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SOME TECHNICAL AND ECONOMIC CONSIDERATIONS ON THE USE OF DIRECT REDUCTION OF IRON ORES IN VENEZUELA

by

Carlos A. Martinez V.

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Signed: C. Martinez V.
Carlos A. Martinez V.

Golden, Colorado

Date: December 1, 1975

Approved: Alfred Petrick
Dr. Alfred Petrick, Jr.
Thesis Advisor

J. Paul Mather
Dr. Jean Paul Mather
Chairman, Department of
Mineral Economics

Golden, Colorado

Date: December 1, 1975.

ABSTRACT

This study points out the most ciritcal technical characteristics that affect the economics of a direct reduction process of iron ores.

Two of the now available industrial processes, the Hojalata y Lamina and the Midrex, are compared technically and economically. The required characteristics and quality of the end product are related to its further use of the electric furnace.

After making an economical selection between the two methods, the selected one is studied in detail using the discounted cash flow analysis and the concept of production function.

The whole situation applies to the present plans of the Venezuelan government to increase raw iron and steel production.

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INTRODUCTION

The Venezuelan government is presently engaged in increasing the steel production and also improving the quality of the iron ore sold in international markets. To accomplish these two purposes, it has been decided to use direct reduction methods, utilizing natural gas as the source of the reducing agent hydrogen.

Presently several plants that use the direct reduction method are under construction in Venezuela. Two of these plants are being built by the Venezuelan government. One of the plants operate with the Hojalata y Lamina Mexican process, and the other one is the Midrex, a German-American process. The reason for building two plants with two different methods of operation is to test which method is the best suited for the existing conditions in Venezuela.

The research done in this work sets off the technical and economical basis on which a selection must be made between the two processes. The scope of the investigation is limited to the two processes under consideration by the Venezuelan government.

Specifically, the objective is to assemble data from several sources describing the technical aspects of the methods that have a direct bearing on their economics and also to evaluate existing cost and investment data to determine the relative profitability of each process.

The first part of the study describes the availability of raw materials and the reasons why it is better advised to proceed with the method of direct reduction instead of the conventional blast furnace. This part of the study ~~has~~ containing a general presentation of the availability of iron ore, scrap, coal, natural gas, electricity, and other materials such as limestone and dolomite. The reserves and chemical composition of natural gas and iron ore are presented, because the economics of the direct reduction method are sensitive to changes in the quality of these two inputs.

The following part includes a description of the present iron ore mining and future process developments. This description includes the background and present operating conditions of the two American companies now operating there, the competing processes that are now under construction or at the start-up stages around the same area, and a brief description of the plans to increase capacity.

Several technical factors must be defined next, because they link the technical and economic conditions of the methods under study. The operating characteristics of the methods and the chemical and physical characteristics of the end products are presented. Then, it is specified how the latter characteristics affect the operating conditions of the electric furnace.

The next chapter deals with the technical description and comparison of the two methods selected. Subjects such as background, present operating conditions, brief mechanical description, and quality of the end products are dealt with in this chapter.

The price of iron ore is calculated next. The units are \$ per metric ton and \$ per unit of iron content. What follows are quotations of the operating and investment costs of the HyL and Midrex processes. The quotations were taken from the five years Metallurgical Plan, made up jointly by the Venezuelan government and foreign technicians.

It must be noted at this point that these data have been revised since first calculated. The purpose of this study is not to calculate investment and operating costs, but to use them to construct the discounted cash flow rate of return and the production function models. The cost information used here by the author was made available to him by the government engineers and other technical personnel. The purpose here is to summarize the technological conditions facing the selected method by computing its production function, which gives the functional relationship between rates of input of factors of production and rates of output per time period.

The other study that exists on the economic evaluation of the direct reduction processes to be used in Venezuela is that contained within the Metallurgical Plan. The method of "comprehensive operating cost" is used to calculate a constant capital recovery factor. This factor is then multiplied by the capital cost of the particular project and finally added to the annual operating costs. That is the way in which the processes are compared. Note that the factor used is the same one for all the situations.

Essentially, then, this study is of value because of the following reasons:

a) Other existing technical works are mostly descriptive. Here, technically, the objective is to correlate the most critical technical factors for the selection of a particular process and the quantitative effect of these factors on the economics of the projects.

b) To present a method with an actual selectivity value, that helps in the economical selection of a specific process among several of the same kind. The basis for the selection is the profitability of the process. This part of the study also contributes to develop the production model that shows the relationship between the ongoing selling price in the iron market and the level of maximum profit where marginal cost equals marginal revenue.

This study will utilize available data. Where specific data are missing, information from similar plants and different processes will be correlated and related to the specific process under study.

Due to the confidential nature of more recent information, no updating was attempted in the data presented in tables. The existing trends and costs existed through 1974.

THE QUALITY AND AVAILABILITY OF THE INPUT MATERIALS

The end-product specifications of directly reduced iron ore in an electric furnace are much more sensitive to changes in the available raw materials as compared to end products obtained from any other classical method of iron ore treatment. This sensitivity must be evaluated starting with the original conditions (i.e. chemical and physical composition) of the raw materials used. For instance, the amount of gangue content in the raw iron ore will go unchanged through all the direct reduction treatment up to the melting of the briquette in the electric furnace.

