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EVALUATION OF STEELMAKING PROCESSES

Topical Report

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EVALUATION OF STEELMAKING PROCESSES

Executive Summary

The AISI Direct Steelmaking Program has formed a task force to examine the existing and some proposed processes for refining the metal produced from the AISI smelter; the analysis is also applicable to blast furnace hot metal. After initial screening, the following processes were examined in detail: Trough Processes (COSMOS, WORCRA, etc.), Posthearth Refining, IRSID Continuous Steelmaking, EAF, and EOF (Energy Optimizing Furnace). The detailed technical analysis included development status and critical issues such as productivity, yield, and degree of refining. Variable operating costs were developed for the IRSID and EOF processes and compared with those for a BOP. A sensitivity analysis for scrap prices ranging from $90 to $130 per net ton was also developed. Considerations regarding the use of hot metal in an EAF are provided in the appendix. Economic analyses for the EAF are best carried out by companies for their special circumstances. Rough estimates of the capital costs, in dollars per annual ton, are also provided for the BOF, EAF, EOF, and IRSID processes.

The major findings are summarized below:

Trough Processes: The detailed analyses indicated that these processes may work in theory, but heat losses would be excessive and the complications far exceeded any advantages. The processes were not as promising as the IRSID process.

Posthearth Refining: There are potential advantages and considerable complications associated with refining in the posthearth of the smelter. At the current time it is not known if a posthearth is required and how it would operate; therefore, further analysis was not conducted.

IRSID: The IRSID process is a reasonably well-proven process that continuously produces metal similar to a BOF. It has similar operating costs and considerably lower capital cost as compared to the BOF.

EAF: The EAF is examined in detail in the Appendix. The process is well proven. Mitsubishi Steel will use EAFs to process blast furnace hot metal next year.

EOF: The EOF would produce steel higher in sulfur, and scrap should be limited to 50 - 60% to avoid operational and residual element problems. The EOF has similar operating costs to the BOF and considerably lower capital costs.

It was concluded that there is insufficient technical or economic incentive to replace a working BOF with any of these processes to refine hot metal. However, if new or incremental steelmaking capacity is required,the IRSID and EOF should be considered. Both have lower capital costs, and the EOF has the flexibility to melt more scrap. The IRSID process could be developed with a pilot plant associated with the AISI smelter. The EOF could be further evaluated with well-designed and controlled tests at an existing EOF facility.

A fully continuous process from ironmaking to casting was considered. The capital and operating cost savings that could be achieved beyond those for direct ironmaking and continuous refining are relatively small. A fully continuous process should not be considered until direct ironmaking and continuous refining are perfected.

EVALUATION OF STEELMAKING PROCESSES

Introduction

The ultimate objective of the AISI Direct Steelmaking Program has been to develop a process or a series of processes to produce steel directly from ore and coal. The processes should be less capital intensive, consume less energy, and have higher productivity or process intensity than the current processes. To date, a direct ironmaking process has been developed which is of immediate interest to many North American steel companies. The objective of the current study is to examine in detail the available and potential processes for refining the metal produced from the AISI smelter.

AISI Direct Steelmaking organized a task force to examine in detail the available processes with regard to their technical feasibility and, for the most promising cases, their economics. After inital screening, several were examined in detail; these are listed below, along with the key investigator.

Process	Investigator
Trough Processes (COSMOS, WORCRA, Iron carbide, etc.)	C. L. Nassaralla
Posthearth Refining	H. M. Pielet
IRSID Continuous Steelmaking	R. J. Fruehan
Electric Arc Furnace	G. J. W. Kor
Energy Optimizing Furnace	R. J. Fruehan

Variable operating costs for the IRSID and EOF processes were developed and compared with those for a BOP by H. R. Pratt and C. G. Davis. They are presented in Table I with indicated scrap usages. Figure 1 compares the variable operating costs developed in Table I for scrap prices ranging from \$90 to \$130 per net ton. Capital costs for the IRSID process were estimated by J. H. Cox, and capital costs for the EOF were provided by Lurgi.

The detailed analyses included development status (pilot plant, commercial, etc.) and the following critical issues:

Productivity
Yield
Refining (S, P, N, etc.)
Scrap (amount, size, residuals, etc.)
Metal feeding
Tapping

Gas removal and cleaning
Environmental issues

When possible, the analysis was based on the available literature. However, in many cases original theoretical or process analysis was required. The analysis for each process was reviewed by the entire committee, and the recommendations reflect the consensus.

As a base case, conventional oxygen steelmaking (OSM), such as BOF and Q-BOP, was considered for the technical and economic evaluation. For a process to be recommended, it must be capable of producing most of the steels produced by OSM and must have an economic or strategic advantage over conventional OSM. Since the raw materials and processing will be the same in most cases, the major economic advantage would be expected to be in capital costs. An example of a strategic advantage is a process which is capable of melting more scrap. Of the processes considered, all are batch processes with the exception of the IRSID Process and the Trough Processes. Even the IRSID Process is batched prior to steelmaking and at the end for ladle refining. The most attractive process in theory is one that is truly continuous. However, it is assumed that it will be necessary at least to batch the metal for final ladle processing. It may be possible to have a continuous process, including continuous tapping of the smelter, continuous desulfurization, and continuous refining. The major incentive for such a process would be lower capital costs by eliminating metal holding equipment. The possible economic advantage of a truly continuous process was also examined.

This report will consist of a brief summary of the major findings of the technical assessment for each process, the economics for the most promising cases, and recommendations. A complete evaluation for each process is in the Appendix.

Technical Evaluations

Trough Processes: A number of continuous steelmaking processes have been proposed based on metal refining in a long horizontal reactor (trough). These processes include WORCRA, COSMOS, the Bethlehem Steel process, and Iron Carbide Steelmaking. These processes may be concurrent or countercurrent with respect to the slag and metal. In some cases pilot-plant-scale data were available. Based on these data and theoretical analysis, critical issues such as geometry, backmixing, control, scrap melting, etc., were considered. In particular, a dispersion model (a modified plug flow) and the use of barriers to control flow were considered.

Based on the analysis, the following major conclusions have been made:

1. Trough processes have been studied extensively.

2. In theory, they can work but require either very large length-to-width ratios or multiple barriers.

3. If barriers are used, these would present a problem in maintenance, and as the openings in the barriers wear, increased control problems will occur.

4. Multiple and individually-controlled oxygen lances are required.

5. The required geometry would result in large heat losses, and consequently less scrap may be melted. Furthermore, scrap may be limited in size.

It was concluded that, whereas these processes could be made to work, their problems outweigh any minor advantages they might have over OSM. Furthermore, other processes look more promising.

Posthearth Refining: It has been proposed that the AISI smelter have a posthearth for separating the phases and tapping. This posthearth could potentially be used for refining. Desulfurization or even decarburization could be performed. Several refining options were considered, including continuously refining to 0.5 to 1.0% C or continuous refining to about 1.0% followed by batch refining to 0.1% C.

A detailed analysis of the potential advantages and problems was made. The advantages included lower capital cost, since there could be a single gas cleaning system for smelting and refining, improved overall yield by recycling refining slag to the smelter, and lower nitrogen contents. Potential problems include refining for sulfur and phosphorus, phase separation, and decrease in smelting productivity.

Based on the fact that no experience with regard to the posthearth currently exists, the following issues should be examined before proceeding further:

1. Determine if a posthearth is necessary for separation of the char and metal from the slag.

2. If it is not, consider if the degree of refining achieved is adequate incentive for having a posthearth.

3. If posthearth refining is seriously considered, the following research should be conducted: water and mathematical modeling of the connection between the smelter and posthearth, a refining model for the process, and potential for use of 0.5 - 1.0% C metal.

IRSID Continuous Steelmaking: The IRSID Process is the furthest developed continuous steelmaking process. A 25 tonne per hour pilot plant was run from 1970 - 1973 producing over 17,000 tonnes of steel. The process can produce steel with carbon contents of about 0.05%; below this level vacuum processing is required. Despite the previous work, a number of issues were identified which could be examined in a pilot plant research program. These include the reactor and decanter sizes, improved methods of metal feeding, and the method and place for scrap additions.

It was concluded that this process is technically feasible and may have adequate lower capital cost incentives.

EAF Using Smelter Metal: The use of hot metal in an EAF is not new; for example, ISCOR uses metal from their COREX in an EAF, and Mitsubishi Steel plans to add blast furnace hot metal to their EAFs at Mouran Works. Nevertheless, there are a number of critical issues which must be addressed. The hot metal to scrap mix is limited by the product requirements. Refractory wear may be slightly greater and electrode consumption may be higher due to oxidation. Total electrode consumption may nevertheless be lower due to lower power consumption. High sulfur contents may present a problem. If a high hot metal charge with high sulfur is used, both hot metal and steel ladle desulfurization may be required. Product mix, furnace capabilities, and hot metal transfer facilities will affect a company's decision to add hot metal to its EAFs.

EOF: The EOF appears to be a good match for the smelter and offers high scrap melting capability when using a 40 - 60% hot metal charge. It has been used commercially and is highly productive.

1. The EOF has the same limitations as the EAF with regard to product requirements and the amount and type of scrap used. It has the advantage over the EAF of not requiring electrical energy or the capital cost associated with it. It has the disadvantage that 100% scrap charges have not been effectively melted.

2. Nitrogen contents will be slightly higher than those from a BOF but should be satisfactory for most products. Phosphorus levels will be similar to those from a BOF, but vacuum processing will be required for less than 0.03% C.

3. As with the EAF, sulfur may be a problem, especially if high sulfur coal is used in the EOF for a high scrap charge.

The EOF appears to be an attractive method of refining the metal from the AISI smelter, and an economic evaluation has been made.

Economic Analysis

Operating Costs: Operating costs are site-specific and depend on local unit costs. In this study, the same unit costs for all cases were used and are those in the AISI Direct Steelmaking "Technoeconomic Assessment" report. The process elements for the BOF were taken from that report. The process elements for the IRSID Continuous Steelmaking and the EOF were supplied by IRSID and Lurgi, respectively. The price of the hot metal is $131.45 per ton as given in the AISI-DOE report. The price for scrap is assumed to be $110 per ton; a sensitivity analysis for the price of scrap was conducted. Desulfurization is a major consideration, and these costs are taken from the analysis in the Appendix.

The operating costs for the oxygen steelmaking processes (BOF, IRSID, and EOF) are shown in Table 1. As expected, the operating costs are similar for all three cases, with those for EOF being slightly higher primarily due to desulfurization costs. Within the certainty of the analysis, the costs for the BOF and IRSID are the same.

The conclusions are sensitive to the assumptions. If the steel produced does not require high quality scrap or scrap prices are lower, the EOF and EAF obviously become more attractive. Conversely, if hot metal costs are lower, the BOF and IRSID costs are reduced more than those for the EOF. A sensitivity analysis for scrap prices ranging from $90 to $130 per ton is shown in Figure 1. Desulfurization is a major cost. If less than 0.01% sulfur is needed, steel desulfurization for the BOF and IRSID process would be required, increasing their cost. In all cases, if steel less than 0.03% C is required, vacuum processing is necessary.

In general, the cost for IRSID, EOF, and BOF depend on the product and assumptions but are reasonably similar, as shown in Table 1. In general, for lower quality products, the EOF may have a slight advantage, while for high quality products, the BOF and IRSID are preferred.

Capital Cost: Capital costs are also site-specific and depend on the assumptions. The assumptions are that the plant is to produce about 1 million tons of steel per year and that a brownfield site is available. The main assumptions are listed below:

1. The building and a cleared site exist.
2. Hot metal and steel ladles exist and are not included.
3. The main crane exists.
4. Oxygen and other utilities are at the battery boundary.

The costs for the IRSID refiner were estimated by Hatch, the BOF by Davis and Pratt from updated studies, the EAF by Hatch and Davis from industry averages, and the EOF by Lurgi. The results are presented in Table 2 with an estimated uncertainty.

Continuous versus Batch Processing

Many experts view the ultimate iron-steelmaking-casting process as being truly continuous from ironmaking through casting. In the current coke oven-blast furnace-desulfurizer-BOF-ladle metallurgy-continuous casting, only ironmaking and casting are continuous with the others being batch processes. This study indicated there were potential capital cost savings with continuous refining, such as with the IRSID process. These arise from higher productivity by eliminating the time for charging and tapping and the need for tilting furnaces. However, as currently conceived, the IRSID process requires batching before and after steelmaking. The questions are: to what extent is it practical to be truly continuous, and are there significant additional cost savings.

It is our view that it is necessary to batch the steel for ladle metallurgy, including reheating and vacuum treatment for final composition, temperature control, deoxidation, and inclusion removal. However, it is possible to tap a smelter continuously, followed by continuous desulfurization and continuous steelmaking. Continuous desulfurization could, for example, be by the GM Process reviewed previously by AISI, and steelmaking could be by the IRSID process. A generic steel would be produced, reducing control problems, which would be treated batchwise by ladle metallurgy, reheating, and vacuum processing, if necessary.

Continuous processing from ironmaking to ladle refining has some definite advantages with regard to temperature losses. Less temperature loss allows for more scrap melting and lower refractory consumption. Continuous processing would also reduce liquid metal handling, reducing labor and capital costs. Capital cost savings include hot metal ladles, cranes, etc. However, in comparison to the capital cost savings obtained by direct ironmaking or continuous refining, the additional savings are relatively small, and the complications would be considerable. Without the buffers that major process batching offers, deviation or interruption in any of the processes would cause major complications in all the processes. Therefore, the ultimate goal of a truly continuous process should not be pursued until direct ironmaking and continuous refining are both demonstrated in commercial-size plants.

Conclusions and Recommendations

1. After screening many possible alternative steelmaking technologies, five were selected for a detailed technical

analysis. An economic analysis was conducted for the two which appeared technically attractive (IRSID and EOF).

2. **Trough Processes** (WORCRA, Bethlehem, etc.): These processes were judged to be technically feasible under certain circumstances, but the potential problems, such as long length, multiple oxygen lances or tuyeres, and metallurgical control, far outweighed any potential advantage. These processes were found to be not as attractive as the IRSID process, which was the only other continuous process evaluated.

3. **Posthearth Refining:** The AISI smelter may require a posthearth in which limited refining may be done. The limitations and lack of experience with such a process led to the conclusion that further work would be required to make an adequate assessment. The need for a posthearth and how it functions will have to be determined.

4. **IRSID Continuous Steelmaking:** The IRSID process is reasonably well-proven on a pilot scale and technically attractive. It is a continuous process and will have a higher productivity per reactor volume than a BOF because it does not require time for charging and tapping. A limited amount of development work is required to better define vessel size, scrap melting, etc. The operating costs are similar to the BOF, but the capital costs may be significantly lower.

5. **Energy Optimizing Furnace (EOF):** The EOF is both technically and strategically attractive. As with the EAF, the products may be limited, and sulfur levels will be high. The operating costs are slightly higher than for the BOF or IRSID processes, primarily due to desulfurization costs.

6. **Truly continuous processing should not be considered until a continuous refining process has been adequately demonstrated.** The advantages of using a truly continuous process from ironmaking to ladle refining were examined. There appear to be significant capital cost advantages for continuous refining. The advantages of further implementation of continuous processing include lower temperature losses and the need for liquid metal handling, including labor and capital (cranes, ladles, etc.). Due to metallurgical requirements, it is considered necessary to batch the steel for final ladle metallurgical processing for cleanliness, chemistry, and temperature control. Whereas linking the smelter directly with continuous steelmaking is a desirable and theoretically possible goal, the economic and processing advantages beyond continuous refining are small.

