Presidential Address:
Oxygen-coal in-bath smelting reduction—a future process
for the production of iron and stainless steel?

by J.P. Hoffman*

SYNOPSIS
In 1988 the Japan Iron and Steel Federation started a research-and-development programme known as DIOS (direct iron ore smelting). Eight Japanese integrated steel producers and the Coal Mining Research Centre, subsidized by the Ministry of International Trade and Industry, are involved in the project. The aim of the programme is to have a 500 t/d pilot plant for the production of iron via an oxygen-coal in-bath smelting converter in operation by mid 1993. The three main components of the DIOS process are as follows:

- a fluidized bed for the pre-reduction of iron ore fines
- in-bath smelting reduction in a converter
- off-gas reforming by the addition of coal fines.

The main drive is to provide a smelting process that is more energy-effective, using coal instead of coke and/or electricity. The advantages claimed include inter alia

- direct use of fine iron ore
- direct use of non-coking coals—low coal consumption
- low capital expenditure—replaces costly coke ovens, agglomeration plants, and the blast furnace
- economical production on mini-plant scale
- less of an impact on the environment.

This concept of 'direct steel' has been carried over to the production of stainless steel direct from the chromite. Several interacting elements are recognized that could have a productivity-limiting aspect. The possibility of the direct production of stainless steel from chrome is discussed briefly with reference to the Highveld of the eastern Transvaal as a possible site for such a plant.

SAMEVATTING
In 1988 het die Japanse Yster en Staal Federasie met 'n nuwe navorsingsprogram begin: DIOS (Direct Iron Ore Smelting reduction process). Agt geïntegreerde Japanse staal produsente en die Steenkool Mynavorsingssentrum, deur die Ministerie van Internasionale Handel en Nywerheid (MITI) gesubsidieer, is hierby betrokke. Die doel van die programme is om 'n middel 1993 'n 500 t/d kooienhawe in die produksie van yster via 'n in-bad smelter omsetter waarin steenkool en suurstof as energiebron gebruik word, in bedryf te hê. Die drie hoofkomponente van die DIOS proses is

- 'n fluid bed vir die voorreduksie van fyn ystererts
- in-bad smeltrieksie in 'n omsetter
- hervorming van afgas deur die inspuit van fyn steenkool.

Die hoofdoel is om 'n smeltriese te ontwikkels wat meer energie effektief is en waarby steenkool in plaas van kook- en/of elektriese energie gebruik word. Die voordele wat gesien word sluit onder andere in

- direkte gebruik van fyn ystererts
- direkte gebruik van nie-kooksteenkool en lae steenkool verbruik
- lae kapitaal investering—vervanging van suurstof en kookstof
ekonomiese produksie op mini-plant skaal
- kleiner impak op die omgewing.

Die konseps van 'direkte staal' was oorgedra na vlektyraal produksie direk van die ertse, weereens deur die vervanging van energie intense en energie kosbaar die dorpelbooggas wat vir die produksie van hoë koolstof ferroooy genominering word. 'n Aantal Japanse maatskappye het reeds 'n sekere mate van sukses bereik.

'N Aantal elemente wat op mekaar interaksie het en wat die produktiwiteit kan beperk, word in hierdie aanbieding ge-analiseer. Die moontlikheid om vlektyraal direk van chromiet af te produseer, word kortlik bespreek met verwysing na die oostelike Transvaalse Hoëveld as 'n moontlike plek vir so 'n aanleg.

INTRODUCTION

It is well known that South Africa has extensive reserves of high-quality hematite iron ore at Sishen and Thabazimbi, and banded-magnetite ores in the northern Transvaal, that can be upgraded with relative ease. It is also a known fact that South Africa has the world’s largest chromite reserves. Most of these chromites are available in a friable or fines form, and these could be exploited in the new smelting-reduction processes. These ores, which are relatively cheap, are concentrated in the Bushveld Complex close to the Highveld.

Fluxes, such as fluor spar, quartz, serpentine, and dolomite, are abundant and in close proximity. All these factors are indicators that South Africa, and the Highveld in particular, is well located for the new smelting-reduction technologies that are now developing in countries such as Japan, Australia, and the USA.

Non-coking coals are estimated to represent 75 per cent of the world’s coal reserves. South Africa is not well endowed in coking coals; nor, for that matter, in good metallurgical coals. However, coals with acceptable properties in terms of their ash and fixed-carbon contents, and their relatively low phosphorus contents, are available in quantity and, in the range of steam coals, the availability is enormous. Compared with other parts of the world,
South Africa has coals that are cheap. Again, the coal occurs on the doorstep of the steel giants in the Highveld.

To use non-coking coal reserves more efficiently, the steel industry has promoted the injection of non-coking coals into the blast furnace, using oxygen to enrich the blast and produce a gas of higher calorific value.

The conventional blast-furnace process is in many respects still regarded as very efficient, and is by far the most important ironmaking process because of its high productivity. However, it requires high-quality coking coal, coke-oven batteries, high-grade iron ores in terms of iron content, and sinter plants to agglomerate the fines available from these sources.

The improvement of the blast-furnace process by the use of oxygen instead of air, which allows a high level of substitution of coke by injected coal, will, besides achieving a significant decrease in coke consumption, increase the productivity and produce a gas of high calorific value for distribution in an integrated steelworks, thus making the blast furnace also more energy-efficient. Edstrom described the full oxygen blast furnace (FOBF) and balance oxygen blast furnace (BOBF) in a recent paper, and compares the economics of these concepts with that of the emerging smelting-reduction processes.

A constraint of the blast furnace is its need for agglomerated ores and coke. The agglomeration of ore and the coking of coal are energy-consuming and expensive (representing 20 to 25 per cent of the operating costs of ironmaking). They also account for more than half of the investment cost for ironmaking based on the blast furnace.

Most of the coke-oven batteries now in use in the world were built in the period 1960 to 1970. If the service life of a coke-oven battery is assumed to be about 35 years, all of these batteries will have to be replaced (or shut down) between the years 1995 and 2010. Also, these older batteries are causing concern because of their polluting emissions and the harm to the environment. To replace a battery of 100 ovens would cost about 200 million dollars. Thus, the economic incentives to develop alternative processes that would avoid the need for both agglomeration and coking are evident.

The above, including the high cost of energy, particularly electrical power, in countries such as Japan and Korea, has led to the development of the smelting-reduction process. In smelting-reduction, the objectives are to use iron-ore fines (preferably) with coal as the reductant and source of energy. Smelting-reduction should be able to completely avoid the costly and environmentally objectionable agglomeration and coking processes. As the ironmaking step consumes approximately 65 per cent of the energy required by an integrated steelworks, the viability of new ironmaking processes will, to a large extent, focus on the consumption and price of energy and the impact on the environment.

In the late sixties, Klöckner at Maxhütte (Germany) developed the OBM process, a bottom-blown oxygen steelmaking process with oxygen and lime injected through submerged tuyères. The K-OBM process combined the metallurgical advantages of the OBM process (with additional energy input into the bath through the post-combustion of converter off-gas rich in carbon monoxide) with the top-blowing of oxygen. The proven ability of solids (lime) injection into the OBM and K-OBM converter led to the development of the K-MS process. Powdered coal or coal fines are injected through submerged tuyères in the converter bottom. The carbon from the injected coal goes into solution, the ash and sulphur are removed in the slag, and the volatile matter is cracked into its carbon and hydrogen components. Oxygen is injected into the bath to continuously oxidize the carbon in solution to carbon monoxide, thereby generating heat in the bath. Secondary oxygen injected through a top lance (or side tuyères) combuts some of the carbon monoxide and hydrogen to carbon dioxide and water respectively in the converter freeboard. The secondary energy released is transferred back into the bath.

If pure amorphous carbon is used and combusted with oxygen to carbon monoxide in an iron bath, 1,4 Gcal per tonne of carbon are available to the iron bath. If a highly volatile coal is used (containing, say, 36 per cent volatile matter), the amount of energy originating from the combustion to carbon monoxide is reduced by an amount necessary to heat up and crack the volatile matter, and almost no excess energy (0,1 Gcal) is transferred to the bath. However, post-combustion to 20 per cent increases the former to 2.4 Gcal, and the latter to 1.3 Gcal (Table I).

The coal injection makes the K-MS process an allothermic basic oxygen furnace process. The additional available energy can be used to melt scrap, direct-reduction iron, or solid pig iron.

The post-combustion of residual carbon monoxide and hydrogen has been exploited in a joint venture between Klöckner and CRA for the smelting-reduction of iron ores. CRA (Australia) had taken the concept of bath smelting utilizing post-combustion and from it developed the HI-Smelt process.

Shibuya estimates that, after the year 2003, the demand for coke in Japan will exceed the supply and, in the year 2020, the shortage will be 210 Mt unless coke-oven batteries are replaced (Figure 1). He suggests four measures:

- increase the percentage of crude steel produced by the smelting-reduction process
- inject more than 200 kg per thermal metre of coal into the blast furnace
- renew existing coke-oven batteries
- adopt new technologies such as the use of formed coke.

One of the most important tasks in the near future will be to find the best combination of measures to provide

<table>
<thead>
<tr>
<th>Table I</th>
<th>Net energy (Gcal/t) available to the bath from carbonaceous reductants at various post-combustion levels</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>Reductant type</strong></td>
<td><strong>Amorphous C</strong></td>
</tr>
<tr>
<td>Combustion to CO at 1500°C</td>
<td>1.4</td>
</tr>
<tr>
<td>20% post-combustion (CO2+H2O = 20%)</td>
<td>2.4</td>
</tr>
<tr>
<td>40% post-combustion</td>
<td>3.2</td>
</tr>
</tbody>
</table>

* VM = Volatile matter
Prerequisites

- Steel production: \(100 \times 10^4\) t/y
- EAF: \(50 \times 10^4\) t/y
- Converter: \(70 \times 10^4\) t/y
- BF (15 or 30 t)
- Fuel rate of BF
  - Coke rate (kg/t BF): 408
  - Coal-injection rate (kg/t BF): 100
- Coke-oven battery
  - Life: 25 yrs
  - Working ratio: '88 actual results

Figure 1—Changes in the production capacity of coke without the rebuilding of coke-oven batteries, and the demand for coke with production sharing by smelting-reduction

an optimum solution to the problems posed by the coke shortage.

The steelmaker will meet increasing ecological constraints concerning emissions such as \(SO_2\), \(NO_x\), and \(CO_2\), as well as dust recycling.