Theoretical costs calculation can always be made in pre-investment cost analysis using average value. The costs thus obtained do not represent the actual costs of the process when in actual operation. The reasons for that are the existing variations in the chemical and physical characteristics of the input materials which are directly related to the quality of the end product and consequently with the costs of the operation.

The input materials of the direct reduction process are metallized pellets and natural gas. These two inputs make up 88 percent of total operating costs. If we exclude the costs of the iron pellets, natural gas amounts to 34 percent of the operating costs.

The next sections include summaries of the quantitative and qualitative data of Venezuelan iron ore, natural gas, and electricity.

Also included in this section are two short comments about the availability of coal and scrap in Venezuela.

<u>Iron Ore</u>

The iron ore reserves of Venezuela are estimated at 2,536 million tons. All of the iron deposits are located at the south-west region of the country officially called the Bolivar State. Some of the areas were given as concessions by the government to American companies and some are classified as national reserves. Table I shows the amounts of reserves and the status of each one of them.

In general the deposits are of a sedimentary origin, mainly hematitic, and to a lesser extent composed of magnetite, limonite, and goethite. Impurities are present in low proportion. Table II shows the average composition of the ore in the main deposits.

TABLE I

Venezuelan Iron Ore Proved Reserves

Group Name	Concessionary	Mill. Tons
Rio Aro	National Reserves	150
Cerro Bolivar	Orinoco Mining Co.	803
San Isidro	Siderurgica del Orinoco	821
Maria Luisa	National Reserves	258
El Pao	Iron Mines Co.	379
Imataca	National Reserves	125

Source: Metallurgical Plan, p. 38 (adapted)

TABLE II

Average Chemical Composition of the Venezuelan Iron Ore

Element %	Cerro Bolivar	El Pao	San Isidro*
Fe nat	58.91	61.91	62.69
SiO_2	2.13	3.29	2.10
Al_2O_3	1.43	2.64	N.A.
P	0.1	0.05	0.15
S	0.02	0.01	0.08
Ignition loss*	4.98	3.20	5.78
Humidity*	7.94	3.45	4.06

Source: Company Reports: Jan-June, 1975

* Metallurgical Plan, p. 39.

Scrap

The production of scrap in Venezuela is limited. The available scrap comes from the oil and auto incustries. At present the steel industry is consuming an average of 140,000 tons per year. Scrap is sold at about 83 U.S. $ per ton (oral communication, 1974). Therefore, scrap represents a cash outlay of approximately 11,620,000 U.S. $ per year to steel producers. In Venezuela, like elsewhere, scrap prices are subject to heavy fluctuations. A stabilizing effect is therefore to be expected from the use of pre-reduced products for the production of steel.

Coal

Venezuelan coals belong to the tertiary period. They are classified as sub-bituminous to lignite and they are not coking coals. These coals can only be used in the steel industry in combination with higher quality imported coals.

Natural Gas

Since the beginning of the Venezuelan oil industry in 1912, 17.3 trillion cu. ft. of natural gas had been flared or otherwise vented to the atmostphere in Venezuela. This amount of wasted gas represents 55 percent of the accumulated gross production of natural gas and about 50 percent of the present reserves in the country. In 1971, the proven reserves of natural gas amounted to 31.6 trillion cu. ft., and adding the probable reserves, the total amounts to 36.4 trillion cu. ft. Table III shows the classification of the Venezuelan natural gas.

TABLE III

Reserves of Natural Gas in Venezuela
(As of December 31, 1971; mill. cu ft.)

Associated Gas (92.09% of total proven)	29,383,062
Non-associated Gas (7.09% of total proven)	2,242,113
Total proven	31,625,175
Total probable	4,790,825
Total Gross	36,426,000

Source: La Salvia, 1972

Also, in 1971, 0.75 trillion cu. ft. of natural gas or 34 percent of a total production of 1.68 trillion cu. ft. were wasted; this is equivalent to 103.2 million barrels of oil or 8 percent of the total oil produced in Venezuela.

By the year 2000, it is estimated that a total of 22 trillion cu. ft. will be consumed in the industrial and domestic markets of the country. Since the reserves amount to a total of 36.4 trillion cu. ft. this leaves a balance of 14.4 trillion cu. ft. that presently have no future commercial and/or industrial use.

The gas available at Puerto Ordaz, the location of the future steel works, comes by pipelines from the Barcelona and Maturin fields. The relationship among the producing areas is shown in Table IV, which shows the gas produced in 1970 and 1971.

TABLE IV

Gas Produced by Areas (mill. cu. ft.)

Area	Total Gross Produced		% of Total	
	1970	1971	1970	1971
Maracaibo	1,015,553	965,406	59.38	57.45
Barcelona	611,020	624,687	35.73	37.18
Maturin	82,884	89,523	4.85	5.33
Barinas	742	636	0.04	0.04
Venezuela	1,710,199	1,680,252	100.00	100.00

Source: La Salvia, 1972

The percentages of associated and non-associated natural gas for each region is shown in Table V.

