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UNITED NATIONS INDUSTRIAL DEVELOPMENT ORGANIZATION

Development and Transfer of Technology Series

No. **11**

**TECHNOLOGICAL
PROFILES
ON THE
IRON AND STEEL
INDUSTRY**

123



UNITED NATIONS

TECHNOLOGICAL PROFILES ON THE IRON AND STEEL INDUSTRY

UNITED NATIONS INDUSTRIAL DEVELOPMENT ORGANIZATION
Vienna

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UNITED NATIONS
New York, 1978

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Preface

The four technological profiles on the iron and steel industry contained in this publication were prepared for the Industrial and Technological Information Bank (INTIB) of the United Nations Industrial Development Organization (UNIDO), which is a component of the UNIDO programme on the development of technology. INTIB is a pilot operation that began in July 1977 for a period of 18 months. Its work is being concentrated on four industrial sectors: iron and steel, fertilizers, agro-industries, and agricultural machinery and implements. Each of these sectors has priority in other UNIDO endeavours also: sectoral studies, consultations, negotiations and technical assistance projects.

The concept of INTIB has its roots in the Lima Declaration and Plan of Action, adopted at the Second General Conference of UNIDO in 1975, and in various United Nations General Assembly resolutions, all envisaging such a service as an essential instrument for the transfer, development and adaptation of appropriate technologies. To increase the share of developing countries in world industrial output from 7 per cent (in 1975) to 25 per cent by 2000, an objective set by the Lima Conference, decision makers must have adequate information on new investments. Those advising the decision makers national industrial information centres, technology development institutes and investment banks—also must have this information.

The novel character of INTIB, as compared with the services previously rendered by UNIDO, is that it is addressed to the selection of technology before the technology is acquired. INTIB not only draws upon the services available in the Industrial Information Section, where it is housed, but also relies on the expertise of specialists in the Industrial Operations Division of UNIDO and outside experts to process the information obtained from sources within and outside UNIDO relevant to technology selection. As a result of this effort, UNIDO is able to supply information in anticipation of, as well as in response to, demand. An example is the series of technology profiles and monographs being prepared, to which this volume belongs, concerning matters to consider when selecting a technology from a variety of alternatives.

The "customers" INTIB is designed to serve include ministries of industry, planning and industrial development institutes, multipurpose technological institutions, and agencies concerned with the transfer of technology. The intention is to serve all those who are responsible for selecting technology, whether in an advisory role or decision-making capacity, in each of the four priority industrial sectors selected for the pilot phase.

Further information about INTIB and its related activities can be had on request by writing to the Chief, Industrial Information Section, UNIDO, P.O. Box 707, A-1011 Vienna, Austria.

These technological profiles were prepared by G. P. Mathur, acting as a consultant to UNIDO; the views expressed are those of the consultant and do not necessarily reflect the views of the secretariat of UNIDO.

EXPLANATORY NOTES

References to dollars (\$) are to United States dollars.

The term "billion" signifies a thousand million.

References to "tons" are to metric tons, unless otherwise specified.

Parentheses around a figure indicate a minus amount (in tables only).

Besides the common abbreviations, symbols and terms, the following have been used in this report:

BF	Blast-furnace
BOF	Basic oxygen furnace
DR	Direct reduction
EF	Electric furnace
HTGR	High-temperature gas-cooled reactor
kVA	Kilovolt-ampere
LD	Linz-Donawitz
LWS	Creusot-Loire and Wendel-Sideler with Sprunck and Co.
MVA	Megavolt-ampere
Nm ³	Normal cubic metres
OBM	Oxygen Bottom Maxhütte
OH	Open hearth
Q-BOP	"Quick, quiet, quality" basic oxygen process
SIP	Submerged injection process

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I. World iron-ore survey

Iron-ore deposits

A large portion of common ores and rock-forming minerals contain appreciable amounts of iron, but there are only six minerals containing sufficient amounts of iron and available in sufficient quantities to serve as potential sources from which iron may be economically obtained. The six iron-bearing minerals and the iron content of each pure mineral are given below.

<i>Iron-bearing mineral</i>	<i>Iron content of pure mineral (percentage of Fe)</i>
Hematite	69.9
Magnetite	72.4
Goethite	62.9
Chamosite	42
Siderite	48.2
Pyrite	46.6

The wide variety of conditions under which iron is concentrated in the earth, the physical and chemical nature of these concentrations, their mineralogical and geological environment, and the complex process which resulted in the concentration of iron in ore deposits, account for the peculiar characteristics of each deposit.

Iron-ore deposits occur extensively throughout the world, as shown by the following brief survey of the major producing areas.

Africa

Deposits in Africa are similar to those of Bilbao, Lake Superior, the Minette Basin and Taberg, and mostly contain hematite-goethite, hematite-magnetite, hematite-pyrite (ochre) and siderite-goethite.

Asia

In south-west Asia, iron-ore deposits are found in Israel, Saudi Arabia and Turkey. Deposits in Israel are hematite and goethite; those in Saudi Arabia are for the most part similar to Lake Superior deposits, and generally contain hematite, sometimes with magnetite mineralization; and those in Turkey are mostly magnetite.

The deposits of middle south Asia occur in Afghanistan, India, Iran, Nepal, Pakistan and Sri Lanka. Deposits in Afghanistan contain hematite and

siderite. Indian iron ores are similar to those of Lake Superior and Taberg, the predominant iron-bearing mineral being hematite and sometimes goethite and magnetite. Magnetite and magnetitic deposits occur in Iran, while hematite and deposits of bedded type are found in Nepal. Pakistan has magnetitic and bedded-type deposits, with magnetite and hematite as the principal iron-bearing minerals. The deposits of Sri Lanka are of residual lateritic type, containing mostly goethite and sometimes magnetite.

In east Asia, deposits occur in China, the Democratic People's Republic of Korea, Japan and the Republic of Korea. Magnetite-hematite ores are the principal iron-bearing minerals in the Chinese deposits, which are similar to those of Lake Superior and the Minette Basin. The deposits of Japan are of residual bog and bedded iron-sand types containing goethite, magnetite and ti-magnetite. The Democratic People's Republic of Korea and the Republic of Korea have mainly magnetitic deposits containing magnetite and occasionally hematite.

Deposits of magnetitic, residual lateritic and bedded iron-sand types are generally found in eastern south Asia, including Burma, Democratic Kampuchea, Indonesia, the Lao People's Democratic Republic, Malaysia, the Philippines, Thailand and Viet Nam.

Europe

In eastern Europe, Bulgaria has ores similar to those of Bilbao, containing goethite, hematite, magnetite and siderite. Ores in Czechoslovakia and the German Democratic Republic resemble Minette Basin deposits, and Poland and Romania have ferruginous carbonates containing siderite-magnetite-goethite.

In northern Europe, Norway has deposits like those found at Lake Superior and Taberg, containing magnetite-hematite minerals. Swedish ores resemble those of Kiruna and Lake Superior, with magnetite and magnetite-hematite minerals. United Kingdom ores are related to those occurring in the Minette Basin and contain chamosite-goethite-hematite.

In southern Europe, the ores of Italy, like those of Yugoslavia, mostly resemble the ore deposits of Bilbao and the Minette Basin and include magnetitic types, with magnetite, siderite and siderite-chamosite minerals. In Greece, lateritic deposits are predominant and goethite is the principal economic mineral. Ores in Portugal are related to Minette Basin

deposits, consisting of hematite, magnetite and occasionally siderite and chamosite. In Spain deposits similar to those of Bilbao are predominant, with hematite-goethite as iron minerals.

In western Europe, Austria has ores like those of Bilbao, containing mostly siderite, while those of the Federal Republic of Germany belong to the class of deposits found in the Minette Basin, consisting mainly of hematite-chamosite-siderite, with occurrences of goethite in some areas. French ores also belong to the class of Minette Basin deposits and contain siderite-goethite.

Latin America

Iron-ore deposits in Argentina are similar to ores found at Lake Superior and in the Minette Basin, and contain hematite and magnetite. Those of Bolivia and Brazil mostly resemble Lake Superior ores and contain hematite. Ores similar to those found at Kiruna also occur. Deposits in Chile are related to those of Kiruna and include magnetitic ores, with magnetite and hematite as the iron-bearing minerals. Deposits in Colombia contain goethite and resemble those occurring in the Minette Basin. Cuba and the Dominican Republic have ores similar to those of Lake Superior and Taberg, consisting mostly of hematite, magnetite and goethite. Deposits in Mexico and Middle America, consisting of magnetite, hematite and goethite minerals, may be generally classified with those found at Kiruna. Those of Peru contain mostly magnetitic ores, and Venezuela has hematite deposits similar to those of Lake Superior.

Northern America

The deposits of Canada are located in Grenville, Labrador, south-west and northern Canada. They are basically similar to those occurring at Lake Superior

and Taberg, and mostly contain hematite, magnetite and goethite. Siderite, pyrites and chamosites are also sometimes found associated. The important deposits in the United States occur in the Cuyuna, Fillmore, Gogebic, Lake Superior, Mesabi and Vermilion regions. They usually resemble those found at Lake Superior, but ores related to those of Clinton, Kiruna, Magnitnaya and Taberg also occasionally occur. The principal minerals are hematite, magnetite and siderite.

Oceania

Deposits in Australia are similar to ores found at Algoma, Clinton and Lake Superior, with goethite, hematite, hematite-magnetite-goethite, magnetite and magnetite-pyrite minerals. New Zealand deposits are of alluvial and sedimentary nature and contain goethite and magnetite as the main iron minerals.

Union of Soviet Socialist Republics

In the Union of Soviet Socialist Republics, the biggest deposits, which resemble those of Lake Superior, are found in the Ukrainian Soviet Socialist Republic (Krivoy Rog and Kursk magnetic anomaly). The eastern slopes of the Urals contain ores similar to those occurring at Taberg, while Turgay and western Siberia have magnetic deposits and ores related to those of the Minette Basin. Large deposits are found in the Caucasus, Kazakhstan and Siberia.

Production and reserves

Production of iron ore in the different producing countries is given in table 1, and world distribution of reserves in table 2.

The above description of world iron-ore resources gives an indication about the varieties of

TABLE 1. WORLD IRON-ORE PRODUCTION

Country or area	Production level (millions of tons)			
	1973	1974	1975	1976 ^a
Algeria	3.130	3.792	3.300	3.200
Angola	6.048	4.980	3.360	3.300
Australia	83.568	96.688	97.365	92.400
Austria	4.211	4.246	3.833	3.784
Belgium	0.116	0.123	0.093	0.063
Brazil	55.019	79.973	88.493	70.000
Bulgaria	2.774	2.684	2.337	2.300
Canada	48.200	47.271	44.745	56.000
Chile	9.650	10.297	11.070	10.500
China	50.000	51.000	51.000	50.000
Colombia	0.442	0.500	0.623	0.600
Czechoslovakia	1.672	1.688	1.773	1.850
Democratic People's Republic of Korea	8.100	8.100	8.200	6.100
Denmark	0.012	0.006		
Egypt	3.130	3.792	3.300	3.200

Country or area	Production level (millions of tons)			
	1973	1974	1975	1976 ^a
Finland	0.885	0.934	0.766	0.700
France	54.754	54.730	50.142	45.543
German Democratic Republic	0.520	0.250	0.590	0.500
Germany, Federal Republic of	6.429	5.670	4.273	3.034
Greece	1.842	2.001	1.965	2.154
Guinea				
Hong Kong	0.151	0.160	0.161	0.037
Hungary	0.681	0.595	0.386	0.631
India	34.426	34.230	40.271	41.400
Iran	0.600	0.620	0.650	0.650
Italy	0.675	0.795	0.739	0.643
Japan	1.007	0.780	0.942	0.800
Liberia	34.620	36.000	36.500	35.000
Luxembourg	3.782	2.686	2.315	2.079
Malaysia	0.516	0.468	0.349	0.300
Mauritania	10.416	11.110	8.500	8.000
Mexico	5.736	4.902	4.621	3.500
Morocco	0.376	0.534	0.554	0.350
Netherlands				
Norway	3.970	3.918	4.064	4.291
Peru	8.964	9.563	7.753	7.000
Philippines	2.256	1.616	1.352	1.150
Poland	1.413	1.296	1.192	1.100
Portugal	0.057	0.024	0.045	0.043
Republic of Korea	0.467	0.493	0.525	0.500
Romania	3.234	3.205	3.065	2.300
Sierra Leone	2.400	2.508	2.500	2.400
South Africa	10.955	11.734	11.191	15.684
Southern Rhodesia	0.550	0.550	0.600	0.600
Spain	6.901	8.613	8.617	7.700
Sudan				
Swaziland	2.148	2.055	2.232	1.932
Sweden	34.727	36.153	30.867	30.526
Switzerland				
Thailand	0.036	0.036	0.032	0.020
Tunisia	0.811	0.820	0.652	0.500
Turkey	1.861	1.531	1.990	1.000
United Kingdom	7.105	3.602	4.490	4.583
United States of America	88.800	85.917	81.351	81.200
USSR	216.104	224.883	232.803	239.000
Venezuela	22.880	26.408	24.104	23.000
Yugoslavia	4.670	5.034	5.239	4.265
Total	853.797	901.564	897.880	877.412

^aEstimated or provisional.

iron ores found. Each of these deposits has its own characteristic features, variations in iron content, mineralogical assemblage, particle size of iron minerals and those of associated economic and gangue minerals etc.

Throughout the world, higher grades of ores are becoming gradually depleted owing to selective mining of one type or another. During mining of high-grade ores, the admixture of low-grade ores, which may be present as overburden and capping or *in situ* along with good-grade ore; takes place. This admixture becomes inevitable where large-scale mechanized mining is resorted to.

Thus, in most cases, some kind of beneficiation of the run-of-mine ore has to be adopted to ensure an

accepted and consistent quality of iron ore of desired chemistry for iron smelting. The prepared burden for iron smelting is of paramount importance, necessitating size reduction, screening into size gradings and improving the chemical composition of the ore by employing beneficiation techniques.

Beneficiation

Different beneficiation techniques are employed to suit a particular ore, depending upon its mineralogical and petrological characteristics. The techniques include crushing, grading, sizing, washing and wet screening, gravity treatment, magnetic

TABLE 2. WORLD DISTRIBUTION OF IRON-ORE MINERAL RESOURCES
(C: chamosite; F: hematite; G: goethite; I: ilmenite; M: magnetite; P: pyrite; S: siderite)

Country or area	Total resources and potential reserves ^a (millions of tons)															Unclas- sified	Others	
	F	M	G	S	F-M	FG	CS	F-S	MP	MS	FMP	MSP	MG	SP	F-MG			FMS
Africa	17 056 (2 405)	2 718 (373)	1 297 (262)	6 (6)	4 254 (603)	1 358 (127)	92 (20)	168 (168)		55 (15)	41 (41)		46 (46)		1 419 (1 250)	1 427 (432)	42 (42)	P-17; FPI-11; FMI-279 (28); FMC-1 000 (1 000)
Asia (including India)	19 427 (8 310)	7 751 (2 673)	3 206 (655)	41 (4)	12 824 (4 206)	566 (317)	2 545 (508)	129 (29)					4 (2)		275 (108)		24 328 (88)	G:SO-4450
India	15 516 (6 982)	2 332 (620)			8 547 (419)	52 (52)	2 540 (508)								53 (53)		12 (12)	
Australia and New Zealand	(1 630)	(660)	(5 951)		(35)	(7 953)		(322)	(144)		(2)				(120)			P-38; FI-1 350 (350); FMI-320; FSC-2 000; FMC-25; GSC-1 500
Canada and the West Indies	12 235 (2 025)	32 411 (8 164)	(1)		55 875 (20 898)	8 370 (4 320)		600	20 (10)	1 000	1 675 (40)	735 (420)						
Europe (including Sweden)	777 (457)	3 122 (3 103)	1 264 (794)	1 900 (1 680)	2 498 (1 278)	839 (139)	9 460 (4 840)	775 (675)	50 (10)	50 (50)					440 (100)	2 240 (2 120)	1 110 (10)	MI-50 (50); GC-5 148 (3 048); SC-10 (2); FSC-50 (30); MSC-680 (230); GSC-1 853 (1 048); FGSC-1 740 (1 640); MGS-38 (38)
Sweden		2 895			475													
South America	84 869 (32 262)	788 (251)	1 948 (301)		2 610 (411)	2 304 (929)							8					
United States of America	3 561 (396)	12 969 (789)	907 (745)		65 555 (3 227)	6 139 (2 267)	343 (343)	50					313 (113)		1 742 (272)	10 254 (4)	3 (3)	FGS-30 (30); FMGS-4 228 (28); others-2 (2)
USSR	29 025 (20 875)	38 325 (21 648)	106 363 (13 894)	1 258 (1 195)	36 255 (29 421)		522 (447)	9 316 (9 316)							300	82 937 (13 687)		

^a Figures without parentheses indicate total resources and those with parentheses show potential reserves.

separation, froth flotation, reduction roasting, thickening and drying. The overall beneficiation flow-sheet may include the use of one or more of the different methods. The criteria for determining and finalizing the flow-sheet depend on the cost economics of the process, the required quality of the end-product and the possibilities of recycling waste products; these are primarily governed by the mineralogical characteristics of the ore under study.

Techniques

Crushing

The mined ore is generally 300-400 mm in size. The minimum and maximum ore sizes required for use in blast furnaces are respectively 10 mm and 30-50 mm. Crushing is done by jaw or gyratory crushers. For some types of ore, such as the Indian iron ores, wet screening of crushed ore has to be carried out because of the sticky nature of the material and the presence of clayey matter mixed with the mined ore. The screen undersize fraction of less than 100 mm is dewatered and slime rejected in a spiral or rake classifier.

Crinding

In some cases, such as with magnetite ores and taconites, the ore is ground, either wet or dry, in ball or rod mills, with a view to liberating iron-bearing minerals from gangue minerals.

Washing

Lateritic ores and the ores admixed with aluminous clayey matter are scrubbed with water in log washers, cylindrical or conical types of washers fitted with lifters, for loosening the adhering fines. The scrubbed ore is then wet-screened on a double-deck wet vibrating screen to separate clean lumpy ore from adhered fines for direct use in the blast-furnace and from free-flowing fines for use in the sinter plant.

Gravity methods

Heavy-media separation An aqueous suspension of ferro-silicon or magnetite, finely ground, is used to separate hematite, goethite or siderite from lighter gangue minerals. The size of ore treated is normally less than 30 mm and greater than 4 mm. However, finer sizes can be treated in heavy-media cyclones.

Jigging Jigs of the Harz or Renier type are used for ore in the size range of 0.5-25 mm.

Humphrey's spiral The size range of feed to spiral is generally 0.1-1.5 mm. Sometimes specular hematite

of 65 per cent fineness, passing 150 micrometres, has been successfully treated on spirals.

Shaking tables The size of feed is almost the same as that used for spirals. Tables are generally employed for recleaning of fine-gravity rougher concentrates.

Cyclones Cyclones are used for recovering heavy minerals from fine gangue particles contained in slimes.