The Clean Air Act Amendments of 1990 (Title III Section 112—Requirements for Coke Ovens) may have a large impact on the steel industry. This title of the 1990 amendments extensively restructures the hazardous air-pollutant programme. Important compliance dates for the EPA and coke industries are listed, starting in December 1992. By 1st January, 2020, all coke-oven batteries must comply with health-based standards—the lowest achievable emission rates (LAER), e.g. 16 seconds of emissions per charge.

The springboard for the development of the smelting-reduction processes was provided by the current state of coal-injection technology as developed by Klöckner; the control of post-combustion; the efficient transfer of energy; and the collection, cleaning, and use of converter off-gas due to pressure from the environmentalists.

**SMELTING-REDUCTION OF IRON ORE**

Smelting-reduction can be defined as the production of liquid iron direct from ores through a process other than that using a blast furnace, with coal and oxygen as sources of energy. It involves the use of primary energy, instead of electrical energy, with the simultaneous gasification of coal. A combination of final reduction and melting in or directly above the iron bath is required, the gases available after a specific degree of post-combustion being used for pre-reduction. It is clearly a two-stage process, consisting of a pre-reducer and a melter-gasifier.

Two distinct processes are identified:
1. Shaft-smelting reduction processes, e.g. Corex, Kawasaki STAR, XR, and
2. Oxygen-coal converter in-bath smelting reduction, e.g. NKK, NCS, CRA—MIDREX.

This Address will discuss the development of the in-bath smelting-reduction process and problems associated with it.

**Smelting-Reduction Processes**

In Japan, the development of direct iron-ore smelting (DIOS) started in 1988 as a joint research project among eight iron-manufacturing companies who had, prior to 1988, been studying the smelting-reduction process individually. This project was sponsored by MITI, the Japanese Ministry of International Trade and Industry. The DIOS pilot plant is scheduled to start operating in July 1993 at NKK's Keihin Works (500 t per day). The concept of the DIOS process is shown in Figure 2.

In 1981, CRA (an Australian resource company) formed a joint venture with Klöckner to develop an iron-bath direct-smelting process based on the highly successful K-MS process. The HI-Smelt2 (high-intensity smelting process) was developed from this joint venture. One of the principal aims of CRA has been to develop a process based on fine ore and coal. At present, the process is at a pilot stage (100 kt per annum), developed by CRA—MIDREX.
For the smelting-reduction in-bath smelter or reactor to be successful, the optimum must be found between the basic six elements of smelting-reduction and the plural interactions between these elements. These elements are as follows:

1. Pre-reduction degree (PRD)
2. Post-combustion ratio (PCR)
3. Reaction kinetics or production rates (RK)
4. Heat-transfer efficiency or energy efficiency (HTE)
5. Cost of production (COP)

Figure 3 shows schematically the interaction of the different elements, each element being seen as the side of a cube.

In-bath smelting-reduction processes consist of two or more separate steps. In the first step, the ore is pre-heated and/or pre-reduced with process gas, and smelting takes place in the second unit. Schematic diagrams of the two systems considered by the Japanese are shown in Figure 4. The pre-reduction reactor could be a shaft or rotary kiln if lumpy ore is used, or a fluidized bed (recycling or refluxing type) if fines are used. To prevent sticking in the case of iron ore, the PRD is limited to 33 per cent (FeO or wustite). It is immediately apparent that, if the PRD is increased from 15 per cent, to 60 per cent, as in Figure 4(b), a second pre-reducing reactor is needed, with a concomitant increase in capital costs. It is equally apparent that, in order to increase the PRD, the PCR has to be decreased so as to leave more carbon monoxide and hydrogen for reduction, it being borne in mind that PCR is defined as the partial oxidation of process gas and mathematically as

$$PCR = \frac{H_2O\% + CO_2\%}{H_2\% + H_2O\% + CO\% + CO_2\%}.$$  \[1\]

According to Fruehan,

$$HTE = \frac{\text{Heat transferred to bath}}{\text{Heat generated by post-combustion}}.$$  \[2\]

**Energy Requirements**

Another objective to consider would be to choose the PCR and PRD in such a way that the consumption of coal is minimized, and that the amount of gas produced is small enough for the scrubbers and gas-cleaning equipment to handle and for reforming after reduction. Fruehan compared four possible combinations:

1. 90 per cent PRD and 0 per cent PCR—conditions found with the Corex process
2. 60 per cent PRD and 40 per cent PCR—countercurrent two-stage shaft-reducing kiln with some gas reforming, Figure 4(a)
3. 30 per cent PRD and 50 per cent PCR—in which case the iron ore is reduced to FeO
4. 0 per cent PRD and 50 per cent PCR—no pre-reduction or pre-heating.

For the simplest case (IV), the initial capital cost is low. However, the coal consumption is high. For a high HTE of, say, 85 per cent, the gas volumes are high and the off-gases are at temperatures of 1880°C.

![Figure 4](image-url)

**Figure 4**—Schematic diagrams of two configurations being considered by the Japanese for their smelting-reduction process (a) Low post-combustion and high pre-reduction ratio system (b) High post-combustion and low pre-reduction ratio system.

**Legend**

- PRR Pre-reduction reactor
- SRF Smelting-reduction furnace
- WB Waste-heat boiler
- PCR Post-combustion ratio
- PRD Pre-reduction degree
Converter off-gas systems can handle gases with temperatures up to about 1760°C. To regulate the off-gas temperatures down to that level, mixtures of oxygen and air may have to be used (62 per cent oxygen and 38 per cent air), and gas volumes would be of the order of 1690 Nm³ per tonne of hot metal (thm). Figure 5 clearly illustrates the interdependence of off-gas temperature and coal consumption as a function of HTE and PCR.

An obvious improvement on Option IV is the utilization of smelter off-gases for pre-reduction and/or pre-heating, thereby reducing the energy requirements in the smelter to those for final reduction and melting, thus efficiently employing part of the chemical energy in the off-gas within the process. Cases II and III represent two such approaches. The other extreme is a process in which no post-combustion is utilized and the chemical energy in the gas is used to pre-reduce the ore to a high degree of reduction, e.g. Corex (Option I). The gas leaving the reduction shaft contains excess energy that is exploited elsewhere in an integrated steelworks.

Coal consumption as a function of PCR and type of coal is summarized in Figure 6. Three hypothetical coals are compared with a reference coal—here, Eikeboom coal with an ash content of 10.3 per cent. Figures 5 and 6 show that, for Eikeboom coal, the off-gas temperature from the converter could reach 2300°C at 50 per cent PCR and 70 per cent HTE. These temperatures are too high and constitute a danger for the converter and off-gas systems.

If the HTE is increased from 70 to 85 per cent, the coal consumption will drop by about 150 kg per tonne of hot metal (thm) at 40 per cent PCR; and the off-gas temperature to 1830°C.

Maeda et al.6 investigated the plural interaction of energy utilization, PCR, and PRD for the smelting-reduction of an iron ore via a molten-iron bath. It was recognized that, for the smelting-reduction process to be successful, energy utilization would have to increase.

During the smelting-reduction of iron ore, the following reaction takes place5:

\[
\text{Fe}_2\text{O}_3 + 5\text{C} + \text{O}_2 \rightarrow 2\text{Fe}(l) + 5\text{CO}
\]

\[1,43\text{ kg} + 1,293\text{ kg} + 0,905\text{ Nm}^3 \rightarrow 1\text{ kg} + 2,413\text{ Nm}^3\]

(25°C) (25°C) (25°C) \rightarrow (1450°C) (1450°C)

The amount of heat generated by the combustion of 1,293 kg of carbon is 10,470 kcal (1 kcal = 4,1868 kJ). The quantity of heat effectively utilized for the reduction of the ore and melting of the iron is 1,998 kcal.

The efficiency of the process \(= \frac{1,988}{10,470} = 0.19\) or 19 per cent. If heat losses are disregarded, the remainder of the energy available is discharged with the off-gas. It is thus clear that, to increase the efficiency, it would be necessary to utilize the energy in the off-gas.

One way to utilize the potential energy in the off-gas is to post-combust some of the carbon monoxide in the free-board above the bath in the converter. If a PCR of 30 per cent is chosen (30 per cent of the carbon monoxide combusted to carbon dioxide) and the off-gas temperature is 1600°C, the energy utilization increases to 36 per cent. This is still not sufficient if compared with the energy utilization of coal-fired power stations at 35 per cent. However, an increase of the PCR will give off-gas temperatures that are too high, as previously shown, with concomitant detrimental effects on the refractories and equipment. Pre-reduction of the ore would be an additional way to utilize more of the potential chemical energy.

If the pre-reduction is done at 800°C (to prevent sticking) and the Fe₂O₃ is reduced to iron, the PCR must be maintained at more than 65 per cent (Figure 7).

If a PRD of 75 per cent and a PCR of 35 per cent are chosen, the following equations apply:

**Coal Analyses**

**COAL ANALYSES**

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<tr>
<th>COAL</th>
<th>F.C.</th>
<th>VOL.</th>
<th>MOIST.</th>
<th>ASH</th>
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<th>F.C.</th>
<th>VOL.</th>
<th>MOIST.</th>
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<td>30.4%</td>
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Figure 5—Influence of post-combustion heat transfer (HT) and degree of post-combustion on coal consumption and secondary off-gas temperature

Figure 6—Influence of coal type and degree of post-combustion on coal consumption

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For pre-reduction:

\[
\begin{align*}
\text{Fe}_2\text{O}_3 + 3\text{CO} & \rightarrow \text{Fe(s)} + \text{FeO(s)} + \text{CO} \\
& + 2\text{CO}_2 \\
1,43 \text{kg} + 1,29 \text{Nm}^3 & \rightarrow 0,625 \text{ kg} + 0,482 \text{kg} + 0,838 \text{Nm}^3 \\
& + 0,451 \text{Nm}^3
\end{align*}
\]

For the equilibrium diagram temperature and post-combustion ratio for iron and iron oxide

![Equilibrium diagram](Image)

For the smelting reduction converter:

\[
\begin{align*}
\text{FeO(s)} + \text{Fe(s)} + 3\text{C} + \text{O}_2 + \text{CO}_2 & \rightarrow 2\text{FeO}_2 + \text{CO} \\
0,48 \text{kg} + 0,625 \text{ kg} + 0,631 \text{kg} + 0,458 \text{Nm}^3 + 0,112 \text{Nm}^3 & \rightarrow 1 \text{ kg} + 1,29 \text{Nm}^3 \\
(800°C)(800°C) & \rightarrow (1450°C)(1450°C)
\end{align*}
\]

The energy utilization now calculated becomes 39 percent. However, for a PRD of 70 percent, it will be necessary to control the PCR at 30 percent to ensure sufficient carbon monoxide in the off-gas for pre-reduction.