TABLE V

Distribution of Gas by Areas (Percent)

Area	Associated Gas	Non-Assoc. Gas
Maracaibo	67.3	4.0
Barcelona	27.7	83.0
Maturin	6.0	13.0
Total	100.0	100.0

Source: Tiratsoo, 1967, p. 221.

Most of the gases are sweet with high contents of methane, ethane, and heavier hydrocarbons, containing occasional substantial proportions of CO_2, but little if any hydrogen sulphide. Table VI shows the chemical composition of the gases available at Puerto Ordaz and that available at Maracaibo.

TABLE VI

Average Chemical Composition of Venezuelan Natural Gas
(Components in percent)

Components	Puerto Ordaz	Maracaibo
Methane	80.22	70.68
Ethane	5.04	13.12
Propane	2.03	6.61
I-Butane	0.34	1.22
N-Butane	0.48	1.89
I-Pentane	0.16	0.60
N-Pentane	0.13	0.49
Hexand Plus	0.26	0.22
Heptanes	-	0.13
CO_2	11.34	4.37
Specific Gravity	0.734	0.802
Calorific Value (btu/cu ft)	1005	1275

Source: Metallurgical Plan, p. 46.

Electricity

In 1972, the installed generating capacity of electricity in Venezuela reached 3,377.8 MW, out of which 1.438.0 MW are generated in steam plants, 907.4 MW as hydropower, 756.9 MW in gas plants, and 275.5 MW in diesel plants.

Several new projects have been developed to increase the generating capacity in iron ore regions. All the electricity of the area is and will be generated as hydropower. The new generating capacity of this area will be in the order of 10,705 MW, distributed as follows:

Macagua	1,315 MW
Caruachi	1,760 MW
Tocoma	1,080 MW
Guri	6,550 MW

Source: The Metallurgical Plan.

Other Materials

The other materials commonly used in the preparation of steel are limestone, dolomite, and silicon. These minerals are found all over the country in adequate quantities to assure that steel production will not be obstructed.

All iron alloys needed for the production of pig iron have been imported in the past, but there are several projects for the construction of plants to produce these materials in the country. Only five percent of the 60,000 tons per year of refractory material has been imported.

PRESENT STATUS OF IRON ORE MINING AND PROCESSING IN VENEZUELA

There are three iron mining companies in Venezuela. One company, Iron Mines Company of Venezuela (I.M.C.O.V.), is a subsidiary of the Bethlehem Steel Corporation, and the other one is Orinoco Mining Company (O.M.C.), a subsidiary of the U.S. Steel Corporation. The third company is Siderurgica del Orinoco.

I.M.C.O.V. started operations in 1950 and all of its output is exported to the parent company's mills in the U.S. The mine is located near El Pao. The installations of the mine consist of a primary, secondary, and a small capacity tertiary crusher. The ore is transported by train (50 Km) to the harbor of Palua on the shores of the Orinoco River. The company is constructing a washing and screening plant at Palua, because the physical and chemical requirements of the ore are becoming more strict.

O.M.C. started operations at Cerro Bolivar in 1954, and it sends about 25 percent of its output to Europe and the rest to the parent company's facilities in the U.S. This company does not have any processing facilities at the mine site. The ore is transported by train (140 Km) to the port at Puerto Ordaz. In the port area the company has crushing, screening, and drying facilities. In 1973 it started operations of a briquetting plant for the production of one million tons per year (nominal capacity) of 86.5 percent metallized briquettes. This plant was constructed under the terms of a joint venture between the owners of the patent, the U.S. Steel Corp., and the Venezuelan government which has the option to buy 25 percent of the plant after two years of operations. The method of production is called the High Iron Briquettes (H.I.B.), and so far the plant has been producing at only 30 percent of its nominal capacity due to technical problems. Both companies, O.M.C. and I.M.C.O.V., use the open pit method of mining.

The third iron ore concession, located at San Isidro, belongs to Siderurgica del Orinoco (Sidor), a subsidiary of a government corporation called Corporation Venezolana de Guayana (C.V.G.). Also owned by Sidor are the most important steel works of the nation. The steel production of Sidor (approximately 1,160,000 tons per year) represents about 85 percent of the nation's total production. The ore mined at San Isidro is transported by truck (150 Km) to Cuidad Guayana where the steel works are located. During the year 1972, one million tons of ore were mined at San Isidro. All of the briquettes so far produced by H.I.B. have been used by Sidor.

Table VII shows the relative importance of these three companies, listing output, domestic sales and exports:

TABLE VII

Iron Ore Companies Operating in Venezuela
(Cum. data is up to 1971 in %. Totals in million tons)

Production				Exports			Sales in Ven.		
IMCOV	OMC	GOV	TOTAL	IMCOV	OMC	TOTAL	IMCOV	OMC	TOTAL
19.0	80.5	0.6	285	20.0	80.0	276	26.0	74.0	10

Sources: Companies reports.