Magnetic separation

Strongly magnetic minerals such as magnetite are separated from non-magnetic minerals employing a low-intensity wet magnetic separator. The separation is often preceded by desliming the feed for better efficiency. Low-intensity dry magnetic separation is used for pre-concentration of strongly magnetic minerals and for treatment of beach sands for recovering ilmenite and other magnetic minerals. High-intensity magnetic separation is used for feebly magnetic minerals such as limonite, specularite, goethite etc. and can be wet or dry. In case dry separation is employed, the ground ore should be almost free from adhering gangue minerals like clays.

Froth flotation

Flotation is employed for fine-grained low-grade non-magnetic ores such as siderite-hematite ores, and specular hematitic ores. The pH of the flotation pulp could be weakly acidic or alkaline depending upon the minerals to be floated and the reagents used. Tall oil, alkyl sulphonates, sodium fluosilicic acid, ligneous tar, fish fats etc. are the common flotation reagents used.

Electrostatic/high-tension separation

Electrostatic or high-tension separation is used for further upgrading fine gravity concentrates, and helps in the removal of undesirable minerals such as apatite, micas, hypersthene etc. from iron-bearing minerals.

Low-temperature magnetizing roasting

Low-temperature magnetizing roasting is employed for fine-grained, non-magnetic or feebly magnetic low-grade ores containing hydrated oxides and sometimes siderite. The roasted ore is then passed through magnetic separators to separate magnetic from non-magnetic gangue minerals.

Dewatering and drying

Fine concentrates are thickened in thickeners, filtered and dried for use. Drying could be partial, depending upon the end use of fine concentrate.

Practice in various countries

Ores that contain 60-65 per cent iron are generally considered of good quality and acceptable for direct use in the blast-furnace for smelting. The presence of the total gangue minerals consisting of oxides of silicon, aluminium and titanium up to a level of about 8 per cent is acceptable. The phosphorus and sulphur contents of the ore should be as low as possible.

A brief description of beneficiation techniques for the various types of iron ore found in different countries is given below. It may be mentioned that the exact process parameters will depend upon the amenability of ore to upgrading, the nature and characteristics of the constituent minerals, the sizes at which different minerals are liberated from each other, the end use of the beneficiated product etc. The description given is, therefore, merely indicative of the broad process techniques in each case.

Canada

Specular hematite These are low-grade friable ores occurring in southern parts of the Labrador-Quebec area. Generally, after autogenous grinding, the ground ore is treated on spirals. If, however, super-high-grade concentrate is needed, then magnetic separation and flotation are sometimes employed.

For specular-hematite-magnetite quartzites of the Lake Carroll and Lake Wabush regions, Humphrey's spiral treatment is adopted to produce concentrates analysing 60-66 per cent Fe.

Hematite-siderite These ores, which occur in Algoma, Wabana and Steep Rock, are subjected to washing, gravity treatment such as heavy-media separation (cyclones/drums) and jigging.

Magnetite Ores from Marmora, Ontario, Moose Mountain etc. are concentrated by low-intensity magnetic separation.

India

Indian iron ores, though generally of high iron content, are characterized by their high alumina content and the presence of clayey matter. This makes the ore sticky, particularly in rainy seasons, with the result that the ore crushing and handling plants come to a standstill during periods of wet weather. All the crushers, bins and bunkers, conveyors and chutes become choked, making screens completely blocked.

The treatment for these types of ores (hematites) consists in scrubbing with water to loosen the clay and then wet-screening with powerful jets of water. The screen undersize containing almost all the water and slimy matter is treated in a classifier. The classifier overflow carries away the slime which is

generally a waste product. This is sent to the water reclamation system. The slimes, in case they contain a higher percentage of iron values, are treated in cyclones. Cyclone underflow, after thickening and filtering, is sent to the agglomeration plant. The classifier sand fraction is then a free-flowing material and can be used for agglomeration directly or after beneficiation by gravity methods. The washed lumps are clean and free from adhered fines. Nearly 30-40 per cent of the total silica in the ore is thus eliminated as slime along with about 20-30 per cent of alumina.

Magnetite-hematite ores These ores are found in the Kudremukh and Ongole areas in southern parts of the country. Magnetic separation after grinding yields a high-grade concentrate analysing over 60 per cent Fe. The non-magnetic tailings containing hematite are treated in Humphrey's spirals for its recovery.

A typical flow-sheet for hematitic ores is given in figure I, and that for a magnetite-hematite ore in figure II.

Sweden

Magnetite ores These types of ores, found in Kiruna, Malmberget and Grängesberg are upgraded by repeated magnetic separations. If hematite is also present, then the non-magnetic tailings are treated in jigs or shaking tables for its recovery. The concentrates analyse over 60 per cent Fe, are generally fine, and require agglomeration.

Hematite ore The ore after coarse crushing is subjected to the Stripa process or heavy-media separation using ferro-silicon as the separation medium. The finer fractions of ore are treated in shaking tables and Humphrey's spiral.

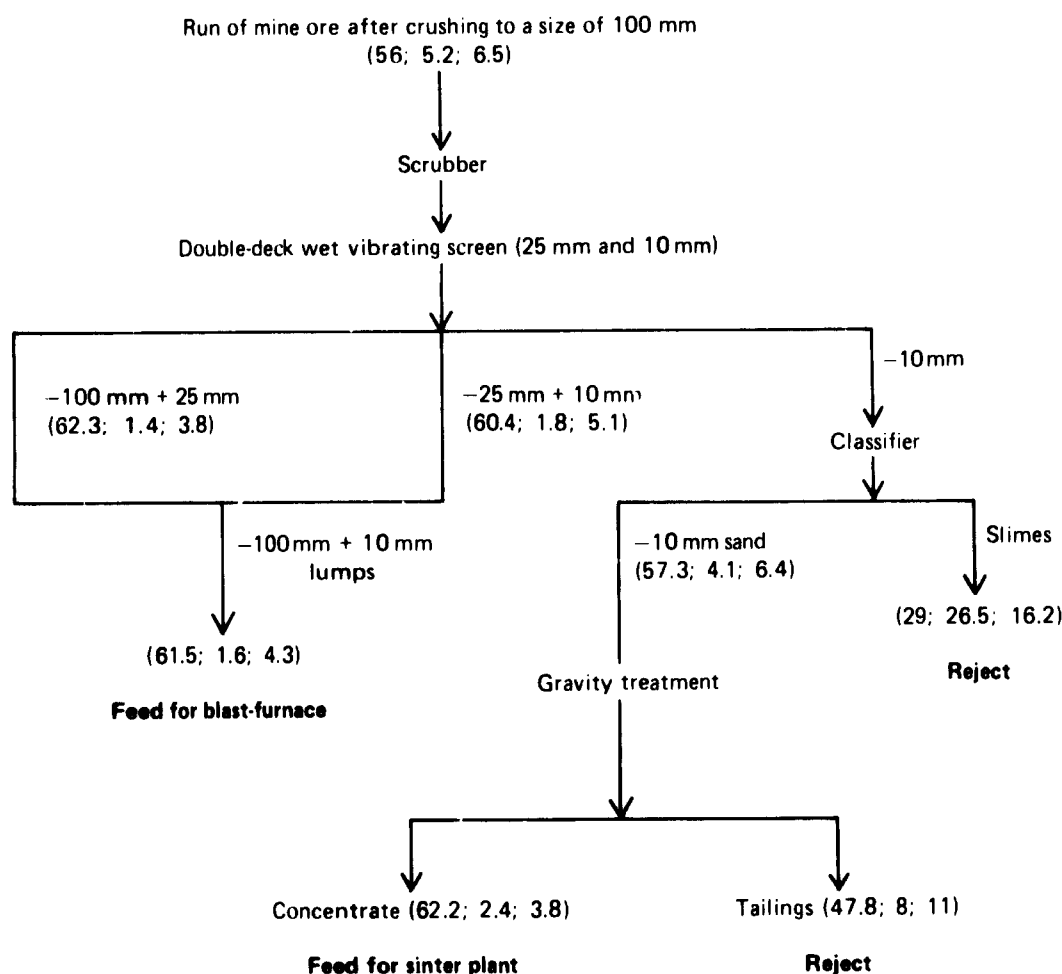
Sometimes flotation is adopted to recover associated economic minerals like apatite. Emulsified tall oil is the reagent used at a pH of 8.5 to recover apatite. Hematite is floated after lowering the pH to about 6. The raw ore analysing 35 per cent Fe and 0.02 per cent P, is upgraded to 65 per cent Fe and 0.01 per cent P. The apatite float analyses 0.3 per cent P.

Ores such as those found at Skarn are first subjected to dry magnetic separation at about 20 mm in size, followed by ball milling and flotation of pyrite. The flotation tailing after high-intensity wet magnetic separation yields magnetite concentrate separately.

Union of Soviet Socialist Republics

Magnetite ores are mainly exploited, as substantial reserves of these ores have been accumulated and they are easier to beneficiate. However, purely magnetic separation treatment becomes economical if

Figure 1. Typical flow-sheet for a hematite ore from India



Note: A minus (-) or plus (+) sign indicates ore sizes of, respectively, less than or greater than the specified dimension. Percentages of Fe, SiO₂ and Al₂O₃ in the ore fractions are given within parentheses (Fe; SiO₂; Al₂O₃).

the proportion of magnetite in the ore exceeds 70-80 per cent and the loss of iron in magnetic tailings does not exceed 12-14 per cent.

The beneficiation plants at Olenegorsk and Krivoy Rog, employing a combination of magnetic separation and gravity methods such as spiral treatment, heavy-media separation and jigging, treat 20 million tons/year. For flotation, the ore is subsequently ground to a degree of fineness allowing approximately 90 per cent of the material to pass through a 200-mesh screen.

United States of America

Brown iron ore After crushing to the required size, the ore is scrubbed and wet-screened to obtain clean-sized lumpy ore and free-flowing fines for use in the sinter plant or for pelletization.

Oxidized ores Generally, after washing, the washed lumps and fines are subjected to gravity methods of beneficiation, namely heavy-media separation, jigging, Humphrey's spiral treatment and hydrosizing. Flotation is sometimes used to recover iron values from fine-grained tailings from the heavy-media circuit.

Taconites The ore, after crushing, is ground in stages using rod and ball mills in closed circuit. After rod-milling, the pulp is passed through a wet magnetic separator to recover magnetic iron oxide liberated in primary grinding. The classifier overflow from the ball mill circuit is deslimed in cyclones and the sand fraction is subjected to anionic flotation to remove siliceous gangue minerals.

Specularite After grinding in stages in an open-circuit rod mill and a closed-circuit secondary ball mill, followed by desliming, the underflow is

ever-increasing and exacting demands of iron smelters, necessarily produce large proportions of fines, sometimes up to 50 per cent by weight of the ore mined, in addition to the fines obtained *in situ*. In the case of magnetite ores, the entire quantity of concentrate is in the form of fines which are utilized for ironmaking after sintering or pelletizing.

Sintering

A sintering plant in an iron and steel works may be likened to a scavenger, making useful agglomerate sinter from a wide variety of wastes such as coke breeze, mill scale, flue dust, blue dust, limestone and dolomite fines. The process has great flexibility in the agglomeration of raw materials with different physical properties and mineralogical compositions.

Earlier batch-sintering machines in sintering plants have been replaced by continuous machines of different makes. Modern continuous sinter plants have large strand areas of 400-500 m², capable of producing 4-5 million tons of sinter per year. The introduction of grate-cooling systems for cooling hot sinter has helped to raise productivity, to reduce solid fuel consumption and maintenance costs, and to produce good-quality, highly-reoxidized fine-grained sinter.

Pelletizing

Pelletizing is resorted to where the ore particles are in very fine form either as a beneficiated product or a naturally-occurring mineral like blue dust. The process consists of two principal steps, namely balling and induration. After grinding the ore, which could be either wet or dry, in open or closed circuit with the mill, dewatering and partial drying, green pellets of desired size are made with the addition of a suitable binder. Pellets can be made in drum, disc or cone types of pelletizer. The balling drum requires separate screening facilities to recirculate undersized pellets back into the balling circuit, whereas for pelletizing discs and cones, separate screening is not normally required, as sizing is done during the balling operation and only pellets of the desired size are discharged. Binders commonly used are bentonite, limestone and hydrated lime.

For making pellets of good quality having adequate green strength and yielding suitably strong heat-hardened pellets of good reducibility, the type of grind, the size to which the ore should be ground and the specific surface area of the ground material, the schedule of drying and pre-heating, and the firing and cooling cycles are important parameters which should be carefully controlled.

Induration of pellets by heat is achieved in vertical shaft furnaces, travelling horizontal grates, grate-kiln combinations and circular-grate pelletizing systems. In all of these, the induration process involves drying of the green pellets, pre-heating to

induration temperature, firing at the required temperature, and soaking for a definite period to create iron oxide or a slag bond formation between the grains, followed by regulated cooling of the product. Strict control of drying and heating cycles is important to maintain product quality and avoid such problems as spalling, premature pellet breakage, and cluster formation.

Cold-bonded pellets A development in recent years has been the introduction of cold induration processes. Cobo, Grancold and several other processes have been developed. Special types of cements (which do not contain sulphur) are used with the pelletizing feed before balling. The green balls, sometimes coated with iron concentrate fines to prevent cluster formation, are allowed to cure and harden for periods of up to five weeks. Eight to ten per cent of cement is normally required.

In case bonding is achieved by the addition of lime, the green pellets after partial drying are allowed to harden at about 120°-150°C in a carbon dioxide atmosphere under pressure.

Cost data

It has been estimated that capital costs for installation of a pelletizing plant of 5 million tons/year in Iran would be \$100 million, and for a plant of 6 million tons/year in Japan, \$111 million.

Arthur D. Little Inc. estimates average pellet costs in the United States to be \$16.6-\$19.3 per short ton of pellets of grade 63.8-64.8 per cent Fe. Average ore costs were computed at \$0.2034-\$0.2688 per short ton unit of contained Fe for an average ore grade of 55.4-65.8 per cent Fe.

Sinter has been costed by discounting lump ore costs by \$0.02 per short ton unit of contained Fe. Capital investment costs for a sinter plant of 100 tons/day have been estimated at \$100,000, and for a 1 000 tons/day plant the cost will be approximately \$550,000.

Pellet production and heat-hardening

World pellet production in various regions using different indurating systems is given in table 3.

The share of the developing countries in 1975 was about 22-23 million tons produced by 14 plants in 10 countries. World production in 1985 is projected to be 435 million tons, approximately 165 million tons of which would be accounted for by the developing countries.

Comparison of sinter and pellets

It is well known that pellets are usually produced near the source of iron ore and sinter is made at the smelting plant. Sinter production permits a much

TABLE 3. WORLD PELLET PRODUCTION IN 1975

Induration system	Production level (millions of tons)						Total
	Australia and New Zealand		Europe		Latin America	Northern America	
	Africa		Eastern	Western			
Shaft furnace	0.85	3.00	-	1.25	1.50	17.30	23.90
Travelling grate	2.00	6.70	21.10	7.43	10.60	37.65	85.48
Grate kiln	2.00	9.85	-	7.50	-	42.25	61.60
Lepol furnace	-	0.30	-	0.45	-	-	0.75
Circular grate	-	-	-	-	0.75	-	0.75
Grancold process	-	-	-	1.60	-	-	1.60
Total	4.85	19.85	21.10	18.23	12.85	97.20	174.08

Source: United Nations Economic and Social Council, Economic Commission for Europe, "Structural changes in the iron and steel industry" (STEEL/GE.3/R.3/Add.1), p. 9.

larger part of the added values to be credited to the country where the ore is processed into iron and steel than does pellet manufacture. This extra benefit usually favours the iron and steel manufacturer, who therefore has a good reason to prefer the use of sinter over pellets.

Technically, a good-quality, self-fluxed or super-fluxed sinter of adequate strength and close sizing possesses metallurgical, physical and chemical properties which are the same as, or sometimes even superior to, those of oxidized pellets. The choice of sinter or pellets as a burden material in similar circumstances, is usually a matter of personal preference, within limits over which the operator has little control. The choice is also dictated by the local conditions and characteristics of the ore.

Pellets are largely preferred in North America, given the huge deposits of taconite available for exploitation. The concentrates produced are finely-divided, excellent feed material for pelletizing.

Japan imports huge quantities of high-grade lumpy ore and fines. The fines are sintered and used in blast furnaces. However, pelletizing plants are being established at works using imported fines and concentrates, probably to avoid pollution hazards, which are subject to stringent pollution control regulations.

Physical form of iron-ore consumption

The production ratios of sinter-pellets and pig-iron listed in table 4 show the enormous efforts being made to improve the burden preparation.

Table 5 shows the changing pattern of iron-ore requirements. It may be seen that the proportion of sinter feed in the iron ore has generally remained unaltered, while the share lost by lumpy ore is gained by pellets.