TABLE 1
COMPARISON OF OPERATING COSTS FOR OSM
(BOF, IRSID, & EOF)

	UNITS	UNIT COS $/UNIT	BOF UNITS/ NTLS	BOF $/NTLS	IRSID UNITS/ NTLS	IRSID $/NTLS	EOF UNITS/ NTLS	EOF $/NTLS	EOF UNITS/ NTLS	EOF $/NTLS
% CHARGE										
HM			80		80		60		40	
SCRAP			20		20		40		60	
METALLICS										
HOT METAL	NT	131.45	0.8447	111.04	0.8447	111.04	0.6320	83.08	0.4200	55.21
ORE	NT	25.00	0.0000	0.00	0.0060	0.15	0.0000	0.00	0.0000	0.00
SCRAP	NT	110.00	0.2197	24.17	0.2197	24.17	0.4440	48.84	0.6680	73.48
REAGENTS										
LIME	NT	50.00	0.0164	0.82	0.0164	0.82	0.0256	1.28	0.0382	1.91
DOLOMITE	NT	50.00	0.0018	0.09	0.0018	0.09	0.0000	0.00	0.0000	0.00
OXYGEN	NT	30.00	0.0681	2.04	0.0681	2.04	0.0716	2.15	0.1120	3.36
NITROGEN	NT	34.00	0.0032	0.11	0.0000	0.00	0.0000	0.00	0.0000	0.00
COKE BREEZE	NT	50.00	0.0000	0.00	0.0000	0.00	0.0000	0.00	0.0402	2.01
TOTAL MATERIAL				138.26		138.31		135.34		135.97
OTHER COSTS										
ELECTRICITY	Kwh	0.037	60.0000	2.22	60.0000	2.22	10.5000	0.39	12.3000	0.46
REFRACTORIES	Lbs	0.50	4.0000	2.00	4.0000	2.00	3.0000	1.50	3.0000	1.50
NATURAL GAS	MMBtu	3.00	0.3333	1.00	0.3333	1.00	0.1800	0.54	0.1800	0.54
LABOR & SUPER	MH	25.00	0.2300	5.75	0.2300	5.75	0.2300	5.75	0.2300	5.75
R&M	$			4.40		3.00		3.00		3.00
T&S	$			4.40		4.40		4.40		4.40
TRANSPORT	$			0.75		0.75		0.75		0.75
SERVICES	$			4.30		4.30		4.30		4.30
OTHER EXPENS	$			6.50		6.50		6.50		6.50
DUST HANDLING	Lbs	0.02	20.0000	0.35	40.0000	0.70				
	Lbs	0.10					40.0000	4.00	40.0000	4.00
DESULFURIZATION	$			6.50		6.50		13.00		13.00
TOTAL OTHER COSTS				38.17		37.12		44.13		44.20
TOTAL COST				176.43		175.43		179.47		180.16

Table 2

Capital Cost Estimates for Refining Processes

Process	Dollars per annual ton
BOF	70 - 75
EAF	72 - 80
EOF	30 - 45
IRSID	35 - 40

Assumptions:

- One million ton per year plant
- Building and main crane exist
- Cleared site
- Utilities at battery limit

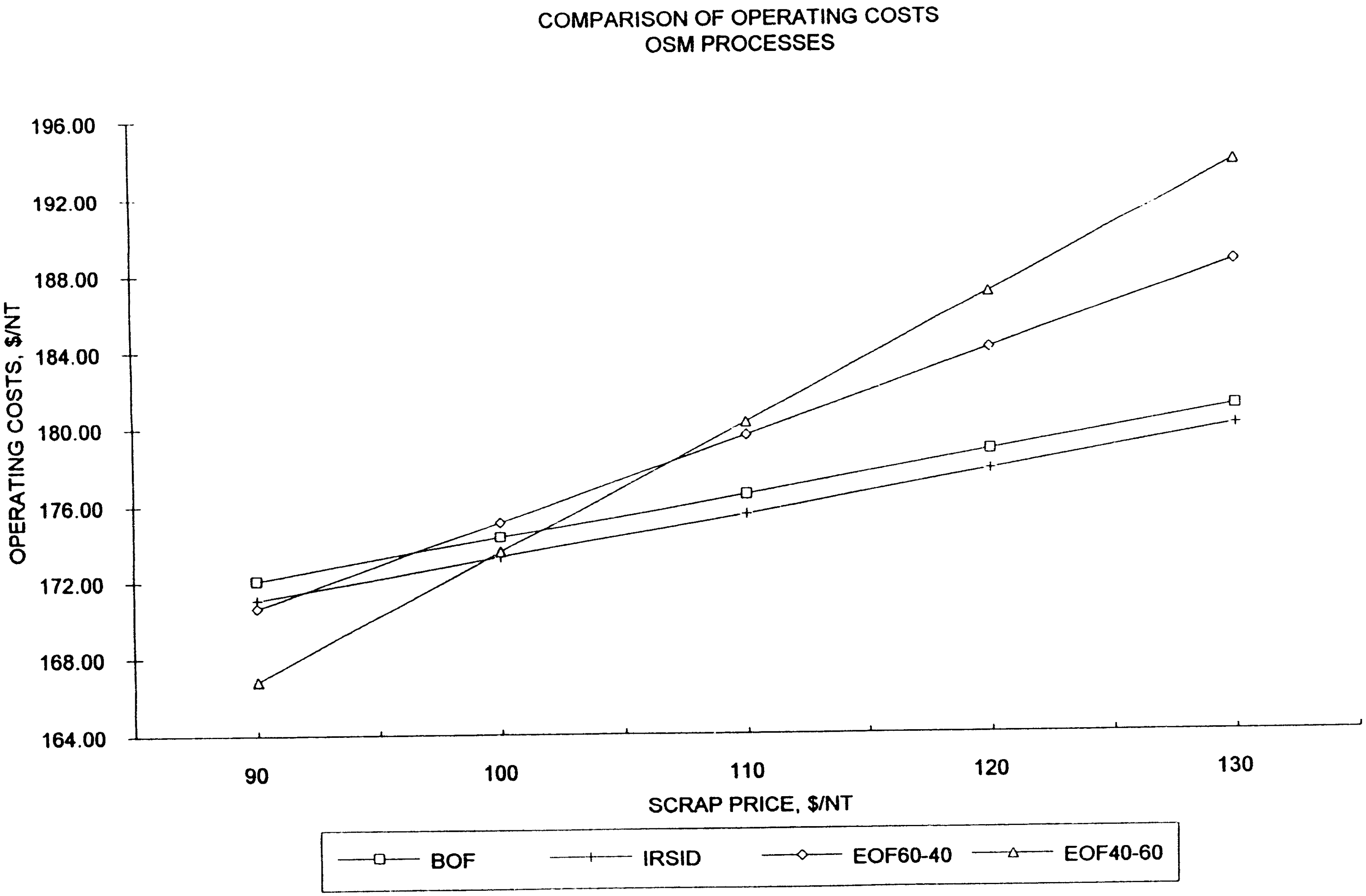
COMPARISON OF OPERATING COSTS
OSM PROCESSES
OPERATING COSTS, $/NT
196.00
192.00
188.00
184.00
180.00
176.00
172.00
168.00
164.00
90
100
110
120
130
SCRAP PRICE, $/NT
BOF
IRSID
EOF60-40
EOF40-60

Figure 1

Appendix

Technical Evaluations

General Evaluation of the Trough Continuous Steelmaking Process

Bethlehem Steel Continuous Steelmaking Evaluation

COSMOS Continuous Steelmaking Evaluation

IRSID Continuous Steelmaking

IRSID Continuous Steelmaking Evaluation

EAF Steelmaking Evaluation

Post Hearth Refining in an Integrated Smelter Refiner

EOF Steelmaking Evaluation

Desulfurization Strategy and Cost

GENERAL EVALUATION OF THE TROUGH CONTINUOUS STEELMAKING PROCESSES

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1- Introduction

The purpose of this report is to conduct an overall evaluation of trough continuous steelmaking processes. The evaluation includes the technical feasibility, concerns and whether the processes respond to the economic and technical drivers. These processes include WORCRA,(1) COSMOS(2) and the Bethlehem process(3) discussed previously in detail. This evaluation may be extended to the Iron Carbide Continuous Steelmaking process (ICCS) proposed by Geiger.(4)

The ICCS process is simply a trough process in which iron carbide is the feed material. The iron carbide consists of Fe_3C, 2-10 percent unreduced FeO, and gangue. In theory, these processes can use any feed, for example: DRI plus carbon, cold pig iron and even scrap plus carbon; there is nothing unique concerning iron carbide. Iron carbide has the following disadvantages, it has to be injected, requires extra energy since it has gangue associated with it, requires energy for FeO reduction and does not allow scrap melting. Therefore, hot metal is a more attractive feed for the process.

2- Decarburization

The decarburization reaction in a trough process is similar to BOP and Q-BOP reactor. The decarburization reaction of iron with oxygen occurs between gas bubbles and liquid metal. The transition between dissolved gas from a liquid metal to the gas phase comprises several stages:

1. Transport of carbon from the metal bulk to the gas-metal interface.
2. Transition of oxygen atoms from the dissolved state to the adsorbed state at the surface.
3. Reaction of the adsorbed oxygen with carbon.
4. Desorption of the CO molecule from the surface.
5. Diffusion of CO molecule into the gas phase.

The slowest of these steps is the rate controlling mechanism for the process.

In the case of decarburization, the rate controlling mechanism depends on the carbon content in the metal. At high carbon (>0.3 percent), the rate is controlled by oxygen flowrate, which means that the decarburization rate is controlled by the oxygen flowrate. At low carbon levels, the rate controlling step is liquid phase mass transfer; this rate is primarily controlled by transport of carbon from molten steel to the FeO at the interface.(5)

The rate equation for high carbon content can be written as

$$\frac{\dot{N}_{O_2}}{W} = k \qquad (1)$$

where $\dot{N}_{O_2}$ is the oxygen flow rate, W is the weight of the bath and k is the rate constant.

The rate equation of liquid phase mass transfer can be written as

$$J_C = mA(C^s - C^b) \qquad (2)$$

where J_C is the flux of carbon, m is the mass transfer coefficient, A is the surface area, C_s and C_b are the concentrations of carbon at the surface of the bubble and bulk.

The mechanism of decarburization can be visualized by formation of iron oxide layer at the gas bubble/liquid metal interface. As the bubble rises, the carbon dissolved in the metal reduces the iron oxide, and if the retention time on the bubble is long enough (the retention time is directly proportional to the depth of the molten metal in the vessel), before the bubble reaches the surface of the bath all iron oxide will be reduced. This happens at high carbon content in the bath. At low carbon content, the diffusion of carbon through the liquid metal to the reaction site is slower, therefore not all the iron oxide is reduced by the carbon before the bubble leaves the bath.

3- Desulfurization and Dephosphorization

The dephosphorization and desulfurization reactions of iron occur in the interface between slag and metal, and these reactions are controlled by liquid phase mass transfer, which means the rate controlling step of these reactions is the transport of phosphorus and sulfur from metal to slag phases. The rate equation of liquid phase transfer for phosphorus and sulfur is similar to Equation (2). In the conventional steelmaking converter, the rate of phosphorus and sulfur reactions are fast enough, to consider that they approach equilibrium conditions. Similar behavior for phosphorus and sulfur is expected in the trough process. The analysis below was done assuming the process reached a steady state, which is expected to be close to the equilibrium conditions.

3.1- Trough Process with Concurrent Flow

The phosphorus level in a concurrent flow trough process will be similar to the ones obtained in the BOP and Q-BOP. It can estimated by performing a simple mass balance inside the reactor, assuming the input phosphorus content in the hot metal ($[P_o]$) is 0.1 percent, the phosphorus partition ratio is 100, and the weights of metal and slag are 225 and 22.5 tones, respectively.

$$[P_o]Wm = [P_f]Wm + (P_f)Ws \qquad (3)$$

Where $[P_f]$ and (P_f) are the final phosphorus content in the metal and slag phases, respectively.

Rearranging Equation (3):

$$[P_f] = \frac{[P_o]}{Wm + L_P Ws} = 0.009 \qquad (4)$$

However, the sulfur level will be higher, since no hot metal desulfurization operation is performed before steelmaking process, and the hot metal produced via smelting process has higher sulfur, once coal is used in the process instead of coke.

3.2- Trough Process with Counter Current Flow

The trough process with counter current flow is an excellent refiner. Since the phosphate capacity of the slag increases along the length of the vessel (from the charging port to the tapping hole), the phosphorus level at the end of the process is about a half of the one obtained in the conventional BOP and Q-BOP practices. On the other hand, the sulfide capacity increases in the opposite way (from the tapping hole to the charging port). Therefore, most of the sulfur removed in the process is transferred to the slag near the charging port. Also the high carbon level, at the charging port, further enhances the desulfurization process.

A mass balance can be written for this process to prove these findings. Lets assume that the trough is divided in 2 zones, a reducing and an oxidizing zone, as shown in Figure 1.

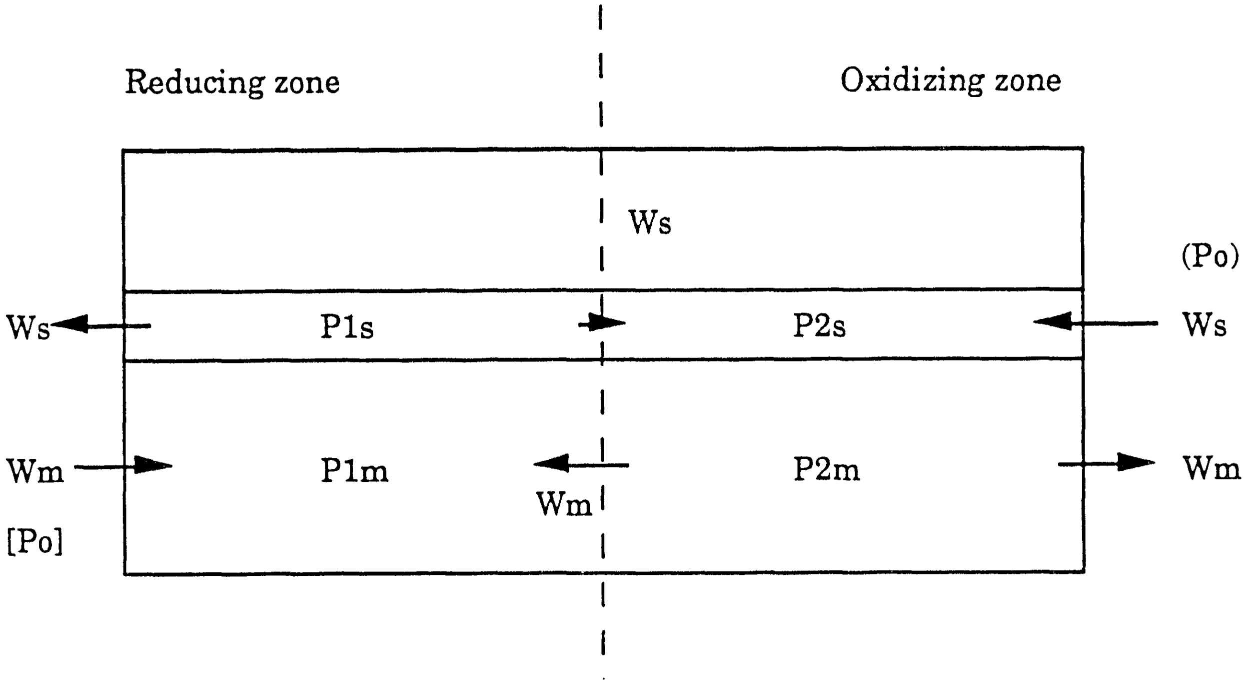


Figure 1.- Schematic of the inputs and outputs in a trough process.

Where Ws and Wm are the weight of slag and metal respectively; $[P_o]$ and (P_o) are the initial P content in the hot metal and slag added to the vessel, $[P_{1m}]$ and $[P_{2m}]$ are the phosphorus content in the metal at reducing and oxidizing zones of the vessel in contact with L_{P1} and L_{P2} (phosphorus partition ratio in the reducing and oxidizing zones, respectively) and (P_{1s}) and (P_{2s}) are the phosphorus content in the slag at reducing and oxidizing zones of the vessel.

A mass balance, at the reducing zone of the vessel assuming that steady state is reached, is represented by Equation (5), also the phosphorus partition ratio (at the reducing zone) can be represented by Equation (6).

$$[P_o]Wm + (P_{2s})Ws = [P_{1m}]Wm + (P_{1s})Ws \qquad (5)$$

$$L_{P1} = \frac{(P_{2s})}{[P_{1m}]} \qquad (6)$$

A mass balance can also be performed at the oxidizing zone of the vessel, and be represented by Equation (7).

$$[P_{1m}]Wm + (P_o)Ws = [P_{2m}]Wm + (P_{2s})Ws \qquad (7)$$

$$L_{P2} = \frac{(P_{1s})}{[P_{2m}]} \qquad (8)$$

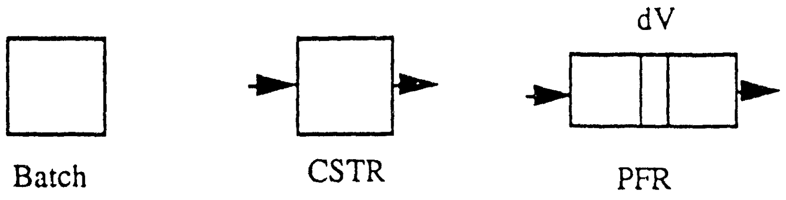
Solving this system of four equations and four unknowns and assuming the $(P_o) = 0$, which means that the flux added will have no phosphorus, one will find:

$$[P_{2m}] = \frac{[P_o]\,W_m^2}{(Wm + L_{P1}Ws)(Wm + L_{P2}Ws) - L_{P2}WmWs} \qquad (9)$$

For example, for $[P_o] = 0.1$; $Wm = 225t$; $Ws = 22.5t$; $L_{P1} = 20$ and $L_{P2} = 100$, the final phosphorus content in the metal will be ($[P_{2m}]$) 0.0044. Using the same equation for sulfur and assuming $L_{s1} = 16$; $L_{s2} = 8$ and case (1): $[S_o] = 0.04$ (without hot metal desulfurization), the final sulfur content in the metal will be ($[S_{2m}]$) 0.01; case (2): $[S_o] = 0.006$ (with hot metal desulfurization), the final sulfur content in the metal will be ($[S_{2m}]$) 0.002.

4- General Behavior of Continuous Processes

A continuous process can be carried out in a continuous stirred tank reactor (CSTR) or a plug flow reactor (PFR). A comparison between batch and continuous reactors (CSTR and PFR) is represented in Figure 2.



Mixing	complete	complete	only in dV
Conc. Gradient	none	none	longitudinal
Conc. Changing	with time	constant	with distant
Reaction speed charging	with time	constant	longitudinal

Figure 2.- Differences between batch and continuous operation.