In 1973 a completely independent private concern authorized Arthur G. McGee and Company to begin with the engineering studies to construct a 400,000 metric tons per year (up to 92.5 percent metallized iron ore) plant. The process used is called the Fluidized Iron Ore Reduction (F.I.O.R.), which was developed by the Esso Research Engineering Company. No details have been published about the progress of the construction of this plant. Similar plants were planned to operate three years after the construction started, and in some cases it took two or three years longer to operate at half capacity. The F.I.O.R. process was tested in a demonstration plant build in Darmouth, Nova Scotia. The capacity of the plant was 300 tons/day, and during four years of operation it produced 50,000 metric tons, or 12.6 percent of the operation at full capacity (McGee, A.G., 1973).

The Plans to Increase Capacity

The long range plans to increase the steel production in Venezuela have been divided in two equal periods of five years each. The first phase of the program comprises from 1975 to 1979, and the second phase from 1980 to 1984.

There are two main ways to increase steel production in a given industry: the blast furnace-basic oxygen furnace (B.F. and B.O.F.) and the direct reduction-electric arc furnace (D.R. and E.A.F.). The goal of the first phase of the program is to increase the total steel production by 1,200,000 metric tons per year. Due to the favorable existing conditions with respect to the availability of iron ore, electricity, and natural gas, it has been decided to use the D.R. and E.A.F. processes to increase steel production. It also has been determined that it is more economical to use in Venezuela the B.F. and B.O.F. way, whenever the producion increase exceeds the 5,000,000 metric tons per year level (oral communication, 1974).

In order to accomplish the production increase the following plants will be constructed: a pelletizing plant that consists of 2 units of 3,000,000 tons per year each, and 11 direct reduction units for a total of 3,840,000 metric tons per year of sponge iron. The task here is to point out the relevant technical, and to a greater extent, the economic differences between the two direct reduction processes chosen by Sidor, the HyL and Midrex.

TECHNICAL FACTORS THAT AFFECT THE SELECTION OF A D.R. PROCESS

The end product of direct reduction is the briquette. In Venezuela the briquettes are going to be used in the electric arc furnaces already in operation, planned, or under construction. The efficiency by which the electric furnace uses the electricity to melt the briquettes is related to the briquette chemical composition. The gangue content requires electricity for its fusion. Also, if some reduction (oxygen removal) must be made, extra electricity must be used in the electric furnace.

Therefore, the methods that produce the briquettes with the highest degree of metallization (lowest oxygen content), has the economic advantage.

The purpose of this section is to summarize the different types of direct reduction processes, the characterization of the metallized pellets, and the metallized end products--the briquettes.

Also summarized in this section are the reasons why the degree of metallization should be the deciding technical factor in the selection of a specific method in contrast with any other factors.

Table VIII shows how the different terms are defined and used to characterize the metallized products of direct reduction.

The total iron content determines the selling price of the briquette. This content is shown in Table VIII, and it is the basis for the production analysis in the last chapter.

Direct reduction processes can be classified on the basis of the type of iron ore feed, the kind of reducing agent used, the equipment, and the physical and chemical characteristics of the end product. Fig. 1 shows how the iron ore feed, the processes themselves, and the end products are characterized.

Iron Ore and Metallized End Product Characteristics

For the conversion of iron ore to crude steel there are three essential tasks:

1. The separation of the iron from oxygen,
2. fusion of the iron and the gangue, and
3. separating out the contaminant elements.

In the conventional process, combination blast furnace-basic oxygen steel plant, the blast furnace performs the first two tasks.

In the process combination direct reduction-electric furnace, the direct reduction only takes over the first task.