Figure III shows the growth pattern of consumption of lumpy ore, sinter and pellets in different

TABLE 4. RELATION BETWEEN PRODUCTION OF SINTER/PELLETS AND PIG-IRON

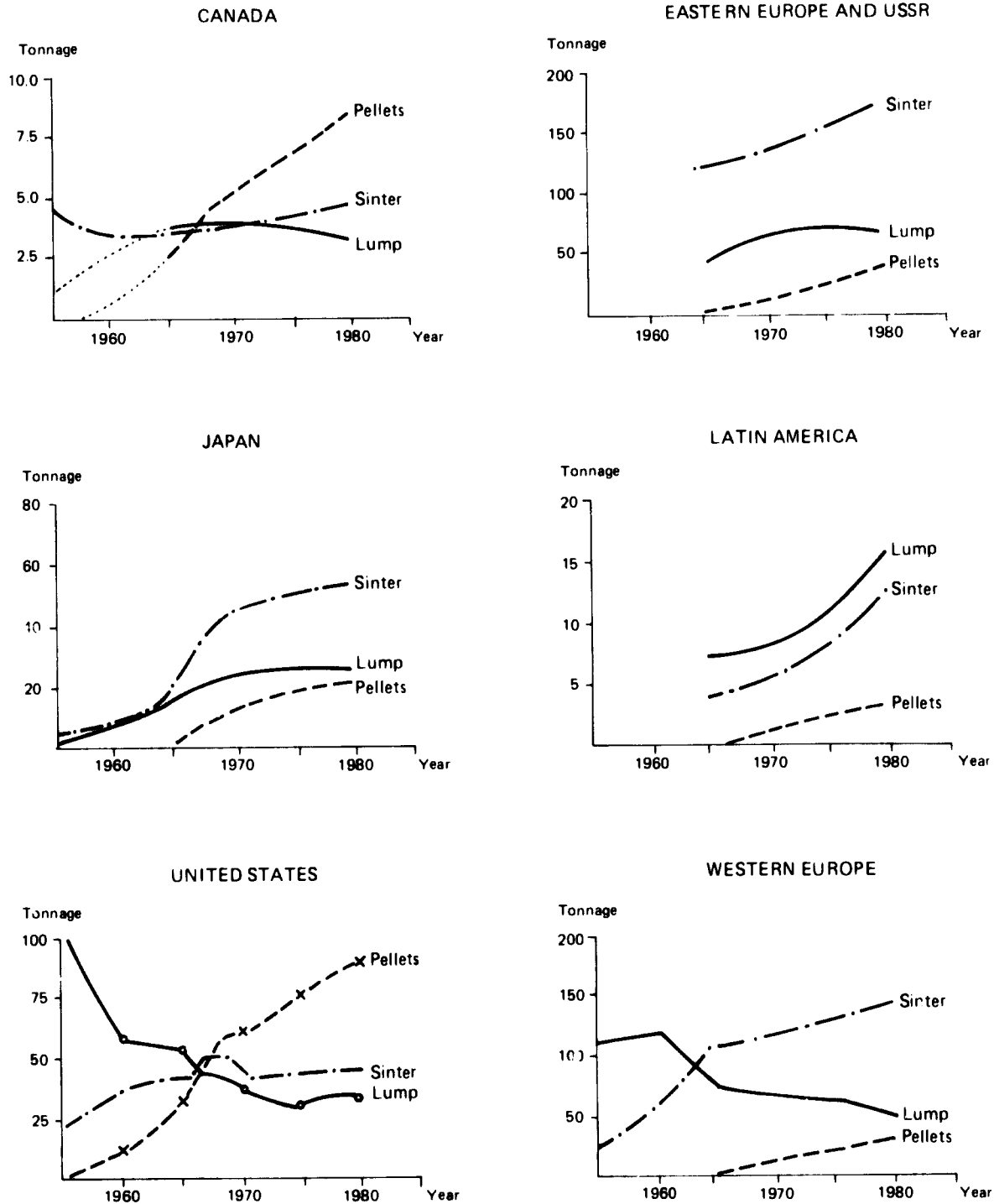
Country	Production ratios (sinter/pellets to pig-iron)	
	1960	1975
Austria	1.127	1.450
Belgium	0.337	1.130
Canada	0.774	1.380
France	0.448	1.699
Germany, Federal Republic of	0.779	1.202
Hungary	-	1.996
Italy	0.788	1.270
Japan	0.670	1.309
Luxembourg	-	2.207
Netherlands	0.733	1.604
Poland	1.292	1.281
United Kingdom	0.398	1.139
United States of America	0.777	1.246
USSR	-	1.732

TABLE 5. CHANGING PATTERN OF IRON-ORE REQUIREMENTS

Type of feed	Iron-ore requirements (percentage of total)			
	1965	1970	1980	1985
Lumpy ore	39.4	33.4	22.0	20.0
Sinter feed	53.0	50.3	48.0	48.0
Pellet	7.6	16.3	30.0	32.0
Total	100.0	100.0	100.0	100.0

regions of the world. It may be seen that sinter consumption predominates in Eastern and Western Europe and in Japan. In the United States and Canada, the pattern was similar until the early 1960s, when pellets became the preferred iron burden material. In Latin America, lumpy ores are still the principal burden material, but sinter consumption is also steadily rising.

Figure III. Projection of iron-ore consumption by physical form
(Millions of tons)



Source: Jack R. Miller, "On-site processing of iron ore in developing countries through the stage of prereducd agglomeration" (ID/WG.146/67), p. 17.

II. Ironmaking

Pig-iron is the intermediate form through which almost all iron must pass in the manufacture of steel. In addition to this, it is used in foundries for the manufacture of a wide variety of iron castings.

Most iron ores may be used to produce pig-iron. In current ironmaking practice, the term "iron ore" is applied to any iron-bearing material that can economically be used at a particular time and place for the manufacture of pig-iron.

All the constituents of the ore that are undesired in ironmaking are impurities. However, the mineralogy of iron ores often does not lend itself to the ready removal of many such impurities by known ore-treatment methods. It is therefore necessary to determine the nature and amount of impurities and accordingly to control the composition of the iron in the smelting-furnace. The wide differences in structure and mineral content of ores from different deposits are responsible for the considerable variation in the beneficiation methods that have been developed to remove or to limit the impurities. However, some impurities, notably silica, alumina and lime, play important positive roles in scavenging other impurities from molten iron. Nevertheless, the ironmaker would prefer to use high-grade ores containing a minimum of these scavengers and to add them only as needed, in controlled amounts, or to blend ores containing known amounts of different impurities.

Influence of burden constituents

Iron ores

Iron compounds

The iron content of ores that are used in blast-furnaces varies widely from about 30 per cent to 71 per cent. Hematites are easier to reduce in iron smelting than magnetites, in spite of the greater amount of oxygen combined with the iron in hematites. Ores with a high content of iron silicate minerals have a low degree of oxidation and are difficult to reduce in the blast-furnace. Goethites and carbonate ores contain combined water and carbon dioxide which are removed in the upper part of the blast-furnace.

In order to obtain low fuel consumption in the blast-furnace, the ore should have a high iron content so that a smaller amount of slag is formed.

Silica

Silica is one of the most important gangue minerals in an iron ore. Together with alumina, it is a main constituent of the acid slag produced during the smelting operation. The amount of silica permissible in the ore is determined by the proper slag volume, which in turn is determined primarily by the sulphur in the charge and, secondarily, by the necessity of having a slag fluid enough to recover the molten iron.

A decrease of about 1.5 per cent in the silica content of the ore will produce a drop in slag volume of about 65 kg per ton of pig-iron. It has been estimated that an increase of 100 kg in the amount of slag per ton of pig-iron raises fuel consumption by about 40 kg of coke per ton of pig-iron.

Alumina

The alumina content of the slag of a coke blast-furnace should not be too low. About 10-15 per cent of alumina increases the fluidity of basic blast-furnace slags and thus makes it possible to use a higher basicity, which facilitates the removal of sulphur.

If the ore is high in alumina, its content in slag may be as high as 25-30 per cent. Such a slag requires a high temperature in the furnace to ensure the right fluidity and produce high-silicon pig-iron for foundry and Bessemer processing as in Indian practice.

If slags contain 40-45 per cent alumina, they can be used in the cement or aluminium industries. In case the alumina content of the slag is low, desulphurization is not quite effective, although this could be offset to some extent by raising the magnesia content.

Lime

Lime is a dominant constituent of a basic slag. Its function is to form a fluid slag with coke ash, ore gangue and other burden impurities.

Magnesia

Magnesia, the other dominant constituent of slags, helps in reducing the viscosity of slag due to high alumina content. Dolomite is generally used in blast-furnaces along with limestone.

Manganese oxides

Most manganese, if present in the ore, passes into the pig-iron, and subsequently a portion of this finds its way into the final steel.

Phosphorus

Almost all phosphorus present in the burden will pass directly through the blast-furnace and enter the pig-iron. A high phosphorus content is a drawback, as extra lime is required for its elimination; thus slag volume and fuel consumption are increased and steel output decreased accordingly.

Sulphur

Sulphur is contributed not only by iron ore but also by limestone and the coke used in the burden. Excepting a small percentage of sulphur which goes out as gas, it is divided between the slag and metal. Satisfactory removal of sulphur requires a basic slag low in iron, a high temperature and a large quantity of slag. The extent of sulphur removal, therefore, depends upon the temperature of the hearth and the ratio of basic oxides of lime and magnesia to the acid oxides of silica and alumina. High silicon in the pig-iron is favourable, and a high manganese content is also an advantage.

Titanium

In blast-furnace smelting, most of the titanium oxide remains unreduced in the slag. It is a strongly carbide-forming element, and the titanium carbide has a low solubility in molten pig-iron. High titanium content gives rise to blast-furnace operating difficulties. However, in electric smelting, the operating difficulties are not many and so higher amounts of titanium can be tolerated.

Vanadium

About 70-90 per cent of vanadium present in iron ore passes into the pig-iron. If the metal is not high in silicon or titanium, a large quantity of the contained vanadium can be oxidized quickly when refined by the Bessemer process. A vanadium-rich slag is poured off for the production of ferro-vanadium, an important metallurgical agent.

Zinc

If zinc content increases beyond 0.2 per cent, operating difficulties begin to appear.

Copper

The entire amount of copper present in the burden will pass into pig-iron and ultimately into steel. Small percentages of copper in steel increases its corrosion resistance. But if its content increases beyond 0.3 to 0.4 per cent, rolling difficulties are encountered.

Chromium

The chromium content of pig-iron is an advantage for alloy steels. For ordinary steels, its presence is mainly a disadvantage.

Nickel

The entire amount of nickel will go into pig-iron from which it cannot be removed by oxidation during the steelmaking process. In some special cases, small amounts of nickel in steel may be of advantage in improving mechanical properties. But in most instances, its presence is undesirable.

Arsenic

An excess of arsenic causes cold brittleness in steel produced from pig-iron containing arsenic. However, up to 0.15-0.25 per cent, is acceptable in ordinary steels, and up to 0.05-0.10 per cent in steels for temper hardening.

Lead

Lead is rare in iron ores. It does not enter pig-iron but damages refractory lining by penetrating it.

Tin

The entire amount of tin present in the ore goes into pig-iron and thence into steel. Even in relatively low amounts, it is harmful in steels because it damages the deep drawing properties and also causes brittleness. The steels should not contain more than 0.05 per cent of tin.

Coke

Coke for blast-furnace consumption must be sufficiently firm and strong to resist shattering by handling, and crushing by pressure from the heavy blast-furnace burden. It should be free of dust and fines, and in pieces not too large for optimum speed of combustion.

With a good coking coal, these physical properties can be controlled only moderately by the coking process. As the coal is heated, it becomes plastic at 350°-475°C, forming a fused mass irrespective of its form when charged into the retort. As bituminous coal is heated through this range of temperatures, volatile matter is given off, rapidly at first, then slowly up to about 950°C. The coals making up a blend, so far as possible, should have about the same plastic range. Slow heating through the plastic range slightly increases the hardness of the coke. The size of the lumps of coke depends largely upon the thickness of the coal charge and whether or not it is heated from one or both sides.

As to the chemical composition, a good metallurgical coke will contain very little volatile matter not over 2 per cent and 85-90 per cent fixed carbon. The remainder is ash, sulphur and phosphorus. The phosphorus content, 0.018-0.040 per cent for making Bessemer iron, should also be low for basic iron. Sulphur content varies from 0.6-1.5 per cent, but is desired as low as possible because coke is the chief source of sulphur in the pig-iron produced. Standard specifications for foundry coke call for a volatile matter content of 2 per cent, maximum sulphur of 1 per cent, maximum moisture of 3 per cent, and a minimum fixed carbon content of 86 per cent.

Shattuck and tumbler tests are also specified, but no standard for combustibility has been adopted. These requirements are controlled through selection of the coal, which should be low in sulphur, free from slate or removable refuse, and give an ash which has a moderately high fusion point in a reducing atmosphere.

There are three principal kinds of coke, classified according to the methods by which they are manufactured, namely low-, medium- and high-temperature coke. All the coke used for metallurgical purposes must be processed in the high temperature ranges if the product is to have satisfactory physical properties.

The most desirable blast-furnace coke is made from mixtures of high- and low-volatile coals, pulverized and blended, and coked in ovens capable of heating the mass to a uniformly high temperature.

Manufacture of metallurgical coke

There are basically two methods for manufacturing metallurgical coke, namely the beehive process and the by-product or retort process. In the beehive process, air is admitted to the coking chamber in controlled amounts for the purpose of burning the volatile products distilled from the coal to generate heat for further distillation. In the modern by-product method, air is excluded from the coking chambers and the necessary heat for distillation is supplied from external combustion of

some of the gas recovered from the coking process. The temperature of coking is somewhat lower than in the beehive ovens. Besides metallurgical coke, coke breeze, coke-oven gas, tar, ammonium sulphate, ammonia liquor and light oil are the principal by-products. Refining of tar and light oil yields a large variety of products such as benzene, naphthalene, pyrene, phenol, pyridine etc.

In order to reduce environmental pollution, extensive studies have been made in the USSR on dry quenching of hot coke instead of wet quenching.

Table 6 contains an estimate of world reserves of coking coals.

TABLE 6. WORLD COKING-COAL RESERVES

Country or area	Coking-coal reserves (billions of tons)	
	Hard	Soft
Africa	12	-
China	101	-
Europe	41	38
India	11	1
Japan	1	-
Latin America	-	1
Oceania	14	11
United States of America	128	60
USSR	166	107
Others	12	1
Total	486	219

About 90 per cent of total world reserves of coking coal exists in relatively few places throughout the world, mainly in developed countries. Developing countries account for only 2.3-2.5 per cent of total world reserves.

Estimates of world coking-coal requirements are given in table 7. The forecast figures take into account the more economical use of coke resulting from improved smelting technologies, such as oil and pulverized-coal injection, oxygen enrichment of the blast, use of pre-reduced ore etc.

Coke production figures for some producing countries in 1974 are given in table 8.

Formed coke

Attempts to develop new processes for the manufacture of artificial solid fuel suitable for metallurgical use have been mainly inspired by the desire to use otherwise unsuitable coal-based materials and by a world-wide shortage of good coking coal. Experiments have been made for the development of a new coking process aimed at obtaining a solid product similar to metallurgical coke by means of blends partially or entirely made up of non-coking coals and called "formed coke". Although many processes have been developed throughout the

TABLE 7. WORLD COKING-COAL REQUIREMENTS

Area	Estimated coking-coal requirements (millions of tons)											
	1975				1980				1985			
	No PR		With PR		No PR		With PR		No PR		With PR	
	BF	Total	BF	Total	BF	Total	BF	Total	BF	Total	BF	Total
Africa and Middle East	5.6	6.6	5.4	6.5	7.4	8.7	6.9	8.1	9.4	10.9	7.2	8.4
Asia	75.0	94.3	67.6	85.4	79.8	91.5	69.9	80.3	87.5	108.9	66.2	96.4
Europe												
Eastern	94.6	132.8	85.8	119.8	108.8	117.1	92.4	123.6	117.1	145.8	89.5	112.0
Western	89.4	129.5	83.5	121.0	107.0	147.0	91.0	125.4	111.0	144.5	87.3	113.6
Latin America	16.3	17.8	15.2	16.5	24.0	26.2	20.2	21.9	24.3	26.8	19.3	21.2
Northern America	75.7	85.1	67.0	75.4	76.9	86.4	70.1	76.9	78.8	88.2	66.3	74.7
Oceania	10.3	12.0	9.8	11.5	15.6	18.3	12.7	14.9	17.4	20.2	12.7	14.4
Total	366.9	478.1	334.3	436.1	419.5	495.2	363.2	451.1	445.5	545.3	348.5	440.7

Note: PR: pre-reduction; BF: blast-furnace.

Source: Jack R. Miller, "On-site processing of iron ore in developing countries through the stage of prereduced agglomeration", paper prepared for the Third Interregional Symposium on Iron and Steel Industry, Brasilia, 14-21 October 1973 (ID/WG.146/67), p. 32.

TABLE 8. COKE PRODUCTION OF SELECTED COUNTRIES

Country	Coke production in 1974 (millions of tons)
Argentina	0.660
Australia	4.916
Austria	1.733
Belgium	8.050
Brazil	1.850
Bulgaria	1.308
Canada	5.233
Chile	0.315
China	28.000
Colombia	0.510
Czechoslovakia	10.898
Democratic People's Republic of Korea	2.300
Egypt	0.360
Finland	0.080
France	12.282
German Democratic Republic	1.829
Germany, Federal Republic of	34.854
Greece	0.372
Hungary	0.766
India	8.199
Iran	0.065
Italy	8.566
Japan	45.632
Mexico	2.071
Netherlands	2.687
New Zealand	-
Norway	0.315
Peru	0.012
Poland	16.929
Portugal	0.196
Romania	1.525
South Africa	3.600
Southern Rhodesia	0.255
Spain	4.243
Sweden	0.481
Turkey	1.241

Country	Coke production in 1974 (millions of tons)
United Kingdom	15.776
United States of America	60.487
USSR	82.641
Yugoslavia	1.323
Total	372.560

world, most installations operate on the basis of two main processes: degassification and transformation of the coal into semi-coke to be briquetted with the aid of a binder; and hot briquetting of coal followed by distillation in special furnaces.

The main advantages that can be achieved with this new technology are the following: enlargement of the quality range of the coals to be used; better uniformity in coke size; extensive automation in the operating plants; better control of pollution problems; and continuity in the coking process from coal preparation up to the final product.

It is well known that one of the main conditions for lowering the coke rate and increasing productivity of the blast-furnace is satisfactory permeability of the burden. Formed coke, with its well-balanced briquettes, is certainly more suitable than conventional coke for this purpose.

Charcoal

Wherever conditions are favourable, charcoal is used in smaller-size blast-furnaces in place of coke. Since charcoal is virtually ash-free compared with

coke, one of the features of the operation is the extremely low slag volume produced. It is unnecessary to use high-grade limestone because silica must be added to the burden to make up minimum slag volume. Low-grade, high-silica limestones may be used for this purpose.

Slag volume varies between 150 and 200 kg/ton, and all slag is tapped with the iron. Slag basicity is usually within the 0.9-1.0 range, but sulphur can be kept at 0.02 per cent maximum despite low volume. The charcoal produced is screened to a size of at least +4 mm before being charged to the furnace. The screen undersize after pulverization is injected through the tuyères of the furnace.

Charcoal is highly reactive, but it is not strong and cannot withstand the abrasion of the charge in a blast-furnace of the usual height. It can therefore be satisfactorily used in a shorter and smaller furnace of capacity up to, for example, 400-500 tons/day. With a well-prepared burden of sinter or pellets, a coke rate of 750 kg per ton of iron should be possible. However, such a furnace would need stoves capable of heating the blast up to 1100°C.

Charcoal for iron smelting in relatively small blast-furnaces is being used in several developing countries which have good forest wealth and a forestation programme. In Brazil, about 3 million tons of pig-iron are smelted in small blast-furnaces using charcoal as the reductant and for heat input. In western Australia, an iron-smelting blast-furnace using charcoal has been in operation for more than two decades. In India, an iron and steelworks at Bhadravati has had an operating charcoal blast-furnace for several years. In Malaysia, iron smelting has been successfully in operation for several years using charcoal made from rubber wood.

Fluxes

Limestone and dolomite are used as fluxes. The functions of these fluxes are twofold: to form a fluid slag with the coke ash, ore gangue, and any other charged impurities; and to form a slag of such chemical composition that it will provide a degree of control of the sulphur content of the pig-iron.

Selection of the proper flux for a given process is chiefly a chemical problem requiring a knowledge of the composition and properties of all materials entering the process.

Almost all of the slag-forming compounds that enter into a smelting or refining process may be classed as either acids or bases by virtue of the fact that they will react with each other to form compounds which are similar to the salts formed in reactions taking place in water solutions. Since one of the functions of a flux is to react chemically with unwanted impurities to form a fusible slag, it will naturally follow that to remove basic impurities, an acid flux will be required, and to remove acid components, a base will be used as the flux.

Acid fluxes

Silica (SiO_2) is the only substance that is used as a strictly acid flux. For this purpose it is available as sand, gravel and quartz, and also as siliceous iron-bearing minerals.