Because of this relatively low utilization of energy, the rate of carbon (coal) consumption is high, with a concomitant high consumption of oxygen. In actual commercial practice, as the consumption of coal increases, both the quantity of slag and the metal loss increase. In addition, the cost of equipment to cope with the increased gas volumes and slag handling increases.

Maeda et al. show that, at a PCR of 50 percent and a PRD of 30 percent, coal consumption becomes a minimum (point D on Figure 8). Minima for coal consumption are predicted for PCRs of 50, 40, 30, and 20 at a PRD of 33 percent—the reduction of \( \text{Fe}_2\text{O}_3 \) to \( \text{FeO} \). From the same graph it is clear that the coal consumption increases alarmingly at a PRD of more than 33 percent (e.g., BC for 20 percent PCR). At point D, where the coal consumption is lowest, the consumption of oxygen and lime would also be low. For low direct costs, a PCR of 50 percent and a PRD of 33 percent should be chosen.

To further increase the energy utilization, off-gas is passed through a waste-heat boiler for the generation of steam, and the surplus gas after pre-reduction is distributed as fuel for re-heating purposes (chemical energy 780 kcal/Nm³). The overall energy efficiency could be increased to over 40 percent.

HTE is related to various practical problems. Tokuda et al. recommend a soft blow with a large distance between the top oxygen lance and the melt surface to give a high PCR and a high heat transfer. They also suggest that post-combustion is a function of \( L/d \), where \( L \) is the distance of the lance tip from the bath surface, and \( d \) is the diameter of the lance nozzle.

**Foaming Slag**

The transfer of heat could be increased by a layer of foaming slag. The virtues of a foamy slag were first appreciated by operators melting direct-reduced iron, (DRI). The development of carbon- and oxygen-injection techniques for slag foaming under ordinary melting conditions is the most significant advance in steelmaking by electric-arc furnace (EAF) since the advent of ultra-high power (UHP). The first observed result of slag foaming was an improvement in the heat-transfer efficiency, which resulted in a reduction in energy consumption and a faster temperature pickup than under flat bath conditions.

In smelting-reduction, a thick layer of slag containing coke in suspension both serves as a shield to prevent oxidation of the metal and supplies effective reduction sites. The coke also carries heat from the slag surface, which is the contact with the post-combustion freeboard to the reaction sites in the slag and metal. Moreover, coke suppresses excess foaming, which leads to stopping and thus limits the production rates in a smelting-reduction converter (Figures 9 and 10).

Since, during the in-bath melting processes, coal and/or iron ore may be injected into an iron bath (some ore may be injected into the slag), large volumes of carbon monoxide and hydrogen gases are evolved from the devolatilization and cracking of the coal volatiles, and carbon monoxide and water arise from the reduction reactions. When a high production rate is maintained, violent foaming is almost inevitable. Foaming and its control are considered to be critical for all in-bath or converter smelting-reduction processes.

Ito and Fruehan studied the phenomenon of slag foaming relevant to smelting-reduction processes for the smelting of iron ore. Figure 11 shows the relationship between the foaming index (Σ) and the basicity of the slag at 1573 K and 1673 K. The foaming index is defined as the retention time of gas in the slag and the foam life (τ). Foaming increases with increasing slag viscosity and...
Although, for the smelting-reduction of iron ore, foam heights of 5 m are possible, a high degree of pre-reduction will reduce foaming and thus increase the production rates. Figure 14 shows that the production rate is determined by the foam height, and can be controlled by the PRD of the iron oxide: Case 1 (30 percent PRD), Case 2 (60 percent PRD), Case 3 (90 percent PRD).

![Figure 9 — Effect of coke-to-slag ratio on slag foaming](image)

Figure 9 — Effect of coke-to-slag ratio on slag foaming

![Figure 10 — Effect of powder and lump-coke addition on slag foaming](image)

Figure 10 — Effect of powder and lump-coke addition on slag foaming

decreasing surface tension (Figures 12 and 13). Ito and Freuhan's model predicts slag-foam heights as high as 3 to 5 m for a typical operation, and this may constitute a serious problem. This value was also reported by Janke and Steffen through extrapolation of work done in a laboratory and by mathematical modelling of CaO-SiO₂-FeO slag systems with MgO or CaF₂ added. Foaming is also a function of temperature, production rate, and FeO content of slag. The production rate, in turn, is a function of the gas velocity on the bath surface or the gas flowrates ($Q_s$ in m³/s):

Foaming index ($\Sigma$) = $\frac{\Delta h}{V_{gs}^*}$

where $V_{gs}^*$ = $\frac{Q_s}{A}$

$\Delta h$ = height of slag or foam

$V_{gs}^*$ = surface velocity of gas

$Q_s$ = gas flowrate

$A$ = area of vessel at the slag-metal interface.

Also, 

$\tau = \frac{-t}{\ln(h/h_0)}$, 

where $t$ is the time required to reduce the foam height from $h$ to $h_0$. 

![Figure 11 — The relation between foaming index ($\Sigma$) and the basicity ratio of the slag at 1573 and 1673 K](image)

Figure 11 — The relation between foaming index ($\Sigma$) and the basicity ratio of the slag at 1573 and 1673 K

![Figure 12 — Dependence of foam stability on surface tension](image)

Figure 12 — Dependence of foam stability on surface tension

![Figure 13 — Dependence of slag height on surface tension](image)

Figure 13 — Dependence of slag height on surface tension
Also, \( h = C \cdot \sum a_{\text{FeO}} \) \[9\]
where \( a_{\text{FeO}} \) is the activity of the FeO in the slag, and \( C \) is a constant that takes into account the reaction surface and the vessel geometry.

Von Bogdandy et al.\(^{11}\) designed a slag for the smelting of reduced iron that would not foam. This slag is very high in FeO (25 to 50 per cent) and would be considered aggressive towards conventional refractories. In contrast, Takahasi et al.\(^{12}\) controlled slag foaming by controlling the Fe in the slag to between 2 and 7 per cent. The feed to their experimental furnace was iron-ore fines (-5 mm) pre-reduced in a bubbling type of fluidized-bed reactor (at 700°C).

Takahasi and his co-workers\(^{12}\) found that slag foaming increased as the charging rate of the iron ore was increased. Takumitsu et al.\(^{13}\) argued that the reduction rate of iron ore is equal to the feed rate, and that the reduction rate can be assessed from the content of FeO in the slag. Slag foaming would thus be a factor limiting the productivity of smelting-reduction furnaces\(^{9}\) (Figure 14).

**Heat Transfer**

Ibaraki et al.\(^{14}\) report that the scaling up of testwork from a 5 t experimental furnace to a 170 t modified basic oxygen furnace led to production rates of 41 t of hot metal per hour. The feed was lumpy coke and iron ore to the slag. Oxygen was blown in via the top lance, and some 8 per cent via bottom tuyères into the melt. The operating temperatures were 1500°C. They found that the transfer of heat could be controlled on a commercial scale and that a large volume of slag increased the kinetic reactions.

Figure 15 shows the HTE as a function of the PCR and the type of reductant (coke and coal with volatile matter of 22 per cent and 30 per cent respectively). It is evident that the PCR increases with lower amounts of volatile matter.

**Gasification**

Smith and Corbett\(^{15}\) describe the coal–iron gasification process, which was the subject of a Japanese–Swedish feasibility study conducted in 1982–1983 as one of the International Energy Agency's co-operative projects. On completion of the studies, it was decided not to pursue the ironmaking process any further. Areas where technical improvements were thought to be required are refractories, post-combustion, cheaper methods of oxygen production, and the removal of carbon dioxide.

In the CGS–STB (coal–gas–smelter Sumitomo top- and bottom-blowing) process, hot metal is produced as a byproduct of gasification\(^{16}\). Gasification is done in the molten-iron bath in a converter fitted with a top oxygen lance and bottom tuyères. When use was made of an Australian non-coking bituminous coal (fixed carbon 55.4 per cent, volatile matter 34.4 per cent, ash 8 per cent), gas containing 66 per cent carbon monoxide, 25 per cent hydrogen, and 5 per cent carbon dioxide with a calorific value of 11 050 kJ/Nm\(^3\) could be generated at a rate of 2000 Nm\(^3\) per tonne of coal. The carbon conversion was more than 98 per cent, and the thermal efficiency was 75 per cent.

If the CGS–STB process is used jointly with steelmaking, it can produce steel or stainless steel\(^{16}\). An alloy with a nickel base of 12 per cent was produced by the smelting of NiO during the gasification of char fed into a bath of dephosphorized hot metal. After deslagging, the base alloy was decarburized (in the same converter) by the STB process. Suitable amounts of charge chrome and lime were added during the refining period. After decarburizing to 0.05 per cent carbon, ferrosilicon and lime were added to recover chromium from the slag and to desulphurize the metal. The product was a 304 type stainless steel. Similarly, by the reduction of chromite in the bath during gasification, an alloy with a 12 per cent chromium base was produced. After further processing, type 410 or 409 steel could be processed. It is equally possible to use the converter gas for the production of direct-reduced iron.

**Refractory Wear**

One of the problems encountered in the new smelting–reduction technologies, especially in the in-bath or converter approach, is attack on the refractories by the slags, which are rich in iron oxides.
Since the conditions in the Corex furnace are no more severe than those in a blast furnace, the hearth refractories are similar to those in a blast-furnace hearth. In the oxygen tuyère area and above, silicon carbide brick is used. The dome is lined with high-alumina refractories (dome temperatures are typically 1050°C), and the shell is spray-cooled with water.

One cannot assume that the refractories now available will necessarily be compatible with the new in-bath smelting-reduction technologies. In the case of the Direct Steel Programme of the AISI (DOE-AISI), the process is designed to allow for the limitations of the available refractories. Process parameters include high temperatures and high concentrations of FeO in the slag, and rapid refractory attack can be expected. Rapid refractory erosion brought the US Direct Steel Programme to an untimely end17.

Parallel to the development of the AISI direct-steel process was research at the University of Missouri at Rolla; sponsored by 13 US steel and refractory companies, this research is proceeding on the development of a more compatible refractory.

Magnesite-carbon refractories were pioneered by North American Refractories Company (NDR Co.), and these were selected for the initial stages of the AISI programme, although the erosion rate is approximately 1 mm per hour. It is also a certainty that water cooling will be prominent in future processes.