TABLE VIII

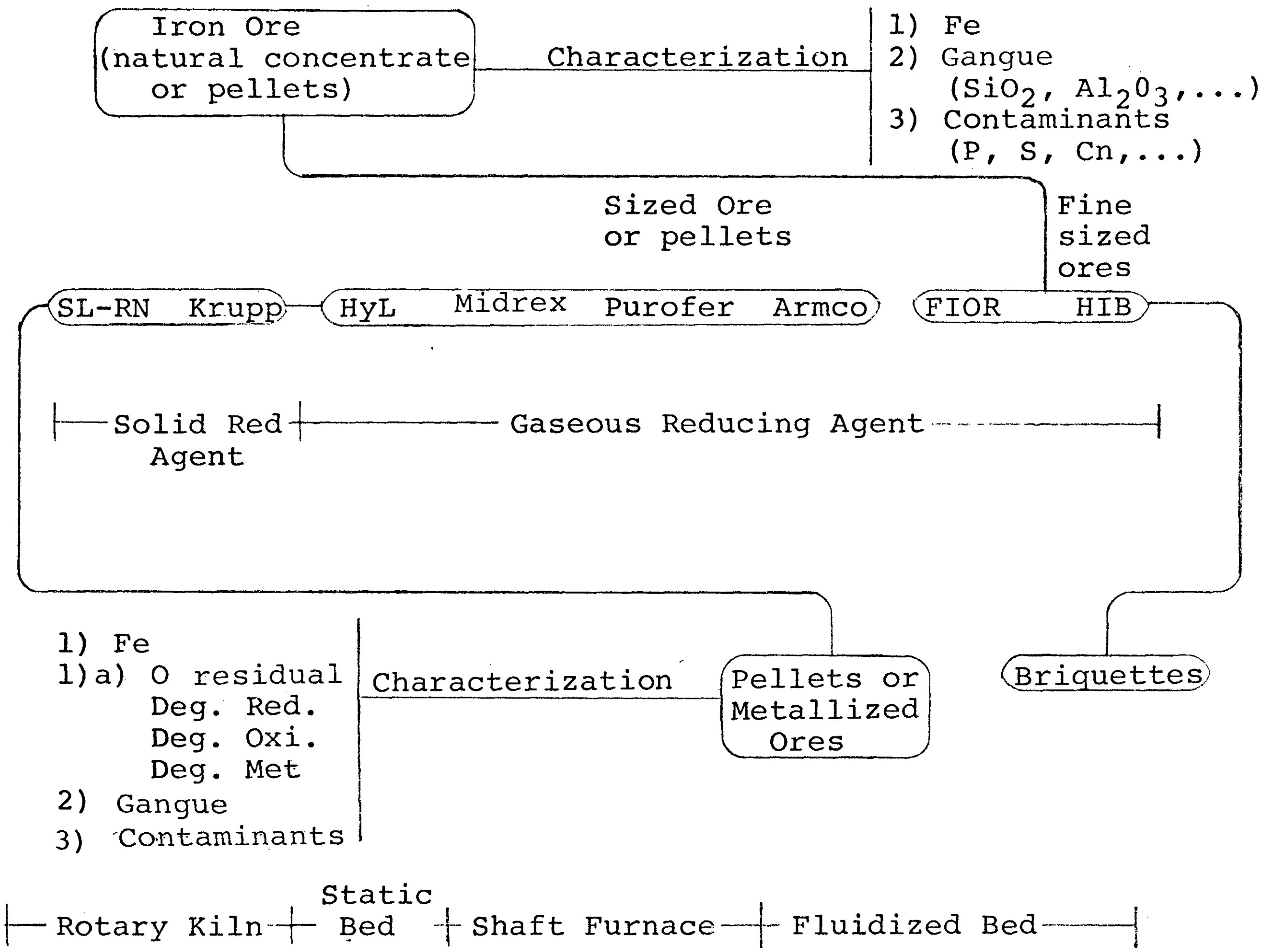
Index of Reduction or Metallization of Pure Iron Oxide

Degree of metallization $= \frac{\text{Fe metal}}{\text{Fe total}}$	75	80	85	90	95	100
Degree of reduction $= \frac{\text{O removed}}{\text{O max assoc with Fe}}$	83.33	86.66	90.00	93.33	96.66	100
Degree of oxidation = 100 - deg. red.	16.66	13.33	10.00	6.66	3.33	0
Composition of iron-iron-oxide phases						
Fe total %	93.33	94.59	95.89	97.22	98.59	100
Fe metal %	70.00	75.67	81.50	87.50	93.66	100
Fe^{++} %	23.33	18.92	14.39	9.72	4.93	0
O %	6.66	5.41	4.11	2.78	1.41	0
Index $\frac{O}{Fe}$ %	7.14	5.72	4.29	2.86	1.43	0

Source: Astier, J.E., 1973

FIGURE I

Characterization of iron ores and direct Reduction Processes



Source: Adapted from Astier, J. E., and others.

The melting of the iron and the gangue is carred out by the electric furnace, which requires electrical energy for this purpose instead of coke. Also, if some reduction of iron oxide is required this is, of course, possible, but it will lead to the use of carbon as reduction agent and increased consumption of electrical energy (about 35 Kw-hr per 0.010 per metric ton 0_2). Otherwise the iron oxide will be slagged off, decreasing the iron yield. A difference of 5% in the degree of metallization represents an increase of

$$\frac{35\ \text{KWH}}{0.010\ \text{MT of } 0} \times 4795\ \text{M.T. of } 0 \times \frac{0.01\$}{\text{KWH}} = \$167{,}825.00$$

in the operating costs of the electric furnace.

Gangue content is a direct consequence of the analysis of the raw materials, and there is no increase or decrease during the prereduction process. Therefore, one of the technical deciding factors that contributes to reach a decision for a particular D.R. method is the oxygen residual or degree of metallization attained by the method.

Summarizing, the prereduction materials to be used in electric arc furnaces will tend to approach residual oxygen in the range of 2.86 index (0/Fe/ton) or less. This means as far as possible, degrees of metallization over 90 percent, degrees of reduction over 93 percent. (from Astier, J.E.)

The Hojalata y Lamina Direct Reduction Process

HyL is a process with more than 15 years of commercial operating experience. A historical profile of HyL plant construction is shown in Table IX.

