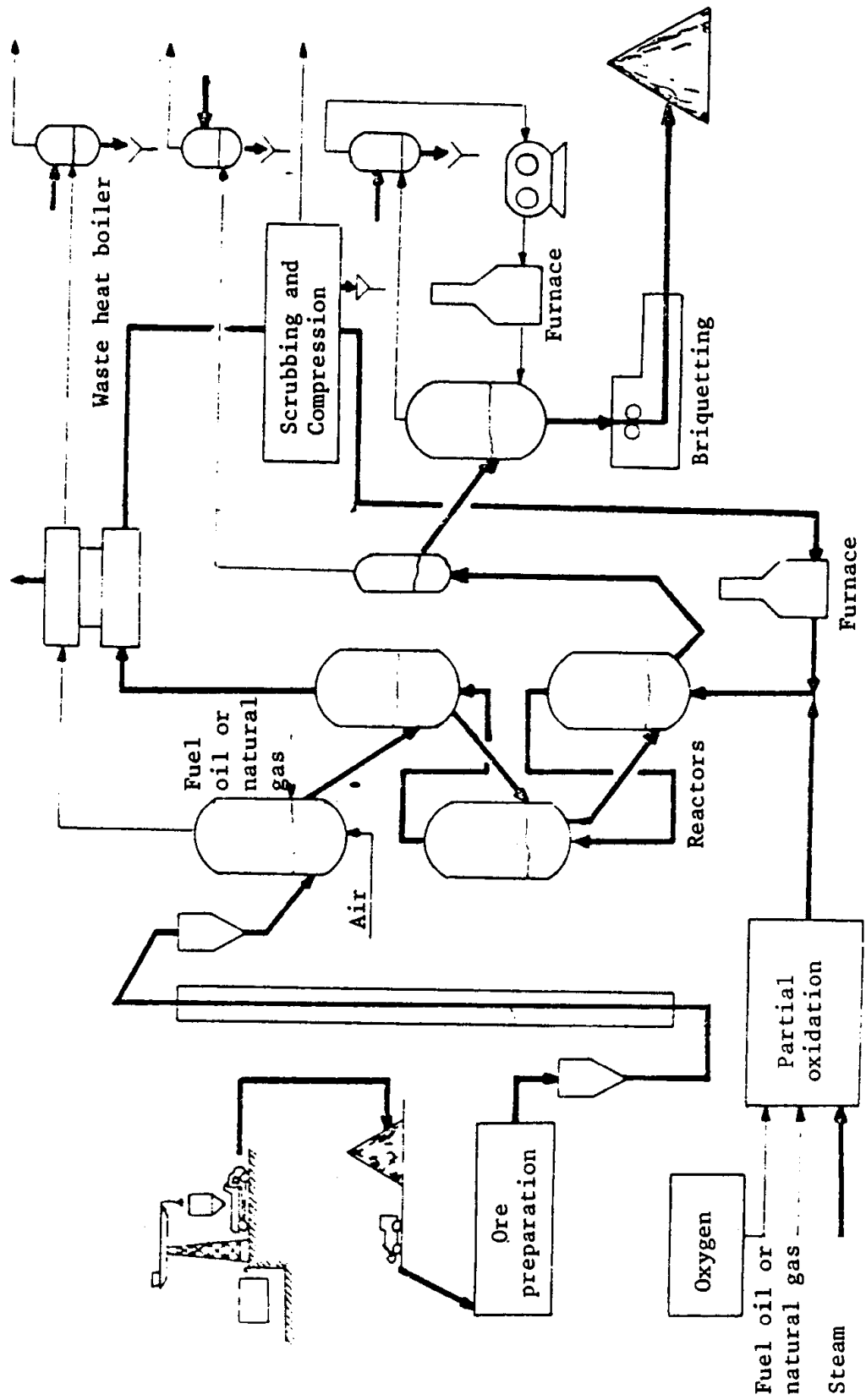


Fig. 1. Diagram of a FIOR plant

HIB PROCESS

1. Characteristics

1.1. Name and history of process

HIB (formerly Nu-Iron; see C 20)

1.2. Principle of process

Reduction by $H_2 + CO$ at approximately $800^{\circ}C$.

1.3. Inventors

E.H. Shipley, Th.F. Reed, Pittsburgh, U.S. Steel Co.

1.4. Companies involved in development

United States Steel Co. (USA)

2. General description (schematic drawing)

See diagram

3. Plant required

Production of the $H_2 + CO$ mixture by catalytic reforming of desulphurised natural gas with steam. Primary crushing of the ore to < 13 mm, drying in the rotary kiln, final crushing in a closed circuit to < 1.65 mm. Pre-heating of the ore to $325^{\circ}C$ in two stages. Fluidised bed reduction furnace in two stages. First stage: reduction to FeO . Second stage: degree of reduction $\approx 80\%$. Degree of metallisation 70%.

4. State of development

4.1. Name of laboratories where the process is situated

United States Steel

4.2. Pilot plant

4.2.1. Location

South Works of U.S. Steel, Chicago, Illinois, USA

4.2.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

Two-stage fluidised bed reactor. For the first stage of the test, a shaft 42 cm in diameter and 3 m high. For the second stage, a shaft 42 cm in diameter and 6.1 m high.

4.2.3. Capacity in tonnes per 24 h of final product

100-200 kg ore per hour, i.e. 2-3 tonnes sponge iron per day.

4.3. Industrial installations

1 plant with 3 production lines

4.3.1. Location

Puerto Ordaz, Ciudad Guyana, Venezuela

4.3.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

Reduction shaft int. dia. 6.7 m, h = 30 m, total height of reduction plant 62 m.

4.3.3. Capacity in tonnes per 24 h of final product - per unit of production

Nominal capacity 40 t per hour per line. Actual production 25-30 t per hour (for $Fe_{met} = 70\%$).

4.3.4. Method of operation (continuous or batch, number of operating hours per day)

Continuous

4.3.5. Coefficient of utilisation in time (number of working days per annum)

At present 60 - 70%

4.3.6. Maximum rated production

Nominal production 1 million tonnes p.a. of sponge iron with 70% metallisation. Maximum production at present corresponds to $\approx$ 0.6 million tonnes p.a. sponge iron with 70% metallisation.

5. Reducing agents

5.1. Raw materials on which the reducing agent is based

CVP natural gas (Compania Venezolana Petroleum)

5.2. Process for conversion of above raw materials into a form for use in reduction

Catalytic steam reforming

5.3. Condition and analysis of reducing agent at moment of use

72.5% H_2 ; 2.5% CH_4 ; 12.5% CO; 10% CO_2 ; 2.5% N_2

5.4. Preheat temperature on entering reduction apparatus

$\approx 870^{\circ}\text{C}$

5.5. Gas velocity in reduction apparatus

Not known

5.6. Analysis and temperature of gas after passing through reduction apparatus

Not known

6. Ores and ferruginous materials

Ore fines rich in iron

6.1. Source

Orinoco Mining Co. (Cerro Bolivar), Venezuela

6.2. Analysis

Moisture content 8%, $\text{Fe}_{\text{tot}} = 58.5\%$

6.3. Size

Crushing to < 13 mm, drying in a rotary kiln heated with natural gas; residual moisture content $< 1\%$; final crushing in a closed circuit to < 1.65 mm.

6.4. Preparation

Pre-heating before reduction to $315\text{--}350^{\circ}\text{C}$

7. Additions

7.1. Material

None

7.2. Analysis

Not known

7.3. Size

Not known

8. Temperature zones in reduction
in °C in the reduction apparatus

Pre-heating of ore in fluidised bed at 315°C.

Reducing gas fed in at 870°C

9. Pressure zones in reduction
in kg/cm² in the reduction apparatus

1.8 bar for the primary reduction

2.1 bar for the secondary reduction

10. Product
of the reduction apparatus

10.1. Form

Powder

10.2. Analysis

Typical analysis: $\text{Fe}_{\text{tot}} = 86.5\%$; $\text{Fe}_{\text{met}} = 60.5\%$; $\text{SiO}_2 = 1.83\%$

$\text{Al}_2\text{O}_3 = 1.93\%$; $\text{CaO} = 0.29\%$; $\text{MgO} = 0.27\%$; $\text{S} = 0.02\%$;

$\text{P} = 0.13\%$

10.3. Degree of reduction

80%

- 10.3.1. Proportion of oxygen removed from iron oxides as a percentage of the total oxygen combined with iron in the ores

80%

- 10.3.2. Metallic iron as a percentage of the total iron content of the product

70%

- 10.4. Utilisation of by-products

None

11. Further treatment of the product (10) of the reduction apparatus

- 11.1. Method of cooling

By inert gas

- 11.2. Crushing, compacting or briquetting

Hot briquetting

- 11.3. Details of final size grading

2" x 1½" x 1"

- 11.4. Method of concentration (e.g. magnetic separation or sintering)

None

12. Final product obtained after treatment according to 11

- 12.1. Form

Briquettes; density ~ 5 g/cm³

12.2. Analysis

See 10.2.

13. Iron yield

13.1. Iron contained in product (10) as a percentage of iron in the charge

Not known

13.2. Iron contained in final product (12) as a percentage of iron in the charge

Not known

14. Utilisation of final product

Destination of final product (blast furnace, electric or other furnace). Technical and economic differences as compared with using pig iron and scrap

In blast furnace; as addition to electric furnace

15. Consumption data per tonne of final product (12)

15.1. Raw materials for reduction (5.1)

Approximately 500 Nm³ natural gas per tonne sponge iron

15.2. Fuels and thermal values required for the entire process (fuel characteristics) (thermal balance)

Not known

15.3. Electrical energy

Total consumption 40 kWh per tonne sponge iron
(including crushing of ore)

15.4. Steam used

Not known

15.4.1. For reducing agent

Not known

15.5. Oxygen

None

15.6. Cooling water

Total water consumption 30 t per tonne sponge iron (for reforming and cooling)

15.7. Refractories

Not given

15.8. Man hours of productive labour

At present 0.9 man-hours per tonne sponge iron (total including maintenance).

290 people including management and administration.

15.9. Labour and materials for routine maintenance

Not known

16. Overall capital investment

16.1. For the plant described in (3) and related to one tonne of capacity for the final product (12)

Not known

16.2. Construction costs of ancillary plant not covered in (3)

Not known

17. Opinions expressed on the process in relation to:

17.1. State of development

First industrial-scale installation

17.2. Practical operating difficulties

Fines sticking to roof; porous bottom of reactor;
hot briquetting - sticking

17.3. Possibilities of exploitation

Rich ores of adequate grain size

17.4. Special requirements to be met by certain materials
(e.g. refractories)

Limited content of ultra-fines in the ore. Natural gas with
a sulphur content of 2 ppm max.

18. Bibliography

18.1. Articles and publications

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- 2) Metal Bulletin Monthly, April 1971
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18.2. Patents

- 1) See Nu-Iron process, C20

HYL PROCESS

1. Characteristics

1.1. Name and history of process

HyL process

1.2. Principle of process

Two-stage reduction of lump ore in retorts using a mixture of H_2 and CO obtained by catalytic reaction of natural gas in the presence of steam.

1.3. Inventors

There is no one single inventor; many people have participated in the development of the process.

1.4. Companies involved in development

The company which developed this process is Hojalata y Lamina S.A. The process is still being developed in a pilot plant and an industrial-scale plant at Fierro Esponja S.A. in Monterrey.

2. General description (schematic drawing)

See Figures

The natural gas is desulphurised and converted to a mixture of $CO + CO_2 + H_2 + H_2O$ using an excess of steam in the presence of the catalyst. The steam is condensed. The reducing gas is brought to a temperature of $870-1030^{\circ}C$; it first reduces partially reduced ore then fresh ore during the secondary phase

of the reduction. To obtain the desired temperature in the reactor part of the reducing gas is burnt with air pre-heated to the same temperature as the reducing gas.

200 t.p.d. plant (Fig. 1);

The plant is composed of two gas reactors and five retorts which give a complete cycle in five hours. After reduction is complete the natural gas is passed through the retort to carburise the sponge iron. The retorts are movable and can be inverted. They are held by pressure against the underside of a stationary gas seal. The sponge iron is discharged at 650°C into containers and is covered with lime to prevent oxidation. The processing takes place in the company's steelworks.

500 t.p.d. plant (Fig. 2);

The plant is composed of one gas reactor and four fixed retorts which give a complete cycle in 12 hours. The fresh reducing gas is passed through the fully reduced product to cool and carburise the sponge iron, is then cooled to condense the water and heated for reduction of the partially reduced ore. The gas leaving the first reduction stage is also cooled to condense the water and heated before entering the reactor containing the fresh ore. The residual gas from this stage is used as fuel in the plant itself. To regulate the temperature of the gas entering the reduction stages a controlled volume of pre-heated air is injected, to burn part of the gas and raise the temperature. The sponge iron is withdrawn when it is cold.

3. Plant required

Gas reactor, retorts, plant for pre-heating the natural gas, steam generator, waste heat boilers, gas cooler, gas pre-heater, means of withdrawal, and other auxiliary equipment.

4. State of development

4.1. Name of laboratories where the process is situated

Not known

4.2. Pilot plant

4.2.1. Location

Monterrey, Mexico

4.2.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

Not known

4.2.3. Capacity in tonnes per 24 h of final product

30 t.p.d.

4.3. Industrial installations

4.3.1. Location

HYLSA (Monterrey, Mexico), capacity 100,000 tonnes sponge iron per annum. (Production 1972: 95,000 t). In service since November 1967.

HYLSA (Monterrey, Mexico), capacity 270,000 tonnes sponge iron per annum. (Production 1972: 260,000 t). In service since June 1970.

TAMSA (Vera Cruz, Mexico), capacity 235,000 tonnes sponge iron per annum. (Production 1972: 220,000 t). In service since March 1967.

HYLSA (Pueblo, Mexico), capacity 350,000 tonnes sponge iron per annum. (Production 1972: 220,000 t). In service since February 1970.

USIBA (Bahia, Brazil), capacity 325,000 tonnes sponge iron per annum. In service since March 1974.

HYLSA (Monterrey, Mexico), capacity 475,000 tonnes sponge iron per annum. In service since January 1974.

All the plants are fed with ore pellets; the raw material for the reductant is natural gas.

4.3.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

200 t.p.d. plant: 5 retorts 2.44 m in diameter and 3.5 m high, with a unit capacity of 13-15 tonnes ore.

500 t.p.d. plant: 4 retorts of a unit capacity of 100 tonnes ore, 3.70 m in diameter and 13.7 m high.

Vera Cruz plant (TAMSA): Retorts 3.80 m in diameter

Monterrey plant (HYLSA): Capacity 475,000 t.p.a.

Retort 4.40 m in diameter.

4.3.3. Capacity in tonnes per 24 hr of final product - per unit of production

See 4.3.2.

4.3.4. Method of operation (continuous or batch, number of operating hours per day).

Batch operation, 24 hours per day

4.3.5. Coefficient of utilisation in time (number of working days per annum)

330 days per annum

4.3.6. Maximum rated production

See 4.3.2.

5. Reducing agents

5.1. Raw materials on which the reducing agent is based

Natural gas or light oil

5.2. Process for conversion of above raw materials into a form for use in reduction

Reaction with excess steam in the presence of a catalyst in the gas reactor. Nickel-based catalyst.

5.3. Condition and analysis of reducing agent at moment of use

Chemical composition: H_2 75%; CO 14%; CO_2 8%; CH_4 3%

5.4. Preheat temperature on entering reduction apparatus

705 - 1050°C

6. Ores and ferruginous materials

6.1. Source

Ores from various deposits - pellets: Alzada, Marcona, CVRD, etc.

6.2. Analysis

Example:

Fe: 66%; S: 0.2%; P: 0.2%; CaO: 0.5%; Al_2O_3 : 0.2%;

4% insolubles. Pellets: Fe: 67.22%; P: 0.052%;

Si: 0.02%; CaO: 1.79%; MgO: 0.75%; Al_2O_3 : 1.03%;

SiO_2 : 1.28%, Misc: 1.1%; basicity: 1.10.

6.3. Size

13-15 mm for lump ore; the pellets are preferably between 9.5 and 16 mm.

7. Additions

None

8. Temperature zones in reduction
in °C in the reduction apparatus

The gases enter at a temperature of 980-1230°C

Reaction temperature 870-1030°C

9. Pressure zones in reduction
in kg/cm² in the reduction apparatus

1st retort: 4.5 kg/cm² abs.

2nd retort: 3.5 kg/cm² abs.

3rd retort: 2.5 kg/cm² abs.

10. Product
of the reduction apparatus

10.1. Form

Sponge iron

10.2. Analysis

Example (Hylsa):

Fe _{tot}	Metallisation	C	O _{resid.}	Gangue	CaO
87.2	85.1	2.2	3.7	4.3	1.9
88.1	90.0	2.2	2.5	4.4	2.0

S and P content similar to ores used (S content reduced).

10.3. Degree of reduction

10.3.1. Proportion of oxygen removed from iron oxides as a percentage of the total oxygen combined with iron in the ores

Not known

10.3.2. Metallic iron as a percentage of the total iron content of the product

85 - 90%

10.4. Utilisation of by-products

Combustible gas

11. Further treatment of the product (10) of the reduction apparatus

11.1. Method of cooling

200 t.p.d. plant: none

500 t.p.d. plant: with cold reducing gas

12. Final product obtained after treatment according to 11

12.1. Form

See 10.1.

12.2. Analysis

See 10.2.

13. Iron yield

- 13.1. Iron contained in product (10) as a percentage of iron in the charge

100%

- 13.2. Iron contained in final product (12) as a percentage of iron in the charge

100%

14. Utilisation of final product

Destination of final product (blast furnace, electric or other furnace). Technical and economic differences as compared with using pig iron and scrap

The sponge iron is charged into electric furnaces in the steelworks (generally near the direct reduction plant). The percentage of sponge iron in the charge varies from 40 to 80%; examples: Pueblo (Hylsa) (1972): ~ 75%; Monterrey (Hylsa): (latest plant): 80%. In Mexico scrap is scarce. The sponge iron shows little oxidation. Pellets with 90% metallisation lost 1% metallisation in a year, with movable covers. Long distance transport, in rainy weather, lost 1.4% metallisation; in the dry season there were practically no metallisation losses.

15. Consumption data per tonne of final product (12)

- 15.1. Raw materials for reduction (5.1)

Not known

- 15.2. Fuels and thermal values required for the entire process
(fuel characteristics) (thermal balance)

Overall heat consumption: 4.3 Gcal (Pueblo 1970-72).

Consumption of natural gas at Pueblo (September 1973):
457 Nm³ per tonne sponge iron (natural gas 93.5% CH₄).

Expected consumption at Monterrey (latest plant):
430 Nm³/t sponge iron (natural gas 96.6% CH₄).

- 15.3. Electrical energy

Not known

- 15.4. Steam used

Not known

- 15.5. Oxygen

Not known

- 15.6. Cooling water

Water consumption: 2.5 Nm³/t sponge iron (Pueblo September 1973).

- 15.7. Refractories

Not known

- 15.8. Man hours of productive labour

0.35 h/t sponge iron (Pueblo September 1973).

- 15.9. Labour and materials for routine maintenance

0.45 h/t sponge iron (Pueblo September 1973).

16. Overall capital investment

For a HyL plant of 350,000 t.p.a. sponge iron the estimated capital expenditure is \$48 per annual tonne of sponge iron. (September 1973), but increasing the capacity reduces this. For example, according to HyL's calculations, for a plant of the order of one million tonnes per year the capital expenditure of \$36.5 per annual tonne, estimated for a guaranteed capacity, may fall to \$30 for the same productive capacity. The attached Figure shows the variations in cost per annual tonne with the design capacity or the production (Figure 3). On the other hand the production costs are reduced by about 8% by doubling the capacity of the plant.

17. Opinions expressed on the process

During the next few years developments of the HyL process are planned in several countries:

Matanzas, Venezuela - SIDOR	2,400,000 t.p.a.
Solwezi, Zambia - TIKALtd.	300,000 t.p.a.
Indonesia - Krakatau Steel	500,000 t.p.a.
Iran - National Iranian Steel Co.	1,200,000 t.p.a.
Iraq - Creusot Loire	600,000 t.p.a.
Pueblo II, Mexico - HYLSA	700,000 t.p.a.

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- 5) G.H. Müller, Metal Progress, January 1960, p.111-115 and 192-193.
- 6) R.A. Labine, Chemical Engineering, 28 December 1959, p.50-53.
- 7) F. Jäger and H. Winterhoff, Stahl und Eisen, 82 (1962) p.290-293.
- 8) D.N. Skillings, Jr., Skillings Mining Review, March 1965, p.1-5.
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- 10) Steel Times, February 1969 p.121-124.
- 11) H.E. Gearhart, K.A. Jackson - Iron and Steel Engineer, March 1971, p.53-58.
- 12) R. Lawrence, Jr. - Seminar on Direct Reduction of Iron Ore, Bucharest (Rumania) 18-23 September 1972.
- 13) J. Celada S., ILAFA Latin American Seminar, The Direct Reduction of Iron Ore and its Use in Latin America, Mexico, November 1973.
- 14) Quintero, R.G. Iron Steel Intern. 48 (1975) No.6 p.437-440.

18.2. Patents

Patent Nos.

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	1.137.058 (1963)
	1.192.008 (1965)
	1.201.377 (1966)
Belgium:	587.846 (1960)
	587.913 (1960)
	597.159 (1960)
	616.156 (1962)
	621.696 (1962)

France:	1.248.793	(1961)
	1.248.987	(1960)
	1.272.891	(1961)
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	1.303.920	(1962)
	1.318.043	(1963)
	1.319.018	(1963)
	1.319.770	(1963)
	1.342.812	(1963)
U.K.	890.189	(1962)
	906.515	(1963)
	941.885	(1963)
	946.965	(1964)
	988.579	(1965)
	952.399	(1964)
	939.910	(1964)
	940.007	(1963)
	1.004.006	(1965)
U.S.A.	3.148.050	(1964)
	3.191.920	(1965)
	3.236.628	(1966)
	3.193.266	(1965)
	3.150.680	(1964)
	3.189.438	(1965)

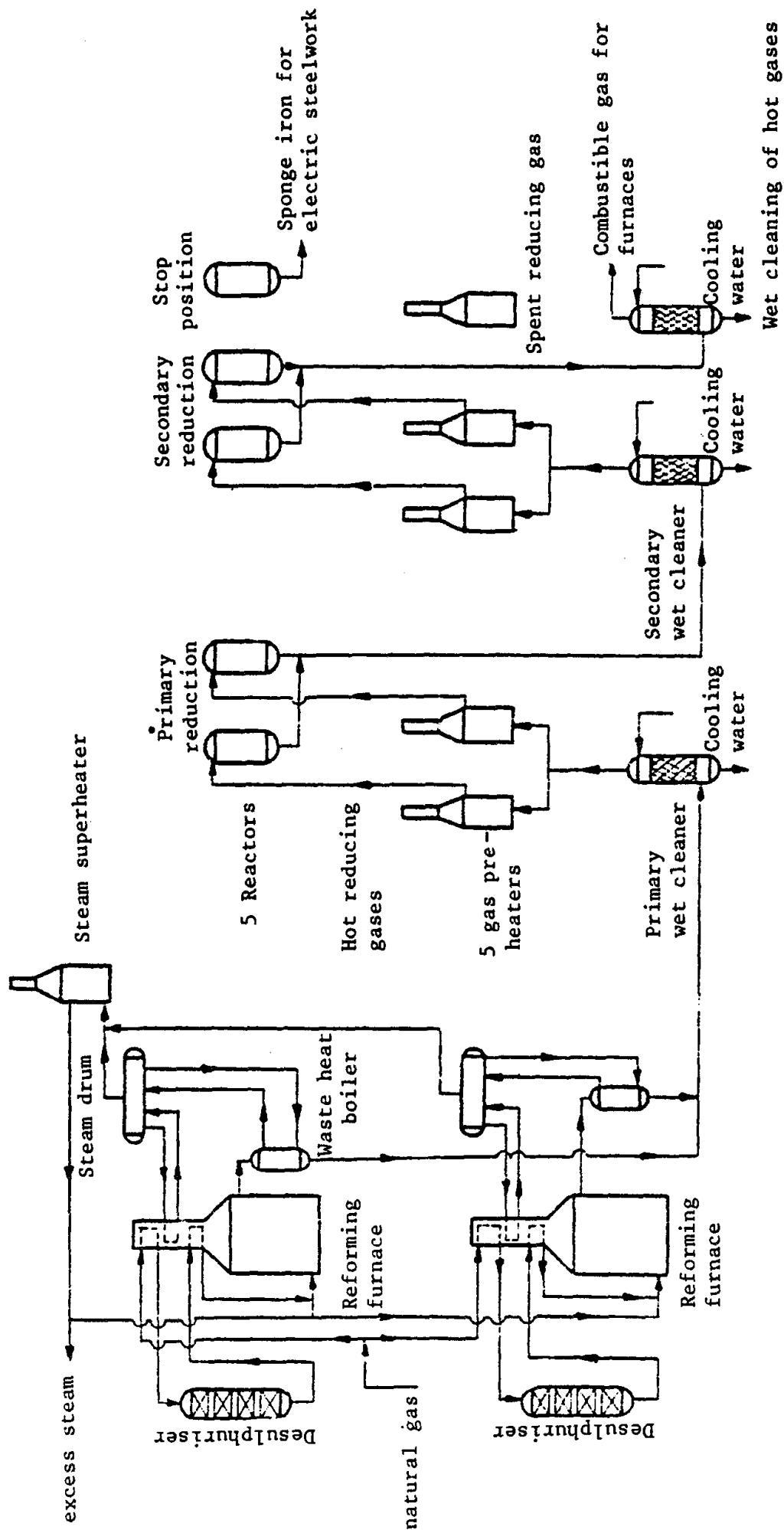


Fig. 1. Conversion and circulation of gas in Hyl process (200 t.p.d. plant)

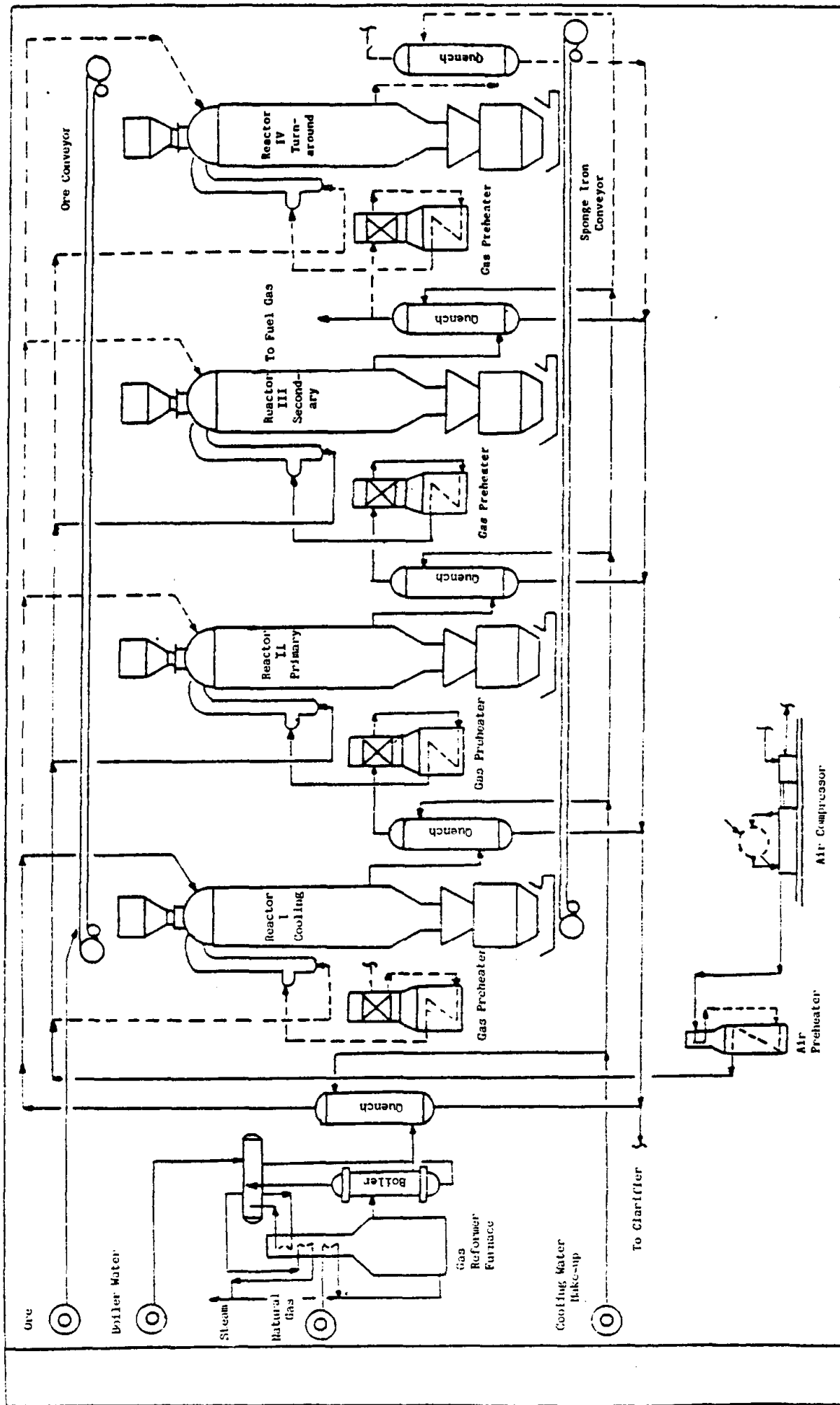


Fig. 2. Diagram of process (500 t.p.d. plant)

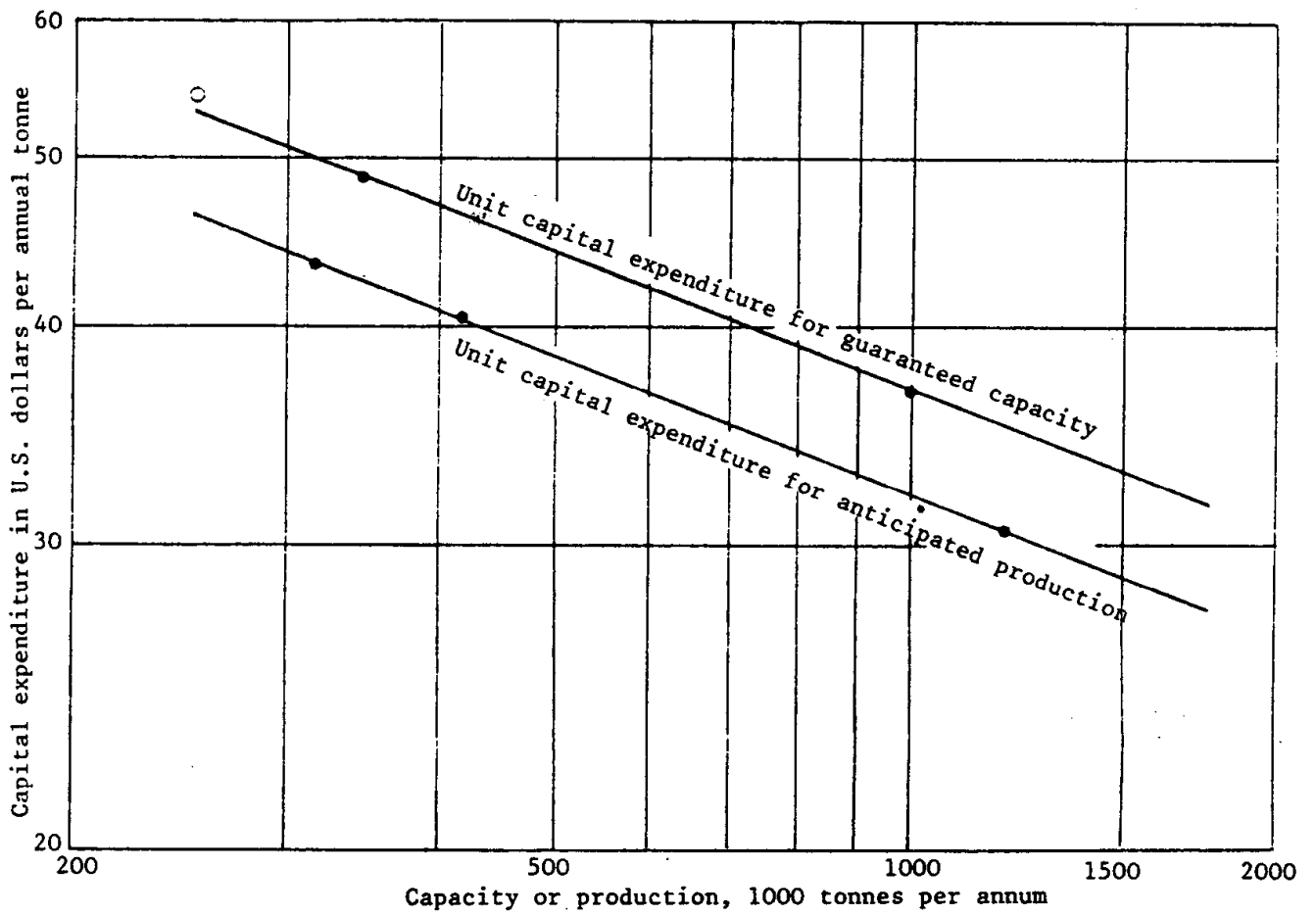


Fig. 3. Unit capital expenditure for HyL plant in relation to capacity

KAWASAKI PROCESS

1. Characteristics

1.1. Name and characteristics of process

Kawasaki process

Following earlier experience with the Krupp-Renn process (Kuji works) and with the production of oxide pellets at the Chiba works, Kawasaki built a pilot plant for 500 t per month with a view to developing a process which is suitable for direct reduction while at the same time processing the fines produced in iron and steel works.

The first industrial installation with a capacity of 5000 t per month was built at the Chiba works in December 1968. Another installation with a capacity of 15,000 t per month was commissioned in March 1972 at the Mizushima works, with satisfactory results so far.

Another installation with a capacity of 15,000 t per month is at present being built at the Chiba works and should be in operation by 1977.

1.2. Principle of process

Firing of pellets on the grate, then reduction in the rotary kiln (Grate-kiln plant).

1.3. Inventors

Kawasaki Steel Corporation

1.4. Companies involved in development

Kawasaki Steel Corporation and
Kawasaki Heavy Industries Ltd.

2. General description (schematic drawing)

See Figures 1 and 2

3. Plant required

Disc pelletiser, travelling grate, rotary kiln, rotary cooler

4. State of development

4.1. Name of laboratories where the process is situated

Chiba and Mizushima works of the Kawasaki Steel Corporation

4.2. Pilot plant

4.2.1. Location

Chiba

4.2.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

Rotary kiln 1.8 m in diameter and 25 m long

4.2.3. Capacity in tonnes per 24 hr of final product

25 t.p.d.

4.3. Industrial installation

4.3.1. Location

a) Chiba works

b) Mizushima works

4.3.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

- a) Grate with an area of 3.15 m x 15 m
rotary kiln 3.6 m in diameter and 40 m long
- b) Grate with dimensions 4 m x 32 m
kiln 5 m in diameter and 50 m long

4.3.3. Capacity in tonnes per 24 hr of final product - per unit of production

- a) 60,000 t.p.y.
- b) 180,000 t.p.y.

4.3.4. Method of operation (continuous or batch, number of operating hours per day)

Continuous

4.3.5. Coefficient of utilisation in time (number of working days per annum)

310 days per year

4.3.6. Maximum rated production

Not known

5. Reducing agents

5.1. Raw materials on which the reducing agent is based

Solid carbon

5.2. Process for conversion of above raw materials into a form for use in reduction

None

5.3. Condition and analysis of reducing agent at moment of use

Coke breeze with approximately 85% C_{fix} , 1.5% volatile matter, 0.7% S and 14% ash

5.4. Preheat temperature on entering reduction apparatus

No preheating of the reducing agent

5.5. Gas velocity in reduction apparatus

Not known

5.6. Analysis and temperature of gas after passing through reduction apparatus

Not known

6. Ores and ferruginous materials

6.1. Source

Blast furnace dusts, blast furnace slurry, sinter fines, LD fines

6.2. Analysis

	Blast furnace dust	Blast furnace slurry	Sinter fines	LD dust
% Fe_{tot}	32.2	28.6	43.5	64.8
% C	38.5	40.8	3.5	-
% S	0.2	0.3	0.2	0.03
% Zn	0.45	0.54	0.03	1.06

6.3. Size

325 mesh:	blast furnace fines	5 %
	blast furnace slurry	59 %
	sinter fines	61 %
	LD fines	90 %

6.4. Preparation

Pelletising of the charge materials

7. Additions

None

8. Temperature zones in reduction
in °C in the reduction apparatus

Gas temperature 1100-1200 °C

9. Pressure zones in reduction
in kg/cm² in the reduction apparatus

Not known

10. Product
of the reduction apparatus

10.1. Form

Reduced pellets

10.2. Analysis

74.3 % Fe_{tot}; 70.6 % Fe_{met}; 0.02 % Zn; 0.01 % Pb

10.3. Degree of reduction

95 %

10.4. Utilisation of byproducts

Not known

11. Further treatment of the product (10) of the reduction apparatus

11.1. Method of cooling

Water cooling in the rotary cooler

11.2. Crushing, compacting or briquetting

None

11.3. Details of final size grading

Pellets with an average diameter of 13 mm,
90 % of the pellets are within the range 5 - 20 mm

11.4. Method of concentration (e.g. magnetic separation or sintering)

None

12. Final product obtained after treatment according to 11

12.1. Form

Pellets

12.2. Analysis

See 10.2

13. Iron yield

85 - 90 %

14. Utilisation of final product

Destination of final product (blast furnace, electric or other furnace). Technical and economic differences compared with the use of pig iron and scrap

Use of the pellets > 5 mm in the blast furnace
 < 5 mm in the sintering plant

15. Consumption data per tonne of final product (12)

15.1. Raw materials for reduction (5.1)

300 - 500 kg/t

15.2. Fuels and thermal values required for the entire process (fuel characteristics) (thermal balance)

Fuel oil 12 - 60 l/t
Heat 3.5 - 4.2 Gcal/t

15.3. Electrical energy

15.3.1. For heating

None

15.3.2. For power

100 - 130 kWh

15.4. Steam used

15.4.1. For reducing agent

None

15.4.2. As energy

30 - 50 kg/t pellets

15.5. Oxygen

0.05 - 0.1 m³/t pellets

15.6. Cooling water

1 - 3 Nm³/ pellets

15.7. Refractories

7 - 10 kg/t pellets

15.8. Man hours of productive labour

0.5 - 1.5 man hours per tonne pellets

15.9. Labour and materials for routine maintenance

5 - 7 % of the total costs

16. Overall capital investment

a) Installations at Chiba works: 5 million US \$

b) Installations at Mizushima works: 15 million US \$

17. Opinions expressed on the process in relation to:

17.1. State of development

The development work is completed. Technical licences are granted on request.

17.2. Practical operating difficulties

Operation is regulated by changes in the analysis of the blast furnace fines

17.3. Possibilities of exploitation

Use for ore fines and all kinds of blast furnace fines

17.4. Special requirements to be met by certain materials (e.g. refractories)

None

18. Bibliography

18.1. Articles and publications

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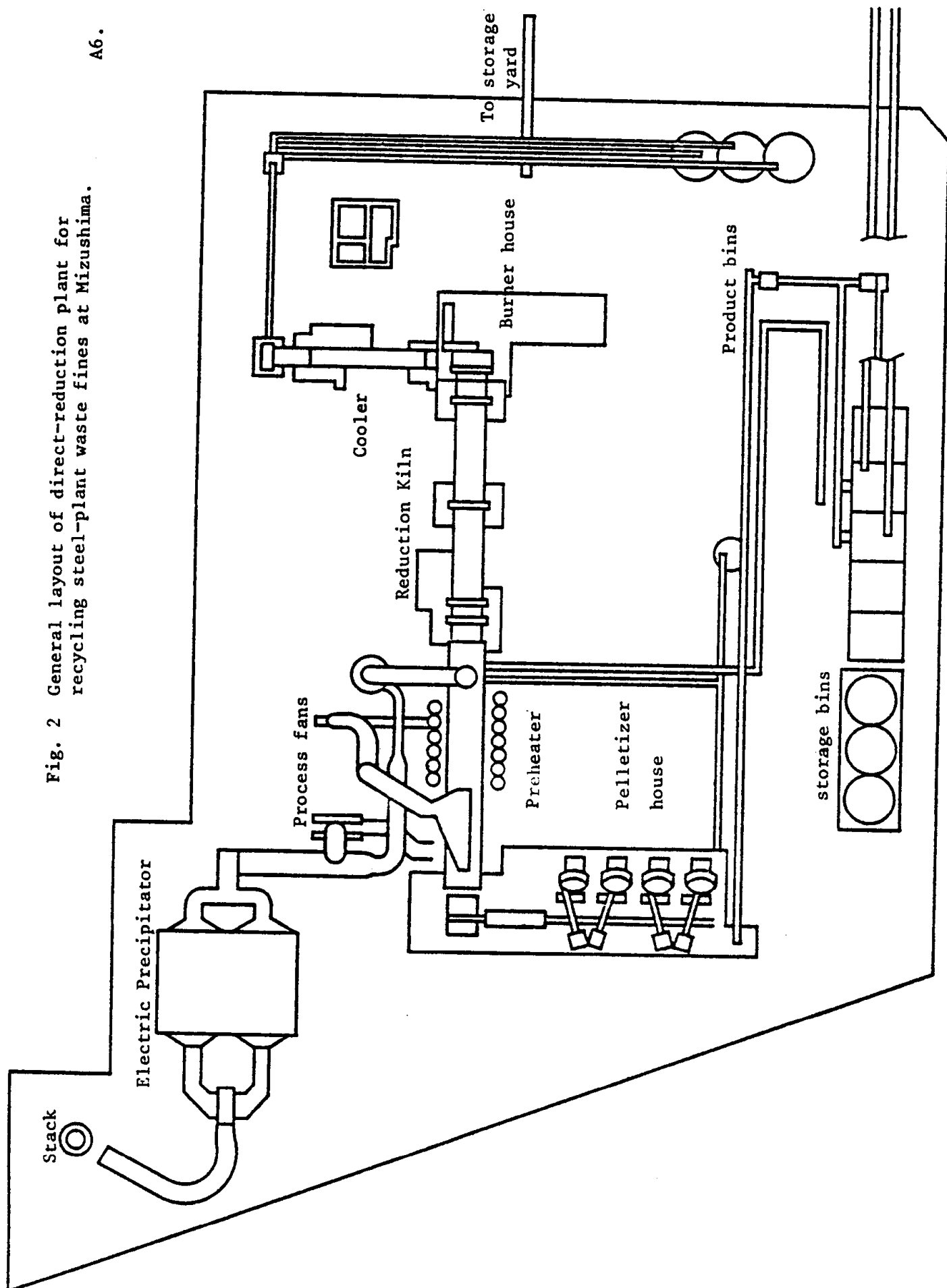
18.2. Patents

5 patents granted and 5 applied for

Fig. 1 Flowsheet of Chiba direct reduction plant

Fig. 2 General layout of direct-reduction plant for recycling steel-plant waste fines at Mizushima.

A6.



KINGLOR-METOR PROCESS

1. Characteristics

1.1. Name and history of process

Process derived from the Echeverria process, the layout being substantially the same with some improvements in plant. In 1968 Monteforno (Switzerland) and Danieli and Co (Italy) set up a joint effort in the field of direct reduction of iron ores. Kinglor-Metor Mineraria & Metallurgica S.p.A. was founded in 1971 (50% holding by Monteforno and 50% by Danieli and Co) with the aim of developing "the processing of ores and the production and processing of steels in accordance with the Kinglor Anstalt processes and patents".

1.2. Principle of process

The same as the Echeverria process. Reduction of iron ore as graded lump ore or pellets with a solid reducing agent and the limestone necessary to neutralise the sulphur in the coal. The raw materials are charged into a chamber furnace of rectangular section and unusual height; the heat is supplied from an outer chamber in which liquid or gaseous fuel is fired. The charge descends through the pre-heating, reduction and cooling zones. The sponge iron is separated by means of a magnetic separator and the residual reducing agent is recycled.

1.3. Inventors

Kinglor Metor (Italy)

1.4. Companies involved in development

Kinglor Metor S.p.A., Buttrio, Italy.

2. General description (schematic drawings)

See Figure 1.

3. Plant required

Ore crushing and screening plant (not needed when using pellets);
Reduction furnace with pre-heating of ore;
Furnace for heating the retort;
Means of cooling and discharging the sponge iron;
Magnetic separator.

4. State of development

4.1. Name of laboratories where the process is situated

In 1971 it was decided to build a pilot plant with two retorts.
The project was prepared in 1972 and put into effect in September 1973.

The pilot plant should also be regarded as an industrial plant since each retort constitutes a reduction unit and six retorts constitute a unit of industrial scale.

4.2. Pilot plant

4.2.1. Location

Buttrio, Italy

4.2.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

Two reactors of rectangular section 300 to 600 x 1,600 mm;
height 12 m (including the pre-heating zone 2 m). Reactors constructed of silicon carbide refractory.

4.2.3. Capacity in tonnes per 24 h of final product

4,000 t.p.a. sponge iron for each reactor (approximately 10 t.p.d.) The complete cycle takes 24 hours - 12 h in the retort for heating and reduction, 12 h for cooling.

4.3. Industrial installations

See 4.2., continuous operation

5. Reducing agents

5.1. Raw materials on which the reducing agent is based

Metallurgical coke, various coals.

5.2. Process for conversion of above raw materials into a form for use in reduction

Not known

5.3. Condition and analysis of reducing agent at moment of use

Not known

5.4. Preheat temperature on entering reduction apparatus

The reducing agent is heated at the same time as the ore in the upper part of the retort.

6. Ores and ferruginous materials

Iron ore, graded lumps, sinter or pellets (variable Fe content)

6.3. Size

Size of lump ore: 6.5 - 25 mm

7. Additions

Addition of limestone, dolomite, etc. to neutralise the sulphur in the fuel

8. Temperature zones in reduction
in °C in the reduction apparatus

Pre-heating temperature: 850°C

Reduction temperature : 1050°C

9. Pressure zones in reduction
in kg/cm² in the reduction apparatus

Not known

10. Product
of the reduction apparatus

10.1. Form

Mixture of sponge iron and residual reducing agent.

10.2. Analysis

Not known

10.3. Degree of reduction

90 - 94% (> 90%)

10.4. Utilisation of by-products

Part of the gas produced during reduction is burned in the pre-heating zone. The excess reducing agent is separated magnetically from the metallurgical product and recycled.

11. Further treatment of the product (10) of the reduction apparatus

11.1. Method of cooling

Indirect, water cooling

11.2. Crushing, compacting or briquetting

Not known

11.3. Details of final size grading

Not known

11.4. Method of concentration (e.g. magnetic separation or sintering)

The material from the retort is screened to 3 mm. The fractions are put through a magnetic separator. The +3 fraction gives rise to a magnetic part (sponge iron) and a non-magnetic part (recycled coal). The -3 fraction gives magnetic fines usable like sponge and non-magnetic fines (waste).

12. Final product obtained after treatment according to 11

Not known

13. Iron Yield

Not known

14. Utilisation of final product

Destination of final product (blast furnace, electric or other furnace). Technical and economic differences as compared with using pig iron and scrap „

The product may be used in the steelworks or foundry. In view of the dimensions and modular characteristics of the plant it is particularly suitable for mini-steelworks.

15. Consumption data per tonne of final product (12)

15.1. Raw materials for reduction (5.1)

1350 kg iron ore (63.5% Fe) per tonne sponge iron

310 kg solid reductant (6,000 kcal/kg) per tonne sponge iron

15.2. Fuels and thermal values required for the entire process (fuel characteristics) (thermal balance)

Solid fuel for reduction (6,000 kcal/kg): 310 kg per tonne sponge iron. Fuel for heating: L.P.G. (12,000 kcal/kg): 134 kg; or natural gas (8,300 kcal/Nm³): 193 Nm³. Thermal balance per tonne sponge iron:- heat input: 3,8 Gcal (100%); utilisation (%): carbon in the sponge iron 1%; drying the materials 3%, coal fines (waste) 7%; losses through the walls: 7%; cooling of the sponge iron: 11%; losses in the off gases: 19%; heating: 14%; reduction 38%. Recycling comprises 24% of the total in circulation (1.2 Gcal). The thermal yield is 52%. The losses without recovery of heat other than recycling of the coal are 48%.

15.3. Electrical energy

80 kWh/tonne sponge iron

15.4. Steam used

None

15.5. Oxygen

None

15.6. Cooling water

Not known

15.7. Refractories

Not known

15.8. Man hours of productive labour

1.5 h/tonne sponge iron (management not included)

16. Overall capital investment

16.1. For the plant described in (3) and related to one tonne of capacity for the final product (12)

The capital expenditure for a small unit (20,000 tonnes per annum) is less than the costs of plant for the other processes with capacities greater than 300,000 t.p.a. These costs do not include offices, various additional departments and some civil engineering work.

17. Opinions expressed on the process:

At present the industrial pilot plant at Buttrio (Udine, Italy), is being used for tests on various raw materials, and the development of new machines and materials. Kinglor-Metor S.p.A. has received the order for the first industrial plant from the Italian company, Indumetal.

The capacity of this plant (comprising two units, extendable to four, each containing six furnaces) will be 40,000 t.p.a.

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18.2. Patents

Not known

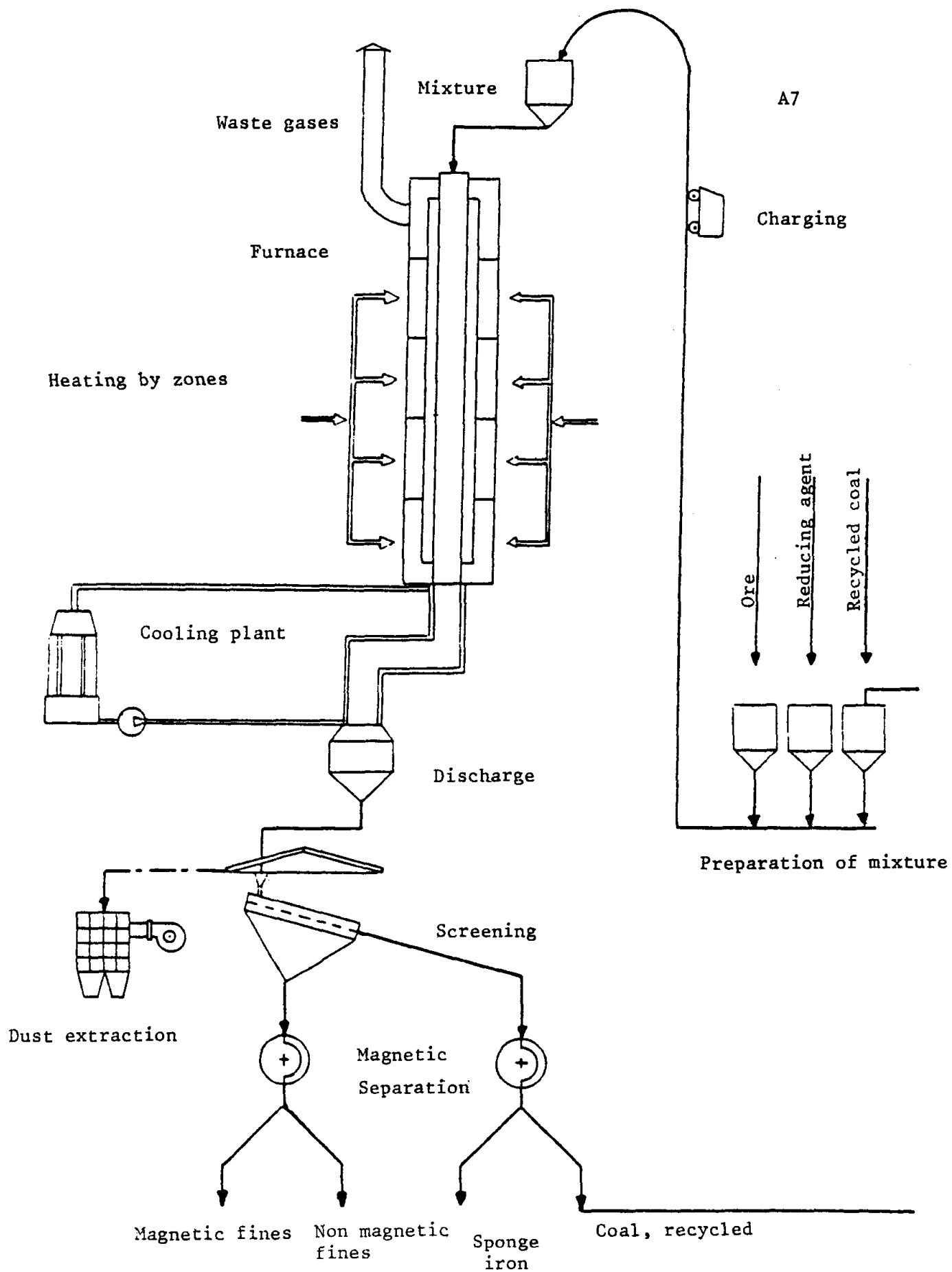


Fig.1. Diagram of Kinglor-Metor process.

KOHO PROCESS

1. Characteristics

1.1. Name and history of process

Koho process

From 1971 until 1975 reduction roasting tests were carried out in a small rotary kiln at the Washibetsu Laboratory of Koho Kogyo KK and the industrial plant, of 7000 t per month capacity, was commenced in October 1971.

1.2. Principle of process

Carbon-containing pellets are processed using a horizontal drier and a rotary roasting reduction kiln.

1.3. Inventors

Koho Kogyo Kabushiti Kaisha

1.4. Companies involved in development

Muroran Works (Nippon Steel Corporation)

2. General description

See flow sheet

3. Plant required

Two pre-driers, rotary mixing crusher, pelletizer, horizontal drier, rotary roasting reduction kiln

4. State of development

4.1. Name of laboratories where process is situated

Washibetsu Laboratory (Koho Kogyo KK)

4.2. Pilot plant

4.2.1. Location

Ashibetsu-cho, Noboribetsu-city, Hokkaido, Japan

4.2.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

Disc pelletizer, 0.9 m in diameter and 0.28 m deep

2 rotary reduction kilns:

one 0.7 m in diameter, 4.4 m long

one 1.0 m in diameter, 12.0 m long

4.2.3. Capacity in tonnes per 24 hr of final product

1) 0.7 m dia. kiln, 1.0 t.p.d.

2) 1.0 m dia. kiln, 5 t.p.d.

4.3. Industrial installations

4.3.1. Location

5, Naka-mach, Muroran-city, Kokkaido, Japan

(on the site of Muroran works, Nippon Steel Corp.)

4.3.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

rotary grinding mill dia. 3.0 m, length 4.0 m

pelletizer dia. 4.5 m, depth 0.7-1.0 m

horizontal drier width 2m, length 50 m

rotary roasting reduction kiln, dia. 3.3 m, length 24 m

4.3.3. Capacity in tonnes per 24 hr of final product - per unit of production

84,000 t.p.y. (of raw materials treated)

4.3.4. Method of operation

Continuous

4.3.5. Coefficient of utilisation with time (number of working days per annum)

310 days per year

4.3.6. Maximum rated production

50,000 t.p.y.

5. Reducing agents

5.1. Raw materials on which the reducing agent is based

Carbon contained in the blast furnace primary and secondary dust.

5.2. Process for conversion of above raw materials into a form for use in reduction

During the process the carbon-containing raw materials are brought to a form suitable for reduction

5.3. Condition and analysis of reducing agent at moment of use

Not known

5.4. Preheat temperature on entering reduction apparatus

150°C (preheat temperature of pellets containing carbon)

5.5. Gas velocity in reduction apparatus

2-3 m/sec (in the rotary kiln)

5.6. Analysis and temperature of gas after passing through reduction apparatus

CO₂ 20-28 %

O₂ 0- 2 %

CO 0- 3 %

N₂ 72-75 %

6. Ores and ferruginous materials

6.1. Source

In-plant dust from Muroran works, Nippon Steel Corporation

6.2. Analysis

	C _{tot}	Fe _{tot}	Fe ₂ O ₃	SiO ₂	CaO	Al ₂ O ₃
BOF dry dust	-	66	92	1	1	0.1
BOF wet dust	-	63	88	2	1	0.1
Sinter plant dust	-	53	71	7	9	3
BF primary dust	31	26	30	6	3	3
BF secondary dust	43	27	34	6	4	3

6.3. Size

	140 μm	140-74 μm	74-44 μm	44 μm
BOF dry dust	0	1.0	1.0	98.0
BOF wet dust	30	18	7.0	45
Sinter plant dust	1	7	13	79
BF primary dust	49	25	10	19
BF secondary dust	8	18	12.0	67

7. Additions

7.1. Material

Bentonite

7.2. Analysis

Not known

7.3. Size

97.9% under 10 microns

8. Temperature zones in reduction
in $^{\circ}\text{C}$ in the reduction apparatus

max. 1150°C

9. Pressure zones in reduction
in kg/cm^2 in the reduction apparatus

None

10. Product of the reduction apparatus

10.1. Form

Pellets

10.2. Analysis

Size, mm	%	Fe _{tot}	Fe _{met}	Fe ₂ O ₃	SiO ₂	Al ₂ O ₃	CaO
16 mm		68	45	7	8	2	6
5-16 mm		73	56	7	9	2	6
5 mm		62	72	7	9	2	5

10.3. Degree of reduction

80-90 %

10.4. Utilisation of the by-products

Product under 5 mm in size is utilised as a sinter material

11. Further treatment of the product (10) of the reduction apparatus

11.1. Method of cooling

Water spray cooling

11.2. Crushing, compacting or briquetting

Portion over 16 mm in size is crushed

11.3. Details of final size grading

Pellets under 16 mm in size

11.4. Method of concentration

None

12. Final product obtained after treatment according to 11

12.1. Form

Granular, over 5 mm and under 16 mm in size

12.2. Analysis

72 % Fe_{tot}, 53 % Fe_{met}, 7 % Fe₂O₃, 7 % SiO₂, 2 % Al₂O₃, 6 % CaO

13. Iron yield

13.1. For product according to 10

97-98 %

13.2. For production

92-93 %

14. Utilisation of final product

Destination of final product (blast furnace, electric or other furnace). Technical and economic differences compared with the use of pig iron and scrap

Utilised as a raw material for the blast furnace

15. Consumption data per tonne of final product

15.1. Raw materials for reduction (5.1)

Coke breeze contained in blast furnace dust

15.2. Fuels and thermal values required for the entire process
(fuel characteristics) (thermal balance)

Fuel: heavy oil (class C)

characteristics:

analysis: C = 86.4 %
H = 11.4 %
S = 2.2 %

specific gravity: 0.942 g/cm³

lower calorific value H_u : 9.840 kcal/kg

Thermal balance:

1) Thermal balance of the reduction kiln (kcal/t final product)

Input			Output		
	kcal	%		kcal	%
Combustion heat of heavy oil	730,000	27.8	Reduction heat of Fe oxide	1,200,000	45.6
Combustion heat of carbon	1,900,000	72.2	Unburnt carbon	120,000	4.6
			Sensible heat of pellets	250,000	9.5
			Sensible heat of waste gas	500,000	19.0
			Radiation losses, etc.	560,000	21.3
Total	2,630,000	100.0		2,630,000	100.0

2) Thermal values required for other uses (kcal/t final product)

raw material preparation	190,000 kcal/t
other	30,000 kcal/t
Total	220,000 kcal/t

15.3. Electrical energy

15.3.1. For heating

None

15.3.2. For power

110 kWk/t

15.4. Steam used

None

15.5. Oxygen

None

15.6. Cooling water

2 m³/t (including cooling water for the machinery)

15.7. Refractories

2 kg/t

15.8. Man hours of productive labour

8 persons per shift

16. Overall capital investment

About 500 million yen

17. Opinions expressed on the process in relation to:

17.1. State of development

Successful

17.2. Practical operating difficulties

Deposits of dust exhausted from rotary kiln on the blade and inside the box of the suction fan.

17.3. Possibilities of exploitation

Good possibility of application with high flexibility as regards requirements in the matter of the condition of the raw materials

17.4. Special requirements to be met by certain materials (e.g. refractories)

None

18. Bibliography

18.1. Articles and publications

None

18.2. Patents

3 patents applied for relating to the process and the pelletizing

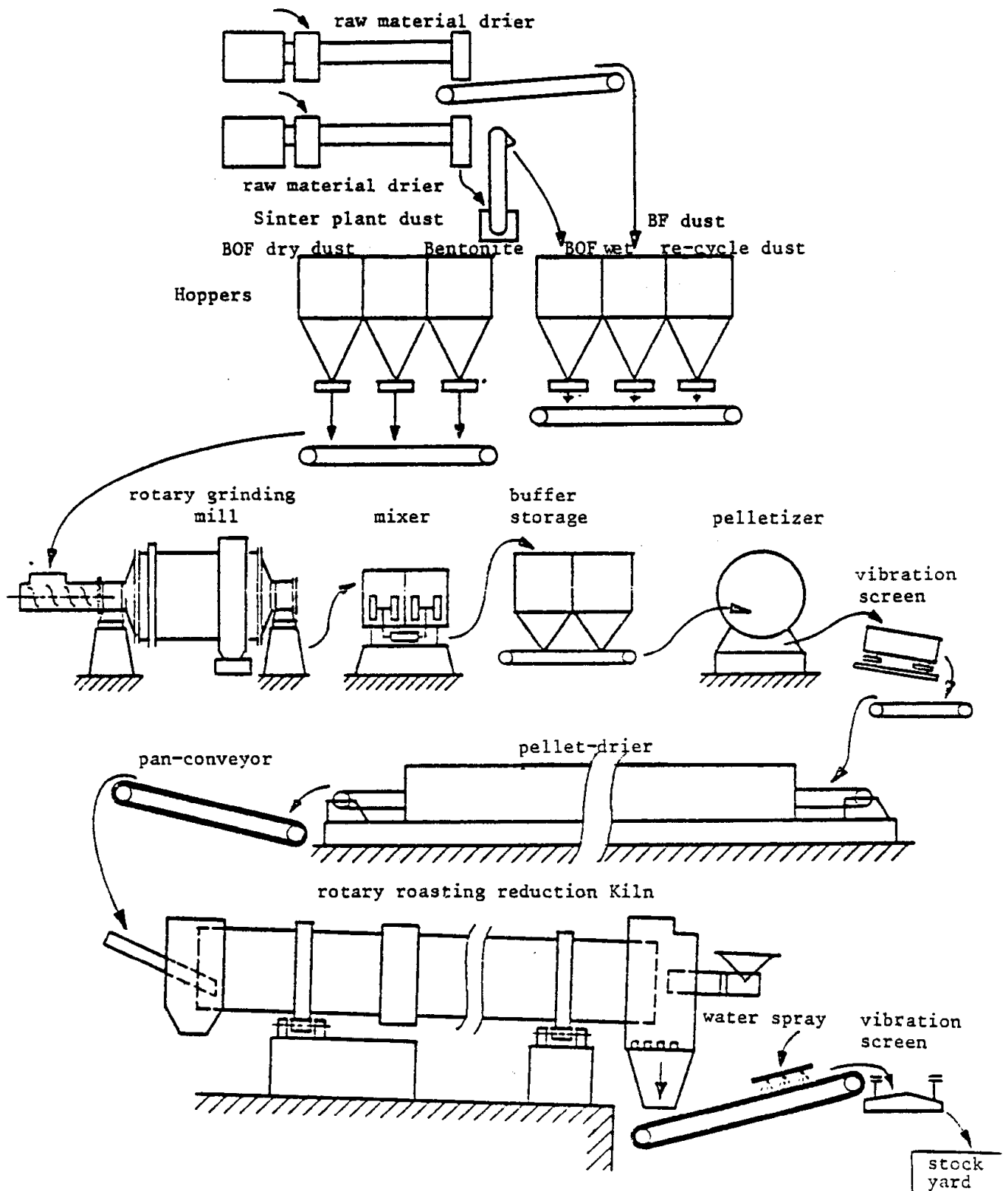


Fig. 1. General description KOHO process.

KRUPP SPONGE IRON PROCESS

1. Characteristics

1.1. Name and history of process

Krupp sponge iron process. First trials on a semi-industrial scale 1957, large-scale test 1961, first industrial plant 150,000 annual tonnes capacity 1973.

1.2. Principle of process

Reduction of rich iron ores in lumps or pellets with solid reducing agents in a rotary kiln to produce sponge iron

1.3. Inventors

See below

1.4. Companies involved in development

Joint development by Fried. Krupp GmbH, Krupp Forschungsinstitut (Research Institute), Essen and Krupp Industrie- und Stahlbau, Rheinhausen

2. General description (schematic drawing)

The characteristic feature of the Krupp sponge iron process (Fig.1) is the use of a rotary kiln. Practically all solid fuels (anthracite, coke fines, long flame gas coal, lignites, low-temperature coke) can be used as reductants. Lump ore or pellets are fed into the rotary kiln together with the reducing agent and limestone or crude dolomite for desulphurising purposes: they are heated in the kiln in the countercurrent to the kiln gases

and reduced to sponge iron. Depending upon its gas content the coal is either blown in at the discharge end or fed in with the ore. The mixture of sponge iron excess fuel and ash is cooled to below 150°C in an adjoining cooling drum by direct water spraying. The sponge iron is separated from the excess fuel and ash by screening and flotation. The kiln is heated from the discharge end with a burner; some of the air for combustion is supplied via several nozzles on the kiln shell. This enables a specific temperature profile to be achieved.

3. Plant required

Rotary kiln in an inclined position with peripheral nozzles for the combustion air. Countercurrent heating; cooling drum for direct and indirect water cooling of the kiln product. The sponge iron is then separated from the excess fuel, ash and desulphurising agent by a combination of screening, magnetic separation and flotation.

4. State of development

4.1. Name of laboratories where the process is situated

Fried Krupp GmbH, Krupp Industrie- und Stahlbau, Rheinhausen

4.2. Pilot plant

4.2.1. Location

Rotary kiln experimental plant, located in Essen

4.2.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

Length of kiln: 13 m

Diameter of kiln inside the refractory brickwork: 0.9 m

Diameter of kiln shell: 1.2 m

Inclination of kiln: between 0.5 and 5%

Speed of rotation of kiln: varies between 0.3 and 2.0 rpm

Travelling grate: 4.0 x 0.5 m

4.2.3. Capacity in tonnes per 24 h of final product

5 to 10 t of sponge iron per 24 hours

4.3. Industrial installations

Sponge iron plant 150,000 annual tonnes, Dunswart Iron and Steel Works (Pty) Ltd.