Information on the refractory and tuyère wear of the new smelting-reduction processes is very sparse. In Steel Times International18 it was stated that the durability of bottom-blowing tuyères has been improved by ‘improved refractory materials’ in the bottom of the vessel, the placing of the tuyères, and a slag coating practice for a 180 t LD–KGC vessel at Mizushima Works. Tuyère life has increased to over 6000 charges or heats, even at flowrates of up to 0.2 Nm³/min per tonne of steel.

However, for the smelting of chromium ore, the tuyère and tube-pipe transportation systems posed a real problem owing to the high wear caused by the abrasive materials being transported. This problem was ‘solved’ by blowing down of the chromite with the top lance (similar to flash smelting).

Kolesnikov et al.19 investigated the factors affecting the life of the bottom tuyères. Besides the temperature and the type of refractory used in the bottom, the tuyère design itself was found to be of major concern. They found that the size of the annular gap of the peripheral channel of the bottom tuyère should be in the range 0.6 to 3.5 mm. With a gap of less than 0.6 mm, the gas stream is affected by the roughness of the tube wall and, when the gap is more than 3.5 mm, metal breakthroughs through the tuyères became more likely. Blowing with nitrogen, which has a cooling power 1.5 times that of argon, improved the tuyère life, and the nozzles also remained clear. The authors also found a good correlation for tuyère wear between millimetres of wear per heat and converter capacity in tonnes (e.g. a 250 t converter has a wear of 11 mm per heat; a 50 t converter has a wear of approximately 2.5 mm per heat).

Klöckner, who are considered to be world leaders in the field, have designed a special tuyère to handle abrasive materials at high injection rates, together with transportation and distribution systems (including pressure vessels) to these tuyères20. The basic technology for coal injection was developed by Klöckner for the K–MS process, and has been used routinely for many years. Considerable attention has been given to the accurate and reliable measurement of the flow of solids.

Ironmaking

It is evident that if, in an oxygen–coal converter, the carbon is replenished by the continuous addition of coal, and the exothermic combustion of the carbon is balanced by the endothermic reduction reactions, the converter produces iron with a certain carbon content. Such a converter is regarded as a much more ‘natural reactor’ for ironmaking than the shaft furnace because of the higher reaction rates due to higher temperatures and the strong stirring. With an iron bath of 10 t and 3 per cent carbon, the carbon content is renewed approximately every 8 minutes20 at a coal-injection rate of 3 t/h.

The Australian CRA–Midrex HI-Smelt process can be regarded as the most advanced in-bath or iron-bath smelting–reduction process. The evolution of this process is considered unique. The non-equilibrium nature of the chemical and thermal processes occurring in the smelt-reduction vessel separates this technology from the Corex, XR, and BSC/Hoogovens processes21.

Whereas, in the DOIS and DOE–AISI processes, deep slag baths are utilized with preferably submerged post-combustion techniques to achieve high PCR and HTE, the HI-Smelt process evolved from a fundamentally different approach. By use of the Klöckner bottom-injection of coal and a horizontal vessel, substantial quantities of metal droplets are splashed up into the freeboard space of the vessel, resulting in an ideal environment for high HTE and mass transfer.

The iron bath is considered to be an intensive reaction medium for the rapid final reduction of pre-reduced ore and the dissolution of carbon from the coal. The bath is also considered to be the ideal thermal and chemical buffer for the process. The bottom injection of coal has several advantages. The cracking of the volatile matter enhances the bath turbulence and facilitates the maximum recovery of carbon, the transfer of energy, the rapid reduction of ore, and adequate mixing of slag with high-carbon metal to ensure low levels of FeO in the slag.

In the HI-Smelt process, hot blast air is used instead of oxygen. The inert nitrogen provides a mechanism for the transfer of sensible heat, and thus limits the temperature of the post-combustion flame, leading to lower off-gas temperatures, and thus less refractory wear in the vessel and less ‘burning’ of further downstream equipment. Off-gas temperatures in excess of 1700°C are known to have a large impact on refractory life, and the engineering requirements for the handling of large volumes of dirty gases are likely to result in high maintenance costs. For the HI-Smelt process, the off-gas temperature would be approximately 1750°C using a hot-air blast (PCR 50 per cent, HTE 80 per cent). For the same pre-conditions and the use of pure oxygen, the off-gas temperature would be 2000°C.
A recirculating fluidized-bed iron pre-reduction reactor is used to recover some of the sensible heat and chemical energy in the off-gas. Excess heat is used to generate steam.

CRA–Midrex have announced the construction of a pilot facility costing 100 million dollars to produce 100 kt of iron per year at Kwinana, 40 km south of Perth. Commissioning is scheduled for early 1993.

**Volatile Matter**

It is important to note that, for the Corex process, the most important property of the coal is its volatile-matter content, since this determines the gasification temperature for the reaction

\[ \text{Coal} + \text{O}_2 \rightarrow \text{CO} + \text{H}_2. \]  

[10]

Coals that are low in volatile matter generate higher temperatures when they are gasified with oxygen, thus releasing more energy for the melting of the direct-reduced iron from the shaft. When coals of high volatile matter are used, the temperatures are lower because the volatile hydrocarbons must be cracked before full gasification can occur. The higher the adiabatic gasification temperature (2000 to 2400°C for South African coals), the more suitable is the coal for the Corex process, and the lower will the coal consumption be. Suitable coals in terms of ash, fixed carbon, and volatile matter are summarized in Figure 16. Methane (CH₄) in the gas serves as a useful indicator of the decomposition of the hydrocarbons.

CH₄ in the Corex raw gas is about 1 per cent. The particle size of the coal is also important and should be about 2 to 50 mm.

**Environmental Impact**

No review of the new or emerging smelting-reduction processes is complete without a consideration of their impact on the environment.

Twenty-five years ago, the ‘greenhouse effect’ was only the subject of speculation among a few scientists, but today, as evidence for the effect is mounting, it is taken far more seriously. More scientists are accepting the claim that a doubling of the pre-1900 level of 275 p.p.m. of carbon dioxide will cause catastrophic warming of the earth.

Most current climatological studies agree that carbon dioxide in the atmosphere has increased from 275 p.p.m. a century ago to 316 p.p.m. in 1960 to 345 p.p.m. in 1988, and is expected to increase to more than 420 p.p.m. by the year 2050. The earth’s mean temperature has risen by about 0.4°C since 1950. Figures 17 and 18 show the increase in carbon dioxide in the earth’s atmosphere and the increase in mean global temperature respectively. Another 1 to 2°C warming is ‘committed to’ in the words of the Environmental Protection Agency due to past emissions. Should the emissions of carbon dioxide double by the year 2050, a total of 1.5 to 4.5°C will be added to the earth’s mean temperature.

Human sources of carbon dioxide comprise the following: industry 29 per cent, electricity generation 28 per cent, transportation 27 per cent, and residential effects 16 per cent. It is thus clear that, to decelerate the emission of carbon dioxide, a drive to higher energy efficiencies should be promoted—an idea also advocated by Schmithals. Efforts have to be made to develop techniques leading to drastic energy savings. It would be ideal to carry out processes involving high-temperature chemistry in such a way that the mass of raw materials and semi-products (or work-in-process) is heated only once. An ideal temperature profile is shown in Figure 19. Schmithals also shows how, through proper energy management, the emissions of carbon dioxide from the German steel industry are declining; this does not take the emission from power stations into account.

It has been reported that the US steelmakers have also shown an impressive achievement in energy conservation. In the years 1989 and 1990, the energy usage was 35 per...
Constituent: particulates, SO$_2$, NOx, CO$_2$.

Particulates, g
180 40
SO$_2$, kg 480 185
NOx, kg 560 220
CO$_2$, kg 497 459

Production stages:
- Reducing
- Melting
- Conditioning
- Casting
- Rolling

Figure 19—Ideal temperature profile in an iron and steelworks for rebar steel

cent lower than that recorded in 1972. In 1972 it took 38.5 GJ to produce and ship 1 t of steel. In 1990, this figure was down to 25.2 GJ.

Von Bogdandy reports that a process based on coal and oxygen such as that with the EOF (energy optimizing furnace) emits less NO$_x$, CO$_2$, and SO$_2$ in producing 1 t of crude steel than an electric-arc furnace. (Table II).

Allen estimated that some control of carbon dioxide emissions can be achieved in operations fired by fossil fuel through increased efficiencies, and that a reduction in carbon dioxide emissions of 10 to 25 per cent will be achieved through a 10 per cent increase in power-generation efficiency from 35 to 45 per cent.

To stabilize the present concentration of carbon dioxide in the atmosphere, a worldwide release of carbon dioxide of 1 Gt per annum should be aimed at. This is not easy to achieve even if carbon dioxide could be sequestered through afforestation, dumped in the ocean to dissolve as bicarbonate, or pumped into depleted natural-gas or oil reservoirs. For example, to deplete 1,4 Gt of carbon dioxide emitted in the USA in 1988 would have required 10 million square miles of new forest (25 per cent of the land area of the USA).

Oceanic surface waters take up atmospheric carbon dioxide according to the reaction

$$\text{CO}_2 + \text{CO}_3^{2-} + \text{H}_2\text{O} \rightarrow 2\text{HCO}_3^{-}.$$  \[11\]

However, the capacity of surface waters to absorb carbon dioxide is limited to perhaps one-tenth of the current fossil-carbon flux because of the small supply of carbonate ion, CO$_3^{2-}$, present in the water, and the future capacity of the oceans to absorb carbon dioxide may decrease as the carbonate ion is consumed. It is possible, but not yet proved, that the deep layers of the oceans could serve as a sink for carbon dioxide by virtue of the reaction

$$\text{CaCO}_3 + \text{CO}_2 + \text{H}_2\text{O} \rightarrow \text{Ca}^{2+} + 2\text{HCO}_3^{-}.$$  \[12\]

Table II

<table>
<thead>
<tr>
<th>Constituent</th>
<th>EAF</th>
<th>EOF</th>
</tr>
</thead>
<tbody>
<tr>
<td>Particulates, g</td>
<td>180</td>
<td>40</td>
</tr>
<tr>
<td>SO$_2$, kg</td>
<td>480</td>
<td>185</td>
</tr>
<tr>
<td>NOx, kg</td>
<td>560</td>
<td>220</td>
</tr>
<tr>
<td>CO$_2$, kg</td>
<td>497</td>
<td>459</td>
</tr>
</tbody>
</table>

* Calculation: Korf Largi Stahl Eng

Note: All the electrical energy generated from coal has been converted to 100 per cent coal equivalent.