TABLE IX

HyL Plant Construction (Mexico)

Location	Year	Nominal Cap. (mt/yr)	Status	Actual Prod.	% Above Capacity	Improvements Made
Monterrey	1955	9,900	?	?	?	Pilot
Monterrey	1957	75,000	Operating	95,000	25	Flexibility
Monterrey	1960	190,000	Operating	260,000	37	Cold and carburized
Veracruz	1967	190,000	Operating	220,000	15	-
Puebla	1970	250,000	Operating	315,000	26	Thermal efficiency
Bahia (Brazil)	1974	225,000	Under Const.	-	-	-
Monterrey	1972	420,000	Under Const.	-	-	-

Source: Sada, C. G., 1974.

Ore reduction occurs in four reactor vessels, which are coupled to gas preheaters and quench towers (see fig. 2a). Conveyors move the ore from storage bins to charge hoppers. A complete reducing cycle for each reactor requires 12 hours and the stages for conversion are as follows: (1) preheating raw ore with partially spent reducing gas (secondary gas)-3 hours; (2) reducing the preheated ore to sponge (exposure to primary gas-3 hours; (3) cooling the sponge with fresh, cool reducing gas, 3 hours; (4) unloading the cooled sponge iron and reloading with raw ore, 3 hours.

The quenched gas is heated to above 1600°F when entering the reactor. The reaction temperature may be varied between about 1600° and 1900°F, depending upon ore reducibility; therefore, the product is not subject to reoxidation to an appreciable degree.

Since the charge remains stationary throughout the reduction cycle, the degree of reduction within each reaction chamber is not completely uniform at the end of the operation. Typically, the final degree of reduction varies from 96 percent at the top to 73 percent at the bottom of the ore bed in the retort. The average degree of reduction varies between 85 and 90 percent. The normal operation of the steelworks at Monterrey had its economic optimum at about 83 percent metallization. Due to the varying degrees of

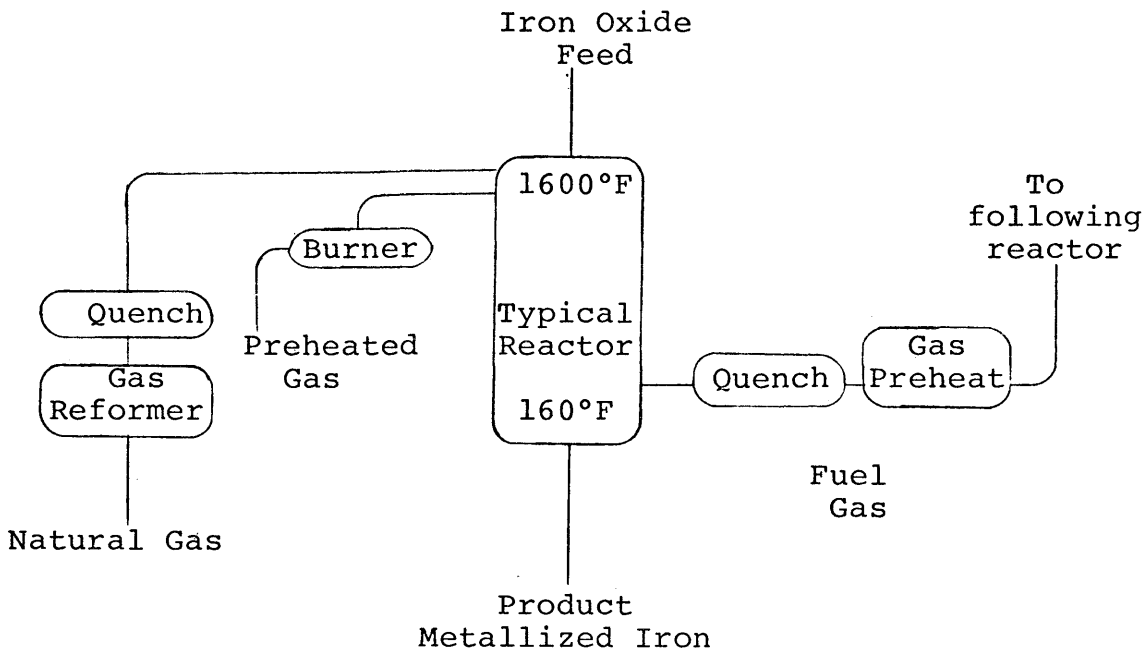


Figure 2a. In the HyL process, the charge remains stationary in the reactors through the reduction cycle.