4.3.1. Location

Benoni/TVL, S. Africa

4.3.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

Length of kiln: 73.5 m

Diameter of kiln inside brickwork: 4.1 m

Diameter of kiln shell: 4.6 m

4.3.3. Capacity in tonnes per 24 h of final product - per unit of production

470 t of sponge iron per 24 hours with a degree of metallisation of about 93%

4.3.4. Method of operation (continuous or batch, number of operating hours per day)

Continuous 24 hours a day

4.3.5. Coefficient of utilisation in time (number of working days per annum)

320 working days per annum

4.3.6. Maximum rated production

470 t of sponge iron per 24 hours

5. Reducing agents

5.1. Raw materials on which the reducing agent is based

Bituminous coal, lignite

5.2. Process for conversion of above raw materials into a form for use in reduction

No special preparation of the reducing agent

5.3. Condition and analysis of reducing agent at moment of use

Where necessary, various sizes of coke dust, anthracite, long flame gas coal, lignite and low temperature coke (max. 25% moisture) are used.

5.4. Preheat temperature on entering reduction apparatus

None

5.5. Gas velocity in reduction apparatus

Not known

5.6. Analysis and temperature of gas after passing through reduction apparatus

Not known

6. Ores and ferruginous materials

The use of sponge iron for direct production of steel requires ores with over 65% Fe (hematite) or over 67.5% (magnetite) with a particle size of between 5 and 25 mm or calcined pellets or fines or concentrates transformed into green pellets. The choice of size depends on the reducibility.

7. Additions

7.1. Material

Desulphurising agents: crude dolomite or limestone; binder for pelletising: bentonite

7.2. Analysis

No particular specifications

7.3. Size

Desulphurising agent 1 to 3 mm with max. 20% undersize, bentonite of commercial size.

8. Temperature zones in reduction
in °C in the reduction apparatus

Maximum temperature of the material 1100°C

9. Pressure zones in reduction
in kg/cm² in the reduction apparatus

Normal atmospheric pressure

10. Product
of the reduction apparatus

10.1. Form

Sponge iron in lumps or pellets

10.2. Analysis

Not known

10.3. Degree of reduction

Degree of metallisation 92 to 95%, total iron content depends on the original ore. The same applies for gangue and phosphorus. Sulphur content: approximately 0.03%

10.4. Utilisation of by-products

Residue: mixture of desulphurising agent and ash (waste); off gas from rotary kiln with 0 to 1 vol. % CO + H₂, about 27 vol. % CO₂ (via chimney into atmosphere); filter dust (waste).

11. Further treatment of the product (10) of the reduction apparatus

11.1. Method of cooling

Indirect cooling with fine jets of water to about 150°C

11.2. Crushing, compacting or briquetting

The material discharged from the kiln is divided into sponge iron, excess fuel and ash; see also section 3.

11.3. Details of final size grading

Final particle size = original size

11.4. Method of concentration (e.g. magnetic separation or sintering)

See 3.

12. Final product obtained after treatment according to 11

12.1. Form

See 10.1. and 11.3.

12.2. Analysis

90 to 92% Fe total iron, 92 to 95% metallisation,
0.03% Si, 0.5% C, depending on the ore

13. Iron yield

13.1. Iron contained in product (10) as a percentage of iron
in the charge

Approximately 95%

13.2. Iron contained in final product (12) as a percentage of
iron in the charge

Approximately 95%

14. Utilisation of final product

Destination of final product (blast furnace, electric or other
furnace). Technical and economic differences as compared with
using pig iron and scrap

Almost completely free from undesirable tramp elements such as
Cu, Zn, Pb, Cd and uniform composition provided that suitable
ores are available; good flow properties for continuous charging
in steel production (electric arc furnace, LD converter)

15. Consumption data per tonne of final product (12)
- 15.1. Raw materials for reduction (5.1)
 1.45 t of ore per t of sponge iron
- 15.2. Fuels and thermal values required for the entire process
 (fuel characteristics) (thermal balance)
 3.8 - 4.0 Gcal/t Fe
- 15.3. Electrical energy
 55 kWh/t sponge iron
- 15.4. Steam used
 None
- 15.5. Oxygen
 Not necessary
- 15.6. Cooling water
 1.5 m³/t sponge iron
- 15.7. Refractories
 3 kg/t sponge iron
- 15.8. Man hours of productive labour
 0.5 h/t sponge iron
- 15.9. Labour and materials for routine maintenance
 Personnel: 0.4 h/t sponge iron; material approximately 6 DM
 per tonne of sponge iron (1975)

16. Overall capital investment

- 16.1. For the plant described in (3) and related to one tonne of capacity for the final product (12)

Capacity expressed in tonnes per day or per year.

Investment costs depend on local conditions and the quality of the raw materials and only approximate values can therefore be given. For a daily output of 625 tonnes of sponge iron with one furnace, which is equal to 200,000 annual tonnes, starting with the raw material bunkers and ending with the bunkers containing the product, the costs in German currency are at present about 40 million DM (1975).

- 16.2. Construction costs of ancillary plant not covered in (3)

Not known

17. Opinions expressed on the process in relation to:

- 17.1. State of development

It has been proved that the process can be used industrially

- 17.2. Practical operating difficulties

No operating difficulties have been experienced

- 17.3. Possibilities of exploitation

The sponge iron process is suitable for countries with rich ore deposits or ores which can be treated to give concentrates rich in iron and with low grade solid fuels.

The process can be used for the pre-reduction of ores containing non-ferrous metals (industrial plant supplied for LARCO, Société Minière et Métallurgique de Larymma, S.A. Larymma-Lokris, Greece;

throughput of laterite iron ores containing Ni: about 4000 t.p.d.; rotary kiln: 4.2 m dia x 90 m (1956); 4.2 m dia x 90 m (1970); 5.2 m dia x 90 m (1972).

Modified process for dezincing recycled materials in a metallurgical smelting plant and making it into lumps; the process has been tried out on a semi-industrial scale (processing of dusts and sludges from blast furnaces and steelmaking furnaces. The metallised product can be reused in the blast furnace).

17.4. Special requirements to be met by certain materials (e.g. refractories)

No special requirements

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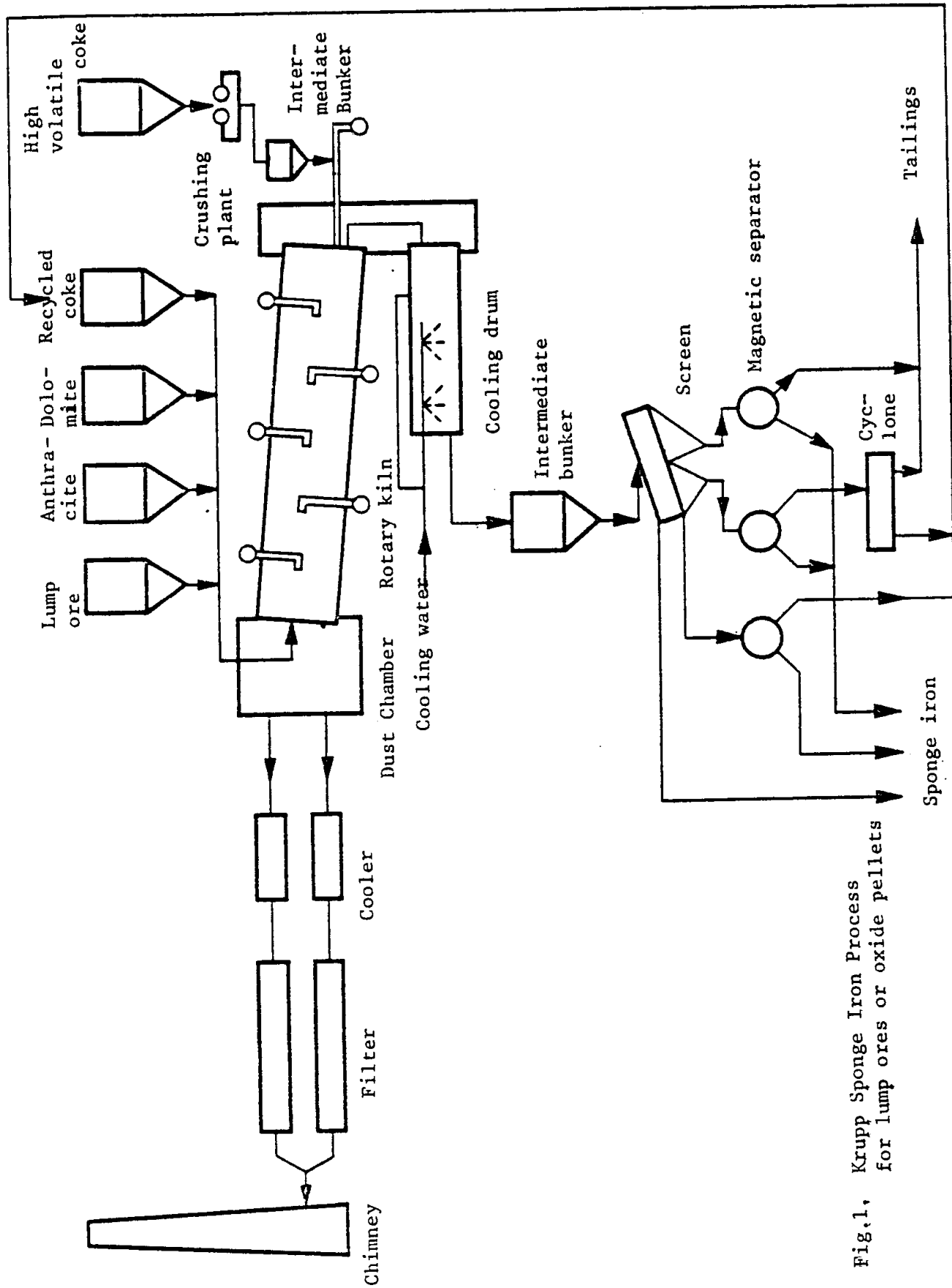


Fig.1, Krupp Sponge Iron Process for lump ores or oxide pellets

MIDREX PROCESS

1. Characteristics

1.1. Name and history of process

Midrex direct reduction process

1965 pilot plant with 1.5 t.p.d. in Toledo

1969 first large scale plant, 2 x 200,000 t.p.a. in Portland,
Oregon

1971 second large scale plant with 1 x 400,000 t.p.a. in
Georgetown, South Carolina

1972 third plant with 1 x 400,000 t.p.a. in Hamburg

1973 fourth plant with 1 x 400,000 t.p.a. in Contrecoeur,
Canada

1.2. Principle of process

Reduction of iron ore in a shaft furnace by reducing gases
in accordance with the counterflow principle.

1.3. Inventors

D. Beggs, W.T. Marston

1.4. Companies involved in development

Midland-Ross Corporation, Toledo, Ohio

Korf-Stahl AG, Baden-Baden, with subsidiaries

2. General description (schematic drawing)

See Figure 1

3. Plant required

Shaft furnace, reformer, recuperator, various compressors for circulating the gas, scrubbers and dust-collecting appliances.

4. State of development

4.1. Name of laboratories where the process is situated

No details

4.2. Pilot plant

4.2.1. Location

Toledo

4.2.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

No details

4.2.3. Capacity in tonnes per 24 h of final product

1.5 t.p.d.

4.3. Industrial installations

4.3.1. Location

- a) Portland, Oregon, USA
- b) Georgetown, S.C., USA
- c) Hamburg, Fed. Rep. of Germany
- d) Contrecoeur, Quebec, Canada

4.3.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

No details

4.3.3. Capacity in tonnes per 24 h of final product - per unit of production

Nominal capacity:

- a) 2 x 200,000 t.p.a.
- b) 400,000 t.p.a.
- c) 400,000 t.p.a.
- d) 400,000 t.p.a.

4.3.4. Method of operation (continuous or batch, number of operating hours per day)

Continuous

4.3.5. Coefficient of utilisation in time (number of working days per annum)

No details

4.3.6. Maximum rated production

No details

5. Reducing agents

5.1. Raw materials on which the reducing agent is based

At the moment, natural gas but the following are also possible:
liquid gas, naphtha, coke oven gas, coal, heavy oil, off-gas
from low-shaft electric furnace

5.2. Process for conversion of above raw materials into a form for use in reduction

The natural gas is catalytically reformed with the recycled top gas from the shaft furnace

5.3. Condition and analysis of reducing agent at moment of use

Dependent on raw materials and energy.

Example using Groningen natural gas

CO ₂	1.0 - 5.0 %
CO	25 - 40 %
H ₂	35 - 55 %
CH ₄	3 - 4 %
N ₂	1 - 10 %

These gas analyses include all Midrex plants previously in operation.

5.4. Preheat temperature on entering reduction apparatus

760 to 900°C

5.5. Gas velocity in reduction apparatus

No details

5.6. Analysis and temperature of gas after passing through reduction apparatus

Dependent on raw materials and operating conditions.

Example using Groningen natural gas

CO ₂	12 - 20 %
CO	12 - 20 %
H ₂	20 - 40 %
CH ₄	1 - 5 %
N ₂	- - 10 %

Temperature: 280 - 400°C

6. Ores and ferruginous materials

6.1. Source

Example of the ores used at Contrecoeur

6.2. Analysis

See Table 1

6.3. Size

See Table 1

6.4. Preparation

Screening of fines < 5 mm to 2% max.

7. Additions

None

8. Temperature zones in reduction
in °C in the reduction apparatus

Dependent on type of ore

9. Pressure zones in reduction
in kg/cm² in the reduction apparatus

No details

10. Product
of the reduction apparatus

10.1. Form

Pellets or lump according to 6.1.

10.2. Analysis

For examples see Table 2

10.3. Degree of reduction

For examples see Table 2

10.4. Utilisation of by-products

Gas is recycled to the reformer for heating and cracking

11. Further treatment of the product (10) of the reduction apparatus

11.1. Method of cooling

Cooling occurs in the lower part of the shaft through a sealed circuit

11.2. Crushing, compacting or briquetting

None

11.3. Details of final size grading

None

11.4. Method of concentration (e.g. magnetic separation or sintering)

None

12. Final product obtained after treatment according to 11

See 10

13. Iron Yield

About 98%, depending on type of ore

14. Utilisation of final product

Destination of final product (blast furnace, electric or other furnace). Technical and economic differences as compared with using pig iron and scrap

Can be used in all metallurgical processes for the production of pig iron and steel. In blast furnaces, cupola furnaces, induction furnaces, electric reduction furnaces, OBM, electric arc furnaces.

Mainly used in electric arc furnaces. Technical and economic differences compared with using scrap:

- higher productivity
- better quality (low in tramp elements)
- less harm to the environment
- better control of the process

15. Consumption data per tonne of final product (12)

15.1. Raw materials for reduction (5.1)

No details

15.2. Fuels and thermal values required for the entire process (fuel characteristics) (thermal balance)

Mean energy consumption at the Hamburg steelworks 3.1 Gcal/t Fe using natural gas. For the plants under construction, which make use of heat recovery, an energy consumption of 2.9 Gcal/t Fe has been calculated.

15.3. Electrical energy

120-130 kWh/t (without using the off-gas from the reformer)

15.4. Steam used

None

15.5. Oxygen

None

15.6. Cooling water

1.5 m³ per tonne

15.7. Refractories

Not known

15.8. Man hours of productive labour

0.3 h per tonne

15.9. Labour and materials for routine maintenance

approximately 3.50 DM per tonne + 0.18^j h per tonne

16. Overall capital investment

16.1. For the plant described in (3) and related to one tonne of capacity for the final product (12)

Capacity expressed in tonnes per day or per year

In battery limits, based on conditions in Fed. Republic of Germany approximately:

160 - 180 DM/annual tonne

16.2. Construction costs of ancillary plant not covered in (3)

No details

17. Opinions expressed on the process

By altering the flow diagram (using top gas for cooling) it would be possible to use ores with a higher sulphur content too, without adverse effects on the reformer.

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Table 1. Chemical and physical properties of various charge materials

Chemical Analysis	Unit	LKAB Pellets	CVRD Pellets	Canadian Pellets Hilton	Canadian Pellets Carol Lake	Brazilian Lump Ore
Fe		68.13	68.12	67.08	65.26	68.46
SiO ₂		1.53	1.01	2.18	5.73	0.70
Al ₂ O ₃		0.74	0.91	0.25	0.33	1.18
CaO	%	0.12	0.59	0.22	0.34	-
MgO		0.30	0.11	1.40	0.26	-
P		0.009	0.027	0.005	0.012	0.042
S		0.002	0.009	0.006	0.002	0.005
Screen Analysis	mm	%	%	%	%	%
	+ 19	4.7	10.0	0.4	4.5	9.8
	+ 16	5.7	12.2	0.5	6.1	12.4
	+ 13	35.6	40.8	14.6	34.3	41.2
	+ 9	39.3	15.5	69.8	45.7	15.1
	+ 6	7.6	14.3	12.52	5.2	14.2
	+ 3	2.2	2.1	0.9	1.0	2.0
	- 3	5.2	3.2	1.0	3.2	3.5
Combustion process		Shaft furnace Magnetite	Travelling grate Hematite	Shaft furnace Magnetite	Travelling grate Hematite	
Raw material						

Table 2. Chemical analysis of various types of sponge iron

Chemical Analysis	CVRD- Pellets	LKAB Pellets	Canadian Pellets	
			Hilton	Carol Lake
Fe _{total}	92.46	93.83	90.03	88.24
Fe _{metallic}	86.12	87.47	84.18	82.12
Metallisation	93.0	93.10	93.50	93.00
SiO ₂	1.71	1.16	2.41	7.63
Al ₂ O ₃	0.91	0.88	0.83	0.44
CaO	0.65	0.40	0.48	0.38
MgO	0.28	0.25	1.78	0.29
P	0.017	0.010	0.001	0.008
S	0.003	0.008	0.005	0.005
Cu	traces	traces	traces	traces

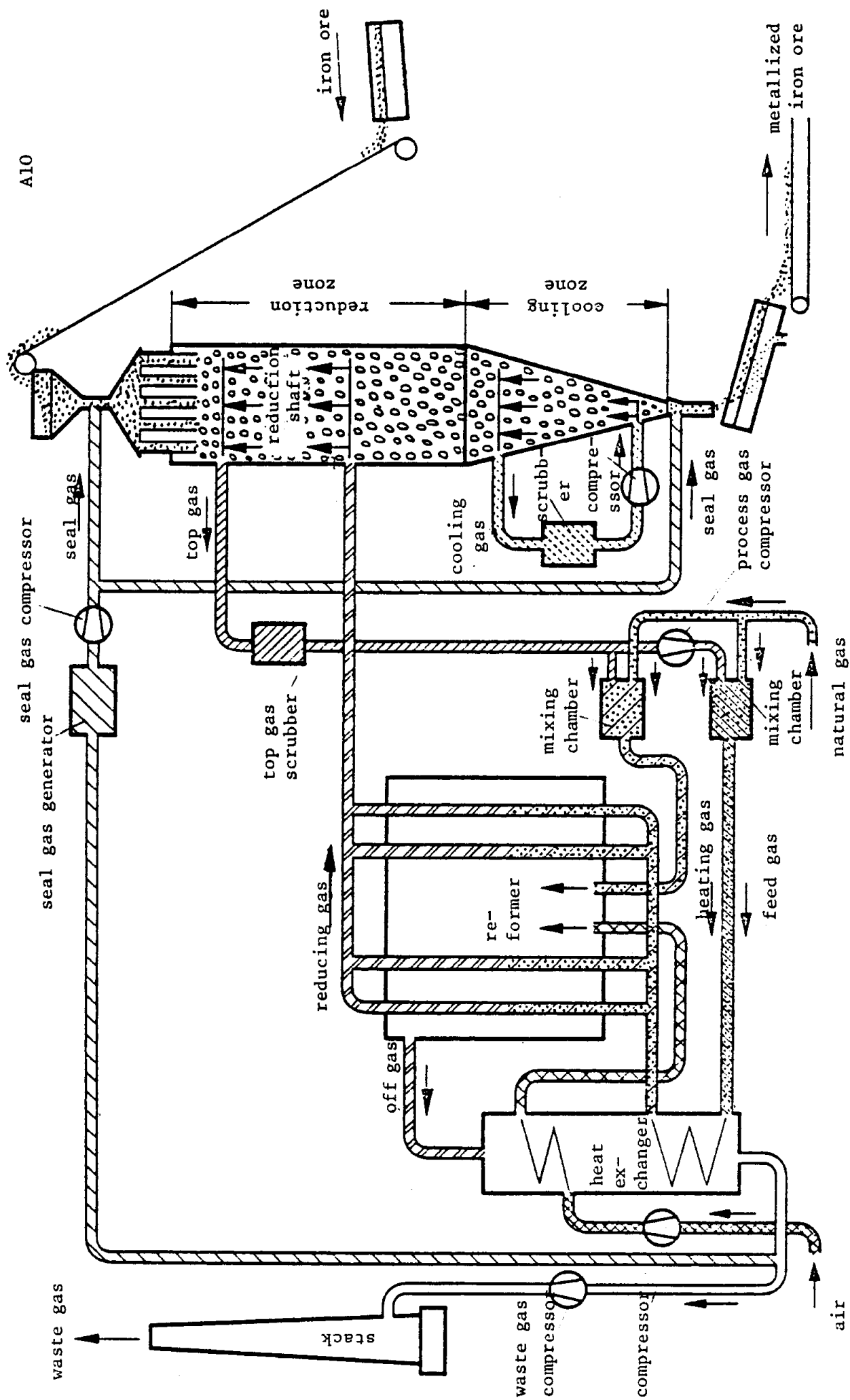


Fig. 1. Flow Sheet of the Midrex Process.

NSC PROCESS

1. Characteristics

1.1. Name and history of process

NSC process (Nippon Steel Corporation)

In 1971 Nippon Steel began with a fundamental investigation of the processes for producing sponge iron. It was decided to develop their own process and a small pilot plant was built at the Hirohata works. Since it appeared feasible to use the process industrially to produce sponge iron, in the next stage a larger experimental plant was built (commissioned at the end of 1976).

1.2. Principle of process

Shaft furnace with gaseous reducing agent

1.3. Inventors

Nippon Steel Corporation, Tokyo (Japan)

1.4. Companies involved in development

Nippon Steel Corporation, Tokyo (Japan)

2. General Description

The latest knowledge from blast furnace technology was used in the development of the NSC process. The reduction unit consists of two parts, a shaft of special shape and a cooling chamber. After reduction of lump ore or pellets in the shaft the sponge iron is cooled to temperatures below 100°C in the cooling chamber.

The reducing gas is produced by partial oxidation of oil, for instance by the Texaco process. A gas with a high reduction potential is obtained from carbon monoxide and hydrogen.

The NSC process has the further feature that - in common with modern blast furnaces - counter-pressure at the throat is used. For this reason the shaft is equipped with a two-bell arrangement. Furthermore, the process is suitable for high reduction temperatures. The gas circuit is closed in this process, so that no problems arise regarding stress to the environment. The MEA process is used to absorb CO_2 in the gas after the reduction.

3. Plant required

Reduction shaft with double bell arrangement at the throat, cooling chamber, oil gasification plant.

4. State of development

4.1. Name of laboratories where the process is situated

Hirohata Works, Nippon Steel Corporation

4.2. Pilot plant

4.2.1. Location

Fuji-machi, Hirohata-ku, Himeji-City, Hyogo Prefecture
(Japan)

4.2.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

Dimensions of the shaft furnace

Internal volume	0.7 m ³
Diameter	0.65 m (at tuyere level) 0.33 m (at the throat)
Height	3.6 m
Number of tuyeres	3
Removal of sponge iron	with rotary table

4.2.3. Capacity in tonnes per 24 hr of final product

8 t sponge iron per 24 hr

4.3. Industrial installations

4.3.1. Location

Fuji-machi, Hirohata-ku, Himeji-City, Hyogo prefecture
(Japan)

4.3.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

Not known

4.3.3. Capacity in tonnes per 24 hr of final product - per unit of production

500 t sponge iron per 24 hr

5. Reducing agents

5.1. Raw materials on which the reducing agent is based

Oil, which is gasified by the Texaco process

In the NSC process, however, natural gas or reduction gas from the gasification of coal can also in principle be used.

- 5.2. Process for conversion of above raw materials into a form for use in reduction

In the Texaco process fuel oil is partially oxidised under pressure at high temperatures, using oxygen, to carbon monoxide and hydrogen.

- 5.3. Condition and analysis of reducing agent at moment of use

Not known

- 5.4. Preheat temperature on entering reduction apparatus

Not known

- 5.5. Gas velocity in reduction apparatus

Not known

- 5.6. Analysis and temperature of gas after passing through reduction apparatus

Not known

6. Ores and ferruginous materials

Iron ore (lump ore or pellets)

7. Additions

Not known

8. Temperature zones in reduction
in °C in the reduction apparatus

The shaft is designed for high reduction temperatures.

9. Pressure zones in reduction
in kg/cm² in the reduction apparatus

The process takes place under excess pressure (pressure range as for the blast furnace process with back pressure).

10. Product
of the reduction apparatus

Sponge iron in the form of the material charged

11. Further treatment of the product (10) of the reduction apparatus

No details

12. Final product obtained after treatment according to 11

No details

13. Iron yield

No details

14. Utilisation of final product

For steelmaking

15. Consumption data per tonne of final product (12)

No details

16. Overall capital investment

The costs for the pilot plant with a capacity of 500 t/24 hr run to about 5 billion Yen (approximately £10,460,000)

17. Opinions expressed on the process

The pilot plant will be operated from October 1976 to March 1978.

The shaft furnace principle is assumed to have the best chance for direct reduction. In comparison with the blast furnace the furnace installations and capacity are much smaller, but the costs of building the shaft furnace for direct reduction are lower and the process can be used in mini-steelworks. Moreover, it is particularly suitable for areas where reducing gas is available. The development of the process is also helped by the difficulties to be expected in the long term as regards supplies of coking coal and by the scarcity of other raw materials.

18. Bibliography

None

PUROFER PROCESS

1. Characteristics

1.1. Name and history of process

PUROFER process, development work carried out since 1958

1.2. Principle of process

Countercurrent reduction of ores in a shaft furnace with gases obtained by reforming hydrocarbons with oxidising gases.

1.3. Inventors

A number of people from the companies listed under 1.4. have taken part in developing the process to its present state.

1.4. Companies involved in development

The work in Oberhausen was carried out initially at the Hüttenwerk Oberhausen AG. This company later became part of Thyssen Niederrhein. Since 1972 all PUROFER activities have been grouped together under Thyssen PUROFER GmbH.

2. General description (schematic drawings)

The shaft furnace is charged with lump ore or pellets. The reducing gas is produced by reforming natural gas, coke oven gas or the like with recycled top gas in regenerative gas reformers and is fed directly into the shaft furnace (Figure 1). If fuel oil or coal is used the cold gas produced in a gasification plant is heated up in a recuperator (Figure 2).

3. Plant required

Shaft furnace, regenerative gas reformers or gasification plant and recuperator, briquetting press.

4. State of development

4.1. Name of the laboratories where the process is situated

Thyssen PUROFER GmbH, Oberhausen.

4.2. Pilot plant

Now a large-scale experimental plant producing at least 300 tonnes of sponge iron a day.

4.2.1. Location

Oberhausen, Rheinland, Federal Republic of Germany

4.2.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

Designed as an experimental plant fitted out for using natural gas and coke-oven gas as energy sources. Hydrocarbons reformed with either recycled top gas or air. Dimensions of shaft furnace suitable for high gas throughput with reducing gas containing nitrogen from reforming with air.

Designed at the ore end for throughput of fired pellets or lump ore.

Connected direct to a pelletising plant thus enabling continuous charging of unfired (green) pellets.

4.2.3. Capacity in tonnes per 24 h of final product

Depending on processed ore and energy sources 300 to 400 per day.

4.3. Industrial installations

4.3.1. Location

- a) Santa Cruz, GB, Brazil (Cosigua);
- b) Ahwaz, Iran (NISIC).

4.3.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

Design of shaft furnace identical in the two cases and suitable for using either fired pellets and/or lump ore, height of burden column 13 m (see 4.3.3.).

Ahwaz, Iran:	Natural gas reformed in two regenerative PUROFER gas reformers of 6 m diameter.
Santa Cruz, Brazil:	Gasification of heavy oil by means of oxygen and steam to form a CO/H_2 mixture. Addition of recycled shaft furnace top gas from which the CO_2 has been scrubbed. Heating of mixture in recuperators which are heated with shaft furnace top gas. Output of heavy oil gasification plant: $36,000 \text{ Nm}^3 (\text{H}_2 + \text{CO})/\text{h}$.

4.3.3. Capacity in tonnes per 24 h of final product - per unit of production

1000 t Fe, corresponding to approximately 1100 t sponge iron.

4.3.4. Method of operation (continuous or batch, number of operating hours per day)

Continuous

4.3.5. Coefficient of utilisation in time (number of working days per annum)

330 d.p.a.

4.3.6. Maximum rated production

400,000 tonnes per annum sponge iron

5. Reducing agents

5.1. Raw materials on which the reducing agent is based

Gaseous and liquid hydrocarbons, coal.

5.2. Process for conversion of above raw materials into a form for use in reduction

Gaseous hydrocarbons reformed to CO and H₂ with recycled top gas or gasification of liquid hydrocarbons or coal using conventional methods, heating cold gases.

5.3. Condition and analysis of reducing agent at moment of use

Reducing and carburising potential of the gas are adjusted to meet the metallurgical requirements. Main influencing parameter is the permissible reduction temperature predetermined by the ore used. Figure 3 shows the course of the Boudouard curve in this system, taking as an example a ratio of H₂/C = 1 in the reducing gas. The content of oxidising constituents in the reducing gas is adjusted in accordance with the trend of the Boudouard curve, i.e. a fairly low degree of oxidation at high reduction temperatures and a fairly high one at low reduction temperatures.

For a degree of oxidation of about 5% a typical composition of reducing gas is obtained using various types of fuel as per the Table below.

Type of Fuel used	Natural Gas	Coke Oven Gas	Heavy Oil
Gas Composition	Vol %	Vol %	Vol %
CO	44.7	31.6	44.3
H ₂	46.7	58.9	49.3
CO ₂	2.0	2.0	3.6
H ₂ O	3.1	3.0	1.2
CH ₄	1.2	1.2	0.5
N ₂	2.3	3.3	1.1

5.4. Preheat temperature on entering reduction apparatus

Maximum possible temperature, determined purely by the fusion behaviour of the ore in question, can be set between

max. approximately 1100°C

min. approximately 700°C

5.5. Gas velocity in reduction apparatus

Velocity in empty shaft about 1.5 m/s under normal conditions, relative to mean cross-section of shaft furnace.

5.6. Analysis and temperature of gas after passing through reduction apparatus

Vol %	Natural Gas	Coke Oven Gas	Heavy Oil
CO	27.3	19.4	26.3
H ₂	30.5	37.9	33.6
CO ₂	18.6	13.2	21.1
H ₂ O	20.0	25.0	17.3
CH ₄	1.2	1.2	0.5
N ₂	2.4	3.3	1.2

The temperature of the top gas is between 300 and 500°C depending on the ore used.

6. Ores and ferruginous materials

6.1. Source

Pellets and hematite lump ores

6.2. Analysis

No particular limitations. For examples of ores processed see Table 1

6.3. Size

Screening at 6.3 mm before charging into shaft furnace. For size ranges used in the pilot plant at Oberhausen see Table 2.

7. Additions

None

8. Temperature zones in reduction
in °C in the reduction apparatus

Depends on the fusion or sticking temperature of the ores,
otherwise no restrictions.

9. Pressure zones in reduction
in kg/cm² in the reduction apparatus

Approximately 2.5 bar at entry to shaft furnace.

Approximately 1.4 bar at furnace throat.

It is possible to use higher pressures, even a higher back
pressure at the furnace throat because mechanical locks provide
a seal against the outside at the top and discharge end of the
shaft furnace.

10. Product
of the reduction apparatus

10.1. Form

As with the ore charged, partial disintegration of the ores
during reduction is possible and acceptable. Examples for
analyses of size distribution corresponding to the ore
analyses listed under 6.3 are given in Table 3.

10.2. Analysis

For examples corresponding to the ore analyses listed under
6.2., see Table 4.

10.3. Degree of reduction

10.3.1. Proportion of oxygen removed from iron oxides as a percentage of the total oxygen combined with iron in the ores

At least 92%

10.3.2. Metallic iron as a percentage of the total iron content of the product

At least 90%

10.4. Utilisation of by-products

None

11. Further treatment of the product (10) of the reduction apparatus

11.1. Method of cooling

No cooling of the product is effected in the reduction vessel. Preferably hot charging in the further processing, e.g. in the steelworks, alternatively hot briquetting with subsequent cooling, where appropriate, in a water bath.

No other treatment of the product in the reduction vessel, sponge iron is discharged hot.

11.2. Crushing, compacting or briquetting

Hot briquetting: production of pillow-shaped briquettes from the sponge iron discharged hot from the reduction furnace.

11.3. Details of final size grading

With hot charging see examples of particle size distribution in Table 3.

11.4. Method of concentration (e.g. magnetic separation or sintering)

None

12. Final product obtained after treatment according to (11)

12.1. Form

As under 11.3.

12.2. Analysis

As under 10.2.

12.3. Reoxidation insignificant with briquettes

13. Iron yield

13.1. Iron contained in product (10) as a percentage of iron in the charge

Dependent on amount of flue dust accumulated which is normally about 20 kg Fe/t sponge iron. No other losses

13.2. Iron contained in final product (12) as a percentage of iron in the charge

See 13.1.

14. Utilisation of final product

Destination of final product (blast furnace, electric or other furnace). Technical and economic differences as compared with using pig iron and scrap

Can be used in all steel and pig iron producing plants and in foundries.

Better quality than scrap; economics depend on location.

Cheaper than scrap if ore and natural gas can be supplied at favourable prices.

15. Consumption data per tonne of final product (12)

All consumption figures are quoted per tonne of iron in the sponge iron since this is the only way of making a correct comparison. To determine the consumption figures per tonne of sponge iron the values given here must be multiplied by the iron contents Fe_{tot} listed in Table 4.

15.1. Raw materials for reduction (5.1)

Dependent on calorific value, see 15.2.

15.2. Fuels and thermal values required for the entire process (fuel characteristics) (thermal balance)

Net calorific value of gases given off approximately

$1,700 \frac{\text{kcal}}{\text{Nm}^3} \text{ top gas}$

	Natural Gas	Coke Oven Gas	Heavy Oil
Energy used Gcal/t Fe	3.3	3.1	3.5

The hot gas leaving the shaft furnace has a net calorific value of about $1,700 \text{ kcal/Nm}^3$ before scrubbing and cooling

15.3. Electrical energy

15.3.1. For heating

None

15.3.2. For power

	Natural Gas	Heavy Oil
kWh/t Fe	130	360 ⁺
Briquetting kWh/t Fe	15	15

⁺ including requirement for oil gasification and production of oxygen

15.4. Steam used

No steam is needed if natural gas and coke oven gas are used. If heavy oil is used steam is required in the gasification plant but this is produced completely within the process. The energy requirement for producing the steam is included in the total energy consumption of 3.5 Gcal/t Fe. 1.4 m³/t Fe boiler feed water is used.

15.5. Oxygen

Natural gas/coke oven gas: -
Heavy oil: 260 Nm³/t Fe

15.6. Cooling water

Natural gas/coke oven gas: 1.0 Nm³/t Fe
Heavy oil: 3.2 Nm³/t Fe

15.7. Refractories

Natural gas/coke oven gas: 3.-- DM/t Fe
Heavy oil: included in 15.9.

15.8. Man hours of productive labour

Natural gas/coke oven gas: 120 man-hours per day
Heavy oil: 168 man-hours per day

15.9. Labour and materials for routine maintenance

Natural gas/coke oven gas: 72 h per day, 8.-- DM/t Fe
Heavy oil: 72 h per day, 11.40 DM/t Fe

16. Overall capital investment

No details

17. Opinions expressed on the process in relation to:

17.1. State of development

2 plants, each with a capacity of 1000 t.p.d. based on natural gas/pellets and heavy fuel/lump ore respectively are in the course of construction and will start operating in 1976.

1 pilot plant has been in operation since 1970 at Oberhausen. Development with regard to large-scale application of the process has been completed. Further development is being carried out on processing green pellets (hardening with simultaneous reduction in the shaft furnace).

17.2. Practical operating difficulties

None

17.3. Possibilities of exploitation

Suitable for both lump ore and pellet charges.

100% lump ore without the addition of pellets can also be processed.
Almost all hydrocarbons and coal suitable as energy sources.

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18.2. Patents

German Patents

<u>No.</u>	<u>Filing date</u>
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PS ^{**) 1 178 869}	27.11. 1959
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P 2 514 703	4. 4. 1975

^{*)} = disclosure patent

^{**) patent;}

^{*** disclosure document}

^{**** patent application}

Table 1. Chemical analysis of ores processed in the pilot plant in Oberhausen

P e l l e t s

Ore %	Carol Lake	Itabira	MPS	MPB 1	Svappa-vaara	MPB 111	MPR	Kiruna	Marcona
Fe _{total}	65.1	66.3	67.9	67.6	66.7	66.4	68.0	65.9	66.5
Fe ₂ O ₃	92.5	92.4	96.0	96.2	93.4	94.2	96.3	92.4	92.6
SiO ₂	5.61	1.5	0.95	1.30	1.18	1.50	1.25	3.85	2.25
CaO	0.40	1.37	0.10	0.35	1.25	0.25	0.18	0.20	0.40
MgO	0.43	0.18	0.30	0.36	0.35	0.30	0.27	0.22	0.92
Al ₂ O ₃	0.35	0.79	0.60	0.75	0.87	0.92	0.78	0.76	0.50
TiO ₂									
P	0.014	0.024	0.01	0.01	0.015	0.019	0.022	0.03	0.018
S	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01

L u m p O r e s

Ore %	Brazil Natural Ore	Hamersley	Feijao-1/4-1"	Feijao 1/2-1 1/2"	Mutuca 1/4-1 1/4"	Mutuca 1/2-1 1/2"
Fe _{total}	66.3	56.23	68.25	67.9	67.6	68.10
Fe ₂ O ₃	91.8	93.06	92.80	92.9	94.6	94.10
SiO ₂	1.18	3.08	0.71	0.85	0.76	0.72
CaO	0.10	0.09	0.10	0.10	0.12	0.15
MgO	0.05	0.03	0.10	0.10	0.10	0.10
Al ₂ O ₃ ⁺	1.62	1.85	0.72	0.76	1.15	0.88
TiO ₂						
P	0.041	0.037	0.08	0.075	0.05	0.058
S	0.020	0.01	0.01	0.01	0.01	0.01

Table 2. Size analysis of ores processed in the pilot plant in Oberhausen

P e l l e t s

Ore	Carol Lake	Itabira	MPS	MPB 1	MPB 111	MPR	Kiruna	Marcona
- 1 mm	0.2	0.1			0.05	0.1		0.3
- 5 mm	0.7	0.5	0.5	0.2	0.14	0.8	0.75	1.3
- 6.3 mm	5.4	1.4	1.1	0.3		1.4	1.60	2.2
- 8 mm	55.0	3.9	1.5	0.6	0.70	2.5	3.10	3.9
- 12.5mm	84.0	58.8	6.1	18.5	39.40	34.3	46.60	70.5
- 16 mm	92.8	93.0	11.3	67.0	87.10	86.9	97.60	98.8
- 20 mm	96.9	98.6	21.4	91.6	94.50	94.5	99.8	100.0
- 25 mm	99.1	100.0	66.0	98.2	98.60	97.3	100.0	
- 31.5mm	100.0		94.5	100.0	100.0	100.0		
+ 31.5mm			5.5					

L u m p. O r e s

Ore	Brazil Natural Ore	Hamersley	Mutuca 1/4-1 1/4"	Mutuca 1/2-1 1/2"	Feijao 1/4-1"	Feijao 1/2-1 1/2"
- 1 mm	0.1		0.3	0.3	0.35	0.3
- 5 mm	0.7	0.7	2.2	1.1	2.32	1.6
- 6.3 mm	1.0	4.6	3.1	1.9	4.2	2.6
- 8 mm	2.6	16.0	5.5	2.7	7.55	4.4
- 12.5mm	70.5	43.7	26.0	7.7	23.70	13.2
- 16 mm	98.8	61.3	40.3	14.5	40.5	22.8
- 20 mm	100.0	75.6	55.2	28.1	66.8	34.7
- 25 mm		90.2	77.0	49.9	92.7	52.5
- 31.5mm		99.1			100.0	
- 40 mm		100.0	100.0	97.6		100.0

Table 3. Size analysis of sponge iron from ores processed in the pilot plant in Oberhausen

P e l l e t s

Sponge Iron	Carol Lake	Itabira	MPS	MPB 1	MPB 111	MPR	Kiruna	Marcona
- 1 mm	0.8	0.8	0.3	0.2	0.25	2.1	2.1	0.3
- 5 mm	6.5	1.8	4.5	0.6	1.05	6.4	7.7	1.3
- 6.3 mm	52.1	2.8	6.9	1.2		8.0	11.6	2.2
- 8 mm	87.8	5.6	11.1	11.4	2.95	11.5	15.9	3.9
- 12 mm	97.0	62.7	32.5	48.5	62.5	67.7	77.8	70.5
- 16 mm	98.8	93.9	52.0	91.0	97.0	98.2	99.2	98.8
- 20 mm	99.3	99.2	69.9	100.0	98.9	99.4	99.8	100.0
- 25 mm	100.0	100.0	86.5		100.0	100.0	100.0	
- 31.5 mm			98.9					
+ 31.5 mm			1.1					

L u m p O r e s

Sponge Iron	Brazil Natural Ore	Hamersley	Mutuca 1/4-1 1/4"	Mutuca 1/2-1 1/2"	Feijao 1/4-1"	Feijao 1/2-1 1/2"
- 1 mm	8.5	4.8	6.2	4.3	4.5	4.0
- 5 mm	27.0	24.6	21.3	16.9	16.0	14.1
- 6.3 mm	33.4	33.0	26.5	21.2	21.5	18.1
- 8 mm	43.3	47.3	35.2	27.5	29.6	24.0
- 12.5 mm	79.2	74.7	64.3	48.5	57.7	50.5
- 16 mm	88.0	84.4	76.4	61.2	75.7	66.6
- 20 mm	93.6	92.9	85.2	73.6	91.0	78.9
- 25 mm	96.8	98.7	93.5	86.5	99.0	90.9
- 31.5 mm	99.5	100.0			100.0	
- 40 mm	100.0		100.0	100.0		100.0

Table 4. Chemical analysis of sponge iron from ores processed in the pilot plant in Oberhausen

P e l l e t s

Sponge Iron %	Carol Lake	Itabira	NPS	HPB 1	Svappa-vaara	MPB 111	MPR	Kiruna	Marcona
Fe _{total}	89.0	92.7	94.8	93.7	93.1	89.4	92.4	90.53	90.0
Fe _{met}	85.3	87.5	90.4	89.7	88.2	83.7	85.7	85.61	82.8
SiO ₂	7.6	2.08	1.35	1.71	1.57	3.29	1.75	5.10	3.05
CaO	0.5	1.65	0.15	0.30	1.65	0.45	0.25	0.35	0.60
MgO	0.58	0.18	0.36	0.35	0.25	0.68	0.37	0.35	1.28
Al ₂ O ₃ ⁺									
TiO ₂	0.46	1.04	0.76	0.94	0.93	1.38	1.10	1.04	0.81
P	0.017	0.031	0.016	0.012	0.025	0.026	0.034	0.035	0.22
S	0.004	0.003	0.004	0.004	0.003	0.005	0.006	0.006	0.007
C	0.35	0.70	0.66	0.68	0.75	0.87	1.69	1.08	1.90
Degree of reduction	96.7	95.6	96.3	96.7	95.9	95.03	94.3	95.76	93.8

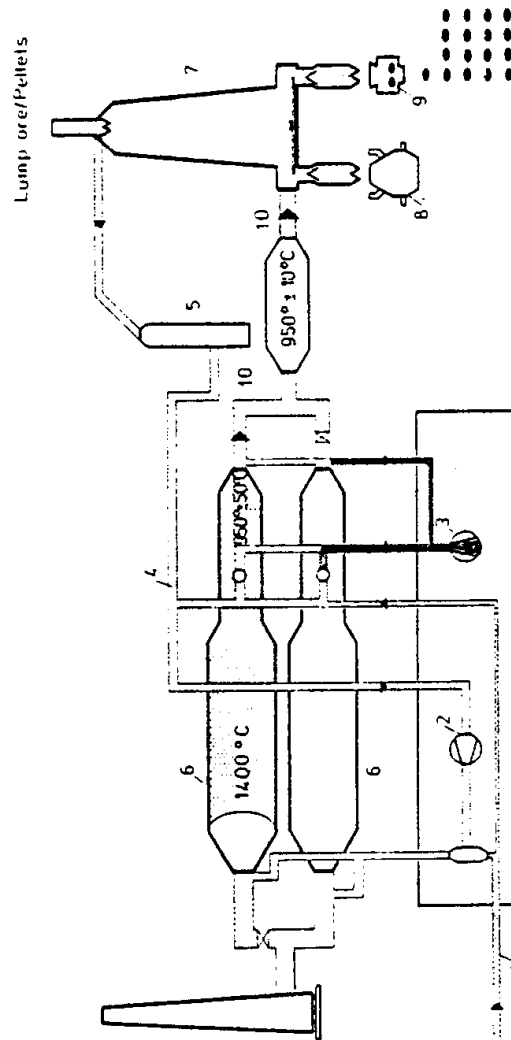
L u a p o r e s

Sponge Iron %	Brazil Natural Ore	Hammersley	Feijao 1/4-1"	Feijao 1/2-1 1/2"	Mutuca 1/4-1 1/4"	Mutuca 1/2-1 1/2"
Fe _{total}	91.0	89.93	93.0	92.9	93.60	94.30
Fe _{met}	80.4	94.68	84.3	85.9	87.70	88.60
SiO ₂	1.66	4.25	0.92	1.10	1.06	0.95
CaO	0.17	0.12	0.12	0.15	0.20	0.20
MgO	0.08	0.05	0.10	0.10	0.17	0.15

Table 4 (Continued)

Sponge Iron %	Brazil Natural Ore	Hammersley	Feijao 1/4-1"	Feijao 1/2-1 1/2"	Mutuca 1/4-1 1/4"	Mutuca 1/2-1 1/2"
Al ₂ O ₃ + TiO ₂	2.37	2.54	0.87	1.05	1.58	1.20
P	0.062	0.051	0.08	0.08	0.052	0.07
S	0.004	0.006	0.005	0.008	0.008	0.008
C	0.46	1.35	1.90	1.75	1.15	1.10
Degree of reduction	91.00	95.45	92.70	94.12	95.10	95.30

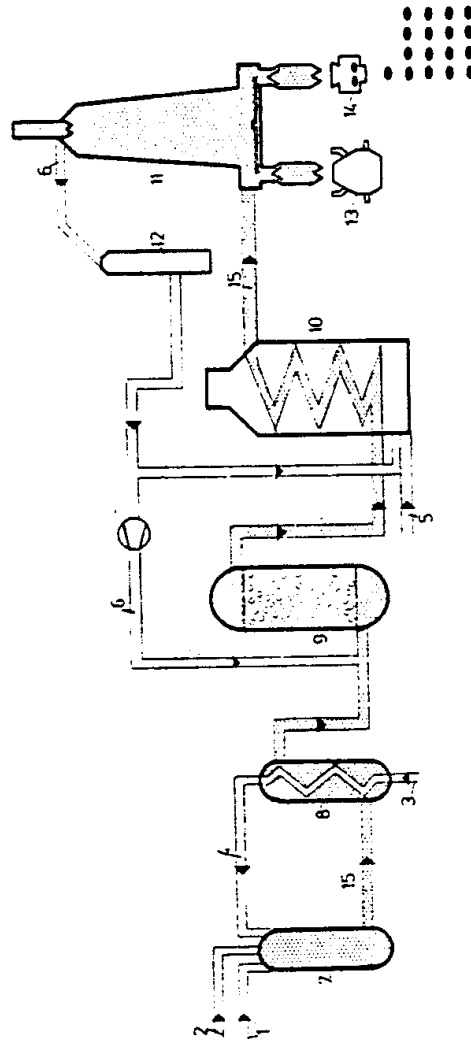
A12



1	Natural Gas up to 30 ppm H_2S	6	Gas Reformers
2	Recycling Compressor	7	Shaft Furnace
3	Combustion Air Fan	8	Container for Hot PUROFER Iron
4	Top Gas	9	Hot Briquetting
5	Scrubber	10	Reducing Gas

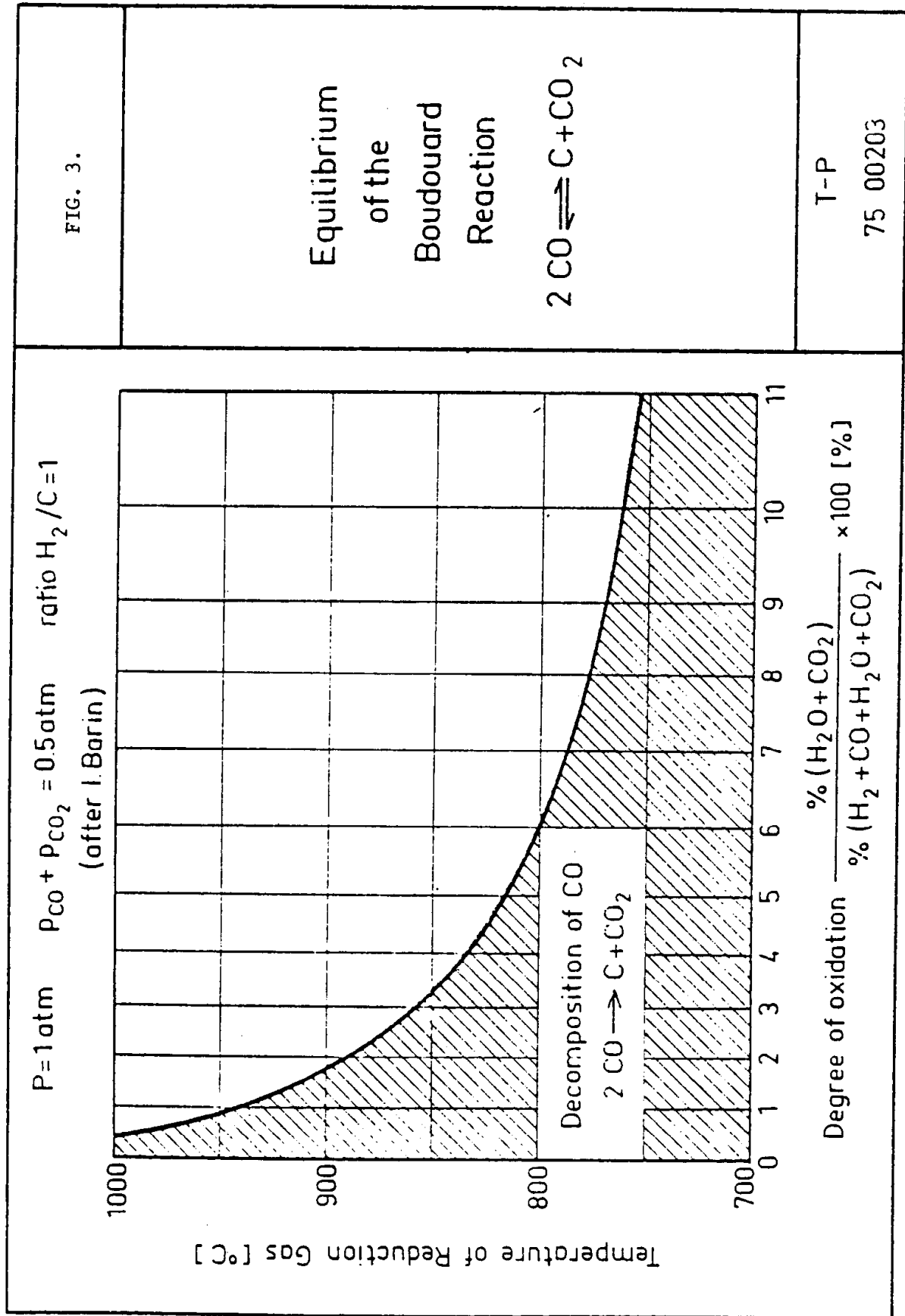
Fig. 1. PUROFER-Process Based on Natural Gas or Coke Oven Gas

A12



1 Heavy Fuel Oil	6 Top Gas	11 Shaft Furnace
2 Oxygen	7 Gasification	12 Scrubber
3 Water	8 Waste Heat Boiler	13 Container for Hot PUROFER Iron
4 Steam	9 CO ₂ and H ₂ S Removal	14 Hot Briquetting
5 Combustion Air	10 Refeathering	15 Reducing Gas

Fig. 2. PUROFER-Process Based on Heavy Fuel Oil



SDR PROCESS

1. Characteristics

1.1. Name and history of process

Sumitomo Dust Reduction Process (SDR)

In 1971 it was decided to develop a new direct reduction process to make use of oxide waste materials from iron and steel works. From October 1971 to March 1972 tests were carried out by Sumitomo Metal Industries and Sumitomo Heavy Industries, the problems of transporting, filtering and drying the slurry, together with pelletising and preheating, being investigated by Sumitomo Heavy Industries and reduction in the rotary kiln by Sumitomo Metal Industries. During 1972 and 1973 Sumitomo Metal Industries carried out a feasibility study and in 1973 work was started on the construction of a plant for processing 20,000 t waste dusts per month at the Wakayama works. The plant was commissioned in May 1975.

1.2. Principle of the process

Agglomerated iron oxide is reduced in a rotary kiln by means of coke fines from the blast furnace dust and bonded together in pellets. In the final stage of the reduction zinc oxide is reduced and vapourised. The vapourised zinc reoxidises in the atmosphere and is collected by bag filters. The carbon monoxide produced is burnt off with air fed in by a pipe in the shell of the rotary kiln. The rotary kiln is heated by a burner at the end.

1.3. Inventors

Not known. The process contains a combination of various ideas.

1.4. Companies involved in development

Sumitomo Metal Industries Ltd. and
Sumitomo Heavy Industries Ltd.

2. General description (schematic drawing)

(See Figure 1)

3. Plant required

(See Figure 1)

4. State of development

4.1. Name of laboratories where process is situated

- a) Niihama Research Laboratories, Sumitomo Heavy Industries Ltd.
- b) Central Research Laboratories, Sumitomo Metal Industries Ltd.

4.2. Pilot plant

4.2.1. Location

Niihama Research Laboratories,
Sumitomo Heavy Industries Ltd.

4.2.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

- a) Disc pelletisers, 1.2 m in diameter
- b) Preheating on a travelling grate 0.6 m x 4 m long
- c) Rotary kiln 1.35 m in diameter, 15 m long and with a capacity of 8 t.p.d.

4.2.3. Capacity in tonnes per 24 hr of final product

Not known

4.3. Industrial installations

4.3.1. Location

Wakayama works, Sumitomo Metal Industries Ltd.,
Japan

4.3.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

See Figure 1.

4.3.3. Capacity

Processing 20,000 t dust per month to 14,000 t sponge iron
per month

4.3.4. Method of operation (continuous or batch, number of operating hours per day)

Continuous

4.3.5. Coefficient of utilisation in time (number of working days per annum)

August 1975:	:	17 days
September 1975	:	20 days

4.3.6. Maximum rated production

At present cannot be estimated. The nominal capacity is greater than the amount of dust available. The present production is about 8000 t per month.

5. Reducing agents

5.1. Raw materials on which the reducing agent is based

Coke fines

5.2. Process for conversion of above raw materials into a form for use in reduction

Grinding the coke fines

5.3. Condition and analysis of reducing agent at moment of use

Not known

5.4. Preheat temperature on entering reduction apparatus

Not known

5.5. Gas velocity in reduction apparatus

Not known

5.6. Analysis and temperature of gas after passing through reduction apparatus

Not known

6. Ores and ferruginous materials

6.1. Source

a) Blast furnace dust

b) LD dust

- c) Dusts from sintering plants
- d) Other dusts from iron and steel works

6.2. Analysis

See Table 1

6.3. Size

See Table 1

6.4. Preparation

Pelletising

7. Additions

7.1. Material

Bentonite as binder for pelletising

7.2. Analysis

Not known

7.3. Size

Not known

8. Temperature zones in reduction
in °C in the reduction apparatus

Maximum temperature in the rotary kiln approximately 1150°C

9. Pressure zones in reduction
in kg/cm² in the reduction apparatus

Normal atmospheric pressure

10. Product of the reduction apparatus

10.1. Form

Pellets

10.2. Analysis

75.88 % Fe_{tot}; 73.96 % Fe_{met}; 97.47 % metallisation;
2.35 % C; 0.3 % S; 0.095 % Zn

10.3. Degree of reduction

Not known

10.4. Utilisation of by-products

As raw material for producing zinc

11. Further treatment of the product (10) of the reduction apparatus

11.1. Method of cooling

Quenching in water

11.2. Crushing, compacting or briquetting

None

11.3. Details of final size grading

Pellets larger than 6 mm

11.4. Method of concentration (e.g. magnetic separation or sintering)

None

12. Final product obtained after treatment according to 11

See 11.

13. Iron yield

13.1. Iron contained in product at (10) as a percentage of iron in the charge

97 - 98 %

14. Utilisation of final product

Destination of final product (blast furnace, electric or other furnace). Technical and economic differences compared with the use of pig iron and scrap

In blast furnace

15. Consumption data per tonne of final product (12)

15.1. Raw materials for reduction (5.1)

165 kg coke fines

15.2. Fuels and thermal values required for the entire process (fuel characteristics) (thermal balance)

Butane (C_4H_{10}) for drying the slurry	55 kg
for preheating the pellets	6 kg
for the rotary kiln	50 kg

15.3. Electrical energy

15.3.1. For heating

None

15.3.2. For power

700 kWh

15.4. Steam used

None

15.5. Oxygen

None

15.6. Cooling water

Not known

15.7. Refractories

No estimates available yet

15.8. Man hours of productive labour

10 men per shift

15.9. Labour and materials for routine maintenance

7 men

16. Overall capital investment

Overall investment, including foundations, approximately
1.2 million US \$.

17. Opinions expressed on the process in relation to:

17.1. State of development

The installation has been running for 5 months. At first there were mechanical troubles during commissioning, but they have since been eliminated. The utilisation of the plant in October

1975 was approximately 25 days. At present there are no technical problems in operating the plant but only mechanical problems which should be eliminated by structural alterations at the end of 1975 or beginning of 1976. Then the utilisation of the plant in time is expected to be over 90% and a reduction in the electrical energy for power is expected.

17.2. Practical operating difficulties

After the best operating conditions had been found by tests, operating the plant proved very simple. There are no problems with sticking of the charge.

17.3. Possibilities of exploitation

The process can only be used for processing the waste materials from iron and steel works, and the high sulphur content of the final product should be borne in mind.

17.4. Special requirements to be met by certain materials (e.g. refractories)

Not known

18. Bibliography

18.1. Articles and publications

1) Pellets and Granules Symposium, October 1974.

Publ.: Australasian Institute of Mining and Metallurgy.

2) Tetsu to Hagane 60, 1974, No.11

18.2. Patents

No patents yet.

Table 1. Dusts used

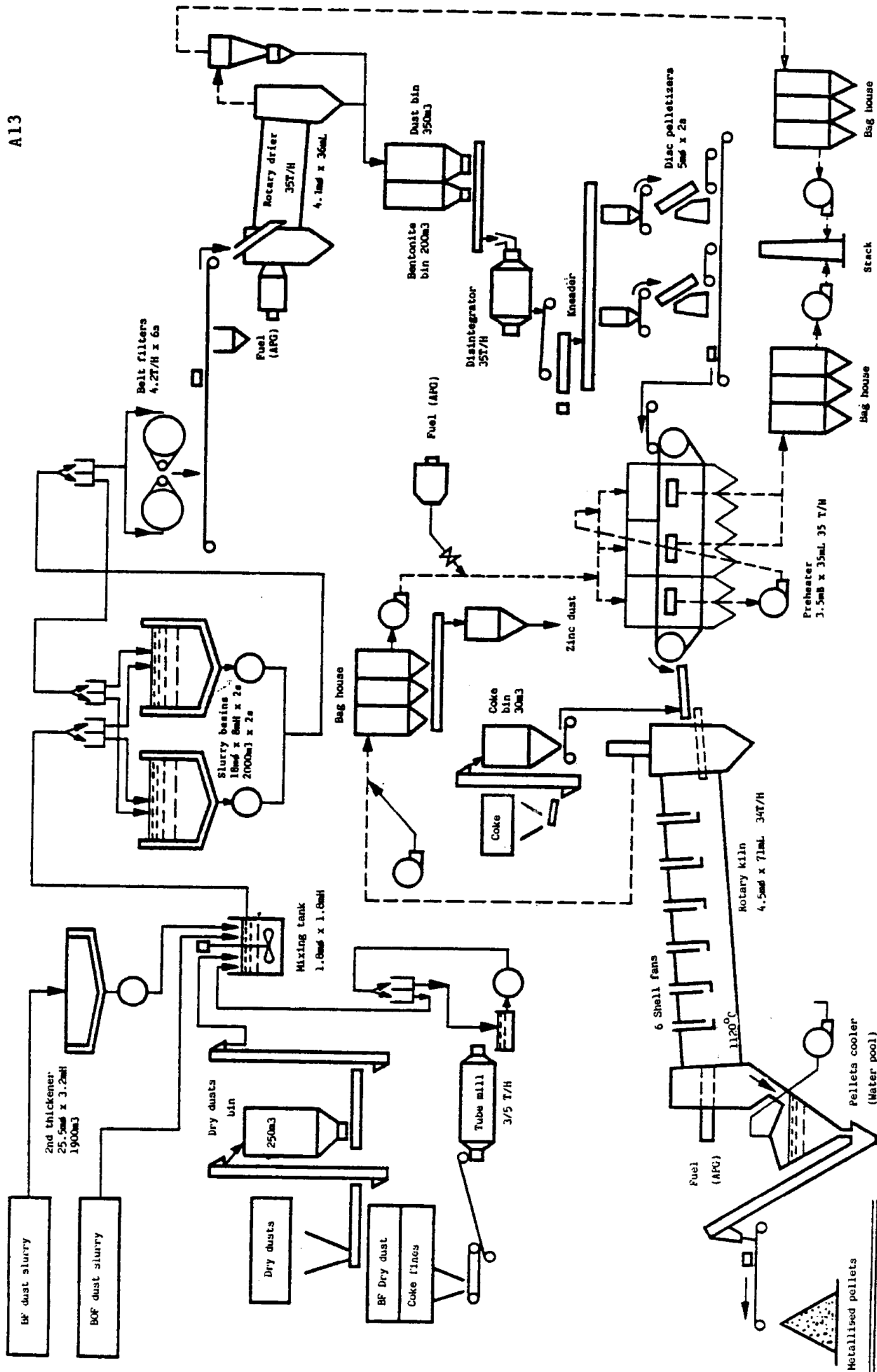
	Sinter plant	Blast Furnace			LD Converter 1		LD Converter 2		LD Converter 3
		Top gas	Casting platform	Ore bunker	Off gas	Dust extraction from casting shop	Off gas	Dust extraction from casting shop	
Dust extraction system*)	EA (dry)	EA and VW (wet)	SF (dry)	SF (dry)	EA (dry)	SF (dry)	EA (wet)	SF (dry)	EA (wet)
Quantity (dry) in t per month	210	4000	280	750	700	85	3600	50	3400
Chemical composition									
% Fe _{tot}	47.8	34.7	29.7	51.8	62.8	53.0	68.5	40.9	66.0
% FeO	4.8	3.1	2.6	3.9	0.8	9.4	51.4	12.9	9.5
% Fe ₂ O ₃	63.0	45.8	39.6	69.6	88.8	65.2	37.0	44.2	82.8
% C	8.3	33.1	33.0	3.7	0.2	4.7	0.6	3.0	0.3
% Zn	0.06	0.77	0.03	0.02	0.14	1.18	0.08	7.4	0.24
% Pb	0.38	0.24	0.01	0.01	0.01	0.03	0.03	0.09	0.04
% Alkalies (R ₂ O)	5.0	0.32	0.5	0.2	2.5	9.5	0.3	2.5	0.7
% (Cl + F)	3.23	0.19	0.13	0.10	1.09	0.96	0.24	2.32	0.17
% Gangue**)	16.9	12.4	14.4	21.6	4.8	5.4	5.0	24.4	4.3

*) EA: Electrical separation; VW: Venture scrubber; SF: Bag filter

**) Gangue: (SiO₂ + Al₂O₃ + CaO + MgO)

Table 1. (Continued)

	Sinter plant	Blast Furnace				LD Converter 1		LD Converter 2		LD Converter 3
		Top gas	Casting platform	Ore bunker	Off gas	Dust extraction from casting shop	Off gas	Dust extraction from casting shop	Off gas	
	Off gas	EA and VW (wet)	SF (dry)	SF (dry)	EA (dry)	SF (dry)	EA (wet)	SF (dry)	EA (wet)	Off gas
	EA (dry)									EA (wet)
Size analysis Proportions %	3	0.5	6	-			4.6		4.6	2.1
	28	11.1	52	4			11.7		11.7	8.9
	69	35.1	91	32			15.4		15.4	15.5
	84	46.0	97	64			17.4		17.4	20.4
	16	54.0	3	36			82.6		82.6	79.6
Specific surface area (cm ² /g)	1930				45.970	22.920		22.780		



SL/RN PROCESS

1. Characteristics

1.1. Name and history of process

SL/RN process, based on research work carried out separately by two groups of companies.

The SL (Stelco-Lurgi) process was originally developed with the aim of producing sponge iron for steelmaking furnaces, using ores rich in iron. If ores with a low iron content were present, the ferruginous materials were separated from the gangue before reduction.

In the RN process (Republic Steel - National Lead), the emphasis was on heat treatment of ores with a low iron content which were divided into their constituent parts - metallic iron and gangue - after reduction.

In order to exploit the experience of both groups to best advantage for industry a joint venture was agreed upon which led in 1964 to the creation of the SL/RN Process.

1.2. Principle of process

Reduction of iron oxides in a rotary kiln with solid reducing agents.

1.3. Inventors

Steel Company of Canada

Lurgi Gesellschaft für Chemie und Hüttenwesen m.b.H.

Republic Steel Corporation

National Lead Company

1.4. Companies involved in development

See 1.3.

2. General description (schematic drawing)

See drawing

3. Plant required

See diagram of the process.

The main unit is a rotary kiln, the characteristic feature of which is the series of shell burners disposed along the length of the kiln. These allow precise regulation of the reaction temperatures.

The material discharged from the kiln is cooled indirectly or directly/indirectly in a cooling drum.

Screens and magnetic separators are required for treating the discharged material.

A briquetting press is sometimes needed for compacting the fine particles of sponge iron. If fine ores are used, a crusher and equipment for making the fines into pellets are required.

This is not the case with titaniferous magnetite (e.g. New Zealand Steel) or ilmenite. These ores can be charged direct into the rotary kiln without being agglomerated.

4. State of development

4.1. Name of laboratories where the process is situated

Test plant of Lurgi Gesellschaft für Chemie und Hüttenwesen m.b.H. in Frankfurt.

4.2. Pilot plant

4.2.1. Location

- a) Birmingham, Alabama
- b) Hamilton, Ontario
- c) Frankfurt/Main

4.2.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

- a) rotary kiln 46 m x 2.75 m
- b) rotary kiln 35 m x 2.70 m
- c) rotary kiln 8 m x 0.50 m int. dia.
rotary kiln 12 m x 0.8 m int. dia.