Although several uncertainties exist. Firstly, the reaction rate may be too slow to cope with the high rate of fossil-fuel use and, secondly, considerable and rapid mixing of the layers does not occur. Only 2 to 8 per cent of the surface waters circulate per year to the deeper parts of the ocean.

In South Africa (at the time of writing), the regulation of pollutant levels falls under three different State departments, depending on the source of pollution:
- inside the works — the Department of Mineral and Energy Affairs
- air pollution to the external environment — the Department of Health
- water pollution to the external environment — the Department of Water Affairs.

The values for airborne contamination or pollutants have not yet been gazetted, but the authorities referred to above use the figures given by the American Conference of Government Industrial Hygienists (ACGIH) as guidelines (Addendum 1). These guidelines are applied with considerable flexibility, cognizance being taken of the difficulties associated with accurate sampling, analysis, and control. Pollution-control equipment is mandatory for furnace off-gas systems and is required to operate a minimum of 96 per cent of the year with an efficiency of more than 99 per cent. No values for the total mass of pollutants emitted are registered. However, the average ambient levels of pollutants for the surrounding environment are expected not to exceed the ACGIH's TLV (threshold limiting value) for general substances by more than 2 per cent and/or 1 per cent of the value for carcinogenic substances.

Cognizance should also be taken of the Environment Conservation Act, Act 73 of 1989.

In Europe and the USA, legislation and regulatory constraints are becoming more stringent, and a company causing pollution may be fined or taxed.

South African companies are becoming increasingly aware of the fact that to wait for regulations and then to act may be more expensive in the long run. It is better to be pro-active and to incorporate pollution-prevention measures into new investments. More companies are also accepting that they should strive for environmental management, rather than just pollution control or pollution prevention. It is also very likely that first-world customers will not trade with countries and/or companies that are not giving attention to managing the environment.

Environmental technology is seen to be a growth industry, especially over the next twenty years. Some estimate that, by the middle of this decade (1995), between 2 and 3 per cent of the GDP of the developed nations could be spent on cleaning the environment.

In order to minimize the emission of carbon dioxide from metallurgical processes, the following should be done:
- energy should be used more efficiently
- processes that form less-damaging pollutants should be chosen
- the emission of polluting substances into the air should be reduced by the production of a safer solid discard product
- energy sources that are less polluting should be used (where possible), e.g. nuclear power.
Air pollution in the Highveld of the eastern Transvaal is of concern, and any new process adopted should be considered in terms of the possible impact on an already overloaded system. Atmospheric pollution and its implications in that area have been fully described by Tyson et al.\textsuperscript{33.}

That area, which consists of some 30 000 km\textsuperscript{2} at a mean altitude of 1700 m above sea level, lies east of the Pretoria-Witwatersrand-Vereeniging complex. It is the main coal-producing region in South Africa, and it generates about 80 per cent of the electrical power of Southern Africa in some of the largest coal-fired power stations in the world: Duvha, Kendall, and Tutuka. Also located here are two major petrochemical plants (Sasol II and III), ferro-alloy smelters, steel plants, foundries, paper mills, and fertilizer plants. The smouldering coal dumps, domestic fires in the townships, motor vehicles, and winter veld fires all contribute to atmospheric pollution.

Els\textsuperscript{34} has reported that the average combined emissions of particulates, SO\textsubscript{2}, NO\textsubscript{x}, CO\textsubscript{2}, CO, and hydrocarbons in that area exceeded 125 Mt in 1983/1984.

The dispersion climatology on the Highveld of the eastern Transvaal rates among the most unfavourable anywhere in the world. High atmospheric stability, clear skies, and low wind speeds are generally associated with prevailing high-pressure systems. The conditions are especially bad during winter, when temperature inversions occur almost every night at high frequencies and at a height of 1200 to 1400 m above ground level. The nocturnal low-level inversions are 150 to 300 m deep (Figures 20 and 21).

The atmospheric pollution (updated by Els\textsuperscript{34} in 1987) is shown in Table III. The major sources of primary pollution in the area are power stations, which emit some 40,58 t of sulphur dioxide per square kilometre per year (compared with the UK 14,34; Germany 14,16; and the former East Germany 30,0). Table III shows that the contribution from ferro-alloy and steel plants is extremely small compared with that from power stations (Addendum 23).

As mentioned earlier, the eastern Transvaal, particularly Middelburg-Witbank, has considerable potential for further developments in the secondary and tertiary industries, contributing factors being the proximity of rich coal and mineral deposits and the existing infrastructures.

Since carbon dioxide contributes 50 to 60 per cent to the total greenhouse effect, the present contribution from coal-fired power plants to the total greenhouse effect is about 4 per cent of the total heat-absorbing gases going into the atmosphere. This value is sufficiently low to indicate that the contribution by the world's coal-fired power stations is minor, and that a significant increase in the use of coal for power generation will have small impact\textsuperscript{35}. However, an increase in the number of coal-fired power stations will result in higher emissions of sulphur dioxide, especially where coals high in sulphur are used, and concomitant prevention methods such as coal washing and gas scrubbing will have to be resorted to.

New-generation power stations based on fluidized-bed combustion are also envisaged. To curb the gaseous emissions from new industry, advanced wet-scrubbing systems may have to be used, but limitations on the availability of water may be a complicating factor.

Even though South Africa's contribution to the greenhouse effect amounts to approximately 2 per cent of the world total in terms of carbon dioxide emissions\textsuperscript{33}, approximately 332 Mt of carbon dioxide emissions were measured as originating from South Africa. Of this, 250 Mt came from the energy industry and 2 Mt from the steel and ferro-alloy industry. Although these figures may be low in terms of the global aspect, South Africans should not be complacent about this. A conscientious effort should be made to comply with international standards and to meet environmental regulations.

<table>
<thead>
<tr>
<th>Constituent</th>
<th>Total emissions t/a</th>
<th>Emissions from metallurgical plants t/a</th>
<th>% of total</th>
</tr>
</thead>
<tbody>
<tr>
<td>Particulates</td>
<td>374 692</td>
<td>33 675</td>
<td>8,99</td>
</tr>
<tr>
<td>SO\textsubscript{2}</td>
<td>1 038 556</td>
<td>1 366</td>
<td>0,13</td>
</tr>
<tr>
<td>NO\textsubscript{x}</td>
<td>355 246</td>
<td>?</td>
<td>?</td>
</tr>
<tr>
<td>CO</td>
<td>339 574</td>
<td>?</td>
<td>?</td>
</tr>
<tr>
<td>CO\textsubscript{2}</td>
<td>123 605 162</td>
<td>387 000 +</td>
<td>0,3 +</td>
</tr>
<tr>
<td>HCS</td>
<td>276 503</td>
<td>?</td>
<td>?</td>
</tr>
</tbody>
</table>

? = Emission not known.
Environmental regulation makes good business sense. ‘If a company is imaginative enough to develop products or processes that help the environment, then that company should be rewarded’ 36.

Since early 1970, German industry has spent 170 billion Deutsche mark on new equipment and additional operating costs to clean up and prevent pollution. Today, the Bavarian forests are recovering, and a variety of fish and waterbirds inhabit the Rhine 36.

**Consideration of New Processes**

There are many requirements besides technical innovation that must be considered before a new process can be adopted 37.

- The process must be of low cost on a worldwide basis if it is to survive an inevitable periodic price recession.
- New plants require a marginal price equal to their cost of production (COP) plus capital amortization. A new process cannot usually replace an older process when the demand for its products is static or shrinking.
- The ‘learning curve’ is a fact of life, and development costs (and royalties where applicable) should be included in the COP.
- The cost of the capital employed must be reasonable.
- The sensitivity of profitability to the development costs for high-technology low-volume projects.
- The costs of energy and conservation should be reasonable.
- Environmental-control costs could equal, or even exceed, the primary processing costs.
- A primary process may need to be located close to low-grade ore deposits in order to offset high transportation costs. Numerical analysis will show the best locality if a number of raw materials are to be transported.

New processes can improve the quality and yield of products through the use of computerized optimization, and artificial intelligence and expert systems. A high-intensity process will be favored because of compact equipment, fast kinetic reactions and thus shorter residence times, fast mass transport (heat transfer), and low capital and operating costs 37.

If iron-ore fines (smaller than 10 mesh) are to be pre-reduced, the reduction could be done in a fluidized- or recirculating-bed reactor. The fluidizing velocity of 0.8 to 1 m/s is considered a constraint, and makes this equipment less economical than a fixed- or packed-bed reactor, which normally operates at velocities of 4 m/s or higher. Fluidized-bed reactors may require approximately four times the capital investment required for packed-bed reactors 38. Thus, pelletizing and then pre-reducing in a packed-bed reactor at high gas velocities may prove to be more cost-effective in terms of capital and operating costs. It could be cheaper (and less complex) to use the chemical and thermal energy of the off-gas to smelt the iron ore in the converter, and to utilize elsewhere the chemical and thermal energy left in the gas.

However, conserving energy could make a process too complex and difficult to operate. It may also prove to be cost-intensive. Thus, the drive to minimize the consumption and cost of energy should not be the criteria for the successful development of new processes; the criteria should rather be simplicity of the process and low investment costs, especially in a developing country where technical skills and capital are in short supply.

**THE PRODUCTION OF STAINLESS STEEL DIRECT FROM ORES**

*Direct stainless steel* is a term that can be applied very broadly. In its widest context, it implies the production of molten metal from the ores of chromium and iron (and possibly also nickel), either together in the correct proportions for stainless steel, or separately and then combined in the liquid-metal streams. Finally, the liquid metal is refined in a converter such as the AOD and continuously cast in near net shape (thin slab or strip). The intermediate stages in which solid ferrochromium is produced are eliminated.

New technology such as smelting-reduction may form part of the route for direct stainless steel. The production of stainless steel or ferrochromium by the smelting-reduction technique suffers from the same interrelated and plural technical elements described previously for the production of iron, and some of these are referred to in this Address with typical examples from experimental studies.

**Raw Materials**

Middelburg (Transvaal), which has the infrastructure needed for the production of ferrochromium (charge chrome) and stainless steel, is ideally located in terms of the supply of chromite (from the Bushveld Complex) and coal (from the Witbank, Highveld, and Eastern Transvaal Coalfields). If high-quality carbon steel and stainless-steel scrap are both in short supply, the iron units can be supplemented by direct iron produced from hematite, or by liquid iron produced from the following:

- titaniferous magnetites (after vanadium extraction) occurring in the Bushveld Complex, again in close proximity to Middelburg
- banded magnetites to the north (Pietersburg area)
- hematite from Sishen, which will involve higher transportation costs.