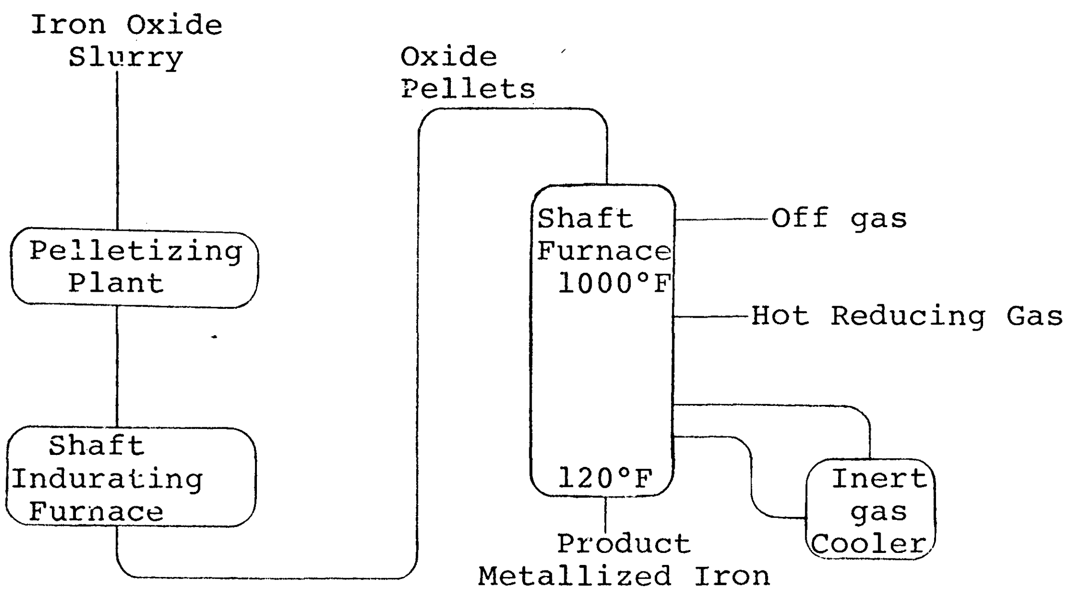


Figure 2b. The Midrex process normally uses an oxide pellet charge which passes downward through a shaft-type furnace.

Source: Both figures from Bertram, J.M., pp. 36 and 37.

reduction, it is desirable to blend the sponge iron before it is used in steelmaking.

Table X shows the chemical analysis of the iron ore used and the resulting analysis of the reduced pellets at two reduction levels. The 85.1 percent metallization column represents the present operating average of the Puebla plant, and the 90 percent metallization column gives the results obtained during periods in which it has been operated at that level.

TABLE X

Analysis of Iron Ore and Sponge Iron at Pueblo Operation

	Iron Ore	Sponge Iron	
Metallization	-	85.1	90.0
Total Iron	67.0	87.2	88.1
Gangue	3.3	4.3	4.4
Lime (added)	1.5	1.9	2.0
Carbon	-	2.2	2.2
Oxygen	28.7	2.9	2.0

Source: Sada, C. G., p. 6.

Table XI shows typical results obtained with HyL sponge iron charge in a 17 ft. dia. electric arc furnace, producing low carbon steel. This performance data reveal a considerable operational improvement over the initial days of sponge iron usage in electric arc furnaces. The Table shows two different charge ratios of sponge iron to scrap, Cases I and II.

TABLE XI

The HyL Sponge Iron in Steelmaking

	1967		1969		1974
	Case I	Case II	Case I	Case II	Case I
Furnace Charge:					
Scrap, %	45	15	40	15	40
Sponge Iron, %	55	85	60	85	60
Ingot Yield, %	88	83	91.4	93	92.4
Tap-to-tap time, min	320	350	304	320	172
Lining life, heats	100	80	120	115	164
Roof life, heats	40	30	43	43	55
Consumption per MT of Ingot Produced:					
Elec. power, Kwh	740	860	680	790	625
Electrodes, Kg	7.85	9.65	8.0	9.7	5.2
Lime, Kg	82	98	82	100	7
Dolomite, Kg	10	14	10.5	115	N.A.
Magnesite, Kg	5	6	5.2	6.6	N.A.
Refractories, Hg	10	12	10.0	10.4	N.A.
Ferro-Alloys, Kg	5.8	5.8	5.8	5.8	N.A.
Lab. and Sup., man-hrs.	1.9	2.1	1.9	2.0	N.A.

Source: 1967: Kobrin, C.L., p. 75.
1969: HyL, Am. Correspondent, p. 123.
1974: Sada, C. G., p. 7.

The Midrex Direct Reduction Process

Two 4,000,000-ton per year plants are now operating in the U.S. One is located at Portland, Oregon, and the other one at Georgetown, North Carolina. A third plant with the same capacity is now operating at Hamburg, West Germany. Another plant of this size is being tested in Contrecoeur, Canada. Two 800,000-ton per year plants are under construction: one at Baton Rouge, Louisiana, and the other one at Osaka, Japan (Mashlanka, W.). A seventh plant is under construction in Kostamuska, Finland; its capacity is unknown.

The Midrex process normally uses an oxide pellet charge which passes downward through a shaft-type furnace. The second step in the process is the metallizing of the oxide pellets. The reduction occurs in the upper section of the furnace. As the pellets descend through the metallizing section they pass hydraulically-operated cylinders, using a slow stroke of one-half hour to break up any clustering that may have developed (see fig. 2b).

Hot reformed gas passes counterflow to the bed of pellets which is descending through the upper section of the furnace. The counterflow principle is of particular importance in order to achieve an optimized chemical and thermal utilization of the reducing gas. By means of the hot gas the pellets get heated up and metallized within the range of 6.5 hours while passing through the furnace.