4.2.3. Capacity in tonnes per 24 h of final product

- a) 50 - 75 t sponge iron per day (processing lean ores)
- b) 100 - 140 t sponge iron per day
- c) 1.5 - 2.8 t sponge iron per day
10 - 12 t sponge iron per day

4.3. Industrial installations

The installations which are in operation or under construction are listed in Table 1.

5. Reducing agents

5.1. Raw materials on which the reducing agent is based

Solid fuels, e.g. coke dust, anthracite, low temperature coke, high volatile coals, lignites. The particle size of the coal depends on the ore used and on the decomposition behaviour of the coal during reduction. It should preferably be less than 20 mm. The reduction temperature should be 100°C below the ash

fusion point. If low volatile coal is used, either oil or gas must be used to heat the kiln. If coal with a high gas content is used, no additional heating is required.

- 5.2. Process for conversion of above raw materials into a form for use in reduction

None

- 5.3. Condition and analysis of reducing agent at moment of use

-

- 5.4. Preheat temperature on entering reduction apparatus

The ore is only preheated in exceptional cases.

- 5.5. Gas velocity in reduction apparatus

The gas velocity in the reduction apparatus is approximately 0 to 5 m/s, increasing towards the kiln inlet.

- 5.6. Analysis and temperature of gas after passing through reduction apparatus

The composition of the off-gas depends on the operating conditions and the reducing coal used. Related to the dry state, the CO_2 content is about 20 to 24%, the O_2 content $< 1\%$, and the proportion of combustible constituents 0 to 10% (anthracite 0%, lignite up to 10%). The moisture content can be between 5 and 20%.

The temperature of the off-gas ranges between 800 and 1000°C .

6. Ores and ferruginous materials

Ores with iron contents of 65 to 68% are ideal. The sizes should preferably range as follows:

lump ore	3 to 20 mm
pellets	8 to 16 mm
fines	0.04 to 3 mm

6.4. Preparation

Pelletising of ore fines

7. Additions

Limestone or dolomite with a particle size of 0.2 to 3 mm can be used as a desulphurising agent. No desulphurising agent is added with coals which have a basic ash.

8. Temperature zones in reduction
in °C in the reduction apparatus

The working temperature in the rotary kiln depends on the reducing agent used and is between 900 and 1100°C in the bed of material.

9. Pressure zones in reduction
in kg/cm² in the reduction apparatus

Reduction takes place at atmospheric pressure

10. Product
of the reduction apparatus

10.1. Form

The form of the ore charged in does not alter during reduction. This means that if lump ore is used, the resulting sponge iron

is the same size as the ore (approximately 3 to 25 mm); if pellets are used, the sponge iron is in pellet form. The fine particles which arise in the kiln (~3 mm) can either be processed directly in the melting furnace or briquetted.

10.2. Analysis

The composition of the sponge iron depends on the analysis of the ore in the charge.

10.3. Degree of reduction

The degree of reduction is adjusted according to what the reduced product is to be used for and is between partial reduction (about 45%) and high degrees of reduction (about 96%).

10.4. Utilisation of by-products

No details

11. Further treatment of the product (10) of the reduction apparatus

11.1. Method of cooling

The material discharged from the kiln is cooled either indirectly or in a direct/indirect manner with water in a cooling drum.

11.2. Crushing, compacting or briquetting

Briquetting is feasible but has so far not been necessary.

11.3. Details of final size grading

See 10.1.

11.4. Method of concentration (e.g. magnetic separation or sintering)

The reduced ore in the material discharged from the kiln is separated from the non-magnetic material by means of screening and magnetic separation.

12. Final product obtained after treatment according to 11

12.1. Form

See 10.2.

12.2. Analysis

See 10.2.

If Hilton pellets are used with 66.7% total Fe, 0.01% S and 3.87% gangue, sponge iron with 93.35% total Fe, 89.85% metallic Fe, 2.50% Fe⁺⁺, 0.012% S and 0.07% C can be produced.

13. Iron yield

13.1. Iron contained in product (10) as a percentage of iron in the charge

The iron yield is between 92 and 98% depending on the ore used.

13.2. Iron contained in final product (12) as a percentage of iron in the charge

No details

14. Utilisation of final product

Destination of final product (blast furnace, electric or other furnace). Technical and economic differences as compared with using pig iron and scrap

If sponge iron is used in an electric arc furnace a substantial increase in output can be obtained. With 75% sponge iron charge,

increases in output of up to 50% were obtained by shortening the melting time. A further advantage of sponge iron is its low and, above all, controllable content of undesirable tramp elements. The consumption of electricity can be considerably decreased by using reduced ore in an electric smelting furnace.

Reduced ore has also been tried out in blast furnaces. A 0.7% increase in capacity and a 0.5% drop in coke consumption can be reckoned on per percentage of metallisation in a blast furnace burden.

15. Consumption data per tonne of final product (12)

Ore approximately 1.45 t, fixed carbon 310 to 390 kg, limestone or dolomite 0 to 70 kg, water 2.5 to 3.5 m³, electrical energy 50 to 70 kWh, repair costs 5 to 10 DM, man hours for production and routine maintenance 0.4 to 0.8 h.

16. Overall capital investment

16.1. For the plant described in (3) and related to one tonne of capacity for the final product (12)

The capital investment costs depend very much on the size and location of the plant. The raw materials, which determine what ancillary equipment is required (e.g. crushing and pelletising of fines), also influence the price.

The cost of an SL/RN plant based on lump ore is about 200 to 300 DM per annual tonne of sponge iron. This figure diminishes, the larger the size of the plant.

16.2. Construction costs of ancillary plant not covered in (3)

No details

17. Opinions expressed on the process in relation to:

17.1. State of development

The process can be applied industrially

17.2. Practical operating difficulties

No details

17.3. Possibilities of exploitation

The process can be used wherever suitable reducing agents are available (see 5). It can be used both for pre-reducing iron ores in combination with electric furnace smelting and for further processing in electric arc furnaces or other steel-making furnaces and also for further processing in blast furnaces. One application which is of interest to the iron and steel industry is processing zinc-containing dust since zinc can be volatilised to a large extent during reduction.

17.4. Special requirements to be met by certain materials (e.g. refractories)

No special requirements are imposed for materials and refractories.

18. Bibliography

18.1. Articles and publications

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pp. 977-89.
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June 1965.
- 3) Janke, W.: Metalurgia, Vol. 22, February 1966, pp.101-06.

- 4) Smith, R.P.: 55. Annual Meeting of American Institute of Chemical Engineers, December 1962.
- 5) Meyer, K., G. Heitmann and W. Janke: Journal of Metals, June 1966.
- 6) Pearce, F.J.: Canad. Min. Met. Bull., 57 (1964) pp. 429-33.
- 7) Peart, J.A., and F.J. Pearce: Journal of Metals, December 1965, pp. 1396-1400.
- 8) Janke, W., and H. Serbent: Intern. Symposium on direct reduction of iron ores, technical and economical aspects. Bucharest, Rumania, September 18 - 23, 1972.

18.2. Patents

Germany:	1 199 296	1 583 954
	1 262 311	6 750 292
	145 876 809	25 011 829
	15 338 520	
Australia:	275 651	
	428 180	
	435 019	
Belgium:	615 733	
Canada:	722 591	
	822 038	
	844 530	
	875 475	
	876 586	
	741 733	
U.K.	1 043 548	
	1 203 532	
	1 240 239	
France:	1 360 612	
	6 902 891	
	6 919 723	
	1 322 423	
India:	88 309	
	119 513	
	121 543	

Italy:	700 744
	868 465
Japan:	463 550
	79 260/74
Mexico:	72 422
New Zealand:	155 203
Peru:	7 017
Spain:	289 176
	275 425
South Africa:	63/2832
	1302/68
	467/69
	3790/69
Sweden:	209 981
	347 293
USA:	3 219 436
	3 663 201
	3 556 772
	3 584 885
	3 235 375
	3 185 464

Table 1. SL/RN plants and rotary kilns for pre-reduction in operation or under construction

Company	New Zealand Steel Ltd. New Zealand	Acos Finos Piratini Brazil	Nippon Kokan KK Fukuyama Japan	Highveld Steel and Vanadium Corp. South Africa	Hecla Mining Arizona USA	Steel Comp. of Canada Canada
Plant size	1 kiln 4 x 75 m	1 kiln 3.6 x 50 m	1 kiln 6 x 70 m with pre- hardening grate 250 m ²	6 kilns 4 x 60 m kiln no.7 ordered start up: late 1976	1 kiln 3.6 x 50 m	1 kiln 6 x 125 m
Ore throughput	190,000 tpy	95,000 tpy	500,000 tpy	1,750,000 tpy	95,000 tpy	520,000 tpy
Ore	Iron sand concentrate 0.04-0.4 mm 60.0% Fe 8% TiO ₂	Pellets 67.0% Fe	BF, BOF-dust pellets and ore fines	Lump ore 55% Fe 1.6% V ₂ O ₅	Leach residue pellets 50-53% Fe	Pellets 66.5% Fe
Coal	sub-bituminous	high-volatile	sub-bituminous	high-volatile	sub-bituminous	sub-bituminous
Product and further processing	High-metallised concentrate, arc furnace, steel	High - metallised pellets, arc furnace, steel	High - metallised pellets, blast furnace, pig iron, Zn,Pb	45% pre-reduced ore, smelter, pig iron, vanadium slag	High - metallised pellets, copper, cementation	High - metallised pellets, arc furnace, steel
Remarks			Start up late 1974	Lurgi designed and delivered the kilns, process developed by HSVC	Start up early 1975	Start up early 1975

Al4

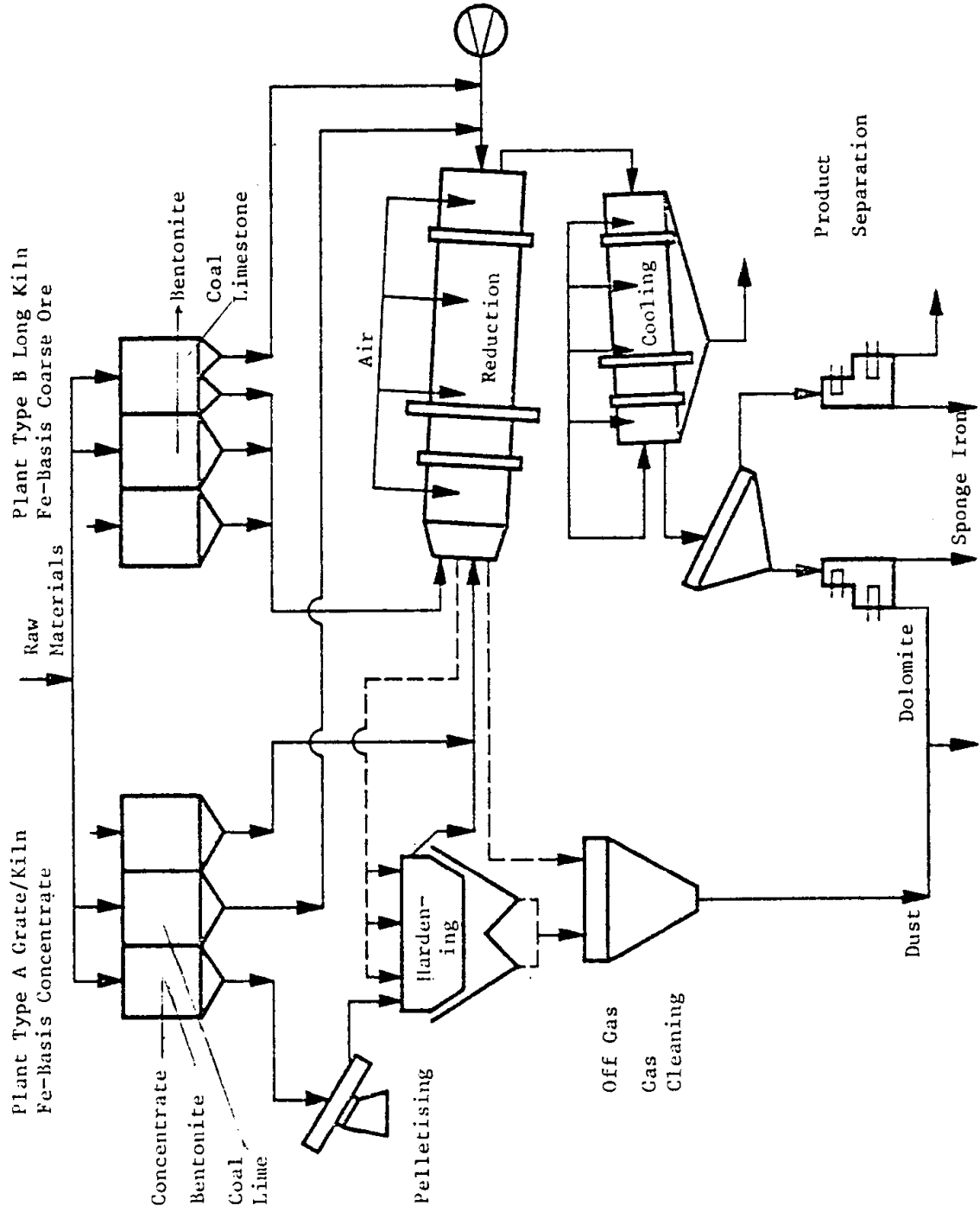


Fig. 1. Principle of SL/RN Process

SPM PROCESS

1. Characteristics

1.1. Name and history of process

SPM process

The process was developed for producing sponge iron for use in the cupola furnace. In an attempt to replace pig iron, tests were started in October 1967 on a pilot rotary kiln by the Kubota Ironworks.

At first green pellets, already containing the reducing agent, were charged to the rotary kiln, but there were difficulties regarding the stacking resistance of the pellets. Then the charge was altered. Ore fines and coal were mixed without comminution and charged directly to the rotary kiln without agglomeration. It was found that under controlled conditions agglomeration and reduction occur simultaneously in the kiln.

In November 1971 Sumitomo Metal Industries carried out tests on processing dusts from iron and steel works. After 10 days' operation the conditions for processing were established. In March 1975 a new plant was commissioned.

1.2. Principle of process

Oxides of iron and zinc are reduced in the rotary kiln by coke fines and coal, which are charged to the kiln with the dust. At the end of the reduction the reduced material is agglomerated because of the rotary movement of the kiln and the heat which is produced by a burner at the end.

1.3. Inventors

Takashi Baba, Toshihiko Kawabata and Susumu Funahashi

1.4. Companies involved in development

Kubota Ironworks Co., Ltd.,
Sumitomo Metal Industries Ltd.

2. General description (schematic drawing)

See Figure 1.

3. Plant required

See Figure 1.

4. State of development

4.1. Name of laboratories where the process is situated

Central Laboratories, Kubota Ironworks Co., Ltd.,
Japan.

4.2. Pilot plant

4.2.1. Location

Funabashi works, Kubota Ironworks Co., Ltd.

4.2.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

Rotary kiln	2.1 m in dia. x 31 m long
Rotary cooler	1.2 m in dia. x 4 m long

4.2.3. Capacity in tonnes per 24 hr of final product

300 t.p.d.

4.3. Industrial installations

4.3.1. Location

Kashima works, Sumitomo Metal Industries Ltd.

4.3.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

See Figure 1.

4.3.3. Capacity in tonnes per 24 hr of final product

Processing 13,000 t dust per month to 8500 t sponge iron per month

4.3.4. Method of operation (continuous or batch, number of operating hours per day)

Continuous

4.3.5. Coefficient of utilisation in time (number of working days per annum)

1975: April 10 days, May 15 days, June 19 days,
July 16 days, August 21 days

4.3.6. Maximum rated production

Not known yet

5. Reducing agents

5.1. Raw materials on which the reducing agent is based

Coke fines from blast furnace dust and anthracite

- 5.2. Process for conversion of above raw materials into a form for use in reduction

None

- 5.3. Condition and analysis of reducing agent at moment of use

Not known

- 5.4. Preheat temperature on entering reduction apparatus

Not known

- 5.5. Gas velocity in reduction apparatus

Not known

- 5.6. Analysis and temperature of gas after passing through reduction apparatus

Not known

6. Ores and ferruginous materials

See Table 1.

7. Additions

None

8. Temperature zones in reduction
in °C in the reduction apparatus

1200°C

9. Pressure zones in reduction
in kg/cm² in the reduction apparatus

Normal atmospheric pressure

10. Product
of the reduction apparatus

10.1. Form

Sponge iron agglomerated to lumps

10.2. Analysis

75 % Fe_{tot}; 58.5 % Fe_{met}; 2.2 % C; 0.12 % P;
0.30 % S; 0.10 % Zn; 4.9 % CaO; 1.2 % MgO;
7.3 % SiO₂; 3.8 % Al₂O₃

10.3. Degree of reduction

Degree of metallisation 75 - 80 %

10.4. Utilisation of by-products

Used for zinc production

11. Further treatment of the product (10) of the reduction apparatus

11.1. Method of cooling

Direct water cooling in the rotary cooler

11.2. Crushing, compacting or briquetting

Screening

11.3. Details of final size grading

Sintered material with a particle size of approximately 7 mm

11.4. Method of concentration (e.g. magnetic separation or sintering)

None

12. Final product obtained after treatment according to 11

See 11.

13. Iron yield

97-98 % Fe in the final product including undersize; the proportion of lump sponge iron over 7 mm is approximately 50 % of the total amount obtained.

14. Utilisation of final product

Destination of final product (blast furnace, electric or other furnace). Technical and economic differences compared with the use of pig iron and scrap

Fractions > 7 mm: for charging to the blast furnace

Fractions < 7 mm: for feeding to sintering plants

15. Consumption data per tonne of final product (12)

15.1. Raw materials for reduction (5.1)

160 kg

15.2. Fuels and thermal values required for the entire process (fuel characteristics) (thermal balance)

Coke oven gas (4.6 Mcal/m³)

for sludge drying 80 m³

for kiln 140 m³

15.3. Electrical energy

220 kWh for power

15.4. Steam used

None

15.5. Oxygen

None

15.6. Cooling water

Not known

15.7. Refractories

Not known

15.8. Man hours of productive labour

10 men per shift

15.9. Labour and materials for routine maintenance

12 men

16. Overall capital investment

Approximately 10 million US \$ including foundations

17. Opinions expressed on the process in relation to:

17.1. State of development

Initial mechanical troubles have since been overcome but the optimum conditions for operating the kiln have not yet been found. In the tests on the pilot plant the proportion < 7 mm was about 30 % of the total yield. This was returned to the rotary kiln. In the industrial installation, however, the fine particles below 7 mm constitute around 60 - 70 % and about half is returned to the kiln.

The degree of sintering of the dusts in the kiln is affected by the temperature, the holding time and the height of the dam at the exit end. Tests to clarify these relationships are still being carried out.

17.2. Practical operating difficulties

The optimum conditions for running the kiln have not yet been found, so that operation is still difficult.

17.3. Possibilities of exploitation

Because of the anthracite added the sponge iron contains sulphur, so that it cannot be used for steelmaking.

17.4. Special requirements to be met by certain materials (e.g. refractories)

There are no special requirements as regards the materials to be charged; only low costs are incurred for pre-treatment, for example for the accurate adjustment of the humidity.

18. Bibliography

18.1. Articles and publications

None so far

18.2. Patents

None granted so far

Table 1. Dusts used

Chemical Composition	Blast Furnace	LD Converter	Ore Stocks	Blooming Mill	Hot Rolling Mill	Pickling Residues
% C	43.15	1.41		23.65	1.27	0.60
% Fe _{tot}	24.24	69.23	41.45	48.52	69.60	68.41
% Fe _{met}	0.87	16.14		0.02	9.68	0.03
% FeO	2.87	61.19				
% Fe ₂ O ₃	31.30	4.22	59.16			
% SiO ₂	5.94	1.82	8.22			
% Al ₂ O ₃	2.84	0.39	3.07			
% CaO	2.02	5.66	15.14			
% MgO	0.59	0.41	2.79			
% P	0.080	0.103				
% S	0.807	0.065		0.012	0.047	0.11
% Zn	3.74	0.55	0.02	0.023	0.120	0.006
% Na ₂ O	0.11	0.16		0.66	0.20	0.007
% K ₂ O	0.22	0.03		0.01	0.012	0.023
% Pb			0.01	0.036	0.006	0.001
% Cl				0.026	0.042	0.358
% F				0.02	0.03	0.01
Size fractions in %						
> 0.50 mm	-	-	0.03	-	-	-
0.50/0.25 mm	0.26	1.36	0.20	4.96	-	0.47
0.25/0.15 mm	3.38	3.80	0.25	3.07	0.79	3.26
0.15/0.074 mm	20.77	12.73	7.62	14.07	3.44	40.34
0.074/0.044 mm	18.80	9.67	14.14	14.98	12.57	15.13
< 0.044 mm	56.79	72.44	77.75	62.92	83.20	40.80
t/m ³ (dry)	2100	5900	3800	1000		



CARBOTHERMIC PROCESS

1. Characteristics

1.1. Name and history of process

Carbothemic process (name given by the writers to a method of reduction)

1.2. Principle of process

Reduction by carbon in an enclosure with neutral atmosphere, heated externally.

1.3. Inventors

Laszlo Visnyovsky, Janos Horvath and Janos Illyes, engineer, director and head of the scientific department respectively at the Hungarian Institute for Iron Research.

1.4. Companies involved in development

No development, work in laboratory and pilot plant.

2. General description (schematic drawing)

The diagram attached shows the laboratory furnace.

3. Plant required

Plant for crushing, screening, briquetting or pelletising of ores.

Charging system for reducing furnace. Any reducing furnace (shaft or tubular furnace, reactor and even covered agglomeration belt) in which a non-oxidising or neutral atmosphere can be maintained.

System for extracting the products, screening, rotary drum for abrasion, magnetic separator.

4. State of development

At laboratory pilot stage

4.1. Name of laboratories where the process is situated

Hungarian Institute for Iron Research

4.2. Pilot plant

At the Institute

4.2.1. Location

Budapest, Hungary

4.2.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

The laboratory furnace is a cylinder 780 mm long and 160 mm in external diameter, internal diameter 90 mm (see attached diagram). The reduction furnace of the pilot plant is of shaft type 6.5 dm^2 in cross-section at the top and 15.1 dm^2 at the bottom for a height of 3 metres, that is an internal volume of 0.44 m^3 .

4.2.3. Capacity in tonnes per 24 h of final product

The tests (with a transit time through the reduction plant of 6 hours) enabled 1.2 tonnes of a briquettes/coke dust mixture to be processed, that is 2.75 t.p.d. per m³ of furnace of materials charged. According to the inventors, the means or processes for recovering heat enabling the materials charged to be pre-heated, would enable 5.5-6 tonnes per day per m³ of furnace to be processed (transit time being reduced to 4 hours).

It appears that the final reduced product represents approximately a quarter, by weight, of the charge.

4.3. Industrial installations

There is as yet no industrial installation

5. Reducing agents

5.1. Raw materials on which the reducing agent is based

Coal or coke with low sulphur content

5.2. Process for conversion of above raw materials into a form for use in reduction

Coal fines or coke dust are used

5.3. Condition and analysis of reducing agent at moment of use

Coke dust, 5-10 mm

5.4. Preheat temperature on entering reduction apparatus

Room temperature; pre-heating takes place in the reduction installation

5.5. Gas velocity in reduction apparatus

Not known

5.6. Analysis and temperature of gas after passing through reduction apparatus

CO₂ + CO in varying proportions, temperature around 700°C

6. Ores and ferruginous materials

6.1. Source

The tests were carried out on ferriferous quartzite concentrates from the Krivoi-Rog basin (U.S.S.R)

6.2. Analysis

These concentrates usually contain 61-63% Fe; the maximum SiO₂ is 14%, P 0.025% and S 0.03%.

6.3. Size

0 - 0.2 mm 80% i.e. 20% above 0.2 mm

6.4. Preparation

Mixing of ore concentrates and a reducing agent and one or more reduction activating agents (tar, K₂CO₃ or NaNO₃); rolling mill scale was also added during pilot tests (see 7.2). Briquetting of this mixture into cylindrical modules 25 mm in diameter and 25 mm high or pelletising.

These briquettes are then charged to the reduction furnace, with an equal volume of coke dust.

7. Additions

7.1. Material

Activating agent (K_2CO_3 or $NaNO_3$); metal powders obtained after processing of the product (see 9) are also recycled by being included in the mixture described in 6.4.

7.2. Analysis

The briquettes used to obtain the final product given under 12 were composed of a mixture of:

Krivoi-Rog concentrates	45%
Mill scale	45%
Coke dust	5%
Coal tar	4%
$NaNO_3$	1%

and had the following contents:

total Fe 58%; C 11%; S 0.2%; Na_2O 0.92%; SiO_2 6.42%

7.3. Size

The additives should be fine

8. Temperature zones in reduction
in °C in the reduction apparatus

The gas temperature in the reduction zone is 1100-1140°C

9. Pressure zones in reduction
in kg/cm² in the reduction apparatus

Not known

10. Product
 of the reduction apparatus

10.1. Form

The briquettes are agglomerated in "packets".

11. Further treatment of the product (10) of the reduction apparatus

11.1. Method of cooling

Cooling in free air in the pilot installation

11.2. Crushing, compacting or briquetting

Screening the products from the reducing plant to separate the briquettes from the residual coke dust which is recycled after magnetic separation.

Abrasion of the briquettes in the drum for about half an hour to remove dust adhering to their surface and screening.

The metal powders from screening after abrasion and those from the magnetic separation of the residual coke dust are recycled to the pelletising plant.

12. Final product obtained after treatment according to 11

12.1. Form

Briquettes or pellets as before reduction

12.2. Analysis

Product obtained by processing briquettes the analysis of which is given under 7.2.

total Fe : 83.4%, metallic Fe : 82.37%, degree of reduction : 99%, C : 1.44%, S : 0.092%, SiO₂ : 9.14%, Al₂O₃ : 3.06%, CaO : 2.00%, MgO : 0.5%, MnO : 0.38%, P : 0.02%

13. Iron yield

No details

14. Utilisation of final product

Destination of final product (blast furnace, electric or other furnace). Technical and economic differences compared with the use of pig iron and scrap

No details

15. Consumption data per tonne of final product (12)

No details

For labour on production the inventors estimate 1 operator and 3 labourers per team for the furnace and 2 labourers per team for charging.

16. Overall capital investment

No details

17. Opinions expressed on the process in relation to:

17.1. State of development

Laboratory pilot study of a method of reducing by means of carbon.

18. Bibliography

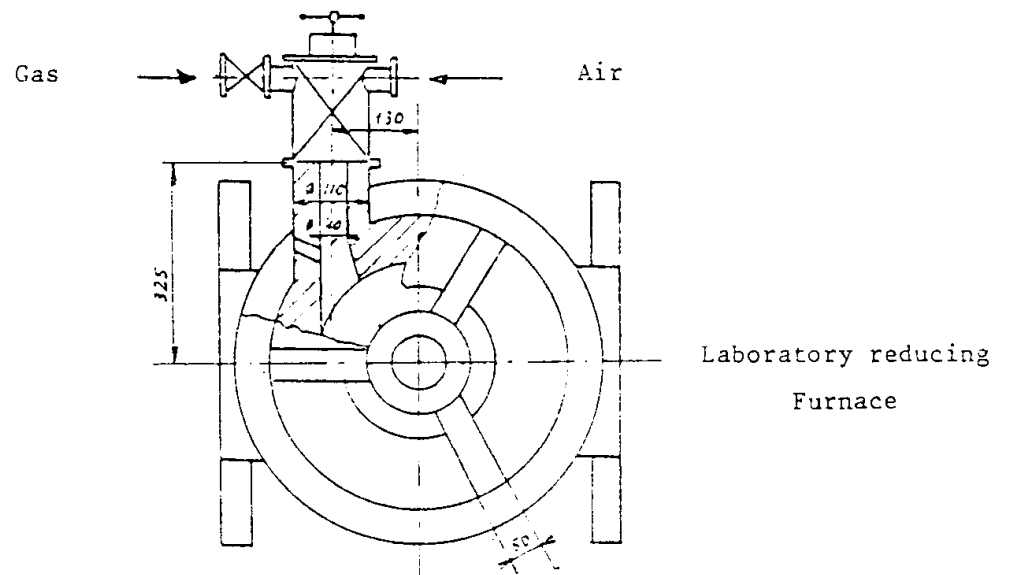
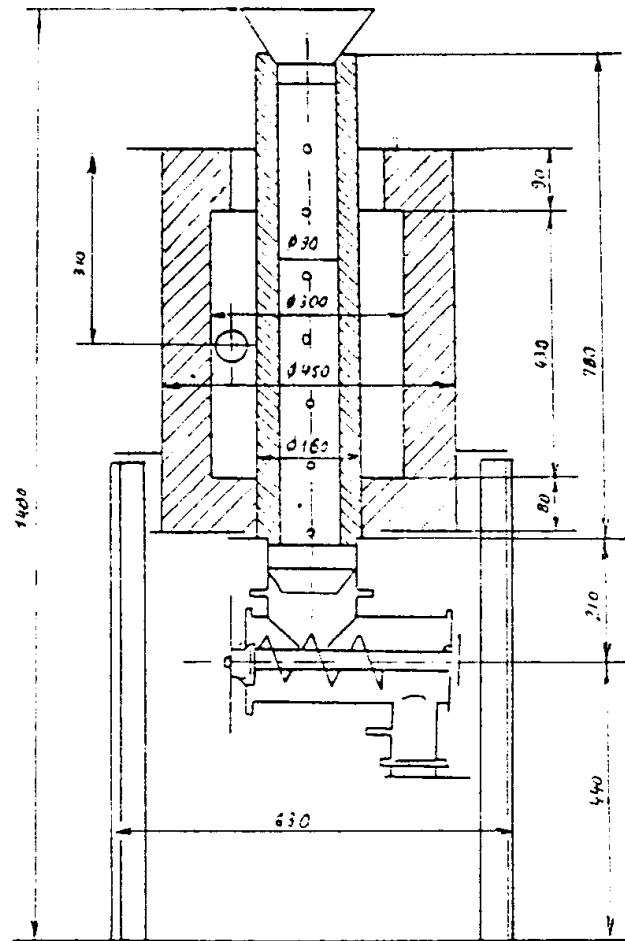
18.1. Articles and publications

Paper presented by the inventors to the "Seminar on the economic and technical aspects of the reduction of iron ore" organised by the Steel Committee of the EEC at Bucharest (Rumania) from 18th to 23rd September 1972.

18.2. Patents

Hungarian patent number 158822

B1



CENTRIFUGE IRONMAKING PROCESS

1. Characteristics

1.1. Name and history of process

Centrifuge Ironmaking Process. Development commenced within the British Steel Corporation

1.2. Principle of process

Reduction of iron ore using low grade coal inside a high speed rotating vessel, the refractory lining of which is protected by the centrifuged product.

1.3. Inventors

E.W. Voice and D.A. Hawkes of British Steel Corporation, following an initial concept published by Professor S. Eketorp in 1954.

1.4. Companies involved in development

British Steel Corporation

2. General description (schematic drawing)

The process is illustrated in Figure 1. The reduction furnace rotates at sufficiently high speed to maintain the liquid constituents inside the furnace as annular layers against the refractory surface. The heaviest liquid constituent (iron) maintains a layer immediately in contact with the refractory surface, hence protecting it from attack by highly corrosive ferruginous slags. The raw materials for the process (coal, ore, limestone, oxygen) are fed through feeders at one end of the reduction furnace. The liquid product spins out from the other end of the furnace when the upper level of the exhaust end dam has been reached.

3. Plant required

As illustrated in Figure 1 and described in Section 2.

4. State of development

4.1. Name of laboratories where the process is situated

Corporate Advanced Process Laboratory

4.2. Pilot plant

4.2.1. Location

Teesside

4.2.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

Rotating cylindrical furnace, length 3 m and internal diameter 0.65 m, rotational speed variable between 160 and 200 rpm.

4.2.3. Capacity in tonnes per 24 h of final product

Maximum rated production of 1 tonne per hour of liquid metal, using a continuous mode of operation

4.3. Industrial installations

None

5. Reducing agents

5.1. Raw materials on which the reducing agent is based

Low grade coal.

- 5.2. Process for conversion of above raw materials into a form for use in reduction

Pit head/as-mined coal of size fraction 25 mm and below.

- 5.3. Condition and analysis of reducing agent at moment of use

Analysis of coal approximately 50% fixed carbon, 35% volatile matter and 9% mineral matter.

- 5.4. Preheat temperature on entering reduction apparatus

No preheat

- 5.5. Gas velocity in reduction apparatus

Approximately 10 m per second

- 5.6. Analysis and temperature of gas after passing through reduction apparatus

Approximate temperature 2000°C. Gas analysis variable within a wide range of experimental conditions

6. Ores and Ferruginous Materials

- 6.1. Source

A wide variety of ores can be used for the process, although the current pilot plant has used Lac Jeannine concentrates exclusively.

- 6.2. Analysis

64% total iron

- 6.3. Size

Below 2 mm.

6.4. Preparation

As received

7. Additions

7.1. Material

Limestone

7.2. Analysis

54% CaO

7.3. Size

Below 16 mm

8. Temperature zones in reduction
in °C in the reduction apparatus

Gas temperature approximately 2000°C and temperature of molten slag and iron within the range 1500°C - 1600°C.

9. Pressure zones in reduction
in kg/cm² in the reduction apparatus

Atmospheric pressure

10. Product
of the reduction apparatus

10.1. Form

Liquid slag and iron

10.2. Analysis

The product analysis is variable due to the experimental nature of the pilot plant. However, the slag iron concentration is normally in the range of 10% - 25% and the product metal in the range 0.5% - 2.5% carbon.

10.3. Degree of reduction

10.3.1. Proportion of oxygen removed from iron oxides as a percentage of the total oxygen combined with iron in the ores

Variable in the range 80% - 97%

10.3.2. Metallic iron as a percentage of the total iron content of the product

All the "useful" product is liquid iron, which can be separated from the liquid slag.

10.4. Utilisation of by-products

As the process is still at pilot plant stage, by-products of the process have not been utilised yet.

11. Further treatment of the product (10) of the reduction apparatus

Not known

12. Final product obtained after treatment according to 11

Not known

13. Iron yield

13.1. Iron contained in product (10) as a percentage of iron in the charge

Yield in the range 70% - 96%

14. Utilisation of final product

Destination of final product (blast furnace, electric or other furnace). Technical and economic differences as compared with using pig iron and scrap

The final product in the form of hot metal can have the same uses as normal blast furnace hot metal. The possible exception to this is that the carbon content in the iron will be lower than the blast furnace which in turn will put some limitations on LD operation. Also some external desulphurisation will be required either before or after the steelmaking process.

15. Consumption data per tonne of final product (12)

15.1. Raw materials for reduction (5.1)

1.5 tonnes of iron ore, 0.3 tonnes of limestone and a low grade coal input varying between 1 tonne and 2 tonnes per tonne of liquid metal product. Oxygen consumption variable over a wide range.

16. Overall capital investment

This information is not yet quantifiable as the process is still at the small pilot plant stage

17. Opinions expressed on the process in relation to:

17.1. State of development

A pilot plant of 1 tonne per hour nominal capacity has been operating at Teesside for approximately five years.

17.2. Practical operating difficulties

The principal operating difficulties are associated with maintaining an extended refractory life for the brickwork in the exhaust end dam of the furnace.

17.3. Possibilities of exploitation

Exploitation will depend on overcoming difficulties put forward under Section 17.2. and the extent to which fuel consumption can compare with existing processes such as the blast furnace.

17.4. Special requirements to be met by certain materials (e.g. refractories)

None

18. Bibliography

18.1. Articles and publications

D.A. Hawkes - "BISRA's Continuous Ironmaking Project".
Proceedings of the meeting "Alternative Routes to Steel" held after the Annual General Meeting of the Iron & Steel Institute at the Hilton Hotel, London, 5/6 May 1971.

18.2. Patents

P.567 Centrifuge Ironmaking

Great Britain	1 270 669	Granted
Australia	456 802	Granted
Belgium	760 174	Granted
Canada	911 173	Granted
France	7 047 500	Granted
Germany	P 2 059 448	Pending

Holland	7 017 807	Pending
Italy	923 229	Granted
Japan	121 899/70	Pending (Filed 29 December 1970)
Sweden	348 225	Granted
USA	3 814 596	Granted

P.7156 Injection System

Great Britain	1 326 456	Granted
Australia	458 667	Granted
Belgium	778 591	Granted
Canada	953 507	Granted
France	7 202 195	Granted
Germany	P 2 202 377	Pending
Holland	7 200 554	Pending
Italy	948 912	Granted
Japan	9 990/72	Pending (Filed 28 January 1972)
Sweden	909/72	Pending (Filed 27 January 1972)
USA	3 740 044	Granted

P.7349 Cooling System

Great Britain	1 405 173	Granted
Australia	53 096/73	Pending
Belgium	796 647	Granted
Canada	165 373	Pending (Filed 6 March 1973)
France	7 308 667	Pending
Germany	P 2 312 224	Pending
Holland	7 303 527	Pending
Italy	980 555	Granted
Japan	28 629/73	Pending (Filed 13 March 1973)
Sweden	7 303 408	Pending (Filed 12 March 1973)
USA	3 806 311	Granted

B2

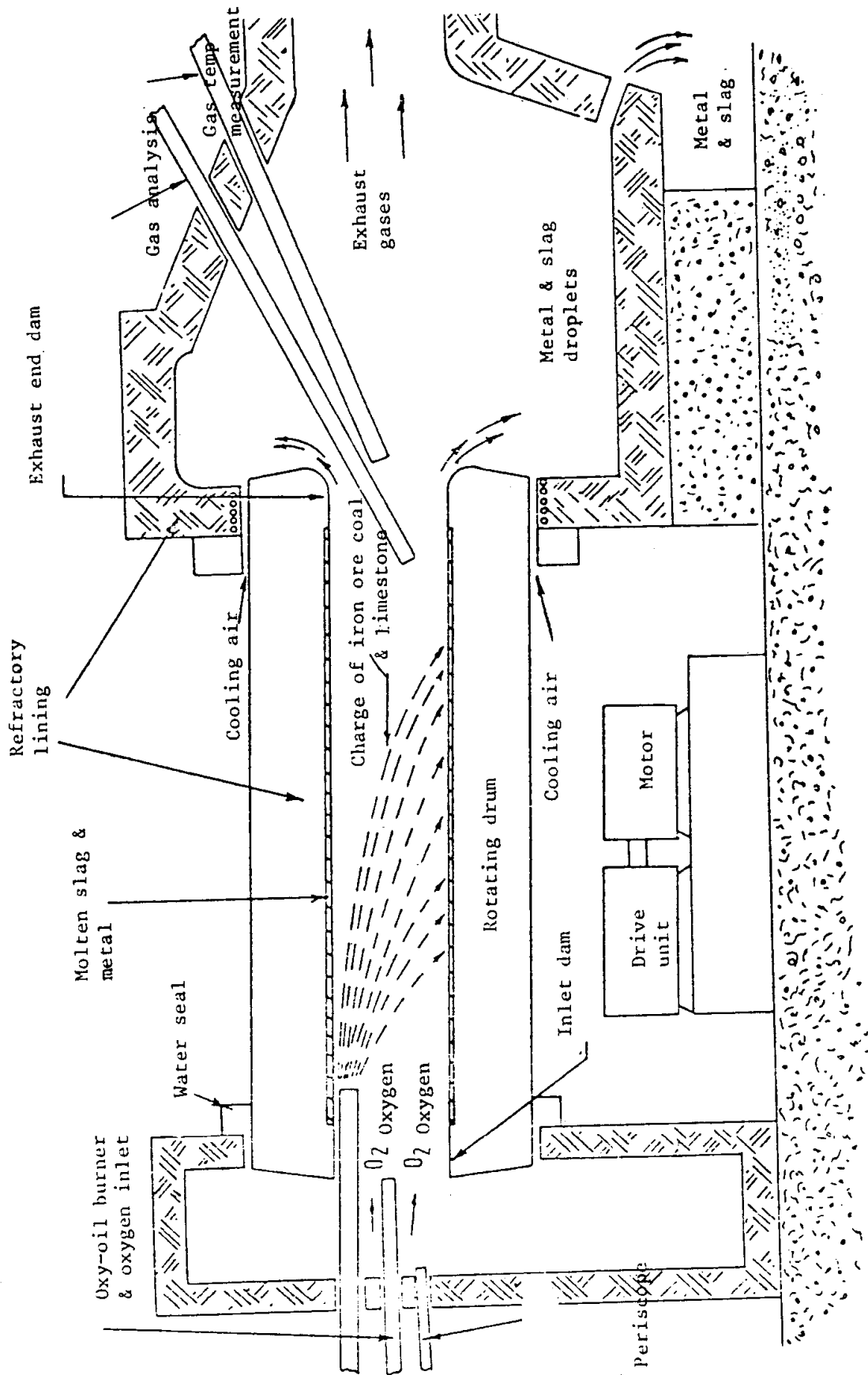


Fig. 1. Basic Concept of CIP Pilot Plant.

ECHEVERRIA PROCESS

1. Characteristics

1.1. Name and history of process

Echeverria process

Difficulties with the supply of small scrap during the period from 1952 to 1957 led the iron and steelmaking company, Patricio Echeverria S.A., to investigate the possibility of producing sponge iron by direct reduction of iron ores. The reduction plant was erected in 1957-58.

1.2. Principle of process

Reduction of lump ore (10/50 mm) or pellets by a solid reductant at about 1050°C. Reduction furnace: shaft furnace heated externally.

The screened iron ore is mixed with the solid reductant (anthracite) and the limestone necessary to fix the sulphur in the coal; this mixture is charged into the upper, "pre-heating" part of the retort.

The pre-heated charge descends to the reduction zone of the retort. The carbon monoxide resulting from the reduction of the iron oxides by the carbon of the solid reductant is burned with air in the pre-heating zone. At the same time, a small proportion of the solid reductant burns in this zone. The pre-heating temperature is regulated by controlling the combustion air flow. The furnace itself is heated externally, by burning producer gas. After reduction the sponge iron, gangue, ash and excess reductant are cooled and discharged.

1.3. Inventors

and

1.4. Companies involved in development

Patricio Echeverria S.A., Legazpia (Guipuzcoa), Spain.

Exclusive licence held by Siderox S.A., 15 rue des Voisins,
Geneva, Switzerland.

2. General description (schematic drawing)

See Figures 1 and 2

3. Plant required

Ore crushing and screening plant (not needed when using pellets).

Reduction furnace (with pre-heating of ore); means of heating
the furnace, cooling and discharge of the sponge iron.

Magnetic separator.

4. State of development

4.1. Name of laboratories where the process is situated

Patricio Echeverria S.A., Legazpia (Guipuzcoa), Spain.

4.2. Pilot plant

No information

4.3. Industrial installations

4.3.1. Location

Plant at Legazpia

4.3.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

Two batteries each of 20 retorts.

Initially the plant comprised only 1 battery.

Two gas generators.

Each retort 10m high and 1m^3 in volume is made of alloy steel which allows continuous operation at high temperature; the lower part (high temperatures) is lined with refractories.

Each retort, placed in a refractory-lined enclosure, is heated externally by producer gas (natural gas or fuel oil may also be used).

Each retort has 2 burners diametrically opposite one another and comprises the following three zones:

- preheating
- reduction
- cooling and discharge

4.3.3. Capacity in tonnes per 24 h of final product - per unit of production

The production capacity of each retort is 1.5 tonnes of sponge iron per 24 hours, i.e. approximately 500 tpy and 20,000 tpy for the whole plant.

4.3.4. Method of operation (continuous or batch, number of operating hours per day)

Process essentially continuous

4.3.5. Coefficient of utilisation in time (number of working days per annum)

330 days per year

4.3.6. Maximum rated production

Not known

5. Reducing agents

5.1. Raw materials on which the reducing agent is based

The reductant is solid; coke, coal, anthracite or charcoal may be used.

5.2. Process for conversion of above raw materials into a form for use in reduction

None

5.3. Condition and analysis of reducing agent at moment of use

The size of the solid reductant is between 8 and 25 mm.

Analysis of types of anthracite used:

	% Volatile matter	% Fixed Carbon	% S	% Ash
Anthracite A	6.0	78.9	0.80	12.4
Anthracite B	8.2	62.5	1.10	27.3

The excess reductant is recovered and recycled.

5.4. Preheat temperature on entering reduction apparatus

The reductant is pre-heated at the same time as the ore in the upper part of the retort by combustion of part of the gases from the reduction. The pre-heat temperature is not given in the literature.

A small amount of solid reductant may burn at the same time as the gas in the pre-heating zone.

5.5. Gas velocity in reduction apparatus

No details

5.6. Analysis and temperature of gas after passing through reduction apparatus

No details

6. Ores and ferruginous materials

6.1. Source

So far, ores from Soria, Bilbao, Cerain and the Rif have been used. Swedish pellets have also been processed.

6.2. Analysis

	% Fe	% CaO	% SiO ₂	% S	Melting % losses
Soria	60.3	-	11.6	0.020	1.3
Bilbao	56.4	-	7.9	0.020	8.8
Cerain	59.6	7.0	0.5	0.192	3.3
Rif	65.2	-	3.3	0.070	2.0
Swedish Pellets	66.1	-	2.4	0.018	0.3

6.3. Size

The ores are crushed to 50 mm and screened at 10 mm. The Lepazpia plant has two ore-crushers of 10 tonnes per hour. The fines < 10 mm may be sent to the steelworks (arc furnace).

6.4. Preparation

No other preparation envisaged

7. Additions

Limestone is added to the charge to fix the sulphur contained in the coal.

8. Temperature zones in reduction
in °C in the reduction apparatus

The temperature of the reduction zone is held between 1050 and 1100°C

9. Pressure zones in reduction
in kg/cm² in the reduction apparatus

Atmospheric pressure

10. Product
of the reduction apparatus

10.1. Form

Sponge iron

10.2. Analysis

Analyses of sponge iron produced from different ores

Ore	Analysis of sponge iron (mean)		
	% metallic Fe	% Total Fe	% S
Soria	85.0	92.2	0.022
Bilbao	78.1	84.6	0.024
Cerain	81.9	91.2	0.097
Rif	87.1	93.3	0.032
Swedish Pellets	85.1	91.8	0.019

10.3. Degree of reduction

90-93% metallisation

10.4. Utilisation of by-products

The gas produced during reduction is partially burned in the pre-heat zone. The excess solid reductant is separated from the metal product and recycled.

11. Further treatment of the product (10) of the reduction apparatus

The products discharged from the retort are screened to the following meshes:

0 - 15 mm

15 - 30 mm

> 30 mm

The three fractions are then passed over magnetic drums where the metallic and non-magnetic phases are separated. This latter is then washed, to separate out the excess reductant from the ash, lime and gangue. The solid reductant thus recovered is recycled.

12. Final product obtained after treatment according to 11

See 10.

13. Iron yield

The total yield of iron may reach 99% in a product processed as under 11.

14. Utilisation of final product

Destination of final product (blast furnace, electric or other furnace). Technical and economic differences as compared with using pig iron and scrap

The final sponge iron may be charged in an open-hearth or arc furnace as 50% of the charge, or to the hot blast cupola as 75%.

15. Consumption data per tonne of final product (12)

15.1. Raw materials for reduction (5.1)

1570 kg iron ore with 60% Fe

50 kg limestone

15.2. Fuels and thermal values required for the entire process
(fuel characteristics) (thermal balance)

For the reduction: 320 kg anthracite with 62.5% fixed carbon.

For gas heating of retort: 1000 kg producer-gas coal (coal with 52.6% fixed carbon, 37.0% volatile matter 8.5% ash and 1.25% S)

15.3. Electrical energy

(for motive power) 110 kWh

15.4. Steam used

No details

15.5. Oxygen

None

15.6. Cooling water

1 m³

15.7. Refractories

No details

15.8. Man hours of productive labour

The total labour is estimated as 4h per tonne of product:
it is made up as follows:

1 engineer,

1 foreman,

30 non-specialised operatives

Each team is composed of 7 men as follows:

- transport of raw materials: 1 man

- charging the retorts: 2 men

- monitoring heating: 2 men

- discharge and transport of sponge iron: 1 man

- gas generator: 1 man

15.9. Labour and materials for routine maintenance

No details

16. Overall capital investment

No details

17. Opinions expressed on the process in relation to:

17.1. State of development

The process appears to have been abandoned. It has not been mentioned in the specialist literature for a long time.

Two other processes, very similar to the Echeverria process and arising directly from it, have recently appeared: the NP (Nuevo Process) and Kinglor-Metor processes.

17.2. Practical operating difficulties

The life of the alloy steel walls of the retort does not exceed 15-18 months. Silicon carbide walls were being investigated with a view to reducing the capital expenditure and maintenance costs. It must be recognised that the process, in its present stage, is not one of the most economic from the thermal point of view. Nevertheless its use at Legazpia seemed to be justified.

17.3. Possibilities of exploitation

The great drawback of this process is the low productivity per retort and the size of the labour force. Therefore it is only suitable for specialised applications (limited tonnage) and places where local labour is cheap.

17.4. Special requirements to be met by certain materials (e.g. refractories)

Only haematites and limonites have been processed successfully. Charging of magnetites has not given encouraging results, very probably because of their lower reducibility.

The melting point of the ores is another criterion for use: if it is too low, there is the risk of formation of rings and clusters on the walls of the retort.

18. Bibliography

18.1. Articles and publications

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- 3) Literature from the promoters (S.A. Siderox, Geneva).
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18.2. Patents

Not quoted

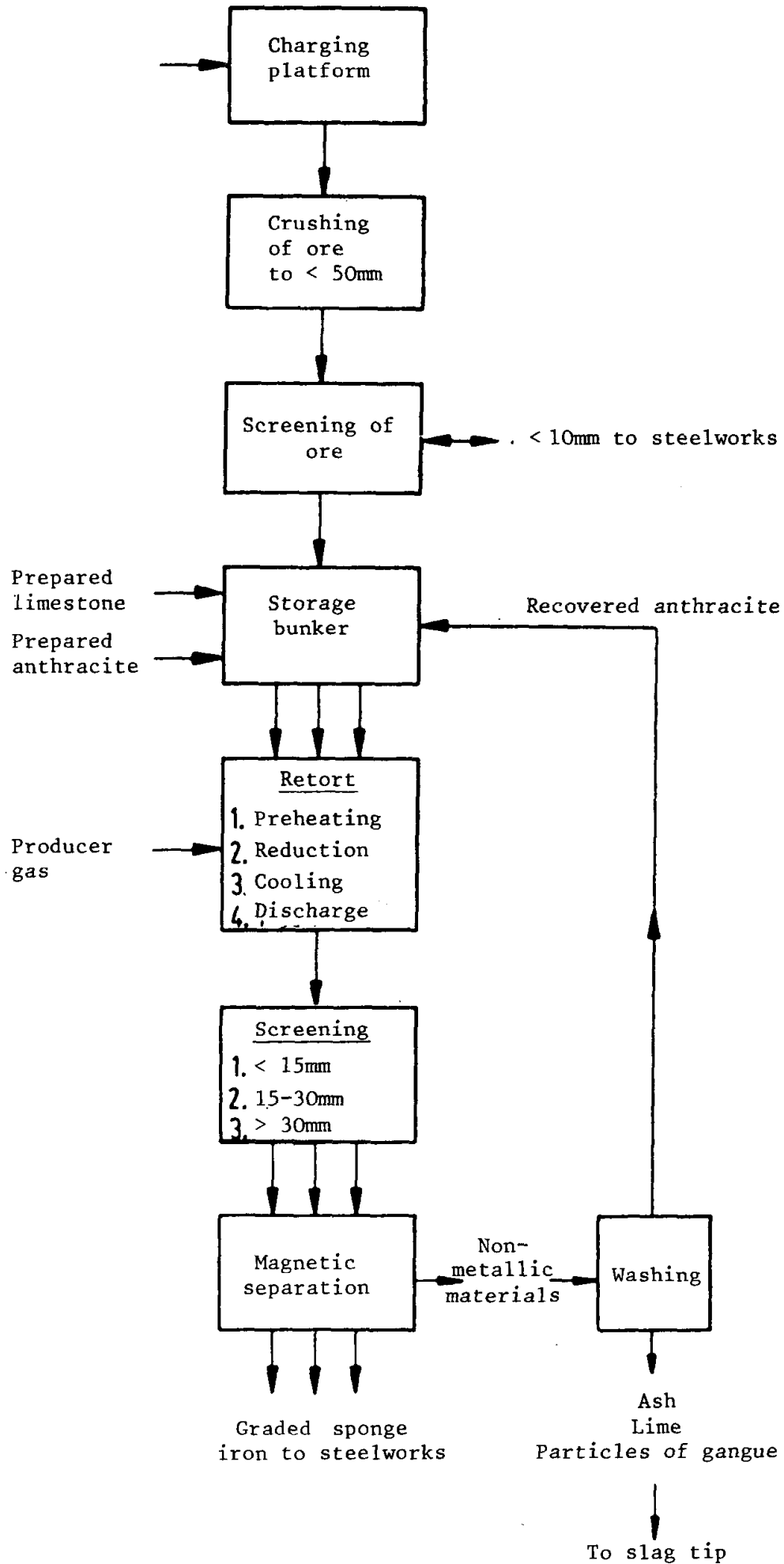


Fig. 1. Echeverria Process

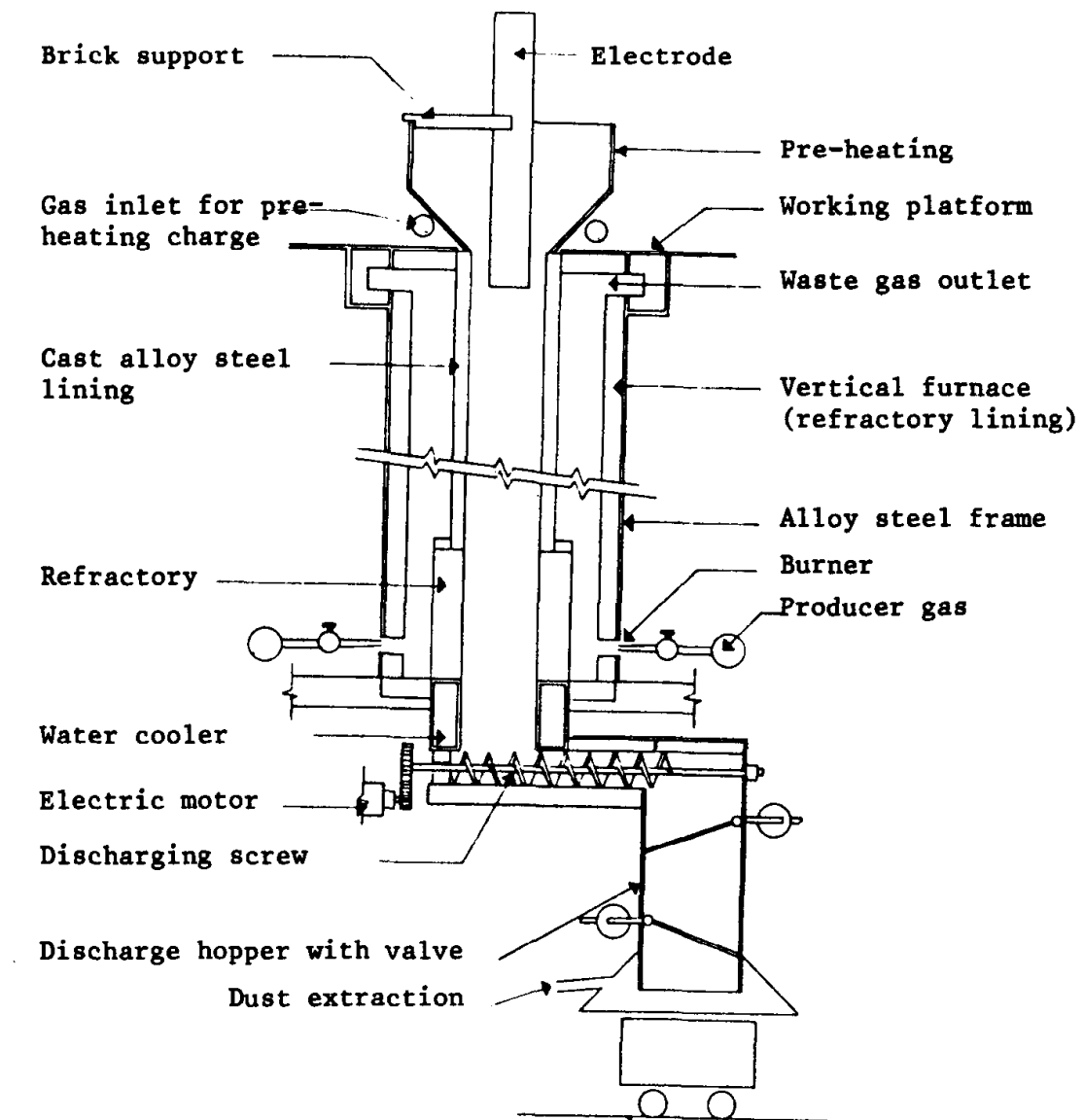


Fig. 2. Echeverria Process

ELKEM PROCESS (with rotary furnace)

1. Characteristics

1.1. Name and history of process

Elkem process (with rotary furnace)

1.2. Principle of process

- a) Heating, drying and/or decarbonatisation and pre-reduction of metal oxides in a rotary furnace.
- b) Transfer of the hot charge to an electric smelting furnace where the melting and final reduction take place.

1.3. Inventors

Fredrik Christen Collin, patent filed in Norway 27th June 1951 by Elektrokemisk A/S, since 1972 become Elkem Spiegeverket A/S.

1.4. Companies involved in development

The process has been developed by Elektrokemisk A.S. (Elkemkuset, Postboks 5430, Majorstna, Oslo 3, Norway) in its laboratories (since 1953) and in pilot plant at Fiskaa, Christiansand, Norway. The process is used in Japan by Nisso Steel Mfg. Co.Ltd. (first tests in 1963) and in Yugoslavia by Rudnici I Zelezara, Skopje, where there are production units, and at Witbank, South Africa, by Highveld Steel and Vanadium Co., where there are 5 production units.

The process is also used for producing ferroalloys. At present 20 units for FeNi and two for FeMn are operational.

2. General description (schematic drawing)

Diagram attached

3. Plant required

Tysland-Hole type electric smelting furnace together with a gas or oil-fired rotary furnace. Equipment for transferring the hot charge from the rotary furnace to the electric furnace and for transferring gas from the electric furnace to the rotary furnace. (see diagram).

4. State of development

4.1. Name of laboratories where the process is situated

See 1.4.

4.2. Pilot plant

4.2.1. Location

Fiskaa Verk, Kristiansand, Norway. (This works comprises one shop for producing ferroalloys, one for making electrode paste and the pilot plant).

4.2.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

One rotary furnace 0.65 m in diameter and 8.75 m long. One electric arc furnace, monophasic, 500 kVA, central electrode 350 mm in diameter. The mean power during smelting is approximately 400 kW. In addition, an electric furnace, monophasic, of a slightly smaller type fitted with a central electrode 200 mm in diameter and in contact with the hearth.

The mean power during operation is approximately 200 kW. Both furnaces are stationary and built like the commercial Elkem furnaces, based on dimensional analysis and the theory of models. Thus they have almost the same thermal yield as the industrial furnaces.

The hot, pre-reduced materials leaving the rotary furnace are weighed and fed into a hopper which is connected to the electric furnace roof by a chute. The solid materials flowing in this chute meet the gases leaving the electric furnace.

4.3. Industrial installations

4.3.1. Location

- a) Semi-industrial installation of Nisso Steel Mfg. Co.Ltd., (Ote-machi Building, Chiyoda-ku, Tokyo, Japan) in its Toyama works.
- b) Industrial plant of Rudnici 1 Zelezara, Skopje, Yugoslavia.
- c) Industrial plant of Highveld Steel and Vanadium Co., Witbank, South Africa.

4.3.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

- a) 1 rotary furnace of 1.3 m internal diameter and 25.4 m long, tilted 2.5%.
1 electric arc furnace. 3 phase, 3 x 400 kVA.
- b) 5 rotary furnaces approximately 100 m long. At present at a standstill.
5 electric arc furnaces. 3 of 34.500 kVA and 2 of 45.000 kVA.
- c) 5 rotary furnaces approximately 60 m long and 4 m in diameter.
5 electric arc furnaces. 4 of 30.000 kVA and 1 45.000 kVA.

4.3.3. Capacity in tonnes per 24 h of final product - per unit of production

- a) 15 t.p.d. in tests
- b) Approximately 200 tonnes iron per day per unit
- c) Approximately 300 tonnes iron per day per unit

4.3.4. Method of operation (continuous or batch, number of operating hours per day)

Continuous operation 24 hours a day.

4.3.5. Coefficient of utilisation in time (number of working days per annum)

Possible to operate 365 days a year.

4.3.6. Maximum rated production

Adding a rotary furnace for pre-reduction and a normal electric melting furnace may enable the production to be increased by 30-50%.

5. Reducing agents

5.1. Raw materials on which the reducing agent is based

Coke dust, semi-coke, coal fines, lignites.

5.2. Process for conversion of above raw materials into a form for use in reduction

The raw materials are used without preparation

5.3. Condition and analysis of reducing agent at moment of use

Solid state, fines (3-20 mm)

	Fixed carbon	Volatiles	Ash
Coke or dust	80 - 90%	1 - 14%	7 - 14%
Coal	50 - 75%	20 - 40%	5 - 10%

5.4. Preheat temperature on entering reduction apparatus

The materials leave the rotary furnace at approximately 1000°C.

6. Ores and ferruginous materials

6.1. Source

Ores and coals from all sources have been treated in the Fiskaa pilot plant; the tests by Nisso in Japan have been concerned with pelletised concentrates of titaniferous sands; the Yugoslav ores are chamosites, limonitised to varying degrees.

6.2. Analysis

Ores or sinter containing 50-65% iron

Analysis of pellets treated in Japan

Fe	SiO	Al ₂ O ₃	TiO ₂	CaO
54.5%	3.6%	2.3%	12.5%	0.5%

6.3. Size

Pellets, sinter or lump ores of < 35 mm

6.4. Preparation

Pelletising or sintering of the concentrates, crushing to < 35 mm of raw ores

7. Additions

7.1. Material

Additions necessary to obtain a fusible and desulphurising slag, for example limestone, silica.

8. Temperature zones in reduction
in °C in the reduction apparatus

Pre-reduction in the rotary furnace at 1000-1050°C

Reduction in the electric furnace at approximately 1600°C

9. Pressure zones in reduction
in kg/cm² in the reduction apparatus

Atmospheric pressure

10. Product
of the reduction apparatus

10.1. Form

See 12.

10.2. Analysis

See 12.

10.3. Degree of reduction

In rotary furnace 50-70%

10.4. Utilisation of by-products

a) The gas from the electric furnace may constitute a by-product if it is not used in the rotary furnace to remove 10-30% oxygen from the oxides.

Output per tonne iron	CO ₂	Composition		N ₂
		CO	H ₂	
600 - 700 Nm ³	15 - 20%	65 - 75%	5 - 10%	2 - 4%

Applications: heating, synthesis of ammonia, nitrogenous fertilisers, etc.

b) Slag from the electric furnace.

Amount and composition similar to the slag which would be produced in the blast furnace using similar ores.

11. Further treatment of the product (10) of the reduction apparatus

No details

12. Final product obtained after treatment according to 11

12.1. Form

Liquid iron

12.2. Analysis

C: 2.5 - 4%, Si: 1 - 2%

13. Iron yield

13.1. Iron contained in product at (10) as a percentage of iron in the charge

No details

13.2. Iron contained in final product (12) as a percentage of iron in the charge.

Approximately 95%

14. Utilisation of final product

Destination of final product (blast furnace, electric or other furnace). Technical and economic differences compared with the use of pig iron and scrap

Used as pig iron

15. Consumption data per tonne of final product (12)

Balance sheets related to 1 t pig iron (Ref.1, 7) in Table 1.

16. Overall capital investment

No details

17. Opinions expressed on the process in relation to:

17.1. State of development

The process is being developed on an industrial scale.

17.2. Practical operating difficulties

As with all the processes treating iron ores in the rotary furnace the pre-reduction temperature limits are narrow.

17.3. Possibilities of exploitation

It may be used for a wide range of ores, particularly for the selective reduction of iron in manganese or titanium bearing ores. This is not possible in a blast furnace.

17.4. Special requirements to be met by certain materials (e.g. refractories)

None

18. Bibliography

18.1. Articles and publications

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- 2) Wright, F.C., Metal Progress, October 1957, p.99-104.
- 3) Collin, F.C., IV. Congr s international d' lectrothermie, Stresa, 25-29 May 1959, n  208.
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- 6) Collin, F.C., Steel Times, 5 March 1965, p.312-315.
Summary of paper presented to Sixteenth Annual Technical Meeting and International Symposium of the Indian Institute of Metals, Calcutta, February 1963 and United Nations Inter-regional Symposium, Prague, November 1963.
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- 9) Andersen, H.C., Shretting, H., and Svana, F.
Proceedings of the ISI Conference ALTERNATIVE ROUTES TO STEEL London, May 1971, p.68-75.
- 10) Andersen, H.C., International Symposium on the Direct Reduction of Iron Ore, Bucarest, Roumania, Sept., 1972.

18.2. Patents

- 1) Norwegian Patent of 27.6.51; French Patent No.1 058 716; US Patent No. 2 694 097.
- 2) Norwegian Patent of 23.4.58; French Patent No.1 222 243.
- 3) Norwegian Patent of 2.5.59 No.131 338; French Patent No.1 253 560; British Patent No.926 545

- 4) Norwegian Patent of 4.10.60; Belgium Patent No.608 749.
- 5) Norwegian Patent of 24.2.61 No.139 223; French Patent No.1 253 560 , addition 81 216; US Patent No.3 197 303.
- 6) Norwegian Patent of 24.2.61 No.139 224; French Patent No.1 315 646.

Initial ore		Sinter	Sinter	Lime	Pellets	Estimate for industrial installation
Fe	%	61.8	58.9		54.5	
SiO ₂	%	3.3	3.1		3.6	
TiO ₂	%	2.5	2.2		12.5	
Al ₂ O ₃	%	3.4	2.4		2.3	
CaO	%	0.6	5.9		6.5	
MgO	%	1.0	1.2		-	
Amount	kg				1930	1830
Limestone	kg				430	385
Silica	kg				155	155
Coal	kg			1/3	435	435
Coke (dust)	kg			2/3	135	135
i.e. fixed C	kg	266	334	352	363*)	363
Slag	kg	320	250	250	800	800
Electrodes	kg	11	8	11	3 - 4	3 - 4
Electricity	kWh	1633	1625	1600	1015	950
Instead of	kWh (1)	2470	2115		2300	2300
Man-hours for production:						1 - 1.5 h/t pig

*) Additional heat for pre-heating the charge is obtained during pilot tests by burning fuel oil; gas from the electric furnace completely (or almost) replaces it on an industrial scale.