Fluxes in the form of dolomite and fluor spar are also fairly close, but lime has to be transported from the northern Cape (Lime Acres, Ulco, and Oupa laas). Table IV summarizes the reserves and annual production of these raw materials in the Republic of South Africa.

**Chromite**

About 60 to 75 per cent of the South African run-of-mine chromite is of the friable type and, after suitable beneficiation, is usually sold to ferrochromium producers in the form of lumps, chips, or fines. Table V gives the size ranges and production figures from the Winterveld Mine in Steelpoort 43, which mines the LG-6 seam. A typical washed friable ore would analyse 45.2 per cent Cr₂O₃, 26.6 per cent FeO, 15.4 per cent Al₂O₃, 9.6 per cent MgO, 2.1 per cent SiO₂, and 0.9 per cent CaO 44. The average chromium-to-iron ratio is 1.5:1.6. The LG-6 seam has the capacity to deliver about 1 Mt of ore per annum.
Table IV
Ore, coal, and fluxes in South Africa (in Mt)

<table>
<thead>
<tr>
<th>Iron Ore</th>
<th>Reserves</th>
<th>Production</th>
<th>Remarks</th>
<th>References*</th>
</tr>
</thead>
<tbody>
<tr>
<td>Hematite</td>
<td>Sishen (NW Cape)</td>
<td>4,200</td>
<td>29.4</td>
<td>Annualized figure for 1991</td>
</tr>
<tr>
<td></td>
<td>Thabazimbi</td>
<td>100</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Titaniferous magnetite</td>
<td>Mapochs</td>
<td>100</td>
<td>5.4</td>
<td>Production all magnetites</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td>Annualized for 1991</td>
</tr>
<tr>
<td>Magnetite</td>
<td>Zandriviersport</td>
<td>50</td>
<td>Nil</td>
<td>Not in production stage</td>
</tr>
<tr>
<td></td>
<td>De Lokoop</td>
<td>7</td>
<td>Nil</td>
<td>Not in production stage</td>
</tr>
<tr>
<td>Limestone and Lime</td>
<td></td>
<td>2,000</td>
<td>16.3 (1990)</td>
<td>80% of unhydrated lime consumption is in metallurgical and chemical industries</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Chromite</td>
<td>Bushveld Complex</td>
<td>2,400</td>
<td>4.5</td>
<td>1991 figures</td>
</tr>
<tr>
<td></td>
<td>UG-2 Reef</td>
<td>200</td>
<td></td>
<td>Indicated reserves</td>
</tr>
<tr>
<td>Coal</td>
<td>Recoverable reserves</td>
<td>55 x 10^3</td>
<td>219.1</td>
<td>Most coals are bituminous: 2% anthracitic, 1.6% metallurgical</td>
</tr>
<tr>
<td>Springs-Witbank Coalfield</td>
<td>12.5 x 10^3</td>
<td>Approx.</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Highveld Coalfield</td>
<td>11 x 10^3</td>
<td>53% of SA's total</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Eastern Transvaal Coalfield</td>
<td>4.5 x 10^3</td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

* Petrick41 and Alberts42 also provide useful information.

Table V
Winterwald chromite (capacity of the mine*: 80 kt/a)43

<table>
<thead>
<tr>
<th>Type</th>
<th>Size range</th>
<th>Share of production, %</th>
</tr>
</thead>
<tbody>
<tr>
<td>Lumps</td>
<td>-100 +25 mm</td>
<td>2</td>
</tr>
<tr>
<td>Chips</td>
<td>-25 +6 mm</td>
<td>7</td>
</tr>
<tr>
<td>Fines</td>
<td>99% -6 mm</td>
<td>85</td>
</tr>
</tbody>
</table>

* The production in 1991 was approximately 55 kt.

At present, all the fines (flotation-plant sinks) originating from the UG2 seam as discards from platinum-recovery plants are being stockpiled. The average chromium-to-iron ratio is 1.35 : 1.30, and the total annual production approximately 1200 kt.

Coal

Since the South African coals were formed from different plant systems and under different conditions, they differ from those in the northern hemisphere (Europe, China, and North America). The coals of the northern hemisphere are generally rich in vitrinite and exinite, and are thus more reactive than the durain-rich types occurring in the southern hemisphere.

South Africa is poorly endowed with coking coals, and the coals regarded as coking or metallurgical coals usually have higher ash contents than, for instance, Australian metallurgical coals. Average analyses of South African bituminous coals are given in Table VI; Table VII gives the average analyses of washed coals. Figure 22 shows a general all-purpose classification of areas of use for South African coals. It should be noted that low-phosphorus coals are preferred without exception for the production of ferrochromium and stainless steel.

For coal-based smelting-reduction processes, the choice of coal is critical, and successes obtained elsewhere may not necessarily hold true for South African coals. Pilot-plant trials may be necessary to evaluate a process or an envisaged reductant in terms of the available coals.

Smelting of Chromite to Produce Ferrochromium or Stainless Steel Direct

The energy requirements for the production of 1 t of liquid ferrochromium from a chromite containing 45 per cent Cr₂O₃ and 25 per cent FeO, at an efficiency of 100 per cent with the alloy and slag tapped at 1700°C (slag-to-alloy ratio 1:0), would be 3194 kWh, which is 24 per cent higher than for the melting of an iron ore of 94 per cent Fe₂O₃. The heating of the raw materials would require about 40 per cent of this total, and the reduction of the FeO + Cr₂O₃ 47 per cent. High-carbon ferrochromium (HCFeCr or charge chrome) produced via a submerged-arc furnace requires about 4000 kWh per tonne, a very important factor for countries such as Japan where electrical energy is very expensive. Even if pre-reduction and hot charging are used,

![Figure 22](image-url) General all-purpose classification for broad areas of use (SABS specifications)
Table VI

<table>
<thead>
<tr>
<th>Consumer</th>
<th>Product (size in mm)</th>
<th>Ash, %</th>
<th>CV*, MJ/kg</th>
<th>Volatiles, %</th>
<th>Total S*, %</th>
</tr>
</thead>
<tbody>
<tr>
<td>General industry</td>
<td>Cobbles (100 x 31,5)</td>
<td>11-27</td>
<td>21-31</td>
<td>20-31</td>
<td>0,6-2,0</td>
</tr>
<tr>
<td></td>
<td>Nuts (40 x 22,4)</td>
<td>10-26</td>
<td>22-30</td>
<td>20-31</td>
<td>0,4-2,4</td>
</tr>
<tr>
<td></td>
<td>Peas (25 x 6,3)</td>
<td>12-27</td>
<td>21-29</td>
<td>20-32</td>
<td>0,6-1,6</td>
</tr>
<tr>
<td></td>
<td>Diff (-6,3)</td>
<td>10-24</td>
<td>19-29</td>
<td>21-32</td>
<td>0,5-2,1</td>
</tr>
<tr>
<td>Sasol (Tvl)</td>
<td>Crushed coal</td>
<td>20-22</td>
<td>23-24</td>
<td>22-23</td>
<td>0,9-1,0</td>
</tr>
<tr>
<td>Sasol (OFS)</td>
<td>Crushed coal</td>
<td>29-31</td>
<td>19</td>
<td>21</td>
<td>0,5-0,6</td>
</tr>
<tr>
<td>Eskom (Tvl)</td>
<td>Crushed coal</td>
<td>23-29</td>
<td>20-25</td>
<td>22-25</td>
<td>0,8-2,2</td>
</tr>
</tbody>
</table>

* Calorific value
† The higher sulphur values were restricted to a few samples

Table VII

<table>
<thead>
<tr>
<th>Product</th>
<th>Moisture, %</th>
<th>Ash, %</th>
<th>CV, MJ/kg</th>
<th>Volatiles, %</th>
<th>Sulphur, %</th>
</tr>
</thead>
<tbody>
<tr>
<td>Metallurgical</td>
<td>2,3-2,6</td>
<td>6,8-7,6</td>
<td>30,7-31,2</td>
<td>31,8-33,4</td>
<td>0,42-0,59</td>
</tr>
<tr>
<td>low-ash coal</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Power-station</td>
<td>2,2-3,1</td>
<td>12,9-16,1</td>
<td>27,0-28,4</td>
<td>23,6-30,3</td>
<td>0,36-0,97</td>
</tr>
<tr>
<td>smalls</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

the energy consumption is still of the order of 2500 kWh per tonne\(^46\).

Uemura\(^47\) reported recently that the overall power consumption of the Shunan Denko Process to produce HCFECr after 75 per cent pre-reduction of the chromite now stands at 2000 kWh per tonne of alloy. Composite pellets are pre-reduced in a rotary kiln and smelted in a submerged-arc furnace. The liquid alloy is charged into an EAF for processing into stainless steel. Uemura reports that 90 per cent of the alloy produced is now charged in the form of liquid alloy. The thermal efficiency of the process has been improved to 70 per cent. The target is 80 per cent pre-reduction using an indirectly heated rotary kiln. The demand for electrical energy will drop to 1500 kWh per tonne of hot metal.

To produce HCFECr with less, or preferably no, electrical energy, smelting-reduction processes were developed in Japan by the Research Association for New Smelting Technologies (RANST) from 1981 to 1987 (from 1988 known as DIOS). Eleven Japanese companies were members of this project. A number of problems were identified, and the research directed to solve these includes work on

- efficiencies
- kinetics of chromite reduction, especially under conditions of high oxidation potential
- separation of the metal from the freeboard by a thick layer of slag
- high generation of dust
- large volumes of slag
- foaming of slags
- refractory attack
- control of the phosphorus content
- large gas volumes from the high-temperature process
- productivity.

Several papers have reported on these problems\(^48-52\), and these are discussed in this Address.

Nishioka\(^48\) describes how smelting-reduction can be done in a basic oxygen furnace charged with pre-treated (desulphurized and dephosphorized) hot metal from a blast furnace (could be Corex or EOD iron), chromite, fluxes, coal or coke, and blowing with oxygen. Decarburaizing to produce stainless steel can be done in the same basic oxygen furnace using the combined blowing technique or in an argon-oxygen decarburaization (AOD) vessel.