Basic fluxes

The chief natural basic fluxes are limestone and dolomite (CaCO_3 and $\text{CaMg}(\text{CO}_3)_2$ respectively). Either dolomite or limestone may be used as a blast-furnace flux, the proportions of each depending on the other constituents of the slag and the amount of sulphur that the slag removes.

Alumina

Although alumina is seldom employed as a flux, it is present in a large number of raw materials as an impurity and is therefore present in slag. In slags it may function as an acid or as a base, depending on the conditions. In highly siliceous slags it may form aluminium silicates, while in the presence of an excess of a strong base such as lime, it may form calcium aluminates.

Fluorspar

For making slags more fusible, a neutral substance like fluorspar (CaF_2) may be added.

Manufacture of pig-iron

Pig-iron is the term applied generally to the metallic product of the blast-furnace when it contains over 90 per cent of iron. This term is used to distinguish it from blast-furnace products such as ferro-phosphorus and other ferro-alloys, including ferro-manganese and spiegeleisen, which are made from manganese ore or a mixture of manganese and iron ores.

Pig-iron can be made in the following ways: in blast-furnaces using coke; in blast-furnaces using charcoal; in electric smelting-furnaces; and in cupolas by melting steel scrap with an excess of carbon. However, most pig-iron is made in blast-furnaces using coke, and only small quantities are made by the other methods.

Blast-furnace operation and technology

The following products are produced in blast-furnaces: iron for steelmaking and castings, and ferro-alloys. The chemical specifications for these products are broadly given in table 9.

TABLE 9. SPECIFICATIONS OF METALLIC PRODUCTS

Product	Composition range (percentage)				
	Silicon	Sulphur (maximum values)	Phosphorus	Manganese	Total carbon
Iron for steelmaking					
Basic pig-iron	1.50 maximum	0.05	0.04-0.90	0.40-2.00	3.50-4.40
Acid pig-iron (Bessemer)	1.00-2.25	0.045	0.040-0.135	0.50-1.00	4.15-4.40
Acid pig-iron (open hearth)	0.70-1.50	0.045	0.050	0.50-2.50	4.15-4.40
Oxygen-steelmaking pig-iron	0.20-2.00	0.05	0.40 maximum	0.40-2.50	3.50-4.40
Iron for castings					
Low phosphorus	0.50-3.00	0.035	0.035 maximum	1.25 maximum	3.00-4.50
Intermediate phosphorus	1.00-3.00	0.050	0.036-0.075	1.25 maximum	3.00-4.50
Bessemer	1.00-3.00	0.050	0.076-0.100	1.25 maximum	3.00-4.50
Malleable	0.75-3.50	0.050	0.101-0.300	0.50-1.25	3.00-4.50
Foundry	3.5 maximum	0.050	0.301-0.900	0.25-0.75	3.00-4.50
Ferro-alloys					
Spiegel	1.00-4.50	0.050	0.14-0.25	16-30	6.5 maximum
Standard ferro-manganese	1.2 maximum	0.050	0.35 maximum	74-82	7.6 maximum
Ferro-silicon	5.0-17.0	0.060	0.30 maximum	1.0-2.0	1.5 maximum
Ferro-phosphorus	1.50-1.75	0.05	15-24	0.07-0.50	1.10-2.00

In the blast-furnace process, iron-bearing materials such as lumpy iron ore, sinter, pellets, mill scale, open-hearth or Bessemer slag, iron and steel scrap, fuel (coke) and flux (limestone or dolomite) etc. are charged into the top of the furnace. Heated air (blast) and, in recent practice, gas, oil or pulverized coal are blown from the tuyères. The blast air burns part of the fuel to produce heat for the chemical reactions involved and for melting the iron, while the balance of the fuel and part of the gas from the combustion remove the oxygen combined with the metal.

To produce 1 ton of pig-iron, the following average amounts of material are required: iron ore (1.7 tons), coke and other fuels (500-600 kg), fluxes (250-295 kg), scrap (330-340 kg), air (1.8-2.0 tons).

Besides 1 ton of pig-iron produced, nearly 0.2-0.4 tons of slag, 2.5-3.5 tons of blast-furnace gas, and about 0.05 tons of flue dust are also produced.

The depletion of world coking-coal resources, together with the increasing cost of metallurgical coke, has necessitated considerable innovations in blast-furnace technology in recent years. The sharp rise in daily output capacities has been accompanied by a steady fall in coke consumption per ton of iron. The average coke consumption has fallen in the United Kingdom from about 1100 kg to 650 kg per ton, in the United States from 940 kg to about 600 kg per ton, and in Japan from 900 kg to below 400 kg per ton. Parallel with the improved fuel efficiency leading to reduced total coke consumption, there has been a realization of the fact that a part of the coke can be replaced by hydrocarbon fuel injected through the tuyères. The various steps taken to reduce coke consumption and increase metal production are briefly described below.

Raw materials

Preparation of iron ore The current modern practice is to use closely-sized lumpy ore of generally 10-50 mm. Studies carried out in Japan have shown that a 1 per cent decrease in the proportion of lumps larger than 35 mm in the burden corresponds to a decrease in the coke rate of approximately 2 kg/ton, and a 1 per cent decrease in fines of less than 6 mm leads to a decrease in the coke rate of 1 kg/ton. Larger furnaces in Japan now use ore of greater than 8 mm and less than 25 mm.

It is well known that the iron ore used should be of as high a grade as possible. Experiments carried out in the USSR have shown that for an increase of 1 per cent Fe in the burden, productivity increases by 2 per cent and coke rate decreases by 3 per cent. It has also been estimated that for every 1 per cent reduction in the alumina content of the ore, the coke and flux rates decrease by 40 kg and 60 kg per ton respectively, and the consequent increase in production of pig-iron made would be about 2.0-2.5 per cent.

Sinter and its effect Due to close sizing of iron ore and limiting the top size of ore for direct charging into the blast-furnace, large quantities of ore fines are generated, both at mine sites and in ironworks. These fines are utilized after sintering.

In order to derive the maximum possible metallurgical advantages from use of sinter in ironmaking, it should have a suitable chemical composition, be self-fluxing or, better still, super-fluxed, and have adequate physical strength to withstand handling without crumbling into fines. The sinter should have neither unsintered particles nor a

glassy structure, and should be non-magnetic, that is, the iron oxide content should be as low as possible.

In modern blast-furnaces, large percentages of sinter, up to 70-80 per cent, are generally used in the burden, and in some cases the burden consists exclusively of fluxed or superfluxed sinter. The size of the sinter used is generally 5-50 mm. Practice in Japan has shown that a 10 per cent increase in sinter ratio in the blast-furnace burden corresponds to a coke rate reduction of approximately 10 kg/ton and to a production increase of about 2 per cent.

Pelletizing Pelletizing, the newest of the agglomerating processes, is being increasingly used in blast-furnaces despite the unquestioned benefits of sinter to blast-furnace performance. The pelletizing process is desirable for agglomeration of finely-divided ore concentrates, blue dust, flue dust and the like. Fluxed pellets of uniform size and of adequate strength, good porosity and reducibility, and desirable swelling index, become ideal feed for ironmaking in the blast-furnace. The process enables use of a very high-grade ore concentrate in the form of pellets, which have advantages almost equal to those of self-fluxing sinters.

Blast-humidity control

Changes in atmospheric humidity have an effect on furnace operation because of the endothermic reaction taking place when steam comes in contact with hot coke. With the modern practice of using high blast temperatures, humidity has also to be adjusted to prevent the furnace from hanging or operating irregularly. The higher the hot blast temperature, the higher the moisture content the hot blast has to be given to obtain a suitable flame temperature in the hearth zone.

The use of very high hot-blast temperatures together with the addition of the proper amount of moisture in the blast has made it possible to increase blast-furnace production rates substantially. Moisture produces more reducing gas per unit volume than dry air does. However, the possibility of controlling the flame temperature of the tuyères by the injection of fuel has almost eliminated the necessity of steam addition. In modern practice, the temperature of hot metal has been controlled by fuel injection in highly industrialized countries such as Japan.

Oxygen enrichment of blast

Enrichment of blast with oxygen reduces the volume of gas required at the same daily productivity, and has therefore an effect identical to the use of a higher top pressure. However, if the blast air is enriched, the flame temperature increases, so that for oxygen contents above 22 per cent, moisture or hydrocarbon fuels must be added to control the flame temperature. It can broadly be assumed that

for every 1 per cent of enrichment a production rate increase of 3-4 per cent can be achieved. The higher the hot-blast temperature, the smaller the improvement in the production rate for each per cent of increase in oxygen content. Oxygen enrichment was first tried as early as 1959 in Japan. It has been established that a 1 per cent enrichment of oxygen brought about a 5 per cent increase in production and a slight decrease in the coke rate. Currently, up to 3.8 per cent enrichment is being adopted.

In a 2 000 m³ blast-furnace in the USSR, when oxygen in the blast was increased from 26.7 to 34.7 per cent, and the natural gas injection increased from 8.6 to 14.3 per cent, productivity increased by 15.3 per cent, with a lowering of the coke rate from 484 to 445 kg per ton of pig-iron and a coke replacement factor of 0.91 kg of coke per cubic metre of natural gas. With a blast containing 25 per cent oxygen and an injection of 100-105 m³ of natural gas per ton of pig-iron, it has also been possible to increase the daily pig-iron production of another 2 014³ blast-furnace at Magnitogorsk, USSR, by 1.8-2.2 per cent, with a reduction in coke consumption of 2.4 per cent per 1 per cent of oxygen.¹ With the actual relation between expenditure on coke, natural gas and oxygen, the greatest economic benefit was obtained with the consumption of oxygen and natural gas, each of which amounted to 90 m³ per ton of pig-iron. It has also been established that the limiting levels of natural gas consumption for blast-oxygen contents of 21, 25, 30, 35 and 40 per cent calculated from empirical data are respectively 20-65, 70-140, 100-210, 130-230 and 150-245 m³ per ton of pig-iron. Gas consumption should be closer to the upper limits when a combined blast is used to obtain a greater lowering of coke consumption, and if the aim is mainly to obtain a rise in productivity, gas consumption closer to the lower limits should be ensured.

Fuel injection

Fuel injection through the tuyères is the most important practice next to raw material preparation in blast-furnace operations. With the development of the means of obtaining higher hot-blast temperatures and the need for controlling the flame temperature, it became apparent that cold hydrocarbon fuels could be injected into the blast-furnace tuyères not only for controlling the flame temperature, but also to replace some of the coke.

In the presence of large quantities of coke, the hydrocarbon fuels can burn only to carbon monoxide and hydrogen; consequently they produce less heat than that produced by the hot coke they replace. So long as a blast-furnace has the stove capacity for obtaining higher hot-blast temperatures, and so long

¹ N. N. Babarykin and others, "Use of combined blast in Magnitogorsk Combine blast-furnaces", *Steel in the USSR*, March 1976, pp. 118-122.

as moisture must be added to the blast to lower the flame temperature, hydrocarbon fuels can be used to advantage because their endothermic effect provides a means of controlling the temperature in the hearth.

Generally, when tuyère-injected fuels are used, the moisture content of the blast must be decreased. Natural gas, coke-oven gas, fuel oil, pulverized coal, tar and slurries of oil and coal have been used in this manner.

Fuel injection has the following favourable features: reduction of coke rate; stabilization of blast-furnace operation and production increase; compensation of shortage in coke-oven capacity or saving of coking facilities; and low investment costs of the blast-furnace injection equipment.

For the effective application of fuel injection, attention should be given to the following points: heat compensation capable of keeping the flame temperature at the tuyère within a certain range; combustion load limits for individual tuyères; changes in permeability, heat exchange, and reducing reactions caused by varying the volume of gas produced per ton of hot metal; injection method (atomizing techniques etc.); and measures to cope with unexpected blow-off and other difficulties.

Heat compensation and combustion load are the most important problems in injecting fuel. Reduction of coke rate brought about by the fuel injection is related to the blast temperature.

According to the operating results of a blast-furnace at Kawasaki, Japan, the limit of oil injection is 30 kg/ton at a constant humidity, that is, with a lower limit of 2 000°C and an upper limit of 2 200°C of the theoretical flame temperature and a blast temperature of 800°C. With a blast temperature of 1 100°C, even an injection of 110 kg/ton is possible.

The limit of the ratio to the amount of oxygen necessary for the perfect combustion of oil is considered to be 1.1-1.2, and this would result in a ratio of oil injection of 110 kg/ton and that of tar injection of about 90 kg/ton.

In recent operations of blast-furnaces at Fukuyama, Japan, oil injection of about 80 kg/ton was applied in combination with oxygen enrichment to reduce the coke rate to about 400 kg/ton. It was however necessary to keep the theoretical flame temperature at a rather high level of 2 300°-2 400°C. It was found possible to reduce the coke rate to 210 kg/ton by injecting reducing gas made from about 220 kg of oil per ton in an experimental blast-furnace.

It has been reported² that injection of coal dust in a 700 m³ blast-furnace at Donetsk, USSR, using 200 kg of lean coal per ton of pig-iron, altered the productivity and coke rates very substantially.

²Yaroshevskii and others, "Blast-furnace operating technology with injection into the hearth of natural gas and 100-200 kg of coal-dust fuel per tonne of pig-iron", *Steel in the USSR*, June 1976, pp. 298-300.

The data obtained over a period of one year are as follows: change in productivity with coal injection 0.2 per cent; reduction in coke consumption 45.1 kg/ton of iron, or 7.5 per cent; reduction in pig-iron production cost about \$1.3 per ton of pig-iron.

High-top-pressure operation

One of the limiting factors in attempting to increase the production rate of a blast-furnace is the lifting effect caused by the large volume of gases blowing upward through the burden. This lifting effect prevents the burden from descending normally and causes a loss rather than an increase in production.

An increase in blast volume reduces the passage time of gas through the furnace, that is, the reaction time. This lowers the utilization ratio of gas, resulting in a rise in top temperature and an increase in coke rate. Pressure drop in the furnace also increases. However, if the blast volume is increased over a certain limit, an imbalance of permeability through the furnace would result in channelling and flooding.

To achieve a uniform ascent of gases and a satisfactory burden descent, it is necessary to keep the gas-flow velocity within certain limits. Use of higher gas pressure in the furnace, leading to decreased gas volume, is a very effective measure for this purpose. Most of the newly-constructed furnaces in Japan are operated with high top pressure. Top pressure has been gradually raised to the present level of 1.0-1.5 kg/cm², and sometimes up to 2.5 kg/cm² in bigger blast-furnaces.

As a result of work done by iron producers in Japan, the following conclusions have been drawn concerning the effects of high-top-pressure operation:

(a) With an increase in top pressure the permeability index increases proportionately;

(b) An increase in top pressure raises productivity; a 0.1 kg/cm² increase in top pressure corresponds to an increase of about 1.5 per cent in productivity;

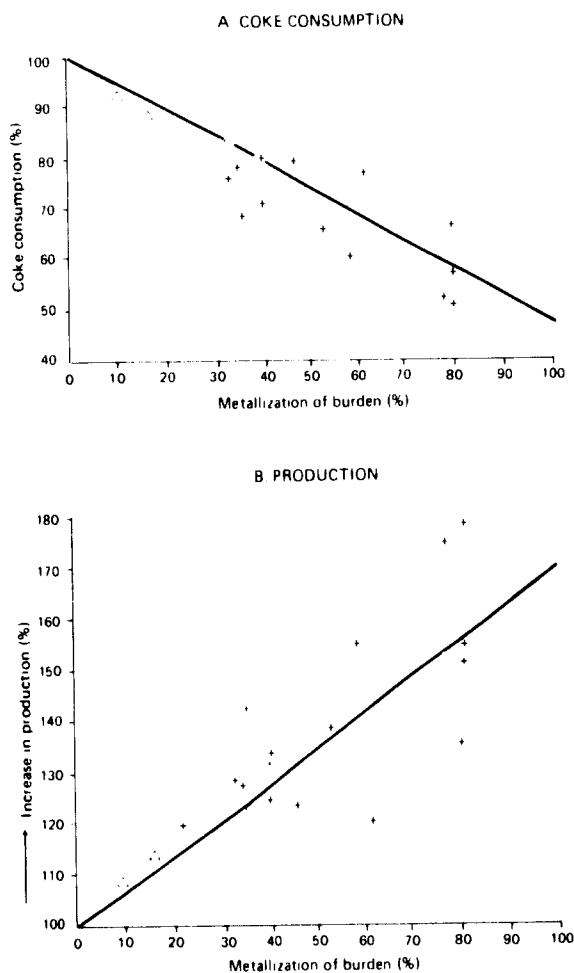
(c) An increase in top pressure reduces solution loss, whereas higher daily productivity results in an increased solution loss; but the overall effect is a slight reduction in the coke rate;

(d) A higher top pressure leads to a better permeability and therefore reduces the number of slips. Higher top pressure slows down the top gas velocity, thus reducing the dust lost.

Pre-reduced burden

Blast-furnaces performance data have shown that coke consumption declines and productivity increases when pre-reduced burden is used, as reflected in figure IV.

Figure IV. Relation of coke consumption and production to pre-reduced burden.



Source: United States Bureau of Mines, Publication No. 166.

Note: +: United States Bureau of Mines data; O, Δ: Steel Company of Canada Ltd. data.

Large-capacity blast-furnace

Steelmaking in integrated iron and steelworks has been changed from the open-hearth furnace process to the converter process. The high efficiency of the latter process and the expansion of unit capacity of rolling facilities have ensured the most economical annual unit productivity of ironworks. To cope with this trend towards larger facilities and the increasing demand for hot metal from converters, there is a strict requirement for a stable supply of large quantities of low-cost hot metal to be produced in blast-furnaces having increased capacities.

Comprehensive data on the factors involved in the operation of larger-capacity blast-furnaces indicate that the production cost of hot metal is lowered.

This advantage, though marked up to an inner volume of 2 500 m³, becomes less over the limit of about 4 000 m³, as has been found in studies carried out by iron and steelmakers in Japan.

In deciding the size of a blast-furnace, the properties of raw materials should be given serious consideration. It should be noted in particular that burden size and strength requirements have recently become more and more stringent. Moreover, blast-furnace capacity is increased by enlarging the hearth diameter and other cross-sectional dimensions, while not raising the furnace height too much. The top pressure for a larger-capacity blast-furnace is also proportionately increased.

Although the operating costs for a larger-size blast-furnace are higher than for a smaller-size furnace, the investment costs are lower in the former case. The overall operating and investment costs are lower than those for a smaller-size blast-furnace. In a larger blast-furnace, the solution loss is higher because furnace height is not great enough in relation to the increase in inner-volume acid, hence the descent time for the burden is shortened. The labour-cost is reduced with the increase in furnace capacity, but the operating cost of the blower increases when high-top-pressure operation is applied for a stable operation. Since the increase in blower operating cost exceeds the decrease in labour cost, the overall operation costs tend to be higher with the increase in furnace capacity.

The construction cost per ton of iron is inversely proportional to a third power of the inner volume. This results in a decrease in unit depreciation cost, interest on capital and running cost.

The Industrial Environmental Research Laboratory of the United States Environmental Protection Agency has estimated the investment and running costs of blast-furnaces of various sizes in the United States using oxidized and metallized pellets. These figures are given in tables 10 through 13.