H-D PROCESS

1. Characteristics

1.1. Name and history of process

H-D process, after the initials of the inventor (Hellbruegge) and the works where it was developed, Dedini S.A.

1.2. Principle of the process

The H-D process operates with a closed gas circuit and a shaft furnace for reduction, with regenerators in which the required amount of off gas is regenerated and superheated. The regeneration of the off gas can be achieved with liquid or gaseous fuels in the regenerators. Between the reduction furnace and the regenerators is an electrically heated carburettor which reduces the residual CO_2 or H_2O vapour content from the regenerators and so cools the superheated gas, thus regulating the temperature.

When solid fuels are to be used, they are gasified in simple gas generators connected parallel with the gas circuit, with a ratio of O_2 and/or off gas to water vapour such that the gas generator does not freeze.

Since O_2 is brought into the system from outside the purging of the excess CO_2 can be economic.

In every case the excess off gas not required for regeneration is used for heating the regenerators.

1.3. Inventors

Heinrich Hellbruegge

1.4. Companies involved in development

Siderurgica Dedini S.A., Piracicaba (Brazil)

2. General description (schematic drawing)

See drawing

3. Plant required

Shaft reduction furnace (electric furnace if synthetic pig iron is to be produced), scrubber for off gas, gas blower, regenerators, electric carburettor (where appropriate hot lime desulphurisation)

4. State of development

4.1. Laboratory development

Gasification tests on heavy fuel oil using CO_2 were carried out in the laboratory.

4.2. Pilot plant

Laboratory plant for 1000 kg Fe/h has been built.

4.2.1. Location

Piracicaba, Sao Paulo (Brazil)

4.2.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

Internal diameter of the reduction furnace 1.2 m

Internal diameter of the carburettor 1.2 m

4.2.3. Capacity in tonnes per 24 hr of final product

1000 kg Fe/h

4.3. Industrial installations

4.3.1. Location

Piracicaba, Sao Paulo (Brazil)

4.3.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

Diameter of reduction furnace 6 m

4.3.3. Capacity in tonnes per 24 hr of final product - per unit of production

700 t Fe/d

4.3.4. Method of operation

Fuel oil is used in the regenerator and charcoal or coke in the carburettor

4.3.5. Coefficient of utilisation in time (number of working days per annum)

Commissioning planned for end of 1977

4.3.6. Maximum rated production

Reduction furnace: 1400 t/d (ancillary equipment so far only provided for 700 t/d)

5. Reducing agents

5.1. Raw materials on which the reducing agent is based

Fuel oil for the regenerators and bituminous coal or charcoal for the carburettor

- 5.2. Process for conversion of above raw materials into a form for use in reduction

Oil preheating

- 5.3. Condition and analysis of reducing agent at moment of use

Not known

- 5.4. Preheat temperature on entering reduction apparatus

Oil 180°C

- 5.5. Gas velocity in reduction apparatus

0.8 m/s, related to empty space and volume in normal state

- 5.6. Analysis and temperature of gas after passing through reduction apparatus

Expected analysis: approximately 30 % CO, 30 % H₂, 20 % CO₂,
20 % H₂O

Expected temperature: approximately 400°C

6. Ores and ferruginous materials

- 6.1. Source

Minas Gerais, Brazil, Mutuca ore

- 6.2. Analysis

66 % Fe

- 6.3. Size

12 - 30 mm

- 6.4. Preparation

Screened and washed

7. Additions
None
8. Temperature zones in reduction
 in °C in the reduction apparatus
On entry 900°C
9. Pressure zones in reduction
 in kg/cm² in the reduction apparatus
Not known
10. Product
 of the reduction apparatus
No data possible yet
11. Further treatment of the product (10) of the reduction apparatus
- 11.1. Method of cooling
In water bath
- 11.2. Crushing, compacting or briquetting
Not known
- 11.3. Details of final size grading
Not known
- 11.4. Method of concentration (e.g. magnetic separation or sintering)
None

12. Final product obtained after treatment according to 11
Continuous melting down to synthetic pig iron.
13. Iron yield
Practically 100% (without taking into account the dust losses in the shaft furnace and losses to the slag during melting down)
14. Utilisation of final product
Destination of final product (blast furnace, electric or other furnace. Technical and economic differences compared with the use of pig iron and scrap)
As for normal iron for steelmaking
15. Consumption data per tonne of final product (12)
 - 15.1. Raw materials for reduction (5.1)
Approximately 50 kg solid carbon from charcoal or coke
Approximately 230 kg fuel oil
 - 15.2. Fuels and thermal values required for the entire process (fuel characteristics) (thermal balance)
2.60 Gcal/t Fe
 - 15.3. Electrical energy
 - 15.3.1. For heating
110 kWh
 - 15.3.2. For power
80 kWh

15.4. Steam used

15.4.1. For reducing agent

320 kg steam/t Fe (produced from waste heat)

15.4.2. As energy

None

15.5. Oxygen

Only necessary if solid fuels are gasified in the gas generator

15.6. Cooling water

Not known

15.7. Refractories

Not known

15.8. Man hours for productive labour

Not known

15.9. Labour and materials for routine maintenance

Not known

16. Overall capital investment

Not known

17. Opinions expressed on the process in relation to:

17.1. State of development

The laboratory installation for 1000 kg Fe/h has been built

17.2. Practical operating difficulties

No data possible yet

17.3. Possibilities of exploitation

The process can use a variety of fuels. If the regenerators are once provided for example for fuel oil, all other liquid or gaseous fuels can be used without modification.

When solid fuels are used the gas generator grates have to be suited to the fuels.

17.4. Special requirements to be met by certain materials
(e.g. refractories)

Not known

18. Bibliography

18.1. Articles and publications

1) Direct reduction of iron ore: H.D. process.

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(ILAFA). Congress in Porto Alegre, Brazil, May 1975.

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18.2. Patents

Applied for: 1 process patent

1 design patent for the electric
carburettor

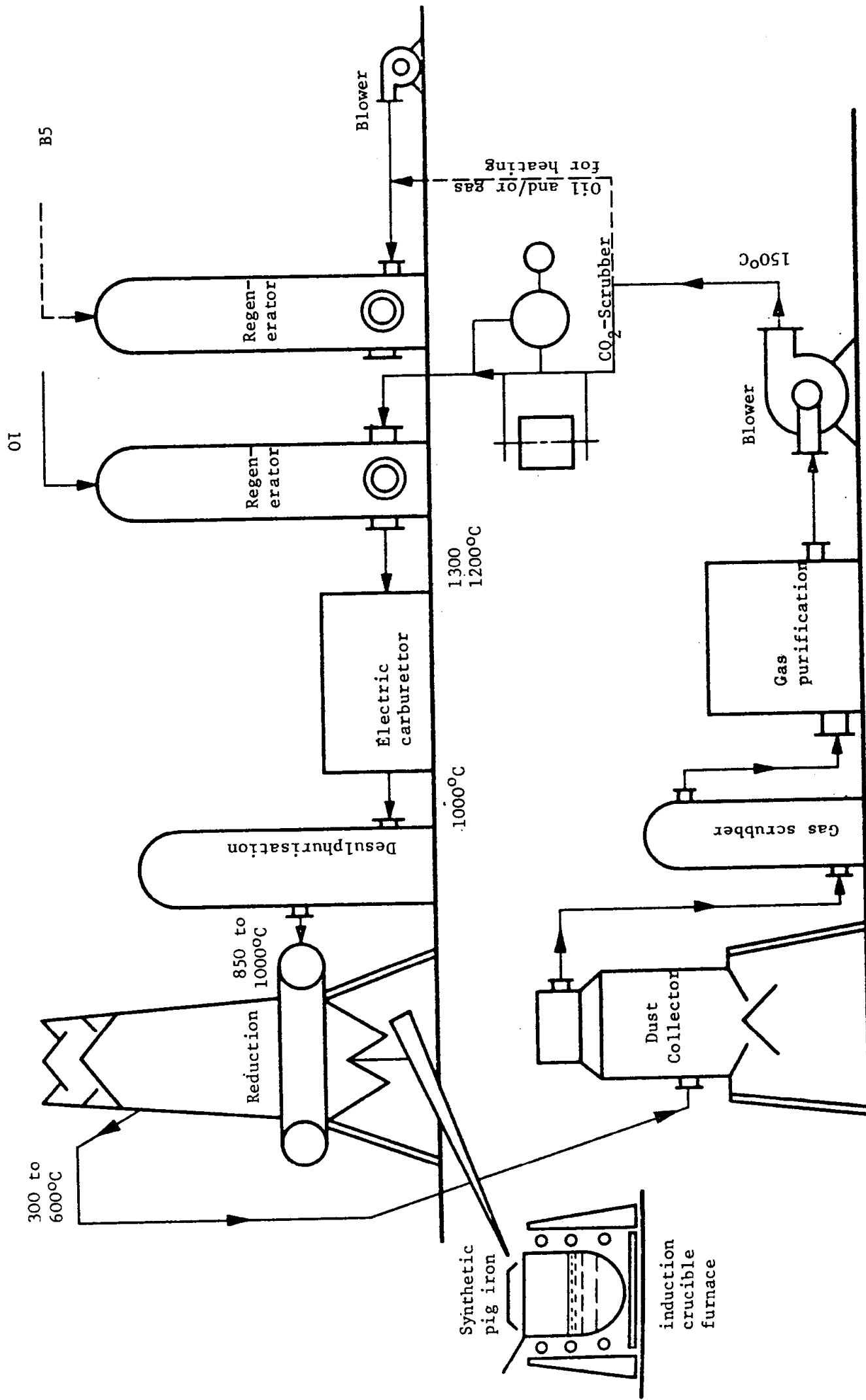


Fig.1. Diagram of the HD Process.

HÜGANÄS PROCESS

1. Characteristics

1.1. Name and history of process

HÜganäs process, invented in 1908

1.2. Principle of process

A concentrate with a high iron content ($> 70 \% \text{ Fe}$) is charged with coal or coke dust and limestone into ceramic muffles; the materials are arranged in alternate layers or concentric rings. The muffles are heated to approximately 1200°C in a tunnel kiln and cooled, at the end of the reaction, away from air. The cooled sponge iron is crushed and freed from the excess fuel and lime adhering. The gas which is liberated during reduction is used directly for heating the muffles. The final sponge iron, of very high purity, is used in powder metallurgy or as high grade scrap for producing special steels.

1.3. Inventors

E Sieurin (1908)

1.4. Companies involved in development

Process developed at HÜganäs AB, Sweden

2. General description (schematic drawing)

See working diagram of tunnel kiln

3. Plant required

Tunnel kiln, muffle cars, installation for separating the sponge iron.

4. State of development

4.1. Name of laboratories where the process is situated

Höganäs AB. Sweden

4.2. Pilot plant

No details

4.3. Industrial installations

4.3.1. Location

1. Höganäs, Sweden
2. Oxelösund, Sweden
3. Riverton, N.J., USA

4.3.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

1. Tunnel kilns, 270 m long, commissioned 1963 - 4
2. Tunnel kiln, no data, commissioned 1954
3. Tunnel kilns, 183 m long commissioned 1954 - 60

4.3.3. Capacity in tonnes per 24 h of final product - per unit of production

1. Production 165,000 tpa in 3 kilns
2. Production 35,000 tpa in 1 kiln
3. Production 67,000 tpa in 2 kilns

- 4.3.4. Method of operation (continuous or batch, number of operating hours per day)

Retort process with continuous operation

- 4.3.5. Coefficient of utilisation in time (number of working days per annum)

No details

- 4.3.6. Maximum rated production

No details

5. Reducing agents

- 5.1. Raw materials on which the reducing agent is based

Coal or coke dust

- 5.2. Process for conversion of above raw materials into a form for use in reduction

No pre-treatment

- 5.3. Condition and analysis of reducing agent at moment of use

Fines, mixed with limestone, no other data.

- 5.4. Preheat temperature on entering reduction apparatus

None

- 5.5. Gas velocity in reduction apparatus

No details

- 5.6. Analysis and temperature of gas after passing through reduction apparatus

No details

6. Ores and ferruginous materials

6.1. Source

Concentrates with high Fe content

6.2. Analysis

Fe content > 70 %

6.3. Size

Fines, no other data

6.4. Preparation

No special preparation

7. Additions

Fine limestone (120 kg per t) to combine with the sulphur
in the fuels

8. Temperature zones in reduction
in °C in the reduction apparatus

Approximately 1200°C

9. Pressure zones in reduction
in kg/cm² in the reduction apparatus

Atmospheric

10. Product
of the reduction apparatus

10.1. Form

Porous cakes or tubes of sponge iron (porosity 70%)

10.2. Analysis

	Fe _{tot}	Fe _{met}	P	S	C
%	98	96	< 0.015	< 0.015	0.1 - 0.3

10.3. Degree of reduction

Degree of metallisation 98%

10.4. Utilisation of by-products

Recovery of excess fuel

11. Further treatment of the product (10) of the reduction apparatus

The cake of sponge iron freed from the excess lime and fuel is crushed to the desired size.

12. Final product obtained after treatment according to 11

See 10

13. Iron yield

No details

14. Utilisation of final product

Destination of final product (blast furnace, electric or other furnace). Technical and economic differences as compared with using pig iron and scrap

Raw material for producing iron powder.

High quality scrap for making special steels

15. Consumption data per tonne of final product (12)

15.1. Raw materials for reduction (5.1)

No details

15.2. Fuels and thermal values required for the entire process (fuel characteristics) (thermal balance)

Approximately 4 Gcal/t of which 3 Gcal in the form of coke and 1 Gcal in the form of fuel oil for heating.

15.3. Electrical energy

100 - 200 kWh/t

15.4. Steam used

None

15.5. Oxygen

None

15.6. Cooling water

No details

15.7. Refractories

No details

15.8. Man hours of productive labour

No details

15.9. Labour and materials for routine maintenance

No details

16. Overall capital investment

No recent data

17. Opinions expressed on the process

Capital investment and conversion costs probably high per tonne of final product. On the other hand the process is proven, with few risks of breakdowns; the electricity consumption is moderate.

18. Bibliography

18.1. Articles and publications

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- 4) E. Barrett, Bulletin 519, Bureau of Mines.
- 5) M. Tigerschild, Stahl und Eisen 70 (1950), pp.401-403.
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- 16) Cavanagh, P.E., Obtaining sponge iron in a tunnel furnace, Brit. Clayworker 60 (1951-1952), pp.238, 274-278, 310-314 and 345-346; 61 (1952-1953), pp.30-35.
- 17) Høganäs A/B, Høganäs Eisenschwammpulver. Helsingborg, 1953.
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- 27) Scortecci, A., and A. Palazzi: J.Iron Steel Inst.
195 (1960) No. 3, pp.267-278.

18.2. Patents

Germany	249 031 (1960)
	478 563 (1927)
USA	1 065 890 (1910)
	174 280 (1927)
Austria	54 245 (1911)
France	599 185 (1925)
UK	252 899 (1925)
	296 235 (1927)

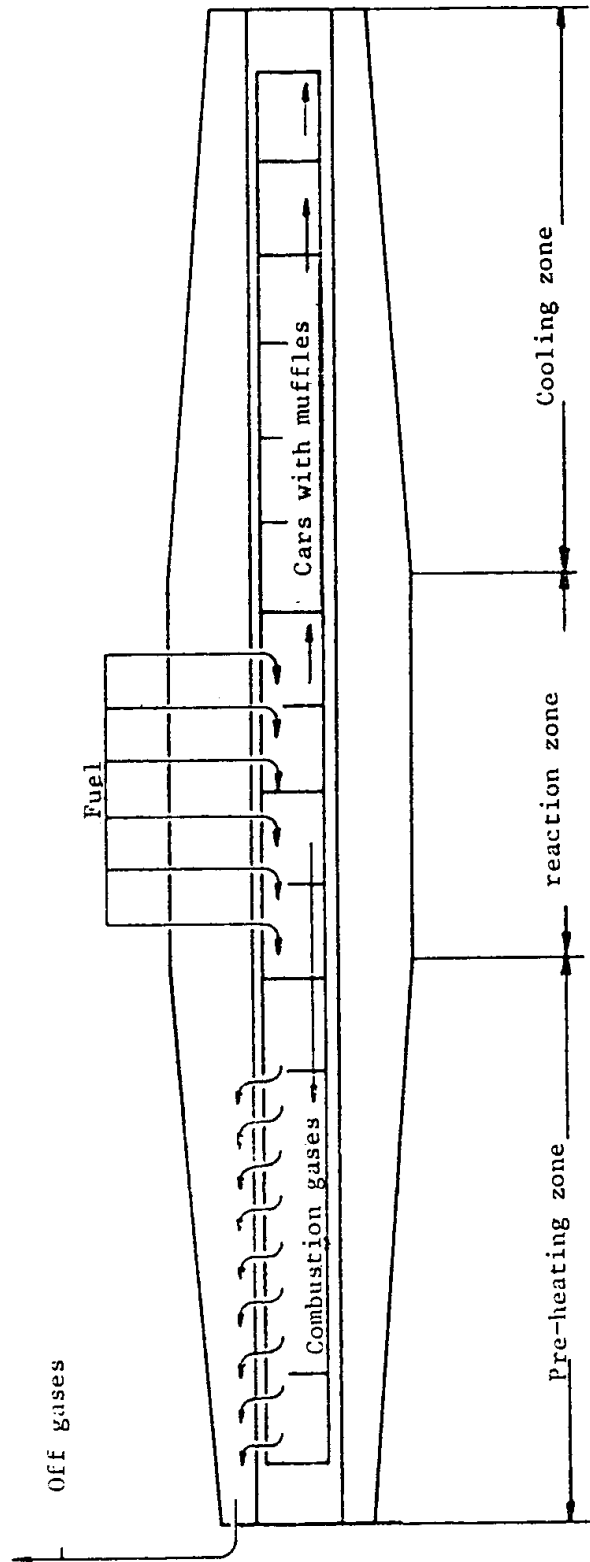


Fig.1. Hoganas process - Working diagram of a tunnel kiln.

I.C.E.M. PROCESS

1. Characteristics

1.1. Name and history of process

I.C.E.M. Process

1.2. Principle of process

Reduction of iron ore as lump or pellets in a vertical shaft furnace using a reducing gas with a high CO and H₂ content. The gaseous reductant is produced by recycling and regenerating a large proportion of the gas in the reduction process.

The regeneration of the gas occurs in two furnaces operating alternately, of Cowper type, in accordance with the reactions of reforming methane with the carbon dioxide and steam present in the recycled gas, the reducing gas being heated at the same time.

1.3. Inventors

Iosif Tripsa, Ovidiu Hatarascu and Gheorghe Atanasiu.

1.4. Companies involved in development

I.C.E.M. (Institute for Metallurgical Research) Bucurest.

2. General description (schematic drawing)

Drawing attached (Figures 1 and 2).

The shaft furnace for reduction is charged from the top with lump ore, using a skip. The pre-heated reducing gas with a content of CO + H₂ > 85% is fed into the furnace from the bottom so that it circulates in counterflow to the descending ore.

Most of the gas drawn off at the top of the furnace is fed back in during the reduction process using a blower.

To this end the gas withdrawn from the furnace first passes through dry and wet cleaners to remove the dust and some of the steam. It is then regenerated and re-heated in two vertical regenerating furnaces of cylindrical shape which operate alternately, one period for the re-heating and another for heating the furnace.

The furnaces for regeneration and pre-heating the gas are filled with refractory granules and have in the centre a layer of catalysts. In these furnaces the gas is fed in after being mixed with a certain amount of methane added according to the amount of CO_2 and H_2O contained in the gas after use for reduction and wet cleaning. The gas which circulates downwards is pre-heated and the CO and H_2O content increases by a reaction, in a catalytic medium, between the methane introduced and the CO_2 and H_2O contained in the gas.

In order to obtain the heat necessary to regenerate the recycled gases and pre-heat the reducing gases, each furnace operates alternately, that is for one period re-heating using a gaseous fuel (methane gas, blast furnace gas, etc.).

Each of the reformers has at the bottom an oxygen-methane burner which can be used to supply extra heat to the reducing gas.

3-5% coke, in lumps, is charged to the reducing furnace with the iron ore, to increase the permeability to the gas of the column of material in the furnace to prevent sticking. The coke can be recovered by magnetic separation after reduction and either reused or charged with sponge iron to a blast furnace or electric furnace.

The sponge iron produced is used to produce pig iron in a blast furnace and to produce steel in an electric furnace.

3. Plant required

Shaft reduction furnace.

Wet and dry cleaners (dust extractors)

Furnaces for regeneration and pre-heating of the gases.

Ancillary equipment (fans, pump, burners, etc.)

4. State of development

4.1. Name of laboratories where the process is situated

Direct reduction laboratory of I.C.E.M. (Institute for Metallurgical Research), Bucarest.

4.2. Pilot plant

4.2.1. Location

Pilot plant at I.C.E.M. Bucarest.

4.2.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

Shaft reduction furnace of truncated cone shape wider at the bottom, height 4.8 m, volume 2.2 m^3 .

The furnace for regenerating the recycled gases and pre-heating the reducing gases has an internal height of 4.00 m and a useful volume of 3.90 m^3 .

4.2.3. Capacity in tonnes per 24 h of final product

Production capacity of the pilot plant approximately
2 t.p.d. sponge iron.

4.3. Industrial installations

In 1972 a 25 t.p.d. semi-industrial scale plant for producing sponge iron was commissioned. In 1972 this plant produced about 1400 t sponge iron, which was used in the blast furnace. The degree of reduction fluctuated within very wide limits, between 30 and 90%. After several modifications and improvements, in 1973 sponge iron was produced with a mean degree of metallisation of 94%; it was melted down in arc furnaces.

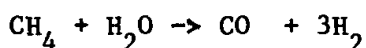
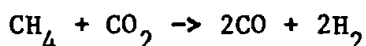
5. Reducing agents

5.1. Raw materials on which the reducing agent is based

Natural gas

5.2. Process for conversion of above raw materials into a form for use in reduction

Catalytic reforming of methane with the CO_2 and H_2O present in the gases leaving the top of the reduction furnace and recycled continuously during the process.



5.3. Condition and analysis of reducing agent at moment of use

The gaseous mixture comprising mainly CO and H_2 of the following composition (dry basis):

CO_2 = 6.5% max

CO = 35-45%

H_2 = 40-50%

CH_4 = 4.5% max

N_2 = 6%

5.4. Preheat temperature on entering reduction apparatus

850°C

5.5. Gas velocity in reduction apparatus

14.4 m/min. or 0.24 m/sec.

5.6. Analysis and temperature of gas after passing through reduction apparatus

The chemical composition of the recycled gases before regeneration is characterised by a high CO₂ content

CO₂ = 20%

CO = 28%

H₂ = 42%

CH₄ = 4%

N₂ = 6%

The temperature of the gases leaving the reduction furnace is approximately 550°C.

•

6. Ores and ferruginous materials

6.1. Source

Hematite

6.2. Analysis

The chemical composition of the ore is as follows:

65.0% Fe

27.8% O₂

7.2% gangue

The gangue is largely SiO₂ and Al₂O₃ in the proportion:

3.3% SiO₂ for 2.5% Al₂O₃

6.3. Size

The size of the ore varies between 0.5 and 100 mm.

6.4. Preparation

Crushing and screening of the lump ore to 15-50 mm. The fine ores are sintered or pelletised.

7. Additions

7.1. Material

Coke, 3-5% of the charge

7.2. Analysis

Chemical composition of a blast furnace coke

7.3. Size

Size between 20 and 40 mm

8. Temperature zones in reduction
in °C in the reduction apparatus

Charging temperature in the lower part of the shaft 770°C and reducing gas temperature 850°C at the inlet.

9. Pressure zones in reduction
in kg/cm² in the reduction apparatus

Reduced pressure (0.075 kg/cm₂).

10. Product
of the reduction apparatus

10.1. Form

Sponge iron

10.2. Analysis

The chemical composition of the sponge iron usually varies between the following limits:

Metallic iron = 73-85%

Total iron = 86-89%

O₂ = 2-4 %

C = 0.5-1.5%

10.3. Metallic iron as a percentage of the total iron content
of the product

85%

11. Further treatment of the product (10) of the reduction apparatus

11.1. Method of cooling

The iron is cooled to about 400°C in the lower part of the reduction furnace.

The sponge iron is withdrawn in metal containers, hermetically sealed, so that the cooling can take place without any risk of oxidation of the iron.

In the semi-industrial plant the sponge iron is cooled to 60°C in a cooling zone below the bustle pipe in the shaft by means of a gas circuit.

11.2. Crushing, compacting or briquetting

Not necessary

11.3. Details of final size grading

85% of the sponge iron is between 15 and 50 mm in size and the rest less than 15 mm.

11.4. Method of concentration (e.g. magnetic separation or sintering)

Separation of the coke where necessary

12. Final product obtained after treatment according to 11

The same form and analysis as for 11

13. Iron yield

13.1. Iron contained in product (10) as a percentage of iron in the charge

99%

13.2. Iron contained in final product (12) as a percentage of iron in the charge

99%

14 Utilisation of final product

Destination of final product (blast furnace, electric or other furnace). Technical and economic differences as compared with using pig iron and scrap

The sponge iron has been used in tests on producing steel in electric furnaces as a replacement for scrap, and has also been used in tests on producing pig iron in a blast furnace with a useful volume of 40 m³, as 35% of the metallic charge. In

comparison with a reference operation, the specific fuel consumption fell to 40% and the productivity of the blast furnace increased up to 20%.

15. Consumption data per tonne of final product (12)

15.1. Raw materials for reduction (5.1)

Natural gas 350 Nm³ per tonne (depending on degree of reduction)

15.2. Fuels and thermal values required for the entire process (fuel characteristics) (thermal balance)

Natural gas 750 Nm³ per tonne as fuel including gas for heating the regenerators (depending on degree of reduction).

The natural gas may be partly replaced by other gaseous fuels (blast furnace gas, coke-oven gas, etc.).

15.3. Electrical energy

15.3.2. For power

70 kWh/tonne for motive power.

15.4. Steam used

No consumption of steam

15.5. Oxygen

Oxygen is consumed only when it is desired to raise the temperature of the gases above that quoted in 5.6.

15.6. Cooling water

5 Nm³ per tonne sponge iron

15.7. Refractories

10 kg per tonne

15.8. Man hours of productive labour

For a plant with a capacity of 100 t.p.d. the direct labour is approximately 2.5 man hours per tonne sponge iron, that is a total daily labour force of 30-32 men.

16. Overall capital investment

No details

17. Opinions expressed on the process in relation to:

17.1. State of development

The process was designed and laboratory tested during 1962. At the same time plans were drawn up for the construction of a pilot plant with a capacity of 2 tonnes of sponge iron per day. Construction of the plant was completed in July, 1963, and the plant was commissioned the same month. In 1972 a 25 t.p.d. semi-industrial scale plant was commissioned. It is intended, when the research work is completed, to build an industrial plant to produce about 200,000 t sponge iron per year.

17.2. Practical operating difficulties

No great difficulties have been encountered during operation of the pilot plant.

17.3. Possibilities of exploitation

The process can be developed on an industrial scale, when the methane consumption would be reduced.

The process is suitable for lump ores and pellets to obtain sponge iron used in iron and steelmaking (blast furnace, steelworks, etc.). The process and the plant can also be adapted to produce iron used in powder metallurgy.

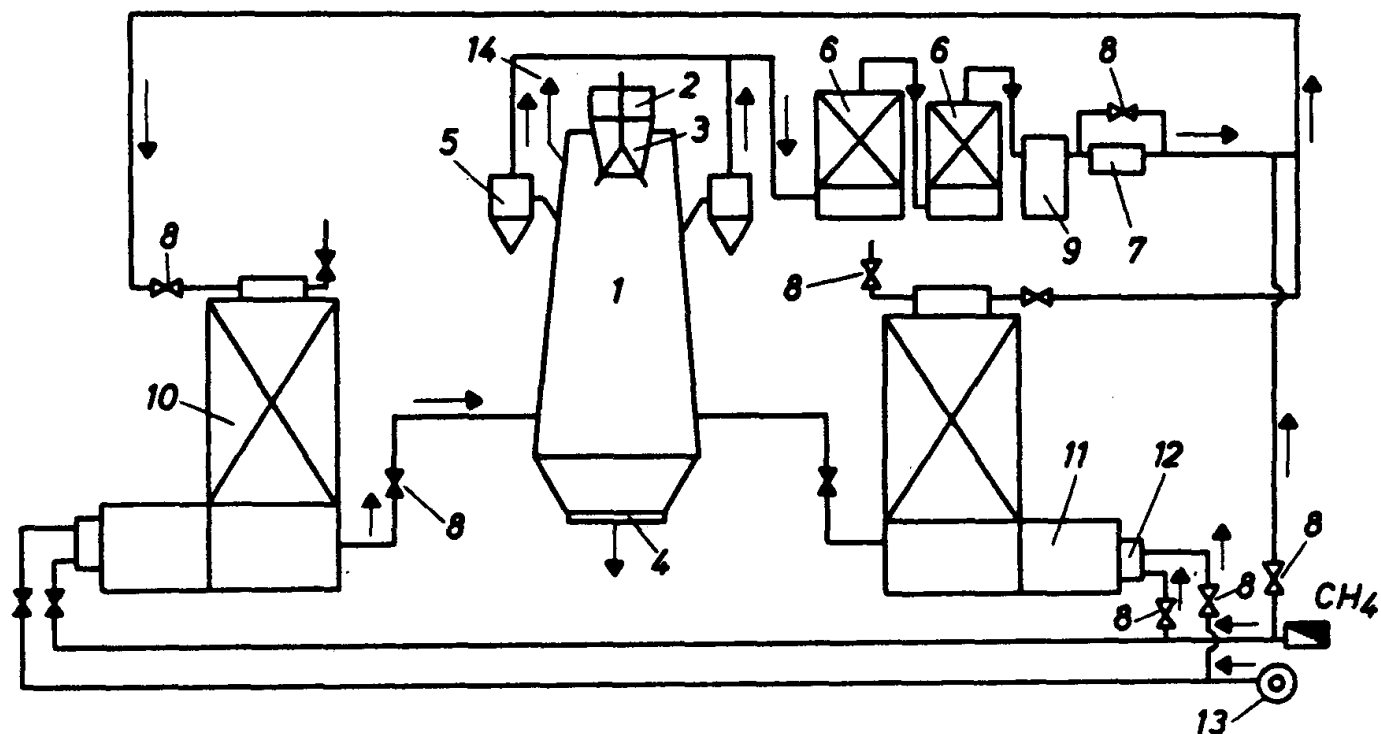
- 17.4. Special requirements to be met by certain materials
(e.g. refractories)

No special requirements

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18.1. Articles and publications

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- 1 Reduction shaft
- 2 Charging system
- 3 Cone
- 4 Withdrawal system for sponge iron
- 5 Cyclone
- 6 Scrubber
- 7 Blower
- 8 Control valves
- 9 Condensers
- 10 Gas reformer
- 11 Combustion chamber
- 12 Burner for methane gas
- 13 Fan
- 14 Gas outlet to atmosphere

Fig. 1. ICEM process

KRUPP - RENN PROCESS

1. Characteristics

1.1. Name and history of process

Krupp-Renn process, developed by the former Fried. Krupp AG, Grusonwerk, between 1932 and 35.

1.2. Principle of process

Process for producing iron granules (compact nodules or "Luppen") from lean to medium fine ores with mainly acidic gangue using fine grained, low grade reductants in a rotary kiln heated by counterflow.

1.3. Inventors

Friedrich Johannsen (original patents).

1.4. Companies involved in development

Fried. Krupp AG

2. General description (schematic drawing)

See diagram

3. Plant required

See diagram of process

4. State of development

4.1. Name of laboratories where the process is situated

Experimental rotary kiln plant, Essen.

4.2. Pilot plant

4.2.1. Location

No further development

4.2.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

Rotary kiln 14 m long, inside diameter of kiln 0.9 m measured inside the brickwork, inside diameter of shell 1.2 m. Tilt between 0.5° and 5° , number of revolutions variable between 0.3 and 2 r.p.m.

4.2.3. Capacity in tonnes per 24 h of final product

About 1 t.p.d. nodules (Luppen), depending on the iron content of the charge.

4.3. Industrial installations

4.3.1. Location

a) Essen - Borbeck (Fried. Krupp)

1 kiln 3.6 m dia. x 50 m, 275 to 300 tonnes of ore processed each day. Large scale experimental plant. Operated from 1935 to 1945 then dismantled.

b) Frankenstein, Silesia (Fried. Krupp)

2 kilns 3.6 m dia. x 50 m, each processing 275 to 300 tonnes of ore containing nickel and iron (garnierite). First kiln built in 1935, second in 1941. Nodules used as charge material for nickel

steels. Both kilns operated up until 1945 and then dismantled by U.S.S.R.; the plant was then rebuilt by the Poles.

c) Salzgitter-Watenstedt, Germany (Fried. Krupp)

3 kilns 4.2 m dia. x 70 m, each processing 500 t.p.d. of Salzgitter crude ore containing 25 to 30% Fe. Built in 1943 to 1944. The nodules containing phosphorus were melted to pig iron in a blast furnace. Two kilns were in operation from 1944 to 1945, the third kiln was still under construction at the end of the war. All three kilns were dismantled in 1950 as reparations.

d) Unterwellenborn, Thuringia, East Germany

2 kilns, each 3.6 m dia. x 60 m, built after 1945, daily throughput of ore about 300 to 400 tonnes per kiln depending on the composition and size. Ores containing 24 to 33% iron and 29 to 26% silica processed.

e) St. Egidien, Saxony, East Germany

2 kilns built after 1945, kiln size 4.2 m dia. x 100 m.

f) Kraluv-Dvur, Czechoslovakia

3 kilns, 3.6 m dia. x 60 m, each processing 300 to 325 t.p.d. brown hematite containing 30 to 35% Fe and a high SiO_2 content. One kiln in operation from 1943, three kilns in use after the war.

g) Minisek, Czechoslovakia

3 Renn kilns (no other information available).

h) Egpovice, Czechoslovakia

10 Renn kilns, built after 1945, 3.6 m dia. x 60 m, (no other information available).

i) Czenstochau, Poland

2 Renn kilns (no other information available).

k) An-shan, formerly Manchuria (Showa Steel Works)

8 kilns, 3.6 m dia. x 60 m, each processing 300 t.p.d. of ore. Quartzitic ore containing 35 to 36% Fe and 47 to 48% SiO_2 . Some of the nodules were fed into the refiner at the steelworks, some were charged into the blast furnaces and others were shipped to Japan. Believed to have been shut down.

l) Seishin, North Korea (Mitsubishi Shogi Kaisha)

4 kilns, 3.6 m dia. x 60 m, each processing 300 t.p.d. of magnetite ore containing 55 to 60% Fe. The nodules were shipped to Japan and charged into open-hearth furnaces or electric furnaces. Believed to have been shut down.

m) Kuji, Japan (Kawasaki Dockyard)

2 kilns, 3.6 m dia. x 60 m, each processing 300 t.p.d. titaniferous iron sands containing 34 to 35% Fe and 5 to 8% TiO_2 .

n) Nihon Yakin KK, Japan

4 kilns, 3.6 m dia. x 70 m, for processing a total of 500 to 600 tonnes of ore containing nickel and iron. The kilns were constructed between 1940 and 1942. After the war, two units processed New Caledonian nickel ore to nickel-containing nodules as charge material for steelworks producing high quality steel.

o) Aviles, Spain

1 kiln, 3.6 m dia. x 60 m, for processing 250 to 275 t.p.d. ore. Iron quartzite containing 30 to 50% Fe and 10 to 30% SiO_2 . Plant has been in operation since Autumn 1954.

p) Salzgitter-Watenstedt, Germany (Rennanlage-Salzgitter-Ruhr GmbH

2 kilns, 4.2 m dia. x 95 m and 2 kilns 4.6 m dia. x 110 m, processing acidic ores from the Harz foothills and intermediate products from wet preparation containing 28 to 34% Fe. Total daily capacity approximately 2,000 tonnes of ore throughput. The first kiln was commissioned in October 1956; the plant ran at full capacity from April 1956 and was shut down in 1963.

r) Essen-Borbeck, Germany (Rennanlage Rhein-Ruhr GmbH)

6 kilns, 4.6 m dia. x 110 m, total ore throughput 4,200 t.p.d. Commissioned 1959/60. Plant shut down in 1963.

No information is available on Renn plants built and operated in the Eastern bloc countries.

4.3.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

See 4.3.1.

4.3.3. Capacity in tonnes per 24 h of final product - per unit of production

See 4.3.1.

4.3.4. Method of operation (continuous or batch, number of operating hours per day)

Continuous - 24 hours per day.

4.3.5. Coefficient of utilisation in time (number of working days per annum)

300 working days per annum.

4.3.6. Maximum rated production

About 350 t.p.d. of nodules in a kiln of 4.6 m dia. x 110 m.

5. Reducing agents

5.1. Raw materials on which the reducing agent is based

Coke dust, anthracite, bituminous or lignite low temperature coke.

5.2. Process for conversion of above raw materials into a form for use in reduction

No special preparation of the reductant

5.3. Condition and analysis of reducing agent at moment of use

Where necessary, the various reductants are crushed to less than 5 mm.

5.4. Preheat temperature on entering reduction apparatus

None

5.5. Gas velocity in reduction apparatus

No details

5.6. Analysis and temperature of gas after passing through reduction apparatus

No details

6. Ores and ferruginous materials

6.1. Source

Iron ores with high SiO_2 contents are suitable.

The Renn process is also suitable for processing oxidic nickel ores (garnierite).

6.2. Analysis

Acid ores containing 30 to 50% Fe.

6.3. Size

The reducibility of the ore depends upon the size and should not exceed 10 mm.

6.4. Preparation

There is not usually any need for special preparation apart from crushing to the desired size.

7. Additions

7.1. Material

7.2. Analysis

Depending on the composition of the ore, limestone, sand or aluminiferous additions are used to obtain CaO to SiO_2 and Al_2O_3 to SiO_2 ratios of between 0.2 and 0.4 in the slag. The total amount of slag should not be less than about 600 kg/t nodules.

7.3. Size

The size of the additions should be matched to that of the ore.

8. Temperature zones in reduction
in °C in the reduction apparatus

The maximum temperature of the material is about 1300°C.

9. Pressure zones in reduction
in kg/cm² in the reduction apparatus

Normal atmospheric pressure

10. Product
of the reduction apparatus

10.1. Form

The material discharged from the kiln is of a doughy consistency and the iron is embedded in the doughy slag in the form of solid nodules.

10.2. Analysis

None. This is because a conglomerate of slag is obtained with iron nodules encased in it.

10.3. Degree of reduction

The degree of reduction in the nodules, calculated from the ratio of the metallic iron to the total iron is 97 to 98%.

10.4. Utilisation of by-products

The off-gas from the rotary kiln contains less than 3% CO and about 19% CO₂. The temperature is between 300 and 600°C. The end slag which is collected during preparation of the material discharged from the kiln has a fine particle size and contains 5% FeO max.

11. Further treatment of the product (10) of the reduction apparatus

11.1. Method of cooling

Direct cooling with water on the plate conveyor or in the drum; pre-comminution of vitreous slag.

11.2. Crushing, compacting or briquetting

Comminution in ball mill with peripheral discharge

11.3. Details of final size grading

Nodules of 1 to 10 mm; a certain amount of larger nodules

11.4. Method of concentration (e.g. magnetic separation or sintering)

The nodules are magnetically separated from the slag

12. Final product obtained after treatment according to 11

12.1. Form

Mainly round granules of iron

12.2. Analysis

94% Fe (sometimes with Ni), 0.8% C.

Sulphur contents depend on the S content of the charge material, e.g. with anthracite having a low S content : 0.2% S, with coke dust : 0.6 to 0.8% S. The sulphur content is not important if the nodules are processed in a blast furnace or in a hot-blast cupola. However, if they are used in a steelworks the proportion of nodules is limited by the S content.

13. Iron yield

13.1. Iron contained in product (10) as a percentage of iron in the charge

No details

- 13.2. Iron contained in final product (12) as a percentage of iron in the charge

Iron yield 92 to 95%

14. Utilisation of final product

Destination of final product (blast furnace, electric or other furnace). Technical and economic differences as compared with using pig iron and scrap

The nodules (Luppen) can be used as a replacement for scrap in blast furnaces and in steelworks. The choice of process for which they are to be used depends mainly on the composition of the raw materials.

15. Consumption data per tonne of final product (12)

- 15.1. Raw materials for reduction (5.1)

No details

- 15.2. Fuels and thermal values required for the entire process (fuel characteristics) (thermal balance)

Total heat requirement 5.5×10^6 kcal/t nodules.

- 15.3. Electrical energy

- 15.3.1. For heating

None

- 15.3.2. For power

75 - 110 kWh/t nodules

- 15.4. Steam used

None

15.5. Oxygen

None

15.6. Cooling water

2 m³/t nodules

15.7. Refractories

9 kg/t nodules

15.8. Man hours of productive labour

1.5 - 3 h/t nodules

15.9. Labour and materials for routine maintenance

Repairs and maintenance approximately 7% of the capital expenditure for machinery and electrical equipment.

16. Overall capital investment

16.1. For the plant described in (3) and related to one tonne of capacity for the final product (12)

No currently valid figures available

16.2. Construction costs of ancillary plant not covered in (3)

No details

17. Opinions expressed on the process in relation to:

17.1. State of development

The process has given satisfactory results, operating on a large scale, over many years.

17.2. Practical operating difficulties

Operating difficulties can be overcome

17.3. Possibilities of exploitation

The importance of the Renn process has declined in the face of growing supplies of rich ores, concentrates and pellets, produced from these concentrates, together with a comparatively low unit iron price on the world market.

17.4. Special requirements to be met by certain materials (e.g. refractories)

The refractories must meet particular requirements as regards quality.

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Documentation de la Sidérurgie, No.10, Rue de Lota, Paris
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18.2. Patents

None

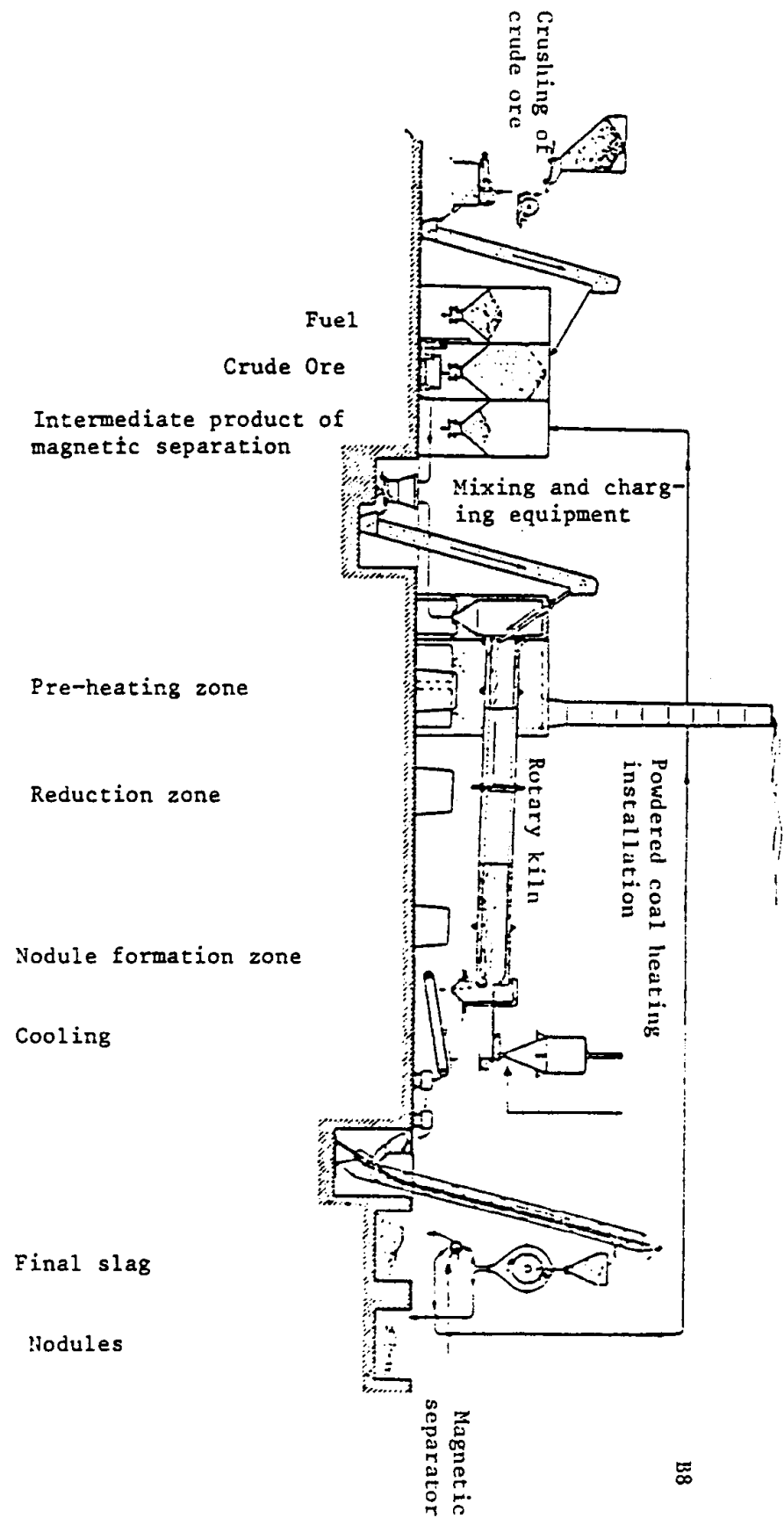


Fig.1. Krupp - Renn Process

NOVALFER DIRECT REDUCTION PROCESS

1. Characteristics

1.1. Name and history of process

Novalfer direct reduction process

1948 : Laboratory studies

1958 : 10 t.p.d. experimental furnace

1968 : 60 t.p.d. plant

1.2. Principle of process

Reduction of iron ores by pure hydrogen or a gas rich in hydrogen in a fluidised bed.

The reduction may take place in one or two stages. The hydrogen is obtained by reforming hydrocarbons with steam.

1.3. Inventors

Guillemonat Patry and O.N.I.A. (Office National Industriel de l'Azote)

1.4. Companies involved in development

A.P.C. (Azote et Produits Chimiques - nitrogen and chemical products) 143, Route d'Espagne, 31053 Toulouse Cedex, France.

2. General description (schematic drawing)

See diagram.

The reduction furnace is charged with powdered ore, pre-dried and pre-heated. It is kept in the fluidised state by a reducing gas rich in hydrogen, obtained by reforming natural gas with a

large excess of steam; the carbon monoxide formed is converted into carbon dioxide by the steam and then removed. The reducing gas is recycled after condensation of the steam. The reduced product is cooled under an inert gas on leaving the reactor.

3. Plant required

Reduction furnace with fluidised bed, pre-heating furnace for the solids, reforming and conversion unit.

4. State of development

4.1. Name of laboratories where the process is situated

No details

4.2. Pilot plant

No details

4.3. Industrial installations

4.3.1. Location

A.P.C., Toulouse.

4.3.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

Reactors 3.4 and 2.0 metres in diameter and 8 metres high

4.3.3. Capacity in tonnes per 24 h of final product - per unit of production

60 t.p.d.

4.3.4. Method of operation (continuous or batch, number of operating hours per day)

Continuous

4.3.5. Coefficient of utilisation in time (number of working days per annum)

300 days per year

4.3.6. Maximum rated production

18,000 t.p.y.

5. Reducing agents

5.1. Raw materials on which the reducing agent is based

Hydrocarbons, preferably natural gas

5.2. Process for conversion of above raw materials into a form for use in reduction

Reforming with steam

Conversion of CO to CO₂

Scrubbing under pressure

5.3. Condition and analysis of reducing agent at moment of use

Leaving the preparation unit, the reducing gas has an approximate composition of:

H ₂	97.3%
CH ₄	2.0%
CO	0.4%
CO ₂	0.3%
H ₂ O	0.3%

The reducing gas, which is a mixture of fresh and recycled gas, has a hydrogen content of 90%.

5.4. Preheat temperature on entering reduction apparatus

Preheating to 850°C

5.5. Gas velocity in reduction apparatus

0.5 to 0.7 m/sec

5.6. Analysis and temperature of gas after passing through reduction apparatus

Not known

6. Ores and ferruginous materials

6.1. Source

Ores and concentrates rich in iron

6.2. Analysis

No details

6.3. Size

Preferably, no fines smaller than 100 microns (10% at most).

7. Additions

None

8. Temperature zones in reduction
in °C in the reduction apparatus

Primary reactor 700°C

Secondary reactor 580°C

9. Pressure zones in reduction
in kg/cm² in the reduction apparatus
Approximately 1.5 kg/cm²
10. Product
of the reduction apparatus
 - 10.1. Form
Powder of same form as ore charged
 - 10.2. Analysis

Iron	85 - 91%
Oxygen	8 - 2%
Gangue	7%
 - 10.3. Degree of reduction
 - 10.3.1. Proportion of oxygen removed from iron oxides as a percentage of the total oxygen combined with iron in the ores
Degree of reduction 78 - 95%
 - 10.3.2. Metallic iron as a percentage of the total iron content of the product
Metallisation 67 - 92%
 - 10.4. Utilisation of by-products
None: recycling of gas
11. Further treatment of the product (10) of the reduction apparatus
 - 11.1. Method of cooling
Cooling under inert gas

11.2. Crushing, compacting or briquetting

None

11.3. Details of final size grading

Powder of similar size to ore charged

11.4. Method of concentration (e.g. magnetic separation or sintering)

Possible enrichment by magnetic separation

12. Final product obtained after treatment according to 11

Same as 10 except:

with enrichment by magnetic separation the product has the analysis:

iron 91 - 97%

O₂ 8 - 2%

gangue 1%

Once cooled the product does not re-oxidise

13. Iron yield

13.1. Iron contained in product (10) as a percentage of iron in the charge

99%

13.2. Iron contained in final product (12) as a percentage of iron in the charge

99%

14. Utilisation of final product

Destination of final product (blast furnace, electric or other furnace). Technical and economic differences compared with the use of pig iron and scrap.

No details

15. Consumption data per tonne of final product (12)

It is necessary to state the degree of reduction DR desired, which may vary between 75 and 92%

15.1. Raw materials for reduction (5.1)

No details

15.2. Fuels and thermal values required for the entire process (fuel characteristics) (thermal balance)

	<u>DR 75%</u>	<u>DR 92%</u>
Natural gas (CH ₄) in Nm ³ /h		
- for producing hydrogen	280	344
- for heating furnaces	100	140
	<u>380</u>	<u>484</u>

15.3. Electrical energy

Electricity in kWh/t	20	70

15.4. Steam used

Steam for producing H ₂	autonomy
Steam for reduction	nil

15.5. Oxygen

None

15.6. Cooling water

	<u>DR 75%</u>	<u>DR 92%</u>
Nm ³ /t	3	5

15.7. Refractories

1 lining every 5 years for the furnace

15.8. Man hours of productive labour

0.4 - 4 h/t

15.9. Labour and materials for routine maintenance

1 - 0.1 h/t

16. Overall capital investment

No details

17. Opinions expressed on the process in relation to:

No details

18. Bibliography

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French Patent No. 1.084.764 16th June 1953
French Patent No. 1.115.194 26th November 1954
French Patent No. 1.174.061 30th March 1957
French Patent No. 1.406.651 27th May 1964

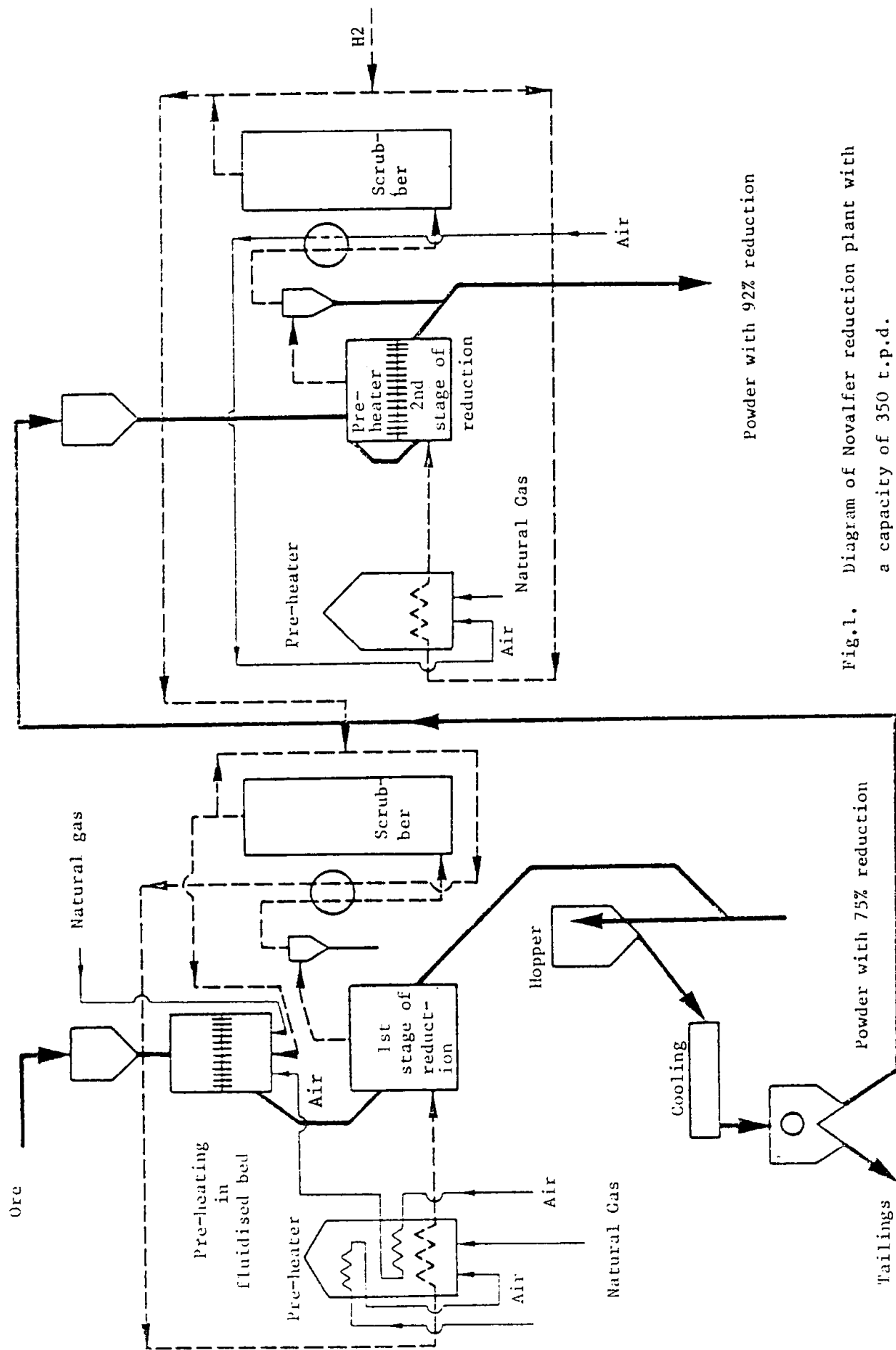


Fig.1. Diagram of Novalfer reduction plant with a capacity of 350 t.p.d.

ROTORED PROCESS

1. Characteristics

1.1. Name and history of process

Rotored process

1.2. Principle of process

Reduction in a vertical rotary furnace of a mixture of fine iron ore and solid reductant, to obtain a liquid metal product, carburised in varying degrees. The solid reducing agent, the iron ore and the necessary additions (to adjust the slag composition and desulphurise) are charged in alternate layers on to the wall of the reactor which rotates at a speed such that the raw materials charged are retained on the wall and so present very large surfaces for chemical and thermal exchange.

The heat necessary for the process is supplied partly by combustion of CO from the reduction to CO_2 and partly by the excess solid fuel (and its gas) or by the auxiliary, liquid or gaseous fuel. The pre-heated air or pure oxygen and the auxiliary fuel are injected by a burner placed in the top part of the reactor. The shape and thickness of the layers may be varied by metering of the raw materials and regulation of the upward and downward movement of an injection lance inside the reactor.

At the end of charging the total thickness of the alternate layers of ore and coal is around 10 cm.

The process takes place in two phases:

- 1) Distribution of the materials in the furnace, pre-heating to 1250°C and pre-reduction in the solid state.

- 2) Melting and where necessary simultaneous completion of reduction in the liquid state and finally discharge of the products from a hole at the bottom.

The charge materials for the reactor may be pre-heated.
The liquid metal and slag produced are discharged to a holding furnace where deslagging and desulphurisation take place.

1.3. Inventors

Prof. A. Ramacciotti, Centro Sperimentale Metallurgico, Rome.

2. General description (schematic drawing)

See drawing

3. Plant required

The pilot plant comprises separate, metered hoppers for iron ore fines and coal to feed the reducing reactor during operation, the reducing reactor, the ancillary equipment for moving the reactor and for regulating the means of feeding in the raw materials and oxygen.

4. State of development

4.1. Name of laboratories where the process is situated

Centro Sperimentale Metallurgico, Rome.

4.2. Pilot plant

4.2.1. Location

Centro Sperimentale Metallurgico, Rome.

4.2.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

Reactor: diameter 500 mm; height 1350 mm; maximum speed of rotation 105 r.p.m.

4.2.3. Capacity in tonnes per 24 h of final product

2.4 t per 24 hours

4.3. Industrial installations

Diagram of industrial-scale plant

5. Reducing agents

5.1. Raw materials on which the reducing agent is based

Various types of solid reducing agents may be used: coke, coals with various volatile contents.

5.2. Process for conversion of above raw materials into a form for use in reduction

As appropriate, drying, crushing and screening.

5.3. Condition and analysis of reducing agent at moment of use

Fines < 3 mm.

6. Ores and ferruginous materials

6.2. Analysis

In the tests rich iron ores were used (63 - 65% Fe) but ores with a lower Fe content (50 - 55%) may also be used.

6.3. Size

< 3 mm

6.4. Preparation

As appropriate drying, crushing and screening

7. Additions

7.1. Material

Fluxes (lime)

7.2. Analysis

No details

7.3. Size

Similar size to the ore

8. Temperature zones in reduction
in °C in the reduction apparatus

No details

9. Pressure zones in reduction
in kg/cm² in the reduction apparatus

None

10. Product
of the reduction apparatus

10.1. Form

Liquid metal

10.2. Analysis

Similar to pig iron (C content 2 - 4%)

11. Further treatment of the product (10) of the reduction apparatus

No details

12. Final product obtained after treatment according to 11

No details

13. Iron yield

13.1. Iron contained in product (10) as a percentage of iron in the charge

95%

14. Utilisation of final product

Destination of final product (blast furnace, electric or other furnace). Technical and economic differences compared with the use of pig iron and scrap

Steelmaking

15. Consumption data per tonne of final product (12)

15.1. Raw materials for reduction (5.1)

1620 kg iron ore (65% Fe) per tonne Fe and
590 kg (coal 90% C_{fix}) per tonne Fe.

- 15.2. Fuels and thermal values required for the entire process
(fuel characteristics) (thermal balance)

530 kg carbon per tonne Fe.

- 15.3. Electrical energy

- 15.3.1. For heating

None

- 15.3.2. For power

1.5 kWh per tonne Fe

- 15.4. Steam used

None

- 15.5. Oxygen

1471 Nm³ air per tonne Fe

- 15.6. Cooling water

No details

- 15.7. Refractories

No details

- 15.8. Man hours of productive labour

0.1 - 0.14 h per tonne Fe.

16. Overall capital investment

- 16.1. For the plant described in (3) and related to one tonne of
capacity for the final product (12)

30,800 It. Lire per annual tonne iron (1974) for a
500,000 t.p.a. plant.

17. Opinions expressed on the process

None

18. Bibliography

18.1. Articles and publications

CSM Report, Rome, September 1972.

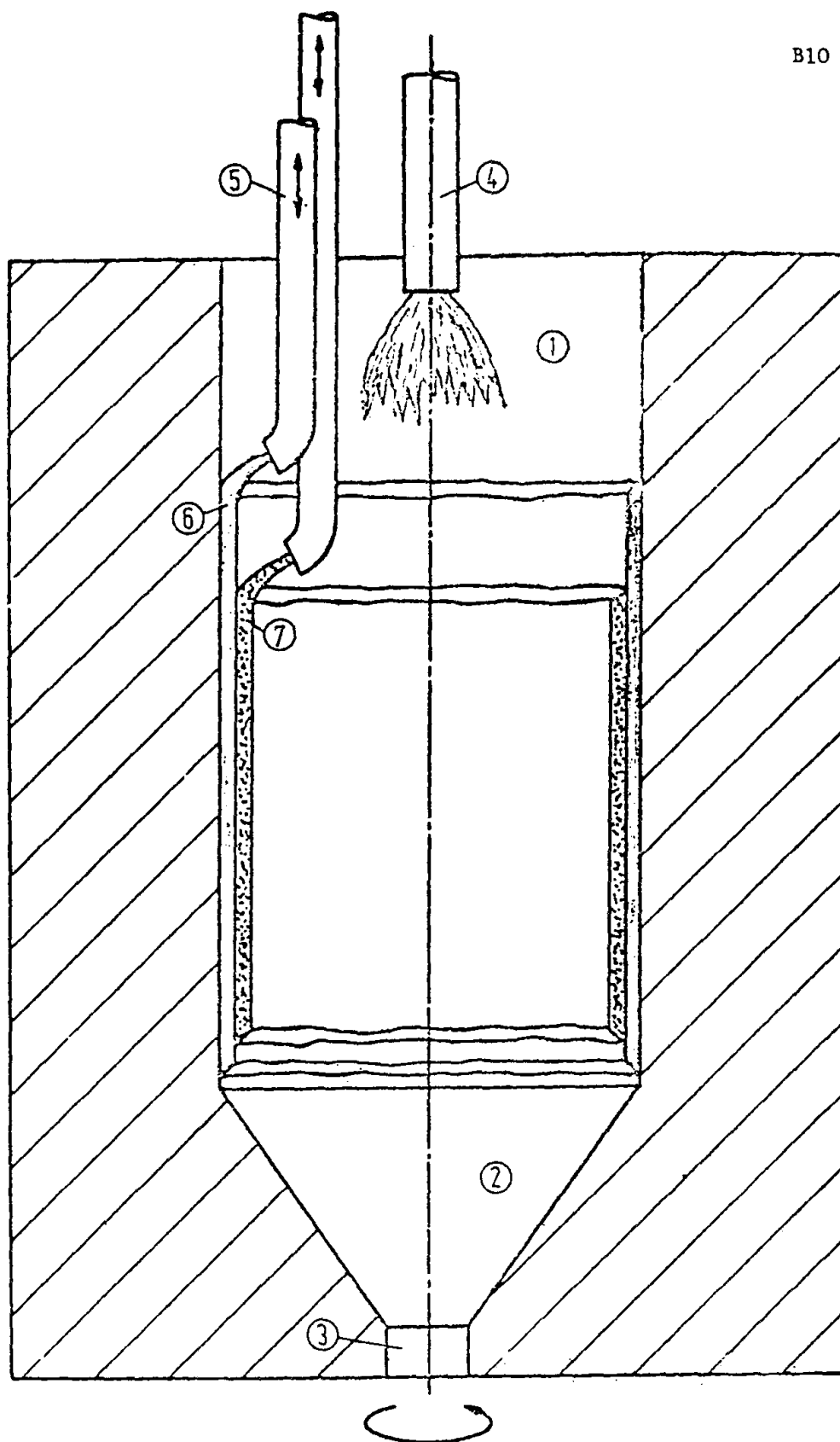
18.2. Patents

Italian Patent No. 919714

Corresponding patents:

- Belgium		No.778.620
- France		No.2.124.286
- Germany		No.2.203.455
- Japan	Appl.	No.9006/72
- UK	Appl.	No.4077/72
- Luxembourg		No.64686
- Holland	Appl.	No.72.00.786
- USA		No.3.833.355

Improvement to Italian patent appl. No.47770 A/75



1) Reactor - cylindrical part	5) Lances for distributing materials
2) Reactor - conical part	6) Coke or coal
3) Outlet for liquid products	7) Iron ore and fluxes
4) Burner	

Fig. 1 Cross section through reduction reactor of Rotored process.

WIBERG-SÖDERFORS PROCESS

1. Characteristics

1.1. Name and history of process

Wiberg-Söderfors process. The Wiberg process was invented in 1918. The first industrial plant was built in 1932 at Söderfors and it was not in continuous production until 1941. The reductant was charcoal at first; from 1950 onwards coke was used for economic reasons.

1.2. Principle of process

Reduction in shaft furnace using a reducing gas produced in a carburettor and continuously regenerated.

1.3. Inventors

Prof. Martin Wiberg

2. General description (schematic drawing)

For details see Figures 1 and 2.

The shaft furnace for reduction is charged from the top with rich lump ore or pelleted concentrates; sinter can also be used. Fed in from the bottom, the hot reducing gas, produced outside the furnace in a carburettor and containing approximately 20% H_2 and 75% CO , passes through the descending burden. Part of the reducing gas (about 75%) is drawn off half-way up and fed to the carburettor to be regenerated. The gas remaining in the reduction furnace is burned using air blown in about three-quarters of the way up the furnace, thus pre-heating the charge. The sponge iron produced is used like high grade scrap. In order to desulphurise

the reducing gas, a shaft furnace charged with graded lumps of dolomite is placed between the carburettor and the reduction furnace. The carburettor is heated by means of three or four pairs of electrodes. Hydrogen is added to the recycled gas in any form (steam, hydrocarbon, etc.) before it enters the carburettor in order to increase its proportion in the reducing gas. As an alternative natural gas or fuel oil may be used, after reforming by CO_2 .

3. Plant required

- Shaft furnace for reduction
- Carburettor
- Desulphuriser

4. State of development

4.1. Name of laboratories where the process is situated

No details

4.2. Pilot plant

- 1) Woxna (1920), 0.5 tonne per 24 hours
- 2) Sandviken (1925), 2 tonnes per 24 hours
- 3) Domnarvet (1930), 2 tonnes per 24 hours

4.3. Industrial installations

4.3.1. Location

- 1) Sandviken, Sweden (Sandvik)
- 2) Hofors, Sweden (SKF)
- 3) Yasugi, Japan (Hitachi Metals)

4.3.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

No details

4.3.3. Capacity in tonnes per 24 h of final product - per unit of production

- 1) Sandviken: 1 furnace - approximately 25,000 t.p.y.
- 2) Hofors: 1 furnace - approximately 25,000 t.p.y.
- 3) Yasugi: 1 furnace - 10,000 t.p.y.

4.3.4. Method of operation (continuous or batch, number of operating hours per day)

Continuous

4.3.5. Coefficient of utilisation in time (number of working days per annum)

97 - 98%

4.3.6. Maximum rated production

Approximately 25,000 - 30,000 t.p.y. per furnace for plants 1 and 2

5. Reducing agents

5.1. Raw materials on which the reducing agent is based

Coal or coke with substantial additions of fuel oil, natural gas, propane or butane (see Figure 2).

5.2. Process for conversion of above raw materials into a form for use in reduction

Gasification by carburettor.