If the production of type AISI 304 stainless steel (18Cr-8Ni) is considered to match or synchronize with a continuous caster having tap-to-tap times of 90 minutes, the production rate, \(d(Cr)dr\), should be more than 0,3 per cent per minute. At present, \(d(Cr)dr\) is about 0,2 per cent per minute, and the process is thus not regarded as sufficiently fast for mass production.

At Kawasaki Chiba Works, the commercial production of stainless steel was achieved by the incorporation of two 85 t K-BOP converters. As previously, the time available for the smelting reduction is limited by the downstream continuous caster. Pre-reduced chromite pellets, fluxes, and lumpy coke are added to the dephosphorized hot metal and, after reduction, the hot alloy is tapped, leaving a heel for the next smelt. After deslagging, the alloy is charged into a second K-BOP, where the crude alloy is refined through decarburaization to stainless steel using a combined blowing technique\(^53\). Table VIII shows the material mix used by Kawasaki to produce stainless steel. From this table it is clear that a limited amount (about 30 per cent) of chromium units came from the chromium ore.

The production of HCFECr by the oxygen-coal in-bath converter process would not require carbon monoxide gas for pre-reduction since chromite cannot be readily reduced by carbon monoxide. Thus, the PCR could be as high as necessary to allow high HTE concomitant with the restraints of refractory wear and off-gas handling. It would be necessary to reach slag temperatures of 1700 to 1800°C. If chromite is to be pre-reduced, it is best done by direct reduction (composite pellets such as used by Shunan Denko and CMI, or chromite fines mixed with reductant in a siliceous slag such as envisaged in the Krupp-CDR). In the
CMI kiln, the degree of pre-reduction attainable is about 55 per cent for chromium and 80 per cent for iron (at a temperature of 1350°C).

Smelting and direct reduction of ore fines can be accomplished in an electric furnace by means of an 'open bath' slag process. This provides higher furnace resistance and thus increased efficiency.

To overcome the high temperatures with higher PCR in the freeboard of a converter, Tanaba of NKK Corporation in Japan proposed a novel idea that was patented. The process takes place in one converter but in two stages.

During stage one, the smelting-reduction process, inert gas is blown from the bottom tuyères, causing a 'fountain' of molten metal on the bath surface. Inert gas is also blown in from the side (2 in Figure 23) to disperse this rising volume of metal into the layer of slag on top of the metal. This dispersion of high-carbon metal in the slag in which the chromite fines float accelerates the reduction of the chromite. Oxygen is blown onto the metal by means of a submerged lance (3), and oxygen is also blown into the slag so that post-combustion takes place mainly in the slag. The slag is agitated by the flow of inert gas from the side tuyères, and the heat generated by the post combustion is transferred to the bath at high LTE. The PCR, defined as

\[
\text{Post-combustion } O_2 = \frac{\text{Decarburizing } O_2 + O_2}{\text{in ore}}
\]

is greater than 30 per cent. The chromite feed can consist of fines (smaller than 1 mm), or pellets. Coke is added as the reductant (additional). The refractory lining is dolomite or mag-chrome.

After the smelting reduction, the slag is discharged, and stage 2 commences. This stage is a decarburization stage using the top lance for oxygen and the bottom tuyères for stirring.

Katayama et al. describe pilot-plant work (on the scale of 5 t/h) done under the auspices of RANST. The special feature of the converter process is the separation of the metal bath from the oxygen jet by a thick layer of slag (Figure 24). This enables the reduction of the chromium oxide in the slag in an atmosphere with a high PCR. Chromite fines or pellets can be used, and the correct choice of raw materials and reductants results in low-phosphorus products. The results indicate that operating with a large volume of slag is possible, and that the oxygen flowrate can be scaled up by 30 times. Slag of appropriate composition accelerated the reduction process without increasing the wear rate of the MgO-C converter lining (Figure 25).

Using composite chrome pellets pre-reduced in a rotary kiln as feed to the smelting-reduction furnace (a converter containing a heel of about 3 t of hot metal), Katayama produced 6 t of charge chrome containing 57 per cent chromium and about 9 per cent carbon per smelt; the slag mass was 5 t. The phosphorus content depended on the reductant used, with petroleum coke giving the lowest values of 0.026 per cent and metallurgical coke up to 0.075 per cent; the latter value is clearly not acceptable. The experimental results are summarized in Figures 26 to 29. Figure 30 shows a concept for commercial-scale exploitation.

The physicochemical properties of slag that are important in the smelting-reduction process are liquidus temperature, viscosity, surface tension, and basicity. Chrome smelting slags are multicomponent systems (MgO–SiO2–CaO–)

---

Table VIII
Material mix in the stainless-steel making process at Kawasaki (kg/t)

<table>
<thead>
<tr>
<th>MF–K–BOP</th>
<th>Smelting-reduction process</th>
</tr>
</thead>
<tbody>
<tr>
<td>MF</td>
<td>K–BOP I</td>
</tr>
<tr>
<td>Normal scrap</td>
<td>351</td>
</tr>
<tr>
<td>Stainless scrap</td>
<td>333</td>
</tr>
<tr>
<td>60% pre-reduced Cr pellets</td>
<td>-</td>
</tr>
<tr>
<td>Alloy (FeCr)</td>
<td>260</td>
</tr>
<tr>
<td>Hot metal</td>
<td>400</td>
</tr>
<tr>
<td>Hot ratio</td>
<td>51.5%</td>
</tr>
</tbody>
</table>

MF Melting Furnace  K–BOP I Smelting reduction converter  K–BOP II Decarburization converter

---

Figure 23—Schematic diagram showing the smelting-reduction of chromium

Figure 24—Specific features of the process as seen by RANST
Al₂O₃–FeO–Cr₂O₃, the base system being the ternary system MgO–Al₂O₃–SiO₂. The viscosity of the slag must be low enough to allow diffusion processes to take place at a sufficiently high rate of reaction between the slag and the alloy for rapid equilibrium to be reached, which affects the kinetics of the process favourably. Ease of separation of the alloy from the slag is another aspect of slag viscosity, and is of particular importance because of the turbulent conditions in the slag during smelting by oxygen blowing. The dissolution of the chromite spinel increases as the basicity of the slag decreases and, in general, the recovery of chromium into the HCFeCr alloy increases to 98 per cent as the basicity decreases to between 0.7 and 0.8. (Chromium recoveries in the conventional submerged-arc furnace are about 85 per cent.)

Muan⁵⁵ points out that a quantitative evaluation of the equilibrium distribution of chromium between alloy and slag requires a knowledge of the high-temperature properties of both phases, an aspect he researched, particularly in the regions of very low oxygen potentials.

Phase relations in the system Cr₂O₃–SiO₂ in contact with metallic chromium show a much larger solubility of Cr₂O₃ in the silicate liquid under low oxygen-potential conditions than there is in air. The liquidus and solidus temperatures are also depressed by approximately 300°C.

Figure 25—Influence of (% MgO) + (% Al₂O₃) on (T.Cr) in slag and the erosion rate of brick in the oxygen converter (Temp: 1590-1620°C, Brick: MgO-C)

Figure 26—Effect of post-combustion on the chromium content of slag

Figure 27—Chromium loss to dust

Figure 28—Examples of the behaviour of chromium in slag

Figure 29—Relation between post-combustion ratio and temperature of the off-gas
composition, slag viscosity, ratio of coke to slag (suppression of slag foaming), apparently changing slag density, temperature, and stirring intensity.

Figure 32 shows that the rate constant \( k_1 \) decreases with chromium content, and becomes more or less constant above a chromium content of about 20 per cent. At a chromium content of less than 20 per cent, the carbon in the alloy contributes to the reduction of the chromium oxides.

Izawa also found that, to limit the erosion rate of the MgO-C refractory lining of the converter (to 1 mm/h or less), the following optimum conditions had to be met:

- Temperature \( \leq 1610^\circ C \)
- PCR \( \leq 60 \) percent
- Slag composition with \((\text{MgO}) + (\text{Al}_2\text{O}_3) \geq 45\) per cent
- The presence of a thick layer of slag, with control of foaming by the addition of coke
- Soft top blowing and high-intensity bottom stirring.

Between October 1986 and December 1987, 1,500t of HCFeCr were produced at the NSC Kimitsu works using the above techniques under the conditions specified.

Crude stainless steel was produced in one of two K–BOP converters using pre-reduced pellets (65 per cent) and dephosphorized pig iron. After smelting, the crude steel was refined to stainless steel in the converter by normal decarburization.

Marukawa \textit{et al}.\textsuperscript{57} used an improved CGS process for the production of a crude stainless alloy (32 per cent chromium, 7 per cent carbon). Lumpy coke and chromite were fed into the slag, and a soft oxygen blast via a top lance burned and heated the coke floating on top of the slag. Through bottom stirring with nitrogen and side stirring with oxygen, the hot coke was mixed into the slag to transfer the heat energy, and was also utilized as a reductant, reducing the chromite dissolved in the slag.

With a \((\text{Cr}_3\%)) of more than 15 per cent, where \((\text{Cr}_3\%)) is the difference between the chromium added as ore and the chromium reduced by smelting–reduction, slag viscosities increased to unacceptable levels, and limited the feed rate and thus the productivity. It was found that the optimum feed rate was such that the \((\text{Cr}_3\%)) was between 5 and 15 per cent (under steady-state conditions). An increase in temperature lowers the \((\text{Cr}_3\%)) and the slag viscosity. The best temperature conditions were found to be as follows:

- Slag 1600 – 1700°C
- Alloy 1550 – 1650°C
- Off-gas 1800°C.

Kouroki \textit{et al}.\textsuperscript{58} co-smelted composite pellets of chromite and composite pellets of iron ore in a plasma-arc furnace, and managed to produce an alloy of 18,4 per cent Cr, 6,45 per cent C, and 1,30 per cent Si, with a chromium yield of 96,5 per cent. Optimal slag conditions were 20 per cent \(\text{CaO} \), 40 per cent \(\text{SiO}_2 \), 30 per cent \(\text{MgO} \), and 15 per cent \(\text{Al}_2\text{O}_3 \).

Huge volumes of gases were produced with very high potential energy yields (both chemical and sensible). For the process to become economically viable, it will be essential to utilize this energy effectively.

Barnes and Finn\textsuperscript{59} studied the pre-reduction and smelting characteristics of UG-2 concentrate and compared them with those of Winterveld chromite ore. The UG-2 ore proved more reducible, and substantial pre-reduction could be obtained at 1250°C (70 per cent reduction after 30 minutes). The chromium-to-iron ratio of UG-2 is typically 1,35, with a higher content of iron rather than a lower
content of chromium. The particles of UG-2 were 99 per cent smaller than 300 μm and this size favours pelletization or injection.