Operating data from some of the larger blast-furnaces in Japan and the USSR are presented in tables 14 and 15 respectively. Technical data on a recently-commissioned blast-furnace at the Linz Works of Voest Alpine A.G., Austria, are also given.

Technical data on a recently-commissioned blast-furnace^a

Hearth diameter	11 m
Total volume	3 055 m ³
Utilizable volume	2 504 m ³
Maximum furnace gas pressure	2.5 bar
Maximum hot-air temperature	1 350°C
Maximum air quantity	360 000 Nm ³ /hour
Tuyères	28
Pig-iron taps	3
Slag taps	1
Daily pig-iron output	5 500 tons
Coke consumption per day	2 300 tons
Burden rate per day	9 500 tons

^aLinz Works of Voest Alpine A.G., Austria.

TABLE 10. COST STRUCTURE IN NEW BLAST-FURNACE FACILITIES
(Annual design capacity, 1.22×10^6 tons of hot metal; capital investment (CI), \$90 million; location, Great Lakes)

<i>Cost components</i>	<i>Costing units or annual cost basis</i>	<i>Dollars per unit</i>	<i>Units consumed per ton of product</i>	<i>Dollars per ton of product</i>
<i>Variable costs</i>				
<i>Raw materials</i>				
Pellets	B.t.u.	0.45	84.7	38.11
Limestone	Tons	5.00	0.25	1.25
<i>Energy</i>				
Purchased coke	Tons	90.25	0.53	47.85
Electric power purchased	kWh	0.016	25.00	0.40
<i>Energy credits</i>				
Blast-furnace	10^6 B.t.u.	2.0	3.8	(7.60)
<i>Water</i>				
Cooling (circulating rate)	10^3 gallons	0.05	11	0.55
Direct operating labour (wages)	Man-hours	7.00	0.15	1.05
Direct supervisory wages	15% labour			0.16
Maintenance labour and material	5% CI			3.69
Labour overhead	35% (L+S)			0.42
<i>Miscellaneous variable costs/credits</i>				
Slag sampling				0.25
Scrap credit	Tons	80.00	0.01	(0.80)
Total variable costs				85.33
<i>Fixed costs</i>				
Plant overheads	65% (L+S)			0.79
Local taxes and insurance	2% CI			1.48
Depreciation (18 years)	5.55%			4.09
Total production costs				91.69
Return on investment (pretax)	20% CI			14.75
Total				106.44

Source: Compiled from a study carried out by the United States Environmental Protection Agency, EPA-600/7-76-034c, December 1976.

Note: B.t.u.: British thermal units; L: direct operating labour; S: direct supervisory wages.

TABLE 11. COST STRUCTURE IN NEW BLAST-FURNACE
(Annual design capacity, 2.6×10^6 tons of hot metal; capital investment (CI), \$156 million; location, Great Lakes)

<i>Cost component</i>	<i>Costing units or annual cost basis</i>	<i>Dollars per unit</i>	<i>Units consumed per ton of product</i>	<i>Dollars per ton of product</i>
<i>Variable costs</i>				
<i>Raw materials</i>				
Pellets	B.t.u.	0.45	84.7	38.11
Limestone	Tons	5.00	0.332	1.66
<i>Energy</i>				
Purchased coke	Tons	90.0	0.53	47.70
Electrical power purchased	kWh	0.016	25	0.40
<i>Energy credits</i>				
Blast-furnace gas	10^6 B.t.u.	2.0	3.8	(7.60)

TABLE 11 (continued)

Cost component	Costing units or annual cost basis	Dollars per unit	Units consumed per ton of product	Dollars per ton of product
<i>Variable costs (continued)</i>				
Water				
Cooling (circulating rate)	10 ³ gallons	0.05	11	0.55
Direct operating labour (wages)	Man-hours	7.00	0.10	0.70
Direct supervisory wages	15% labour			0.11
Maintenance labour and material	5% CI			3.00
Labour overhead	35% (L+S)			0.28
Miscellaneous variable costs/credits				0.25
Slag sampling				(0.80)
Scrap credit	Tons	80.00	0.01	
Total variable costs				84.36
<i>Fixed costs</i>				
Plant overheads	65% (L+S)			0.53
Local taxes and insurance	2% CI			1.20
Depreciation	5.55%			3.33
Total production costs				89.42
Return on investment (pretax)	20% CI			12.00
Pollution control				4.57
Total				105.99

Source: Compiled from a study carried out by the United States Environmental Protection Agency, EPA-600/7-76-034c, December 1976.

Note: B.t.u.: British thermal units; itu: long ton unit (22.4 lb of contained Fe); L: direct operating labour; S: direct supervisory wages.

TABLE 12. COST STRUCTURE IN NEW BLAST-FURNACE WITH REDUCED COKE RATE
(Annual design capacity, 2.6 x 10⁶ tons of hot metal; capital investment (CI), \$152 million; location, Great Lakes)

Cost component	Costing units or annual cost basis	Dollars per unit	Units consumed per ton of product	Dollars per ton of product
<i>Variable</i>				
Raw materials				
Pellets	itu	0.45	84.7	38.11
Limestone	Tons	5.00	0.225	1.12
Energy				
Purchased coke	Tons	90.00	0.515	46.35
Electric power purchased	kWh	0.016	24	0.38
Energy credits				
Blast-furnace gas	10 ⁶ B.t.u.	2.00	3.69	(7.38)
Water				
Cooling (circulating rate)	10 ³ gallons	0.05	10.6	0.53
Direct operating labour (wages)	Man-hours	7.00	0.10	0.70
Direct supervisory wages	15% labour			0.11
Maintenance labour				
Maintenance materials and supplies	5% CI			2.92
Labour overhead	35% (L+S)			0.28
Miscellaneous costs/credits				0.25
Slag sampling				(0.80)
Scrap credit	Tons	80.00	0.01	
Total variable costs				82.57

<i>Cost component</i>	<i>Costing units or annual cost basis</i>	<i>Dollars per unit</i>	<i>Units consumed per ton of product</i>	<i>Dollars per ton of product</i>
<i>Fixed costs</i>				
Plant overheads	65% (L+S)			0.53
Local taxes and insurance	2% CI			1.17
Depreciation (18 years)	5.55%			3.25
				<u>87.52</u>
Total production costs				87.52
Return on investment (pretax)	20% CI			11.70
Pollution control				4.06
				<u>103.28</u>
Total				103.28

Source: Compiled from a study carried out by the United States Environmental Protection Agency, EPA-600/7-76-034c, December 1976.

Note: B.t.u.: British thermal units; itu: long ton units (22.4 lb of contained Fe); L: direct operating labour; S: direct supervisory wages.

TABLE 13. COST STRUCTURE IN NEW SPONGE IRON (93% METALLIZED) FACILITIES
(Annual design capacity, 1.2×10^6 tons of hot metal; capital investment (CI), \$168 million; location, Great Lakes)

<i>Cost component</i>	<i>Costing units or annual cost basis</i>	<i>Dollars per unit</i>	<i>Units consumed per ton of product</i>	<i>Dollars per ton of product</i>
<i>Variable costs</i>				
<i>Raw materials</i>				
Pellets	itu	0.45	8.5	38.25
Limestone	Tons	5.00	0.140	0.70
<i>Energy</i>				
Purchased fuel	10^6 B.t.u.	2.00		
Coal	Tons	25.00	0.625	15.62
Purchased steam	10^6 B.t.u.	3.00		
Electric power purchased	kWh	0.018	56.0	0.90
Miscellaneous				
<i>Water</i>				
Process (consumption)	10^3 gallons	0.50		
Cooling (circulating rate)	10^3 gallons	0.05	4	0.20
Direct operating labour (wages)	Man-hours	7.00	0.20	1.40
Direct supervisory wages	15% L			0.21
Maintenance materials and supplies	4% CI			5.60
Labour overhead	35% (L+S)			0.56
				<u>63.44</u>
Total variable costs				63.44
<i>Fixed costs</i>				
Plant overheads	65% (L+S)			1.05
Local taxes and insurance	2% CI			2.80
Depreciation (18 years)	5.55%			7.77
				<u>75.06</u>
Total production costs				75.06
Return on investment (pretax)	20% CI			28.00
				<u>103.06</u>
Total				103.06

Source: Compiled from a study carried out by the United States Environmental Protection Agency, EPA-600/7-76-034c, December 1976.

Note: B.t.u.: British thermal units; itu: long ton unit (22.4 lb of contained Fe); L: direct operating labour; S: direct supervisory wages.

TABLE 14. OPERATING RESULTS OF FUKUYAMA BLAST-FURNACES (JAPAN)

Item	August 1966 (1 blast-furnace)	February 1968 (2 blast-furnaces)	July 1969 (3 blast-furnaces)	April 1971 (4 blast-furnaces)
Inner volume (m ³)	2 004	2 626	3 016	4 197
Hearth diameter (m)	9.8	11.2	11.8	13.8
Production (tons/day)	4 639	6 064	6 834	10 017
Productivity (tons/day/m ³)	2.32	2.31	2.27	2.39
Coke rate (kg/ton)	469	469	465	437
Oil rate (kg/ton)	34	26	40	52
Fuel rate (kg/ton)	503	495	505	489
Sinter rate (%)	70	64	76	80
Slag volume (kg/ton)	253	260	274	290
Blast volume (Nm ³ /min)	4 073	5 309	5 842	7 722
Blast pressure (kg/cm ²)	2.24	2.61	2.93	3.61
Top pressure (kg/cm ²)	0.59	0.99	1.36	2.10
Blast temperature (°C)	1 112	1 146	1 159	1 200
O ₂ enrichment (%)	0	0	0.6	1.4
Si in pig-iron (%)	0.71	0.69	0.66	0.71
S in pig-iron (%)	0.038	0.037	0.038	0.032
CaO/SiO ₂ in slag (%)	1.23	1.17	1.16	1.13
Coke ash (%)	9.2	9.1	10.6	10.5
Drum index	92.4	93.2	91.8	92.0

TABLE 15. DATA ON NEW BLAST-FURNACES IN THE USSR

Item	5 000 m ³ blast-furnace	5 500 m ³ blast-furnace
Hearth diameter (m)	14.7	15.1
Useful height (m)	33.5	34.3
Daily production (tons)	12 900	14 000
Burden (kg/ton of pig-iron)		
Sinter	1 000	837
Pellets	680	810
Coke (dry)	375	370
Energy resources (m ³ /ton of pig-iron)		
Blast (including 6% losses)	830	820
Process oxygen	158	156
Natural gas (including 3% losses)	149	147
Blast volume (m ³ /min)	7 500	8 000
Blast temperature (°C)	1 400	1 400
Yield of smelting products per ton of pig-iron		
Slag (kg)	325	290
Top gas (m ³)	1 380	1 370
Flue dust (kg)	25	25
Manganese in pig-iron (%)	1.2	1.2
Sulphur in pig-iron (%)	0.035	0.020
Top gas calorific value (kcal/m ³)	1 190	1 180
Top gas pressure (bar)	2.5	2.5

Electric ironmaking techniques

There are about 100 electric ironmaking furnaces throughout the world, with a total capacity of 3-4 million tons of pig-iron per year. It was at the end of the nineteenth century that the first electric furnaces were built for smelting iron ore and

producing liquid metal. Furnaces of this type currently in use are the open-bath arc furnace and the low-shaft or submerged-arc furnace.

Open-bath arc furnace

In this type of furnace the electrode is in contact with the slag, but it is not immersed in the solid charge. The open-bath arc furnace is being used at a plant at Sorel, Quebec, Canada, for the melting of ilmenite with selective reduction of iron and production of a low-carbon hot metal and titaniferous slag. It is used in the Strategic-Udy process

Submerged-arc furnace

The submerged-arc furnace is an intermediate type between the shaft and the open-bath furnaces. It was industrially developed by Tysland and Hole in Norway, where, since the first 600 kW Tysland-Hole furnaces, a number of larger furnaces have been built, including four 20-25 MW furnaces of Tysland-Hole type in Moi Rana, and one unit of the same size in Svelgen. The larger installations in different countries are given in table 16.

Steelworks in Japan have independently developed electric pig-iron smelting furnaces based on relatively small open furnaces of 1-10 MW capacities. Most of the units in operation smelt the locally available beach sand concentrates. Taking into account the Japanese furnaces and the smaller open type of furnace, total world smelting capacity is estimated at approximately 1 000 MW for a production of 3-4 million tons of pig-iron annually.

Consumption of electrical energy has averaged about 2 200 kWh per ton of pig-iron produced, and

TABLE 16. LARGE TYSLAND-HOLT FURNACES IN VARIOUS COUNTRIES

Country	Number of furnaces	Smelting capacity (MW)
Canada	1	10
Finland	1	10
India	2	20
Israel	2	29
Italy	13	102.5
Japan	2	13
Norway	6	108
Peru	2	20
Philippines	2	20
Portugal	1	10
Spain	1	6.5
Sweden	6	54
Switzerland	1	8.5
Venezuela	9	180
Yugoslavia	3	30
Total	52	621.5

electrode paste consumption between 8 and 15 kg/ton. About 1 500 kg of sintered ore and 400-430 kg of coke are generally consumed to yield 1 ton of pig-iron.

Pre-reduction of smelting material

With regard to direct reduction processes, a clear distinction exists between pre-reduced material for iron smelting in the blast-furnace or electric furnace, on the one hand, and sponge iron for direct steelmaking in an electric furnace, on the other. In the former case, the end-product is pig-iron. A brief review of the use of pre-reduced material for making pig-iron is given below.

Since considerable pre-reduction takes place in the shaft of the blast-furnace itself, the advantages of using a pre-reduced charge are not very significant in blast-furnaces. On the other hand, use of a pre-reduced charge for electric smelting results in a substantial economy in the consumption of coke, electricity and fluxes. Productivity also tends to improve with the use of a pre-reduced charge. The extent of improvement depends upon the degree of metallization of the pre-reduced iron ore.

Of the large number of processes for pre-reduction and making sponge-iron, the following are the most promising:

- (a) Solid reductants
- External heating
 - (i) Echeverria
 - (ii) Kinglor Meteor
 - Internal heating
 - (i) Krupp
 - (ii) SL/RN

(b) Gaseous reductants

- Shaft furnace
 - (i) Midrex (gas recycling)
 - (ii) Purofer (gas recycling)
 - (iii) Armco (no gas recycling)
- Fixed bed
 - HvL
- Fluidized bed
 - (i) HIB (under pressure)
 - (ii) Fior (under pressure)
 - (iii) Novalfer (without pressure)

The processes used in commercial production are briefly outlined below.

The rotary-kiln process is a continuous process which uses non-coking coals as the reducing agent. It is advantageous to those countries where coal is available in abundance along with good quality iron ore. Both high-grade lumpy iron ore and high-grade pellets can be utilized.

The Midrex process uses a continuous shaft furnace with reformed natural gas as the reducing agent. The recycling of gases together with the advantages of a continuous process results in low energy requirements.

The HvL process is a batch process using a fixed-bed reactor and reformed natural gas as the reducing agent.

The Armco process is similar to the Midrex process.

The high iron briquette (HIB) process involves fluidized bed reduction using steam-reformed natural gas. The fine ore is first pre-heated, then reduced by steam-reformed natural gas at a temperature of about 700°C followed by hot-briquetting and cooling.

Economic considerations

In comparing the capabilities of electric reduction furnaces and blast-furnaces, it is difficult to conceive of an electric furnace plant having an ironmaking capacity of 2 million tons or more; the blast-furnace, with its large output, is more suited for this. For plants of up to approximately 1 million tons capacity, the electric reduction furnace could be considered, as electric furnaces would offer the advantage of greater flexibility than a large blast-furnace.

It may, however, be noted that the trend toward larger electric furnaces employing pre-heated and pre-reduced charges will make possible the production of 500 tons/day per furnace in the near future, with possibilities of higher tonnages of up to 1 000 tons/day. Furnaces of this type could be

considered for plants of 1.5 million tons, such as those projected for eastern Siberia or Africa, where tremendous hydroelectric power potential exists.

Concerning capital costs, there does not appear to be a great deal of difference between blast-furnaces and electric reduction furnaces of comparable capacity. For large or medium-size plants, annual costs may be more or less equal per ton of iron produced by either a modern blast-furnace or an electric reduction furnace, each with ore preparation but without stocking facilities, coke-oven plants, power stations and mines.

If investment costs are comparable for blast-furnace and electric furnace installations, then the choice between them is largely a matter of the relative cost of electricity and the cost and availability of coking coals. Studies in the USSR have shown equivalent costs for the two processes if 1 kg of coke yields the equivalent of 3.5-4 kWh, and 108 000-120 000 kVA furnaces with six electrodes and an output of 1 000 tons/day are employed.

In conclusion, it may be pointed out that the choice between the two processes would be mainly dependent on availability of cheap electric power and coal. While developing countries have a dominant share of world resources of gas, oil and hydroelectric power potential, and a good share of iron ores, they suffer from a lack of coal, particularly coking coal. It would therefore be more desirable for many developing countries to consider adopting the electric furnace method, which has the added advantages of flexibility of size and comparatively lower plant costs.

From a technological viewpoint, the substitution of charcoal for coke is entirely acceptable and feasible. However, its utilization will call for a programme of reforestation in order to ensure adequate supplies of charcoal on a long-term basis. The paramount importance of preserving environmental balance in developing countries would make it even more urgent to undertake suitable reforestation programmes.

III. Steelmaking

Pig-iron consists of the element iron combined with numerous other chemical elements, the most common of which are carbon, manganese, phosphorus, sulphur and silicon. Depending upon the composition of the raw materials used in iron-making—principally iron ore (beneficiated or otherwise), coke and limestone—and the manner in which the furnace is operated, pig-iron may contain 3.0-4.5 per cent of carbon, 0.15-2.5 per cent or more of manganese, as much as 0.2 per cent of sulphur, 0.025-2.5 per cent of phosphorus and 0.5-4.0 per cent of silicon. In refining pig-iron to convert it into steel, all five of these elements must either be removed almost entirely or at least reduced drastically in amount.

Modern steelmaking processes, including the pneumatic processes, are divided into two general classes from the chemical standpoint: acid processes and basic processes. Carbon, manganese and silicon can be removed with relative ease by any of the processes, either acid or basic. The removal of phosphorus and sulphur requires special conditions that can be met only by the basic processes involving the addition of lime to the chemical system to form a basic slag that is capable of forming compounds with phosphorus and sulphur during refining operations, thereby removing them from the metal.

Processes

There are five different steelmaking processes, namely the open-hearth, pneumatic, continuous, electric and atomic energy processes, with several modifications of each.

Open hearth

The open-hearth furnace is both reverberatory and regenerative. The charge is melted on a refractory hearth, which is shallow in relation to its length, by a flame passing over the charge so that both the charge and the relatively low roof above the hearth, built of refractory brick, are heated by the flame. The hot gases from the combustion of fuel pass out of the reverberatory furnace chamber through passages into regenerative chambers containing fire brick.

There are two types of open-hearth process, namely the acid and the basic open-hearth processes.