By performing a simple mass balance in a reactor, one can see that degree of the dispersion in the system defines the type of reactor where the process takes place. The contributions to backmixing of fluid flowing in the x direction can be described by the following equation:(6)

$$\frac{\partial C}{\partial t} = D\frac{\partial^2 C}{\partial x^2} \qquad (10)$$

where the parameter D is called the longitudinal dispersion coefficient, which characterizes the degree of backmixing during flow. Equation (10) can be written in a dimensionless form where $z = (ut + x)/L$ and $\theta = t/t = tu/L$

$$\frac{\partial C}{\partial \theta} = (\frac{D}{uL})\frac{\partial^2 C}{\partial z^2} - \frac{\partial C}{\partial z} \qquad (11)$$

where the dimensionless group (D/uL), is the inverse of Peclet number (N_{Pe}), and measures the extent of longitudinal dispersion. Peclet number represents the ratio of material transferred by bulk flow to material transport by Eddy diffusion. L is the length of the reactor and u is the velocity in the direction of the flow. Therefore,

$\frac{D}{uL} \to 0$ negligible dispersion, PFR

$\frac{D}{uL} \to \infty$ large dispersion, CSTR

It means that when the diffusion plus convection inside the vessel is small the reactor approaches a ideal plug flow reactor. On the other hand, when the diffusion and convection are large, the reactor approaches a continuous stirred tank reactor. Therefore, IRSID(7) and AISI processes approach continuous stirred tank reactor and WORCRA and Bethlehem processes approach plug flow reactor.

5- Continuous Stirred Tank Reactor

Normal BOF and Q-BOP steelmaking are batch stirred tank reactors. For continuous processing the AISI Continuous Steelmaking is a CSTR and the IRSID process a modified emulsion CSTR.

For the continuous stirred tank reactors, it was shown that

$$\%C_f = \frac{P(\%C_0)}{P + k} \tag{12}$$

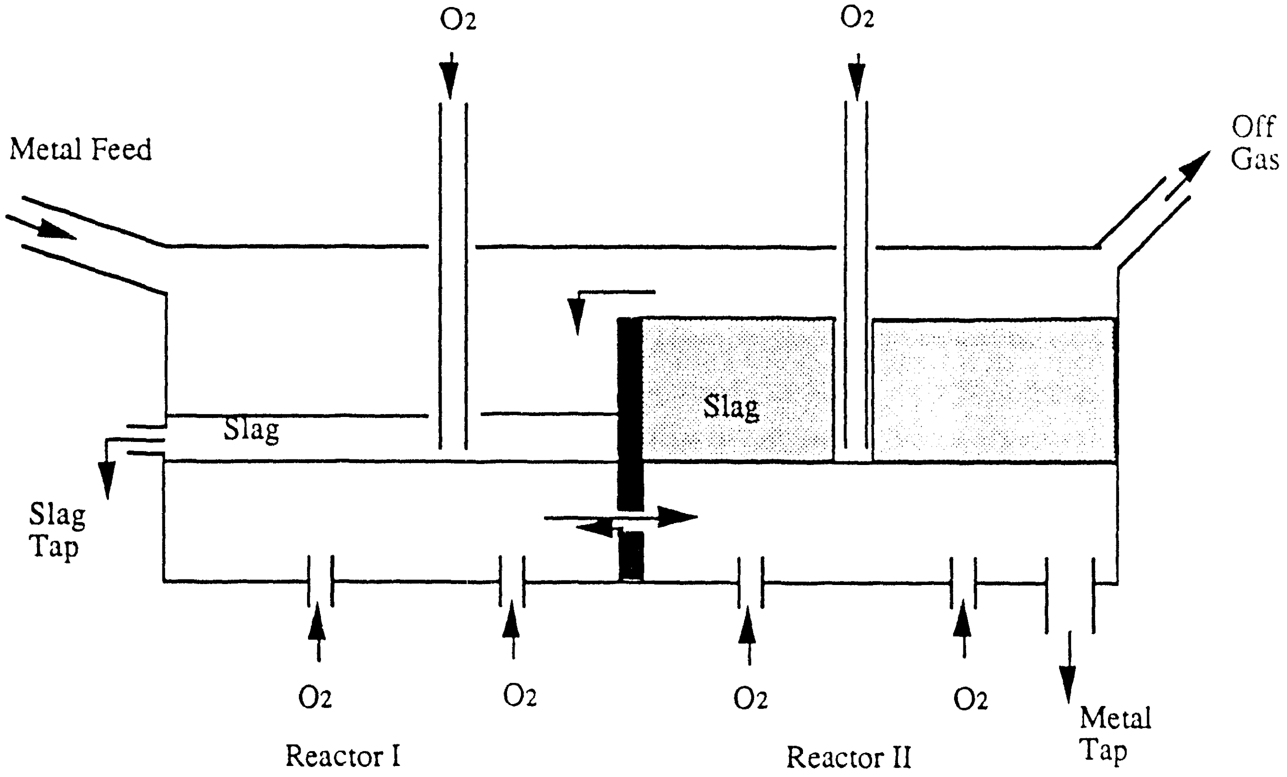
where $\%C_f$ is the final carbon, $\%C_0$ is the initial carbon in the molten metal, k is the decarburization constant and P is the specific production rate $\left(\frac{P'}{W_s} = \frac{\text{Production rate}}{\text{Weight of iron in the vessel}}\right)$.

It was shown that for CSTR reasonable production rates could be achieved. However there would be excessively high yield losses (>25%) because decarburization is being done at a low carbon level, where it is inefficient and excess oxygen reacts with iron. Reducing the oxygen reduces k correspondingly resulting in higher carbon content.

6- Effect of Barriers

The trough process with plug flow can be considered to be an infinite number of CSTR in series. The effect of backmixing in trough processes can be minimized with barriers. A barrier is a partition with an opening for metal flow.

As an example, a commercial steelmaking furnace producing 300t/hr consisting of two 100 tonne CSTR separated by a barrier, will be considered (Figure 3). As shown in actual processes by AISI and water modeling by CMU-USX, there may be considerable backmixing.(8)



Metal Flow

1%C ⇒ $P' + Q_B$

0.05% ⇐ Q_B

Figure 2.- Schematic diagram of a trough process with a barrier.

For such device equation (12) is modified to

$$\%C_f = \frac{(P+\beta)\,\%C_I}{P+\beta+k} \tag{13}$$

where β is the specific backmixing(Q_B/W_V), Q_B is the backmixing (tonnes/hour), $\%C_I$ is the carbon content from the first half of the reactor I.

An analysis indicates that if $\%C_I$ is about 1% the productivity can be achieved. The calculation is not to sensitive to β as long as $\beta << P$; in this case β was assumed to be 0.2 or Q_B = 20 tonne/hour. It was also assumed k = 60 hr^{-1} (1.0 min^{-1}) as in a normal BOF. If the final carbon content is to be 0.05%, the production can be achieved but the iron yield loss to FeO is high (1,300 kg of Fe per minute or about 25%). The high FeO slag does flow counter currently to Reactor I. Not all of this FeO could be reduced in fact the FeO would be supplying 75% of the oxygen required for decarburization, extra oxygen gas maybe required to achieved to the necessary slag-metal mixing. Furthermore, there would be a considerable control problem, this problem can be minimized in three ways: (i) increasing the number of chambers; (ii) increasing the final carbon content; and (iii) using an O_2-Ar(CO_2) mixture in reactor II. For example, by reducing the oxygen blow rate in reactor II by 50 percent, the final carbon content increases to 0.09 percent, and consequently the amount of Fe oxidized decreases to 350 kg/min. A similar result can be obtained by using a 50-50 (O_2-Ar) gas mixture, which would not be economical.

A critical concern would be the size of the opening in the barrier to limit backmixing. At the AISI pilot plant, in actual trials, it was found that

$$Q_B = 300\ A \tag{14}$$

where A is the area of the opening in tonne/m^2.

This was in reasonable agreement with water modeling at CMU-USX.(8) The modeling study indicated that Q_B increased with stirring energy, decreased with forward flow, and depended on the vessel dimensions. Recent work indicates that backmixing will be decreased due to the large net forward flow.

A crude extrapolation indicated that for the 200 tonnes reactor being considered, the area of the opening is very roughly given by

$$Q_B \approx 200\ A \qquad (15)$$

For $Q_B = 20$ tonne/min, $A = 0.1\ m^2$, assuming that the opening would have a shape of a square, it would correspond a square with 31.6 cm side. This backmixing will reduce productivity, decrease yield, and cause control problems.

Backmixing can be reduced by the barrier and backmixing can be reasonably compensated for if the amount is known. The problem is maintaining the barrier, keeping the open area constant, and controlling the process.

7- Dispersion Model

Trough processes do not behave as an ideal plug flow reactor, however a model is available to characterize PFR, where the ideal plug flow of fluid is superimposed with some degree of backmixing. This is called the dispersion model,(9) and it is visually represented in Figure 3.(6)

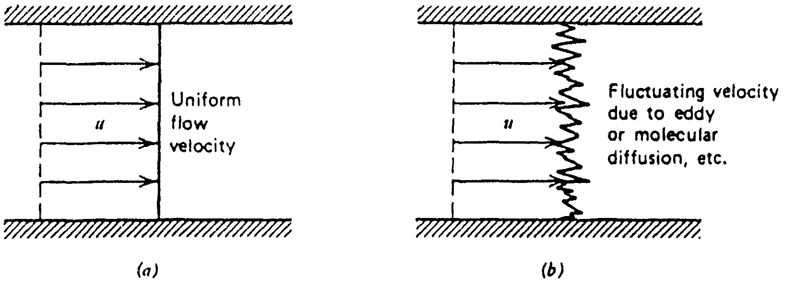


Figure 3.- Velocity profiles in the case of (a) plug flow and (b) plug flow with axial mixing (dispersion model).

7.1- Application in Ferrous Metallurgy

In the pilot plant trials of the WORCRA continuous steelmaking process, a copper tracer was used to determine the metal flow characteristics and an intermediate to large dispersion was observed.(10) This can be seen in Figure

4, where the tracer distribution C curve for the WORCRA process is shown. The second peak in the curve is due to the internal recirculation of the metal in the "bowl" region of the furnace.

The model that best describes this process is tank in series model, which fits the dispersion model for intermediate and large extents dispersion. This model simulates the behavior of the actual vessel by replacing the reactor by ***n*** identical stirred tank reactors operating in series. It is then possible to obtain the value of ***n*** that gives the best fit of the measured tracer curve of the actual reactor. The system is characterized by the variance of the distribution and the inverse of Peclet number (D/uL).

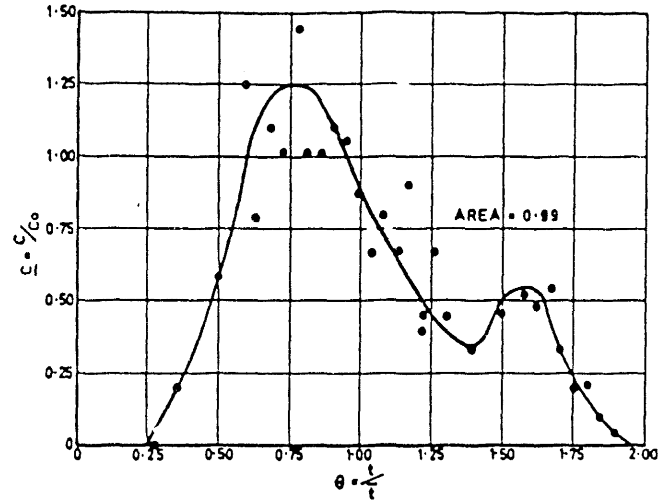


Figure 4.- C curve for metal flow WORCRA pilot plant, Cockle Creek.(10)

For a closed vessel, the variance of the tracer distribution is given by;

$$\sigma_t^2 = \frac{\Sigma ct^2}{\Sigma c} - \left(\frac{\Sigma ct}{\Sigma c}\right)^2 \tag{16}$$

where, **c** is the concentration of trace at time **t** after tracer addition, and

$$\sigma^2 = \frac{\sigma_t^2}{t^2} = \frac{2D}{uL} - 2\left(\frac{D}{uL}\right)^2\left(1 - e^{-uL/D}\right) \tag{17}$$

The number of stirred tanks is given by the following equation

$$n = \frac{1}{\sigma^2} \tag{18}$$

The value of D/uL of 0.064 was found and indicates an intermediate amount of dispersion in the furnace, which equivalent to 8 to 9 stirred tanks in series. It is known that when the number of stirred tanks in series tend to infinite the metal flow tends to ideal plug flow.

Sir Geoffrey Taylor(11) showed that for turbulent flow the Eddy diffusivity is so greater that the molecular diffusion coefficient is not very important. The effect of turbulence is to flatten the velocity profile and reduce dispersion. He also showed that dispersion coefficient D would be given by

$$D \cong 0.6\, a\, u\, Re^{-1/4} \tag{19}$$

where **u** is the average velocity, **a** is half of the width of the vessel and **Re** is Reynolds number. Therefore a large length/width ratio would tend to decrease the influences of both longitudinal diffusion and velocity gradient. A rough generalization that is valid for first order reactions is that if the Reynolds number is greater than 10^4 and if the L/W ratio of the reactor is at least 50, the deviation from plug flow because of longitudinal dispersion may be neglected.

In the WORCRA process, the velocity was approximately 2 ft per min and the actual mean residence time of metal was calculated to be 22.2 minutes, therefore the length of the launder was 44.4 ft. The calculation of the cross section area of the launder is 0.019 m^2 for hot metal flow rate of 5 tonnes/h. Assuming that the cross section area would have a square shape, it would correspond a square with 14 cm side. Therefore the ratio length/width (L/W) would be approximately 97.

In the Bethlehem process(3) the results of a tracer experiment was not reported, however the dimensions of the vessel were given; they were 22' 1" length by an average of 12" width and 4.5" depth. Assuming now that the cross section area of the WORCRA process would have similar proportion between width and depth as the Bethlehem process, the ratio of L/W would be 59. The Reynolds number for the WORCRA process is 9.3×10^2, for density of iron of 7 g/cm^3, viscosity of 6.1 cP, velocity of 1.01 cm/sec and assuming a circular cross-section area, the equivalent radius would be 8 cm.

A rough estimate of the size of 50t/h vessel can be made, by assuming that the production intensity of the WORCRA process is similar to the Bethlehem process 2 t/m^3.h, and that the production intensity is constant. For a 50 t/h production rate, the volume of the vessel needed will be 25 m^3. Assuming a cylindrical shape and keeping the dimensionals proportional to the Bethlehem process, the length of the vessel will be 75.6 ft and the diameter will be 1.9 ft, for the Peclet number calculated for the WORCRA process.

7.2- Application in Non-Ferrous Metallurgy

Several non-ferrous continuous trough processes had been tested on pilot plant scale and few of them can be found in industrial scale, processes such as Noranda(12) (for copper) and QSL(13) (for lead).

The Noranda process consists of lined cylindrical vessel (68ft and 11in length and a diameter of 16ft and 5in), where air or oxygen-enriched air is injected into the matte phase through the tuyeres located along one side of the vessel. It is a counter current process and its operation consists of charging pellets of concentrated and silica flux onto the surface of the slag, blowing air or oxygen-enriched air through the tuyeres. The blister copper is intermittently tapped from the bottom of the reactor.

In order to understand flow characteristics of the Noranda process, tracer experiments were conducted in the water model of the Noranda reactor. In the case of eight tuyeres, uniformly distributed, the inverse of the Peclet number (D/uL) was 0.13 (it corresponds to two CSTR in series), when sixteen tuyeres were used the inverse of the Peclet number was 0.08 (it corresponds to 6 to 7 CSTR in series). These results indicates that the longitudinal dispersion is somehow minimized by increasing the number of the tuyeres.

In order to confirm this finding, a tracer test was conducted in the Noranda pilot reactor, using radioactive copper oxide as a tracer, and no longitudinal concentration gradient was observed. It was concluded that the flow in the pilot reactor could be considered close to CSTR. Tests were also performed in the Noranda prototype reactor using silver as a tracer, it was found that 45 percent of the matte volume behaves as CSTR, 1 percent as PFR, and 54 percent as a stagnant or dead volume. The large dead volume was attributed to the fact that in the prototype reactor the matte layer extends well beyond the active tuyere zone.

In the Noranda process, copper sulfide is oxidized to metallic copper; however this reaction cannot proceed until almost all iron sulfide in the matte is oxidized to FeO. The oxygen injected through the tuyeres is used to oxidized FeS to FeO and S to SO_2, which means that just part of the oxygen introduced to the system will come out as a gas (one mole of oxygen will generate one mole of SO_2). In the case of a continuous steelmaking process, the oxygen used in the process will be used to decarburize the hot metal, therefore for each mole of oxygen injected into the system, two moles of carbon monoxide will be generated.

The QSL process is used for production of lead bullion of low and high lead bearing feed materials. The reactor is divided into 4 zones: an oxidation zone and three reduction zones by three refractory partition walls. Concentrated, residues, fluxes and solid fuel are added through the roof of the oxidation zone. The raw materials fall into a melt consisting of slag and lead bullion, and oxygen is blown into the melt through tuyeres. The lead is gradually reduced from the slag as it flows to the opposite end of the reactor. The slag is tapped at the end of the three reduction zones, whereas the lead settles to the bottom and flows back towards the oxidation zone, where it is discharged via a siphon, whereas the primary slag passes into the first reduction zone.