5.3. Condition and analysis of reducing agent at moment of use

Reducing gas ($1400 \text{ Nm}^3/\text{t}$)

74.4%	CO	21.1% H_2
3.2%	CO_2	1.3% H_2O

5.4. Preheat temperature on entering reduction apparatus

950 - 1000°C

5.5. Gas velocity in reduction apparatus

No details

5.6. Analysis and temperature of gas after passing through reduction apparatus

5.6.1. Recycled gas ($865 \text{ Nm}^3/\text{t}$)

54.7% CO	6.6% H_2
23.0% CO_2	15.7% H_2O

($700\text{--}800^\circ\text{C}$)

5.6.2. Excess gas ($175 \text{ Nm}^3/\text{t}$)

38% CO	11.5% H_2
40% CO_2	10.5% H_2O

5.6.3. Waste gases ($1500 \text{ Nm}^3/\text{t}$)

68.7% N_2	15.8% O_2
12.0% CO_2	3.5% H_2O

(approximately 150°C)

6. Ores and ferruginous materials

6.1. Source

No special restrictions

6.2. Analysis

Approximately 66% Fe, but ore containing 50% Fe has already been processed; low P content.

Example: Fe 66%; P 0.005%, S 0.006%.

6.3. Size

Pellets, sinter or lump ores (15-20 mm)

Pellets are best (25-30 mm dia.). The pellets should be highly reducible, with high mechanical strength during reduction (no swelling, disintegration or softening).

6.4. Preparation

Pelletising or sintering on the grate

7. Additions

Crude dolomite (30 kg/t) for the desulphurisation.

Addition of CaO (5%) as binder for the very rich ore pellets.

8. Temperature zones in reduction
in °C in the reduction apparatus

Preheating zone: 930-950°C

800°C during recycling

1000°C in the combustion zone

9. Pressure zones in reduction
in kg/cm² in the reduction apparatus

Low pressure; no figures

13. Iron yield

See under 10

14. Utilisation of final product

Destination of final product (blast furnace, electric or other furnace). Technical and economic differences as compared with using pig iron and scrap

As substitute for high grade scrap; used for making special steels in the electric furnace.

15. Consumption data per tonne of final product (12)

15.1. Raw materials for reduction (5.1)

Depends on the Fe content

15.2. Fuels and thermal values required for the entire process (fuel characteristics) (thermal balance)

No details

15.3. Electrical energy

Comparison per tonne sponge iron (fuel and electricity)

	<u>Hitachi Metals (18.1.16)</u>	<u>SKF Hofors (18.1.17)</u>
Pellets	1.350 kg	not stated
Dolomite	not stated	30 kg
Coke	128 kg	85 kg
Fuel oil	50 kg	-
Propane/butane	-	45 kg
Electrodes	0.8 kg	1.2 kg
Electricity	870 kWh	880 kWh

The porosity of the pellets has a great influence on the reducibility and so on the power consumption. Suzumoto (18.1.16) states that increasing the porosity from 21.3% to 28.5% reduces the power consumption by 6% and the coke consumption by 20%.

15.4. Steam used

No details

15.5. Oxygen

No details

15.6. Cooling water

9 Nm³/t

15.7. Refractories

No details

15.8. Man hours of productive labour

0.7 - 1.3 h/t

15.9. Labour and materials for routine maintenance

No details

16. Overall capital investment

No recent information

17. Opinions expressed on the process in relation to:

17.1. State of development

Long established process; no more development

17.2. Practical operating difficulties

Sometimes sticking prevents the burden descending. A certain amount of dust is in circulation.

17.3. Possibilities of exploitation

A scrap substitute produced in this way is useful when large supplies of cheap electricity are available, or when special products are to be obtained for which a high level of purity of the metal is required (e.g. ball bearings).

17.4. Special requirements to be met by certain materials (e.g. refractories)

No details

18. Bibliography

18.1. Articles and publications

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18.2. Patents: F.M. Wiberg

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German patent	No. 356.762 (1919)
Austrian patent	No. 82.588 (1919)
Swiss patent	No. 90.120 (1920)
French patent	No. 501.733 (1919)
Spanish patent	No. 130.334 (1919)
French patent	No.1.073.811 (1952)
British patent	No. 717.851 (1954)
U.S. patent	No.2.635.957 (1957) use of gas generators

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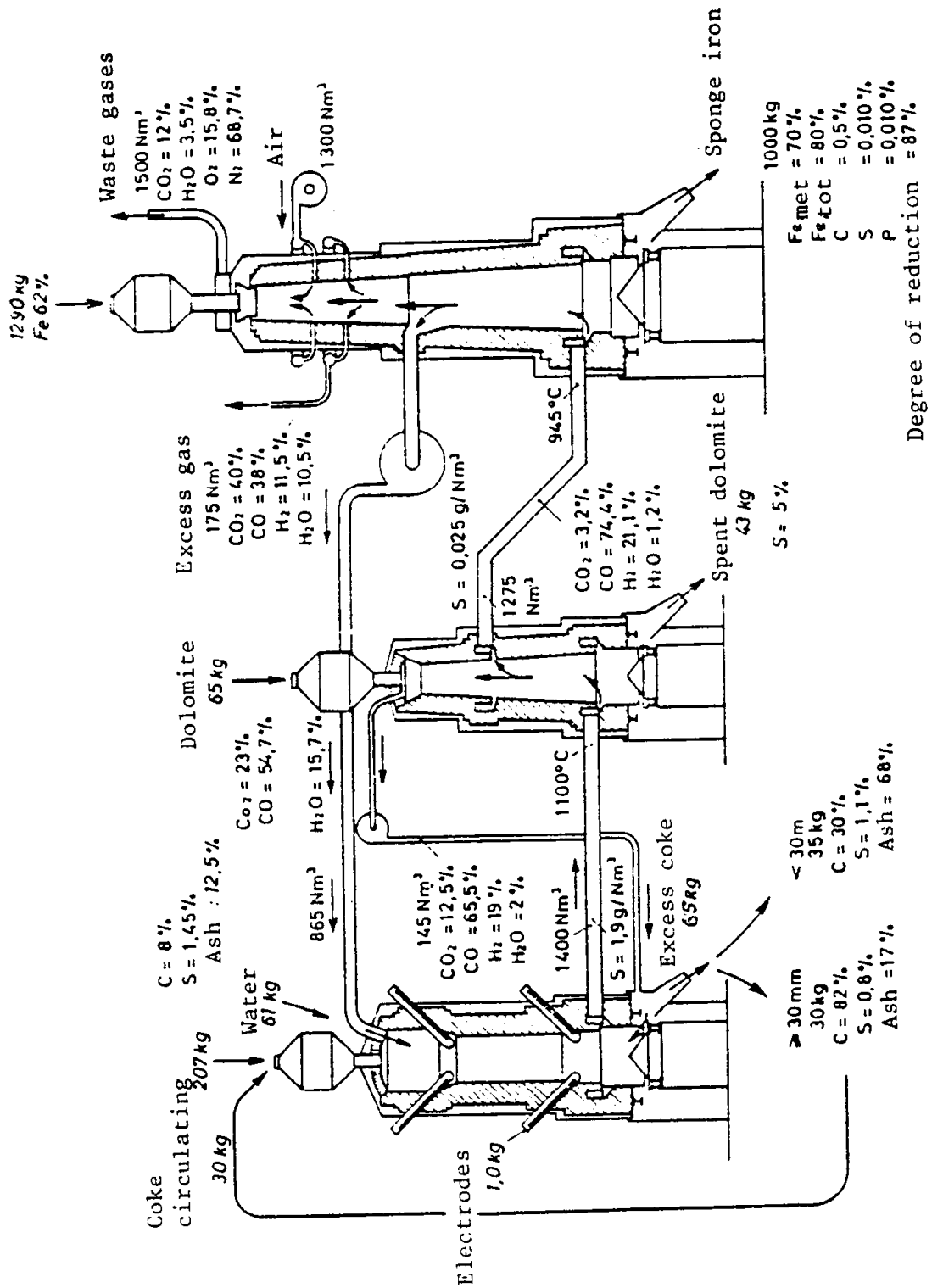


Figure 1 Materials balance.

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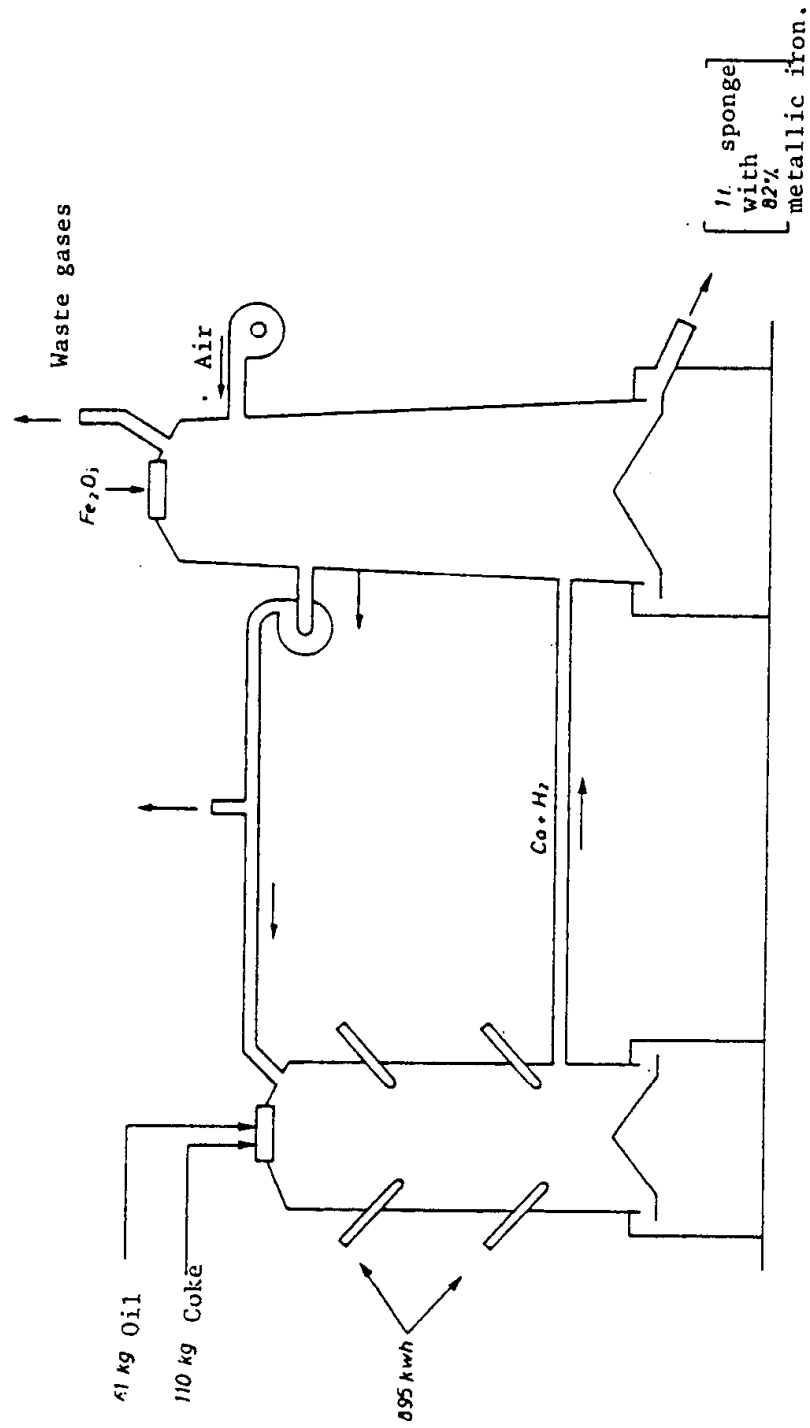


Fig. 2. Wiberg process with coke + gas oil
(Usual practice in most Swedish plants)

AACHEN ROTARY KILN REDUCTION PROCESS

1. Characteristics

1.1. Name and history of process

Aachen rotary kiln reduction process.

If we classify the reduction processes, excluding the blast furnace process, in a systematic manner, we see that there is no rotary kiln process with direct injection of liquid or gaseous reducing agents into the iron ore charge. The Aachen Rotary Kiln Reduction Process should fill this gap. After preliminary investigations on a laboratory scale which were started in 1960, an experimental plant has now been developed.

1.2. Principle of process

Reducing agents (oil or natural gas) and air are introduced, separately from one another, via the axis of rotation of the rotary kiln to the outer surface of the kiln. The reductant is injected directly into the iron ore charge through special burners and is cracked under the influence of heat and partially combusted with an exactly measured amount of air. The soot which forms and the gas components carbon monoxide and hydrogen reduce the iron oxides as they pass through the bed of ore. The working temperature which, to prevent agglomeration, should not exceed 900 to 1000°C (depending on the type of ore), is maintained by subsequent combustion of the reducing components which have not been used. The process is continuous.

1.3. Inventors

Professor Dr.-Ing. Werner Wenzel

Professor Dr.-Ing., Dr.-Ing.h.c. Hermann Schenck

1.4. Companies involved in development

Rheinstahl AG Essen

2. General description (schematic drawing)

See figure

3. Plant required

In addition to the actual reduction unit the only other plant required is that for preparing the ore (crushing, screening or pelletising) and for subsequent treatment of the product (cooling, briquetting).

4. State of development

4.1. Name of laboratories where the process is situated

Institut für Eisenhüttenwesen der Rheinisch-Westfälischen Technischen Hochschule, Aachen (Institute of Metallurgy of the Rhineland-Westphalia Technical University).

4.2. Pilot plant

4.2.1. Location

Institut für Eisenhüttenwesen der Rheinisch-Westfälischen Technischen Hochschule, Aachen (Institute of Metallurgy of the Rhineland-Westphalia Technical University).

4.2.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

length of rotary kiln: 4.50 m
inside diameter: 0.80 m

4.2.3. Capacity in tonnes per 24 h of final product

Approximately 2 t per 24 h

4.3. Industrial installations

None

5. Reducing agents

5.1. Raw materials on which the reducing agent is based

Oil or natural gas (so far, only tests with light fuel oil have been carried out (sulphur content: < 1%); there is no practical experience with other oils or natural gas)

5.2. Process for conversion of above raw materials into a form for use in reduction

Cracking and partial combustion directly in the heated bed of ore.

5.3. Condition and analysis of reducing agent at moment of use

See 5.1.

5.4. Preheat temperature on entering reduction apparatus

No preheating

5.5. Gas velocity in reduction apparatus

Up to 1 m/s

5.6. Analysis and temperature of gas after passing through reduction apparatus

Average analysis of the dry off-gas:

CO ₂	=	12.7%
CO	=	10.8%
H ₂	=	8.9%
CH ₄	=	1.3%
N ₂	=	<u>66.3%</u>
		100.0%

Temperature of off-gas: 150 to 250°C

6. Ores and ferruginous materials

Tests so far with Venezuelan ore (1-15 mm) and Carol Lake Pellets. The proportion of fines (< 1 mm) should be small in order to prevent dust losses in the off-gas.

7. Additions

None. In the case of reducing agents with a high sulphur content, desulphurising agents may be added (dolomite).

8. Temperature zones in reduction
in °C in the reduction apparatus

900 to 1000°C

9. Pressure zones in reduction
in kg/cm² in the reduction apparatus

1 atm

10. Product

10.1. Form

Like the charge material

10.2. Analysis

75% Fe_{met}

0.06 to 0.08% S

0.02% C

Proportions of gangue according to type of ore

10.3. Degree of reduction

10.3.1. Proportion of oxygen removed from iron oxides as a percentage of the total oxygen combined with iron in the ores

So far achieved: 85-90% oxygen reduction; higher values possible.

10.3.2. Metallic iron as a percentage of the total iron content of the total oxygen combined with iron in the ores

No details

10.4. Utilisation of by-products

For analysis of off-gas see 5.6.

Quantity of off-gas: approximately 150 Nm³

11. Further treatment of the product (10) of the reduction apparatus

11.1. Method of cooling

Quenching in water tank

11.2. Crushing, compacting or briquetting

Fines under 5 mm briquetted

11.3. Details of final size grading

Same as size of the ore charge

11.4. Method of concentration (e.g. magnetic separation or sintering)

None

12. Final product obtained after treatment according to 11

12.1. Form

Briquetted or granular sponge iron

12.2. Analysis

See 10.2.

13. Iron yield

13.1. Iron contained in the product at (10) as a percentage of iron in the charge

100%

13.2. Iron contained in final product (12) as a percentage of iron in the charge

No details

14. Utilisation of final product

Destination of final product (blast furnace, electric or other furnace). Technical and economic differences compared with the use of pig iron and scrap

Can be used in blast furnaces

Melting in electric arc furnaces

15. Consumption data per tonne of final product (12)

15.1. Raw materials for reduction (5.1)

None

15.2. Fuels and thermal values required for the entire process
(fuel characteristics) (thermal balance)

Total heat requirement of the 2 t.p.d.

experimental plants:

5.4 Gcal/t Fe.

15.3. Electrical energy

No details

15.4. Steam used

No details

15.5. Oxygen

No details

15.6. Cooling water

No details

15.7. Refractories

No details

15.8. Man hours of productive labour

No details

15.9. Labour and materials for routine maintenance

No details

16. Overall capital investment

No details

17. Opinions expressed on the process in relation to:

17.1. State of development

Tests carried out up until now on a semi-industrial scale.
No industrial application. Meanwhile the Allis-Chalmers
process uses direct reduction on the principle described.

17.2. Practical operating difficulties

No critical technical problems have arisen so far

17.3. Possibilities of exploitation

Reduction of ores rich in iron, fines to small lumps (see 6).

17.4. Special requirements to be met by certain materials
(e.g. refractories)

Materials consumption, particularly refractory materials,
as with other rotary kiln processes

18. Bibliography

18.1. Articles and publications

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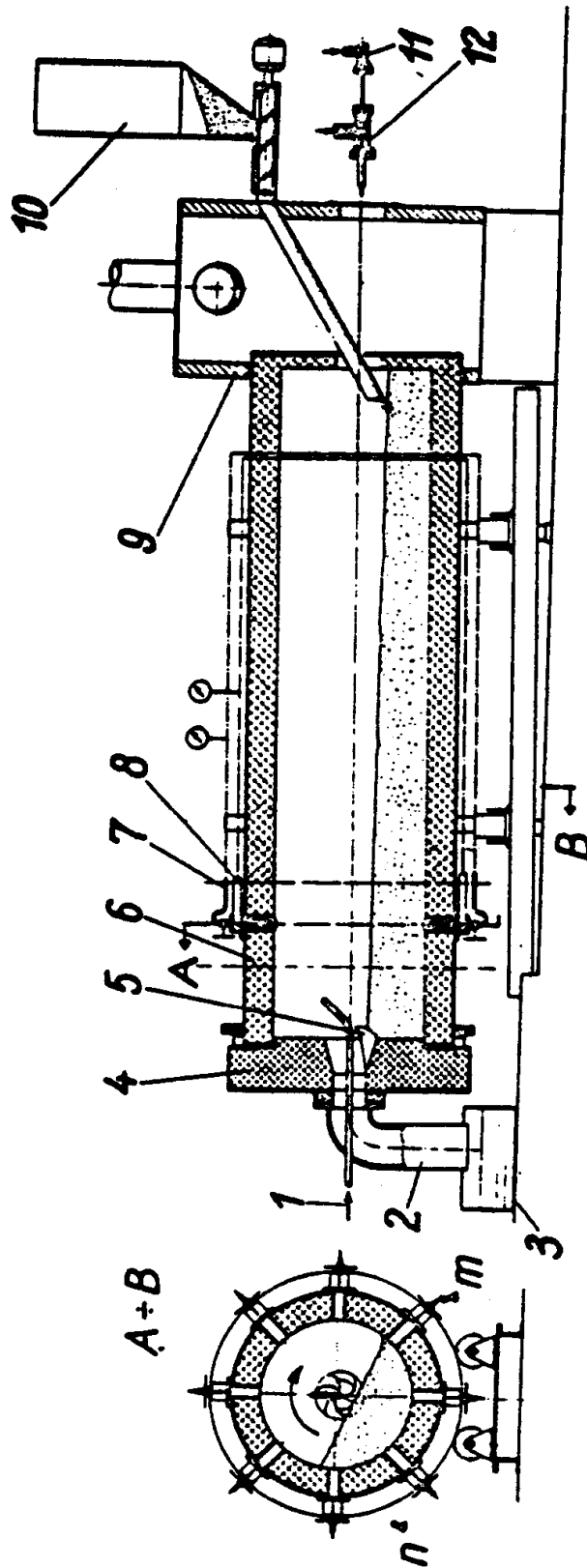
Filed in Germany under No. 35 532 VIa/18a

German Auslegeschrift No:

1 189 096 1 221 654

Also filed in many industrialised countries.

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1. Combustion air	7. Primary air inlet
2. Discharge of sponge iron	8. Oil inlet
3. Tank of water for cooling sponge iron	9. Flue leading to stack
4. Protective plate	10. Hopper for raw materials
5. Conveyor shovel	11. Oil inlet
6. Rotary kiln	12. Air inlet
	m,n Means of opening and closing

Fig. 1 Reduction process in Aachen rotary kiln.

ARTHUR D LITTLE PROCESS

1. Characteristics

1.1. Name and history of process

Arthur D Little direct reduction process

1.2. Principle of process

Reduction of ore fines in a fluidised bed by $H_2 + CO$ at $700-900^{\circ}C$.

1.3. Inventors

Arthur D Little, Acorn Park, Cambridge, Massachusetts,
U S A.

1.4. Companies involved in development

Arthur D Little Inc., Acorn Park, Cambridge, Massachusetts,
U S A.

2. General description (schematic drawing)

See Figure 1.

3. Plant required

Production of the $H_2 + CO$ mixture by partial combustion of natural gas or other hydrocarbons with air. Any of the other processes for producing $H_2 + CO$ mixtures may be used.

No recycling. The off gases are used for pre-heating the raw materials or for power generation. Pre-heating of the ore is carried out in a rotary kiln, shaft furnace or preferably in a fluidised bed.

A fluidised bed reactor in several stages is used for the reduction.

4. State of development

4.1. Name of laboratories where the process is situated

Arthur D Little Inc.

4.2. Pilot plant

4.2.1. Location

Cambridge, Massachusettes

4.2.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

Fluidised-bed reactor of 10 cm internal diameter

Pilot plant with two stages, 30 cm internal diameter
and total height of 9 m.

4.2.3. Capacity in tonnes per 24 h of final product

Approximately 1 tonne sponge iron per day

4.3. Industrial installations

No industrial-scale plant

5. Reducing agents

5.1. Raw materials on which the reducing agent is based

Natural gas

5.2. Process for conversion of above raw materials into a form for use in reduction

The reducing gas mixture is produced, in a burner of the Texaco process type, by partial combustion of natural gas or other fuels containing hydrocarbons.

5.3. Condition and analysis of reducing agent at moment of use

The reduction gas produced by partial combustion of methane with air contains 20 - 25% CO, 41% H₂ and 38.5% N₂.

5.4. Preheat temperature on entering reduction apparatus

Before reaction, the ore is pre-heated to about 860°C.

In the first stage the reducing gas is fed in at a temperature of 900°C.

5.5. Gas velocity in reduction apparatus

Up to 90 cm/sec.

5.6. Analysis and temperature of gas after passing through reduction apparatus

The gases leaving the final stage contain 20% H₂, 11% CO, 20% H₂O, 9% CO₂ and 40% N₂.

6. Ores and ferruginous materials

6.1. Source

Ore fines with high iron content

6.2. Analysis

Ore from Cerro-Bolivar, Venezuela, with 64.5% Fe.

6.3. Size

0 - 2 mm

6.4. Preparation

Crushing

7. Additions

None

8. Temperature zones in reduction
in °C in the reduction apparatus

750 - 900°C

9. Pressure zones in reduction
in kg/cm² in the reduction apparatus

1 - 3 atm

10. Product

10.1. Form

Reduced, powdered ore

10.2. Analysis

Iron content up to 95%

10.3. Degree of reduction

No details

10.4. Utilisation of by-products

No details

11. Further treatment of the product (10) of the reduction apparatus

11.1. Method of cooling

No details

11.2. Crushing, compacting or briquetting

Briquetting

11.3. Details of final size grading

No details

11.4. Method of concentration (e.g. magnetic separation or sintering)

None

12. Final product obtained after treatment according to 11

12.1. Form

Sponge iron briquettes

12.2. Analysis

Same as product 10

13. Iron yield

Except for a slight loss of dust, no loss during reduction

14. Utilisation of final product

Destination of final product (blast furnace, electric or other furnace). Technical and economic differences as compared with using pig iron and scrap

Used for steelmaking. The advantages or disadvantages compared with scrap are not yet known in detail.

15. Consumption data per tonne of final product (12)

15.1. Raw materials for reduction (5.1)

Ore fines with 65% Fe, 1.62 t/t Fe.

15.2. Fuels and thermal values required for the entire process
(fuel characteristics) (thermal balance)

400 m³ natural gas

15.3. Electrical energy

240 kWh per tonne

15.4. Steam used

None

15.5. Oxygen

None

15.6. Cooling water

No details

15.7. Refractories

No details

15.8. Man hours of productive labour

No details

15.9. Labour and materials for routine maintenance

No details

16. Overall capital investment

No details

17. Opinions expressed on the process in relation to:

17.1. State of development

Experimental work has been done on a pilot plant

17.2. Practical operating difficulties

The major difficulty lies in the sintering of the ore in the fluidised bed reactor.

17.3. Possibilities of exploitation

The process would be used preferably with rich ores of sizes suitable for a fluidised bed and if a cheap supply of natural gas or fuel oil is available.

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- 7) Hyde, A.W.: Development work on the ESSO-Research-Little-Process. Paper presented to the Chicago Conference, May 1960.

18.2. Patents

US Patent No. 2,894,831, July 14, 1959. B.S. Old and R.W.Hyde. Process of fluidised bed reduction of iron ore followed by electric furnace melting.

US Patent No. 2,990,269, June 27, 1961. R.W.Hyde. Refining of ores with hydrocarbon gases.

US Patent No. 3,020,149, February 6, 1962. B.S. Old, R.W.Hyde, H.P.Meissner, S.V.Margolin, F.C.Schora -- Beneficiation of ores.

US Patent No. 3,031,293, April 24, 1962. H.P.Meissner. Iron ore reduction process in a fluidised bed.

US Patent No. 3,228,669, January 11, 1966. B.S. Old, R.W.Hyde, F.C. Schora. System for beneficiation of ores.

BASSET PROCESS

1. Characteristics

1.1. Name and history of process

Basset process

The process was first used for producing wrought iron directly from ores.

1.2. Principle of process

Reduction using solid fuel (coal) in an inclined rotary kiln producing at the same time molten iron and cement clinker.

1.3. Inventors

Lucien Basset

1.4. Companies involved in development

The process has been applied with varying degrees of success in France, Spain, Portugal, Japan and Denmark and by the firm F.L.Smith & Co. (Aalborg, Denmark).

2. General description (schematic drawing)

See drawing

3. Plant required

Rotary kiln of the same type as standard cement kilns with some design changes (the "production of clinker" is not dealt with).

4. State of development

4.1. Name of laboratories where the process is situated

No details

4.2. Pilot plant

No details

4.3. Industrial installations

4.3.1. Location

Denmark

4.3.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

Two kilns of 35 and 50 m in length

4.3.3. Capacity in tonnes per 24 h of final product - per unit of production

Total capacity for the two kilns is about 100 t.p.d.

4.3.4. Method of operation (continuous or batch, number of operating hours per day)

Continuous operation

4.3.5. Coefficient of utilisation in time (number of working days per annum)

About 330 days per annum

4.3.6. Maximum rated production

No details

5. Reducing agents

5.1. Raw materials on which the reducing agent is based

Coal, oil.

5.2. Process for conversion of above raw materials into a form for use in reduction

No details

5.3. Condition and analysis of reducing agent at moment of use

Small particles of coke are used in the Danish plant (86 % C, 1 % S and 10 % ash). The process also works using oil or coal dust as fuel.

5.4. Preheat temperature on entering reduction apparatus

No details

5.5. Gas velocity in reduction apparatus

No details

5.6. Analysis and temperature of gas after passing through reduction apparatus

No details

6. Ores and ferruginous materials

6.1. Source

Roasted pyrites, scrap which is economically suitable (chips, galvanised or tin-plated thin sheet scrap) etc.

6.2. Analysis

Roasted pyrites: approximately 67 % Fe, 2 % SiO₂, 2 % S.

6.3. Size

Fine ores

6.4. Preparation

None

7. Additions

7.1. Material

Limestone

7.2. Analysis

Approximately 54 % CaO, 2 % SiO₂

7.3. Size

No details

8. Temperature zones in reduction
in °C in the reduction apparatus

No details

9. Pressure zones in reduction
in kg/cm² in the reduction apparatus

No details

10. Product
of the reduction apparatus

10.1. Form

Molten iron (by-product: cement clinker).

10.2. Analysis

C	Si	Mn	P	S
4.2-4.6 %	0.1 %	0.8-4.5 %	0.1-0.2 %	0.01 %

10.3. Degree of reduction

10.3.1. Proportion of oxygen removed from iron oxides as a percentage of the total oxygen combined with iron in the ores

Some of the iron is present in the slag in the form of FeO and iron granules. The metallic iron is recovered and fed back into the process.

10.3.2. Metallic iron as a percentage of the total iron content of the product

100 %

10.4. Utilisation of by-products

About 320 kg clinker are produced per tonne of iron. The ratio of CaO to SiO₂ is 2.8 to 3.0. In contrast to normal cement clinker this clinker contains ferrous instead of ferric oxide, a small amount of CaS and 5 % C.

11. Further treatment of the product (10) of the reduction apparatus

None

12. Final product obtained after treatment according to 11

No details

13. Iron yield

No details

14. Utilisation of final product

Destination of final product (blast furnace, electric or other furnace). Technical and economic differences as compared with using pig iron and scrap

The end product is pig iron.

15. Consumption data per tonne of final product (12)

No details

16. Overall capital investment

16.1. For the plant described in (3) and related to one tonne of capacity for the final product (12)

Capacity expressed in tonnes per day or per year.

No recent figures available

16.2. Construction costs of ancillary plant not covered in (3)

No details

17. Opinions expressed on the process in relation to:

17.1. State of development

Industrial plant commissioned in 1939.

17.2. Practical operating difficulties

The Basset process has not been used for several years. The plant in Aalborg (Denmark) produces synthetic pig iron from scrap.

17.3. Possibilities of exploitation

Fine solid fuels and powdered ores are used which have a high sulphur content and also contain zinc.

17.4. Special requirements to be met by certain materials
(e.g. refractories)

No details

18. Bibliography

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18.2. Patents

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manufacture of iron and steel directly
from the ore and improved apparatus
therefor - 30 November 1920.
- No. 1 419 801 - Basset, Lucien Paul - Process of making
iron or steel - 13 June 1922.
- No. 1 435 686 - Basset, Lucien Paul - Process for the
direct manufacture of iron and steel
- 14 November 1922.
- No. 1 534 051 - Basset, Lucien Paul - Process for the
direct reduction of ores and particularly
of iron ore - 21 April 1925.

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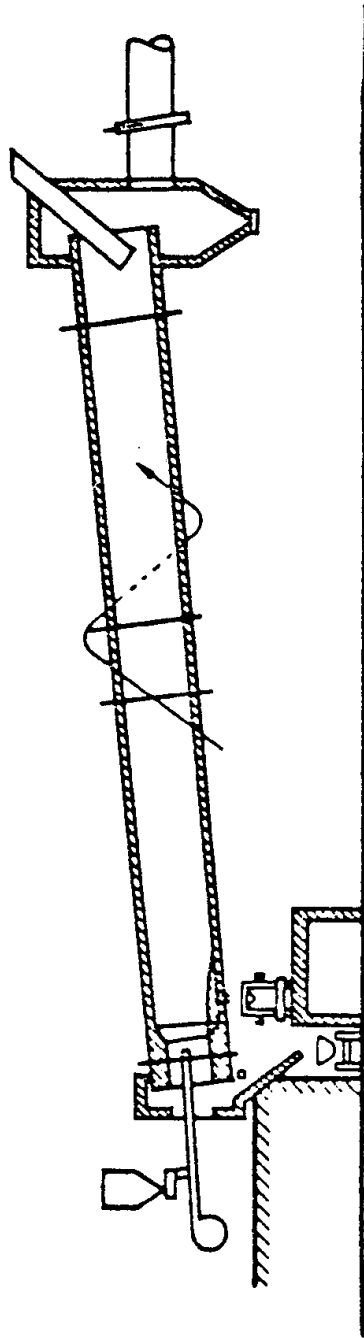


Fig. 1. Rotary Kiln for the Basset process.

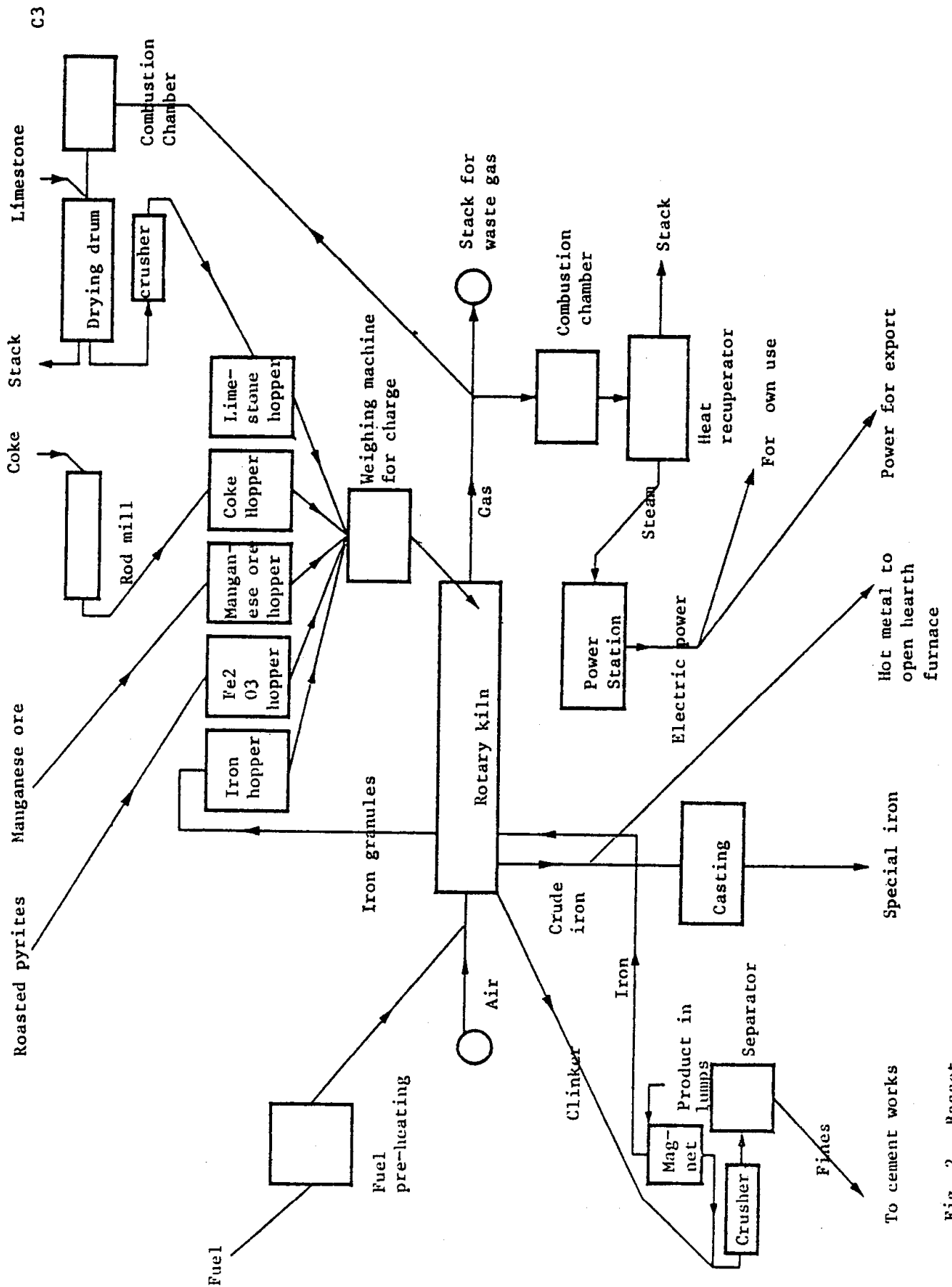


Fig. 2. Basset

BOUCHET-IMPY PROCESS

1. Characteristics

1.1. Name and history of process

Bouchet-Imphy process

1.2. Principle of process

Process for reducing iron ore, pre-reduced products, sponge iron and iron powder and scrap to give liquid iron or steel in equipment within which an oxidising medium above the reducing medium gives good utilisation of the off-gases from the reduction.

1.3. Inventors

M. Jean Bouchet, Société Métallurgique d'Imphy

1.4. Companies involved in development

The process was developed by the Société Métallurgique d'Imphy, now an integral part of the Société Creusot-Loire, 12 rue de la Rochefoucauld, Paris 9^e, and applied jointly by this company and the Office National Industriel de l'Azote (nitrogen) (ONIA), 38 avenue Hoche, 75 Paris 8^e or Route d'Espagne, 31 Toulouse.

2. General description (schematic drawing)

See drawing

3. Plant required

Retort or rotary kiln for reduction, of cylindrical shape tapered at either end, the axis of revolution of the whole being horizontal or slightly sloping. The whole steel unit has a charging and tapping door and is lined with magnesite bricks (centre part of diagram). The plant is operated on a semi-industrial scale.

The so-called autothermic natural gas reformer is composed of a burner and various chambers for the injection, mixing and dissociation of the gases produced by the burner and the natural gas injected. The whole unit is movable and has the outward appearance of a cylinder the axis of which is an extension of that of the reduction retort to which it is connected by a joint during its working period (left-hand part of diagram). A similar unit runs on liquid hydrocarbons.

Chamber for recuperation and removal of the off-gases (right-hand part of diagram).

4. State of development

4.1. Name of laboratories where the process is situated

None

4.2. Pilot plant

(Dates from 1957)

4.2.1. Location

Pamiers Works (09 Pamiers, France) of the Société Métallurgique d'Imphy.

4.2.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

Internal dimensions of the reduction retort:

diameter: 1.2 m, total length: 3.5 m

length of cylindrical part: 2 m

total volume: 2.6 m^3

weight of refractory (magnesite): 11 tonnes

4.2.3. Capacity in tonnes per 24 h of final product

The useful capacity per charge is 800 kg, that is 2.5 t.p.d. in batch operation.

4.3. Industrial installations

A plant on a semi-industrial scale has been in existence since 1963.

4.3.1. Location

The plant was built initially at the Office National Industriel de l'Azote, 31 Toulouse, France, and then moved to the Aciéries et Usines Métallurgiques de Decazeville, 12 Decazeville, France.

4.3.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

Internal dimensions of the reduction retort:

diameter: 1.6 m, total length: 5.7 m

length of cylindrical portion: 3.7 m

total volume: 7.5 m^3

weight of refractory (magnesite): 32 tonnes

4.3.3. Capacity in tonnes per 24 h of final product - per unit of production

Production capacity with rich hematite ore in batch operation: 9 - 12 t.p.d. In continuous operation: iron 30 t.p.d., steel 22 t.p.d. It is estimated that an industrial plant of three 60 m³ retorts could have a unit production capacity of approximately 1000 t.p.d. (see 18.1.1.)

4.3.4. Method of operation (continuous or batch, number of operating hours per day)

Batch operation, the solid materials being charged into the empty retort after each tapping or semi-continuous operation, in which the materials are charged at regular intervals and some of the products tapped.

4.3.5. Coefficient of utilisation in time (number of working days per annum)

No details

4.3.6. Maximum rated production

The hourly production rate (start of operation to tapping) varies from 1.5 tonnes per hour with rich hematite ore to 2 tonnes per hour with wustite and 3.0 tonnes per hour with iron powder with 91% total Fe and 7% oxygen bound with the iron.

5. Reducing agents

5.1. Raw materials on which the reducing agent is based

Coal (from anthracites or graphites to flame coals) and natural gas (or liquid hydrocarbons). It should be noted that the gas is primarily intended to provide additional heat by combustion with the oxygen introduced into the reduction unit.

5.2. Process for conversion of above raw materials into a form for use in reduction

The coal is used as received, charged with the ore into the reduction unit.

The natural gas is partially burned; the gases evolved ($\text{CO}_2 + \text{H}_2\text{O}$) when mixed with the rest of the natural gas cause dissociation to $\text{CO} + \text{H}_2$; these operations take place in the equipment described under 3.2. Liquid hydrocarbons are used in a similar plant or directly in the furnace.

5.3. Condition and analysis of reducing agent at moment of use

The reductant is fine-grained: 60 - 95% fixed carbon; 5/15 mm coal may be used in the semi-industrial plant. The natural gas is reformed as under 5.2.

5.4. Preheat temperature on entering reduction apparatus

The coal is charged cold; the gases from the reforming of the natural gas enter the rotary retort at 1250°C .

5.5. Gas velocity in reduction apparatus

The mean velocity of the gases in the furnace is 1.5 - 2 m/sec.

5.6. Analysis and temperature of gas after passing through reduction apparatus

$$\text{Off-gases } \frac{\text{CO}_2}{\text{CO} + \text{CO}_2} = 80 - 85\%; \text{ H}_2 : 0.2 - 0.4\%$$

6. Ores and ferruginous materials

6.1. Source

Iron ores and complex ores of various origins, pre-reduced products, iron powders and sponge iron and scrap.

6.2. Analysis

Iron content varying from 30% for lean ore to 92% for iron powders and 100% for scrap.

6.3. Size

Preferably 1-25 mm for the semi-industrial plant

6.4. Preparation

Brought to desired size, pre-heating where necessary.

7. Additions

7.1. Material

Addition of lime where necessary to control the basicity index.

7.2. Analysis

No details

7.3. Size

No details

8. Temperature zones in reduction
in °C in the reduction apparatus

Reduction starting around 900-1000°C and continuing up to 1300°C.

9. Pressure zones in reduction
in kg/cm² in the reduction apparatus

Atmospheric pressure or slight negative pressure

10. Product
 of the reduction apparatus

10.1. Form

Liquid iron or steel

10.2. Analysis

Analysis depends on the raw materials charged and the refining.
The carbon may be reduced to 0.02%.

10.3. Degree of reduction

10.3.1. Proportion of oxygen removed from iron oxides as a percentage
 of the total oxygen combined with iron in the ores.

94% for lean ore - 97% for rich ore

10.3.2. Metallic iron as a percentage of the total iron content of
 the product

100%

11. Further treatment of the product (10) of the reduction apparatus

None

12. Final product obtained after treatment according to 11

Same as described under 10

13. Iron yield

13.1. Iron contained in product (10) as a percentage of iron
 in the charge

91% for lean ore, 93% for rich ore,
94% for wustite, 95% for iron powders

- 13.2. Iron contained in final product (12) as a percentage of iron in the charge

Same as 13.1.

14. Utilisation of final product

Destination of final product (blast furnace, electric or other furnace). Technical and economic differences compared with the use of pig iron and scrap.

Liquid iron or steel

15. Consumption data per tonne of final product (12)

Two examples are given in Table 1 for two different methods of operation

16. Overall capital investment

No details

17. Opinions expressed on the process in relation to:

- 17.1. State of development

The process has passed from the pilot to the semi-industrial scale for raw materials containing more than 60% total iron.

- 17.2. Practical operating difficulties

There are no practical difficulties in the operational process: a certain degree of automation may be envisaged through linking the oxygen and methane flow rates to the analysis of the off-gases.

17.3. Possibilities of exploitation

The process has the advantage that it can use coal instead of coke and produce steel directly from ore and coal. The use of natural gas or hydrocarbons gives the plant great flexibility. Capital expenditure is limited and consumption rates are not uneconomic, particularly for the pre-reduced products. However, the form of the unit does not at present allow sufficiently economic reduction of very lean ores (the slag weights per tonne produced are too great and limit production).

18. Bibliography

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18.2. Patents

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France, 29 July 1959, No. 1 240 639

France, 26 July 1963, No. 1 377 329

France, 30 April 1964, No. 1 411 035

France, 16 July 1964, Patent of addition No. 86 154/1 411 035

France, 21 July 1964, Patent of addition No. 86 218/1 411 035

France, 2 December 1964, No. 1 427 045

France, 2 December 1964, No. 1 429 310

France, 16 December 1964, No. 1 426 324

France, 11 March 1965, No. 1 436 670

France, 18 March 1965, No. 1 436 799

France, 7 May 1965, No. 1 442 523

Table 1. Comparisons per tonne of liquid metal

No.1: Rich ores		
Product obtained	Iron	Steel (0.10%C)
Ore charge	1.56t, 68.5% iron $\text{Fe}_2\text{O}_3 = 97.4\%$ $\text{SiO}_2 = 0.6\%$ $\text{P} = 0.035\%$	1.59 tonne of same ore
Reductant	0.460 tonne anthracite	0.460 tonne anthracite or 0.385 tonne graphite
Additions	$\text{CaO} = 0.040$ tonne	$\text{CaO} = 0.040$ tonne
Consumption of powdered MgO	0.025 tonne	0.050 tonne
Heat input		
Methane - preheating	60 Nm^3	60 Nm^3
Methane - melting	60 Nm^3	65 Nm^3
Oxygen - melting	380 Nm^3	440 Nm^3
Product		
C %	3	0.1
Mn %	0.1	0.1
Si %	0.1	0.1
P %	0.030	0.022
S %	0.12	0.0050 (graphite) 0.035 (coal)
Off-gases		
$\frac{\text{CO}_2}{\text{CO} + \text{CO}_2}$ ratio	0.85	0.85

Table 1 (Continued)

Product obtained	Iron	Steel (0.10%C)
Mean temperature i.e. mean enthalpy Hourly production (start to tap)	1400°C 1.7 tonne	1500°C 600 Mcal 1.5 tonne
For an industrial-scale plant one would expect per tonne of iron: 370 kg anthracite 260 Nm ³ oxygen 500 Mcal methane or fuel oil		
<u>No.2:</u> Pre-reduced input corresponding to the formula of wustite Product obtained: liquid steel Reductant : 0.360 tonne coal with 87% C Methane - preheating: 60 Nm³ Methane - melting : 50 Nm³ Oxygen - melting : 330 Nm³ Hourly production : 2.0 tonnes (for a magnetite-type input the values are of the order of 0.390 tonnes coal, 60 Nm³ methane for preheating 50 Nm³ methane and 350 Nm³ O₂ for melting hourly production 1.7 tonnes)		

Table 1 (Continued)

As a matter of information, this process is also used to melt iron powder:
for example starting with an iron powder containing 91.5% total iron and 7% oxygen combined with
the iron the following results are obtained:

Product obtained :	liquid steel with very low carbon content
Reductant :	0.110 kg fixed carbon (graphite or coal)
Methane - melting :	35 Nm ³ (charge not preheated)
Oxygen - melting :	220 Nm ³
Hourly production :	3.0 tonnes

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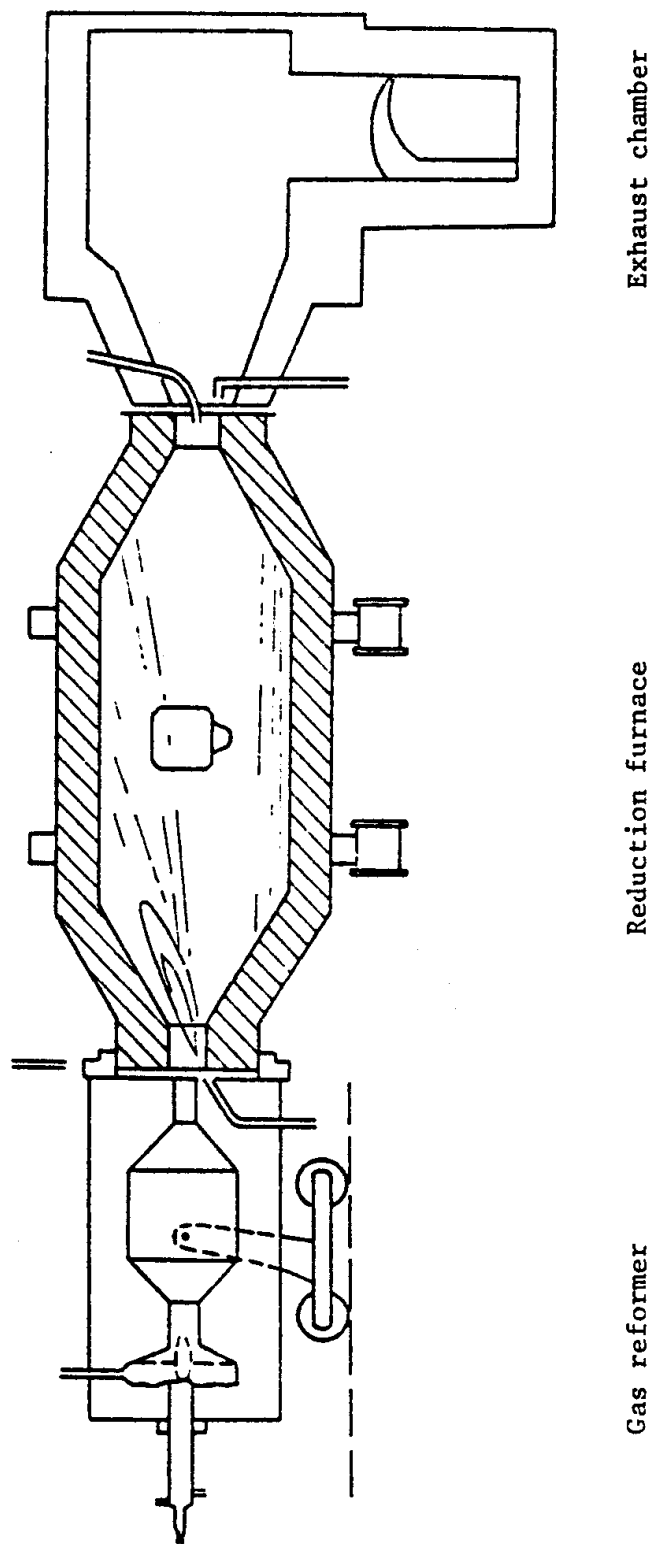


Fig. 1. Bouchet-Imphy Process

CO-C-IRON PROCESS

1. Characteristics

1.1. Name and history of process

CO-C-Iron Process

1.2. Principle of process

Deposition of carbon and pre-reduction of iron ore by means of CO in a fluidised bed. Subsequent final reduction in externally heated chambers.

1.3. Inventors

Dr Hermann Th. Brandi, Mülheim (Ruhr)

Prof. Dr. Ewald Wicke, Hamburg/Münster

1.4. Companies involved in development

Phoenix-Rheinrohr AG, Düsseldorf,

Metallurgy Department, Ruhrort Works.

2. General description: (schematic drawing)

See drawing

3. Plant required

Production of a gas with a high CO content. Fluidised bed furnace for deposition of carbon and pre-reduction (1st stage). Chamber furnace for final reduction (2nd stage) followed in some cases by briquetting.

4. State of development

4.1. Name of laboratories where the process is situated.

Phoenix-Rheinrohr AG, Düsseldorf, Metallurgy Department,
Ruhrort Works.

4.2. Pilot plant

4.2.1. Location

Experimental plant under construction

4.2.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

Fluidised bed reactor about 6m high, 1m in diameter, chamber furnace with similar dimensions.

4.2.3. Capacity in tonnes per 24 h of final product

About 2 to 5 t.p.d. of iron

4.3. Industrial installations

None

5. Reducing agents

5.1. Raw materials on which the reducing agent is based

Non-coking coal, heavy oil

5.2. Process for conversion of above raw materials into a form for use in reduction

Gasification

5.3. Condition and analysis of reducing agent at moment of use

No details

5.4. Preheat temperature on entering reduction apparatus

No details

5.5. Gas velocity in reduction apparatus

30 to 50 cm/s

5.6. Analysis and temperature of gas after passing through reduction apparatus

550°C; approximately 40% CO₂, 40% CO, 20% H₂.

6. Ores and ferruginous materials

6.1. Source

Ore fines with a high Fe content, mainly from Venezuela and Labrador.

6.2. Analysis

65% Fe

6.3. Size

About 0 to 2 mm

6.4. Preparation

None

7. Additions

None

8. Temperature zones in reduction
in °C in the reduction apparatus

1st stage approximately 600°C, 2nd stage approximately 900°C.

9. Pressure zones in reduction
in kg/cm² in the reduction apparatus
1.1 kg/cm²
10. Product
of the reduction apparatus
 - 10.1. Form
Reduced iron powder
 - 10.2. Analysis
Approximately 92% Fe, in some cases 4%C in addition to gangue
 - 10.3. Degree of reduction
 - 10.3.1. Proportion of oxygen removed from iron oxides as a percentage of the total oxygen combined with iron in the ores
95 to 98%
 - 10.3.2. Metallic iron as a percentage of the total iron content of the product
95 to 98%
 - 10.4. Utilisation of by-products
No details
11. Further treatment of the product (10) of the reduction apparatus
 - 11.1. Method of cooling
No data

11.2. Crushing, compacting or briquetting

In some cases briquetting

11.3. Details of final size grading

No details

11.4. Method of concentration (e.g. magnetic separation or sintering)

No details

12. Final product obtained after treatment according to 11

12.1. Form

Reduced iron powder, same size as ore used

12.2. Analysis

No details

13. Iron yield

13.1. Iron contained in product (10) as a percentage of iron in the charge

Approximately 100%

13.2. Iron contained in final product (12) as a percentage of iron in the charge

No details

14. Utilisation of final product

Destination of final product (blast furnace, electric or other furnace). Technical and economic differences as compared with using pig iron and scrap

Steelmaking

15. Consumption data per tonne of final product (12)

No details

16. Overall capital investment

No details

17. Opinions expressed on the process in relation to:

17.1. State of development

No further development

17.2. Practical operating difficulties

No details

17.3. Possibilities of exploitation

No details

17.4. Special requirements to be met by certain materials
(e.g. refractories)

No details

18. Bibliography

18.1. Articles and publications

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- 2) Wicke, Hedden, Lültz: Reduction of fine ores in two stages on a basis of carbon monoxide (CO-C-iron process). Stahl u. Eisen 79 (1959) pp. 129-34.
- 3) Lülth, F: Reduction of iron ores with carbon monoxide via deposited carbon as an intermediate stage. Thesis, Hamburg, 1959.
- 4) Graue, Willems, Artificial scrap. Umschau 60 (1960) pp. 13-15.

18.2. Patents

No details

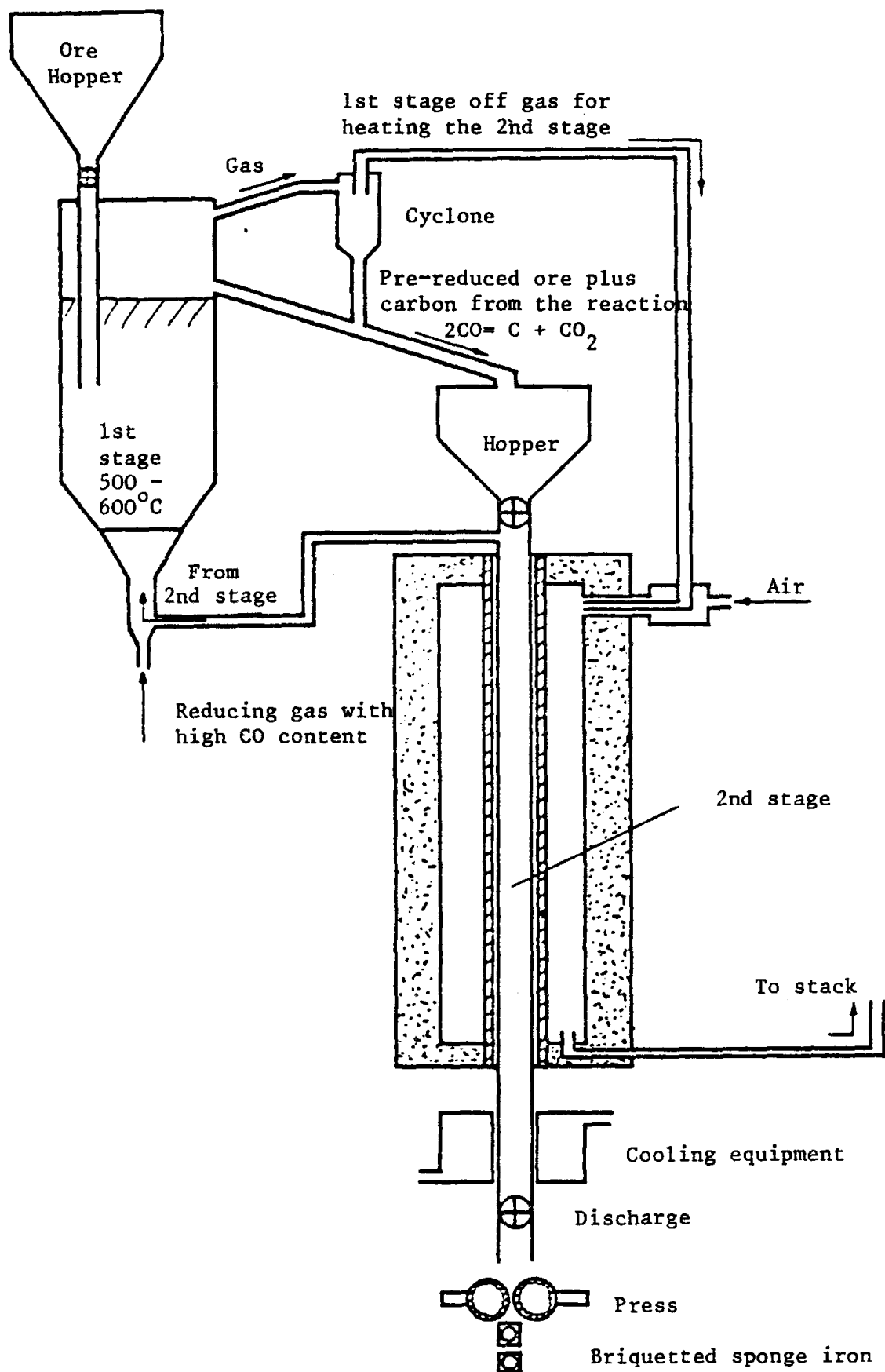


Fig. 1. CO-C- Iron process.

DE SY PROCESS

1. Characteristics

1.1. Name and history of process

De Sy process (process of direct reduction in a two-hearth or multi-hearth furnace induction heated at mains frequency).

1.2. Principle of process

Induction-heated, two-hearth furnace. The primary winding fed at the mains frequency is water-cooled. The secondary winding is formed by the metal bath. The bath undergoes vigorous stirring. Production takes place in two simultaneous stages - reduction of the iron oxides in hearth 2 (Fig.1) and carburisation of the liquid iron in hearth 1 in a strongly reducing medium. This latter hearth contains a large excess of carbon partially immersed in the iron by means of a plunger. Hearth 2 is charged with cold ore, pre-heated or even pre-heated and pre-reduced ore, reduced by the recovered gases.

The furnace is charged progressively. During tapping 25-30% of the metal produced must be retained to start off the next heat.

1.3. Inventors

Professor A de Sy, Rijksuniversiteit, Gent (Belgium).

1.4. Companies involved in development

Metallurgy Laboratory of the Rijksuniversiteit.

2. General description (schematic drawing)

The diagram, Fig.1, refers to the pilot furnace; the industrial scale furnace would be best fixed and with continuous withdrawal;

it would have a more or less rectangular bath, very long and narrow. It would be supplied with 3-phase current (Fig.2).

3. Plant required

Two or multi-hearth furnace, induction heated, as appropriate complemented by a rotary furnace for pre-heating or pre-reduction by the recovered gases, and/or by carbon mixed with the ore.

4. State of development

4.1. Name of laboratories where the process is situated

Laboratorium voor Metallurgie en Metallografie, Rijksuniversiteit, Gent, Belgium.

4.2. Pilot plant

4.2.1. Location

Gent, Belgium

4.2.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

The 150 kVA pilot furnace has a reduction shaft of oval cross-section 0.4 x 0.7 m.

4.2.3. Capacity in tonnes per 24 h of final product

1.2 t per 24 hours, with cold ore (Fe_2O_3) charge; up to 4.8 t per 24 hours with pre-heated and pre-reduced charge (depending on degree of pre-reduction).

4.3. Industrial installations

None

5. Reducing agents

The reducing agent should be rich in carbon and low in ash and sulphur, for example anthracite 0-5 mm or charcoal.

6. Ores and ferruginous materials

6.1. Source

Very rich iron ore

6.2. Analysis

In the discontinuous process (tilting furnace, progressive filling; tapping approximately 70% of contents) the iron content of the ore should be > 65% for a neutral or basic gangue and > 67-68% for an acid gangue. In the continuous process (fixed furnace with continuous discharge of metal and slag) the iron content of the ore should be at least 60%.

6.3. Size

Fines or ore crushed at least to 25 mm

6.4. Preparation

Crushing and screening as under 6.3.

7. Additions

Lime, where the ore is siliceous

8. Temperature zones in reduction
in °C in the reduction apparatus

In the reduction phase the bath temperature is best kept between 1500 and 1600°C.

9. Pressure zones in reduction
in kg/cm² in the reduction apparatus

Atmospheric
10. Product
of the reduction apparatus
 - 10.1. Form

Liquid iron or steel
 - 10.2. Analysis

The tests gave the following results:

Iron: 4.45% C, 0.18% Mn, 0.055% P, 0.011% S.
Tool steel: 0.85% C, 0.42% Mn, 0.065% P, 0.012% S, 0.23% Si.
 - 10.3. Degree of reduction

High, but no figures
 - 10.4. Utilisation of by-products

Possible utilisation of gases from the reduction shaft for pre-heating or pre-reduction of the ores.
11. Further treatment of the product (10) of the reduction apparatus

None
12. Final product obtained after treatment according to 11

See 10.

13. Iron yield

See 10

14. Utilisation of final product

Destination of final product (blast furnace, electric or other furnace). Technical and economic differences as compared with using pig iron and scrap

Iron for conversion or steels for direct use

15. Consumption data per tonne of final product (12)

15.1. Raw materials for reduction (5.1)

1.6 - 1.7 t rich hematite per tonne of product.

15.2. Fuels and thermal values required for the entire process
(fuel characteristics) (thermal balance)

350 kg carbon with cold ore.

380 kg carbon with ore pre-reduced to 33% and heated to 970°C.

420 kg carbon with ore pre-reduced to 70 - 75% and heated to 970°C.

15.3. Electrical energy (heating)

1830 kWh with cold ore.

1010 kWh with ore pre-reduced to 33%.

610 kWh with ore pre-reduced to 70 - 75%.

15.4. Steam used

None.

15.5. Oxygen

None.

15.6. Cooling water

No details

15.7. Refractories

No details

15.8. Man hours of productive labour

No details

15.9. Labour and materials for routine maintenance

No details

16. Overall capital investment

No details

17. Opinions expressed on the process in relation to:

17.1. State of development

The process, studied only on the laboratory scale, has been abandoned.

17.2. Practical operating difficulties

Limitation of slag volume (otherwise boil too vigorous).

17.3. Possibilities of exploitation

Possibility of continuous working, provided the production capacity is adequate; minimum of 2 - 4 tonnes per hour. At this rate a continuous flow of iron and slag becomes possible (see fig. 2 and reference 18.1.4).

Possible applications:

- production of iron for conversion into steel by one of the conventional methods.
- production of very pure hematite iron.

17.4. Special requirements to be met by certain materials (e.g. refractories)

In furnace 2 the refractory is in contact with a very oxidising slag. It is advisable to use magnesite bricks.