In the acid open-hearth process, the hearth of the furnace is of acid brick construction. The initial charge consists of cold pig-iron or cold pig-iron and scrap. No ore can be added with the charge, for iron oxide, being a base, would react with the acid refractory lining and destroy it rapidly. The melting of scrap alone would also be undesirable, for its oxidation products would have a similar detrimental effect. In this process, only silicon, manganese, carbon and a trace of phosphorus are eliminated. None of the sulphur is eliminated. In fact, the finished steel may contain a slightly higher percentage of both phosphorus and sulphur than the amount present in the charge.

Acid open-hearth pig-iron usually has the following specifications: silicon—less than 1.5 per cent; manganese—1.0-2.5 per cent; phosphorus and sulphur—under 0.045 per cent; and carbon—4.15-4.40 per cent.

The composition of acid open-hearth slag is generally as follows: SiO_2 —52-56 per cent; FeO —20.5-29 per cent; MnO —10-20.5 per cent; P_2O_5 —0.02-0.045 per cent; Al_2O_3 —3.1-4.2 per cent; CaO —0.7-5.4 per cent; and MgO —0.12 per cent to trace element.

In the basic open-hearth process, the hearth of the furnace is lined with basic refractory material such as magnesite and burned dolomite to permit charging of limestone and use of a basic slag for removal of phosphorus and sulphur.

Basic open-hearth pig-iron usually has the following specifications: silicon—under 1.5 per cent; manganese—0.4-2.0 per cent; sulphur—under 0.05 per cent; phosphorus—under 0.9 per cent; and carbon—2.5-4.40 per cent.

The basic open-hearth process has undergone changes in the course of time. The earlier "ore practice" has been changed to "oxygen roof-lance practice". The use of oxygen increases flame temperature and the rate of heat transfer to the charge, thereby speeding up the melting of high-scrap charges. It also compensates for deficiencies in air supply and regenerator capacity. In modern oxygen roof-lance practice, the flow of oxygen to the furnace is begun immediately after the addition of hot metal and is continued throughout most of the refining period. It has been reported that there is a saving in heat time of 10-25 per cent and a decrease in fuel consumption of 18-35 per cent when oxygen roof-lancing is used.

Pneumatic processes

As in the case of other steelmaking methods, there are two chemical types of pneumatic process—acid and basic. In both types, air, high-purity oxygen or a combination of these and other oxidizing gases are blown under pressure through, on to or over the surface of molten pig-iron to produce steel. If air is used for blowing, its nitrogen content serves no useful purpose and actually removes heat from the system. Nitrogen absorbed during blowing is considered an undesirable impurity in the finished steel.

There are several ways in which the oxidizing gas can be supplied in a pneumatic process.

Bottom-blown converter

The bottom-blown converter has been the principal type used in both the acid and basic air-blown pneumatic processes for steel production. The blast travels the full length of the molten bath, thus representing the extreme of submerged blowing practices.

The bottom-blown acid process known as the acid Bessemer process was formerly used to produce most of the world's steel supply. Iron of the correct chemical composition and temperature is required for the process, generally in accordance with the following specifications: silicon—1.1-1.7 per cent; manganese—0.4-0.7 per cent; phosphorus—0.09 per cent maximum; sulphur—0.03 per cent maximum; and carbon—4.0-4.5 per cent.

The blow to produce steel lasts for a period of only 10-15 minutes, and speed of operation is very rapid. The ratio of silicon to manganese in pig-iron should be 2-2.5, as the former is a source of heat. Oxygen enrichment of air-blast helps in reducing blowing time and permits greater utilization of cold iron and scrap.

The bottom-blown basic process, known as the basic Bessemer process, or Thomas, or Thomas-Gilchrist process, was never extensively adopted in view of the development of the basic open-hearth process.

The chemical composition of a typical blast-furnace iron for the bottom-blown basic Bessemer process generally includes the following elements: silicon—0.2-0.5 per cent; manganese—0.6-1.0 per cent; phosphorus—1.4-2.0 per cent; and sulphur—0.03-0.05 per cent. The chemical composition and properties of steels produced by this method more closely approach the composition and properties of basic open-hearth steels of similar grade than do comparable steels made by the acid Bessemer process. But the nitrogen content of bottom-blown basic pneumatic steels is definitely higher than that of basic open-hearth steel. For this reason, the properties of air-blown steels made by the basic Bessemer process, while more similar to basic open-hearth steels than are acid Bessemer steels, are still inadequate for certain applications because of their higher strength, lower ductility and susceptibility to strain ageing.

Side-blown converter

The chemical reactions which occur in the side-blown acid converter are similar to those occurring in the bottom-blown acid converter. However, in the former, all the tuyères are above the liquid level of the bath and enter through the side of the vessel. The steel produced is much hotter from a similar iron charge than that produced by the acid bottom-blown converter, and thus the silicon content of the iron can be somewhat lower. Nitrogen content of the finished steel is obviously much lower than in the steel made by the bottom-blown acid process. By enriching the air blow with oxygen, the blowing time is reduced.

Stora-Kaldo process

In this process oxygen is introduced at an angle with respect to the surface of the liquid metal bath contained in a tilted, rotating vessel. In contrast to bottom-blown Bessemer vessels and top-blown oxygen vessels, the Kaldo furnace is tilted at about 15°-20° from the horizontal while operating, and can be rotated about its longitudinal axis at speeds up to 30 rev/min.

As in all top-blown pneumatic processes, phosphorus is eliminated simultaneously with carbon, unlike in the basic Bessemer process, where it is eliminated only after all the carbon has been removed.

Rotor process

The rotor process was developed in Oberhausen, Federal Republic of Germany, and is often referred to as the Oberhausen process. It employs the same rotary principle as the Kaldo process, except that the speed of rotation is 0.5-2 rev/min. Another distinct feature is that two oxygen lances, one for high-purity oxygen and the other for the commercial variety of oxygen, are used. Tap-to-tap time for a 66-ton vessel is about 2 hours.

Linz-Donawitz (LD) process

In the basic oxygen process, substantially pure oxygen is introduced from above the surface of the bath in a basic-lined vessel. The LD process was developed in Austria and initially designed to employ pig-iron produced from local ores that are high in manganese and low in phosphorus content. The basic oxygen process has been readily adapted to the processing of pig-iron of medium and high phosphorus content. The rapid growth of the LD process outstripped steel production by the basic open-hearth process. All grades of steel, including high-carbon, low-alloy and stainless steels, are now being produced by the LD process.

The process has been extensively employed, particularly in Europe, Japan and the United States. Continued improvement in this technology has permitted reduction of heat time to 30-40 minutes, as compared with earlier heat times of 50-60 minutes. This has been mainly possible due to intensified oxygen blowing rates and use of multi-hole lances. For example, with a 400-ton converter employing a 7-hole lance capable of blowing about 1 400 cubic metres of oxygen per minute, the tap-to-tap time is about 35 to 40 minutes.

To increase scrap usage in the LD process from the normal 20-30 per cent to 40-50 per cent range, pre-heating of the scrap by oxy-fuel lance and the use of silicon carbide and calcium carbide have been adopted.

Continuous improvements in the quality of refractory linings have increased lining life from 400-800 heats up to, in some cases, approximately 2 000 heats. Besides lining quality and the use of dolomitic lime as part of the flux charge, improved operating techniques and control of the silicon content of the hot metal have influenced lining life.

Table 17 provides data on installations employing the Linz-Donawitz process throughout the world.

Bottom-blowing oxygen process

The Oxygen Bottom Maxhütte (OBM) process is based on the idea that blowing pure oxygen through the converter bottom would have the inherent advantages of a quieter blow and better mixing. The process was developed by Eisenwerk-Gesellschaft Maximilianshütte m.b.H. in the Federal Republic of Germany, jointly with L'Aire Liquide of Montreal, Canada, and the first heat was made in December 1967. By March 1968, regular production was started. The unique feature of the process is shielding of the oxygen tuyère by a larger-diameter pipe through the annulus of which hydrocarbons, such as propane or natural gas, are blown. The endothermic decomposition of the hydrocarbon at the mouth of the tuyère effectively cools the tuyère. With this concentric tuyère, pure oxygen could be introduced into a steel bath without excessive refractory erosion.

The vessel for the OBM process is similar to the Bessemer converter, except that only 6-15 tuyères are used as against 250-300 in the Thomas converters. The inner oxygen tube is made of copper and the outer tube is of stainless steel. The converter shell is lined with dolomite bricks and the bottom is rammed with tar dolomite. Oxygen with lime powder is injected through the inner pipe and natural gas through the outer tube. The natural gas inflow is about 9 kg/cm², which is reduced to about 4 kg/cm², and the flow rate is about 300 m³/h. The line pressure of oxygen is about 15 kg/cm² which is reduced to about 13 kg/cm² in the converter. The

TABLE 17. TOTAL ANNUAL CAPACITIES OF INSTALLATIONS EMPLOYING THE LINZ-DONAWITZ PROCESS

(Millions of tons of raw steel)

Country	Current installations	Planned installations
Algeria	0.45	11.30
Argentina	1.60	1.25
Australia	6.72	2.81
Austria	5.20	
Belgium	15.15	
Brazil	7.38	8.88
Bulgaria	1.70	
Canada	9.20	2.38
Chile	1.10	0.91
China	4.15	
Czechoslovakia	3.70	0.80
Egypt	1.20	
Finland	2.15	
France	17.40	1.00
Germany, Federal Republic of	45.52	2.54
Greece	0.90	
Hungary		1.10
India	2.96	4.18
Iran	2.00	
Italy	14.62	0.50
Japan	122.78	11.20
Luxembourg	6.14	0.96
Malaysia	0.18	
Mexico	5.00	6.00
Morocco		1.00
Netherlands	7.15	
Norway	0.70	
Peru	0.33	1.80
Philippines		2.12
Poland	8.00	4.00
Portugal	0.50	
Republic of Korea	2.20	
Romania	7.00	3.50
South Africa	6.69	10.54
Spain	7.58	1.13
Sweden	1.97	1.80
Tunisia	0.18	
Turkey	2.90	2.30
United Kingdom	18.61	7.55
United States of America	81.06	5.10
USSR	40.20	44.20
Venezuela		6.00
Yugoslavia	2.70	1.50
Total	464.97	148.35

Source: Metal Bulletin Monthly, August 1977.

oxygen flow rate can be varied from 4 000 to 10 000 m³/h. The ratio of oxygen to natural gas for ignition is approximately 9:1, and during the blowing period this is changed to 30:1. Natural gas consumption averages 6 m³ per ton of steel and oxygen consumption about 64 m³ per ton.

Comparison of the OBM and LD processes

The advantage of the OBM process as compared with the oxygen top-blowing processes are mainly as follows:

(a) Unlike the oxygen top-blowing process, only about a quarter of the iron is evaporated; consequently, less red fume is developed;

(b) The iron-oxide content of the slag, which in contrast with other steelmaking methods approaches equilibrium with the metal bath, is substantially lower. This, together with the results indicated in point (a) above, means a 2.5 per cent increase in yield compared with the oxygen top-blowing processes, which shows a decisive economic advantage of the OBM process;

(c) The intense agitation of the bath by the introduction of the refining gas through the bottom leads to a more rapid dissolution of the scrap, whereby the total refining time in the converter is reduced to 10 minutes;

(d) The simultaneous introduction of lime powder and oxygen makes it possible to obtain completely splash-free refining, which has considerable importance and is independent of the pig-iron composition. Besides advantages relating to process control, the design of the gas-cleaning equipment is also favourably affected;

(e) The amount of scrap to be processed by the OBM process is increased by about 35 per cent compared with the oxygen top-blowing processes. The reasons are as follows: the lower iron evaporation losses; the elimination of cooled lances; shorter refining time; and the possibility of pre-heating the converter during the charging operation by introducing through the tuyères oxygen and natural gas in stoichiometric ratio;

(f) OBM plants require only two thirds of the building height necessary for oxygen top-blowing processes. For this reason, OBM converters are particularly suitable for installation in existing open-hearth steel plants;

(g) Owing to the short blowing periods and the slopping-free blowing behaviour, a considerable increase in production can also be achieved by the use of large OBM converters as opposed to LD converters.

LWS process

A variation of the OBM process is the LWS process (Creusot-Loire and Wendel-Sidelor with Sprunck and Co.). In the LWS process, fuel oil instead of natural gas is used for shielding the oxygen stream. The first commercial operation started in 1971 with a 30-ton converter, and the process, like the OBM, is used for refining high-phosphorus iron. The use of oil in the LWS process ensures better safety in operation compared with the use of propane or natural gas. The process is expected to be attractive in places where natural gas is not available and propane is expensive.

Q-BOP process

The OBM process was adapted to large furnaces by U.S. Steel and given the name of Q-BOP process. The letter "Q" stands for "quiet, quick, quality", and emphasizes the advantages of the oxygen bottom-blowing process. The metallurgical performance of the Q-BOP process has proved to be excellent. Phosphorus could be reduced to a low level. The ability to introduce powdered lime into the steel bath results in good desulphurization, and steels with a sulphur content of 0.02 per cent are produced from hot metal containing 0.07 per cent sulphur. Nitrogen levels at turndown are generally below 0.0025 per cent and hydrogen content at about 2.6 parts per million.

The process developed for treating low-phosphorus iron and for use on large converters with high tonnages involves an elaborate tuyère-gas control system to ensure that the different gases are supplied in the right quantity and sequence to the converter. The blowing time for a 200-ton Q-BOP converter is about 12 minutes, as against 17 minutes for an LD converter.

SIP process

The submerged injection process (SIP) is an offshoot of the OBM process and was developed by the Sydney Steel Corporation, Canada, on its 200-ton tilting open-hearth furnace. The tuyères are located below the bath on the back slope of the hearth. The refining time is reduced to some 12 minutes for an oxygen rate of about 900 m³/min.

Advantages claimed for the SIP are high production rates, a 2-4 per cent increase in yield, and less particle content in the waste gases due to quiet bath conditions. Since the end burners, as in the conventional open-hearth process, are retained, the scrap-melting ability of the SIP process is greater than that of the other processes, and there is no difficulty in melting up to 60 per cent scrap in the charge. Results of experimental heats in a 30-ton Q-BOP converter in the United States indicate that the following products are comparable to similar products from conventional open-hearth furnaces or LD converters: rimmed and mechanically-capped low-carbon steels of both hot- and cold-rolled quality; deep-drawing tin plate and other tin-mill products including corrosion-resistant types; structural quality plates including carbon-manganese steels and low-alloy high-strength steels, as well as high-carbon rail steels.

It has been estimated by M. N. Dastur and Company Private Limited, India, that capital investment on a bottom-blowing installation should be lower than that required for an LD plant. In another study carried out by the above-mentioned consultancy firm, it has been shown that revamping and expanding the capacity of a Thomas converter

The NRIM process, developed in Japan and currently in an experimental stage, involves the use of multi-stage trough-type furnaces.

Electric processes

Electric current can be used for heating in two ways: by utilizing the heat generated in electrical conductors by their inherent resistance to the flow of current; and by utilizing the heat radiated by the electric arc.

Resistance method

Two general methods of heating by resistance are possible. One is the indirect method, in which the charge is heated by radiation and conduction from separate resistors through which current is passed. This method, however, is impracticable for steel-making. The other is the direct method, in which the current is passed through the metal charge or bath itself. A high-voltage low-amperage current is transformed to low-voltage high-amperage current and passed through the bath or charge. The bath acts as a secondary circuit for the current, which is generated from a primary circuit by induction. This method is therefore called induction heating.

Electric arc method

Arc heating may be applied in two general ways. First, the arcs may be made between electrodes supported above the metal in the furnace, which is thus heated solely by radiation from the arc. This is commonly known as indirect arc heating. Second, the arc may be made between the electrodes and the metal. This method, known as direct arc heating, allows current to flow through the bath, so that heat developed by the electrical resistance of the metal is added to that radiated from the arcs. This system using a non-conduction bottom has been successful in making steel in electric furnaces.

There are three different types of furnaces used for steelmaking by the electric arc method: the open-bath arc furnace, which is comparable to an open hearth furnace; the electric shaft furnace, which is the electrical version of the blast-furnace; and the low-shaft or submerged-arc furnace, which lies between the first two.

Open-bath furnaces In open-bath furnaces, the electrode is in contact with the slag, but it is not immersed in the solid charge. The furnace operates as an isothermal zone where the reactions are limited to the direct reduction of the oxides, the eventual carburization of the liquid metal and the formation of slag.

A recent development of the open-bath furnace is the Lubatti furnace, the bath of which is not

actually open but is rather covered by a very small layer of charge material.

The Strategic-Udy process employs this type of furnace using pre-heated highly reduced material. A 33 000 kVA industrial unit employing this process has been commissioned in Venezuela.

Electric shaft furnaces A few furnaces of this type were made (500-600 kw in Sweden), but their performance was not quite satisfactory compared with that of open-bath or submerged-arc furnaces. The electric shaft furnace did not become popular and was finally abandoned.

Submerged-arc furnaces Most electric ironmaking plants at present make use of the submerged-arc furnace. Tysland-Hole furnaces of 50 000 kVA capacity or more are operating in several parts of the world. A 60 000 kVA furnace has been constructed and is operating at Moi Rana, Norway.

From the time of its invention, the electric furnace has been used for making special steels and has therefore been characterized by modest heat size and low hourly output. Its potential increased in the 1960s, thus leading to the ultra-high power (UHP) concept. The main features of this techniques are as follows:

(a) There is an increase in transformer power, making possible the regular attainment of specific power levels of 500-600 kVA/ton. A 100-ton furnace at Fos, France, and an 80-ton furnace at Himeji, Japan, have capacities of 77 MVA and 60 MVA respectively;

(b) Thermal efficiency is much improved using short, stable ores and high voltage and power, the power factor falling from 0.9 to 0.71.

Furnaces in the 80-100 ton range are quite common in many countries. Italy, South Africa, the United Kingdom, the United States of America and the USSR are some of the countries having furnaces of 150-ton capacity or more. A company in the United States for example, has a furnace of 400-ton and another of 700-ton capacity.

Auxiliary heating arrangements are made outside the furnace for reasons of economy in electric power and in order to enhance productivity.

Atomic energy and steelmaking

The application of atomic energy to steel production would involve the use of a reactor to generate both electricity and the hot reducing gas needed in the direct reduction process. The following two methods have been considered feasible:

(a) One method involves using the heat from a high-temperature gas-cooled reactor (HTGR) to reform hydrocarbon gas for the direct reduction of

iron ore, the temperature being indirectly supplied by the hot helium from the reactor. The direct-reduction sponge iron thus obtained would be refined to steel in an electric furnace powered by heat from the reactor;

(b) Another method involves using reactor heat to obtain hydrogen (by splitting water), which is then used for the direct reduction of iron ore, thus avoiding the use of fossil fuel.

The overriding problem at present is to design a safe, long-life catalytic reformer that can be heated by hot helium gas from the reactor. The Federal Republic of Germany, Japan, Switzerland, the United States and the USSR are active in developing this technology. Belgium, France, Italy and the United States have also joined together to develop nuclear steelmaking technology. However, the first use of nuclear fission in steelmaking is not expected to take place before the period 1985-1990, and perhaps even 5-10 years later.