The partition walls reduce the backmixing caused by the high turbulent bath and also help to keep a concentration gradient along the reactor's length. The partition walls have an underflow for the exchange of slag and metallic lead. The lifetime of these walls is shorter than that of the shell lining, because the partition walls are heated on both sides.

In the oxidation zone, the following reactions take place:

$$PbS + O_2 = Pb + SO_2 \quad (20)$$

$$PbS + 3/2O_2 = PbO + SO_2 \quad (21)$$

$$2PbO + PbS = 3Pb + SO_2 \qquad (22)$$

In the zone the PbO content in the slag is aimed at 25-30 percent. As can be seen from equation (20) to (22) just part of the oxygen injected through the tuyeres is coming out of the process in the gas form, and each mole of oxygen corresponds to one mole of SO_2.

The task of the reduction zones are to reduce the PbO from the slag to Pb. To accomplish this task pulverized coke is injected through the tuyeres with air and oxygen. The reduction mechanism visualized as:

(i) The oxygen, coke and air are quickly heated on the introduction into the bath at the tuyeres, and the coke is burnt to CO.

(ii) The CO bubbles rise into the slag phase and react with the oxides (Fe_2O_3, PbO and ZnO), the reduced Pb enters into the metal phase and Zn fumes and CO_2 rising with the bubbles.

(iii) The zinc, the unreacted CO are then completely and partially post combusted in the gas space above the slag layer, respectively.

No tracer study has been reported with this type of reactor, therefore no information on the flow characteristic of the vessel is known. However, it is expected that the degree of dispersion in QSL process, as well as in the Noranda process, would be smaller than the mixing present in the continuous steelmaking trough process.

8- General Evaluation and Conclusions

It is clear that whereas none of the trough processes worked in the past with enough effort and several constraints they could be made to work. However, there must be an economic driver and there must be a reasonable technical feasibility. They also must be reasonably promising in comparison to other continuous processes. The major economic driver would be reduced capital. The vessel would be stationary and possible with higher productivity due to the time saved associated with charging and tapping.

With regard to the trough processes, a number of issues have been identified including:

- Backmixing - leading to control and yield problems.
- Scrap melting - due to the geometric considerations there will be high heat loss leading to lower scrap melting.
- Multiple oxygen tuyeres or lances - complex control.
- Metal feeding control.
- If the unit is counter current, there is usually 30 percent (10-50 percent) of the metal in the slag phase this will add to backmixing.
- Barrier - If a barrier is used, maintaining it and keeping a constant open area will be difficult.
- Counter current flow trough process is an excellent refiner.

Based on the above concerns and extensive previous work, the potential capital savings most likely do not overcome the drawbacks. Furthermore, in comparison the IRSID process appears more attractive. The IRSID process is a simple high intensity reactor with no backmixing problems, with somewhat simpler control and has proven to be reasonable successful.

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Bethlehem Continuous Steelmaking Evaluation

Process Description:

Hot metal is, charged at one end of the trough-like furnace, continuously decarburized by progressive oxygen lancing of the metal. Fluxes and coolants are also added through lances along the length of the furnace. The slag and metal flow concurrently and are discharged at the opposite end of the reactor. The carbon level in the product is controlled by manipulation of the oxygen blowing rate. A schematic diagram of the Bethlelem Continuous Steelmaking Process is shown in Figure 1.

Development Status:

The pilot plant work was done in a 5 t/hr unit at the Bethlehem Steel Corporation Research Center, Homer Research Laboratories.

Critical Issues:

Productivity: Not available.

Yield: Not available.

Refining: The progressive oxygen blow removes silicon, carbon, manganese, phosphorus and sulfur as a function of distance along the furnace. The carbon level varies from 0.03 to 1.1 percent, which are similar or higher than in the BOF. The highest rate was found to be at the two initial feet of the charging port, however carbon is removed throughout the length. And most of the silicon is removed in the first 5 ft. The manganese level is similar to the BOF, and most of its removal occurs in the first 5 ft of the furnace. The phosphorus level is also similar to the BOF, however it is claimed that a slight reversion occurs at the end of the process. The sulfur level is 2 to 4 times higher than in the conventional BOF. No information is available on the nitrogen levels.

Scrap Melting: The coolant (ore/metallized pellets/scrap) is added through lances, its size should be in average 14 mesh.

Metal Feeding and Control: The feeding practice was far from satisfatory. Initially, the hot metal was charge from a lip-pouring ladle into a preheated tundish with an alumina nozzle. However, the formation of skulls across the top of the tundish made the visual observation of the metal surface very difficult, and consequently fluctuations in the metal level and in the feed rate were observed. Then, they tried to charge the hot metal directly into the furnace from the ladle, but slag built up on the lip of the ladle caused significant fluctuations in the feed rate. A modified teapot spout was then used, where a steel pipe imbedded in the refractory from the lid down into the ladle, well below the metal surface. The metal feeding rate was determined with a crane and regulated with a hand wheel.

Refractories: In the first campaign, a severe erosion at the slag line near the first two oxygen lances was found. The problem persisted in the second campaign, however no as severe. To extend the furnace life, the furnace was sprayed with high magnesia mix after each run.

Metal and slag Tapping: A slag skimmer was used to divert the slag over a dam approximately 1/2 in. higher than the metal dam. They claimed a succesful separation, with clean metal flowing from one channel and slag flowing from the other.

Gas Removal: At steady state little fume was observed from the roof of the furnace, however a heavy dust gas stream was discharged from the flue. Fume losses were lower than in the conventional processes.

Environmental Concerns: It is expected to be similar to the conventional BOF.

Technical Assessment

This process has several drawbacks, and therefore is not recommended to be pursued any further. Their drwbacks are:

1. Backmixing effect associated with the processes that attempt to approach plug flow.

2. Large energy losses, due to the large surface area of metal refractory contact.

3. Smaller size of scrap should be used because of not enough energy is available for melting and channel dimensions are inconveniently small.

4. Phosphorus reversion at the end of the refining process, due to lower slag basicity, low oxygen blowing rates, lack of slag-metal dispersion, and increasing temperatures.

Reference:

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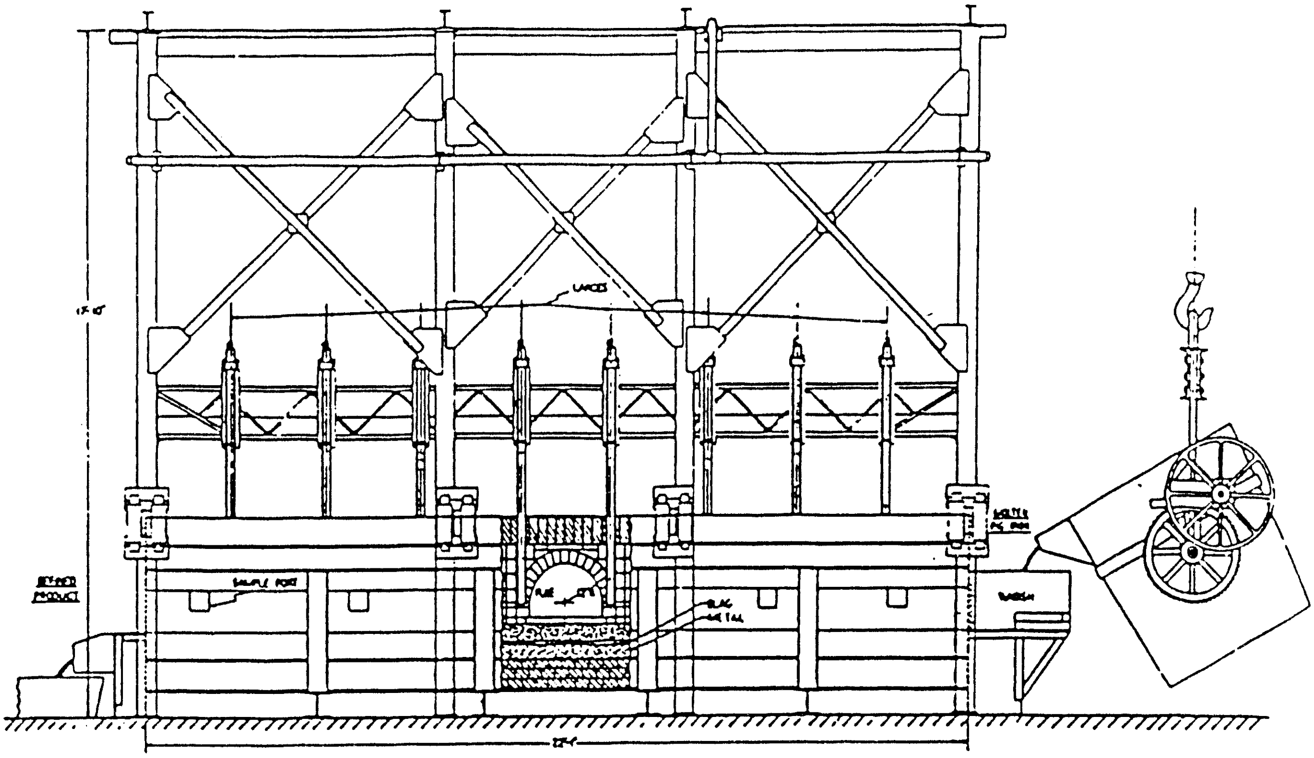
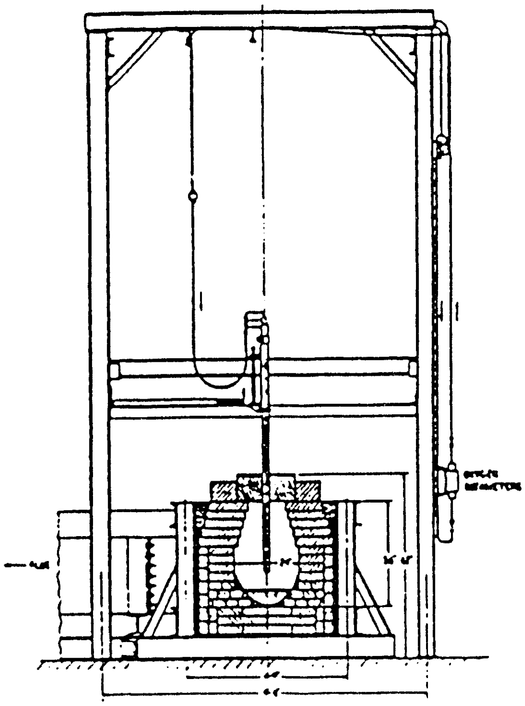


Front Elevation



Side Elevation

Figure 1 .- Continuous steelmaking forehearth, 5-thp capacity.(1)

NRIM Continuous Steelmaking Evaluation

Process Description:

The NRIM consists of a multi-stage trough, where the steelmaking reactions are separated into three groups, desiliconization and dephosphorization, decarburization, and the final control of steel grade. The first and second trough units have settling flow zone on which metal-slag separation takes place. The hot metal is continuously fed into the first trough, where all the four lances are used for oxy-flux injection, in this stage all the silicon and up to 84 percent of the phosphorus are removed. The metal is then fed into the second trough, where two of the lances are used for oxy-flux injection and the remaining lances are used for oxygen blow. In this stage, the rest of the phosphorus is removed and decarburization takes place. The steel is fed into the third and last trough, which is used as a grading furnace, and from there the steel is tapped. The slag produced in the first and second stages flows into their respective slag bucket. Figures 1 and 2 show the NRIM continuous steelmaking furnace.

Development Status:

The pilot plant facilities consisted of three trough furnaces at 7.8 t/hr of hot metal flow rate, and it was located at the National Research Institute for Metals, Japan. The first trough furnace was used as a channel for transporting hot metal and the last two furnaces were used to fill the roles of the first and second trough.

Critical Issues:

Productivity: Not available.

Yield: No reference is made to the yield of this process, however each unit has its settling zone to promote slag-metal separation and two built skimmers, one at the entrance and the other near the tap hole.

Refining: The manganese level is higher than in the BOF, phosphorus level will be lower, since in this process hot metal and steel dephosphorization are accomplished in the first and second stages, respectively. Sulfur level is about 3 times higher than BOF level, however, hot metal desulfurization could be included before this process. The carbon level is higher that in a BOP process (0.06 percent). Nitrogen level is unknown.

Scrap Melting: Some light and small scrap could be added in the first trough, however, this practice was not tried before.

Metal Feeding and Control: The metal should be fed into the reactor at a constant rate. The holding furnace was equipped with load cells measuring its gross weight and feeding rate of the hot metal was regulated by tilting the furnace in accordance with the prescribed time-weight diagram. However, feeding rate by manual control method deviates considerably (± 50 kg/min for a feeding rate of 130 kg/min). Therefore a tundish was designed to keep the deviation always within ±10 percent for the aimed value of 130 kg/min.

Refractories: Refractories in the blowing zone eroded physically by vigorous movement of hot metal and chemically by emulsified slag. The refractories in the settling zone were chemically attacked only at the slag line. The skimmers suffered from the more severe attacks because their both sides are exposed to slag and metal. Construction problems occurred due to the trough rectangular shape.

Metal and slag Tapping: All the three vessels have a side wall tap hole, as shown in Figure 1. They also have a slag-off port and a skimmer, to decrease the slag carried over. The metal is being continuously tapped during the process.

Gas Removal: The flow rate of waste gas was not measured, however it is expected to be at the same order as in the conventional BOF.

Environmental Concerns: The environmental concerns with the NRIM process would be similar to a BOF. Due to the continuous nature of the process, the gas cleaning system could be smaller than in a BOF.

Technical Assessment

The small scale pilot plant and short running time of about 100 minutes do not answer all the technical questions satisfactory. The significant heat losses through the walls of the vessels, refractory construction problems, waste gas flow rate were problems never approached. A considerable amount of pilot plant work will be necessary to answer these questions and clarify the process as a whole.

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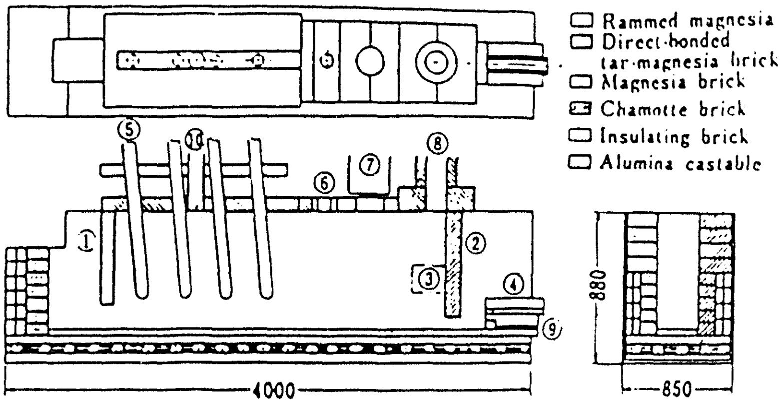

1. Inlet skimmer
2. Outlet skimmer
3. Slag-off port
4. Overflow dam
5. Lance
6. Gas sampling hole
7. Burner
8. Flue
9. Tap hole
10. Coolant feeding chute

Figure 1.- Construction of NRIM continuous steelmaking furnace (unit furnace).(1)

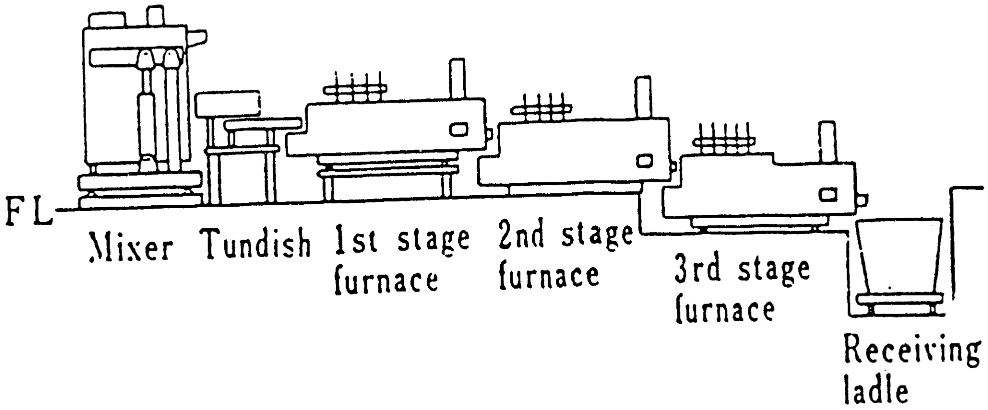

Figure 2.- Multi-stage continuous steelmaking equipment.(1)

WORCRA Continuous Steelmaking Evaluation

Process Description:

Hot metal is continuously fed into the circular zone, where slag (from the refining zone) and metal flow concurrently. In this zone, most of the desiliconization and part of the desulfurization and dephosphorization take place. From the circular zone the metal goes to the refining zone, where the decarburization and the remaining dephosphorization take place. The steel is then tapped. The slag generated by flux addition, at the far end of the refining zone, flows back slowly countercurrent to the metal. This slag changes its character and composition as it flows back towards the high carbon circulating zone. The slag, from the circular zone, goes to the slag clean-up zone where any entrained shot can settle and FeO in the slag can react with any reactant added in that section. The floor of the slag clean-up zone is raised relative to the rest of the furnace bath bottom towards the circular zone, so that any metal, that settles out, is automatically returned to the turbulent circulating bath of the refiner. Figure 1 shows a schematic design of the WORCRA Continuous process.