18. Bibliography

18.1. Articles and publications

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- 2) De Sy, Electric direct reduction process, Journal of Metals, April 1959. 11 (4) pp. 265-69.
- 3) De Sy, "New electrical process for direct reduction of iron ore", Revue de métallurgie, December 1958, 55. No.12, pp. 1161-1170.
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18.2. Patents

No details

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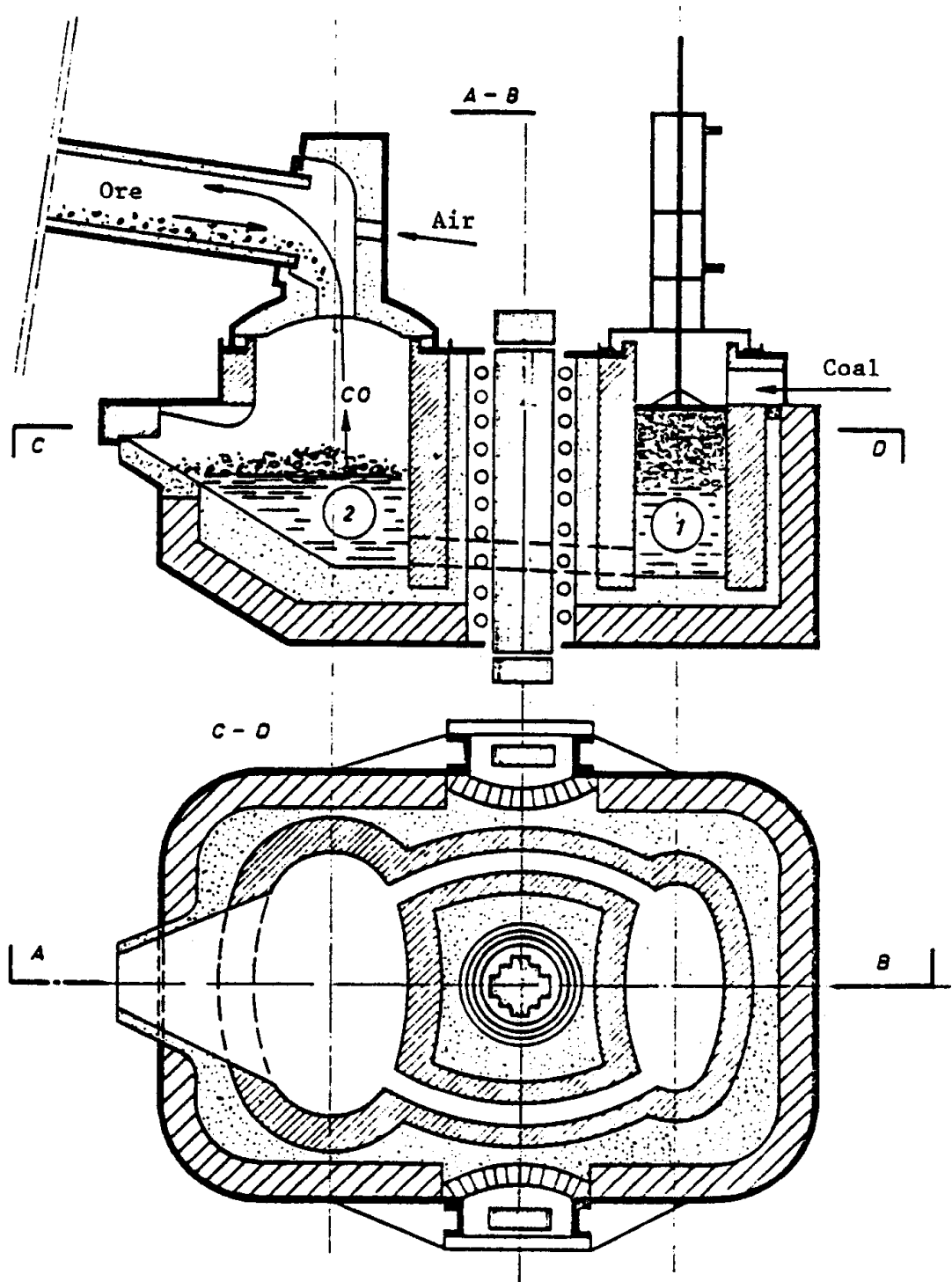


Fig.1. De Sy Process

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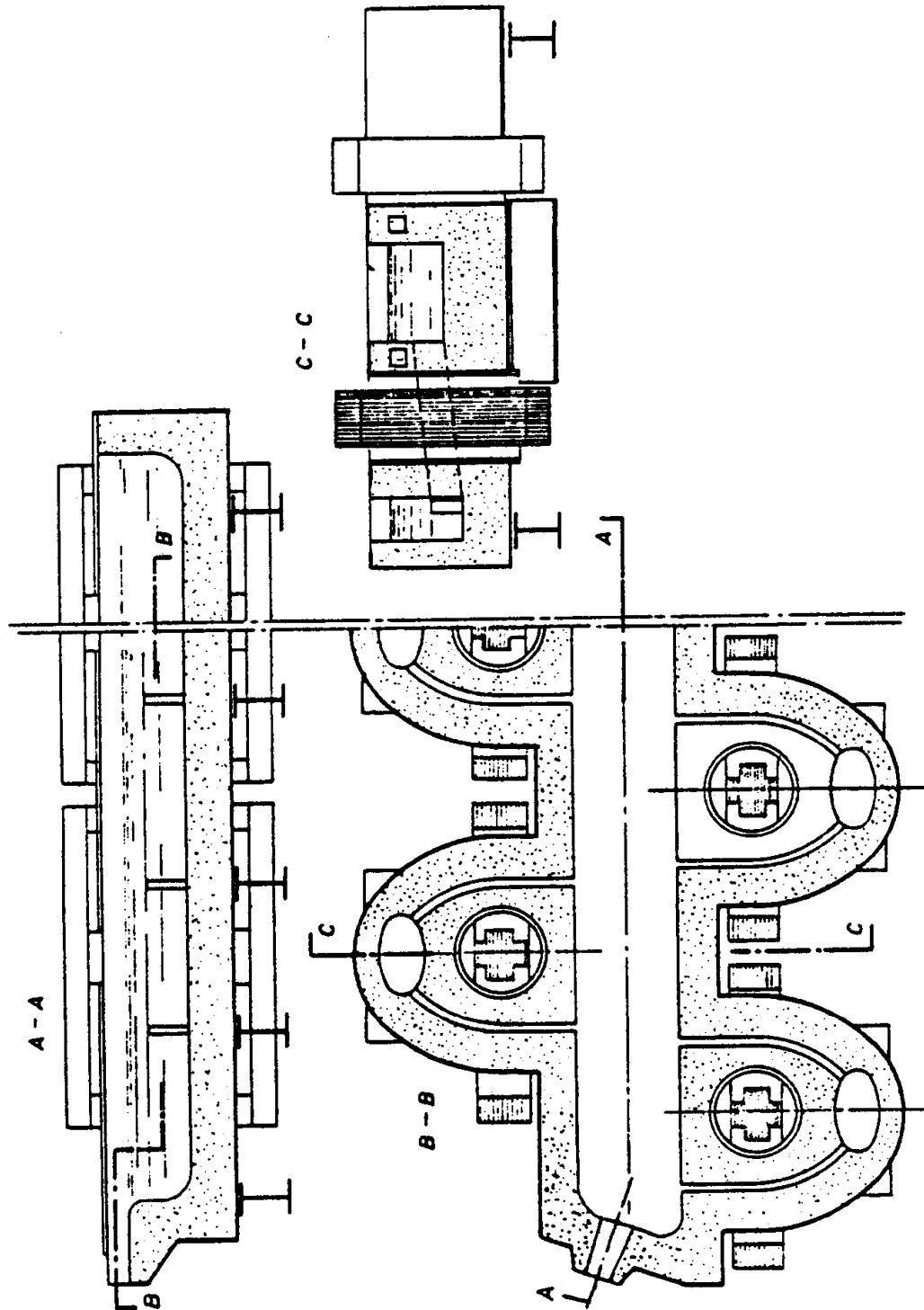


Fig.2. De Sy Process.

D.L.M. PROCESS

1. Characteristics

1.1. Name and history of process

D.L.M. (Dwight-Lloyd-McWane) process

1.2. Principle of process

Pre-heating and pre-reduction on a Dwight-Lloyd travelling grate; final reduction and melting in an electric furnace.

1.3. Inventors

Samuel L. Case, McWane Cast Iron Pipe Co., Birmingham (1957).

1.4. Companies involved in development

McDowell Company Inc. (Dwight-Lloyd Division), 113, St.Clair Ave., N.E., Cleveland 14, Ohio (USA).

2. General description (schematic drawing)

See diagram

The process comprises the following operations:

- 1) Preparation of the charge by mixing coal, ore fines and additions (previously crushed to 10 mesh, i.e. 1-65 mm) then wet grinding (to about 100 mesh, i.e. 0.15 mm), thickening and filtration. The filter cakes then receive the addition of recycled material and are "conditioned" (that is broken down and mixed in a drum with a shaft with paddles) then disc pelletised.

- 2) Sintering and carbonising of pellets on a continuous strand of Dwight-Lloyd type comprising drying, ignition and carbonising sections.
- 3) Screening the hot sinter and charging into an electric furnace where reduction is completed and melting takes place.

3. Plant required

- a) A Dwight-Lloyd sinter strand with preparation plant for crushing, pelletising, pre-drying hood, carbonisation and recycling of the gases.
- b) Electric melting furnace.

4. State of development

- 4.1. Name of laboratories where the process is situated
MacDowell (see 1.4)

4.2. Pilot plant

4.2.1. Location

Cleveland, Ohio, USA

4.2.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

Sinter strand 2 ft. wide. 15½ ft long (0.60 x 4.80 m) - building housing installations approximately 15 x 15 m.

4.2.3. Capacity in tonnes per 24 h of final product

Approximately 8 tonnes of iron per day.

4.3. Industrial installations

A project was being developed in 1967 for the MacWane Cast Iron Pipe Co.

4.3.1. Location

Mobile, Alabama, USA

4.3.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

No details

4.3.3. Capacity in tonnes per 24 h of final product - per unit of production

500 - 700 tonnes of iron per day

4.3.4. Method of operation (continuous or batch, number of operating hours per day)

Continuous operation

5. Reducing agents

5.1. Raw materials on which the reducing agent is based

Non-coking coals

5.2. Process for conversion of above raw materials into a form for use in reduction

Mixing with ore fines to be reduced and additions (proportion 1/3 coal, 2/3 ore), wet ground fine (100 mesh = 0.15 mm), thickening and filtration.

5.3. Condition and analysis of reducing agent at moment of use

Solid state, finely ground (100 mesh, i.e. 1.5 mm)

5.4. Preheat temperature on entering reduction apparatus

Preheating takes place in the unit itself (Dwight Lloyd continuous strand)

6. Ores and ferruginous materials

6.1. Source

Tests have been carried out on a number of ores, the results quoted being obtained with Mesabi ores.

6.2. Analysis

Ore with 51.5% Fe as received (8% water)

6.3. Size

Ground to 10 mesh (1.65 mm)

6.4. Preparation

Mixing with coal and additions then grinding (5.2) thickening and filtration. To the filter cakes are added recycled fines, they are broken down and mixed in a drum and then finally pelletised on a disc to pellets of about $\frac{1}{2}$ " diameter.

7. Additions

7.1. Material

Limestone

7.2. Analysis

No details

7.3. Size

As 6.3.

8. Temperature zones in reduction
in °C in the reduction apparatus

Approximately 1100°C in the bed on the Dwight-Lloyd strand.

9. Pressure zones in reduction
in kg/cm² in the reduction apparatus

Not given but near atmospheric pressure

10. Product
of the reduction apparatus

This is a process in two stages - pre-heating and pre-reduction on a grate followed by completion of reduction and melting in an electric furnace.

10.1. Form

Hot pellets at about 900°C after pre-reduction: liquid iron after melting phase.

10.2. Analysis

No details

10.3. Degree of reduction

10.3.1. Proportion of oxygen removed from iron oxides as a percentage of the total oxygen combined with iron in the ores.

The ore is pre-reduced to 55 or 60% before being charged into the electric furnace.

10.3.2. Metallic iron as a percentage of the total iron content of the product

No data for the pre-reduced product, 100% for the iron

10.4. Utilisation of by-products

No details

11. Further treatment of the product (10) of the reduction apparatus

No treatment

12. Final product obtained after treatment according to 11

12.1. Form

Liquid iron

12.2. Analysis

No details

13. Iron Yield

No details

14. Utilisation of final product

Destination of final product (blast furnace, electric or other furnace). Technical and economic differences as compared with using pig iron and scrap

Similar to a normal pig iron

15. Consumption data per tonne of final product (12)
- 15.1. Raw materials for reduction (5.1)
 850 kg coal
- 15.2. Fuels and thermal values required for the entire process
 (fuel characteristics) (thermal balance)

 Same as 15.1.
- 15.3. Electrical energy

 900 - 1000 kWh for reduction and approximately 70 kWh for
 the rest of the plant, plus 5 kg electrode paste per tonne
 of pig iron.
- 15.4. Steam used

 None
- 15.5. Oxygen

 None
- 15.6. Cooling water

 38 Nm³
- 15.7. Refractories

 No details
- 15.8. Man hours of productive labour

 Approximately 1 h per tonne iron
- 15.9. Labour and materials for routine maintenance

 No details

16. Overall capital investment

No new figures since 1967; then the promoters gave:

Number of units	1	2	3
Tonnes iron per day	362	724	1,086
Tonnes iron per annum	122,000	244,000	366,000
Total cost in millions of \$	7.5	13.5	19.5
Annual cost per tonne iron in dollars	61.5	55.5	53.0

17. Opinions expressed on the process in relation to:

17.1. State of development

No new information since 1967

17.2. Practical operating difficulties

No details

17.3. Possibilities of exploitation

The DLM process forms part of the overall research undertaken to improve the economic conditions in the use of electric pig iron furnaces. The weight of coal to produce one tonne of metal is still very high.

18. Bibliography

18.1. Articles and publications

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- 3) McDowell, R.C. Ironmaking by the Dwight-Lloyd McWane process. Communication Milling and Metallurgical Session of American Mining Congress, Las Vegas, 13 October 1960.
- 4) Ban, T.E., and Worthington B.W., Technology of Dwight-Lloyd McWane Ironmaking. Communication Metallurgical Society of AIME, Philadelphia, 18 October 1960, quoted in Mining Journal, 4 November 1960, p. 505 Summarised in Journal of Metals, 12 December 1960, pp. 937-939.
- 5) Ban, T.E., Worthington, B.W., and Thompson, C.D., New Process for Foundry Iron, American Foundrymen's Society Transactions, volume 69, 1961.

18.2. Patents

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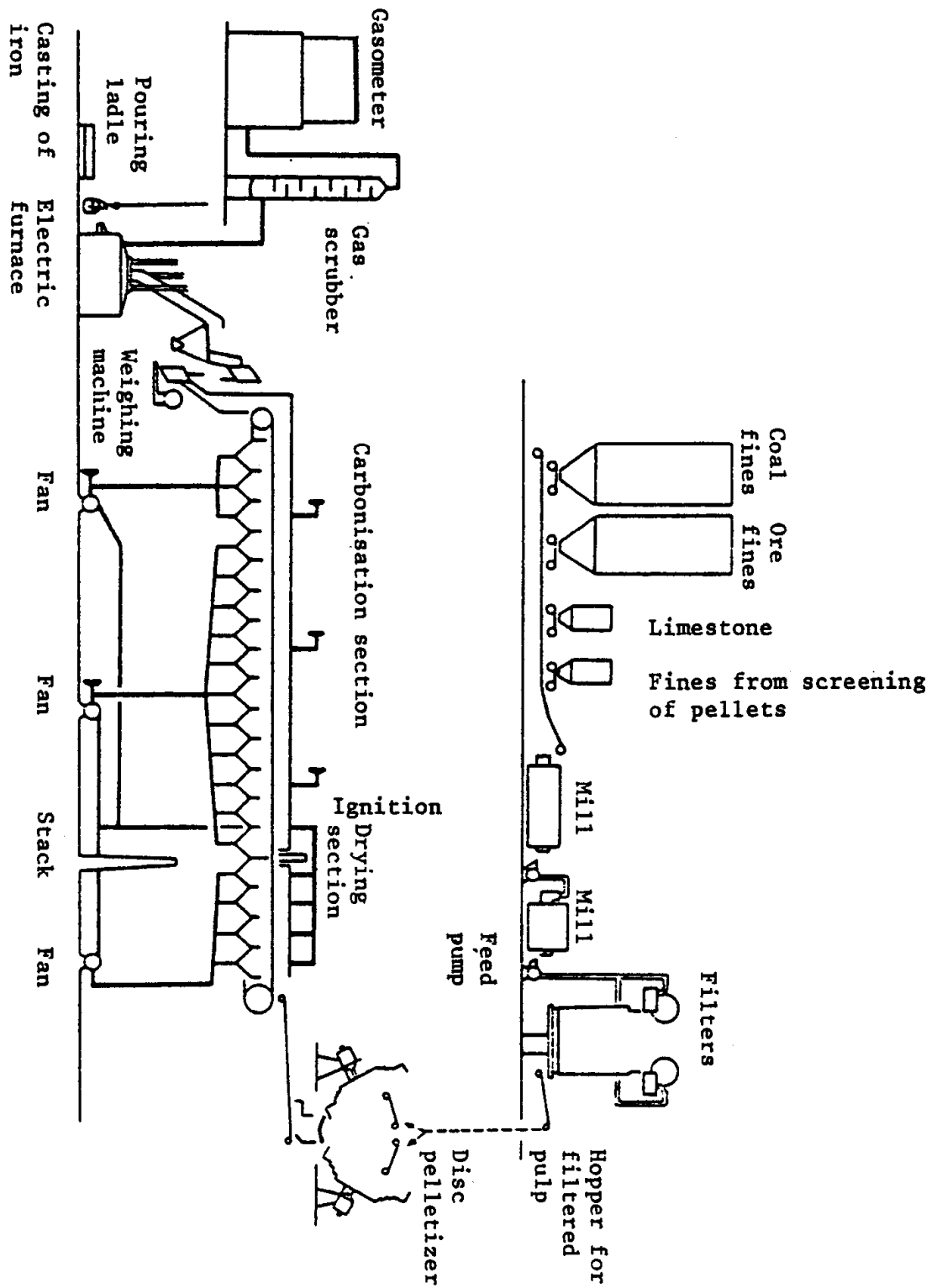


Fig. 1. Dwight-Lloyd-MacWane

DORED PROCESS

1. Characteristics

1.1. Name and history of process

Dored (Dommarvet Reduction) process.

Process developed from the Kaldo process.

In the Kaldo steelmaking process a given quantity of ore is added to lower the temperature of the bath; the iron oxides in this ore react with the carbon dissolved in the melt. A similar practice has been followed in the Dored process to make iron from iron ores and solid carbon added in the form of coke, anthracite or any coal.

1.2. Principle of process

In the Dored process the molten metal is produced directly from ores and a solid reductant in a Kaldo-type rotary furnace. The ore is reduced in a reactive layer above the slag (Figure 1).

1.3. Inventors

No details

1.4. Companies involved in development

Metallurgical Research Centre of Stora Kopparbergs Bergslags A.B at Dommarvet, Sweden.

2. General description (schematic drawing)

See figures 1 and 2.

3. Plant required

Rotary reduction furnace

Rotary furnace for pre-heating the ores by means of the gases from the reduction furnace.

Oxygen lance

Desulphurisation ladle where necessary.

Bunkers, weigh hoppers and conveyors for ore, coke and lime.

4. State of development

4.1. Name of laboratories where the process is situated

The process has been developed at the Metallurgical Research Centre of Stora Kopparbergs Bergslags at Domnarvet, Sweden.

4.2. Pilot plant

4.2.1. Location

The tests have been carried out at Domnarvet.

4.2.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

A rotary furnace of 3 tonnes nominal capacity.

Refractory lining of tar-bonded dolomite.

Useful diameter inside refractories 1.5 m.

Useful length 2.1 m.

Speed of rotation: approximately 20 r.p.m.

4.2.3. Capacity in tonnes per 24 h of final product

Production of hot metal: approximately 12 t per 24 hours.

The tests were carried out for a limited period on the Kaldo furnace at the Domnarvet steelworks. The furnace had a capacity of 30 t. The capacity during the tests was 108 tpd.

4.3. Industrial installations

4.3.1. Location

The industrial plant was at Domnarvet. It was commissioned in the summer of 1965 and dismantled in 1971.

4.3.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

Capacity of reduction furnace 25 t.

Speed of rotation: up to 30 r.p.m.

Very small angle of inclination of furnace.

4.3.3. Capacity in tonnes per 24 h of final product - per unit of production

Productivity estimated at 150 tonnes of hot metal per day = 50,000 t.p.y.

4.3.4. Method of operation (continuous or batch, number of operating hours per day)

Essentially continuous process.

The metal and slag are tapped every 4 - 5 hours.

4.3.5. Coefficient of utilisation in time (number of working days per annum)

No details

4.3.6. Maximum rated production

No details

5. Reducing agents

The reducing agent is solid; it is generally small coke or coke fines, introduced at ambient temperature. Anthracite may be suitable; other types of coal still have to be tested.

6. Ores and ferruginous materials

6.1. Source

Any rich or concentrated ore may be suitable.

6.2. Analysis

Example of analysis of ore processed in the pilot plant:

61.1% Fe
6.1% SiO₂
0.4% P

6.3. Size

Fine ores up to 10 mm or concentrates

6.4. Preparation

No special preparation is required (neither sintering nor pelletising).

7. Additions

Additions of lime to adjust the composition of the slag.

8. Temperature zones in reduction
in °C in the reduction apparatus

1300 - 1350°C

9. Pressure zones in reduction
in kg/cm² in the reduction apparatus

Atmospheric pressure

10. Product
of the reduction apparatus

10.1. Form

Hot metal

10.2. Analysis

Example of analysis: C 3.62%
Si 0.01%
Mn 0.09%
S 0.124%
P 0.027%

The very flexible operation of the reduction process allows the analysis of the metal produced to be controlled provided a suitable slag composition is chosen. It is possible to produce an iron with a very low P content from phosphoric ores. The sulphur content may then be high, necessitating desulphurisation in the ladle.

10.3. Degree of reduction

Very high (liquid iron)

10.4. Utilisation of by-products

All the carbon monoxide produced during the reduction is burned by blowing oxygen into the reduction furnace. The heat of combustion liberated is transferred to the bath partly by radiation and partly by contact with the hot walls. The sensible heat of the gases leaving the reduction furnace is recovered in a rotary furnace for drying and pre-heating the ores (up to 700°C).

11. Further treatment of the product (10) of the reduction apparatus

None

12. Final product obtained after treatment according to 11

See under 10.

13. Iron yield

The yield of the operation depends on the procedure adopted:
when making low-phosphorus iron from ores containing phosphorus,
the iron content of the slag may reach 4%.

Composition of the slag for an iron of the analysis given under
10.2:

CaO	40.3%
SiO ₂	28.3%
MnO	2.9%
FeO	5.5%

The dust from the off-gases (recovered by cyclone) contains
3 - 4% of the iron charged.

14. Utilisation of final product

Destination of final product (blast furnace, electric or other
furnace). Technical and economic differences as compared with
using pig iron and scrap.

The liquid iron is converted into steel in a Kaldo converter
at the Domnarvet steelworks.

15. Consumption data per tonne of final product (12)

- 15.1. Raw materials for reduction (5.1)

The consumption depends on their Fe content

15.2. Fuels and thermal values required for the entire process
(fuel characteristics) (thermal balance)

Coke for reduction and carburisation: 377 kg per tonne of iron.

Coke for heating: 43 kg per tonne of iron.

The thermal balance of the process covers two items:

- almost complete combustion of the CO formed during reduction, by blowing-in oxygen above the bath; recovery of part of the heat in the bath through rotation of the furnace,
- pre-heating of the ore in a conventional rotary furnace by the off-gases from the reduction furnace.

15.3. Electrical energy

For rotation: 4 100 kW motors (30 tonne furnace).

Total electricity, including manufacture of the oxygen,
300 - 400 kWh per tonne iron.

15.4. Steam used

None

15.5. Oxygen

350 Nm³ oxygen per tonne iron (purity of oxygen: 95%)

15.6. Cooling water

No details

15.7. Refractories

See 17.

15.8. Man hours of productive labour

No details

15.9. Labour and materials for routine maintenance

No details

16. Overall capital investment

No recent information

17. Opinions expressed on the process

The consumption of refractories in the industrial plant proved much higher than originally expected. This problem was not convincingly solved in spite of special studies and the industrial plant was shut down in 1968.

18. Bibliography

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18.2. Patents

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C8

Ore (can be pre-heated) and coke

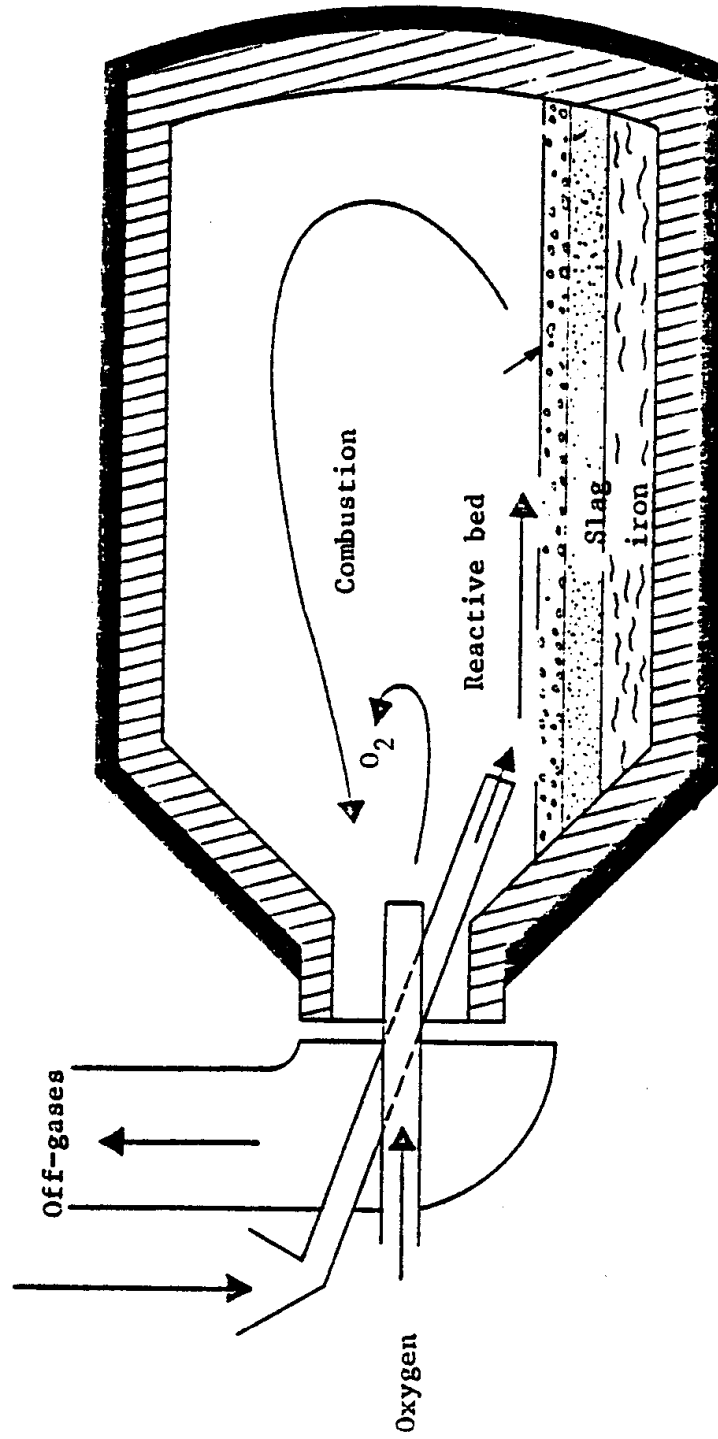


Fig. 1. Dored Process

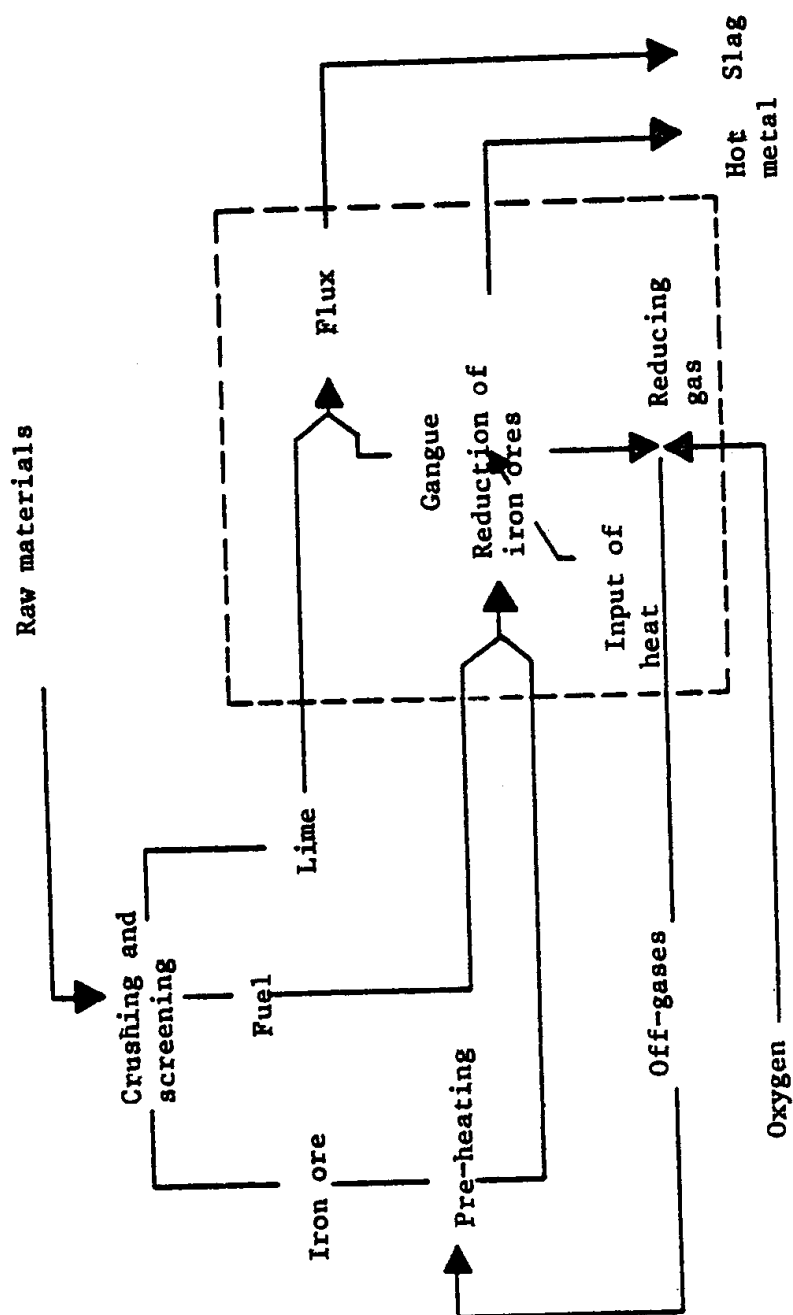


Fig. 2 Dored process

ELECTRIC FLUIDISED - BED PROCESS

1. Characteristics

1.1. Name and history of process

Electric Fluidised - Bed Process
(Wenzel/Schenck Process)

The electric fluidised-bed process was developed by the Institut für Eisenhüttenwesen of the Rheinisch-Westfälische Technische Hochschule, Aachen (Institute of Metallurgy, Rhineland-Westphalian Technical University) after numerous investigations in the field of reduction in fluidised beds and of the physical, particularly electrical behaviour of a directly heated fluidised bed. From 1959 until the present day, development work has led to an experimental furnace (250 kVA) and then to a semi-industrial furnace (400 kVA).

1.2. Principle of process

A fluidised bed consisting of fuel ranging from powder form to fine grained material is heated by means of electrical resistance with immersed electrodes. Ore in powder or fine grained form is fed to the electric fluidised bed from above and is reduced and melted as it falls through the very hot fluidised carbon. The carbon monoxide which forms acts as the gas component for maintaining the fluidised bed. Sintering of the ore is prevented by the excess carbon present (heterogeneous fluidised bed). Stability of the fluidised bed is achieved through the quantity of reducing gas which is dependent on the throughput (autogenous fluidised bed); no external source of gas is required.

1.3. Inventors

Professor Dr.-Ing. Werner Wenzel
Professor Dr.-Ing. Dr.-Ing. h.c. Hermann Schenck

1.4. Companies involved in development

DEMAG-Elektrometallurgie, Duisburg

Gesellschaft für Elektrometallurgie, Düsseldorf, Nürnberg

Stolberger Zink AG, Aachen

2. General description (schematic drawing)

See drawing

3. Plant required

See drawing

4. State of development

4.1. Name of laboratories where the process is situated

Institut für Eisenhüttenwesen of the Rheinisch-Westfälische
Technische Hochschule in Aachen.

4.2. Pilot plant

4.2.1. Location

Institut für Eisenhüttenwesen of the Rheinisch-Westfälische
Technische Hochschule, Aachen
Stolberger Zink AG, Aachen

4.2.2. Dimensions of apparatus and plant (e.g. height/length and
diameter of the shaft of a reduction furnace)

crucible and shaft diameter	900 mm
height of furnace chamber	1800 mm
hearth area	0.64 m ²
height of fluidised bed	300 to 500 mm
reaction chamber	0.2 to 0.3 m
volume of fluidised bed	

4.2.3. Capacity in tonnes per 24 h of final product

Approximately 2.5 t pig iron/24 hours

Approximately 1.2 t ferrochromium (75% Cr)/24 hours

Approximately 2.0 t zinc/24 hours

4.3. Industrial installations

No operating installation, small-scale plant see 4.2.

5. Reducing agents

5.1. Raw materials on which the reducing agent is based

Anthracite, bituminous coals with varying volatile contents,
such as: lignite, coke, low-temperature coke and charcoal

5.2. Process for conversion of above raw materials into a form
for use in reduction

Usually no pretreatment

5.3. Condition and analysis of reducing agent at moment of use

Ranging from powdered to fine grained, size 0 to 1 mm dia.

Ash content not critical) does however affect
Content of volatile substances not critical) consumption of
electricity

Reducing gases from volatile constituents are present as CO and
H₂ (CH₄ < 2%) and are desirable for the prereduction.

5.4. Preheat temperature on entering reduction apparatus

Plant for preheating to 800°C and prereduction plant are
under construction.

5.5. Gas velocity in reduction apparatus

15 to 25 cm/s (1500°C) depending on particle size of carbon

5.6. Analysis and temperature of gases after passing through reduction apparatus

Temperature 1000 to 1400°C depending on temperature of fluidised bed

60 to 90% CO)

2 to 10% CO₂) depending on charge and temperature

5 to 30% H₂) of fluidised bed

0 to 2% CH₄)

6. Ores and ferruginous materials

6.1. Source

Ores from all deposits; also roasted pyrites

6.2. Analysis

Not specified

6.3. Size

0 to 5 mm, 0 to 10 mm possible

6.4. Preparation

No preparation needed with fines; coarse ores have to be crushed

7. Additions

7.1. Material

The usual additions for obtaining a desired slag composition e.g. for desulphurising the pig iron. The furnace has already been operated with highly acidic and highly basic slags.

7.2. Analysis

See 7.1.

7.3. Size

0 to 15 mm

8. Temperature zones in reduction
in °C in the reduction apparatus

The maximum temperature is given by the volatilisation temperature of the molten reaction products. Temperatures up to 1800°C have been obtained.

9. Pressure zones in reduction
in kg/cm² in the reduction apparatus

300 to 500 mm water gauge, higher pressures possible

10. Product
of the reduction apparatus

10.1. Form

Liquid iron, liquid ferroalloys, liquid calcium carbide, zinc vapour

10.2. Analysis

Can be controlled to a large extent by the charge and temperature.

10.3. Degree of reduction

10.3.1. Proportion of oxygen removed from the iron oxides in relation to the total amount of oxygen combined with the iron in the ores

100%

10.3.2. Metallic iron as a percentage of the total iron content of the product

100%

10.4. Utilisation of by-products

600 to 900 Nm² waste gas depending on coal charge.

Composition and quantity of slag depend on the charge, temperature and height of coal fluidised bed.

11. Further treatment of the product (10) of the reduction apparatus

11.1. Method of cooling

Condensation in the case of zinc.

11.2. Crushing, compacting or briquetting

No details

11.3. Details of final size grading

No details

11.4. Method of concentration (e.g. magnetic separation or sintering)

No details

12. Final product obtained after treatment according to 11

12.1. Form

Pig iron, semi-steel, molten zinc, ferroalloy, calcium carbide.

12.2. Analysis

See 10.2.

13. Iron yield

13.1. Iron content in product (10) as a percentage of iron in the charge

Almost 100%, less the slag losses. The proportion carried out with the gas is recycled.

13.2. Iron contained in the final product (12) as a percentage of iron in the charge

See 10.1 and 13.1.

14. Utilisation of final product

Destination of final product (blast furnace, electric or other furnace). Technical and economic differences compared with using pig iron and scrap

Steel production, foundry products, alloys, manufacture of acetylene

15. Consumption data per tonne of final product (12)

The same as for the low shaft electric furnace. No special data.

16. Overall capital investment

16.1. For the plant described in (3) and related to one tonne of capacity for the final product (12)

Capacity expressed in tonnes per day or per year

Not determined, similar to that of the low shaft electric furnace. (The costs per annual tonne will be lower than in a low shaft electric furnace since plant with a capacity above 500 t.p.d. is possible; furthermore, the electricity

consumption per t of end product is lower through prereduction by the waste gas obtained from the furnace; transformer and power losses are lower since furnace voltages used are approximately 3 times higher than those used in the low shaft electric furnace).

16.2. Construction costs of ancillary plant not covered in (3)

In principle, no ancillary installations are needed; preheating and prereduction form a single unit with the furnace installation.

17. Opinions expressed on the process in relation to:

17.1. State of development

Developed to a stage where it can be operated. So far no industrial application.

17.2. Practical operating difficulties

No details

17.3. Possibilities of exploitation

Application of the process covers smelting and reducing all raw materials in powder form or of small size, more particularly for the reduction of oxide ores. It is also possible to produce ferroalloys, calcium carbide and zinc. Further advantages, in addition to processing fine grained ores and using low grade fuels, can be seen in particular in the application of fairly high furnace voltages (>500 V) which provides the possibility of operating large production plants.

Since the electric fluidised bed is not dependent on a high temperature zone because of the favourable heat transfer and material exchange conditions within the fluidised layer, the volatilisation losses usual in the production of ferroalloys do not occur or only occur to a slight degree.

- 17.4. Special requirements to be met by certain materials
(e.g. refractories)

The experimental furnace (4.2.) is operated with graphite electrodes (76 mm dia.); in furnaces with a capacity of more than 3000 kVA Soderberg electrodes would be used.

18. Bibliography

18.1. Articles and publications

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Stresa 1959.
- 2) Wenzel, W.: Klepzig Fachberichte 69 (1961) (10) pp. 361-68.
- 3) Kallfelz, P.: Dr.-Ing.-Diss. RWTH Aachen (1967)

18.2. Patents

Patented in all Western industrial countries; in Germany under
No.1 060 064 dated 10.12.1959.

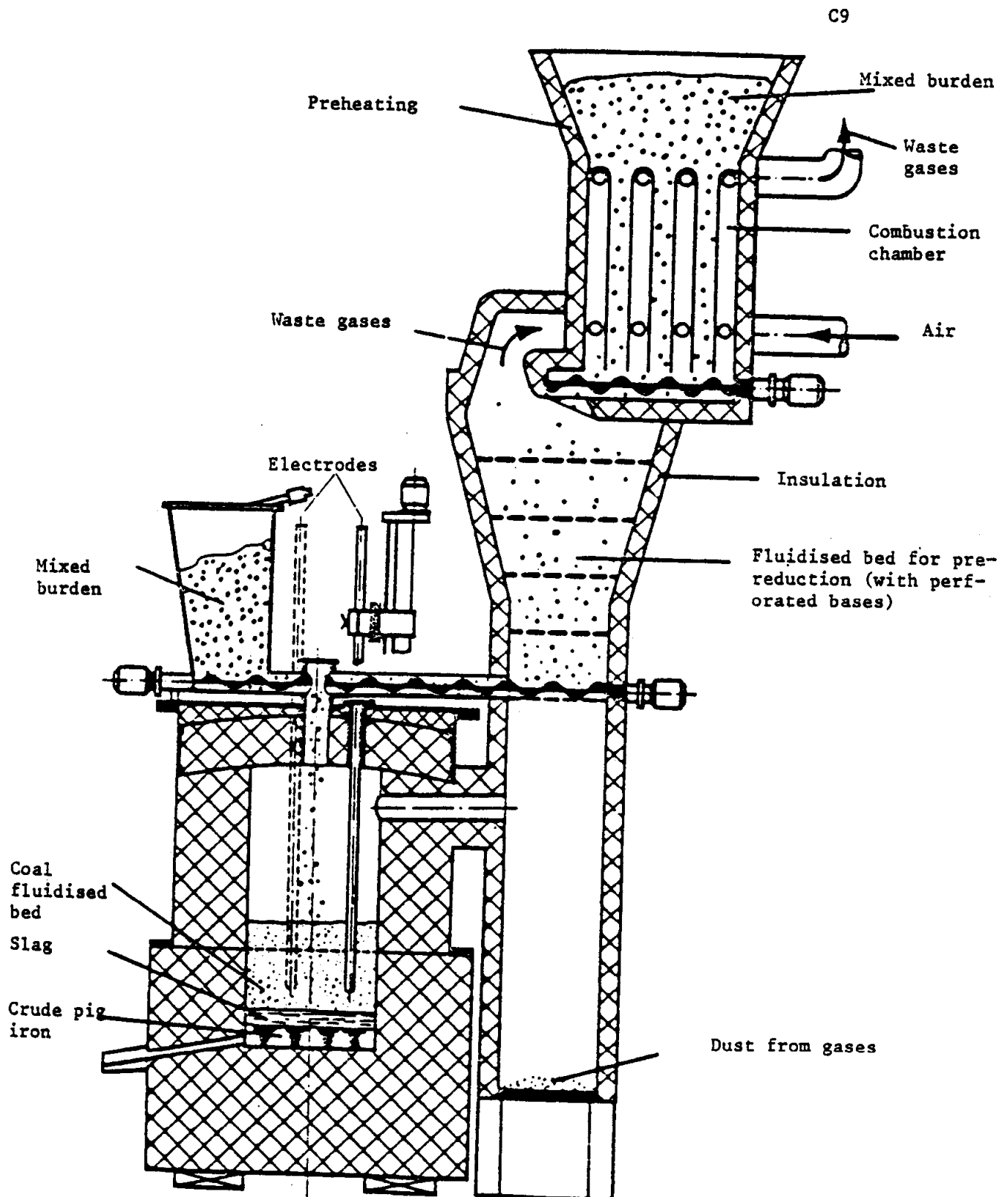


Fig. 1. Electric Fluidised - Bed Furnace

ELKEM PROCESS (with shaft furnace)

1. Characteristics

1.1. Name and history of process

Elkem process (with shaft furnace)

1.2. Principle of process

Pre-heating and pre-reduction of pelletised ores in a shaft furnace feeding an electric smelting furnace.

1.3. Inventors

Elektrokemisk A/S, Oslo, patent filed in Norway, 12th February 1946. Elektrokemisk A/S is now Elkem Spiegeverket A/S (Since 1972).

1.4. Companies involved in development

The process was developed by Elektrokemisk A/S (Elkemhuset, Postboks 5430, Majorstna, Oslo 3, Norway) in its pilot plant at Fiskaa, Christiansand, Norway. The process is used in Norway by Norsk Jernverk and in Portugal by Minas de Vila Cova.

2. General description (schematic drawing)

See Figure 1

3. Plant required

Electric smelting furnace of Tysland-Hole type with modified roof for collecting the gases and receiving the pre-heated and pre-reduced charge in shafts above the furnace.

4. State of development

4.1. Name of laboratories where the process is situated

See 1.4.

4.2. Pilot plant

4.2.1. Location

Fiskaa Verk, Kristiansand S, Norway

4.2.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

A 15 kVA electric arc furnace, monophasic, and several test shaft installations for studying pre-heating and pre-reduction.

4.2.3. Capacity in tonnes per 24 h of final product

Approximately 6 - 7 tonnes per day

4.3. Industrial installations

4.3.1. Location

a) Minas de Vila Cova S.a.r.l. Production halted.

Registered Office: R. Antonio Cardoso, 184 Apartado 135,
Porto;

Works: Campea, Vila Real (Portugal)

b) A/S Norsk Jernverk; pre-reduction and pre-heating at a stand still.

Registered Office: Sorkedalsvein 6, Oslo;

Works: Mo i Rana (Norway).

4.3.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

a) One 13,200 kVA electric arc furnace with 9 shaft furnaces for pre-heating the charge.

- b) Two 60,000 kVA electric arc furnaces each with 9 shafts for pre-heating the charge. External diameter of the arc furnace 16 metres, diameter of electrodes 1.9m.

4.3.3. Capacity in tonnes per 24 h of final product - per unit of production

- a) Production capacity of furnace : 140 tonnes of iron per day.
- b) Production capacity per day : 500 tonnes.

4.3.4. Method of operation (continuous or batch, number of operating hours per day)

Continuous operation, 24 hours a day.

4.3.5. Coefficient of utilisation in time (number of working days per annum)

It is possible to operate 365 days per year.

4.3.6. Maximum rated production

Capacity varies according to the raw material specification (richness in iron, amount of slag formed). Reduction is complete, the end product being iron.

5. Reducing agents

5.1. Raw materials on which the reducing agent is based

High volatile coals.

5.2. Process for conversion of above raw materials into a form for use in reduction

Crushing and mixing with the ores to form pellets. During the final reduction in the electric furnace, these coals

produce a gas which is recovered, purified and partially burned (incomplete combustion) before being fed into the pre-reduction shafts.

5.3. Condition and analysis of reducing agent at moment of use

The gas fed into the pre-reduction shafts has an approximate composition of 10% CO, 25% CO₂ and 65% N₂.

5.4. Preheat temperature on entering reduction apparatus

800 - 900°C.

5.5. Gas velocity in reduction apparatus

No details

5.6. Analysis and temperature of gas after passing through reduction apparatus

150 - 200°C

6. Ores and ferruginous materials

6.1. Source

Rich concentrates.

6.2. Analysis

Of the order of 70% Fe, after pelletising (6.4) 57% Fe, 12% C.

6.3. Size

Fine concentrates, after pelletising 3 - 25 mm.

6.4. Preparation

Pelletising or mixing with coal and a cement which favours cold hardening, storage for several days to reach maximum hardness.

7. Additions (to electric furnace for final reduction)

7.1. Material

Limestone or silica to obtain a fusible and desulphurising slag, coke where necessary.

7.2. Analysis

No details

7.3. Size

Dust

8. Temperature zones in reduction
in °C in the reduction apparatus

Pre-heating and pre-reduction in the shafts : 800 - 950°C,
reduction in the electric furnace : approx. 1600°C.

9. Pressure Zones in reduction
in kg/cm² in the reduction apparatus

Normal atmospheric pressure.

10. Product
of the reduction apparatus

10.1. Form

Molten pig iron

10.2. Analysis

No details

10.3. Degree of reduction

10.3.1. Proportion of oxygen removed from iron oxides as a percentage of the total oxygen combined with iron in the ores

Up to 80% of the oxygen may be removed in the shafts.

10.4. Utilisation of by-products

Slag from the electric furnace

For works a) : 300-350 kg/tonne iron

For works b) : 400-450 kg/tonne iron with the mean analysis:

SiO ₂	TiO ₂	+	Al ₂ O ₃	CaO	MgO	FeO
37%			9 - 10 %	40-42%	6-8%	1%

11. Further treatment of the product (10) of the reduction apparatus

None

12. Final product obtained after treatment according to 11

12.1. Form

Liquid iron

12.2. Analysis

For example at works a) : C 3.8%; Si 3.5%; Mn 0.6%

13. Iron yield

Approximately 95%

14. Utilisation of final product

Destination of final product (blast furnace, electric or other furnace). Technical and economic differences as compared with using pig iron and scrap

Usual applications of pig iron

15. Consumption data per tonne of final product (12)

Per tonne iron, results from pilot plant:

Raw material	Norwegian concentrates pelleted with Longyear coal	Hematite ore fines pelleted with Pocahontas coal

Fixed carbon (coal & coke dust)	290 kg	260 kg

Electricity	1,340 kWh	1,270 kWh

16. Overall capital investment

No details

17. Opinions expressed on the process:

The industrial installations are inoperative

18. Bibliography

18.1. Articles and publications

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- 6) Anderson, H.C., paper presented to International Symposium on sheet reduction of iron ores, Bucharest, September 1972.

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- 4) Norway, filed 22 December 1956, no. 122,949; France no. 1,192,450; German Democratic Republic no. 1,132,167.
- 5) Norway, filed 22 January, 21 February and 25 September 1958; France no. 1,223,457.
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- 12) Norway, filed 4 October 1960; Belgium no. 608,749.
- 13) Norway, filed 27 December 1960, no. 138,543; France no. 1,314,598; Brit.Pat. no. 926,097.

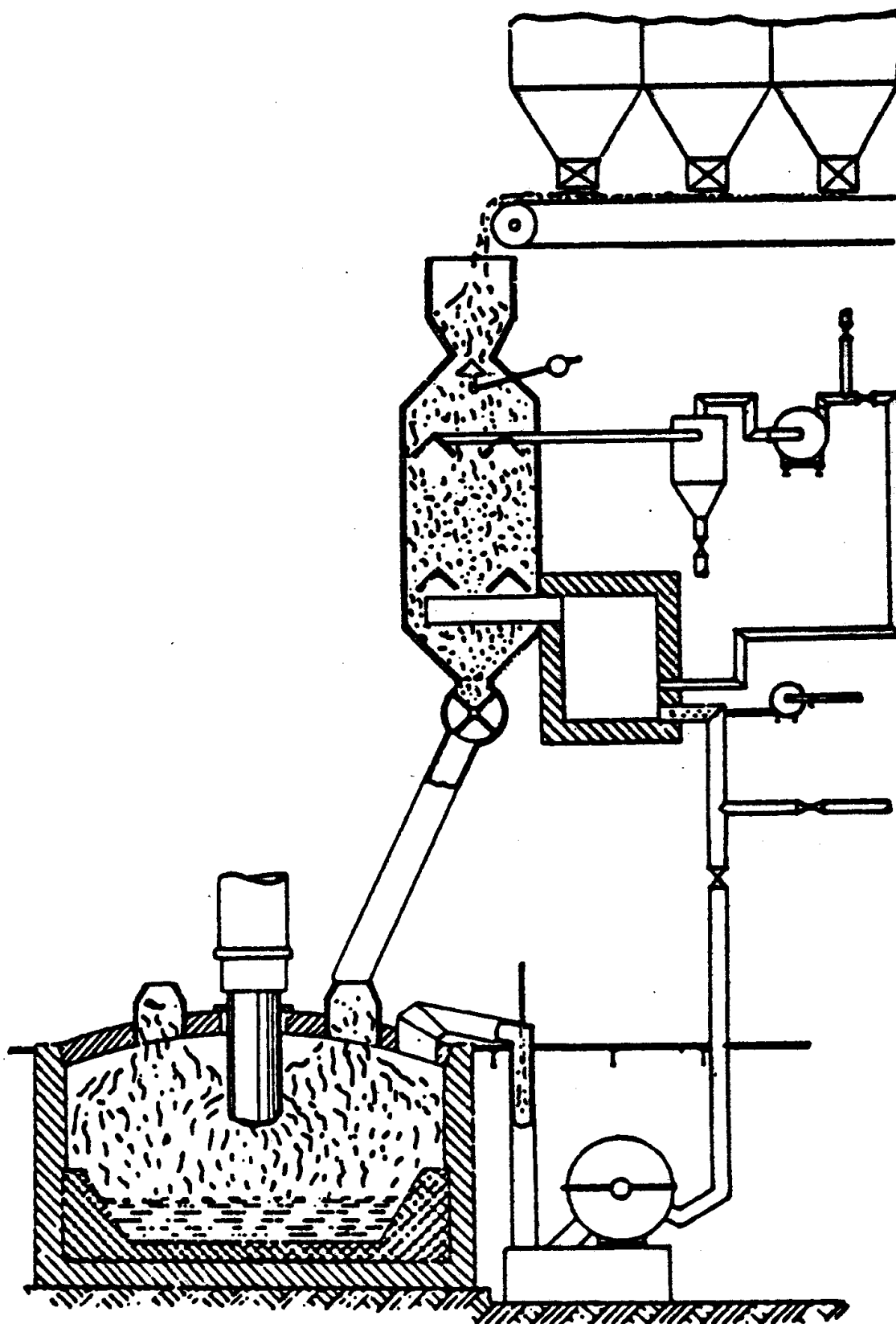


Fig. 1. Principle of Elkem process with preheating in a shaft furnace

FINSIDER PROCESS

1. Characteristics

1.1. Name and history of process

Finsider process

1.2. Principle of process

Reduction in a shaft furnace by means of gas from the gasification of coal by oxygen.

1.4. Companies involved in development

Centro Sperimentale Metallurgico (ex Istituto Siderurgico Finsider).

2. General description (schematic drawing)

See drawing

3. Plant required

3.1. Oxygen gas-producer

3.2. Shaft furnace

3.3. Plant for pelletising and roasting pellets

3.4. Plant for pre-heating and pre-reduction in a rotary kiln

3.5. Means of recycling the gas

4. State of development

4.1. Name of laboratories where the process is situated

Centro Sperimentale Metallurgico S.p.A.

4.2. Pilot plant

Centro Sperimentale Metallurgico S.p.A.

4.2.1. Location

Genova-Cornigliano

4.2.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

Height of shaft of reduction furnace: 9,288 mm

Internal diameter of the shaft of the reduction furnace: 1950 x
843 mm

4.2.3. Capacity in tonnes per 24 h of final product

15-30 t.p.d.

5. Reducing agents

5.1. Raw materials on which the reducing agent is based

Coke, oxygen and steam

5.2. Process for conversion of above raw materials into a form for use in reduction

Gasification

5.3. Condition and analysis of reducing agent at moment of use

Gas of type 70 % CO, 30 % H₂.

5.4. Preheat temperature on entering reduction apparatus

900 - 1000°C

5.5. Gas velocity in reduction apparatus

No details

5.6. Analysis and temperature of gas after passing through reduction apparatus

Analysis: 45.6 % CO; 19.4 % CO₂; 18.2 % H₂; 8 % H₂O

Temperature: 260°C

6. Ores and ferruginous materials

6.1. Source

Various

6.2. Analysis

Magnetites enriched to 62 % Fe

6.3. Size

Fines from magnetic enrichment

6.4. Preparation

Pelletising

7. Additions

No details

8. Temperature zones in reduction
in °C in the reduction apparatus
900 - 1000°C
9. Pressure zones in reduction
in kg/cm² in the reduction apparatus
Normal atmospheric pressure
10. Product
of the reduction apparatus
 - 10.1. Form
Sponge iron in the form of pellets
 - 10.2. Analysis
Depending on the materials used: 90 % total iron for a
degree of reduction of 90 %.
 - 10.3. Degree of reduction
According to the experimental runs: 42 - 88 %
11. Further treatment of the product (10) of the reduction apparatus
 - 11.1. Method of cooling
Discharge and keeping in closed containers
 - 11.2. Crushing, compacting and briquetting
No treatment
 - 11.3. Details of final size grading
Any

- 11.4. Method of concentration (e.g. magnetic separation or sintering)
No enrichment

12. Final product obtained after treatment according to 11

12.1. Form

Sponge iron in pellets

12.2. Analysis

Depending on the materials used, as under 10.2.

13. Iron yield

- 13.1. Iron contained in product (10) as a percentage of iron
in the charge

Practically 100 %

- 13.2. Iron contained in final product (12) as a percentage of
iron in the charge

Practically 100 %

14. Utilisation of final product

Destination of final product (blast furnace, electric or
other furnace). Technical and economic differences as
compared with using pig iron and scrap

Used in steelworks as artificial scrap (virgin charge)

15. Consumption data per tonne of final product (12)

15.1. Raw materials for reduction (5.1)

Coke (in kg/t):

Degree of reduction 41%		Degree of reduction 88%	
per t sponge iron	per t metallic iron	per t sponge iron	per t metallic iron
248	562	423	562

Natural gas (for pre-heating) in kg/t:

23	145	66	88

Natural gas (for re-heating) in kg/t:

9	59	11	15

15.2. Fuels and thermal values required for the entire process
(fuel characteristics) (thermal balance)

See 15.1.

15.3. Electrical energy

No details

15.4. Steam used

15.4.1. For reducing agent
in the gas producer (in kg/t):

Degree of reduction 41%		Degree of reduction 88%	
per t sponge iron	per t metallic iron	per t sponge iron	per t metallic iron
126	794	214	285

For regeneration (in kg/t):

Degree of reduction 41%		Degree of reduction 88%	
per t sponge iron	per t metallic iron	per t sponge iron	per t metallic iron
105	660	2567	3411

15.5. Oxygen

In the gas-producer (in kg/t):

205	1291	349	464

In the superheater (in kg/t):

7	42	54	72

15.6. Cooling water

No details

15.7. Refractories

No details

15.8. Man hours of productive labour

No details

15.9. Labour and materials for routine maintenance

No details

16. Overall capital investment

16.1. For the plant described in (3) and related to one tonne of capacity for the final product (12)

Capacity expressed in tonnes per day or per year

Depreciation evaluated at 3.20 dollars per tonne corresponding to 50 dollars per annual tonne investment.

17. Opinions expressed on the process in relation to

17.1. State of development

Research carried out with financing by the High Authority of the European Coal and Steel Community.

17.2. Practical operating difficulties

No details

17.3. Possibilities of exploitation

Possible to use enriched ores. The final product can be used in the steelworks as artificial scrap particularly for producing special products for which a high degree of purity of the metal is required or in the blast furnace as pre-reduced charge.

18. Bibliography

18.1. Articles and publications

1) Palazzi, A., L. Palumbo and V. Giordano: Research on direct reduction of iron ores in the shaft furnace.

E.C.S.C. High Authority, October 1964.

2) Scortecci, A.: Reports of the Journées internationales de sidérurgie, Liège, 18-28 June, 1958, p.224.

3) Internal reports of the Istituto Siderurgico Finsider.

18.2. Patents

No details

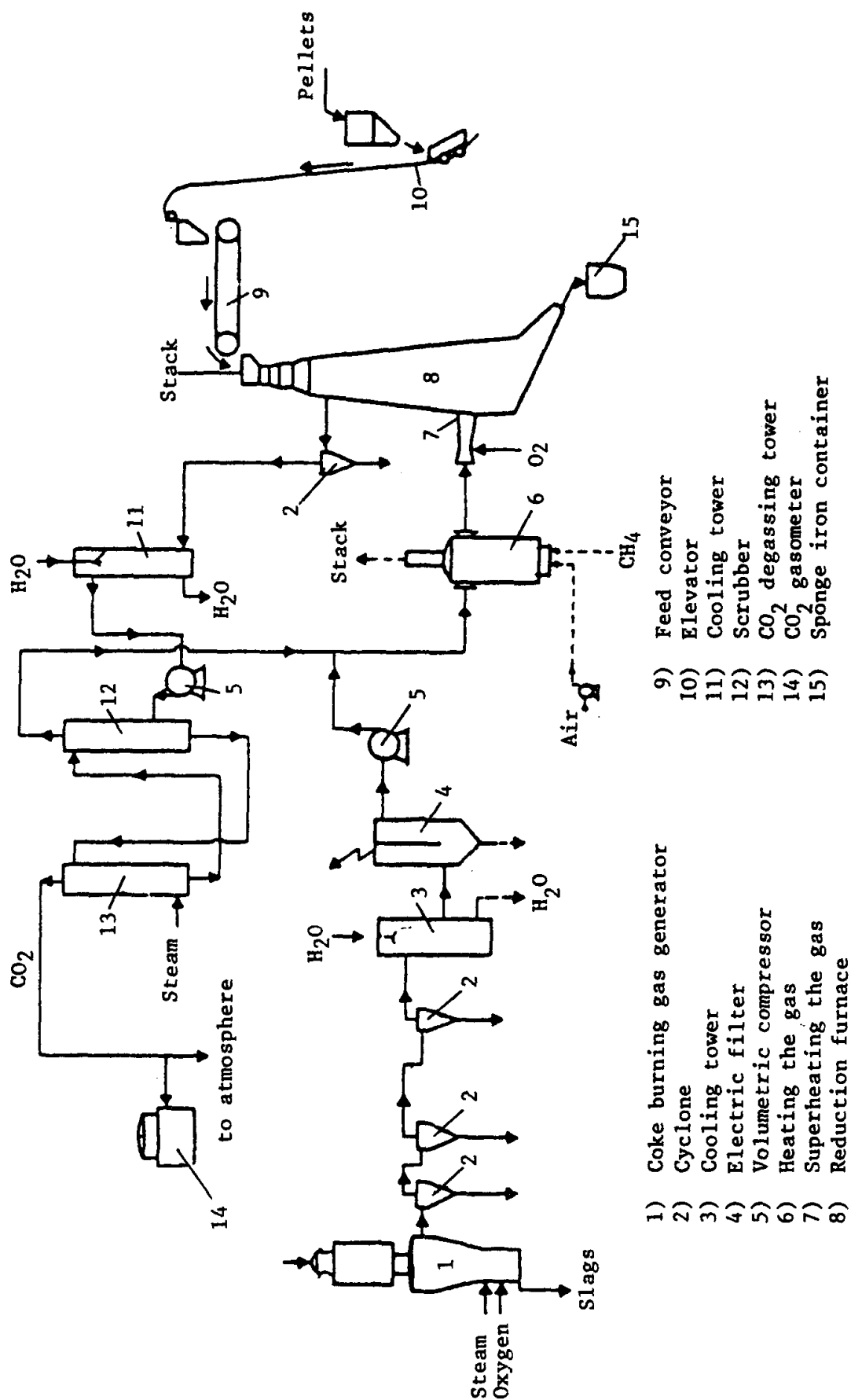


Fig. 1. General diagram of the Finsider pilot plant (without pre-reduction in the rotary kiln)

FLAME SMELTING

1. Characteristics

1.1. Name and history of process

Flame smelting process (originally Cyclosteel)

1.2. Principle of process

Reduction is done in two stages:

In the first, fine ore is partially reduced in a fluid bed reactor at a moderate temperature.

Subsequently the hot partially reduced ore is introduced with powdered coal and oxygen into a high temperature reactor which produces liquid metal. Reduction and rapid fusion is followed by separation of metal and slag.

The high temperature reducing gas leaving the second reactor is utilised to carry out the preheating and reduction in the first stage.

1.3. Inventors

B.I.S.R.A.

2. General description (schematic drawing)

See Figure 1

3. Plant required

In addition to the reactor system, an oxygen plant is required and also a coal preparation plant.

4. State of development

4.1. Name of laboratories where the process is situated

B.I.S.R.A., Grangetown.

4.2. Pilot plant

4.2.1. Location

See Bibliography 18.1.

4.2.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

High temperature reactor 1 m dia. 1 m high.

4.2.3. Capacity in tonnes per 24 h of final product

20 tonnes/day

4.3. Industrial installations

None

5. Reducing Agents

5.1. Raw materials on which the reducing agent is based

Coal

5.2. Process for conversion of above raw materials into a form for use in reduction

No details

5.3. Condition and analysis of reducing agent at moment of use

CO/CO₂ ratio in first stage approximately 10.

6. Ores and ferruginous materials

No details

7. Additions

No details

8. Temperature zones in reduction
in °C in the reduction apparatus

Over 1500°C

9. Pressure zones in reduction
in kg/cm² in the reduction apparatus

No details

10. Product
of the reduction apparatus

10.1. Form

Liquid metal

10.2. Analysis

No details

10.3. Degree of reduction

No details

10.4. Utilisation of byproducts

No details

11. Further treatment of the product (10) of the reduction apparatus

No details

12. Final product obtained after treatment according to 11

No details

13. Iron yield

No details

14. Utilisation of final product

Destination of final product (blast furnace, electric or other furnace). Technical and economic differences compared with the use of pig iron and scrap.

No details

15. Consumption data per tonne of final product (12)

No details

16. Overall capital investment

No details

17. Opinions expressed on the process in relation to

17.1. State of development

The high temperature reactor operated successfully over long periods but the operation of the fluidised bed with the dirty gas from the second stage gave rise to difficulties. Economic circumstances led to the project being abandoned in 1962 and the pilot plant dismantled.

18. Bibliography

18.1. Articles and publications

- 1) Goodeve, C.: Steelmaking since Bessemer. The advancement of science No.50, September 1956. British Ass. Sheffield, 30th August 1956, p.8. See also J. Iron and Steel Inst. 184 (1956) No.2, p.113, British Steelmaker 22 (1956) No.10, p.286, Iron and Coal Trad. Rev. 173 (1956) No.4,607, p.6, Engineering 182 (1956) No.4,722, p.295.
- 2) Cyclosteel, a new process for direct steelmaking. British Chemical Engineering, October 1956, p.291.
- 3) From ore, steel. Chemical Week, January 1957, p.95.
- 4) Cyclosteel. Steel, 6th May 1957, p.83.
- 5) Bhat, G.K.: Cyclosteel - Steel direct from ore. Iron Age 179 (1957) No.21, p.129.
- 6) Unterweiser, P.M.: Steelworks of the future. Iron Age 180 (1957) No.12, p.139.
- 7) Robiette, A.G.: The new metallurgical processes. Iron Coal Trad. Rev. 177 (1958) No.4,705, p.209-16.
- 8) Gascuel: Development of metallurgical processes. Pact (1959) No.6, p.497/99.
- 9) Johnson, T.W., and J. Davison: Reduction of iron ore in Flame-Smelting-process. J. Iron and Steel Inst. 202, May 1964, p.406.

18.2. Patents

No details

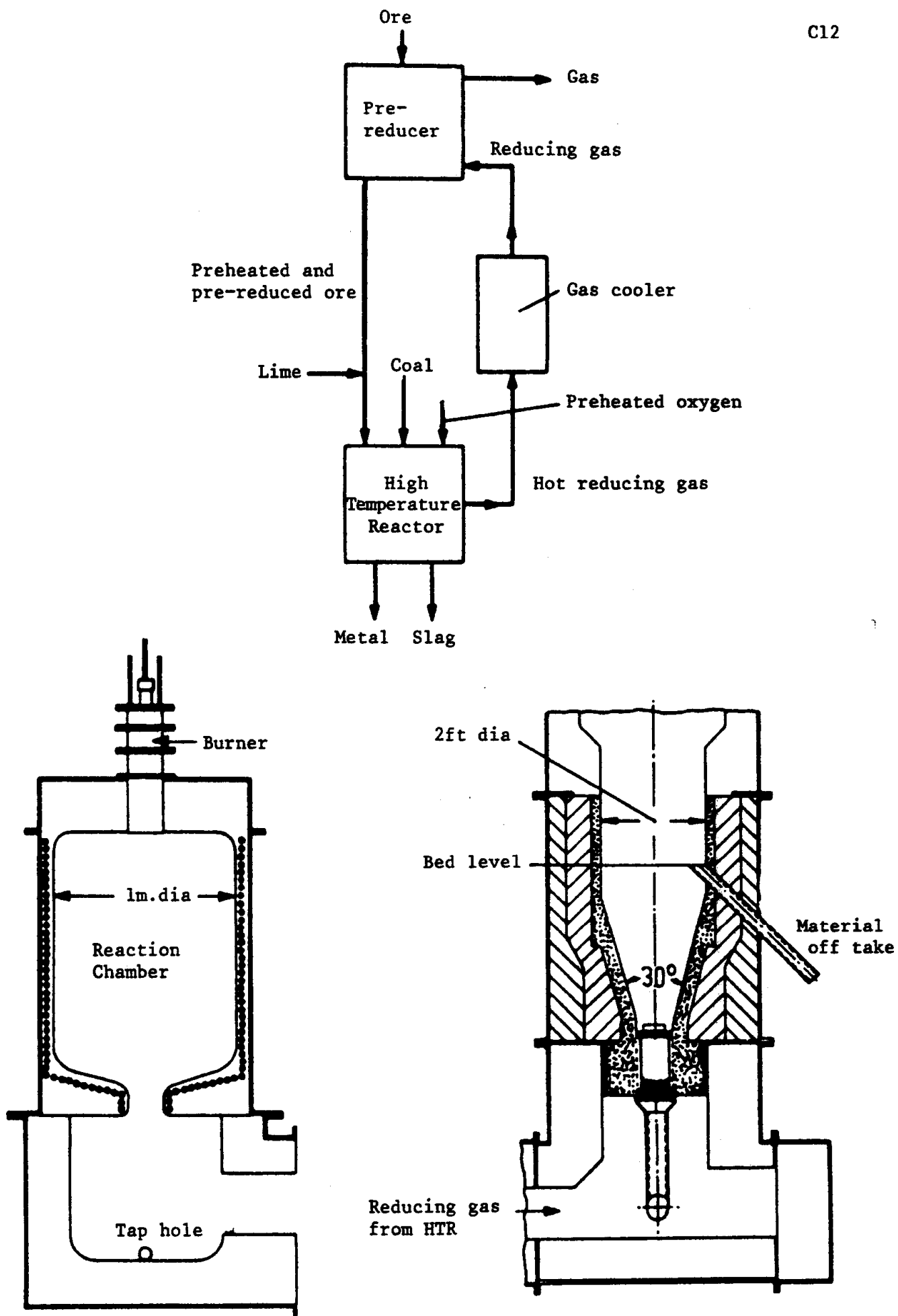


Fig. 1. Flame Smelting

FREEMAN PROCESS

1. Characteristics

1.1. Name and history of process

H Freeman process

1.2. Principle of process

Reduction of pellets of iron ore concentrates in a rotary kiln, with circulation of the reducing gases and the material to be reduced in the same direction.

1.3. Inventors

H Freeman

1.4. Companies involved in development

Freeman Corporation (formerly Ferrum Ltd.),
Cap de la Madeleine, Trois Rivières, Quebec Province.

1.5. Projects

2. General description (schematic drawing)

See drawing.