Share of each process in steel production

Steelmaking has undergone a substantial transformation in the last two decades or so, owing to the introduction on an industrial scale of pure-oxygen melting processes, the development of electric furnaces for the production of ingot steel, and the introduction of metallurgical processing downstream of the steel production furnace.

The steel output of some countries shows striking differences in the choice of technology. This is due to the specific infrastructure of the steel industry in each country.

Table 20 shows the relative importance of the various steelmaking processes in different countries.

Process selection

Of the many different steel production processes employed in the past, only a few have survived in the face of stiff competition. Many decisions have to be made when formulating plans for the establishment of new iron and steelworks, or when considering a major expansion of an existing plant. One of the most important of these is the process route by which the available raw materials are to be converted into the required product mix.

If the general location of the steelworks is decided upon, certain parameters will already be fixed, such as the local market demand and its pattern of growth, which in turn determines the size of the plant and its range of products, the cost of the services available etc.

The choice of processes is strongly influenced by the following factors: the cost and availability of iron ore and steel scrap, of coking and steam coal, oil, natural gas, electricity, charcoal and other fuels, of

capital for the purchase and construction of facilities, and of trained workers and technical staff; transportation facilities available to move the large tonnages of raw materials involved; environmental considerations, primarily the disposal of water and waste rock from mining, and slag from smelting and refining operations; the possible risk involved in choosing a particular technology which may be at various stages of development; and the cost of available capital.

Optimum works scale

With the increasing size of most production units, the optimum size of an integrated steelworks has risen accordingly. There is no general law governing the economics of plant size. Each individual case must be carefully examined to determine the optimum solution. To arrive at a suitable plant scale, the most economical rolling mill capacities to match the current and forecast market demands are first chosen. Then working backwards through the process stages, optimum capacities of all other production units are determined.

The approximate range of economic viability for the capacities of some major units is given below, thus illustrating the complexity of the balancing problem.

APPROXIMATE RANGE OF ECONOMIC VIABILITY OF MAJOR PRODUCTION UNITS

<i>Production unit</i>	<i>Viable capacity range (millions of tons per year)</i>
Blast-furnace	1.0-4.0 (iron)
Sinter plant	Up to 8.0 (sinter)
Coke-oven battery	Up to 2.0 (coke)
	<i>Raw steel</i>
LD steelmaking shop	Up to 10.0
Slab mill	2.0-6.0
Bloom mill	1.5-6.0
Continuous slab caster	Up to 2.0
Continuous bloom caster	Up to 1.0
Continuous billet caster	Up to 0.5
	<i>Steel products</i>
Plate mill	0.5-3.0
Hot strip mill	1.0-6.0
Cold strip mill	0.1-2.5
Structural mill	0.3-1.5
Bar mill	0.01-1.0
Rod mill	0.1-1.0

Cost data

The economic advantages of bottom-blowing installations as compared with LD plants have already been mentioned. M. N. Dastur and Company Private Limited, India, have also estimated the relative investment costs for different steelmaking processes, including the cost of oxygen plant, ingot-casting facilities and calcining plant. These cost figures are given in table 21.

TABLE 20. SHARE OF EACH STEELMAKING PROCESS IN THE STEEL PRODUCTION OF VARIOUS COUNTRIES

Country	Share of each steelmaking process (percentage)												Steel production (millions of tons)		
	Electric			Open hearth			Oxygen			Thomas			1960	1970	1975
	1960	1970	1975	1960	1970	1975	1960	1970	1975	1960	1970	1975	1960	1970	1975
Austria	12.6	12.1	11.8	31.3	16.3	7.4	56.1	71.6	80.8	-	-	-	3.163	4.078	4.068
Belgium	3.1	3.6	5.8	5.8	2.2	1.4	-	52.6	92.8	91.1	41.6	-	6.698	12.607	11.584
Bulgaria	10.0	20.1	21.3	90.0	26.1	22.8	-	53.8	55.9	-	-	-	0.260	1.800	2.265
Canada	10.8	13.4	18.6	89.2	86.6	23.6	-	-	57.8	-	-	-	5.164	11.212	13.025
Czechoslovakia	12.3	11.7	10.9	83.3	68.3	64.2	-	18.0	23.9	3.9	2.0	1.1	6.768	11.480	14.324
Finland	60.1	22.3	17.0	39.9	13.8	8.7	-	63.9	74.2	-	-	-	0.278	1.169	1.618
France	8.7	9.8	12.8	29.9	18.8	7.2	0.5	29.2	64.5	60.9	42.2	15.5	17.181	23.774	21.530
Germany, Federal Republic of	6.4	9.4	11.2	47.3	26.5	16.9	2.5	56.7	70.4	43.8	8.2	1.4	34.032	45.041	40.415
Hungary	9.8	9.3	9.2	90.2	90.7	90.8	-	-	-	-	-	-	1.887	3.110	3.671
Italy	38.6	40.5	43.0	55.9	28.0	11.3	-	31.5	45.7	5.5	-	-	8.461	17.277	21.836
Japan	20.1	16.8	16.4	68.0	4.1	1.1	11.9	79.1	82.5	-	-	-	22.138	93.322	102.314
Luxembourg	2.0	1.8	1.4	-	-	-	-	37.6	70.1	98.0	60.6	28.5	4.084	5.462	4.624
Poland	7.7	4.8	8.7	92.3	81.2	65.9	-	14.0	25.4	-	-	-	6.661	11.750	15.007
Portugal	24.7	18.9	22.0	-	-	-	75.3	81.1	78.0	-	-	-	0.968	0.385	0.430
Romania	10.9	9.3	12.7	89.1	62.0	49.9	-	28.7	37.4	-	-	-	1.671	6.517	9.549
Spain	71.4	35.2	36.1	-	26.4	9.8	14.8	38.4	54.1	13.8	-	-	1.938	7.429	11.242
Sweden	48.0	39.5	40.9	34.0	24.4	15.7	4.0	32.3	43.4	14.0	3.8	-	3.219	5.497	5.611
Union of Soviet Socialist Republics	4.5	9.2	10.0	88.4	72.6	64.7	4.4	17.2	24.6	2.7	1.0	0.7	56.627	115.889	141.325
United Kingdom	6.9	20.0	27.6	84.9	48.3	22.0	0.1	31.7	50.3	8.0	-	-	24.571	28.316	20.198
United States of America	8.5	15.3	19.4	87.0	36.5	19.0	3.3	48.1	61.6	1.2	-	-	90.067	122.120	108.250
Yugoslavia	8.0	19.3	26.4	92.0	73.9	62.4	-	6.9	11.2	-	-	-	1.442	2.228	2.916

Source: United Nations Economic and Social Council, Economic Commission for Europe, "Structural changes in the iron and steel industry" (STEEL/GE.3/R.3/Add.1), pp. 17-19.

TABLE 21. RELATIVE INVESTMENT COSTS
(Basis: 1 million ingot tons per year)

Process	Investment per annual ingot ton (\$US)	Relative investment index
Open hearth	75	100
LD	47	63
OBM/LWS (Q-BOP)	42	60
Electric arc furnace	56	75

Source: SEAIISI Quarterly, January 1976.

A study carried out by Paul Marshall in 1976 of estimated capital costs of some new steel-producing facilities is summarized in table 22.

The table shows a wide range of capital cost variations, namely \$210-\$750 per ton of raw steel at 1975 prices.

Table 23 compares indexes of capital costs for blast-furnace/basic-oxygen-furnace (BF-BOF) and direct-reduction/electric-furnace (DR-EF) installations for different plant capacities.

As shown in table 23 capital cost decreases by about 48 per cent and 37 per cent as plant capacities

TABLE 22. ESTIMATED CAPITAL GOODS

Process sequence	Capacity (tons/year)		Cost (\$/ton)		Total cost of raw steel (millions of dollars)
	Raw steel	Finished steel	Raw steel	Finished steel	
35-ton EF, continuous caster, hot mill	50 000	47 000	240	250	12
DR, 35-ton EF, continuous caster, hot mill	50 000	47 000	340	361	17
70-ton EF, continuous caster, hot mill	100 000	94 000	230	245	23
Two 150-ton EF, continuous caster, merchant mill	100 000	85 000	210	247	21
DR, two 150-ton EF, continuous caster, merchant mill	500 000	450 000	320	355	160
Two 150-ton EF, ingot casting, primary and merchant mills	500 000	450 000	426	475	213
One 1 500 tons/day BF, coke oven, one 100-ton BOF, continuous caster, merchant mill	500 000	375 000	370	500	185
Three 200-ton OH, continuous caster, merchant mill	500 000	450 000	610	670	305
Three 200-ton EF, continuous caster, hot mill, merchant mill	500 000	450 000	340	380	170
Three 200-ton EF, continuous caster, hot mill, heavy structural mill	1 000 000	900 000	346	385	346
DR, three 200-ton EF, continuous caster, hot mill, heavy structural mill	1 000 000	900 000	606	670	606
Three 200-ton EF, ingot casting, primary mill, hot mill, merchant mill	1 000 000	750 000	390	520	390
One 6 000 tons/day BF, sinter plant, coke plant, two 150-ton BOF, continuous caster, hot mill, merchant mill	2 000 000	1 600 000	477	600	955
DR, six 200-ton EF, continuous caster, hot mill, cold mill, galvanizing	2 000 000	1 600 000	482	603	965
Fully integrated plant BF, coke plant, BOF, continuous caster for merchant, bar and structural mills, continuous hot mill, hot and cold sheet mills, coated products, plate mill	8 000 000	6 000 000	675-750	900-1 000	6 000

Source: UNIDO/ICIS.25.

Note: BF, blast-furnace; BOF, basic oxygen furnace; DR, direct reduction; EF, electric furnace; OH, open hearth furnace.

TABLE 23. INDEX OF CAPITAL COSTS OF SELECTED PLANTS

Plant capacity (millions of tons)	Index of capital costs	
	BF-BOF	DR-EF
0.2	213	170
0.3	180	145
0.4	164	131
0.5	155	121
0.6	146	114
0.7	140	111
1.0	129	109
2.0	115	108
3.0	110	107
5.0	100	

Source: Commodities Research Unit Survey.

Note: BF-BOF: blast furnace/basic oxygen furnace;
DR-EF: direct reduction/electric furnace.

increase from 0.2 to 3.0 million tons for BF-BOF and DR-EF plants, respectively.

Tables 24 and 25 show capital costs for a 3-million-ton BF-BOF plant and for a 500 000-ton DR-EF plant, respectively.

The estimates given in tables 24 and 25 do not take into account costs for mining, power, housing, contingencies and interest paid during the construction period.

In a study conducted by UNIDO, the operating cost for 3-million-ton BF-BOF installations located in Brazil, Japan, the United States and Western Europe has been estimated at \$134.70 per ton.

The percentage distribution of inputs in relation to total operating costs is shown in table 26.

The capital investment and operating cost for a new electric furnace with a capacity of 1.71 million tons per year are given in table 27. Cost figures for a new oxygen furnace of similar capacity are given in table 28.

TABLE 24. CAPITAL COSTS OF A 3-MILLION-TON BF-BOF PLANT

Item	Installed steel-making capacity (\$/ton)	Capital cost structure (%)	Capital costs (millions of dollars)
Coke plant	56.0	8.6	168.0
Blast furnace	75.0	11.6	225.0
Basic oxygen furnace	43.0	6.6	130.0
Continuous casters	43.0	6.6	131.0
Mixed rolling facilities	188.0	29.0	563.0
General facilities	42.0	6.5	125.0
Subtotal (fixed assets)	447.0	68.9	1 342.0
Engineering, procurement and inspection (5% of fixed assets)	22.0	3.4	
Administration, advisory and expediting costs (6% of fixed assets)	27.0	4.2	
Pre-operating expenses (3% of fixed assets)	13.0	2.0	
Subtotal (project implementation and pre-operating expenses)	62.0	9.6	
Fixed capital costs	509.0	78.6	
Infrastructural investment	72.0	11.1	
Subtotal (fixed capital costs plus infrastructure)	581.0	89.7	
Working capital (15% of fixed assets)	67.0	10.3	
Subtotal	648.0	100.0	
Interest paid during implementation	42.0		
Total	690.0		

Source: UNIDO/IC. S.25.

Note: BF-BOF: blast-furnace/basic oxygen furnace.

TABLE 25. CAPITAL COSTS OF A 500 000-TON DR-EF PLANT

Item	Installed steel-making capacity (\$/ton)	Capital cost structure (%)	Capital costs (millions of dollars)
Direct reduction (350 000 tons)	77.0	25.5	38.4
Electric furnace (500 000 tons)	40.0	13.2	20.0
Six-strand caster (500 000 tons)	45.0	14.9	22.5
Merchant bar mill (450 000 tons)	63.0	20.9	31.5
Subtotal (fixed assets)	225.0	74.5	112.4
Engineering, procurement and inspection (5% of fixed assets)	11.0	3.6	
Administrative, advisory and expediting costs (3% of fixed assets)	7.0	2.3	
Pre-operating expenses (3% of fixed assets)	7.0	2.3	
Subtotal (project implementation)	25.0	8.2	
Fixed capital costs	250.0	82.7	
Infrastructural investment (15% of fixed assets)	34.0	11.3	
Subtotal (fixed capital costs plus infrastructure)	284.0	94.0	
Working capital (8% of fixed assets)	18.0	6.0	
Subtotal	302.0	100.0	
Interest paid during implementation	10.0		
Total	312.0		

Source: UNIDO/ICIS.25.

Note: DR-EF: direct reduction/electric furnace.

TABLE 26. COST STRUCTURE OF A BF-BOF PLANT
(Percentage)

	Blast-furnace shop	Basic oxygen steelmaking
Raw materials and primary energy	84.7	93.0 ^a
Utilities	1.9	1.0
Labour	5.2	0.8
Overheads	3.3	0.5
Maintenance (4% of investment)	2.0	2.9
Local taxes and insurance	0.1	0.4
Depreciation (5.5% of investment)	2.8	1.4
Total	100.0	100.0

Source: UNIDO/ICIS.25.

^aLiquid metal resulting from the previous process and transfer costs.

TABLE 27. COST STRUCTURE OF A NEW ELECTRIC FURNACE SHOP
(Annual design capacity, 1.71×10^6 tons of steel, capital investment (CI), \$65 million; location, Great Lakes)

Cost components	Costing units or annual cost basis	Dollars per unit	Units consumed per ton of steel	Dollars per ton of steel
<i>Variable costs</i>				
Raw materials				
Reduced pellets	Tons of iron	102.88	0.75	77.16
Scrap	Tons	80	0.32	25.60
Energy				
Electric power purchased	kWh	0.016	600	9.60
Electrodes	lb	0.55	10	5.50
Direct operating labour (wages)	Man-hours	7.00	0.3	2.10
Direct supervisory wages	15% labour			0.32
Maintenance labour and materials	6% CI			2.27
Labour overhead	35% (L+S)			0.85
Miscellaneous variable costs/credits				
Refractories				2.00
Fluxes, oxygen, miscellaneous nonmetallics				1.00
Metallic additions				1.50
Total variable costs				127.90
<i>Fixed costs</i>				
Plant overheads	65% (L+S)			1.57
Local taxes and insurance	2% CI			0.76
Depreciation (18 years)	5.55%			2.10
Total production costs				132.33
Return on investment (pretax)	20% CI			7.60
Total				139.93

Source: United States Industrial Environmental Research Laboratory, EPA-600/7-76-034c.
Note: L: direct operating labour; S: direct supervisory wages.

TABLE 28. COST STRUCTURE OF A NEW BASIC OXYGEN FURNACE
(Annual design capacity, 1.71 million tons of steel, capital investment (CI), \$45 million; location, Great Lakes)

Cost components	Costing units or annual cost basis	Dollars per unit	Units consumed per ton of steel	Dollars per ton of steel
<i>Variable costs</i>				
Raw materials				
Hot metal (93% Fe)	Ton	106.42	0.83	88.33
Scrap	Ton	80.00	0.35	28
Energy				
Electric power purchased	kWh	0.016	30	0.48
Energy credits (term to be specified)				
Carbon monoxide	10^6 B.t.u.	2.00	0.44	(0.88)
Water				
Cooling (circulating rate)	1 000	0.05	2	0.10
Direct operating labour (wages)	Man-hours	7	0.25	1.75
Direct supervisory wages	15% labour			0.26
Maintenance labour and materials	8% CI			2.11
Labour overhead	35% (L+S)			0.70

<i>Cost components</i>	<i>Costing units or annual cost basis</i>	<i>Dollars per unit</i>	<i>Units consumed per ton of steel</i>	<i>Dollars per tons of steel</i>
<i>Variable costs (continued)</i>				
<i>Miscellaneous variable costs/credits</i>				
Oxygen	Ton	10	0.08	0.80
FeMn, lime, spar				3.00
Slag, disposal, hot metal scrap treatment				1.00
Total variable costs				125.61
<i>Fixed costs</i>				
Plant overheads	65% (L+S)			1.31
Local taxes and insurance	2% CI			0.53
Depreciation (18 years)	5.55%			1.45
Total production costs				128.90
Return on investment (pretax)	20% CI			5.26
Total				134.16

Source: United States Industrial Environmental Research Laboratory, EPA-600/7-76-034c.

Note: B.t.u.: British thermal units; L: direct operating labour; S: direct supervisory wages.

IV. Steel casting

A brief account of different processes of steel casting, including continuous casting, is given below. Information about continuous-casting machines installed in various countries, criteria for process selection, and cost data are also included.

Production of steel castings

As far as mechanical properties are concerned, steel castings are inferior to wrought-steel products. However, steel castings cover a very large field and include steel-mill service items such as charging boxes, blast-furnace bells, cinder pots, rolls for certain types of rolling mills etc.; transport industry items such as castings for couplings, journal boxes, brake-shoes, cylinders, valves, engine beds etc.; and several hundred other items of use in the chemical, petroleum, mining, agricultural and construction industries etc.

There are two classes of castings, static castings, and centrifugal castings. Static castings are made by using the proper type of moulds and applying atmospheric pressure and gravity to form castings when molten steel is poured into the moulds. Centrifugal castings involve the use of centrifugal action to perform the function of gravity in static casting for the flow of liquid metal. Tubes, pipes, bushings, sleeves etc. are manufactured by horizontal centrifugal casting techniques involving the rotation of the mould on a horizontal axis. Gears, piston rings, impellers, propellers, turbine diaphragms etc. are produced by the vertical centrifugal casting method. Centrifugal castings are sounder, have fewer inclusions, and can be produced at a greater rate than static castings.

The mechanical properties of cast steels can be developed by suitable heat treatment and addition of alloying elements. Nickel, chromium, manganese, molybdenum, and vanadium are the alloying elements commonly used for improving the properties of the castings.

The common heat-treatment methods applied to steel castings are outlined below.

Annealing

Annealing relieves the tensile and yield strength and increases ductility. Machinability is also improved.