The reducibility of a chromite can be estimated from the ratio

\[ R : \text{non-R} = \frac{\text{Cr}_2\text{O}_3 + \text{MgO} + \text{Al}_2\text{O}_3}{\text{FeO} + \text{Fe}_2\text{O}_3 + \text{SiO}_2}. \]  

[13]

Slatter\textsuperscript{60} states that, all other factors being equal, the ratio \( R : \text{non-R} \) provides the best general measure of reducibility (an ore with a low ratio being more reducible than one with a high ratio).

On this basis, the UG-2 ore has better reducibility than Winterveld chromite. This is confirmed by the experimental observations of Barnes and Finn (Figure 33). Winterveld chromite is considered to be a highly reducible ore, but the UG-2 chromite (marked CW in the figure) required 4226 MJ of energy per tonne, versus 5407 MJ per tonne for Winterveld, for the reduction reactions (21.8 per cent less) or for the overall smelting (12.3 per cent less)\textsuperscript{59}.

The pre-reduction of chromite by solid carbonaceous materials was investigated by Lekatou\textsuperscript{61} in a study of the literature published during the past decade. He came to the conclusion that the rate of reduction of chromite is favoured by a low chromium-to-iron ratio, low contents of MgO and \( \text{Al}_2\text{O}_3 \) in the chromite, and the use of coals with high contents of fixed carbon and volatile matter. He also found that the reduction is accelerated considerably when carbon monoxide and hydrogen are present in the gas. This points towards possible increased production rates if composite pellets and pre-reduction gas rich in CO and \( \text{H}_2 \) (reformed gas) are used.

The role played by carbon and carbon monoxide in the course of chromite reduction is uncertain, and the sluggish rate of reduction of chromite with carbon monoxide gas in the absence of carbon has not yet been explained satisfactorily. It is not clear how the presence of iron oxides, which produce carbon dioxide gas on reduction, enhances the reduction rate of chromite, while a small quantity of carbon dioxide in the gaseous atmosphere is sufficient to inhibit the progress of reduction.

Niayesh\textsuperscript{62} concludes that the carbonaceous reduction of chromite occurs via an intermediate gaseous reduction step, and that carbon cannot reduce chromite at a reasonable rate in the effective absence of carbon monoxide gas. Since \( \text{FeO} \) acts as a generator of carbon dioxide on reduction, the effective reduction of \( \text{Cr}_2\text{O}_3 \) will start only once the reduction of \( \text{FeO} \) is virtually complete.

It would thus be possible, under carefully controlled conditions, to selectively reduce \( \text{FeO} \) (and some \( \text{Cr}_2\text{O}_3 \)) which, on smelting, would produce an iron–chromium alloy with a chromium content that is desirable for the manufacture of stainless steel.

**CONCLUSIONS**

(1) The oxy-coal in-bath smelting–reduction process has advanced technically to a stage where, for the production, at least of iron, demonstration plants are being designed and erected.

(2) The viability of new processes for ironmaking, or the production of ferrochromium and stainless steel direct from the ores, will to a large extent focus on the consumption and efficient use of energy and the impact on the environment.

(3) The applicability of smelting–reduction processes, particularly oxy-coal in-bath processes, may not yet be practical in a mass-production operation. Only full-scale trials will show whether this new technology of smelting–reduction will have the productivity and specific investment costs to make it a real competitor to the continuously improving blast-furnace type of process for the production of iron, and to the processes using direct-current transferred-arc or conventional submerged-arc furnaces, in which pre-reduced chromite is used for the production of ferrochromium, transferring liquid charge chrome to a refining vessel such as the AOD vessel for the production of stainless steel.

(4) Once they have been proved commercially, smelting–reduction processes may be suitable for medium-sized mini-mills, where coke ovens and a blast furnace cannot be justified. Production rates of 500 kt per annum will then be applicable.

(5) Improvements regarding refractories, cheaper oxygen production, and the removal or reforming of carbon dioxide may be key areas for the success of smelting–reduction.

(6) Slags rich in iron oxide are detrimental to conventional refractories, and the problem at this stage is largely unresolved.

(7) Although slag foaming is considered necessary for the transfer of heat, it may be a limiting factor to increased and economic production rates.

(8) The use of ore fines and injection techniques to supply either ores or coals to the bath, and the control of post-combustion and efficient energy transfer, are factors that are important to a successful operation.
Concern for the environment may dictate the production routes that will be followed in the future.

The availability of energy at a reasonable price has been the major factor governing the commercial operation of smelting-reduction processes for the production of ferrochromium or stainless steel. The non-availability of coking coals and environmental concern are the main factors favouring smelting-reduction processes for the production of iron and steel.

In efforts to lower or decrease the overall emissions of carbon dioxide, techniques are to be developed that could lead to drastic energy savings or operating efficiencies that are higher than those of conventional power stations (typically 35 per cent).

The availability of good-quality scrap (containing low residual elements such as copper, phosphorus, and tin) is of concern. The smelting-reduction of low-phosphorus ores using low-phosphorus coals may alleviate the phosphorus problem.

South African coals suitable for smelting-reduction by use of the in-bath technique are still to be identified.

Vertical oxy-coal in-bath smelting-reduction converters may have productivity limitations.

Chromite has to be pre-reduced to a high degree of metallization to make it an attractive feedstock for a smelting-reduction reactor.

Since the dephosporization of chromium-containing alloys is difficult and costly, direct stainless-steel production via a smelting-reduction technique has to rely on the use of low-phosphorus raw materials.

Smelting-reduction techniques (shaft or oxy-coal in-bath) can be considered to be a strong contender in South Africa because of:

- the limited availability of coking coals
- the abundant supply of bituminous coals
- the availability of good-quality iron ore, lumpy and fines
- the availability of almost unlimited resources of chromite fines.

The oxygen-coal in-bath smelting-reduction of iron ores could become a commercial process by 1998, and the production of stainless steel direct from the ores by the year 2000.

ACKNOWLEDGEMENTS

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ADDENDUM 1: ENVIRONMENTAL LEVELS

Table A1 gives some guidelines to the more general pollutants. For other pollutants, the general guideline is that the environmental level must not exceed 1/50 of its threshold limit value (TLV). (TLV is the threshold limit value for substances in workplace air of the American Conference of Government Industrial Hygienists.)

There are guidelines for the pollutant levels in stacks for different processes. For example, the concentrations of HCl, SO2, and NH3 must not exceed 35 mg/m3. The general guideline for particulate matter is 120 mg/m3.

Table A1

<table>
<thead>
<tr>
<th>Pollutant</th>
<th>1 hour</th>
<th>24 hour</th>
<th>Year</th>
</tr>
</thead>
<tbody>
<tr>
<td>Sulphur dioxide</td>
<td>780</td>
<td>265</td>
<td>80</td>
</tr>
<tr>
<td>Nitrogen oxides</td>
<td>1180</td>
<td>540</td>
<td>270</td>
</tr>
<tr>
<td>Oxygen</td>
<td>240</td>
<td>100</td>
<td></td>
</tr>
<tr>
<td>Lead</td>
<td>-</td>
<td>-</td>
<td>25</td>
</tr>
<tr>
<td>Dust particles</td>
<td>350</td>
<td>150</td>
<td>-</td>
</tr>
<tr>
<td>Smoke (ft/m³)</td>
<td>50</td>
<td>3</td>
<td></td>
</tr>
</tbody>
</table>

ADDENDUM 2: ESTIMATES* OF 1987 POLLUTANT EMISSIONS TO THE ATMOSPHERE IN THE HIGHVELD OF THE EASTERN TRANSVAAL

<table>
<thead>
<tr>
<th>Source</th>
<th>1987 emissions (t/a)†</th>
<th>1987 emissions (t/a)†</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Particulates</td>
<td>SO2</td>
</tr>
<tr>
<td>Power stations</td>
<td>(83.28)*</td>
<td>(91.37)</td>
</tr>
<tr>
<td></td>
<td>355,843</td>
<td>1,110,585</td>
</tr>
<tr>
<td>Brickworks</td>
<td>(0.02)</td>
<td>(0.03)</td>
</tr>
<tr>
<td></td>
<td>?</td>
<td>2,634</td>
</tr>
<tr>
<td>Ferro-alloy works</td>
<td>(6.49)</td>
<td>(0.13)</td>
</tr>
<tr>
<td></td>
<td>27,713</td>
<td>1,557</td>
</tr>
<tr>
<td>Steelworks/foundries</td>
<td>(2.94)</td>
<td>(0.01)</td>
</tr>
<tr>
<td></td>
<td>12,582</td>
<td>78</td>
</tr>
<tr>
<td>Sawmills</td>
<td>(0.49)</td>
<td>(0.04)</td>
</tr>
<tr>
<td></td>
<td>2,110</td>
<td>499</td>
</tr>
<tr>
<td>Paper and pulp mills</td>
<td>(0.01)</td>
<td>(0.01)</td>
</tr>
<tr>
<td></td>
<td>44</td>
<td>79</td>
</tr>
<tr>
<td>Petrochemical plants</td>
<td>(0.10)</td>
<td>(0.08)</td>
</tr>
<tr>
<td></td>
<td>433</td>
<td>7,419</td>
</tr>
<tr>
<td>Domestic/municipal</td>
<td>(4.60)</td>
<td>(3.04)</td>
</tr>
<tr>
<td>combustion</td>
<td>19,465</td>
<td>36,984</td>
</tr>
<tr>
<td>Coal dumps</td>
<td>(4.47)</td>
<td></td>
</tr>
<tr>
<td>Motor vehicles</td>
<td>X</td>
<td>54,390</td>
</tr>
<tr>
<td></td>
<td>4,466</td>
<td>X</td>
</tr>
<tr>
<td>Other</td>
<td>(1.08)</td>
<td>(0.32)</td>
</tr>
<tr>
<td></td>
<td>4,608</td>
<td>3,903</td>
</tr>
<tr>
<td>Total</td>
<td>427,264</td>
<td>1,217,728</td>
</tr>
<tr>
<td>variance since 1984</td>
<td>+ 14%</td>
<td>+ 17%</td>
</tr>
</tbody>
</table>

* The values given are based on measurements and calculations
† X denotes relatively small emissions, and ? denotes that the emission is not yet known
‡ The SO2 emissions assume 1.2% sulphur in the coal and a 30% retention of SO2 on the fly-sah
§ The values in brackets indicate the percentages of total emissions for a given pollutant.