Development Status:

The WORCRA pilot plant treated 4 to 5 tHM/hr. Figure 1 shows the schematic representation of WORCRA pilot plant furnace at Cockle Creek, Australia.

Critical Issues:

Productivity: Not available.

Yield: The Worcra process claims high metal recovery, usually between 98-99 percent. This is possible because the FeO and Fe (shot) lost in the slag were very low . The fume was also considerably less than in the conventional BOF.

Refining: Sulfur removal equal to the ones achieved in BOF practice, however it will be obtained with half lime used in the conventional processes. Phosphorus refining was better the one obtained in the conventional BOP and

Q-BOP operation. Manganese level is similar the ones achieved in the conventional BOF, however the carbon level is higher (from 0.52 to 0.16 percent). Nitrogen level is unknown. Oxygen efficiency is higher than the BOF (15 to 20 percent less oxygen than is required for BOP equivalent), because of the reuse of some of the oxygen which has gone into the slag as FeO , and is recovered in the decarburization reaction when the slag flows towards the circular zone of the refiner.

Scrap Melting: The particulate (minus 1 in.) scrap coolant can be added in a similar manner as the flux. Which is via two ports, one over the circular zone and the other over the refining zone. The furnace could consumed 14 percent cold scrap. It is expected that the extent which scrap can be used will probably be lower than in the BOF, because of the very extensive cooling surface of the refining channels.

Metal Feeding and Control: Due to the unsteady rate of hot metal flow from the cupola, used as the production source, it was not easy to maintain steady carbon levels in the output steel over long period of time. To solve this problem it was suggested: (i) a tilting holding vessel; (ii) good quality nozzles in a constant head tundish and/or; (iii) an electromagnetic 'pump'.

Refractories: Cooling pipes were inserted at the slag line in all regions where the erosion was significant (these regions were not specified in the papers). The lining contour was then stabilized and the only refractory troubles were those associated with the usual shrinkage cracking of highly basic refractories when they were cooled at the end of a campaign. During furnace operation there is no temperature cycling, condition which favors longer refractory life.

Metal and slag Tapping: The presence of a slag dam at the steel outflow end enables the tapping of metal relatively slag free. At the slag clean-up zone, it is possible to tap the slag with low FeO and Fe, since most of the metal shot can be settled and the FeO can be reduced by adding a doxidizer.

Gas Removal: The gas cleaning system is expected to be smaller and simpler than in the conventional BOF. It is also expected that considerably less fume will be generated in this process, because of the better 'distribution' of the exothermic heat under the several (smaller) oxygen jets and partly because the backward-flowing slag is vigorously foaming and acts as a 'scrubber', collecting the dust.

Environmental Concerns: It is expected to be similar to the conventional BOF.

Technical Assessment

This process has critical short-comings, therefore it should not the consider as an alternative route to continuous steelmaking. Some of its most critical limitations are:

1. It is not an autogenous process, because of the large energy lost due to:
    (i) the entrainment of cold air from the atmosphere;
    (ii) very extensive cooling surface of the refining channels because their lengthiness.

2. It is impossible to use ordinary size scrap, because the size of the channels and energy limitation.

3. Backmixing effect associated with no plug flow type of furnace.

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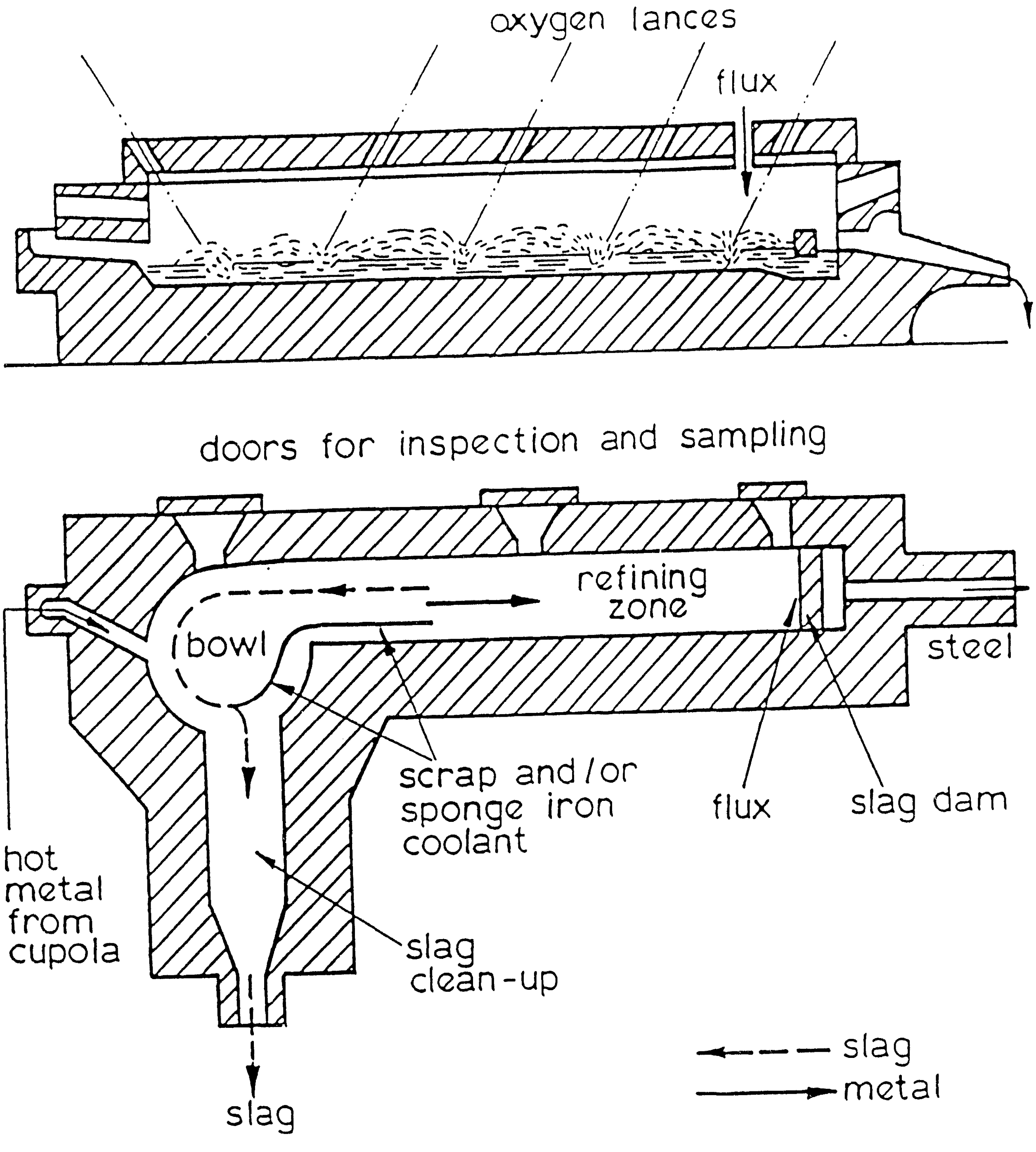

Figure 1.- Sectional plan and elevation of L-shaped pilot plant WORCRA Steelmaking furnace.(1, 2)

COSMOS Continuous Steelmaking Evaluation

Process Description:

The cosmos steelmaking furnace consists of three-zone trough-type vessel for smelting and refining. Separation of these zones may be accomplished by baffles or simply by physical length and control of flow of the slag and metal. The hot metal or hot prereduced pellets from the shaft furnace are top charged into zone 1 of the three-zone trough furnace where coal, oxygen, and fluxes are injected through bottom-mounted tuyeres to smelt and reduce the pellets to molten iron. Since, it is possible to deslagging in each zone of the furnace, partial dephosphorization is accomplished in the first zone. Zone II is designed primarily for sulfur removal and possibly phosphorus by adding fluxes into the bath. Also, in the second zone some oxygen may be blown, therefore desiliconization and decarburization can be achieved. In the third zone, a mixture of oxygen and argon is bottom injected to attain the desired final carbon level in the steel with a minimum flux addition. The refined steel is then tapped into a conventional ladle. Figure 1 shows a schematic diagram of the COSMOS process.

Development Status:

A COSMOS prepilot reactor (15tHM/hr) was constructed at Universal, Pennsylvania (see Figure 2). In this reactor just decarburization reaction was investigated. It is claimed that the continuous decarburization of iron containing 2 to 3 percent carbon was achieved. They also claimed that all four stages (prereduction, pellet smelting, iron desulfurization, and continuous decarburization) have been individually demonstrated, and they could be combined into a viable process.

Critical Issues:

Productivity: Not Available.

Yield: Not available, however it is expected to be lower than in the conventional BOF. Slag foaming was observed during COSMOS trials. This process does not have a settling vessel, therefore most of the metal entrainment in the slag will not have time to decant before tapping.

Refining: In the prepilot reactor just decarburization was investigated. Backmixing was found to be a problem in the process, therefore the desired low carbon levels were never obtained.

Scrap Melting: The hot metal used had low silicon content (>0.05%), consequently scrap melting was never tried. It is expected that small size scrap could be used, however the extent to which scrap can be used will probably be lower than in the BOF, because the large heat loss associated with the trough process.

Metal Feeding and Control: The metal charging rate was measured by load cells which support the ladle platform and was controlled by trottling the slide gate. The metal from the ladle passed through a shroud and down a feed runner to a reservoir, which fed the reactor through a 4-inch diameter horizontal hole located at the reactor floor level.

Refractories: The erosion of the refractory did not seem to be a problem, however the longest trial last ninety minutes . They had experience refractory construction problems due to the rectangular shape of the trough.

Metal and slag Tapping: The slag and metal were tapped through the same tap hole at the end of the vessel.

Gas Removal: The exhaust system installed on the reactor did not effectively remove the fumes released during charging and tapping. Heavy fuming was observed while the reactor was being filled, which settled to minimal amounts during operation. The heaviest fuming occurring while draining the reactor.

Environmental Concerns: It is expected to be similar to the conventional BOF.

Technical Assessment

Besides all the short-comings associated with trough-type of process, such as: backmixing effect, large energy losses, impossibility of using ordinary size scrap. Trials with the three zone COSMOS process was never attempted. The prepilot reactor just investigated the decarburization reaction, where problems with backmixing, and refractory design limited the investigation.

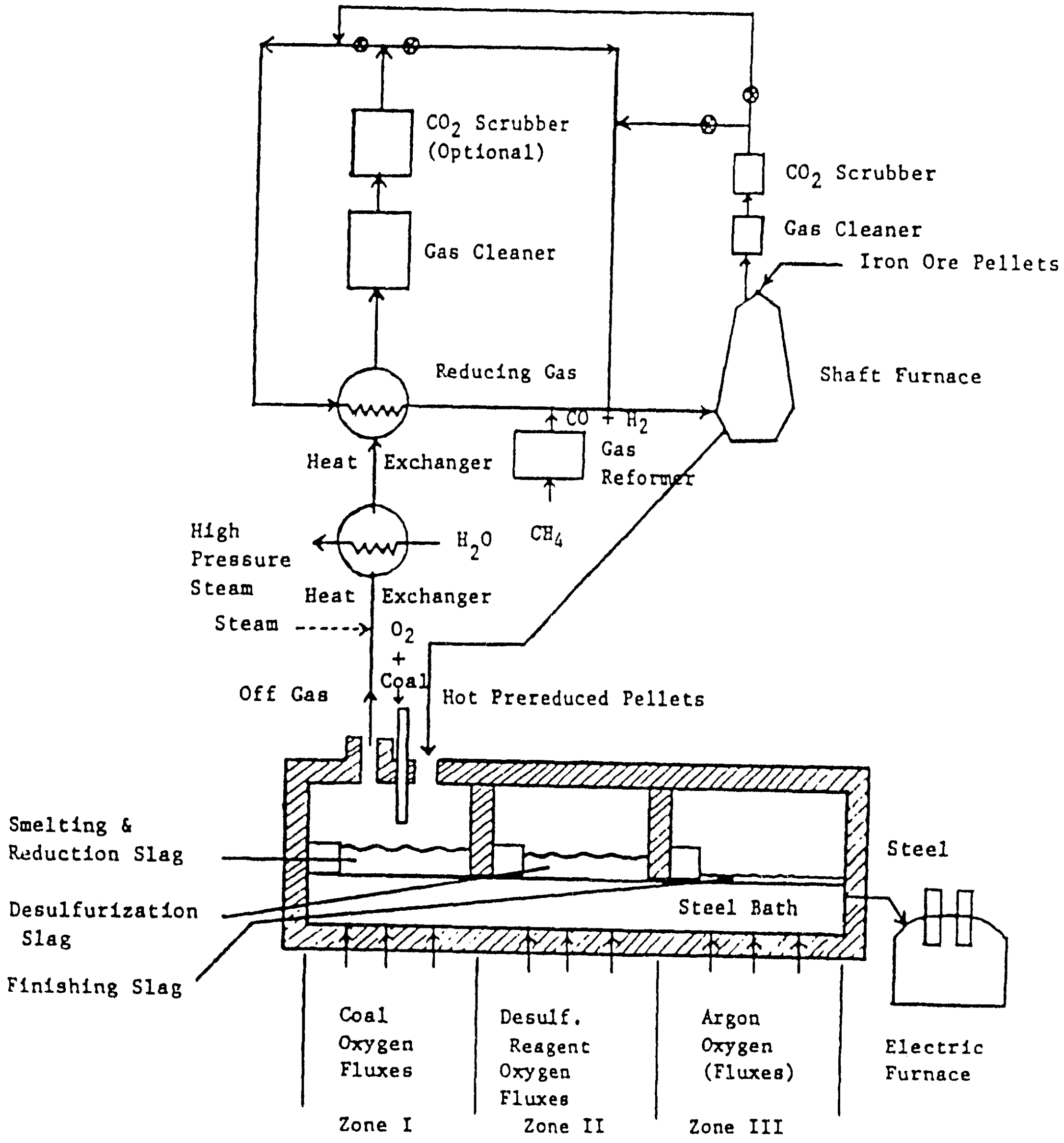

Figure 1.- COSMOS Process flow diagram

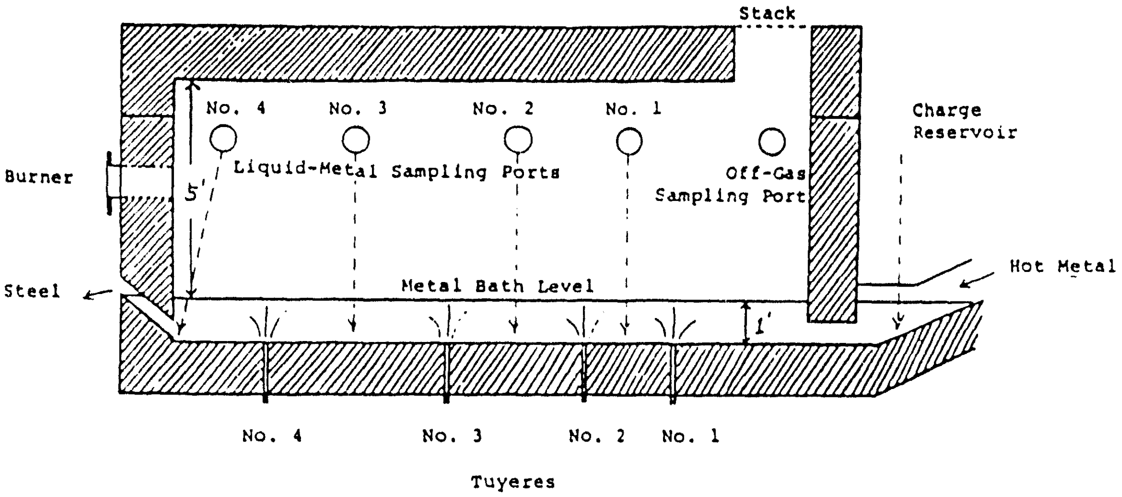

Figure 2.- COSMOS prepilot reactor .

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IRSID Continuous Steelmaking

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Of the existing continuous steelmaking processes the IRSID process was the most extensively developed and is one of the most promising. The process was tested on a 10-12 tonne per hour scale at IRSID at Maizieres-les-Metz and up to 25 tonnes hour at Hagondange. Despite their publications and discussions with IRSID no model or full understanding of the process is available to evaluate the process. In order to make such an evaluation a process model has been developed.

Reaction mechanism in normal OSM

In normal batch oxygen steelmaking, such as the BOF, the oxygen jet and reaction gases cause an extensive slag-metal emulsion with about 30-35% of the metal in the emulsion. The rate of decarburization at high carbon levels is limited simply by the oxygen blowing rates. Below some critical carbon content, about 0.3%, the rate is limited by liquid phase mass transfer. The rate of mass transfer is first order with respect to carbon content. The metal drops in the emulsion are decarburized to some extent and then enter the liquid metal pool in the bottom of the furnace. The average carbon content of the metal drops is slightly less than in the bulk metal and, therefore, the rate of decarburization can be related to the bulk metal concentration. The production rate for a given furnace is limited by the rate of gas removal. That is, there is a free volume requirement for a given rate of gas generation.