The process consists in reducing, with carbon monoxide supplied by a mixture of limestone and coke dust, pellets of concentrates specially prepared so as to obtain sponges of metallic iron. The rotary kiln in which the reduction takes place is heated at the entry for materials by a burner, and it has at the exit a regulatable additional air supply to burn the gas (CO) not used in the reduction.

3. Plant required

For the preparation of pellets:

Ball Mill

Mixer and extrusion equipment

Pelletising drum

Air roasting furnace

For the preparation of the reducing mixture:

Crusher for limestone and coke

Hopper for mixing pellets and reductants

For the reduction (equipment shown on the drawing):

Specially equipped rotary kiln

Cooling conveyor with water jacket

For complementary treatment as under 11:

Cyclones for separating the dusts and metallic pellets

Ball crusher when iron powders are produced (see 18.2.2).

4. State of development

4.1. Name of laboratories where the process is situated

Freeman Corporation

4.2.1. Location

Trois Rivières, Quebec, Canada.

4.2.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

10 m reduction kiln, internal diameter 1.2 m.
(33 ft x 4 ft).

4.2.3. Capacity in tonnes per 24 h of final product

10 tonnes per day

4.3. Industrial installations

Some time ago preliminary discussions with US Steel Corporation and Union Carbide and projects for Brazil or Canada.

5. Reducing agents

5.1. Raw materials on which the reducing agent is based

Mixture of equal parts of coke dust and limestone

5.2. Process for conversion of above raw materials into a form for use in reduction

Mixing and crushing to between 0.15 and 0.55 mm (100 and 30 mesh).

5.3. Condition and analysis of reducing agent at moment of use

Mean size 0.245 mm (60 mesh)

Analysis: Coke dust with 85 % fixed carbon

Limestone with 80 % CaCO_3 .

5.4. Preheat temperature on entering reduction apparatus

The preheating takes place in the rotary reduction kiln.

5.5. Gas velocity in reduction apparatus

No details

5.6. Analysis and temperature of gas after passing through reduction apparatus

900 - 1000°C.

6. Ores and ferruginous materials

6.1. Source

Natural rich ores or enriched ores

6.2. Analysis

At least 65-70 % iron

6.3. Size

These materials are crushed very fine in the preparation plant (See 3.1.1).

6.4. Preparation

Crushing with addition of a binder (flour, molasses, bentonite, etc.) in the proportion of 0.5-2 %; this binder is added after the crusher with wet ores. Addition of 10 % water and mixing followed by extrusion, drum pelletising, with drying on leaving by the off gases from the reduction kiln. Roasting in air of the pellets 3-19 mm in diameter (1/8 - 3/4 in), the mean recommended being 12.7 mm (1/2 in).

Mixing with reductants before charging (18.2.2).

7. Additions

See 5.

8. Temperature zones in reduction
in °C in the reduction apparatus

900 - 1050°C; 1000°C is aimed for to favour the reduction of Fe_2O_3 by CO. (18.1.1).

9. Pressure zones in reduction
in kg/cm² in the reduction apparatus

No details
10. Product
of the reduction apparatus
 - 10.1. Form

Sponge iron in the form of pellets
 - 10.2. Analysis

Starting with a hematite oxide with 70 % iron:
0.02 % C; 0.01 % S; 0.4 % Si; 1.5 % Fe₃O₄, 98.07 % Fe_{met}
(18.2.2).
 - 10.3. Degree of reduction

98.5 % metallisation
11. Further treatment of the product (10) of the reduction apparatus
 - 11.1. Method of cooling

Cooling in the conveyor-extractor, by water jacket.
 - 11.2. Crushing, compacting or briquetting

Separation of the pellets from the powdery limestone waste (lime) and coke ash in a cyclone apparatus. Crushing the pellets if necessary when producing iron powder.
 - 11.3. Details of final size grading

The pellets are the same size as on entering the kiln,
i.e. on average $\frac{1}{2}$ in (12.7 mm).

- 11.4. Method of concentration (e.g. magnetic separation or sintering)

None

12. Final product obtained after treatment according to 11

- 12.1. Form

Pellets or iron powder

- 12.2. Analysis

(See 10.2).

13. Iron yield

No details

14. Utilisation of final product

Destination of final product (blast furnace, electric or other furnace). Technical and economic differences as compared with using pig iron and scrap

Pellets: foundry, additions to the electric furnace.

Iron powder: powder metallurgy, coating electrodes.

The excess reducing powder obtained as under 11.2 could be used in the manufacture of cement.

15. Consumption data per tonne of final product (12)

According to Mr Freeman this is an example of the results of operating a pilot plant of 2 t.p.d. (18.1.1. and 18.2.2).

15.1. Raw materials for reduction (5.1)

To produce one tonne of reduced iron pellets starting from 1,430 kg ore pellets with 70 % iron:

- 600 kg coke dust with 85 % fixed carbon,
- 600 kg limestone with 80 % CaCO_3 .

15.2. Fuels and thermal values required for the entire process (fuel characteristics) (thermal balance)

370-400 kg fuel oil for heating the kiln

16. Overall capital investment

No details

17. Opinions expressed on the process in relation to:

17.1. State of development

The process seems to have been seriously studied by its inventor on a 2 t.p.d. pilot plant (kiln 3 m long) (Ref. 18.1.1. and 18.2.) since 1949, then on a 10 t.p.d. pilot plant (kiln 10 m long) since 1957 (Ref.18.1.2.).

18. Bibliography

18.1. Articles and publications

- 1) H Freeman - Direct Iron in Canada. The Canadian Mining and Metallurgical Bulletin, August 1956.
- 2) H Freeman - A New Direct Iron Process in Relation to ore and Steel Production. The Canadian Mining and Metallurgical Bulletin, August 1959.
- 3) H Freeman - Reasons for the Co-Current Kiln in Direct Reduction. Chicago Congress, May 1960.

18.2. Patents

U.S.A. Patent No.2,792,298 of 14th May, 1957 (filed 9th April, 1954) corresponds to:

- France: 1,128,520 (applied for 7th April, 1955,
published 7th January, 1957);
- Belgium: 537,058 (2nd April, 1955);
- U.K.: 768,018.
- U.S.A.: 2,855,290 7th October 1957 (filed 12th July, 1956),
corresponds to
 - Canada: (filed 4th April, 1956)
 - France: 71,666 (1st addition)/ 1,128,520 (applied for 2nd
April, 1957, published 13th January, 1960);
 - U.K.: 831,006 (2nd April 1957)
 - U.S.A.: 3,080,156 5th March, 1963 (filed 23rd March, 1960),
corresponds to
 - U.K.: 894,348 (18th April 1962).

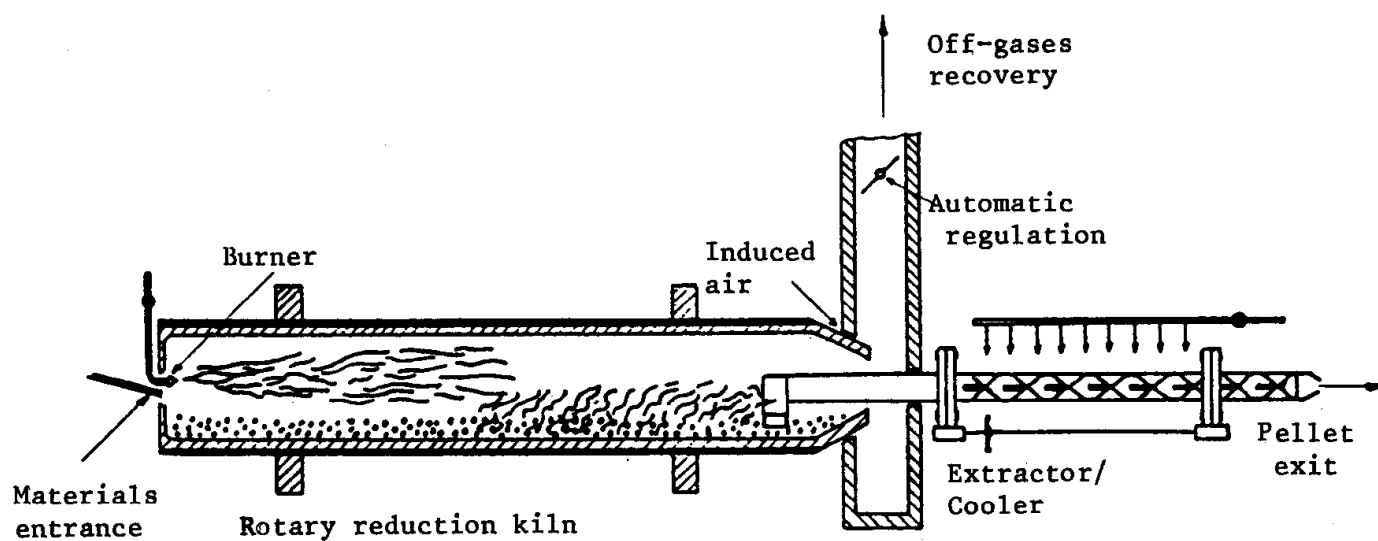


Fig. 1. Freeman Process

HEAT-FAST PROCESS

1. Characteristics

1.1. Name and history of process

Heat-Fast process for producing pre-reduced pellets.

1.2. Principle of process

Production of partly reduced pellets from iron ore concentrates by pelletising with a solid reductant and heating to reduction temperature.

1.3. Inventors

Surface Combustion Division of Midland Ross Corporation, 2375 Dorr Street, P O Box 407, Toledo, Ohio 43601, U.S.A.

1.4. Companies involved in development

The inventing company is developing it. The version of this process intended to prepare oxide pellets (i.e. without addition of solid reductant) has been retained by National Steel Pellet Co., (company formed by National Steel Corp., Weirton, West Virginia and Hanna Mining Company, 100 Erieview Plaza, Cleveland, Ohio, U.S.A.).

2. General description (schematic drawing)

See Figures.

We reproduce the diagram of the version of the process leading to oxide pellets: for pre-reduced pellets the equipment would be almost identical (fig.1), and the diagram of the pilot plant at Cooley for the production of pre-reduced pellets (fig.2).

3. Plant required

Conventional pelletising drum.

Drying oven with continuous grate.

Rotary hearth furnace for roasting and reduction.

Shaft furnace for cooling and, where appropriate, the final reduction.

4. State of development

4.1. Name of laboratories where the process is situated

Toledo Laboratory of Surface Combustion, where there is a small pilot plant of 5 - 10 tpd capacity.

4.2. Pilot plant

4.2.1. Location

Pilot plant of Midland Ross Corporation, Cooley, Minnesota, U.S.A.

4.2.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

Drying oven with continuous grate 6.1 m (20ft) long by 1.2 m (4ft) wide. Rotary hearth furnace of ordinary ceramic 6.1 m (20ft) in external diameter and 3.7 m (12ft) internal diameter.

Shaft furnace 1.35 m in diameter and 6 m total height from the charging level to the discharge door.

4.2.3. Capacity in tonnes per 24 h of final product

100 - 200 tpd oxide pellets. No information on the capacity for pre-reduced pellets.

4.3. Industrial installations

No industrial installation for producing pre-reduced pellets, but an 8000 tpd plant for oxide pellets at Keewatin, Minnesota, U.S.A.

5. Reducing agents

5.1. Raw materials on which the reducing agent is based

Coal, coke, etc.

5.2. Process for conversion of above raw materials into a form for use in reduction

Fine grinding

5.3. Condition and analysis of reducing agent at moment of use

Powder

5.4. Preheat temperature on entering reduction apparatus

The crushed coal is mixed with the iron ore concentrates before pelletising and drying at approximately 150°C.

5.5. Gas velocity in reduction apparatus

No details

5.6. Analysis and temperature of gas after passing through reduction apparatus

No details

6. Ores and ferruginous materials

6.1. Source

Various sources

6.2. **Analysis**

Tests have been carried out on concentrates with 57 % Fe,
and on others with 68 - 69% Fe.

6.3. **Size**

Enriched concentrates

6.4. **Preparation**

Pelletising after mixing with the reducing coal and a binder
(for example bentonite).

7. **Additions**

No special additions

8. **Temperature zones in reduction**
in °C in the reduction apparatus

The reducing apparatus, a rotary hearth, is divided into three zones and in each of these zones the temperature and atmosphere are controlled. The first zone is for preheating from 150°C to about 1000°C by means of burners fired with natural gas; in the second zone the reduction starts and the temperature increases from 1000°C to 1150°C; in the third zone reduction continues and the temperature varies from 1150°C to 1250°C, then a slight cooling brings the products to 1100°C before they are fed into the vertical shaft cooler.

9. **Pressure zones in reduction**
in kg/cm² in the reduction apparatus

No details

10. Product
 of the reduction apparatus

10.1. Form

Pellets 12.7 - 25.4 mm in diameter

10.2. Analysis

No details

10.3. Degree of reduction

No details

10.3.1. Proportion of oxygen removed from iron oxides as a percentage
 of the total oxygen combined with iron in the ores

No details

10.3.2. Metallic iron as a percentage of the total iron content of
 the product

Tests have given degrees of metallisation between 60 and 90%.

10.4. Utilisation of by-products

No details

11. Further treatment of the product (10) of the reduction apparatus

11.1. Method of cooling

Cooling in shaft furnace to 120°C. A Duplex process using gas obtained by air reforming (H₂ 40%, CO 20%) consists in continuing reduction to a degree of metallisation of 95% before cooling.

11.2. Crushing, compacting or briquetting

No details

11.3. Details of final size grading

No details

11.4. Method of concentration (e.g. magnetic separation or sintering)

None

12. Final product obtained after treatment according to 11

12.1. Form

Metallised pellets

12.2. Analysis

80 - 90% iron

13. Iron yield

No details

14. Utilisation of final product

Destination of final product (blast furnace, electric or other furnace). Technical and economic differences as compared with using pig iron and scrap

No details

15. Consumption data per tonne of final product (12)

No details

16. Overall capital investment

No details

17. Opinions expressed on the process

None

18. Bibliography

18.1. Articles and publications

- 1) The Iron Age (U.S.A.), 18 February 1965 p.30 and 8 November 1965 p.31.
- 2) Metal Bulletin (G.B.), 23 February 1965.
- 3) Iron and Steel Engineer (U.S.A.), March 1965, 42 No.3 p.189.
- 4) Engineering and Mining Journal (U.S.A.), April 1965.
- 5) Mining Engineering (U.S.A.), 17 April 1965, p.49 and 18 February 1966, p.34.
- 6) Journal of Metals (U.S.A.), 17 May 1965, p.459.
- 7) L.W. Smith and D.N. Vedensky, A new process for roasting oxidised pellets. Revue de l'Industrie Minérale (France) 48,no.5, May 1966,pp.379-388.
- 8) L.W. Smith and D.N. Vedensky, Heat Fast - Process for Pre-reduced pellets. Paper presented to "Congrès International sur la production et l'utilisation des minerais réduits" (International congress on production and utilisation of reduced ores) Evian (France), 29, 30 and 31 May 1967, pp.57-65.

18.2. Patents

Basic patent: U.S.A.no.2.793.109 of 21 May 1957.

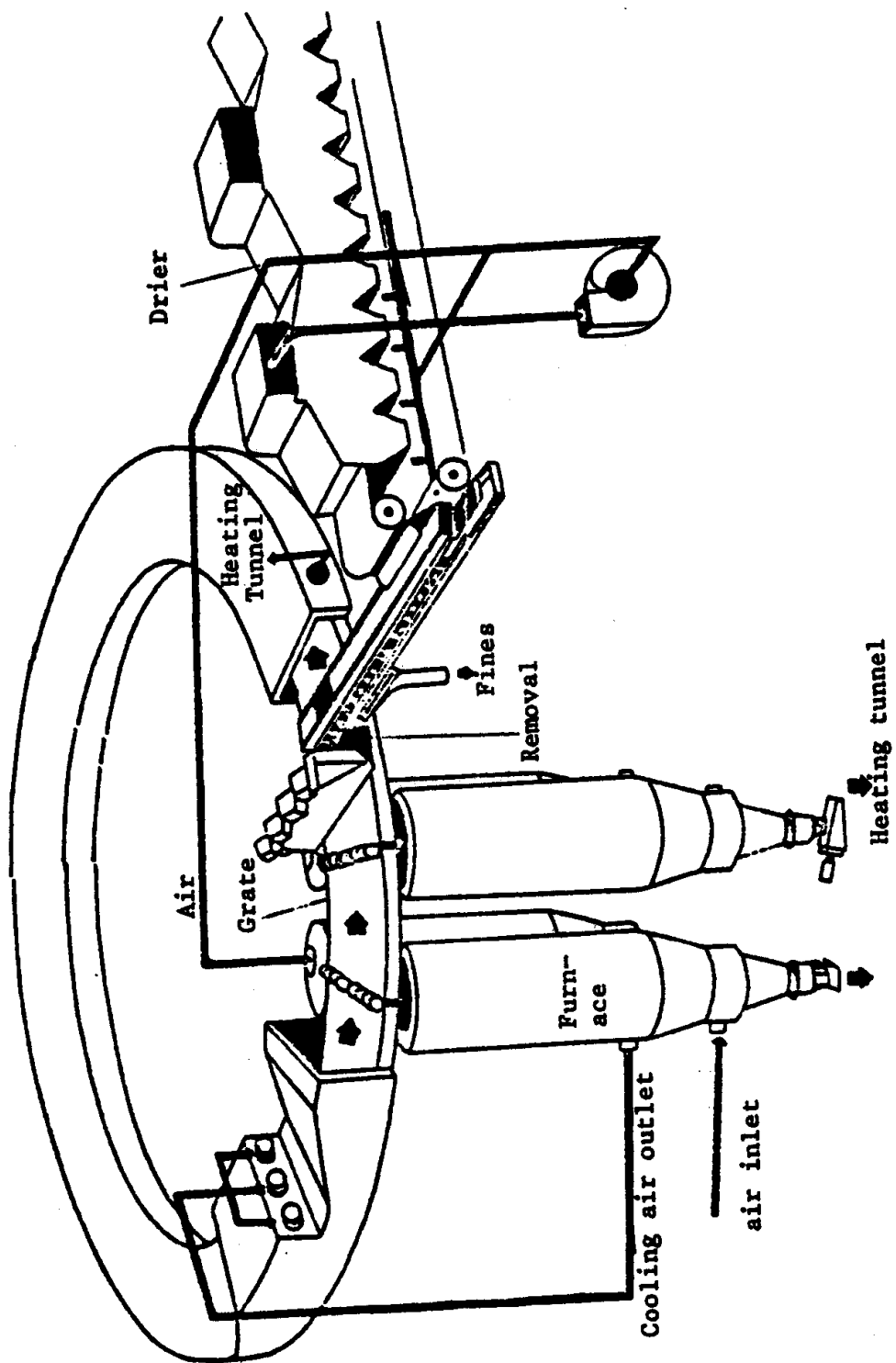


Fig.1 Heat Past Process.

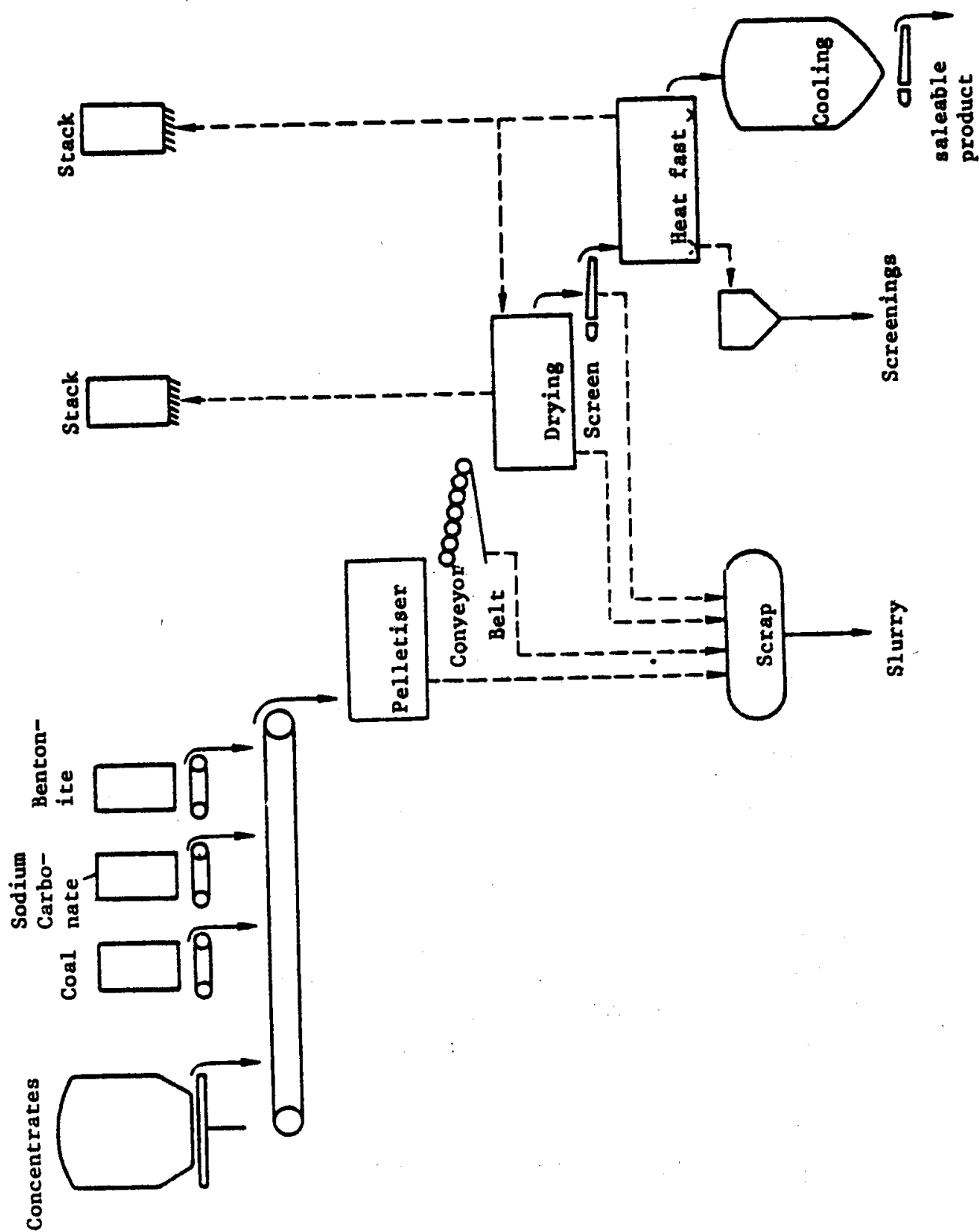


Fig. 2. Diagram of Cooley pilot plant

H - IRON PROCESS

1. Characteristics

1.1. Name and history of process

H-Iron process

1950: pilot plant (20 tpd), Brownsville, Texas

1953: pilot plant (10 tpd), Hydrocarbon Research Inc.,
Trenton, New Jersey

1960: Industrial plant (50 tpd), Alan Wood Co.,
Conshohocken, Pa.

1961: Industrial plant (120 tpd), Bethlehem Steel Co.,
Vernon, California

1.2. Principle of process

Reduction by hydrogen at 480°C under pressure

1.3. Inventors

Hydrocarbon Research Inc., 115 Broadway, New York, U.S.A.

1.4. Companies involved in development

Brownsville, Texas

Hydrocarbon Research Inc., Trenton, New Jersey

Alan Wood Co., Conshohocken, Pa.

Bethlehem Steel Co., Vernon, California

2. General description (schematic drawing)

See drawing

3. Plant required

Ore preparation plant, preheating in a rotary furnace, production of pure hydrogen (e.g. by cracking natural gas or by gasification of oil or coal with special purification). Purification plant and gas recycling. Fluidised bed reactor (possibly with several stages). Device for briquetting the reduced iron ore and for deactivation.

4. State of development

4.1. Name of laboratories where the process is situated

No details

4.2. Pilot plant

4.2.1. Location

Trenton, New Jersey, USA.

4.2.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

Fluidised bed reactors 85 cm in diameter and 9 m high.

4.2.3. Capacity in tonnes per 24 hr of final product

5 t ore can be put through in 8 to 12 hours, which corresponds to a maximum output of 10 t sponge iron powder per day.

4.3. Industrial installations

4.3.1. Location

a) Conshohocken, Pennsylvania, USA.

b) Vernon, California.

4.3.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

- a) 1.65 m in diameter and 29 m high; the fluidised bed reactor consists of three stages one on top of the other.
- b) 1.95 m in diameter and 33 m high; the fluidised bed reactor consists of four stages one on top of the other.

4.3.3. Capacity in tonnes per 24 hr of final product - per unit of production

- a) 50 tpd
- b) 120 tpd

4.3.4. Method of operation (continuous or batch, number of working days per annum)

The production process is continuous with treatment in 3 fluidised beds one after the other. The final product is removed when it has reached a degree of reduction of 98 % in the bottom fluidised bed. The contents of the second fluidised bed (degree of reduction 87 %) are passed to the bottom bed and the contents of the top one (degree of reduction 47 %) to the second. Then 15 t are charged pneumatically into the top fluidised bed via a charge hopper. During this process the flow of hydrogen is not interrupted.

4.3.5. Coefficient of utilisation in time (number of working days per annum)

No details

4.3.6. Maximum rated production

No details

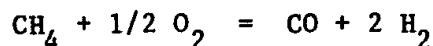
5. Reducing agents

5.1. Raw materials on which the reducing agent is based

Oil, natural gas, coal, coke oven gas.

5.2. Process for conversion of above raw materials into a form for use in reduction

After gasification by the Texaco-HRI process of partial oxidation the gas is purified. The principle is the combustion of CH_4 with oxygen in accordance with the equation



then finally conversion of the CO by steam.

5.3. Condition and analysis of reducing agent at moment of use

The HRI process prefers a gas with 99 % hydrogen and < 0.1 % CO. Tests were carried out with gas with 0.8 - 2.5 % CO, up to 5 % N_2 , < 1.5 % CH_4 . No technical problems occurred but there was a noticeable increase in the manufacturing costs.

The 50 tpd industrial plant uses a gas with 97 % H_2 , 3 % N_2 and traces of CO and CH_4 .

5.4. Preheat temperature on entering reduction apparatus

540°C

5.5. Gas velocity in reduction apparatus

30 - 45 cm/s (on average 33.5 cm/s)

The calculated velocity in the work quoted under 18.1.4. was 40 - 80 cm/s.

5.6. Analysis and temperature of gas after passing through reduction apparatus

The composition of the gas depends on the type of ore and the number of stages. With 3 stages we have a gas with 5 % H_2O , 92 % H_2 , 3 % N_2 and traces of CH_4 and CO .

The temperature is about $480^{\circ}C$.

6. Ores and ferruginous materials

6.1. Source

Widely differing ores

6.2. Analysis

Element	Magnetite concentrate from Sweden	Magnetite concentrate Alan Wood (USA)	Mill scale	Cerro Bolivar (Venezuela)
% Fe	72	72.1	74.8	68.5
% O_2	27.2	27.6	24.5	28.4
% gangue	0.8	0.3	0.7	2.1
% SiO_2	-	-	-	-
Element	Peruvian ore	Concentrate from Benson (USA)	Martite (USA)	Lean ore from Peru
% Fe	60.0	62.0	62.2	33
% O_2	26.8	26.8	24.0	-
% gangue	13.2	13.2	14.0	-
% SiO_2	-	-	-	50

6.3. Size

0.04 - 0.8 mm

6.4. Preparation

No details

7. Additions

None

8. Temperature zones in reduction
in °C in the reduction apparatus

The initial temperature of the reducing gas is 540°C. The reduction in the individual stages takes place at about 480°C.

9. Pressure zones in reduction
in kg/cm² in the reduction apparatus

35 kg/cm², the normal pressure being 28 kg/cm².

10. Product
of the reduction apparatus

10.1. Form

Sponge iron powder

10.2. Analysis

No details

10.3. Degree of reduction

Variable, up to 98 %

- 10.3.1. Proportion of oxygen removed from iron oxides as a percentage of the total oxygen combined with iron in the ores

No details

- 10.3.2. Metallic iron as a percentage of the total iron content of the product

No details

- 10.4. Utilisation of by-products

No details

11. Further treatment of the product (10) of the reduction apparatus

- 11.1. Method of cooling

Air excluded as the product is passed to the briquetting press

- 11.2. Crushing, compacting or briquetting

Briquetting

- 11.3. Details of final size grading

No details

- 11.4. Method of concentration (e.g. magnetic separation or sintering)

Concentration by means of magnetic separation is provided for when lean ores are used.

12. Iron yield

- 12.1. Form

Conshohocken: briquettes (approximately 15 x 3.5 x 2 cm).

Ore fines or briquettes (density 5.5 g/cm³).

Vernon: briquettes (10 cm long, 7.5 cm high) or ore fines.

12.2. Analysis

Example of the analysis of the iron powder:

Conshohocken: Fe_{tot} 96.5 - 97.5; Fe_{met} 95.0 - 96.0

C	0.013 - 0.040
P	0.008 - 0.010
S	0.005 - 0.020
Mn	0.02 - 0.10
SiO ₂	0.5 - 1.0
Cu	0.01 - 0.02
Mo	0.01 - 0.02
V	0.01 - 0.02
Ni	0.01 - 0.02
Ca	0.01 - 0.02
Bi	0.001 - 0.002
Sn	0.001 - 0.002
Pb	0.001 - 0.002
Cr	0.001 - 0.002
Co	0.001 - 0.002
Al ₂ O ₃	0.2 - 0.4
TiO ₂	0.3 - 0.6
MgO	0.08 - 0.10

13. Iron yield

With the exception of the cases where the ore is concentrated after reduction, there are obviously no losses during reduction. However, it is different when the product is used.

13.1. Iron contained in product (10) as a percentage of iron in the charge

No details

13.2. Iron contained in final product (12) as a percentage of iron in the charge

No details

14. Utilisation of final product

Destination of final product (blast furnace, electric or other furnace). Technical and economic differences compared with the use of pig iron and scrap

During the tests the sponge iron was melted down with or without addition of scrap in an electric arc furnace without difficulties. No noticeable reoxidation of the iron powder was found. The melting down times were a little longer for scrap than for sponge iron. The consumption of power and refractories in the furnace were comparable.

In addition the iron powder was sintered. Part was used as welding electrode coating (for electrodes with very high welding speed), etc.

15. Consumption data per tonne of final product (12)

15.1. Raw materials for reduction (5.1)

For hydrogen production: 225 l oil, 245 Nm³ natural gas

15.2. Fuels and thermal values required for the entire process (fuel characteristics) (thermal balance)

Fuel: 280 l oil, 220 Nm³ natural gas

15.3. Electrical energy

Oil: 165 kWh; natural gas 80 kWh

15.4. Steam

No details

15.5. Oxygen

No details

15.6. Cooling water

Oil: 30 Nm³; natural gas: 30 Nm³

15.7. Refractories

No details

15.8. Man hours of productive labour

28/100 h/t + 25 % other.

15.9. Labour and materials for routine maintenance

Estimated at approximately 3 % of the costs.

Note: These figures relate to a reduction plant for 2000 t iron ore per day.

16. Overall capital investment

None

17. Opinions expressed on the process in relation to:

17.1. State of development

In the literature a plant with a capacity of 1000 tpd is discussed, the operation of which is said not to be satisfactory from the economic point of view. Details of this plant are not yet known, however.

There are no reliable predictions of the possibilities of the further development of this process. According to information which was not checked the process seems to have been abandoned.

17.2. Practical operating difficulties

No details

17.3. Possibilities of exploitation

No details

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18.2. **Patents**

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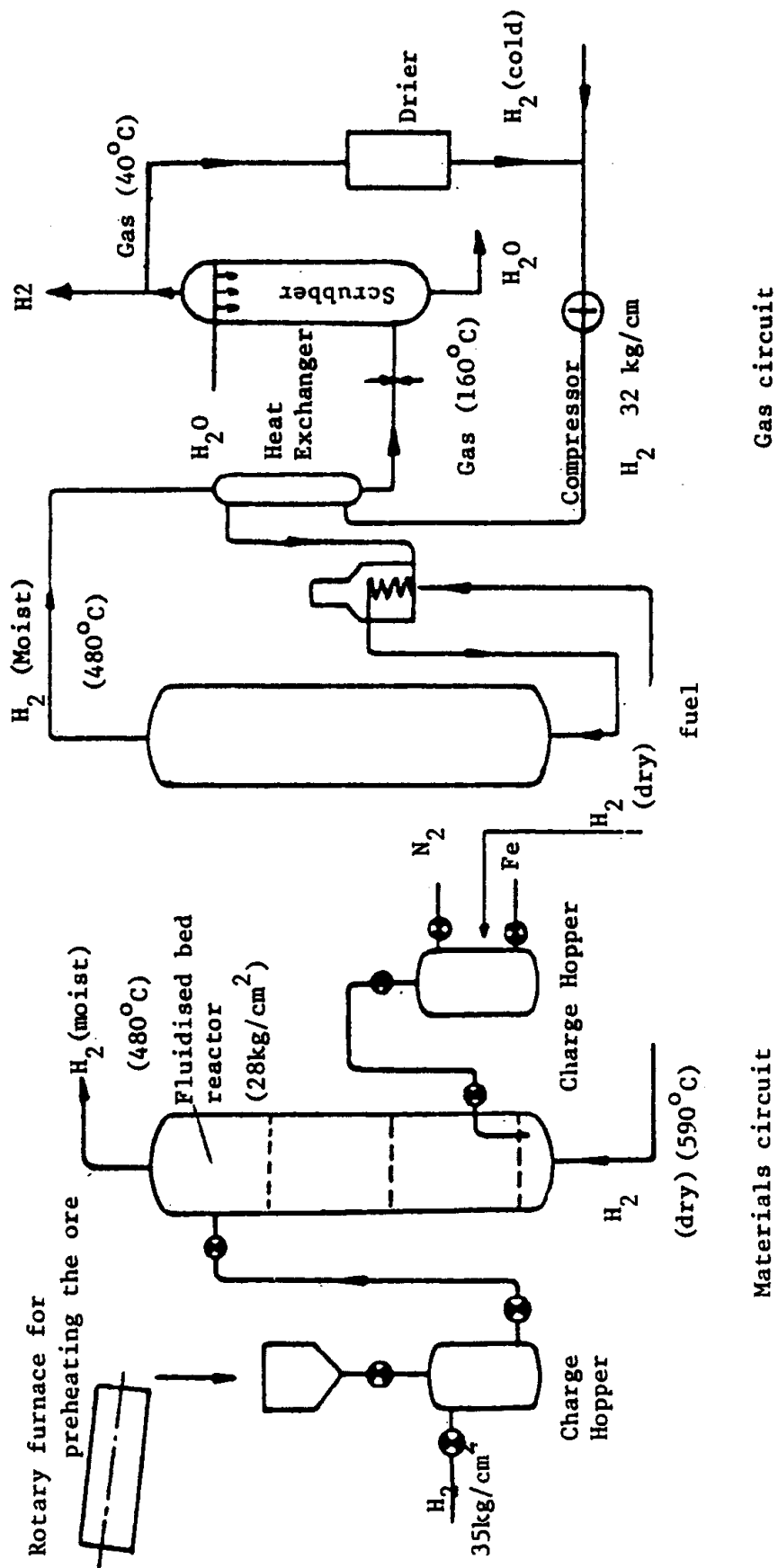


Fig. 1 H-Iron Process.

JET SMELTING

1. Characteristics

1.1. Name and history of process

Jet Smelting Process

1.2. Principle of process

Fine iron oxide concentrate is entrained in a high temperature flame of oxygen and natural gas. The particles of ore decrepitate and the increase in surface is significant. The flame is blown vertically downwards into a bath of molten metal and slag. This consequently reduces dust losses.

In order to accelerate the later stages of reduction the bath is maintained at a high carbon content (by injecting powdered graphite) so that the final reduction takes place in the slag on top of the metal bath.

1.3. Inventors

P.E. Cavanagh

1.4. Companies involved in development

Ontario Research Foundation,
Sheridan Park,
Ontario, Canada.

2. General description (schematic drawing)

See drawing

3. Plant required

Apart from the jet fusion apparatus itself, an oxygen plant is also required.

4. State of development

4.1. Name of laboratories where the process is situated

The process has been studied in the laboratories of the Ontario Research Foundation pilot plant for iron ore concentration and steel making.

4.2. Pilot plant

4.2.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

The cylindrical body of the furnace is in steel with an external diameter of 1.6 m. The internal diameter is 1.10 m, and the inside height 1.22 m.

5. Reducing agents

5.1. Raw materials on which the reducing agent is based

Oxygen and natural gas.

5.2. Process for conversion of above raw materials into a form for use in reduction

Special burner in the upper part of the reactor.

5.3. Condition and analysis of reducing agent at moment of use

No details

5.4. Preheat temperature on entering reduction apparatus

No details

5.5. Gas velocity in reduction apparatus

High velocity gas flame

6. Ores and ferruginous materials

6.1. Source

No details

6.2. Analysis

Magnetite and hematite concentrates in the range 63-68% Fe.

6.3. Size

Usually - 60 mesh (< 0.250 mm)

7. Additions

7.1. Material

Lime

8. Temperature zones in reduction
in $^{\circ}\text{C}$ in the reduction apparatus

The flame temperature drops from 1980°C at the top to 1540°C at the bottom.

9. Pressure zones in reduction
in kg/cm^2 in the reduction apparatus

The reduction takes place at atmospheric pressure.

10. Product
 of the reduction apparatus

10.1. Form

Liquid

10.2. Analysis

Pig iron of variable carbon content

10.3. Degree of reduction

The degree of reduction of the ore to liquid metal (more or less carburised) lies between 20 and 70%. In order to improve this result the ore has been pre-reduced.

11. Further treatment of the product (10) of the reduction apparatus

The product may be subjected to a suitable refining process depending on the analysis obtained.

12. Final product obtained after treatment according to 11

No details

13. Iron yield

No details

14. Utilisation of final product

Destination of final product (blast furnace, electric or other furnace). Technical and economic differences compared with the use of pig iron and scrap

No details

15. Consumption data per tonne of final product (12)

The following results relate to estimates by R.L. Cavanagh for a pilot plant of one tonne per day.

15.1. Raw materials for reduction (5.1)

1520 kg of ore at 66% Fe.

15.2. Fuels and thermal values required for the entire process (fuel characteristics) (thermal balance)

1600 Nm³ natural gas per tonne. There is evidently a considerable excess of reducing gas.

15.3. Electrical energy

No details

15.4. Steam used

None

15.5. Oxygen

1240 Nm³ O₂ per tonne

16. Overall capital investment

No details

17. Opinions expressed on the process:

No details

18. Bibliography

18.1. Articles and publications

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18.2. Patents

No details.

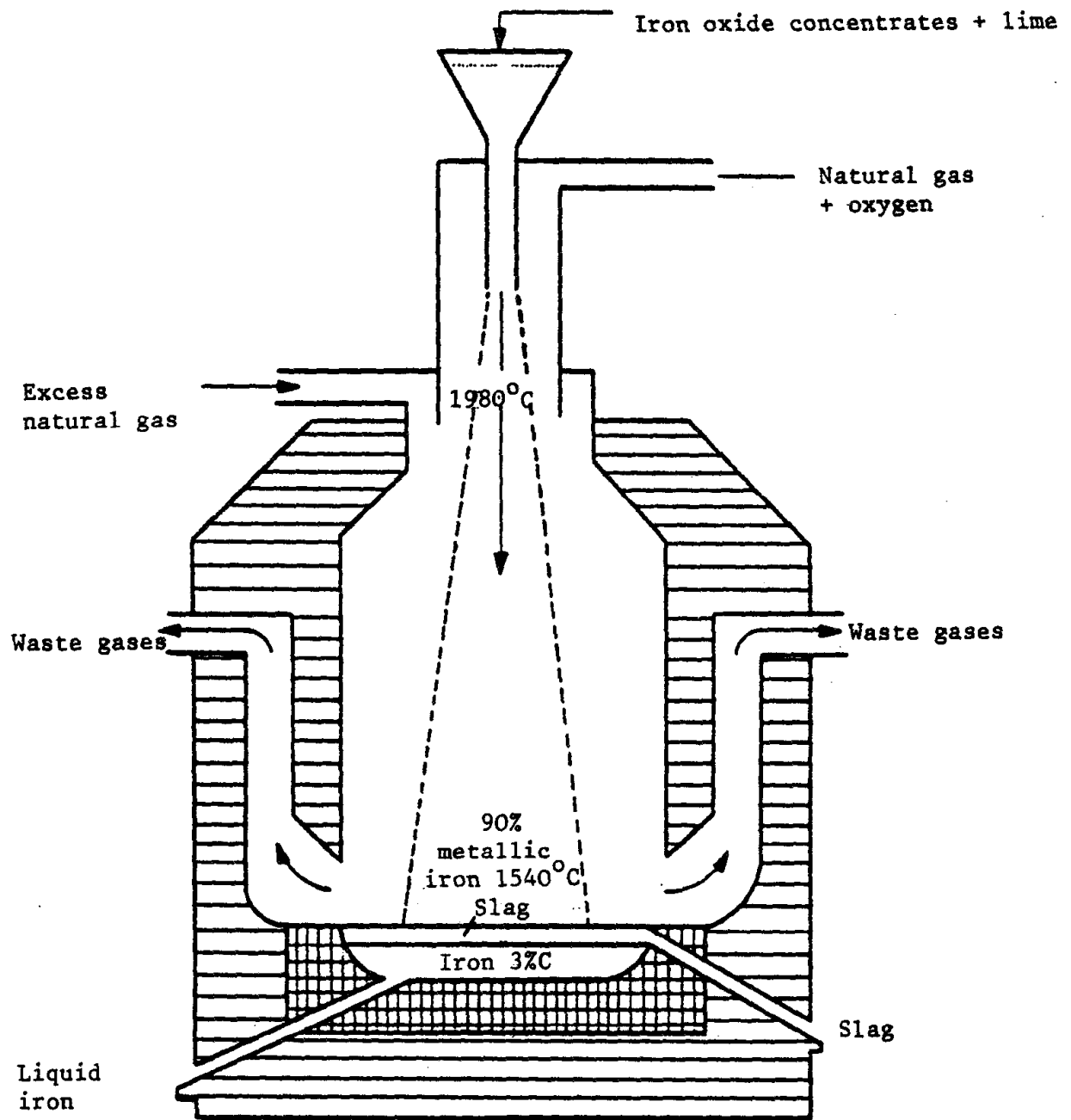


Fig. 1 Jet Smelting

KALLING-AVESTA-PROCESS

1. Characteristics

1.1. Name and history of process

Kalling (Avesta method)

The process was developed by Avesta Iron and Steel Works during the years 1928-35.

1.2. Principle of process

Reduction by means of solid fuel (coal) in a rotary kiln. The heat is supplied to the reduction zone by an electric current across the charge between two annular electrodes.

1.3. Inventors

Bo Kalling

1.4. Companies involved in development

Avesta Iron and Steel Works.

2. General description (schematic drawing)

See drawing

3. Plant required

Rotary reduction furnace.

Magnetic concentration plant.

4. State of development

4.1. Name of laboratories where the process is situated

No details

4.2. Pilot plant

4.2.1. Location

Avesta (Sweden)

4.2.3. Capacity in tonnes per 24 h of final product

5-6 tonnes metallic Fe per 24 h.

5. Reducing agents

5.3. Condition and analysis of reducing agent at moment of use

Charcoal or coke, of small size.

6. Ores and ferruginous materials

6.1. Source

Swedish ores

6.2. Analysis

60 - 65 % Fe.

6.3. Size

Fine ore

- 7. Additions
No details

- 8. Temperature zones in reduction
in °C in the reduction apparatus

No details

- 9. Pressure zones in reduction
in kg/cm² in the reduction apparatus

No details

- 10. Product
of the reduction apparatus

No details

- 10.1. Form

Sponge iron

- 10.2.
to No details
- 10.4.

- 11. Further treatment of the product (10) of the reduction apparatus

- 11.1. Method of cooling

In a cooler on leaving the furnace

- 11.2. Crushing, compacting or briquetting

No details

11.3. Details of final size grading

No details

11.4. Method of concentration (e.g. magnetic separation or sintering)

Desulphurisation and magnetic enrichment.

12. Final product obtained after treatment according to 11

12.1. Form

Dust

12.2. Analysis

Total Fe	Metallic Fe	S	Degree of reduction
94.5	88	0.010	93 %

13. Iron yield

No details

14. Utilisation of final product

Destination of final product (blast furnace, electric or other furnace). Technical and economic differences compared with the use of pig iron and scrap

No details

15. Consumption data per tonne of final product (12)

15.1. Raw materials for reduction (5.1)

Coke and coal of small size: 530 kg per tonne Fe in the sponge

- 15.2. Fuels and thermal values required for the entire process
(fuel characteristics) (thermal balance)

No details

- 15.3. Electrical energy

- 15.3.1. For heating

1,000 kWh per tonne Fe in the sponge (reduction and
desulphurisation)

- 15.4

to No details

- 15.9

16. Overall capital investment

No details

17. Opinions expressed on the process:

The pilot plant at Avesta operated for long periods producing
iron powder, but it ceased operation several years ago.

18. Bibliography

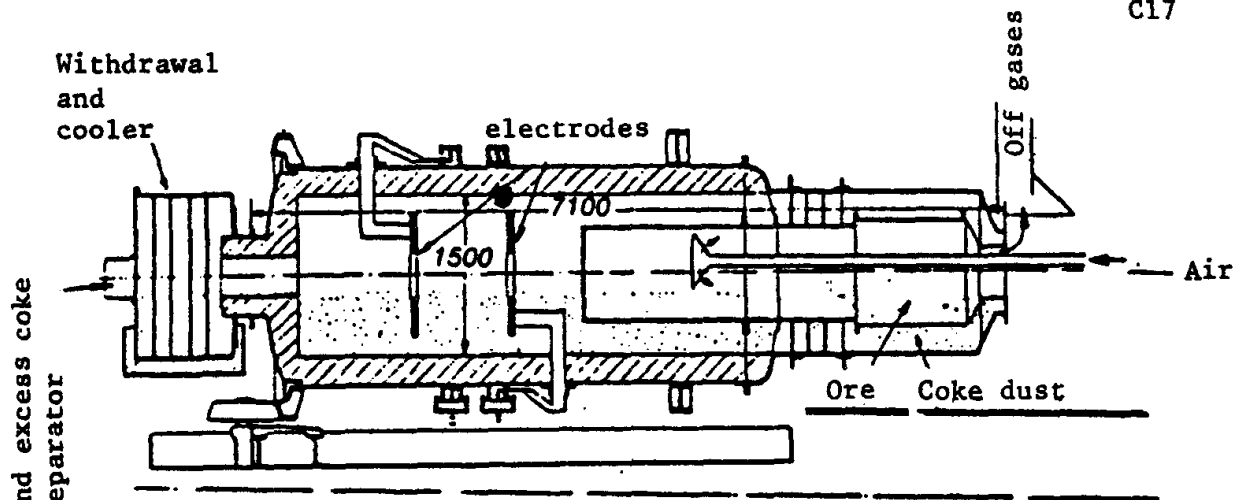
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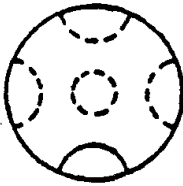
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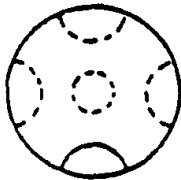
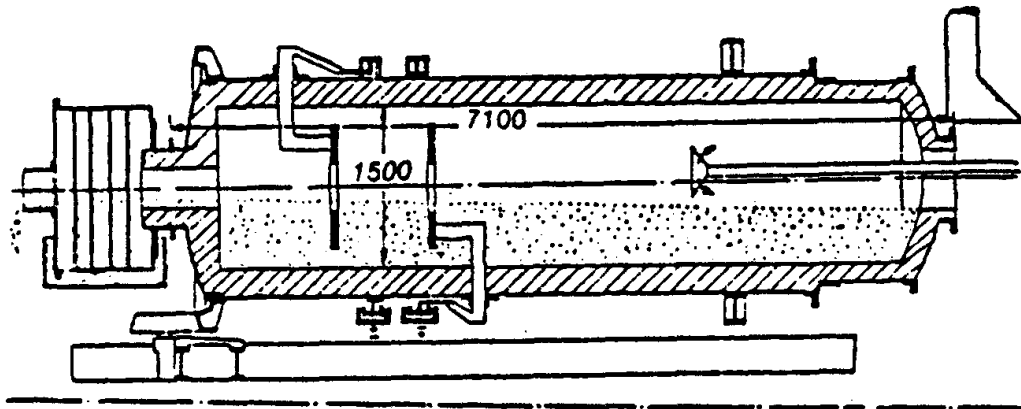
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Sponge iron and excess coke
to magnetic separator



1. Avesta kiln (Sweden)
for producing sponge iron
First design.



2. Modified design.

Fig. 1 Kalling-Avesta Process.

KALLING DOMNARVET PROCESS

1. Characteristics

1.1. Name and history of process

Kalling (Domnarvet method)

1.2. Principle of process

Reduction by means of solid fuels (coal) in a rotary kiln. The heat necessary is supplied by the combustion of part of the fuel by means of hot air fed in along the axis of the kiln.

1.3. Inventors

Bo Kalling

1.4. Companies involved in development

Domnarvet Iron and Steel Works.

2. General description (schematic drawing)

See drawing

3. Plant required

Rotary reduction kiln

Magnetic concentration plant

4. State of development

4.1. Name of laboratories where the process is situated

Preliminary tests at the Högfors works in 1939.

4.2. Pilot plant

4.2.1. Location

Domnarvet (Sweden)

4.2.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

Internal diameter: 1 m, length: 3.50 m.

4.2.3. Capacity in tonnes per 24 h of final product

3 tonnes Fe reduced per 24 h.

4.3. Industrial installations

No details

5. Reducing agents

5.1. Raw materials on which the reducing agent is based

Small coke (80%C) crushed to 2 mm

5.2

to No details

5.6

6. Ores and ferruginous materials

6.1. Source

Swedish siliceous hematite (Striberg)

6.2. Analysis

Fe	MnO	MgO	CaO	Al ₂ O ₃	SiO ₂	F ₂ O ₅	O
55.4	0.1	0.7	2.1	1.0	18.0	0.04	0.01

6.3. Size

< 12 mm

6.4. Preparation

Crushing

7. Additions

No details

8. Temperature zones in reduction
in °C in the reduction apparatus

1075-1150°C

9. Pressure zones in reduction
in kg/cm² in the reduction apparatus

No details

10. Product
of the reduction apparatus

10.1. Form

Crude sponge iron

10.2. Analysis

Fe _{tot}	Fe _{met}	Mn	MgO	CaO	Al ₂ O ₃	SiO ₂	P ₂ O ₅	S	C
68.0	57.5	0.01	0.8	2.5	2.1	23.1	0.04	0.21	1.2

10.3. Degree of reduction

10.3.2. Metallic iron as a percentage of the total iron content of the product

84.5%

11. Further treatment of the product (10) of the reduction apparatus

11.1. Method of cooling

Water cooling in a rotating drum

11.2. Crushing, compacting or briquetting

None

11.3. Details of final size grading

None

11.4. Method of concentration (e.g. magnetic separation or sintering)

Magnetic separation

12. Final product obtained after treatment according to 11

12.1. Form

No details

12.2. Analysis

Total Fe	Metallic Fe	S	C	Degree of reduction
91.8	83.0	0.23	0.35	90.3

13. Iron yield

13.1. Iron contained in product at (10) as a percentage of iron in the charge

No details

13.2. Iron contained in final product (12) as a percentage of iron in the charge

91 %

14. Utilisation of final product

Destination of final product (blast furnace, electric or other furnace). Technical and economic differences compared with the use of pig iron and scrap

No details

15. Consumption data per tonne of final product (12)

15.1. Raw materials for reduction (5.1)

-

15.2. Fuels and thermal values required for the entire process (fuel characteristics) (thermal balance)

Total consumption of 650 kg per tonne of iron contained in the sponge.

16. Overall capital investment

No details

17. Opinions expressed on the process in relation to:

17.1. State of development

A plant with an annual production of 10,000 tonnes was built in Kenya (4) but is no longer in operation

18. Bibliography

18.1. Articles and publications

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18.2. Patents

Cf. Kalling (Avesta) process

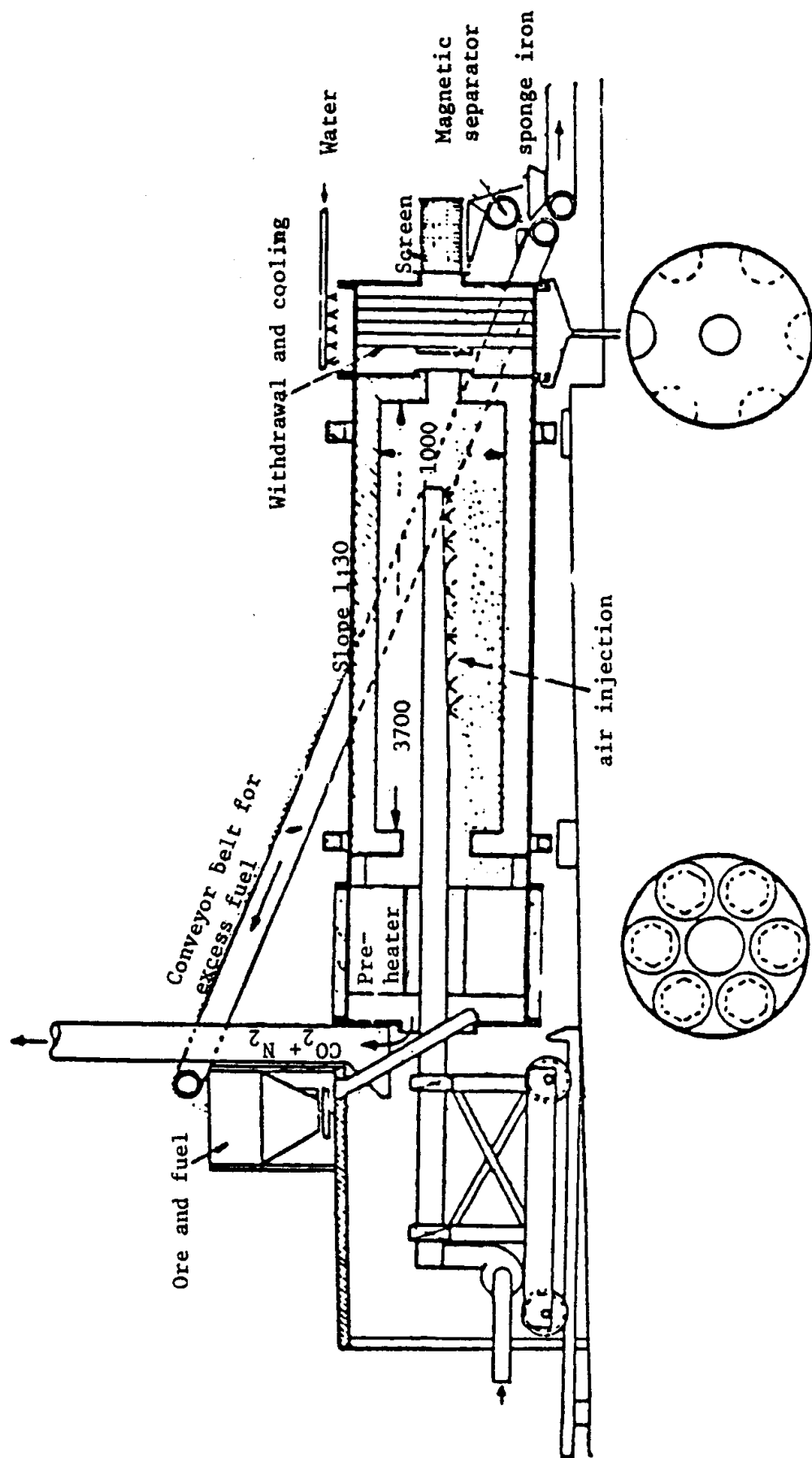


Fig. 1 Sponge iron kiln, pilot plant.

MADARAS PROCESS

1. Characteristics

1.1. Name and history of process

The Madaras process is a modified version of the Hyl process; its promoter, J D Madaras, was in fact involved in the development of the Hyl process.

1.2. Principle of process

Batch reduction of rich pellets by reformed natural gas. The sandy, very lean ores (approximately 7% Fe) of Arizona are beneficiated by magnetic concentration; the concentrates up to 70% iron are then pelletised and reduced. The reduction furnace is a shaft furnace with stationary charge capable of operating under pressure. The reducing gas, reformed from natural gas and containing about 60% H₂ and 38% CO, passes through the bed of pellets under a pulsating pressure of about 2 kg/cm². Reduction occurs at a temperature of approximately 1040°C (1900°F). The sponge iron is either converted into iron powder, or briquetted, or charged directly into an arc furnace to be converted into steel.

1.3. Inventors

Julius D Madaras

1.4. Companies involved in development

Arkota Steel Corp., Coolidge, Arizona, U.S.A.

2. General description (schematic drawing)-

See Figure

3. Plant required

Catalytic reformer)
Thermal reformer) for natural gas
reduction furnace.

Ancillary equipment:
charging and discharging of furnace;
purification of gases after reduction.

4. State of development

4.1. Name of laboratories where the process is situated
Arkota Steel, Coolidge, Arizona.

4.2. Pilot plant

4.2.1. Location

Coolidge, Arizona, U.S.A.

4.2.2. Dimensions of apparatus and plant (e.g. height/length and
diameter of the shaft of a reduction furnace)

Reduction furnace - diameter 2.4 m

- height 3.0 m

4.2.3. Capacity in tonnes per 24 h of final product

75 t per 24 h = 25,000 tpy per furnace

Batch operation; the charge of 14 - 16 tonnes of pellets
remains stationary during reduction.

4.3. Industrial installations

None

5. Reducing agents

5.1. Raw materials on which the reducing agent is based

Natural gas

5.2. Process for conversion of above raw materials into a form for use in reduction

The natural gas is first desulphurised and then reformed in the presence of steam and a $\text{Ni-Al}_2\text{O}_3$ based catalyst. The steam is then removed by condensation and a set proportion of fresh methane and recycled gases is added to the reformed gas. These recycled gases have first been cooled and the dust extracted. The mixture of these gases then passes into a refractory-lined thermal reformer, heated by combustion of natural gas.

5.3. Condition and analysis of reducing agent at moment of use

60% H_2 and 38% CO

5.4. Preheat temperature on entering reduction apparatus

1040°C (1900°F)

5.5. Gas velocity in reduction apparatus

Pulsed injection of the gas at 15 - 20 cycles per minute.

5.6. Analysis and temperature of gas after passing through reduction apparatus

No details

6. Ores and ferruginous materials

6.1. Source

Very lean, titaniferous magnetites of the Arkota deposits,
30 km from the works.

6.2. Analysis

6 - 7% Fe, approximately 6% Ti, high P content

6.3. Size

Sand

6.4. Preparation

The ferruginous sands pass along a magnetic belt, which gives a concentrate containing 62% Fe. This primary product is finely crushed in a wet ball mill. Wet magnetic separation increases the iron content of the concentrate to 71%; the gangue content is less than 1.5%. The concentrates are pelletised; the pellets are dried and then fired to give them the desired strength.

7. Additions

None

8. Temperature zones in reduction
in °C in the reduction apparatus

Approximately 1040°C (1900°F)

9. Pressure zones in reduction
in kg/cm² in the reduction apparatus

2.1 - 2.8 kg/cm² (30 - 40 psi)

10. Product

10.1. Form

Sponge iron, of the shape of the initial pellets.

10.2. Analysis

No details

10.3. Degree of reduction

No details

11. Further treatment of the product (10) of the reduction apparatus

The sponge iron is either crushed to powder or briquetted

12. Final product obtained after treatment according to 11

See 11

13. Iron yield

No details

14. Utilisation of final product

Destination of final product (blast furnace, electric or other furnace). Technical and economic differences as compared with using pig iron and scrap

The sponge iron is melted and treated in an arc furnace. The iron obtained is very pure and contains less than 0.01% C + S + P.

15. Consumption data per tonne of final product (12)

15.1. Raw materials for reduction (5.1)

Concentrated ore (pellets): no figures.

15.2. Fuels and thermal values required for the entire process
(fuel characteristics) (thermal balance)

417,000 kcal for reforming

390,000 kcal for heating the gas

500,000 kcal for heating the ore

Assuming a heat yield of 75%, this requires 180 Nm³ natural gas per tonne of product. To this must be added the 150 Nm³ methane used to produce the reducing gas. The total consumption of natural gas is thus 330 Nm³ per tonne of product.

15.3. Electrical energy

No details

15.4. Steam used

No details

15.5. Oxygen

No details

15.6. Cooling water

No details

15.7. Refractories

No details

15.8. Man hours of productive labour

No details

15.9. Labour and materials for routine maintenance

No details

16. Overall capital investment

No figures possible

17. Opinions expressed on the process

J D Madaras, who also helped to develop the Hyl process, has drawn the following comparison between the two processes (see 18.1): The Madaras process uses only 60% of the gas needed for the Hyl process. This is partly due to the better penetration of the reducing gas in the charge, because of the pulsating pressure.

Capital expenditure is only 40% of that for Hyl.

Process and plant for reforming natural gas are less costly.

Process in 1 stage as against 2 for Hyl.

It is not known whether any development work has been done on the Madaras process since the last description in the 3rd edition of 1967.

18. Bibliography

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18.2. Patents

Not known.

NU - IRON PROCESS

1. Characteristics

1.1. Name and history of process

Nu-Iron process (formerly U.S. Steel process or Shipley process)

Note: the process has been further developed as the HIB
(High Iron Briquette) process. (See A 4)

1.2. Principle of process

Reduction with H_2 and CO at $600-700^{\circ}C$ in the fluidised bed.

1.3. Inventors

E.H. Shipley and Th.F. Reed, Pittsburgh (U.S. Steel Corporation)

1.4. Companies involved in development

United States Steel Corporation, (USA)

2. General description (schematic drawing)

See drawing

3. Plant required

H_2O/CO mixture producer, gas scrubber and return, fluidised
bed ore preheater, fluidised bed reactor in two stages. 1st
stage, mean composition FeO ; 2nd stage O_2/Fe ratio = 0.152 max.

4. State of development

4.1. Name of laboratories where the process is situated

U.S. Steel Corporation

4.2. Pilot plant

4.2.1. Location

South Works of U.S. Steel, Chicago, Illinois, USA.

4.2.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

Two stage fluidised bed reactor: for the first experimental phase a shaft 42 cm in diameter and 3 m high was used; for the second phase a shaft of the same diameter and 6.10 m high was used.

4.2.3. Capacity in tonnes per 24 hr of final product

100-200 kg ore per hour, corresponding to 2-3 tpd sponge iron.

4.3. Industrial installations

No details

5. Reducing agents

5.1. Raw materials on which the reducing agent is based

Natural gas

5.2. Process for conversion of above raw materials into a form for use in reduction

Conversion of the natural gas with steam using catalysts at 870-980°C.

5.3. Condition and analysis of reducing agent at moment of use

According to the patents of Shipley and Reed, a gas with 74.7 % H_2 , 13.8 % CO, 1.5 % H_2O , 0 % CO_2 and 10 % N_2 is used.

5.4. Preheat temperature on entering reduction apparatus

870°C

5.5. Gas velocity in reduction apparatus

0.45 m/s

5.6. Analysis and temperature of gas after passing through reduction apparatus

According to the patent of Reed the gas composition is as follows:

In the first fluidised
bed stage (end product

FeO)

7.2 % CO

6.6 % CO_2

48.7 % H_2

16.0 % H_2O

10.0 % N_2

In the second fluidised
bed stage (end product

Fe)

9.4 % CO

4.4 % CO_2

58.9 % H_2

17.3 % H_2O

10.0 % N_2

6. Ores and ferruginous materials

6.1. Source

Venezuela

6.2. Analysis

Fine iron-rich ores

6.3. Size

< 12.7 mm

6.4. Preparation

The ore is crushed in a closed circuit to < 1.65 mm

7. Additions

None

8. Temperature zones in reduction
in °C in the reduction apparatus

Preheating of the ore in the fluidised bed at 850-1000°C,
entry temperature of the reduction gas 800-930°C,
reduction temperature in the two stages 600-760°C

9. Pressure zones in reduction
in kg/cm² in the reduction apparatus

0 to 4 kg/cm²

10. Product
of the reduction apparatus

Powder

10.1. Form

No details

10.2. Analysis

No details

10.3. Degree of reduction

Variable up to 95 %

10.4. Utilisation of byproducts

No details

11. Further treatment of the product (10) of the reduction apparatus

11.1. Method of cooling

No details

11.2. Crushing, compacting or briquetting

Briquetting

11.3. Details of final size grading

No details

11.4. Method of concentration (e.g. magnetic separation or sintering)

No details

12. Final product obtained after treatment according to 11

12.1. Form

Sponge iron powder or briquettes

12.2. Analysis

No details

13. Iron yield

Except where the ore is concentrated after reduction, there are no losses at this stage of treatment. However, this does not apply for the processing of the product.

14. Utilisation of final product

Destination of final product (blast furnace, electric or other furnace). Technical and economic differences compared with the use of pig iron and scrap

Used in steelworks.

Few details are known regarding the advantages or special features for use instead of scrap.

15. Consumption data per tonne of final product (12)

No details

16. Overall capital investment

No details

17. Opinions expressed on the process in relation to:

17.1. State of development

The process has meanwhile been developed further as the HIB process. It is in large-scale industrial use (see A 4).

17.2. Practical operating difficulties

Risk of fritting together of the ore

17.3. Possibilities of exploitation

Cf. A 4.

17.4. Special requirements to be met by certain materials
(e.g. refractories)

No details

18. Bibliography

18.1. Articles and publications

- 1) Hegarty, A. Iron and Steel, 30, 1957, No.11, pp. 483-485.
- 2) Chem. Ing. 29, 1959, p. 42.
- 3) Reed, T.F., I.C. Agarwal and E.H. Shipley: Nu-Iron, a fluidised bed reduction process. Paper presented to the Congress in Chicago, May 1960, cf. J. Metals, 12, 1960, No. 4, pp. 317-20.
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- 9) Reed, T.F., J.C. Agarwal and E.H. Shipley. Iron Ore Reduction, 1962, pp. 174-188.
- 10) Astier, J. Rev. Métallurg., 58, 1961, No.1, pp. 3-11; cf. J. Metals, 13, 1961, No. 4. pp. 305-307.

- 11) McQuaid, H.W. Iron Steel Eng. 35, 1958, No.1, pp. 68-78.
- 12) Lubker, R.A. and K.W. Bruland. Amer. Inst. of Chem. Eng., New York, 1963, pp. 84-92.

18.2. Patents

- U.S. Patent no. 2,752,234 (Shipley), filed 7th July 1955, granted 26th June 1956.
- U.S. Patent no. 2,848,316. Process for gaseous reduction of iron ore. U.S. Steel, 7th June 1956, 19th August 1956.
- U.S. Patent no. 2,864,688 (Reed), filed 28th January 1958, granted 16th December 1958.
- U.S. Patent no. 2,876,091. Method of reducing iron oxide. U.S. Steel. 31st December 1957, 3rd March 1959.
- U.S. Patent no. 2,915,379, U.S. Steel, 1st December 1959.
- U.S. Patent no, 2,929,703, U.S. Steel, 22nd March 1960.
- French Patent no. 1,216,401, 21st November 1958.
- French Patent no. 1,216,401, 21st November 1958.
- French Patent no. 1,217,047, 15th November 1958.