Normalizing

Normalizing is similar to the annealing process. Harder steel with higher yield and tensile strength is obtained. Internal stresses are removed by tempering the normalized steel.

Quenching and tempering

Quenching and tempering are confined principally to high-carbon and alloy-steel castings where high strength and resistance to impact and/or abrasion is required. The steel is first annealed or normalized, reheated and quenched. Tempering treatment follows immediately.

Flame hardening

If different hardnesses are required in two different zones of a casting, as in the case of a pinion gear which should have wear-resistant teeth with a machinable bore, the flame-hardening technique is adopted. The casting is first annealed or quenched and then only the surfaces to be hardened are heated to the hardening temperature by a torch or induction-heating apparatus. The heated parts are then quenched in water.

Ingot casting

After a heat of steel is properly refined, the liquid steel is tapped into a refractory-lined open-top steel ladle. Additions of alloying materials and deoxidizers are made during tapping of a heat. The molten steel is poured into a series of moulds of the desired shape and dimensions, and after solidification, the ingot is stripped off the mould.

Ingot moulds are of two principal types, one with the big end down, the other with the big end up.

The big-end-down moulds are further classified as open-top and bottle-top. The big-end-up moulds are also similarly classified as open-bottom, closed-bottom and plug-bottom. The moulds are made of cast iron, the inner walls of which may be plain-sided, corrugated or fluted.

The rate of solidification of molten steel in the mould depends on the thickness, shape and temperature of the mould; the amount of superheat

of the liquid steel; the type of steel and its chemical composition etc. The presence of pipes and blow-holes, segregation, internal fissures, cracks and non-metallic inclusions are some of the factors which may be controlled by appropriate measures.

The ingot steel, after reheating in soaking pits is rolled into bloom, slab or billet. These rolled primary products are then further rolled into the desired shapes and cross-section.

Bottom-pressure casting

A process that may be used to bypass the ingot and primary mill stages in the production of wrought steels is the bottom-pressure casting method. A ladle filled with mother steel is placed in a pressure vessel. This vessel is covered with a lid in which a pouring tube is inserted that dips down into the mother steel almost to the bottom of the ladle. A goose-neck connects the pouring tube to the mould in the casting position. When air-pressure is applied to the pressure vessel, molten steel rises in the pouring tube and goose-neck and enters the mould. The rate of casting is controlled by regulating the pressure of air.

The mould is enclosed in a flask which has a gate that retains the molten metal in the mould after the cast is completed. The gate is closed after the mould has been filled and the pressure vessel is then exhausted to the atmosphere. After the casting has solidified, the mould is removed by the stripping machine, and the drag and the casting are separated from the cope using special equipment. The casting is then placed on the cooling conveyor.

Continuous casting

Until recently, steel in the form of blooms, slabs, and billets, was produced mainly by hot-rolling of ingots to form blooms and slabs. Billets resulted from the further hot-rolling of blooms. However, some blooms and slabs are still produced by other means of hot-working, such as forging by hammering or pressing.

Research and development in many countries, has resulted in industrialization of the method of continuous casting of molten steel directly into the form of slabs and billets, bypassing the ingot stage and the necessity for hot-rolling operations.

The attractiveness of casting molten steel continuously into useful shapes led to a long series of attempts to develop various designs of machines. The problems posed due to high melting point, high specific heat and low thermal conductivity of steel were gradually overcome.

When molten steel comes into contact with the walls of the water-cooled mould, a thin solid skin forms. Due to thermal contraction, the skin separates from the mould shortly after solidification. The rate

of heat abstraction from the casting being slow, molten steel persists within the interior of the section for some distance below the bottom of the mould. The thickness of the skin increases due to the action of water sprays as the casting moves downward and gradually becomes completely solidified.

The mass of solid steel casting is supported as it descends by driven pinch rolls that also control line speed by controlling the rate of withdrawal of the casting from the mould. Oscillation of the mould up and down for predetermined distances at controlled rates during casting helps to prevent the casting from sticking in the mould.

The successful application of the continuous casting process in a steel plant is dependent upon many factors other than just mechanical equipment and feasibility. Some of the important factors which would determine the number of casting strands or ladle position are as follows: tons of liquid steel per heat, or, in other words, size of the furnace; tap-to-tap time of the furnace; possibility of programming the tapping time; total tons of steel to be cast per day; shape of the cast product; and casting rate.

The tapping temperature of steel is generally 1650°-1690°C for Concast installations, depending on the life of the furnace refractory and the taphole condition. This is about 30°-40°C hotter as compared with steel tapped for small ingot-making.

The desired physico-chemical properties of billets produced by the conventional method are attained by close control of the process beginning with the ingot-making stage, soaking and rolling. During the soaking and rolling stages, homogeneity of cast structures is obtained by diffusion processes. On the other hand, the continuous casting process will affect the physico-chemical properties, in the as-cast condition, of the billets so produced. In order to ensure that finished products using continuously cast billets will have properties comparable to those of finished products using conventionally produced billets, the following factors should be given careful consideration: surface configuration, which includes, among other things, deformation and bending; surface defects, such as pin-holes and oscillation marks; scum; cracks—internal, longitudinal and transverse; segregation of components; kinds and distribution of non-metallic inclusions; reduction of cross-sectional area; grain size; fatigue. The principal types of continuous casting machines in commercial use are described below.

In the vertical continuous casting machine, the casting is supported in a vertical position and the continuous length of casting is parted by gas-cutting in the vertical position. The cut-off piece is received by a tilting basket mechanism that lowers it to a horizontal position. Since the machine height is 17 m or more, this type of continuous caster requires a tall building or deep pit.

In the curved continuous casting machine, the solidified product, after cooling by water spray, is bent by a series of rollers from the vertical to the horizontal position. Cutting-off is accomplished on the horizontal casting. The machine height is thus reduced to about 10 m, and consequently to that of the building.

A curved mould is employed in the bow type of continuous casting machine, which is maintained in oscillation. The cooling chamber is also curved. The height of this machine is therefore reduced to one third, or less than, that of the vertical type of machine.

Pre-heated and insulated ladles and a tundish are used. The latter may have one or more nozzles that feed the metal to the mould, which is made of copper and cooled by copious amounts of water.

Growth pattern of continuous casting

The growth of continuous casting has been considerable during the last two decades. Vertical machines requiring buildings of great height, are now coupled with bending and straightening devices and curved-mould assembly. The speed of withdrawal has risen to 1.5-2.0 m/min for slabbing machines, and to 4 m/min for square sections. Thus output per strand is 200 tons/h for flats and 15-36 tons/h for square sections. Uninterrupted series of more than 200 heats have been possible with the use of improved distributor refractories, rapid change-over systems for closing mechanisms and proper synchronization of processing furnaces and casting machines.

Table 29 shows the trend in the growth of continuous casting.

It is estimated that by 1980 nearly one third of all steel production in the world will go through the continuous casting route. Over 50 per cent of the machines now in use produce billets, about 20 per cent are for blooms and the rest make slabs.

Continuous casting accounted for 30 per cent of the steel output in Japan in 1975, 20 per cent in the

TABLE 29. GROWTH OF CONTINUOUS CASTING

Year	World steel production (millions of tons)	World casting capacity (millions of tons)	Annual growth (per centage)	Percentage share of continuous casting in steel production
1955	266	0.38		0.1
1960	325	1.65	34	0.5
1970	598	57.40	42	9.6
1975	650	140.00	20	21.5

Source: United Nations Economic and Social Council, Economic Commission for Europe, "Structural changes in the iron and steel industry" (STEEL/GE.3/R.3/Add.1) p. 31.

Federal Republic of Germany, Italy and Spain, 15 per cent in France, and less than 10 per cent in the United States of America, and the USSR. There were 651 mills in 1976. It has been estimated that by the end of 1977 there will be 734 mills in 66 countries.

The utilization rate (ratio of output to production capacity) on a global basis has risen from 38.5 per cent in 1970 to 64 per cent in 1974.

Choice of process

There are a number of technical and economic factors which influence the decision to choose one of the three casting processes, namely conventional small-ingot, conventional large-ingot and continuous casting. The parameters influencing the choice of the process are as follows: production scale, type and quality of product, yield and balance of materials, investment and energy requirements, operating costs, manning, technical requirements in supporting sectors, and management and control requirements.

The conventional ingot-casting and primary-mill route involves considerable cost resources. With continuous casting the two independent functions, namely, casting and primary rolling, are combined into one simple process. The operations involved in the two processes are shown below.

STEPS IN CASTING PROCESSES

Ingot casting/primary mill

1. Furnace tapping
2. Ladle transfer to casting pit
3. Casting into moulds
4. Transfer of ingot moulds to stripper yard
5. Stripping
6. Transfer of ingots to pit or pusher furnace
7. Ingots placed into pit or pusher furnace
8. Heating the ingots
9. Transfer of ingots to primary mill
10. Primary rolling
11. Grinding of rolled products
12. Transfer to rolling mill

Continuous casting

1. Furnace tapping
2. Ladle transfer to casting platform
3. Continuous casting
4. Subdividing of cast strands
5. Transfer of cast material to rolling mill

The applicability of the three processes to different types of steel products, is shown in table 30.

The energy requirements for a blooming mill and continuous casting are 35-55 kWh/ton and 10-30 kWh/ton, respectively. The water requirement is more for a continuous casting installation, namely 12-18 m³/ton per hour, as against 4-7 m³/ton per hour for a blooming mill.

TABLE 30. APPLICABILITY OF CASTING PROCESSES TO DIFFERENT TYPES OF STEEL PRODUCT

Steel product	Small-ingot direct rolling	Large-ingot blooming	Continuous casting and breakdown
Reinforcing bar	1	1	1
General structural bar and section	1	1	1
Low-carbon wire rod	2	1	1
High-carbon wire rod	2	1	1
Cold-drawing quality steel	7	3	4
Cold-heading quality steel	8	5	4, 5
Mechanical structural carbon steel	6	5	4, 5
Low-alloy steel	8	5	7
High-alloy and stainless steel	8	3	7

Source: ID/WG.146/41, p. 7.

- Note: 1 Easily applicable
 2 Only applicable for low quality level
 3 Applicable, but surface conditioning required
 4 Applicable, but square bloom of over 150 mm recommended and surface conditioning required
 5 Applicable, but surface conditioning and guaranteed internal quality required
 6 Difficult
 7 Doubtful
 8 Impossible

However, continuous casting cannot be applied universally for the entire range of a flat-product mix. For all types of output, there are often local factors to be considered in making a choice. The most critical considerations are those of scale and volume, yield and capital cost, and steel quality. Continuous casting does not produce a satisfactory rimmed steel slab or bloom.

Cost data

Investment costs depend very largely on geographical conditions, layout of the plant, product range, tariffs, economic policies of the country, local market costs, freight, labour costs etc. These factors vary greatly from country to country.

The small-ingot process is labour-intensive, requiring more employees per ton of production and increasing markedly as the output rises. The large-ingot blooming process is a large-scale process, so that an increase in capacity utilization can effectively increase the productivity per employee. The continuous casting of square section products has a production limit per machine of 500 000-600 000 tons.

A comparative idea of operating costs for the three processes is given in table 31. However, it may be mentioned that the figures should be taken as indicative only. The current actual cost per ton will be largely influenced by inflationary trends. Since there will be variations in the local prices for copper moulds for continuous casting and for blooming-mill rolls, comparison of operating costs gives only an approximation.

The figures show that the operating costs of continuous casting are the lowest, the small-ingot process comes next, and the large-ingot blooming process is, comparatively, the most expensive.

Arthur D. Little Incorporated has estimated capital investment to be about \$US 800 000 for a 300 tons/day continuous casting plant, and about \$US 4 000 000 for a 4 000 tons/day plant.

A paper presented at the Third Interregional Symposium on Iron and Steel Industry, Brazil, 1973 (ID/WG.146/117) estimated investment cost per ton of installed capacity for a slab-casting machine of 6 000 casts per annum to be 30 per cent lower than for the conventional process.

Another paper presented at the above-mentioned Symposium (ID/WG.146/68) worked out, for different capacities, comparative capital-cost figures for continuous-casting and for the conventional process. The information is summarized in table 32.

TABLE 31. RELATIVE OPERATING COSTS FOR CASTING PROCESSES

Process	Items	Consumption	Cost (\$/ton)
Small ingot	Bricks and refractories	7.0-10.1 kg/t	} 4.2-5.8
	Moulds and plates	8.9-15.3 kg/t	
Large-ingot blooming	Bricks and refractories	7.2-15.0 kg/t	} 6.6-10.4
	Moulds and plates	8.4-16.6 kg/t	
	Heavy fuel oil	20-40 l/t	
	Rolls	0.4-0.7 kg/t	
	Electricity	35-55 kWh/t	
Continuous casting	Bricks and refractories	5.0-18.0 kg/t	} 3.0-4.8
	Moulds	70-500 heats/mould	

Source: ID/WG.146/41, p. 13.

TABLE 32. COMPARATIVE CAPITAL COSTS

Plant capacity (millions of tons per year)	Conventional pouring process		Continuous casting process	
	Capital cost (%)	Yield factor (%) ^a	Capital cost (%)	Yield factor (%) ^a
0.8	100	83	45-55	97
1.2	100	87	65-75	96
1.5	100	85 ^b	80-90	95

^a Approximate.^b Killed steel.

In a survey made by the Organisation for Economic Co-operation and Development on continuous casting of steel in the USSR, it was stated that the cost of converting liquid steel into cast slabs, is about 25 per cent less than converting it into rolled slabs, and the casting process costs could be reduced further if the productivity of the machine could be increased by introducing new steelmaking equipment. At a plant at Lipetsk, two electric furnaces with 25 MVA transformers currently produce steel which is continuously cast in two twin-strand machines. The steels generally cast are transformer and dynamo steels, rimming carbon steels, killed carbon and semi-killed carbon steels in various slab sizes with overall section dimensions ranging from 24³/₈ in. x 5⁷/₈ in. to 40¹/₈ in. x 6³/₄ in. Typical casting speeds quoted are in the range 32-36 in./min for the

large slabs. The two machines, when operating with ladles of 90- to 95-ton capacity, have produced up to 800 tons in 8-9 casts/day.

The consumption of liquid metal per ton of slabs of transformer steel has been found to be respectively 26.3 and 24.3 per cent lower than at Kuznetsk and at Dneprospetsstal and Zaporozhstal taken together. At Kuznetsk, the processing costs incurred in the arc-furnace shops, on the roughing mills, and on deoxidizers and addition materials, amount to 20.1 roubles per ton. At Dneprospetsstal and Zaporozhstal, they amount to R 34.0 per ton. Hence, the savings from continuously casting the slabs are estimated at R 7.7 and R 8.3 per ton respectively. The difference in costs for continuously casting the metal, plus scarfing the cast slabs, as against casting ingot and rolling on a slabbing mill, is given as an average of R 0.8 per ton of cast slabs, so that the real saving in these comparisons is placed at R 8.5 and R 9.1 per ton. This cost saving is expected, however, to increase still further with an increase in the production of cast slabs.

The economic advantage of changing over to continuous casting, while largely due to the increased yield of sound metal or reduction in metal waste, also includes a reduction in the extra capacity of steelmaking units and roughing mills which would otherwise be required to meet the planned rates of expansion. There would, therefore, be corresponding savings on these operations as well as reduced consumption of deoxidizers and flux materials on the liquid metal that is saved.

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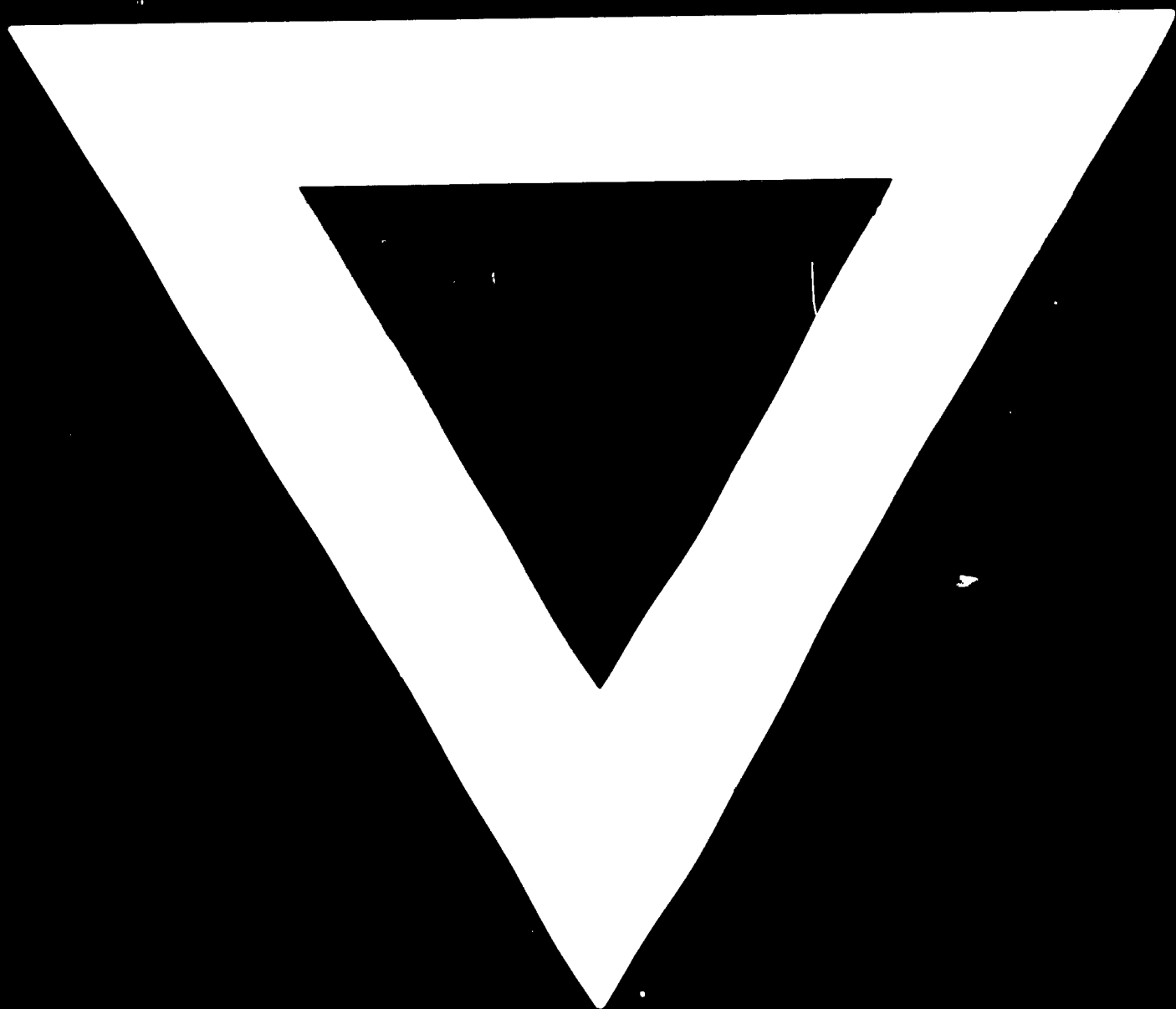
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