Reaction mechanism in the IRSID Process

The reaction mechanism for the IRSID Process is not known. As the metal enters the reactor it is hit with an oxygen jet causing nearly all of the metal to be emulsified. Observations indicate there is no liquid metal pool. In this case the metal drops remain in the emulsion during their entire retention time in the reactor. The rate depends on how long the drops are in the reactor since their carbon content is changing with time. The metal which overflows into the decanter is the average carbon content of the metal in the reactor. The retention time of the metal drops in the reactor depends on the reactor size.

IRSID Process Reaction Model

Obviously the IRSID process is more complex than a normal BOF and no model exists for the process. In order to extrapolate the results obtained by IRSID and evaluate the process a reasonable model is essential. The assumptions for the present model are given below:

1. The rate of decarburization of the metal drop is controlled by liquid phase mass transfer of carbon.

2. There is always sufficient oxygen present as oxygen gas or FeO in the slag.

3. The metal drops react independently and do not combine.

4. It is recognized that there is a distribution of metal drops in the emulsion that are decarburized at different rates due to differing area to volume ratios. However, it is assumed that the average rate of decarburization for the weight average size droplets adequately describes the system.

5. The carbon content of the metal leaving the reactor is the average of all the drops in the emulsion.

6. There is sufficient slag to hold all of the metal in the emulsion. It is recognized that whereas the AISI smelter produces metal with no silicon, it is assumed Si may be added to enhance desulfurization prior to steelmaking. Silicon could be added in any case and, therefore, it is assumed there is adequate slag volume.

The rate of decarburization of a single droplet is controlled by mass transfer and the carbon content is given by:

$$C_D = C_o \, e^{-kt} \tag{1}$$

where C_D = carbon content of a drop after time t

C_o is the initial carbon content

k is the overall rate constant for mass transfer

$$K = \frac{A_D \rho m}{W_D} \tag{2}$$

A_D = area of drop

m = mass transfer coefficient

ρ = density of the metal

W_D = weight of the drop

The reactor is not plug flow but rather a continuous stirred tank reactor (CSTR). That is, some of of the particles only react for a short time and may have a high carbon content while others are very low in carbon. For a CSTR the residence time distribution f(t) is given by:

$$F(t) = \frac{1}{t_r} e^{-t/t_r} \tag{3}$$

f(t) is the faction of drops in the bath for time t and t_r is the average residence time. The average or final carbon content of the metal leaving the reactor is given by:

$$C = \int_0^\infty C(t)f(t)dt \tag{4}$$

$$C = \frac{C_o}{t_r} \int_0^\infty e^{-(k + \frac{1}{t_r})t} dt \tag{5}$$

$$C = \frac{C_o}{t_r k + 1} \tag{6}$$

Equation (6) is a very simple but useful expression for evaluating the IRSID process. It is interesting to note that the rate of decarburization is reasonably fast and most of the drops are fully decarburized. The carbon in the final metal is the result of the drops which only spend a short time in the reactor.

Evaluation of (k) and (t_r)

In order to use equation (6) knowledge of k and t_r is required. k can be evaluated from normal BOF operations. At the end of the BOF cycle, at low carbon, the following relationship is reasonable.

$$C = C_c e^{-k'(t - t_c)} \tag{7}$$

where k' is the overall decarburization rate in the BOF, C_c is the critical carbon content and time when the rate is controlled by mass transfer and is first order.

$$k' = \frac{A_D' \rho m}{W} \tag{8}$$

where,

A_D is the area of drop in the slag in a BOF.
W = the weight of the metal in the BOF

For typical oxygen blowing rates and slag volumes in a BOF the value of k' is about 0.017 s^{-1}.

In comparison to the BOF in the IRSID process all of the metal is in the emulsion. This causes A_D

to increase correspondingly. With all the metal rather than 30% in the emulsion the value of k is estimated to be 0.055 s^{-1}.

IRSID estimated the average residence time in their experimental reactor to be 6 to 7 minutes and scaled the 50 tonne/hour vessel accordingly. This residence time was not supported and an accurate value is required for this evaluation.

It has been found that there is a required reactor volume to hold the slag metal emulsion in a BOF which depends on the rate of CO formation. For 50 tonnes per hour removing 4% carbon the volume of CO and CO_2 at 1923 K is 7.3 m^3/s. Based on normal BOF operations this required 5.5 m^3 which is larger than the reactor proposed by IRSID. Therefore, the entire reactor volume is full of emulsion and the gas is removed at a faster rate than a normal BOF.

The reactor volume at any time is filled with slag, metal and gas. In normal BOF, 70% of the emulsion is gas. In the IRSID process, it may be even higher due to the faster rate of gas generation. If the gas occupies 70% of the volume, the retention time of the metal and slag is given by:

$$t_r = \frac{0.3\ V_R}{\dot{V}_s + \dot{V}_m} \tag{9}$$

where,

V_R = Reactor Volume

$\dot{V}_s$ = Volume flow rate of slag

$\dot{V}_m$ = Volume flow rate of metal

Assuming 100 kg of slag per tonne of metal and 50 tonnes/hour, t_r is estimated to be about 5.5 minutes. This is considered the highest retention time and it could be significantly smaller (~ 3 minutes). In fact, if only the volume to the overflow is considered, the retention time is about 3 minutes.

For an average residence time of 3 minutes (180 s), input carbon of 4.5%, and a decarburization constant of 0.055b s[-1] the final carbon content would be about 0.4%. If the residence time is 6 minutes (360 s) the final carbon level would be 0.2%. t_r can simply be increased by increasing the volume of the reactor. For the above calculations, the largest uncertainty is in t_r and this should be examined further using actual plant data or modeling techniques.

Evaluation of IRSID Results

Despite the publications and correspondence with Usinor Sacilor, details of their tests are not known. From diagrams the size of the reactor at Hagendange was estimated to be 0.70 m^3. Assuming 70% of the volume is gas at 20 tonnes per hour the residence time is about 3 minutes. They indicate that for the low P and Si hot metal, the initial carbon was about 1.2%. Using equation (6), and 0.055 s^{-1} and 180 s respectively for k and t_r the final carbon content would be about 0.1% as observed. They also indicate that metal containing 3.8% carbon was also refined but with no details. In this case, equation (6) predicts about 0.35% C as compared to about 0.09% reported.

Decanter

The function of the decanter is to have slag-metal separation after the reactor. This has apparently been scaled according to the retention time, i.e. volume is in proportion to volume. The actual slag-metal separation process is not clearly understood. It may be that larger bath metal depths may require longer retention times since the separation distances are greater. Therefore, the scaling up of the decanter may be more complicated and further work would be useful.

Conclusion

A reactor process model has been developed to evaluate the IRSID process. However, more detailed information is required to fully evaluate the process accurately. In particular, the operating data from IRSID and a better evaluation of the retention time. However, the model is useful for scaling results and evaluating the process. In particular,

1. The final carbon content depends on the initial carbon content. A certain percentage of the initial carbon remains.
2. The amount of carbon removed increases with retention time. The retention time is simply proportional to the reactor volume and inversely to the production rate.
3. By this evaluation the proposed reactor may be too small to produice 50 tonnes/hour.
4. The scaling for the decanter seems reasonable. However, a better understanding of the process would be useful for it to be as efficient as possible, i.e. the highest production for a given decanter size and geometry.

IRSID Continuous Steelmaking Evaluation

Process Description:

Hot metal is continuously fed into a reactor. An oxygen jet impinges onto the metal causing almost all of the metal to enter the emulsion. The emulsion overflows into a decanter or settling vessel in which slag-metal separation takes place and the metal is tapped.

Development Status:

The IRSID process is most likely the most fully developed continuous process. At IRSID a facility consisting of one or two refining stages treated metal at 10-12 tonnes per hour. A 25 tonne pilot plant was run from 1970-73 at Hagendange.

Critical Issues

Productivity: As discussed, the production rate for a given reactor may be slightly less than projected by IRSID. However, due to the intensified reaction and continuous nature of the process the productivity per unit volume will be at least twice that of a BOF.

Yield: IRSID claims upwards of 97% yield. Iron losses to the slag occur when the oxygen flow rate exceeds the equivalent rate of decarburization. If these are carefully matched yield loss would be similar to a BOF.

Refining: Sulfur and manganese levels will be similar to the BOF since these follow equilibrium. Phosphorus is limited in OSM by kinetics and a model similar to that for decarburization would apply. Dephosphorization may require longer retention times than anticipated with the proposed reactor producing 50 t/hr. As with decarburization, phosphorus removal can be improved by increasing the reactor volume. Nitrogen removal is unknown and may be less than in a conventional BOF. This requires a detailed examination.

Scrap Melting: The amount of scrap which theoretically can be melted is simply determined by the heat balance and can easily be computed. However, due to low retention times the size of the scrap may be limited. The scrap should also be added nearly continuously for a stable operation; this may be difficult.

Metal Feeding and Control: The metal must be fed into the reactor at a constant known rate. The crude system used by IRSID is adequate for pilot studies but a more sophisticated and precise system is required.

Refractories: The IRSID results indicate a high refractory consumption in the jet impact area. On the other hand, refractory consumption may be less per ton than a conventional BOF because of less temperature cycling. Refractory consumption can only be accurately assessed with a long pilot plant campaign.

Metal and Slag Tapping: The decanter is ideal for submerged tapping as in an EBT (EAF) or by a slide gate type mechanism. the metal can be tapped relatively slag free, which is desirable for clean steel, continuously or intermittently. The slag could be tapped periodically through the same or separate tap hole.

Gas Removal: The gas removal rate from the reactor is relatively high and would carry excessive dust with it. However, the dust could settle out in the decanter and the fume loss would be less than in a BOF.

Environmental Concerns: The environmental concerns with the IRSID process would be similar to a BOF. The gas cleaning system could be 50% smaller than in a BOF due to the continuous nature of the process. That is, the gas is being removed continuously as opposed to a BOF in which there is a peak removal rate for only about one third of the time.

Technical Assessment

Based on the extensive work done by IRSID it is reasonable to conclude the process should work as envisioned. However, several critical aspects should be examined before full commercialization. These concerns include:

1. The relationship between production rate, reactor volume and retention time.
2. The minimum size and optimum design of the decanter.
3. Metal feeding system and control.
4. Scrap feeding.
5. Nitrogen and phosphorus refining.
6. Refractory consumption.

With a carefully planned pilot plant program these issues could be addressed with a minimum amount of trials. If continuous refining, in general, if attractive the IRSID process should be considered a leading candidate.

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EAF STEELMAKING USING FEED MIXTURES OF SMELTER HOT METAL, SCRAP AND DIRECT REDUCED IRON

Process Description

Smelter hot metal (HM), mixtures of various kinds of scrap (SM) and, possibly, direct reduced iron (DRI) are charged to an electric arc furnace (EAF) and melted. Fluxes and coal or coke as well as oxygen and natural gas are added as in conventional modern 100 percent scrap-based EAF steelmaking. Tapping of the liquid steel is conventional and is, according to modern technology, achieved by eccentric bottom tapping (EBT), thus minimizing furnace slag carryover into the tap ladle.

Critical Issues

- Productivity

  At least as good as modern 100 percent scrap-based UHP-EAF steelmaking, may be somewhat higher depending on the relative quantity of HM charged. In the case of relatively large quantities of DRI in the charge, productivity may be somewhat impaired.

- Yield

  As in conventional 100 percent scrap-based EAF steelmaking, depends on the specific charge composition; this will be addressed in more detail later.

- Refining

  Provided the EAF shop is equipped with a sufficiently large gas exhaust system, the decarburization rate (thus productivity) should not be limited by the supply of oxygen. For example, a 200 tonne heat of which 60 percent is HM (120 tonne) requires an oxygen blowing rate of about 400 Nm^3/min. It is noted that Timken's new EAF shop is equipped with a gas exhaust capability of about 1000 Nm^3/min.

- Metal Feeding and Control

  Scrap and DRI may be fed continuously (Consteel process). HM is fed intermittently. It has been shown[1] that the continuous feeding of HM to the EAF leads to unacceptably high iron yield losses. The combination of continuous scrap feeding and intermittent feeding of HM may require special handling equipment as yet unknown in conventional EAF shops.

- Refractories

  The processing of relatively large amounts of HM (up to 60% of the charge) may have an adverse effect on refractory life as compared with conventional EAF practice. For smaller amounts of HM in the charge (e.g. 25%) refractory life is expected to be similar to that in conventional EAF practice.

- Metal and Slag Tapping

  Good separation of metal and slag is achieved by the use of EBT or a submerged tap hole, as in Timken's new EAF shop.

- Gas Removal

  The EAF must be equipped with a sufficiently large gas exhaust and cleaning system to be able to handle the large quantity of off-gas generated by feed mixtures containing 60% HM. Modern EAF shops such as Timken's Faircrest plant have this capability.

- Environmental Concerns

  The dust generation rate in a BOF is about 2 percent of the tap weight and, thus roughly equal to that in an EAF. Hence, EAF feed mixtures containing large amounts of HM are not expected to generate more dust than a 100% scrap feed. However, the dust generated by a 60% HM + 40% SM feed mixture is expected to be classified as hazardous, like conventional EAF dust.

Technical Assessment

Feeding an EAF with mixtures of HM, scrap and DRI and melting and decarburizing such mixtures is technically feasible. In fact, Iscor has been feeding up to 46 percent Corex hot metal to an EAF together with scrap and published some results.2)

For easy reference, a Table containing a summary of the most important data estimated for an EAF fed with the indicated mixtures of HM, scrap and DRI is included (p. 3). A more detailed discussion of these data and the assumptions on which they are based is presented in the following sections.

SUMMARY OF MOST IMPORTANT DATA

	STEEL AT TAP				ENERGY,[6] $\frac{\text{kWh}}{\text{tonne}}$	
EAF FEED	[%P]	[%S][4]	DESULF.	$\Delta N/N_0$ (%)[5] RANGE	SCRAP PREHEAT	
					NO	TO 500 °C
60% HM[1] + 40% SM[2]	0.006	0.08	HM, STL.	78-99.9	160	130
60% DRI[3] + 40% SM	0.008	0.014	STL.	35-88	490[7]	460[7]
30% HM + 30% DRI + 40% SM	0.007	0.05	HM (?), STL.	62-99	330	300
25% HM + 75% SM	0.006	0.05	HM (?), STL.	46-96	310	250
25% DRI + 75% SM	0.007	0.02	STL.	16-59	440[7]	380[7]

1) HM: SMELTER HOT METAL (5%C, 0.15%S, 0.023%P)

2) SM: SCRAP MIXTURE OF 50% NO. 1 BUNDLES, 50% NO. 1 BUSHELING (0.025% (MAX.)S, 0.023 (MAX.) P, 2 σ LIMITS)

3) DRI (HBI): 6.8% FeO, 3.85 GANGUE

4) [%S]: STEEL SULFUR CONTENT WITHOUT DESULFURIZATION OF EITHER HM OR STEEL

5) $\Delta N/N_0$ (%): NITROGEN REMOVAL RELATIVE TO NORMAL EAF PRACTICE (BASED ON CO EVOLUTION)

6) ESTIMATED ELECTRICAL ENERGY REQUIREMENT ON THE BASIS OF: 1 TONNE OF LIQUID STEEL + 100 KG OF SLAG AT 1650 °C; HM TEMPERATURE IN FEED: 1400 °C

7) MAY BE ABOUT 20% LOWER FOR DRI WITH 4%C, HOT CHARGED TO THE EAF

Required Composition of the Feed to an EAF

On the basis of the typical levels of trace and/or residual elements present in hot and cold rolled sheet as well as in the automotive grades CAV 01, CAC 01 and CAC 85, as supplied by C. L. Nassaralla (USX-US Steel), it is concluded that chromium and copper are the residual elements that are critical in determining the composition of the feed mixture of HM, scrap and DRI required to produce these grades of steel. To be able to produce the most demanding grade CAC 85 with [%Cr] < 0.04 and [%Cu] < 0.06, as well as the slightly less demanding grades CAV 01 and CAC 01, a feed mixture of 60% HM or DRI and 40% SM is required. The scrap mixture SM is to consist of 50% no. 1 dealer bundles and 50% no. 1 dealer busheling. Other sources of scrap are unsuitable because of their relatively high chromium and/or copper contents. The upper 2σ limits for no. 1 bundles and for no. 1 busheling were taken as 0.1% Cr and 0.1% Cu and 0.09% Cr and 0.08% Cu, respectively.

The less demanding products belong to the category of hot and cold rolled sheet and can be produced in an EAF fed with a mixture of 25 percent HM or DRI and 75 percent SM. The scrap mixture, SM, is the same as mentioned before.

Sulfur and Phosphorus Contents of Steels Produced in an EAF

For the purpose of the following estimates it is assumed that the smelter hot metal (HM) contains 0.15 percent sulfur and 0.02 percent phosphorus. The scrap mixture (SM) has the following maximum concentrations of sulfur and phosphorus, representing 2σ upper limits: 0.025% S and 0.023% P. It is noted that other sources of scrap have higher sulfur and phosphorus contents. The sulfur and phosphorus contents of DRI are assumed to be 0.01 and 0.033 percent, respectively.