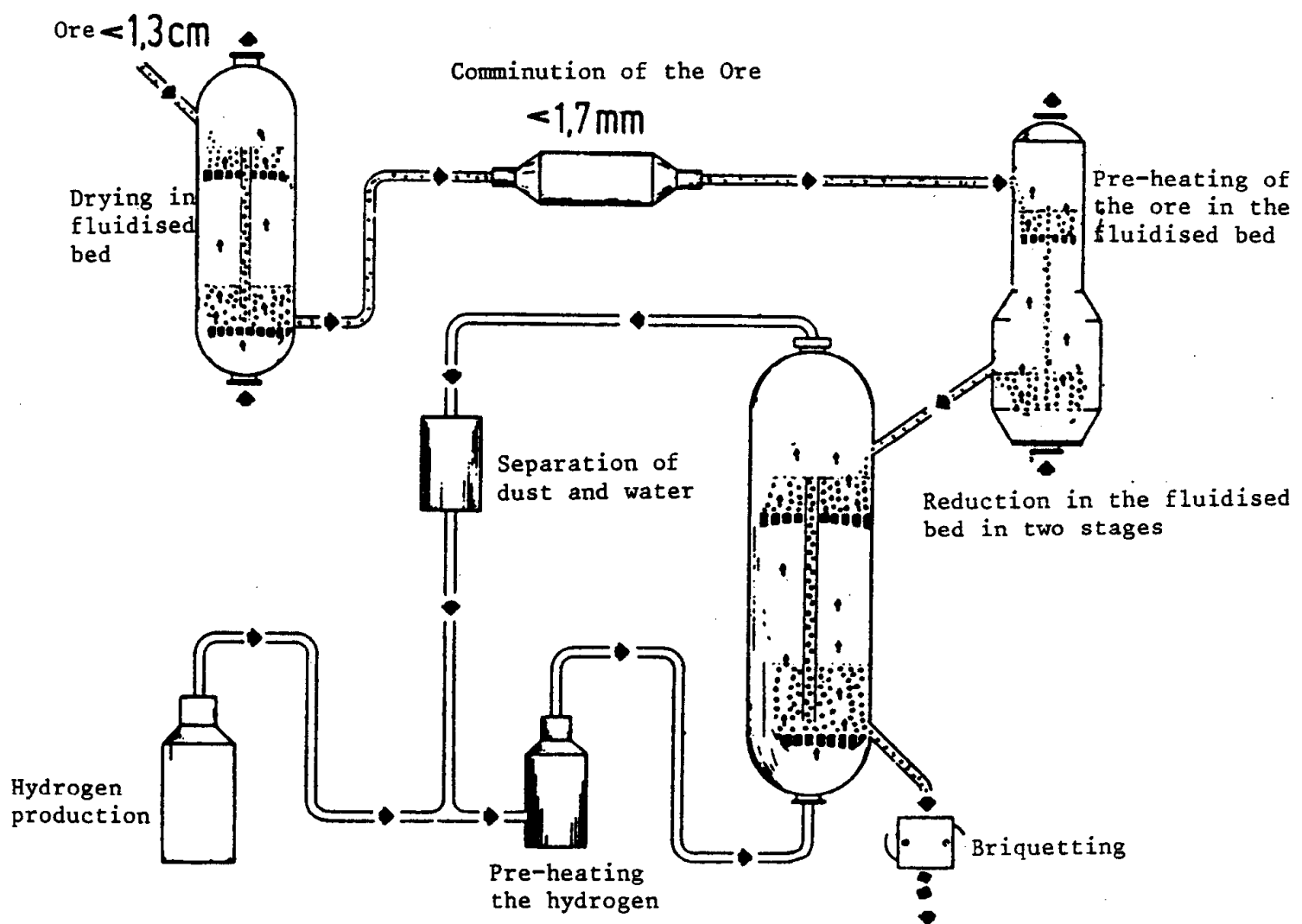


Fig. 1. Diagram of the Nu-Iron process

ORCARB PROCESS

1. Characteristics

1.1. Name and history of process

Orcarb process

1.2. Principle of process

Pelleting and agglomeration of ores with coking coals having a high volatile content, pre-reduction in a rotary kiln and melting in an electric furnace.

1.3. Inventors

E G Lesher

1.4. Companies involved in development

Swindell-Dressler Corporation, Pittsburgh, Pennsylvania

2. General description (schematic drawing)

See drawing.

Extensive pre-reduction takes place in a rotary kiln and the final reduction accompanied by melting in an electric furnace. The originality of the process lies in the preparation of the materials, in a manner specific to Orcarb. The ore concentrates and flux are first fed by screw conveyor to a rotary kiln heated internally by an axial burner to a temperature around 600°C. The product obtained is poured through an insulated hopper on to a conveyor with high volatile coal fines and fed at approximately 350°C into an unlined metal drum, rotating at a peripheral speed of 25 m/min. The drum (Orcarb retort) is heated externally; softening, pelletising and semi-coking of the coal occur, and the

product leaving the retort is fed into the rotary pre-reduction kiln.

The temperature in the rotary reduction kiln is 1030-1190°C. The necessary heat is supplied by a burner through the discharge part. The product may be used directly for making steel or stored and/or transported. The intermediate product, "an agglomerate of coal and iron oxides" may also be used as blast furnace burden.

The tar and gas produced in the agglomeration phase are used for the thermal requirements of the same cycle. When the iron oxide fines contain oxides of zinc and lead, these latter are also reduced and the respective metals may be recovered from the off-gases. The operation is continuous.

3. Plant required

One pre-heating kiln
One heated pelletising retort
One rotary pre-reduction kiln
One electric arc furnace

4. State of development

4.2. Pilot plant

Swindell-Dressler Pilot Plant

4.2.1. Location

Indianola, Pennsylvania, USA, approximately 15 km NE of Pittsburgh

4.2.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

Pre-heating kiln: 0.92 m internal diameter, 3.66 m long (3 x 12 ft)

Rotary pre-reduction kiln: 0.76 m internal diameter, 6.1 m long (2.5 x 20 ft), refractory lining 15 cm (6") thick.

Electric arc melting furnace: hearth 76 cm (2.5 ft) internal diameter, three 10.2 cm (4") electrodes in a triangular arrangement.

4.2.3. Capacity in tonnes per 24 h of final product

1.1 - 1.6 tonnes per 24 hours in continuous operation

4.3. Industrial installations

None

5. Reducing Agents

5.1. Raw materials on which the reducing agent is based

High volatile coking coal, preferably with low sulphur content.

5.2. Process for conversion of above raw materials into a form for use in reduction

Semi-coking

5.3. Condition and analysis of reducing agent at moment of use

Fines 0 - 5 mm (below 4 mesh)

5.4. Preheat temperature on entering reduction apparatus

Approximately 470°C

6. Ores and ferruginous materials

6.1. Source

Ore fines may be used, and also material recovered from iron and steel works (dust, scale, etc.).

6.2. Analysis

Adequate iron content

6.3. Size

Fine and very fine material (much of it <5 microns) which could not be used for normal sintering or pelletising.

7. Additions

7.1. Material

Fluxes

8. Temperature zones in reduction
in °C in the reduction apparatus

1030 - 1190°C

9. Pressure zones in reduction
in kg/cm² in the reduction apparatus

No details

10. Product
of the reduction apparatus

Degree of reduction in rotary kiln 80%, varying with the amount of coal used.

11. Further treatment of the product (10) of the reduction apparatus

Final melting and reduction in the electric furnace

12. Final product obtained after treatment according to 11

12.1. Form

Liquid metal

12.2. Analysis

More than 2% C, less than 7% Si

13. Iron yield

90 - 95%

14. Utilisation of final product

Destination of final product (blast furnace, electric or other furnace). Technical and economic differences as compared with using pig iron and scrap

Same as pig iron

15. Consumption data per tonne of final product (12)

Figures for the pilot plant

15.1. Raw materials for reduction (5.1)

The composition of the solid charge varies according to the characteristics of the materials to be reduced.

For example we may have (by weight):

75% ferruginous materials

25% coal

or

65% ferruginous materials

25% coal

10% flux

During the "agglomeration" heating the coal loses 25-30% of its weight, producing gas and tar which can be used in the same plant.

50-80% of the initial sulphur remains in the pre-reduced product.

15.2. Fuels and thermal values required for the entire process
(fuel characteristics) (thermal balance)

The heat consumption for the heating, in addition to the coal admixed, is distributed as follows:

50% in the drier/pre-heater

19% in the agglomeration kiln

31% in the reduction kiln

Thermal yield: 40%

Additional heat:

3.5% tar oil (8250 kcal/kg)

20.1% carbonisation gas (7150 kcal/Nm³)

76.4% natural gas (8900 kcal/Nm³)

Total consumption, including the reducing coal, for an ore with 65% Fe and with an iron recovery of 95% is 4,520,000 - 5,020,000 kcal per tonne of product.

15.3. Electrical energy

1,100 KWh is required per tonne of metal produced in the electric furnace.

16. Overall capital investment

No recent figures available

17. Opinions expressed on the process in relation to:

17.1. State of development

Process studied on pilot plant since the beginning of 1959.
(See 18.1. 4).

17.2. Practical operating difficulties

No details

17.3. Possibilities of exploitation

Process particularly suitable for processing iron oxide fines recovered in cleaning plants.

18. Bibliography

18.1. Articles and publications

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Tidsskr Kjerni, Bergvesen Met., 1954 14 November,
p. 159-165.
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p. 944-948 and Journal of Metals, October 1955, 7, no.10;
p.1, 114-1 118.
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San Francisco, 8 March 1960, Iron and Steel Engineer, U.S.A.,
(April 1961), 38, No. 4, p. 123-125⁽¹⁾
- 5) Pilot plant studies new direct reduction process, Iron and Steel Engineer, February 1960, p. 149-150.

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- 7) R.H. Caughes, Mgr - The Orcarb process for steel plant waste oxides. Swindell-Dressler Company, November, 1971.

18.2. Patents

USA Patent No. 2,794,728

USA Patent No. 2,918,364

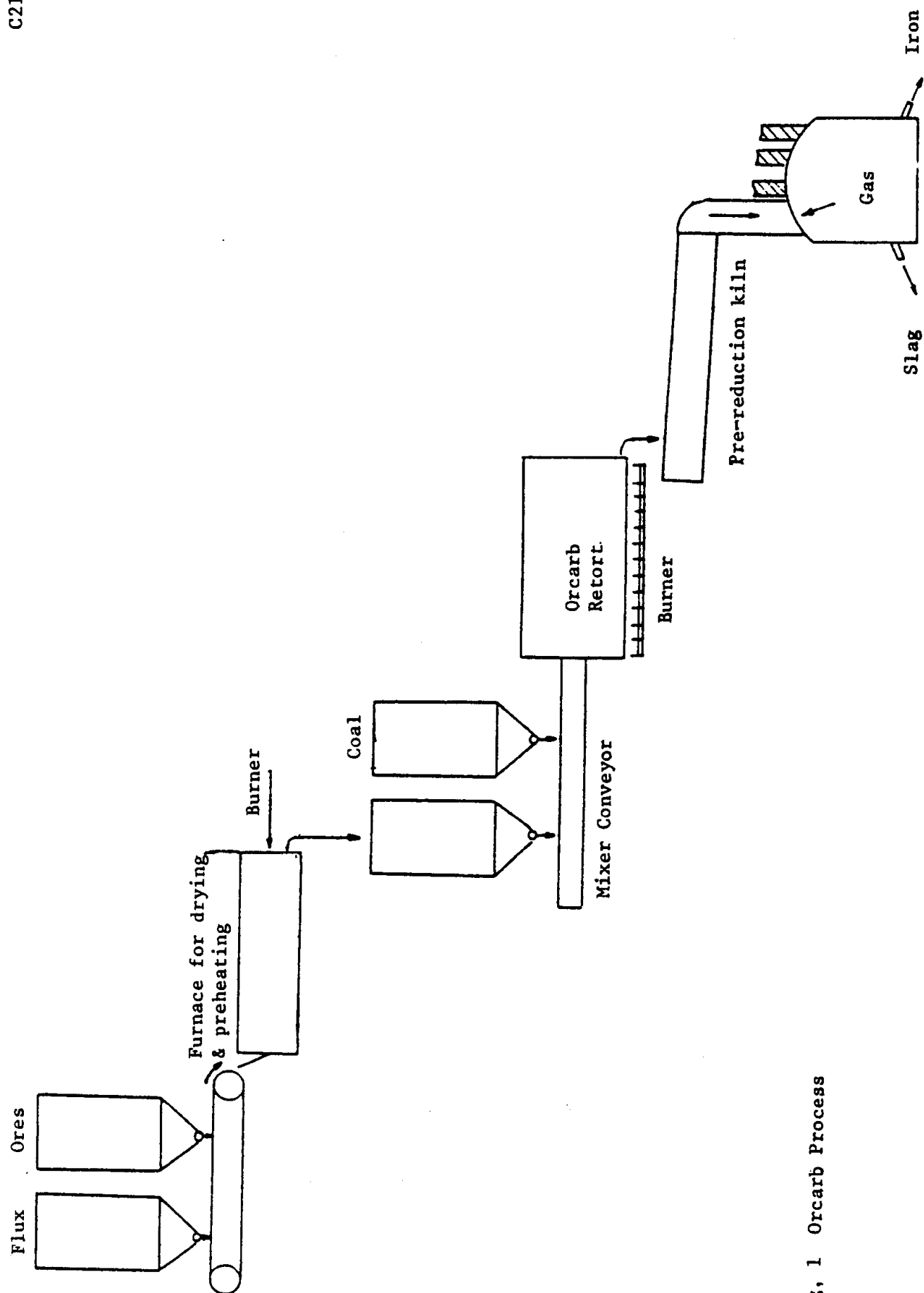


Fig. 1 Orcarb Process

ORF PROCESS

1. Characteristics

1.1. Name and history of process

ORF Direct Steel Process

1.2. Principle of process

Magnetite concentrates (it is a necessary condition that the concentrates contain not more than 0.5% gangue) are oxidised to hematite and subsequently reduced direct to steel by a mixture of $\text{CO} + \text{H}_2$ (3:1). Hematite concentrates may also be used; an alternative reducing gas is H_2 . The bed of concentrates is reduced on a moving band passing through the furnace. At the exit the reduced product can be rolled directly into flat plates.

1.3. Inventors

Ontario Research Foundation

2. General description (schematic drawing)

No details

3. Plant required

Furnace surrounding a moving belt. The reducing gas is passed through the furnace from bottom to top through the bed of ore.
Rolling mill at the exit of the furnace.

4. State of development

4.1. Name of laboratories where the process is situated

Ontario Research Foundation

4.2. Pilot plant

No details

4.3. Industrial installations

No details

5. Reducing agents

5.1. Raw materials on which the reducing agent is based

No details

5.2. Process for conversion of above raw materials into a form for use in reduction

No details

5.3. Condition and analysis of reducing agent at moment of use

Gaseous reducing agent made up of CO and H₂ in the proportion of 3:1.

CO :	70 %	CO ₂ + H ₂ O :	2 %

H ₂ :	26 %	N ₂ etc :	2 %

5.4. Preheat temperature on entering reduction apparatus

1100°C

5.5. Gas velocity in reduction apparatus

No details

5.6. Analysis and Temperature of gas after passing through reduction apparatus

CO :	60 %	CO ₂ + H ₂ O :	15 %
H ₂ :	23 %	N ₂ etc :	2 %

6. Ores and ferruginous materials

6.2. Analysis

Magnetite super-concentrates. They may not contain more than 0.5% gangue, about 72% Fe. (Dry basis).

7. Additions

No details

8. Temperature zones in reduction
in °C in the reduction apparatus

About 1100°C

9. Pressure zones in reduction
in kg/cm² in the reduction apparatus

No details

10. Product
of the reduction apparatus

10.1. Form

More or less compact sheet of steel

10.2. Analysis

No details

10.3. Degree of reduction

The oxides of iron are completely reduced

11. Further treatment of the product (10) of the reducing apparatus

Rolling

12. Final product obtained after treatment according to 11

12.1. Form

Finished rolled product (essentially sheets of maximum width 90 cm and maximum thickness 6.3 mm, alternatively wire and rod of diameter less than 25.4 mm).

13. Iron yield

No details

14. Utilisation of final product

Destination of final product (blast furnace, electric or other furnace). Technical and economic differences compared with the use of pig iron and scrap

No details

15. Consumption data per tonne of final product (12)

No details

16. Overall capital investment

No details

17. Opinions expressed on the process in relation to:

No details

18. Bibliography

18.1. Articles and publications

- 1) P.E. Cavanagh, Direct iron ore reduction, Journal of Metals, December 1958, pp. 804-809.
- 2) P.E. Cavanagh, Direct reduction processes for iron ore, Revue technique luxembourgeoise, 50, no.2, April-June 1958.
- 3) Journées internationales de sidérurgie Liège, Luxembourg, Charleroi, 18-28 June 1958, comptes rendus published by C.N.R.M.

18.2. Patents

No details

REPUBLIC STEEL PROCESS

1. Characteristics

1.1. Name and history of process

Republic Steel

1.2. Principle of process

Extremely pure concentrates are reduced to metallic powder. This is rolled directly into semi-solid strip compact. This strip compact passes through the furnace in which it is heated to 1200°C in a reducing atmosphere and hot rolled into a dense sheet of the desired thickness.

1.3. Inventors

Republic Steel Corporation (USA)

1.4. Companies involved in development

Republic Steel Corporation

2. General description (schematic drawing)

No details

3. Plant required

No details

4. State of development

4.1. Name of laboratories where the process is situated

The research centre of Republic Steel at Independence, Ohio
(near Cleveland).

4.2. Pilot plant

Republic Steel constructed a pilot plant but this was closed
in 1959.

5. Reducing agents

No details

6. Ores and ferruginous materials

Fine super-concentrates

7. Additions

No details

8. Temperature zones in reduction
in °C in the reduction apparatus

No details

9. Pressure zones in reduction
in kg/cm² in the reduction apparatus

No details

10. Product
 of the reduction
- 10.1. Form

 Metal powder
11. Further treatment of the product (10) of the reduction apparatus

 Primary rolling into a semi-solid strip compact. Secondary hot
 rolling to dense sheet of specified thickness.
12. Final product obtained after treatment according to 11

 Sheet
13. Iron yield

 No details
14. Utilisation of final product

 Destination of final product (blast furnace, electric or
 other furnace). Technical and economic differences compared
 with the use of pig iron and scrap

 No details
15. Consumption data per tonne of final product (12)

 No details
16. Overall capital investment

 No details

17. Opinion expressed on the process in relation to:

17.1. State of development

Republic Steel has completed a general study of the process and is investigating extrapolation on an industrial scale. There are no plans for further development of this process.

18. Bibliography

18.1. Articles and publications

- 1) Two new research laboratories. Republic shows direct strip making technique, Journal of Metals, August 1959, p. 508.
- 2) Direct reduction moves closer, Republic goes from ore to strip without melting, Iron Age, 25 June 1959, p.54.
- 3) Republic scores in direct reduction, Steel, 22 June 1959, p. 44-45.

18.2. Patents

No details

STELLING PROCESS

1. Characteristics

1.1. Name and history of process

Stelling process

1.2. Principle of process

Reduction by CO at 600°C in a fluidised bed.

1.3. Inventors

Otto Stelling and Ivan Pereswettoff-Morath with the help of various Swedish companies.

1.4. Companies involved in development

None

2. General description (schematic drawing)

See drawing

3. Plant required

Production of CO in a coke carburettor.

Recycling of the gases in the carburettor.

Pre-heating of the ore in a fluidised bed with oxidation of the magnetite to hematite (900°C).

Pre-reduction stage in fluidised bed at 750°C.

Reduction stage in fluidised bed at 600°C.

4. State of development

4.1. Name of laboratories where the process is situated

Royal Technical School, Stockholm.

4.2. Pilot plant

4.2.1. Location

Royal Technical School, Stockholm

4.2.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

1 fluidised bed reactor, 35.5 cm in diameter, 2.5 m high

4.2.3. Capacity in tonnes per 24 h of final product

Approximately 100 kg sponge iron per day

4.3. Industrial installations

None

5. Reducing agents

5.1. Raw materials on which the reducing agent is based

Coke

5.2. Process for conversion of above raw materials into a form for use in reduction

Gasification

5.3. Condition and analysis of reducing agent at moment of use

Gas 96% CO and 4% CO₂

Gases with 85% CO or 5% CO₂, 65% CO or 2% CO₂,

10-15% H₂O and 1.5% S have been tested.

- 5.4. Preheat temperature on entering reduction apparatus

300°C

- 5.5. Gas velocity in reduction apparatus

Tests at 17 cm/sec have been reported

- 5.6. Analysis and temperature of gas after passing through reduction apparatus

Depend on the nature of the gases used

6. Ores and ferruginous materials

- 6.1. Source

Concentrates from Swedish mines at Örling, Grangesberg, Rud, Småltarmossa, and Balsjö, and washed pyrites residues.

- 6.2. Analysis

Analysis of mixture of Swedish concentrates:

Fe	62.5 %
Fe ₂ O ₃	62.4 %
FeO	24.3 %
SiO ₂	5.6 %
CaO	1.8 %
MgO	1.8 %
Al ₂ O ₃	0.9 %
MnO	0.7 %
P ₂ O ₅	0.019 %
S	0.22 %
Cu	0.019 %
loss at red heat	2.1 %

- 6.3. Size
 No details
- 6.4. Preparation
 No details
7. Additions
 None
8. Temperature zones in reduction
 Oxidation of ore (if magnetite) at 900°C
 - pre-reduction at 750°C
 - final reduction and carburisation at 600°C
 - production of reducing gases at 950°C cooling to 300°C
 before entry to reduction reactor
9. Pressure zones in reduction
 in kg/cm² in the reduction apparatus
 Low pressure
10. Product
 of the reduction apparatus
- 10.1. Form
 Reduced ore and powdered carbide
- 10.2. Analysis
 Approximately 30% FeO and 50% Fe₃C

10.3. Degree of reduction

10.3.1. Proportion of oxygen removed from iron oxides as a percentage of the total oxygen combined with iron in the ores

Approximately 87%

10.3.2. Metallic iron as a percentage of the total iron content of the product

No details

10.4. Utilisation of by-products

No details

11. Further treatment of the product (10) of the reduction apparatus

11.1. Method of cooling

Not given, but cooling to 200°C

11.2. Crushing, compacting or briquetting

None

11.3. Details of final size grading

No details

11.4. Method of concentration (e.g. magnetic separation or sintering)

None

12. Final product obtained after treatment according to 11

Same as product obtained under 10'

13. Iron yield

Unless the ores are concentrated after reduction there do not seem to be any losses at this stage in the process. The same does not apply to the use of the product.

14. Utilisation of final product

Destination of final product (blast furnace, electric or other furnace). Technical and economic differences as compared with using pig iron and scrap

No special information

15. Consumption data per tonne of final product (12)

No details

16. Overall capital investment

No details

17. Opinions expressed on the process:

None

18. Bibliography

18.1. Articles and publications

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- 2) Stelling, O., Carbon Monoxide Reduction of Iron Ore, Journal of Metals, April 1958, p.290-295.
- 3) Scortecchi, A. "Réduction directe des minerais de fer".
Paper presented to the Congrès International de la sidérurgie, Luxembourg 1958.
- 4) Unterweiser, P.M., Iron Age, 30 January 1958, no.5, p. 93-95.

18.2. Patents

French patent no. 1.094.043 and two Swedish patents.

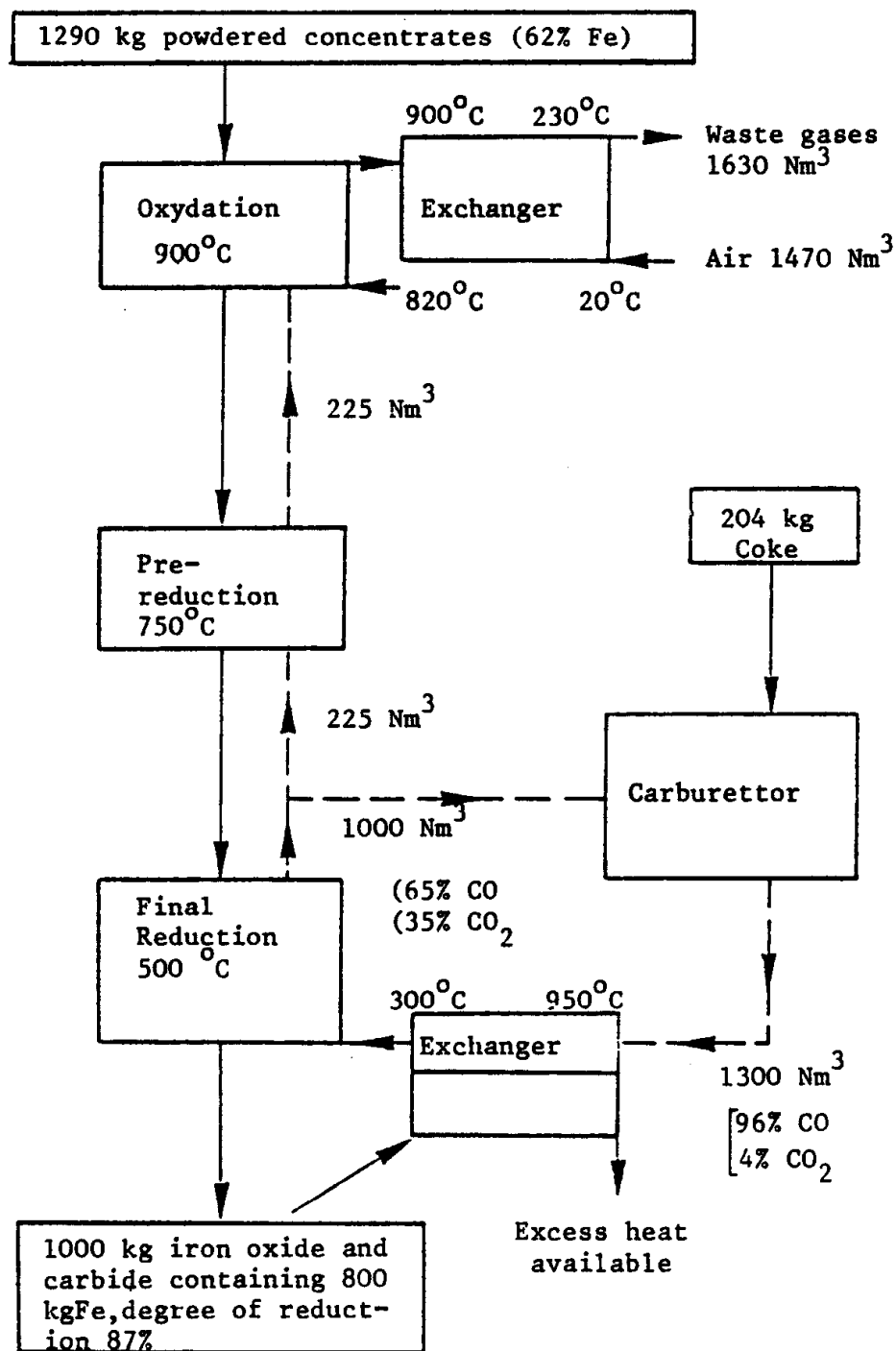


Fig. 1 Stelling Process

STRATEGIC -UDY PROCESS

1. Characteristics

1.1. Name and history of process

Strategic-Udy process

1.2. Principle of process

Pre-heating and pre-reduction in a rotary kiln; final reduction and melting in an electric furnace or other steelworks plant.

1.3. Inventors

Dr Marvin J Udy and Strategic Materials Corp.

1.4. Companies involved in development

Strategic Materials Corporation, 161 East 42nd Street,
New York 17, N.Y. and Koppers Company Inc., Koppers Building,
Pittsburgh/19 Pa.

2. General description (schematic drawing)

See drawing

The mixture of ore, limestone and coal is continuously charged into the rotary kiln, heated in part by the high CO content gases from the electric furnace; the ore is pre-reduced, roasted and heated considerably, and the coal is semi-coked. The products obtained are fed around the periphery of a 3-phase arc furnace, directly in the melting zone, using insulated hoppers.

The originality of the process lies in the operation of the electric furnace, where the space above the electrodes is free, the electrodes being very close to the slag (less than $\frac{1}{2}$ " = 13 mm) or even dipping into it (up to 3", 76 cm, into the bath), and heating is almost entirely by resistance. In plan, the electrodes are 2 m apart or more (Fig.2).

3. Plant required

- a) Plant for preparing the raw materials (crushing and screening)
- b) Rotary furnace for pre-reduction
- c) Electric furnace for final reduction

4. State of development

4.1. Name of laboratories where the process is situated

Strategic Metallurgical and Chemical Processes Ltd., a subsidiary of Strategic Materials Corp.

4.2. Pilot plant

4.2.1. Location

Niagara Falls, Ontario, Canada.

4.2.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

Rotary furnace 24 m (80 ft) long, external diameter 1.8 m (6 ft) and internal diameter 1.4 m.

3 electric furnaces of 1000kVA; oxygen plant, electric furnace, x-ray spectrograph.

4.2.3. Capacity in tonnes per 24 h of final product

Approximately 15-20 tonnes

4.3. Industrial installations

All the Strategic-Udy industrial installations (Matanzas, Niagara Falls) have been abandoned.

5. Reducing agents

5.1. Raw materials on which the reducing agent is based

The reducing agent is coal, high volatile coal, lignite or anthracite. Coals with 0.40% volatiles, lignites with 48% volatiles and coals with up to 3% sulphur have been used successfully.

5.2. Process for conversion of above raw materials into a form for use in reduction

Crushing to 13 mm.

5.3. Condition and analysis of reducing agent at moment of use

Coal fines; the fixed carbon content may vary between 30% and 85%.

5.4. Preheat temperature on entering reduction apparatus

This coal is heated in the pre-reduction unit, the rotary furnace, at the same time as the ores with which it is mixed.

5.5. Gas velocity in reduction apparatus

No details

5.6. Analysis and temperature of gas after passing through reduction apparatus

No details

6. Ores and ferruginous materials

6.1. Source

Tests have been carried out on various ores - ordinary, lateritic and titaniferous, waste from copper production, etc; in particular ores containing sulphur (up to 1% S) or phosphorus (up to 0.8% P) and even fines of 200 (Tyler) mesh (i.e. 0.074 mm) have been processed.

6.2. Analysis

The figures for consumption of materials given under 15 are those for ores containing 57-59% iron, but much leaner ores down to 33% iron can be processed.

6.3. Size

Fines, less than 6 mm and up to 10 mm.

6.4. Preparation

Where required crushing to approximately 6 mm ($\frac{1}{4}$ ") then mixing with the reducing coal and flux.

7. Additions

7.1. Material

Limestone or lime

7.2. Analysis

No special requirement

7.3. Size

Crushing to about 6 mm as for ore

8. Temperature zones in reduction
in °C in the reduction apparatus

The temperature inside the rotary furnace is limited only by the softening point of the ores; in practice it does not exceed 1100°C to prevent sticking and ringing.

9. Pressure zones in reduction
in kg/cm² in the reduction apparatus

Atmospheric pressure

10. Product
of the reduction apparatus

10.1. Form

On leaving the rotary pre-reduction furnace the product appears solid and it is almost the same size as when it went in, that is less than 10 mm.

10.2. Analysis

No details

10.3. Degree of reduction

50 - 60%, in the rotary furnace an even higher rate of reduction of the order of 80% has been attempted.

10.4. Utilisation of by-products

No details

11. Further treatment of the product (10) of the reduction apparatus

The complete reduction and melting take place in an electric arc furnace, 3-phase; the coal coked in the rotary furnace completes the reduction, and more coal may be added in amounts which vary according to the degree of reduction required.

12. Final product obtained after treatment according to 11

12.1. Form

Liquid metal, iron or semi-steel

12.2. Analysis

Carbon content varies, from 0.6% to 4%

13. Iron yield

The only iron losses are to the slag; the iron content of the slag depends largely on whether the furnace is operated to obtain iron or semi-steel.

14. Utilisation of final product

Destination of final product (blast furnace, electric or other furnace). Technical and economic differences as compared with using pig iron and scrap

No special data; iron or semi steel

- 15. Consumption data per tonne of final product (12)
For ores with approximately 58% Fe
- 15.1. Raw materials for reduction (5.1)
430 kg coal with 80% fixed carbon
- 15.2. Fuels and thermal values required for the entire process
(fuel characteristics) (thermal balance)
Reducing coals (see 15.1) plus approximately 8 kg electrodes
- 15.3. Electrical energy
- 15.3.1. For heating
1110 kWh
- 15.3.2. For power
28 kWh
- 15.4. Steam used
- 15.4.1. For reducing agent
None
- 15.4.2. As energy
75 kg
- 15.5. Oxygen
None
- 15.6. Cooling water
No details

15.7. Refractories

No details

15.9. Labour and materials for routine maintenance

No details

16. Overall capital investment

No details

17. Opinions expressed on the process

After the tests on the pilot scale at Niagara Falls industrial-scale units were built, but were later abandoned.

In June 1972 the rotary furnace at Niagara Falls was taken over by Allis Chalmers who modified it to study its own process.

18. Bibliography

18.1. Articles and publications

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18.2. Patents

U.S. - Patent Office n°:

2674 531, Granted 6 April 1954 - Iron Recovery

2805 929, Granted 10 September 1957 - Processes for obtaining Iron from Material Containing Iron Oxides.

2805 930, Granted 10 September 1957 - Process of Producing Iron from Iron-Oxide material.

See also the following patents:

Ferro-Manganese - U.S. Patent n^o:

2549 994, 24 April 1951

2775 518, 25 December 1956

2830 890, 15 April 1958

2830 891, 15 April 1958

Ferro-Chrome - U.S. Patent n^o:

2582 469, 15 January 1952

2845 342, 29 July 1958

Ferro-Ti and Al - U.S. Patent n^o:

2830 892, 14 April 1958

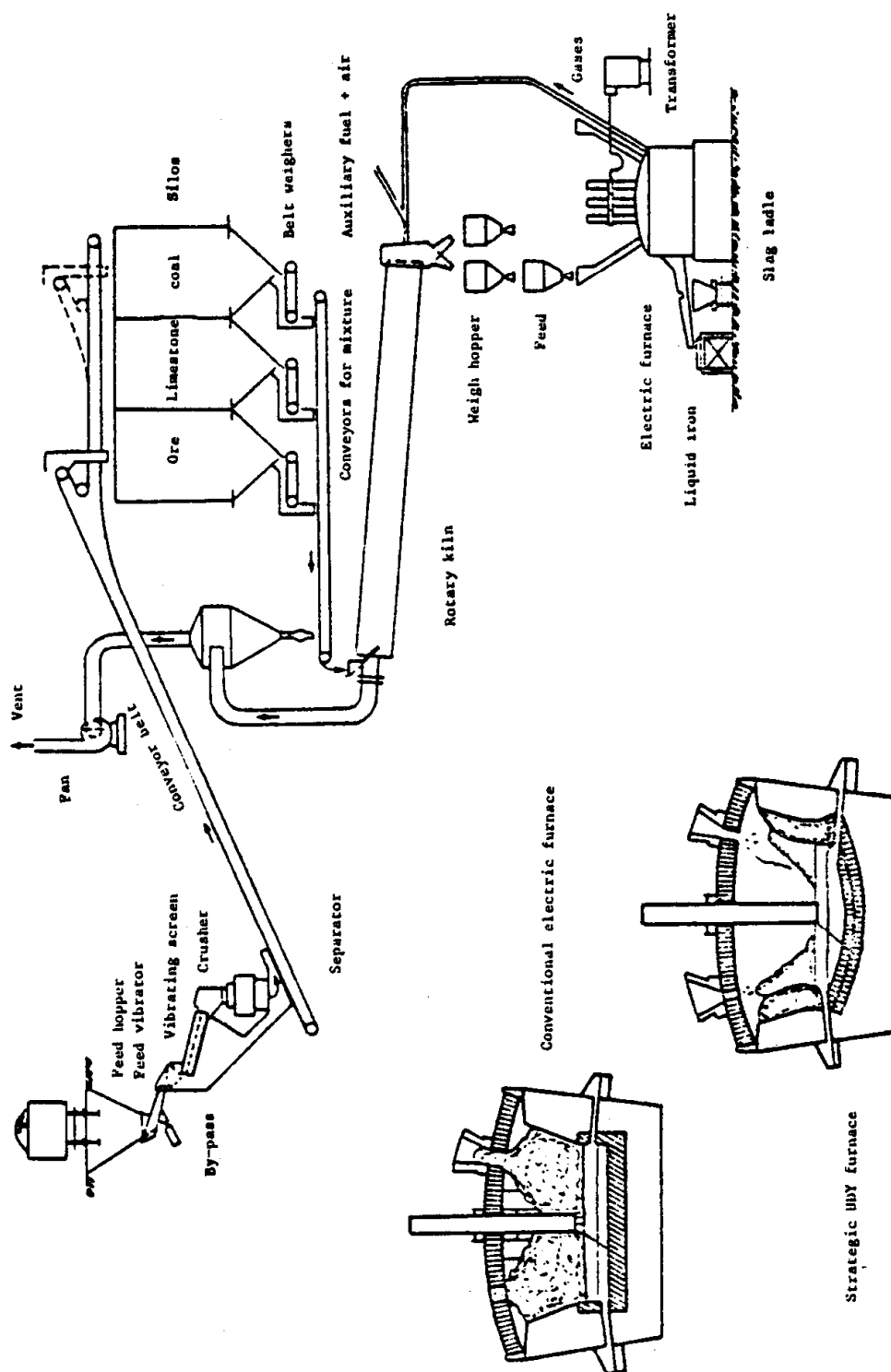


Fig. 1 Strategic-Udy-Process

STÜRZELBERG PROCESS

1. Characteristics

1.1. Name and history of process

Stürzelberg process

1.2. Principle of process

Reduction with the aid of solid fuels (coal) in a horizontally rotating reverberatory furnace, producing molten pig iron.

1.3. Inventors

F Eulenstein and A Krus

1.4. Companies involved in development

Sachtleben AG (Cologne) with collaboration from DEMAG (Duisburg)

2. General description (schematic drawing)

See drawing

3. Plant required

Rotating reverberatory reduction furnace; in addition to rotating about the longitudinal axis the furnace can pivot about a horizontal axis running perpendicular to the main axis of rotation. The furnace is mounted on a turntable so that firing can take place from both sides.

Preheating furnace.

Plant for preheating the air and drying the coal.

4. State of development

No further development

4.2. Pilot plant

None

4.3. Industrial installations

4.3.1. Location

Stürzelberg (Düsseldorf)

4.3.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

The steel shell of the furnace is 13 m long and has an inside diameter of 4.20 m. The lining is 0.35 m thick and consists of a tar impregnated dolomite monolith.

4.3.3. Capacity in tonnes per 24 h of final product - per unit of production

30 to 45 t.p.d. (three tapings) (13)

4.3.4. Method of operation (continuous or batch, number of operating hours per day)

Batch operation

4.3.5. Coefficient of utilisation in time (number of working days per annum)

No details

4.3.6. Maximum rated production

No details

5. Reducing agents

- 5.1. Raw materials on which the reducing agent is based
Coke, anthracite, carbonised lignite, carbon residue of oil in small pieces.
- 5.2. Process for conversion of above raw materials into a form for use in reduction
No details
- 5.3. Condition and analysis of reducing agent at moment of use
No details
- 5.4. Preheat temperature on entering reduction apparatus
None
- 5.5. Gas velocity in reduction apparatus
No details
- 5.6. Analysis and temperature of gas after passing through reduction apparatus
Temperature from 1300 to 1400°C

6. Ores and ferruginous materials

- 6.1. Source
Pyrites residue (13). South American ore with 67% Fe and 4% SiO₂ (14)
- 6.2. Analysis
Pyrites residue containing zinc.
N.B.: Other ores can also be used (e.g. ore fines containing zinc, lead and tin; red mud, titaniferous ores).

6.3. Size

Ore fines, no more details

6.4. Preparation

Dwight-Lloyd sinter heated to 500°C using heat from the off gases.

7. Additions

7.1. Materials

Lime or limestone

7.2. Analysis

No details

7.3. Size

8 to 15 mm

8. Temperature zones in reduction
in °C in the reduction apparatus

No details

9. Pressure zones in reduction
in kg/cm² in the reduction apparatus

Normal atmospheric pressure

10. Product
of the reduction apparatus

10.1. Form

Molten pig iron

10.2. Analysis

Pig iron produced in the normal way has the following composition:

Grade	% C	% Si	% Mn	% P	% S
Special pig iron	4.0-4.8	0.015	0.2-0.4	0.001-0.02	0.01
Special pig iron	4.0-4.8	0.015	0.2-0.5	< 0.09	< 0.04

10.3. Degree of reduction

10.3.1. Proportion of oxygen removed from iron oxides as a percentage of the total oxygen combined with iron in the ores

100 %

10.3.2. Metallic iron as a percentage of the total iron content of the product

100 %

10.4. Utilisation of by-products

No details

11. Further treatment of the product (10) of the reduction apparatus

None

12. Final product obtained after treatment according to 11

See 10

13. Iron yield

No details

14. Utilisation of final product

Destination of final product (blast furnace, electric or other furnace). Technical and economic differences as compared with using pig iron and scrap

The end product is special pig iron

15. Consumption data per tonne of final product (12)

15.1. Raw materials for reduction (5.1)

Reducing agent: approximately 700 kg small lump coke per tonne of pig iron (13)

980 kg (14)

15.2. Fuels and thermal values required for the entire process (fuel characteristics) (thermal balance)

About 600 kg of powdered coal (13). 680 kg coke and powdered coal (14).

16. Overall capital investment

16.1. For the plant described in (3) and related to one tonne of capacity for the final product (12)

Capacity expressed in tonnes per day or per year

For an annual production of 85,000 t Fe (240 t.p.d.) (4 furnaces) the construction costs would run to 3,500,000 dollars (41 dollars per annual tonne) (14). No more recent data available.

17. Opinions expressed on the process in relation to:

17.1. State of development

The plant operated regularly for several years in Stürzelberg, after being installed in 1961. The plant has since been dismantled.

17.2. No details

17.3. Possibilities of exploitation

The high fuel consumption limits the use of plants to areas where large quantities of low-grade fuel are available at sufficiently low prices and for ore fines having a high zinc, lead, tin etc. content.

17.4. Special requirements to be met by certain materials (e.g. refractories)

The ore must have a high Fe content and should not be too hard or subject to decrepitation. The fuel for the burner can be powdered coal, naphtha or gas.

18. Bibliography

18.1. Articles and publications

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- 2) Badin, W., and A Krus: Desulphurisation of pig iron with calcined lime in a Stürzelberg rotary furnace. Stahl u. Eisen 58 (1938) pp. 1457-60.
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18.2. Patents

No details

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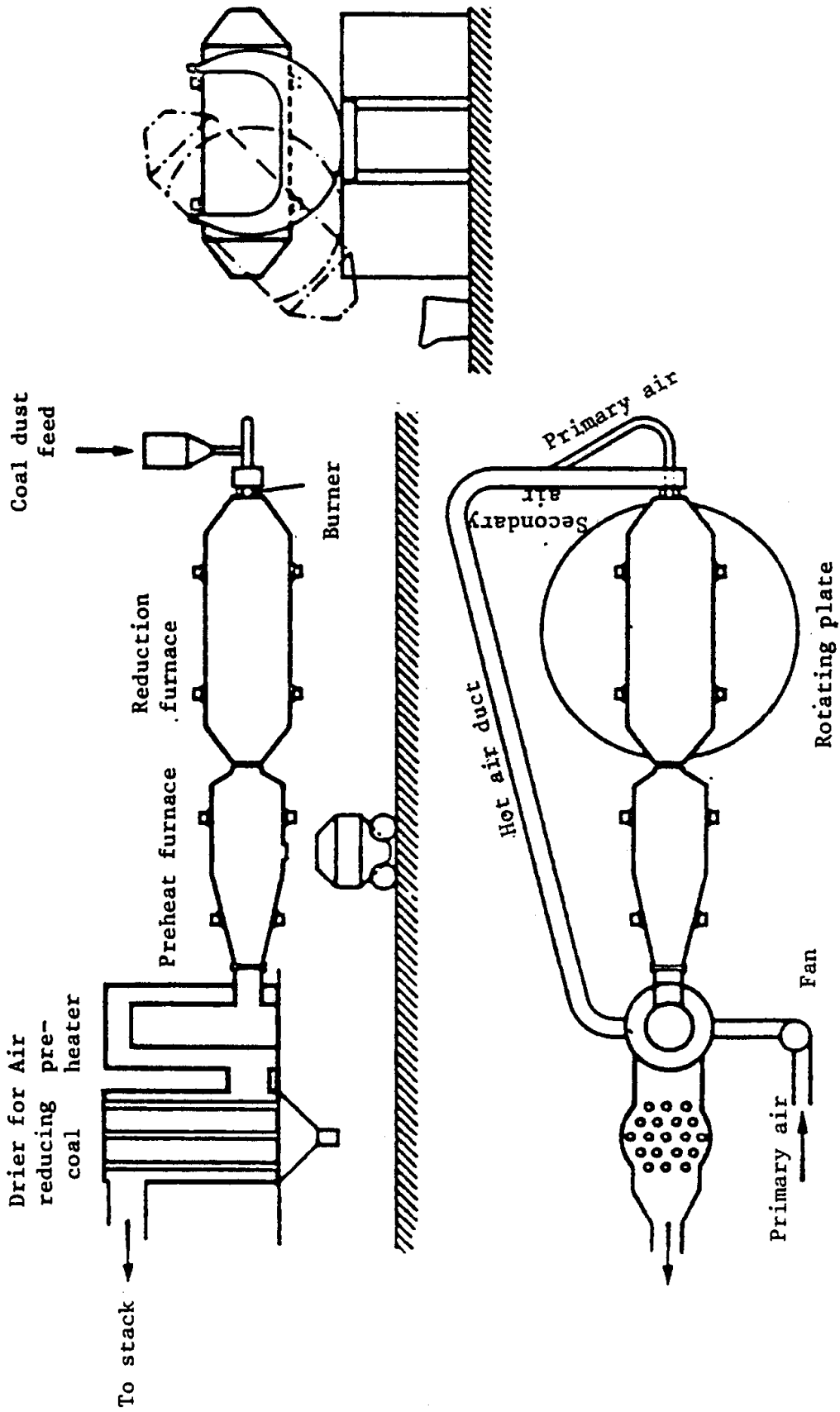


Fig. 1. Stürzelberg Process

UGINE PROCESS

1. Characteristics

1.1. Name and history of process

Ugine direct reduction process.

1.2. Principle of process

a) Reduction in a rotary furnace by a solid reducing agent at low temperature.

b) Where appropriate transfer of the hot charge (600°C) to an electric arc-furnace or Kaldo furnace for steelmaking.

1.3. Inventors

Société d'électro-chimie, d'électro-métallurgie et des aciéries électriques d'UGINE

(Head office: 10, rue du General Foy, Paris 8e).

1.4. Companies involved in development

Inventing company.

2. General description (schematic drawing)

See drawing

3. Plant required

a) Rotary reduction furnace resting on rollers and driven by a motor through a reduction gear unit.

b) "Feed tower" for charging, comprising waste gas offtake and feed hopper.

- c) Means of discharging consisting of a hood with axial burner and a thermally-insulated chamber with double seal and containers also insulated.

4. State of development

4.1. Name of laboratories where the process is situated

Laboratories of the pilot plant

4.2. Pilot plant

4.2.1. Location

Aiguebelle (Savoie) works of the Société d'électro-chimie, d'électro-métallurgie et des aciéries électriques d'UGINE - 73 Aiguebelle (France).

4.2.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

Rotary furnace 35 metres long, external diameter 1.65 m, slope 4%. Lining of alumino-silicate bricks with 40-45% alumina, of varying thickness (see diagram).

4.2.3. Capacity in tonnes per 24 h of final product

15 - 30 t.p.d. in continuous operation

4.3. Industrial installations

The process has not gone beyond the pilot stage

5. Reducing agents

5.1. Raw materials on which the reducing agent is based

Gas coke, charcoal, lignite, sawdust.

- 5.2. Process for conversion of above raw materials into a form for use in reduction

The reducing agent is mixed with the ore to be reduced before being charged into the furnace, in an amount limited to that theoretically necessary for reduction.

- 5.3. Condition and analysis of reducing agent at moment of use

Product crushed to < 3 mm. For gas coke the fixed carbon content is 85 - 90%.

- 5.4. Preheat temperature on entering reduction apparatus

Preheating takes place in the reduction unit.

- 5.5. Gas velocity in reduction apparatus

Not determined

- 5.6. Analysis and temperature of gas after passing through reduction apparatus

Not determined.

6. Ores and ferruginous material

- 6.1. Source

Tests have been carried out on ores from North Africa, Western France and Brazil.

- 6.2. Analysis

Ores from North Africa

1:	Fe _{total}	53 to 54 %
	Fe ₂ O ₃	76 to 77 %
	SiO ₂	2 to 3 %
	Al ₂ O ₃	2 to 3 %
	CaO	7 to 8 %

MnO	< 1 %
MgO	< 0.5 %
Loss at red heat	8 to 9 %

2:	Fe _{total}	47.2 %
	Fe ₂ O ₃	67.5 %
	SiO ₂	3.3 %
	Al ₂ O ₃	0.5 %
	CaO	12.3 %
	MnO	1.9 %
	MgO	1.4 %
	Loss at red heat	13.5 %

Ores from Brazil

Fe _{total}	68.8 %
Fe ₂ O ₃	98.0 %
SiO ₂	1.4 %
Loss at red heat	0.5 %

6.3. Size

Ore fines less than 6 mm

7. Additions

With pelletising, addition, after crushing, of a binder (bentonite or dextrine) and water

8. Temperature zones in reduction
in °C in the reduction apparatus

1000 - 1150°C

9. Pressure zones in reduction
in kg/cm² in the reduction apparatus

Atmospheric pressure

10. Product
of the reduction apparatus

10.1. Form

Sponge iron or pellets

10.2. Analysis

Product obtained with

North African Ore 1	%Fe	74 - 75	:	%C	3 - 4
North African Ore 2	%Fe	63 - 64	:	%C	-
Brazilian Ore	%Fe	81	:	%C	2 - 4

10.3. Degree of reduction

10.3.1. Proportion of oxygen removed from iron oxides as a percentage
of the total oxygen combined with iron in the ores

North African Ore	83 - 85 %
Brazilian Ore	89 %

10.3.2. Metallic iron as a percentage of the total iron content of
the products

North African Ore	74 - 77 %
Brazilian Ore	80 - 85 %

10.4. Utilisation of by-products

The gases are passed to the stack

11. Further treatment of the product (10) of the reduction apparatus

11.1. Method of cooling

The product may be collected in a thermally insulated container to be charged hot to the electric arc furnace. Otherwise it is withdrawn directly, protected from the air, into a water-cooled vessel.

11.2. Crushing, compacting or briquetting

None

11.3. Details of final size grading

No details

11.4. Method of concentration(e.g. magnetic separation or sintering)

None

12. Final product obtained after treatment according to 11

Same as product obtained under 10.

13. Iron yield

13.1. Iron contained in product (10) as a percentage of iron in the charge

93 - 95 %

13.2. Iron contained in final product (12) as a percentage of iron in the charge

Same as 13.1.

14. Utilisation of final product

Destination of final product (blast furnace, electric or other furnace). Technical and economic differences as compared with using pig iron and scrap

Tests have been carried out using the hot product (approximately 600°C) in an arc furnace, and a killed mild steel was obtained with an iron yield of 93 - 95% and a power consumption of 1250 - 1300 kWh/t steel. Added cold to the Kaldo furnace it has proved 1.5 times as efficient a coolant as scrap

15. Consumption data per tonne of final product (12)

For rich ores the consumption of reducing agent is equivalent to about 340 kg carbon per tonne iron. The heat input may be supplied by approximately 200 kg fuel oil.

16. Overall capital investment

No details

17. Opinions expressed on the process:

None

18. Bibliography

18.1. Articles and publications

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18.2. Patents

No details

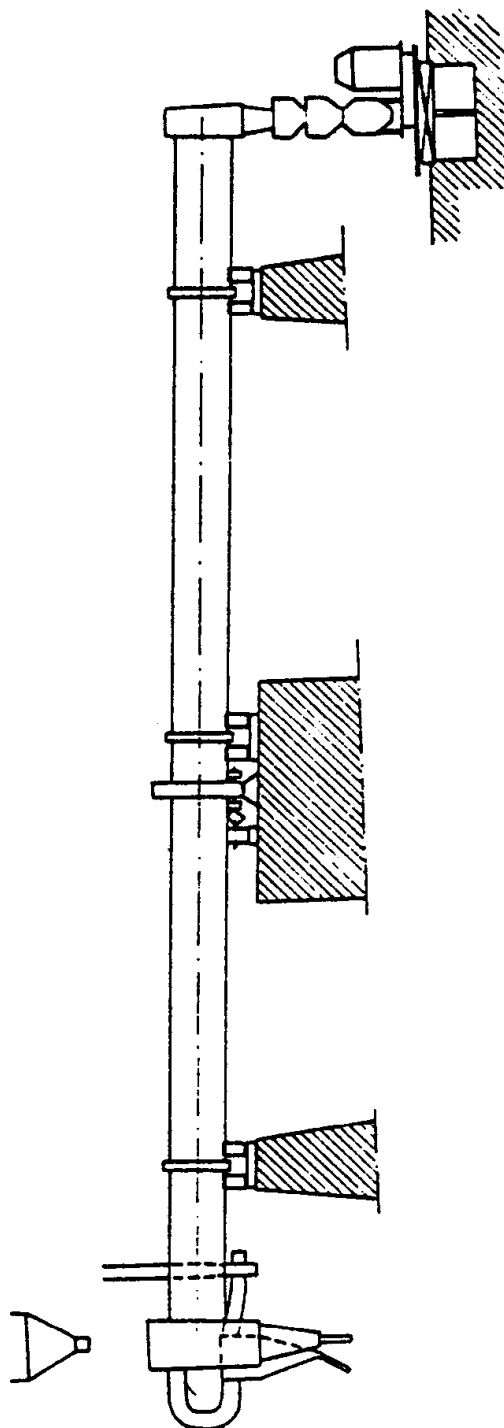


Fig. 1 Uguine Process

VÖEST PROCESS

1. Characteristics

1.1. Name and history of process

Vöest process (Vereinigte Österreichische Eisen-und Stahlwerke).
In 1961 Vöest filed a patent application for its process of direct production of liquid steel.

1.2. Principle of process

The iron ore is reduced by gas in a reduction zone (shaft furnace); the solid, reduced product is then directly converted into liquid steel in a combined zone for melting and production of reducing gas. The product from the reducing zone still contains a certain percentage of FeO which reacts in the melting zone with the carbon in the bath.

The melting furnace has electric heating for starting up and for supplying supplementary heat. An oxy-coal burner is directed on to the bath. By regulating the supply of coal and oxygen on the one hand and electric heating on the other it is possible to maintain a reducing atmosphere. The coal introduced is thus used primarily for producing reducing gas, the melting temperature being regulated by the electric heating. By regulating the coal/oxygen ratio it is also possible to maintain the carbon content of the final steel at the desired level.

1.3. Inventors

1.4. Companies involved in development

Vereinigte Österreichische Eisen-und Stahlwerke,
Vöest, at Linz/Donau, Austria.

2. General description (schematic drawing)

Figure 1 shows the layout of the pilot plant

3. Plant required

- Reduction furnace
- Furnace for melting and for producing reducing gas
- Weigh feeders for coal and ores
- Oxygen supply

4. State of development

4.1. Name of laboratories where the process is situated

Laboratory of Vöest, Linz, Austria

4.2. Pilot plant

4.2.1. Location

Linz/Donau, Austria

4.2.2. Dimensions of apparatus and plant (e.g. height/length and diameter of the shaft of a reduction furnace)

Reduction furnace: useful diameter 350 mm;
 height 12 m;

Melting furnace: useful diameter 1 m;
 height 1.3 m;

Refractory linings:

- reduction furnace: alumino-silicate refractories
- "elbow" between reduction furnace and melting furnace:
 carbon bricks
- melting furnace: magnesite bricks, 300 mm thick

4.2.3. Capacity in tonnes per 24 h of final product

12 t liquid steel per 24 hours. In principle the process is continuous but during the tests the plant operated discontinuously.

4.3. Industrial installations

None

5. Reducing agents

5.1. Raw materials on which the reducing agent is based

Solid fuels, for instance coals or coke.

Drying to 1.5% H₂O

Size : 0-2 mm

Analysis of coal used during tests:

Moisture content : 11.3% at start; 1.5% in use

Ash : 7.10% (dry basis)

Volatiles : 20.78% (dry basis)

Sulphur : 1.00% (dry basis)

Hydrogen : 4.45% (dry basis)

Oxygen and
nitrogen : 5.39% (dry basis)

Total carbon : 82.06% (dry basis)

Net calorific
value : 7.699 kcal/kg (dry basis)

Analysis of ash

Fe₂O₃ : 17.15%

SiO₂ : 39.32%

Al₂O₃ : 26.09%

CaO : 6.33%

MgO : 5.63%

P : 0.22%

S : 1.35%

- 5.2. Process for conversion of above raw materials into a form for use in reduction

The solid fuels are converted into reducing gases in an oxygen burner.

- 5.3. Condition and analysis of reducing agent at moment of use

The composition of the reducing gas alters according to the coal/oxygen ratio: In principle a CO/H₂ ratio of 3:1 and a maximum CO₂ content of 15% are aimed for.

Approximate constituents of gas:

CO ₂	:	15%
CO	:	65%
H ₂	:	20%

- 5.4. Preheat temperature on entering reduction apparatus

No details

- 5.5. Gas velocity in reduction apparatus

No details

- 5.6. Analysis and temperature of gas after passing through reduction apparatus

Off gases : temperature : 500-550°C

Analysis (depending on operation and starting gas); approximately:

CO ₂	:	60%
CO	:	25%
H ₂	:	15%

During the tests the plant and method of operation were continually improved; during the last tests, quoted in 18.1, the rate of utilisation of the CO reached 68.8%, and that of hydrogen 28.8%.

6. Ores and ferruginous materials

6.1. Source

The tests were carried out with various ores: Tuollavaara concentrates, ores from Menara, Conakry and Brazil.

6.2. Analysis

Menara ore: 52.9 % Fe, 7.64 % SiO_2 , 0.85 % Al_2O_3 , 0.10 % CaO, 0.58 % MgO, 0.08 % P

Tuollavaara concentrate: 70.38 % Fe, 1.04 % SiO, 0.18 % Al_2O_3 , 0.10 % CaO, 0.33 % MgO, 0.039 % P, 0.15 % S.

6.3. Size

Optimum size of ores 0-5 mm

6.4. Preparation

No special treatment

7. Additions

In order to neutralise the sulphur introduced by the coal, ground limestone is added (0.5 mm). Thus the sulphur content was reduced to 0.066%.

8. Temperature zones in reduction
in °C in the reduction apparatus

Temperature in the melting unit approximately 1550-1650°C

9. Pressure zones in reduction
in kg/cm² in the reduction apparatus

Atmospheric pressure

10. Product
 of the reduction apparatus

10.1. Form

Liquid steel

10.2. Analysis

The carbon content may be varied by regulating the coal/oxygen ratio. Range of compositions obtained during tests:

C	:	0.01	-	2.42%
Si	:	0.015	-	0.120%
Mn	:	0.020	-	0.103%
P	:	0.003	-	0.064%
S	:	0.066	-	0.360%
Cr	:	0.12	-	0.16% (addition of Conakry ore)
O ₂	:	0.018	-	0.178%
N ₂	:	0.006	-	0.012%

10.3. Degree of reduction

Very high, no precise data; depends on the FeO content of the slag.

For example:

Analysis of steel produced:

C	:	1.20%
Mn	:	0.20%
Si	:	1.00%
P	:	0.20%
S	:	0.25%

Amount and analysis of slag:

Amount : 135.2 kg

Analysis :

FeO : 10.3%

CaO : 46.8%

CaS : 3.3%

Al₂O₃ : 12.0%

MgO : 4.6%

SiO₂ : 21.5%

MnO : 0.5%

P₂O₅ : 1.0%

10.4. Utilisation of by-products

None

11. Further treatment of the product (10) of the reduction apparatus

None

12. Final product obtained after treatment according to 11

See 10

13. Iron yield

The FeO content of the slag depends on the carbon content of the steel.

14. Utilisation of final product

Destination of final product (blast furnace, electric or other furnace). Technical and economic differences as compared with using pig iron and scrap

Special steels

15. Consumption data per tonne of final product (12)

15.1. Raw materials for reduction (5.1)

To produce one tonne of steel from Brazilian ore:

1464.2 kg ore (69% Fe)

15.2. Fuels and thermal values required for the entire process
(fuel characteristics) (thermal balance)

827.9 kg moist coal (5.1)

Thermal balance for conditions under 15.1 (acc. 18.1)

Input

- Combustion of coal by oxygen	
99.5% pure	838,500 kcal
- Electric heating	89,240 kcal
- Chemical reactions:	
$2 \text{ CaO} + \text{SiO}_2 = 2 \text{ CaO} \cdot \text{SiO}_2$	15,511 kcal
$2/5 \text{ P}_2 + \text{O}_2 + 6/5 \text{ CaO} = 2/5 (3 \text{ CaO} \cdot \text{P}_2\text{O}_5)$	2,477 kcal
$\text{Fe} + \frac{1}{2} \text{O}_2 = \text{FeO}$	2,645 kcal
	948,373 kcal

Output

Reduction of Fe_2O_3 with a mixture of $\text{CO} + \text{H}_2$	21,350 kcal
Reduction of Fe_2O_3 with C	157,994 kcal
Reduction of SiO_2	5,864 kcal
Reduction of MnO	250 kcal
Reduction of P_2O_5	763 kcal
Desulphurisation: $\text{FeS} + \text{CaO} + \text{C} = \text{CaS} + \text{Fe} + \text{CO}$:	1,992 kcal
Sensible heat of steel	340,000 kcal
Sensible heat of slag	58,950 kcal
Enthalpy of exit gases	227,366 kcal
Cooling losses	133,844 kcal
	948,373 kcal

- 15.3. Electrical energy
109 kWh per tonne steel
- 15.4. Steam used
No details
- 15.5. Oxygen
Oxygen, 99.5%: 726 m³/t steel:
of which: 659 m³ for combustion of coal as under 15.1
and: 67 m³ for pre-heating the furnace
(expressed in KW = 508 kWh/t steel)
- 15.6. Cooling water
No details
- 15.7. Refractories
Not given, but probably high consumption (FeO in slag)
- 15.8. Man hours of productive labour
No details
- 15.9. Labour and materials for routine maintenance
No details
16. Overall capital investment
No information (laboratory installation)
17. Opinions expressed on the process
According to private information from the promoters, the process described here has been abandoned.

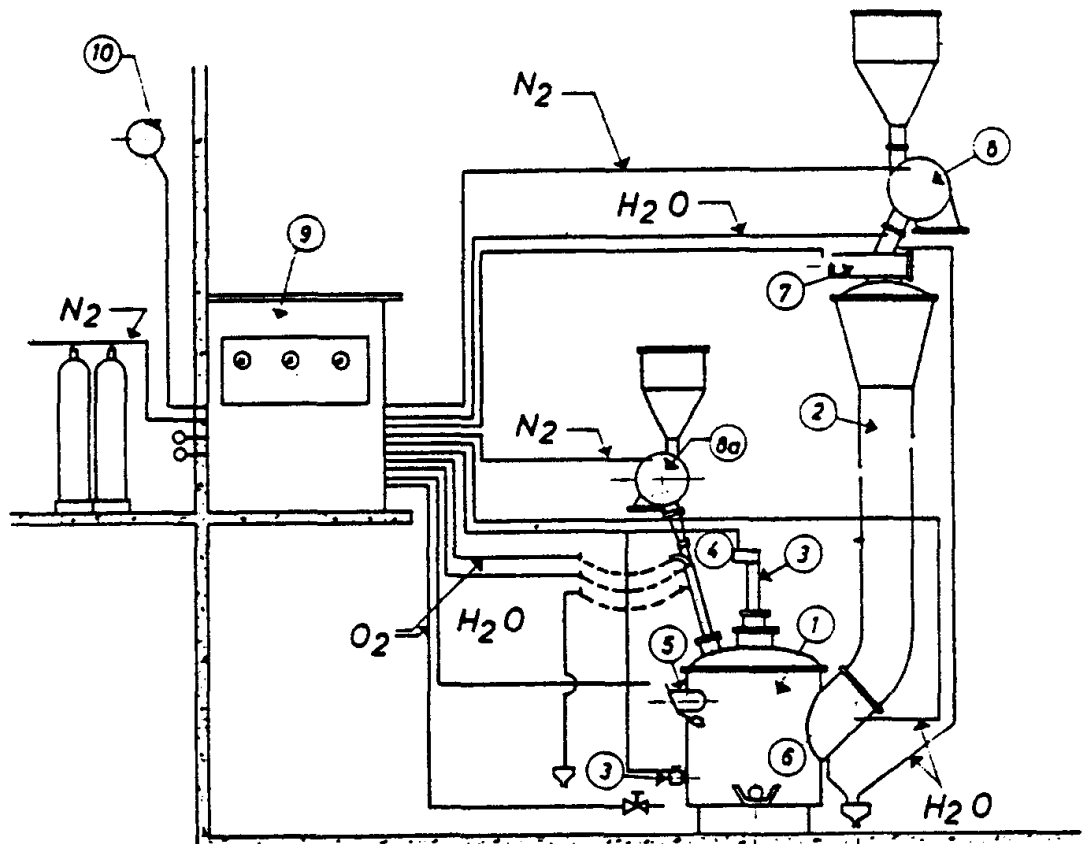
18. Bibliography

18.1. Articles and publication

- 1) R.F. Rinesch et A. Krausz, "Versuchsergebnisse der VBest von einem neuen Verfahren der direkten Herstellung von flüssigem Stahl" (Test results from VBest on a new method of direct production of molten steel). Presented at Eisenhüttentag 1966, Leoben, 10-13 June, 1966.
- 2) R.F. Rinesch et A. Krausz, Berg-und Huttenm. Monatshefte, Nov. 1966, Heft 11, pp. 563-578.

18.2. Patents

British patent No. 1.007.376



- (1) Melting furnace
- (2) Reduction furnace
- (3) Electrode
- (4) O₂ Lance
- (5) Explosion door
- (6) Melt
- (7) Off gases
- (8) Ore feed hopper
- (8a) Coal feed hopper
- (9) Control
- (10) O₂ Inlet

Fig. 1. Voest Process

Images have been losslessly embedded. Information about the original file can be found in PDF attachments. Some stats (more in the PDF attachments):

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