From a series of recently obtained steel and slag analyses at Timken's Faircrest plant, it was determined that the slag-metal distribution ratios, (%S)/[%S] and (%P)/[%P] were about 3 and 30, respectively. Using these distribution ratios, together with a slag to metal weight ratio of 0.1 (100 kg slag per tonne of steel), mass balance calculations give the following sulfur and phosphorus contents for the various feed mixtures to the EAF.

TABLE I

	Feed Mixture	[%S]	[%P]
1.	60% HM + 40% SM	0.08	0.006
2.	60% DRI + 40% SM	0.014	0.008
3.	30% HM + 30% DRI + 40% SM	0.05	0.007
4.	25% HM + 75% SM	0.05	0.006
5.	25% DRI + 75% SM	0.02	0.007

As far as the phosphorus content is concerned, all steel grades under consideration can be produced by feeding any of the indicated mixtures to the EAF. However, it is clear from Table I that the required sulfur content of the various products, varying between 0.008 and 0.013% S, can only be attained by either desulfurizing the steel, or the HM prior to charging into the EAF, or both.

The most demanding grade is CAC 85, requiring [%S] < 0.008. This sulfur content can be obtained by desulfurizing the steel produced by feeding mixture 2 (Table I). When using feed mixture 1, desulfurization of the HM to 0.01% S or lower would be required, possibly followed by some steel desulfurization. For grades CAV 01 and CAC 01 and all hot and cold rolled sheet products it is required that [%S] < 0.012. This may be obtained by desulfurizing the steel produced by feed mixture 3 from 0.05 to ~ 0.01% S or, alternatively, by desulfurizing the HM first.

The overall conclusion is that for maximum flexibility the EAF shop would have to be equipped with HM as well as with steel desulfurization installations.

Nitrogen Content of Steels Produced in the EAF

To estimate the final nitrogen content of the steels produced by feeding the various mixtures to the EAF, use was made of Appendix B to the 1992 EPRI report "Nitrogen Control in Electric Furnace Steelmaking." Here the nitrogen removal in EAF's is shown as a function of the specific CO-evolution rate expressed in scf/ton of steel. Although the data presented in the EPRI report are specifically for the trials at North Star Steel, recent data obtained for the nitrogen removal in one of Timken's EAF's fall within the band of data presented in Appendix B. This indicates that the relationship has general validity and it was decided to apply the upper and lower limits of the band of data to estimate the anticipated degree of nitrogen removal for the various feed mixtures in Table I, relative to a conventional 100 percent scrap-based EAF practice. The specific CO-evolution rate was estimated from the carbon and iron oxide contents of the HM and the DRI, respectively. The results are shown in Table II.

TABLE II

Feed Mixture	$\Delta N/N_0$ (%)1) Range	Remarks on Product Requirement
1. 60% HM + 40% SM	78-99.9	Meets all
2. 60% DRI + 40% SM	35-88	All, except CAC 01 (?)
3. 30% HM + 30% DRI + 40% SM	62-99	Meets all
4. 25% HM + 75% SM	46-96	All, except CAC 01
5. 25% DRI + 75% SM	16-59	May meet hot and cold rolled sheet

1) $\Delta N/N_0$ (%): nitrogen removal relative to normal EAF practice (N_0 ~ 100 ppm).

The actual nitrogen content of the steels produced from the various feed mixtures may be estimated from the above values together with N_o - 100 ppm, the average nitrogen content of EAF steels produced from a 100 percent scrap charge. The remarks in the last column of Table II are based on these estimates and it is seen that all product requirements can be met by all the feed mixtures, except mixture 5. It is possible that the requirement for grade CAC 01 (30-50 ppm N) may not be met by mixture 4 and is possibly difficult to meet by using feed mixture 2. In general, it can be concluded that the presence of HM in the feed mixture practically guarantees a sufficiently low nitrogen content in the product such that the requirements for automotive grades can be met.

Electrical Energy Requirements

In modern up-to-date EAF's equipped with oxy-fuel burners, carbon and oxygen injection for foamy slag practice and sometimes scrap preheating, the electric energy input approximately equals the sensible heat of the liquid steel and slag produced.3-6) This implies that heat losses such as those to the exhaust gas, the water-cooled panels, electric and radiation losses, are made up by the other forms of energy input into the EAF. This approximate equality of electrical energy input and the enthalpy of liquid steel and slag forms the basis of the present estimates of the electrical energy required for the various feed mixtures. In the calculations the enthalpy (sensible heat) as well as the carbon content (chemical heat) of the liquid HM in the feed are taken into account, together with the average

TABLE III

ELECTRICAL ENERGY, kWh/TONNE OF STEEL

FEED MIXTURE	WITHOUT SCRAP PREHEATING		WITH SCRAP PREHEATING TO 500° C	
	TEMP. HM (°C)		TEMP. HM (°C)	
	1300	1400	1300	1400
60% HM 1) + 40% SM 2)	175	160	140	130
60% DRI 3) + 40% SM	490 4)		460 4)	
30% HM + 30% DRI + 40% SM	335	330	305	300
25% HM + 75% SM	315	310	255	250
25% DRI + 75% SM	440 4)		380 4)	

1) HM: SMELTER HOT METAL (5%C)

2) SM: SCRAP MIXTURE OF 50% NO. 1 DEALER BUNDLES AND 50% NO. 1 DEALER BUSHELING

3) DRI: DIRECT REDUCED IRON, 6.8% FeO, 3-8% GANGUE

4) MAY BE ABOUT 20% LOWER FOR DRI WITH 4%C AND HOT CHARGED TO THE EAF

aluminum, manganese and silicon contents of the scrap mixture SM. In the case of DRI, the enthalpy required to reduce the iron oxide as well as that needed to heat the gangue constituents is taken into account.

The results of the calculations are summarized in Table III, where the estimated electric energy requirements are shown for the various feed mixtures, for two temperatures of the HM feed (1300 and 1400°C) and with or without scrap preheating to 500°C. All numbers are based on 1 tonne of liquid steel in the presence of 100 kg of slag at 1650° C (total enthalpy of steel and slag is 430 kWh/tonne of steel).

It is seen from Table III that the replacement of HM by DRI in the feed mixture leads to a significantly higher energy requirement.*) Specifically, the data show that the replacement of each weight percent of HM by DRI results in an increase of 5.5 kWh/tonne in the energy requirement. It is further seen that the effect of preheating the scrap to 500°C on the energy requirement is most pronounced for feed mixtures 4 and 5, leading to a decrease of about 60 and 140 kWh/tonne, respectively. For feed mixtures 1-3, containing 40% SM, the energy savings from scrap preheating are about 30 kWh/tonne.

The temperature of the HM in the feed has a relatively minor effect, a decrease in HM temperature of 100°C leads to an increase in the energy requirement of 5 to 15 kWh/tonne, depending on the feed mixture. This means that a drop in HM temperature as a result of prior desulfurization is of minor concern in terms of subsequent energy requirements during melting.

Raw Material Requirements

The raw material requirements per tonne of liquid steel produced from the various feed mixtures are summarized in Table IV (p. 8). For comparison the raw materials used in conventional modern EAF steelmaking using a 100 percent scrap charge are also shown. The ranges are obtained from a number of recent publications in the Proceedings of the 4th European Electric Steel Congress (1992) and are considered to represent the most up-to-date practices.

The raw material requirements estimated for each specific feed mixture considered here are based on the average between the minimum and maximum of each range for the conventional EAF practice and taking into account the carbon content of the HM, the SiO_2-content of the DRI, etc. It is emphasized that the data in Table IV should be used primarily to compare the various feed mixtures with each other; the absolute values are less important. For example, when comparing 60% HM + 40% SM with 60% DRI + 40% SM, the data indicate that the lime usage is expected to increase by about 50 percent when replacing HM by DRI while the natural gas and coal (coke) usage both increase by 150 percent and the oxygen usage decreases by about 35 percent. The use of DRI containing 4% carbon will decrease the usage of natural gas and coal while the oxygen usage may increase somewhat.

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*) A recent article in the Iron and Steelmaker (July 1993, p. 44) describes Hylsa's work on producing DRI containing about 4% carbon and hot charging the material to an EAF. Based on some of the information in the article, it is estimated that the use of this technology would result in an energy requirement about 20% lower than that indicated in Table III (footnote 4).

TABLE IV

RAW MATERIALS REQUIREMENTS PER TONNE OF LIQUID STEEL FOR THE VARIOUS FEED MIXTURES TO THE EAF (100% SCRAP = CONVENTIONAL PRACTICE)

	AMOUNT TO CHARGE (kg)						
FEED MIXTURE	HM[1]	DRI[2]	SM[3]	LIME (kg)	NG[4] (Nm^3)	O_2 (Nm^3)	COAL (COKE) (kg)
100% SCRAP	–	–	1110	10–20	2–5	20 – 40	10 – 20
60% HM + 40% SM	632	–	444	15	1.4	46	6
60% DRI + 40% SM	–	645	444	23	3.5	30	15
30% HM + 30% DRI + 40% SM	316	323	444	19	2.5	38	10
25% HM + 75% SM	263	–	833	11	2.6	36	11
25% DRI + 75% SM	–	269	833	18	3.5	30	10

1) HM: SMELTER HOT METAL (~ 95% YIELD)

2) DRI (HBI): ~ 1.2% SIO_2 (~ 93% YIELD)

3) SM: 50% NO. 1 BUNDLES, 50% NO. 1 BUSHELING (~ 90% YIELD)

4) NG: NATURAL GAS (OXY-FUEL BURNERS)

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1. G. J. W. Kor, "Iron Yield Loss in Continuous EAF Steelmaking," memorandum to AISI Direct Steelmaking, June 3, 1993.

2. O. Puehringer et al., Stahl und Eisen, 1991, Vol. 111, p. 37.

3. W. Dening et al., Stahl und Eisen, 1984, Vol. 104, p. 68.

4. J. A. Vallomy, Met. Plant and Technol., 1989, no. 6, p. 20.

5. J. A. Vallomy et al., Proc. 4th Euro. Electr. Steel Congr., 1992, p. 39.

6. P. Broussard et al., ibid., 1992, p. 233.

June 13, 1993

Post-Hearth Steelmaking in an Integrated Smelter-Refiner[1]

Process Description

Post-hearth steelmaking is a modification the two-zone horizontal vessel steelmaking concept. As shown in Figure 1, high-carbon metal will overflow from the smelter into an attached refiner. We will inject lime and oxygen into the metal in the refiner (at higher pressure than in the smelter) to remove sulfur and carbon. By removing gas and slag from the smelter only, the CO gas from the refiner will force the refiner gas and slag upstream into the smelter. The slag's CaO content will replace the lime that would otherwise have to be added to the smelter and its FeO will be rereduced to Fe. The CO gas will be postcombusted in the smelter. This design eliminates metal back mixing, and keeps the carbon char particles in the smelter slag and gas out of the oxidizing zone -- two problems that hampered the production of low carbon metal during the operation of the horizontal vessel.

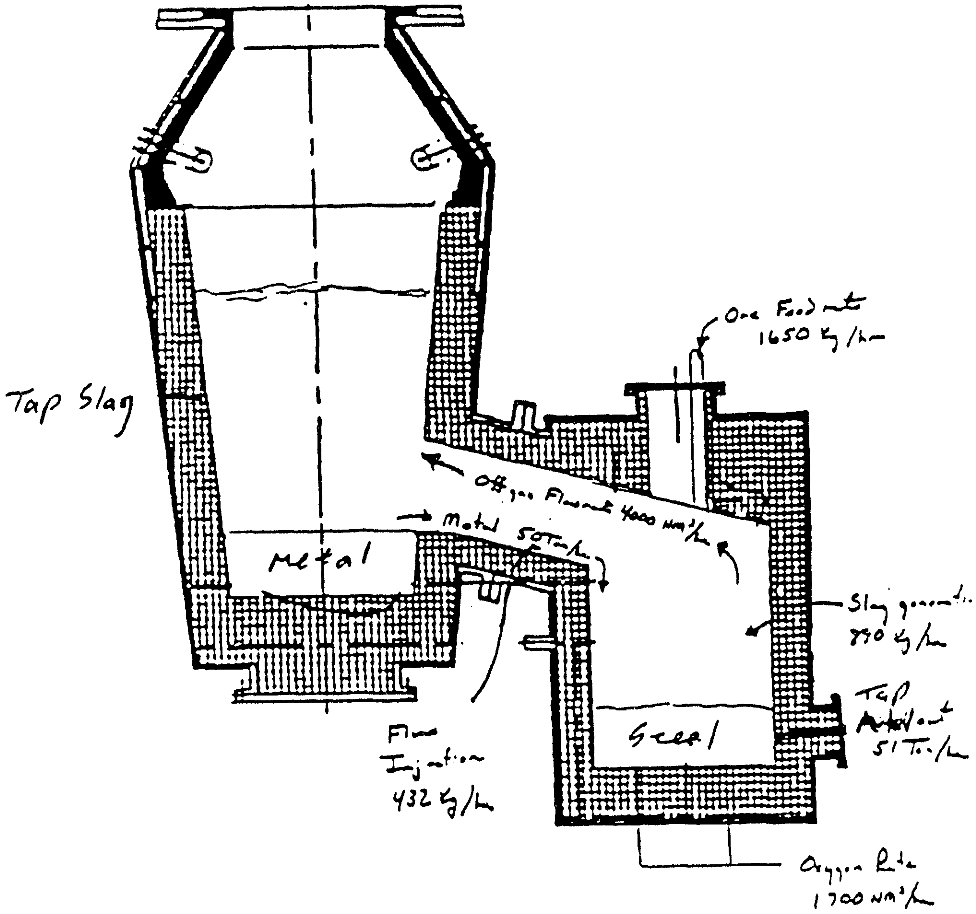

Figure 1. Arrangement of Smelter Vessel and Post-Hearth

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1. This summary is based on a detailed analysis in a memorandum from H. Pielet and M. Schlichting to R. Fruehan.

The system is amenable to a number of operating schemes. For example, we could continuously smelt, refine, and tap 0.3-1% C metal, or we could decarburize continuously to 0.3-1% C, then rapidly decarburize to 0.1% C, possibly with a mixture of oxygen and inert gas, and batch tap.

Development Status

The process builds on our Pilot Plant experience with the continuous refiner and horizontal two-zone smelter. We are already planning to have essentially all of the equipment we will need to try it. Ken Downing estimates that we could set up the post-hearth for steelmaking trials at almost no cost.

Even with the problems encountered during the operation of the horizontal smelter, it produced metal at less than carbon saturation. Carbon as low as 1.00 % was measured during Trial 18.

Critical Issues

Refining (S, P, N, etc.): The smelter, with its oxygen-blown, 5% FeO slag, and carbon-saturated metal has an oxygen activity somewhere between the BF and BOF. Its sulfur and phosphorous distribution ratios of 5 and 10, respectively, are similar to those in a BOF. This implies that, for sulfur and phosphorous removal, the smelter is closer to a BOF than a BF. The smelter metal is about 0.015% P and 0.05% S.

To use the hearth as a settling chamber for the smelter slag, both the smelter metal and the smelter slag will flow into the hearth. Without continued oxygen input to the slag, the excess carbon in the smelter slag may reduce the FeO in the slag. The phosphorus in the slag will then revert to the metal. To use the hearth to desulfurize, we would in any case have to kill the slag, again causing phosphorous to revert to the metal.

On the other hand, we could use the hearth to decarburize, as proposed here. However, because of the high sulfur content of the smelter metal, even a sulfur partition ratio in the hearth of 15, which is higher than is normally achieved in a BOF, will result in a sulfur content of the hearth metal of 0.024%. External desulfurization will therefore be required. However, the metal will already be dephosphorized, and economic analysis indicates that it is no more expensive to desulfurize downstream by tap stream and ladle desulfurization than to desulfurize upstream by scrap selection and hot-metal desulfurization.

Thus we appear to have three choices for using the hearth: 1) use the hearth to settle the smelter slag, blowing enough oxygen into the hearth to prevent phosphorus reversion, 2) use the hearth to settle the smelter slag, allowing the carbon in the slag to reduce the FeO and accepting the phosphorous reversion in order to desulfurize, or 3) use the hearth to decarburize, producing a 1% C, 0.01% P, 0.03% S metal.

Since there will be no air infiltration, $\underline{N}$ could be lower. At higher pressure, $\underline{H}$ may be higher.

Metal Feeding and Control: Smelter metal will overflow into the post-hearth, just as we are already planning for post-hearth settling and desulfurization. However, we will probably need to insert a slag underflow dam in the connection between the smelter and the post-hearth to control the amount of slag in the post-hearth and to control the extent of backmixing between the smelter and hearth slags. The similarity of sulfur and phosphorous partition ratios for the smelter slag and BOF slag implies that backmixing between the smelter and hearth slags may not be particularly detrimental. Even some excess char brought into the hearth slag may simply serve to reduce the FeO formed in the hearth, just as current foamy-slag EAF practice simultaneously blows oxygen into the metal and carbon into the slag.

Since the decarburization off-gas is only about 1/17 the smelter off-gas, the connection may not have to be large. However, it is not clear if gas should flow from the hearth to the smelter through the same connection as the metal and slag or through a separate, higher connection. We will have to do water-modeling and heat transfer calculations to properly design the connection between the smelter and the hearth.

Post-hearth decarburization to 0.3-1%C will be simply limited by oxygen supply. Control will be based on the apparent production rate of the smelter and monitoring of the tap carbon, particularly if we tap continuously. If we batch tap, it may be possible to estimate the metal level in the hearth from the back-pressure on the bottom-blowing tuyeres.

Productivity: Decarburizing to 0.3-1% C should be easy, but it is not clear if there is an economic benefit to producing such an intermediate carbon level. Carryback of FeO and metal droplets in the hearth slag into the smelter should reduce the process intensities in the smelter and hearth by only about 3%.

FeO formation in the hearth slag would reduce the apparent process intensity of the smelter, but not the metallic yield, since the FeO would be rereduced when the hearth slag flows up into the smelter. Including inert gas such as CO_2 from the prereducer off-gas with the decarburization oxygen, would decrease the FeO in the hearth slag when producing 0.1-0.3%. We should calculate the effect of inert gas addition on the overall process economics.

The process intensity of the refining hearth should be greater than a BOF.

Batch operations in the hearth, such as tapping, desulfurization, and decarburization may adversely affect the smelter vessel and prereducer.

Yield: Metallic yield from decarburization will be very high since any FeO produced in the hearth will be rereduced when the hearth slag moves up to the smelter, or by char in any smelter slag allowed into the hearth. Tapping the smelter slag directly from the smelter may decrease the smelter yield.

Start Up: Preload the hearth with coal, then inject oxygen and inert gas to keep the hearth hot and push back the smelter slag until the smelter metal starts overflowing into the hearth.

Scrap Melting: Pilot Plant personnel have successfully added DRI to the smelter vessel. Presumably, shredded scrap would also work. However, adding heavy scrap is a serious problem, since adding into in a sealed, pressurized vessel will surely complicate scrap addition.

Refractories: Refractory wear in the hearth will depend on the temperature and slag FeO as a function of time. The tunnel will be at a lower temperature than the hearth, and the metal flowing through it will be high-carbon. Therefore the main refractory wear will probably be from FeO-containing slag flowing upstream into the smelter. With combined inert-gas/oxygen decarburization, refractory wear should be comparable to BOF and better than IRSID. We may want to adjust the hearth slag, for example by adding dolomite, to make it less erosive to the hearth and tunnel refractories.

Metal and Slag Tapping: We will tap metal only from the hearth. For batch tapping, we could use the smelter tapping procedure. For continuous tapping, we could use a slide-gate or a teapot spout with an overflow about 1.5 m above the taphole.

We will tap slag only from the smelter, using whatever procedures we develop for the smelter.

If we choose to refine and tap the metal continuously and then accumulate it in a ladle, we will have to reheat the metal in the ladle. Any continuous process faces this problem. IRSID should have data on the heat losses from continuous tapping.

Gas Removal: We will postcombust the decarburization CO from the hearth in the smelter. The hearth off-gas will become simply a part of the smelter off-gas. If refining is not continuous, we will have to adjust smelter operation for variations in hearth off-gas flow rate.

Environmental Issues: The smelter-hearth will have only one slag, one metal, and one off-gas stream. This will at least reduce the gas and slag volumes required for further refining to remove the remaining sulfur and carbon.

Technical Assessment

Continuous smelting, refining, and tapping of 0.3-1% C, 0.01% P, 0.03% S metal from a two-zone smelter-refiner with an overflow dam for metal and counter-current flow of metal to gas and slag appears technically feasible. It will allow us to postcombust the decarburization CO, use a single off-gas system, and use the lime for adjusting basicity in the smelter to do some desulfurization in the hearth.

However, if work with the third vessel indicates that we need to settle entrained metal from the smelter slag in the hearth, and that we can settle the entrained metal, we may not be able to decarburize in the hearth. In that case, we could either blow some oxygen to keep the slag oxidizing enough to retain the phosphorous it has already picked up in the smelter, and desulfurize outside the smelter-hearth, or we could significantly reduce the oxygen activity in the slag so we can desulfurize, and accept the phosphorous reversion.

It is not clear if the current smelter-hearth configuration can be adapted for hearth decarburization. We will have to carefully design the connection between the smelter and the hearth to control motion of the gas and slag formed in the hearth up into the smelter. This will require water-modeling and heat transfer calculations. It may require an entirely different smelter-hearth design.

We should be able to add DRI and shredded scrap to the smelter vessel. However, adding heavy scrap to a pressurized vessel is a significant unresolved problem.

Because the metal will still need further refining, we need to understand if there is an economic use for a continuous supply of 0.3-1% C, 0.01% P, 0.03% S metal. One possibility that would give us both a way to melt scrap and a way to use the partly-refined metal would be to tap directly into an EAF.

Continuous smelting, continuous refining to 0.3-1% C, fast decarburization to 0.1% C, possibly with a mixture of oxygen and inert gas, and batch tapping is less feasible. It would allow us to avoid the costs and heat losses of transporting the metal to a separate refiner, achieve high metallic yield, and have only the one stream of metal, one stream of slag, and one stream of gas to remove from the combined vessel, instead of streams from both a smelting vessel and a refining vessel. However, we would have to determine the resulting process intensity and the effect of discontinuous refining on the smelter and prereducer.

An advantage to post-hearth steelmaking is that we are already planning to have essentially all of the equipment we will need to try it. Also, we will have to solve nearly all of the critical issues such as connection design, start up, the effect of discontinuous refining and tapping, and refractory wear, to successfully carry out the post-hearth settling trials that we have already planned.

Recommended Program

I recommend the following program.

1. Determine if we need the post-hearth to settle entrained metal from the smelter slag, and if we can settle the entrained metal in the hearth.

If we do need the post-hearth for slag settling, decide if we want to blow some oxygen to keep the phosphorous in the slag, or if we would rather desulfurize in the post-hearth at the expense of phosphorus reversion. Include several decarburization trials to low carbon to confirm the expected high yield losses.

If we do not need or cannot use the post-hearth to settle the smelter slag, carry out the following program.

2. Determine if there is an economic use for a continuous supply of 0.3-1% C, 0.01% P, 0.03% S metal.

3. Water-model and do heat-transfer calculations to design the connection between the smelter and the post-hearth. Determine if the current smelter-hearth design can be modified or if it must be radically changed.

4. Develop a model to calculate the variation of the chemistry of the metal in the hearth (C, S, P, N, H) with time as a function of smelter production rate, tapping rate, flux addition rate, and oxygen injection rate. Use the model to develop favorable operating sequences.

5. Carry out both continuous and batch post-hearth decarburization trials.

Study

1. Separation of the metal, slag, and char phases in the hearth.

2. Composition of the metal in the hearth as a function of smelter production rate, tapping rate, flux addition rate, and oxygen injection rate.

3. Effects of hearth refining on the smelting process.

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EOF Process Evaluation

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Process Description

The EOF is essentially a batch oxygen steelmaking process with a high degree of post combustion, coal additions and scrap preheating. A schematic of the process is shown in Figure (1). The combination of post combustion, coal and scrap preheating allow for scrap charges theoretically up to 100%. However, the process operates usually with a scrap charge of 40% to 60%.

Hot metal is charged into the vessel in the conventional manner, usually a one tonne heal is present from the previous heat. Scrap is added from the lower scrap preheating chamber. With large scrap additions a second addition is made about 5 minutes into the blow. The scrap temperature is typically 800-1200°C. When the scrap is added from the lower chamber it is refilled from an upper chamber. For a 50/50 scrap/hot metal mix about 64 m^3/tonne of oxygen is used with about one third into the bath and two thirds above the bath for post combustion. The post combusted gas is used to preheat the scrap and, therefore, the post combustion energy does not have to be transferred completely to the liquid metal to be useful. The blowing time is about 30-35 minutes and a tap to tap time is about one hour.

Current Status

The EOF has been running on a commercial basis in Pains Brazil for nearly ten years with over 100,000 heats produced. In 1992 they produced 500,000 tonnes of steel with one 30 tonne furnace with about a 50/50 charge mix. I witnessed the operation of that furnace and was impressed with its productivity. The scrap charging system works well and process control is satisfactory.

In addition there is an 80 tonne furnace at Tata Steel in India and a 60 tonne furnace at AFS in Italy. It can be concluded that the EOF is a reasonably well proven process when working with scrap charges of 40 to 60%.

Critical Issues

Residuals: When using high scrap charge ratios the quality of the steel with regards to residuals is an issue. J. Kor discussed this issue in detail for the EAF and the same conclusions apply to the EOF. Basically, for high quality, flat rolled scrap must be limited to about 40%. However, this puts no restrictions on the EOF since it can operate effectively with 60% hot metal and most grades of steel can be produced competitively with other processes.

Productivity: The EOF is a high productive process capable of producing 100 tonne heats in less than one hour. The productivity is similar to a state of the art EAF and close to that of a BOF.

Yield: Yield will be similar to a BOF and has been proven to be about 92-94%.

Refining: Phosphorus removal will be similar to a BOF and lower phosphorus levels could be obtained due to high scrap rates. Nitrogen will be slightly higher than a BOF due to higher scrap ratios and coal additions to the process. Nitrogen levels of 30-40 ppm at tap are normally achieved. Carbon levels of 0.03-0.05% can be achieved. Effective production of lower carbon will require vacuum processing.

Sulfur will be a major concern with the EOF. The sulfur content will be higher than an EAF in some cases because coal is used. Hot metal desulfurization is not an option with the high scrap rates. For the following assumed conditions the sulfur levels are given below:

$^{(\%S)}/[\%S] = 4$ %S in HM 0.15%
Slag weight 140 lbs/ton %S in Scrap 0.025%
Coal sulfur 0.8%

It is assumed no sulfur is being carried over with smelting slag. The sulfur partition ratio is higher than an EAF due to extensive stirring. The slag weight is based on the CaO addition and could vary depending on conditions.

Charge	Steel Sulfur (±0.01%)
60 HM - 40 Scrap	0.08%
50 HM - 50 Scrap	0.08%
40 HM - 60 Scrap	0.08%

These values will depend on the actual conditions. The sulfur content is about the same for all cases because the higher scrap charge requires more coal for melting which increases the sulfur load by about as much as the decrease in the amount of sulfur from the hot metal.

Scrap Melting: The EOF is capable of melting large quantities of scrap. The scrap should be loose for effective preheating and addition. The EOF has not been proven for very high scrap rates (<75%) however it should be feasible to use 40% to 60% scrap. Less scrap can be used but then the process looses its energy efficiency, ie. not all of the energy is utilized and a BOF or IRSID process is more attractive. The amount of scrap may also be limited due to residuals as discussed previously.

Metal Feeding and Tapping: The EOF is a batch process and metal feeding and tapping will be similar to a BOF.

Gas Removal: In general the EOF uses a bag house for gas cleaning. The amount of gas is less than for an EAF because it is a closed system with little infiltrated air. There is more dust than for an EAF and the dust is generally classified as EAF dust.

Environmental Issues: As mentioned, the dust is classified as an EAF dust when using a high scrap charge. There would be about 50% more dust than an equivalent EAF resulting in higher disposal costs.

Technical Assessment

The EOF is a reasonably well proven technology. It has the ability to melt large quantities of scrap and is a good match with the proposed AISI smelter. An AISI smelter producing 400,000 tonnes of hot metal could be used with an 80 ton EOF could produce over 800,000 tonnes of steel per year. Possibly not all products could be produced due to residuals, nitrogen contents may be too high for some grades of steel. Both hot metal and steel desulfurization may be necessary or there will be a high cost for steel desulfurization.

Development Plan

The EOF is currently in operation but well controlled tests will be required to determine the performance details for the present application. The most cost effective program most likely would be, if possible, to conduct tests at AFS in Italy with Korf Lurgi. As few as ten well planned and executed tests should be sufficient to determine sulfur content, yields, nitrogen content, etc.

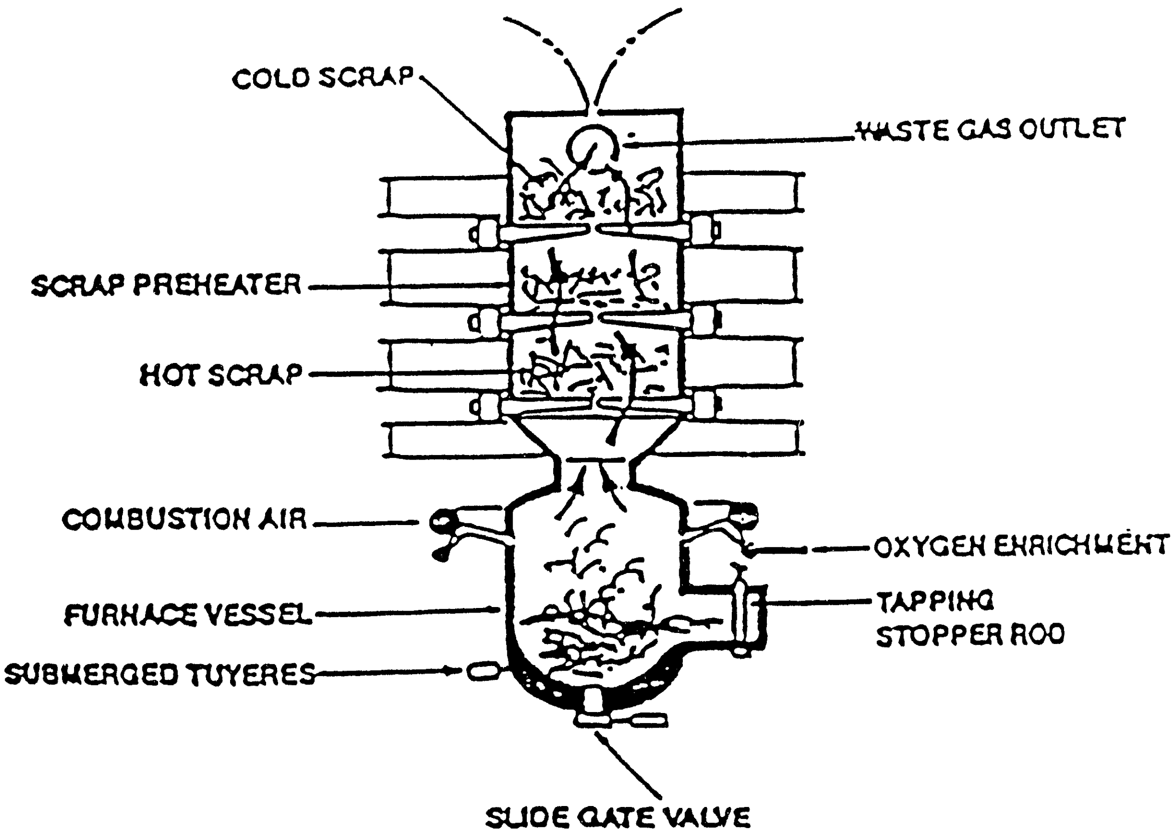

Figure 1 : Schematic diagram of EOF with 100% scrap change

Desulfurization Strategy and Cost

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In order to perform an economic comparison of the various processes the desulfurization strategy and economics must be considered. Both the strategy and the economics depend on the assumptions regarding sulfur input, sulfur partition in steelmaking, final sulfur and the cost of the various processes. These assumptions are listed below:

%S Hot Metal	0.15%
%S Scrap	0.025%
%S Aim	0.01%
$\frac{(\%S)}{[\%S]}$ BOF-IRSID	6
$\frac{(\%S)}{[\%S]}$ EAF-EOF	4
Slag weight	140 lbs/ton
Scrap percentage BOF-IRSID	20%

Desulfurization cost	$/ton
Hot metal 0.15 to 0.003	$8
Steel 0.11 to 0.01	$16
Steel 0.04 to 0.01	$10

The cost for hot metal desulfurization was extrapolated slightly from the previous AISI analysis, the cost for high sulfur steel is based on a detailed analysis from LTV and for the lower sulfur content cost is an industry average. The high sulfur case includes reheating costs. The slag weight is less than normal because the hot metal is low in silicon. In the evaluation it was assumed that the hot metal desulfurization slag is adequately raked. There will also be a sulfur drop due to normal clean steel practices in the ladle. The sulfur partition ratios are for optimized state of the art operations. The process, low cost strategy, steel sulfur level and cost to achieve 0.01% are given below. These are only estimates and the strategy and cost could differ based on specific conditions.

Process $\frac{\%HM}{\%Scrap}$	Strategy	%S (Prior to steel desulfurization	Cost $/ton
BOF-IRSID $\frac{80}{20}$	Hot Metal	~0.01	6.25
EAF$\frac{40}{60}$	Steel	~0.06	11
EOF 40/60	Steel	~0.08	13
EAF 60/40	Steel	~0.08	13
EOF 60/40	Steel	~0.08	13

These estimates are considered to be only estimates (±$1.50). In no case did both hot metal and steel desulfurization appear attractive. If the aim sulfur is 0.02% it significantly changes the strategy and the cost. For both the EAF and EOF for the 60/40 case hot metal desulfurization at $5 per ton of steel will produce steel with less than 0.02%S. If less than 0.01% sulfur is required steel desulfurization for IRSID and BOF will be required increasing cost significantly.

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