Until today, the Hamburgh reduction plant has exclusively processed Swedish Malmberget pellets (Mashlanka, W., pp. 17-19). Due to their analysis and their chemical and physical properties, the pellets produced in the shaft furnace from a magnetite concentrate are qualified especially for direct reduction or as metallized iron ore for further processing in the electric arc furnace. Some of the characterizations of Malberget pellets are shown in Table XII.

TABLE XII

Malmberget Pellets-Before and After Direct Reduction (%)

	Malmberget Pellets	Metallized Product
Fe Total	57.78	92-96
Fe met	-	91
Deg. of met	-	95
SiO_2	1.55	2.00
Al_2O_3	0.48	0.70
P	0.022	0.025
S	0.002	0.010

Source: Mashlanka, W., p. 17, 18.

The chemical composition of the metallized iron ore when using Malmberget pellets appears relatively consistent as indicated by the values shown in Table XII. Typically, the degree of metallization varies from 92 to 99 percent

(94 to 99 percent degree of reduction). After having started with trial operations in October 1971, continuous operation of the Hamburg plant actually began in March 1972. By November 1972, approximately 170,000 tons of metallized pellets were produced.

The carbon content of the metallized product varies between the following ranges.

Campaign A aim 0.8%	n = 57
Campaign B aim 1.0%	n = 88
Campaign C aim 1.2%	n = 148
Campaign D aim 1.6%	n = 66

Source: Maschlanka, W., and others, p. 20.

The data above and in Table XII show that the chemical composition of the metallized iron ore using Malmberget pellets appears relatively consistent. To mill workers engaged in metallized pellet processing it is of advantage to obtain the carbon required for certain steel grades already continaed in the charged materials (Maschlanka, 1972).

THE ECONOMIC PARAMETERS

The basic variables needed to make a discounted cash flow rate of return model are: inputs and outputs, selling prices, fixed and variable costs of production, investment costs, economic life of the project, and income tax rate.

The situation of the iron industry in Venezuela is that of a price taker. The selling price of the briquettes must therefore be competitive in the international markets. The international trade of metallized pellets and briquettes is a fairly new one, and statistical data are nonexistent. Technically, one of the main objectives of briquette production is for it to be a substitute for scrap.

A rational way to calculate the market price internationally will be to compare the amount of scrap substituted by the briquettes and multiply it by the scrap value. This opportunity cost is the first calculation made in this section.

The second part of this section deals with the calculation (by interpolation) of the investment and fixed costs for the two processes under study here.

The costing was made for the Metallurgical Plan by a group of Venezuelan and international engineers. The basis for the calculation were the published data in specialized magazines, brochures of the processes, and other sources of information.

Price Calculation

In order to calculate the selling price of the pellets for prereduction (oxidized pellets) and the briquettes (metallized pellets), a method similar to the one used by O.M.C. (O.M.C. Oct., 1973) to calculate the selling price of the 86.5 percent Fe briquettes, will be used here. The calculation is as follows:

(1) Average selling price of Scrap No. 1 in Philadelphia, according to the quotations of the "American Metal Market," during the first five days of each quarter (\$/gross ton).

(2) The above average times the amount of scrap replaced by one gross ton of pellets.

$$\text{Amount} = \frac{(\text{Pellets Fe Cont.}) \times (\text{Yield in the elec. furnace})}{(98\% \text{ Fe in Scrap}) \times (94\% \text{ yield in elec. furnace})}$$

Result: Value of the replaced scrap by one gross ton of pellets (\$/gross ton).

(3) Subtract the additional cost incurred in the melting of a gross ton of pellets in place of the cost of melting a gross ton of scrap. [This amount always fluctuates

according to the performance (e.g. carbon content) of the metallized products in the furnace].

Result: Value of the pellets that replace the scrap.

(4) Subtract the ocean freight between Puerto Ordaz and Philadelphia, including the ship unloading charges.

(5) Divide by the percentage of iron content of pellets.

Result: Value of pellets (f.o.b. ship) in Puerto Ordaz, including the Orinoco river toll, that replaces the scrap in Philadelphia ($ per Fe unit per gross and metric ton respectively).

The only available published data for the calculation of step (3) above are the yield of the HyL process: 92.4 percent (see Table XI). There are no published data for the calculation of step number 4. Therefore, some appropriate assumptions will be made.

The O.M.C. briquettes (86.5 percent Fe) have an additional cost of $10.66 per gross ton, as compared with the melting of an equal amount of scrap. These briquettes have an ingot yield of 90.7 percent.

The cost data of Tables XIII through XVI (the Metallurgical Plan), were calculated on an equal basis: 90 percent metallization, a pellet yield of 93 percent, and an efficiency of 90 percent (330 days of continuous operation).

The iron content of the oxidized pellets will be assumed to be 65 percent, which is the degree of metallization claimed by one of the bidding corporations (Marubeny and Kobe Steel Ltd.) for the construction of the oxidized pellet plant.

On this basis, the following assumption will be used to calculate the selling price of the oxidized pellets and briquettes:

Ingot yield for the oxidized pellets: 80 percent.

Ingot yield for the metallized pellets: 92 percent.

Additional cost for the oxidized pellets: $15.00.

Additional cost for the metallized pellets: $9.00.

Accordingly, the estimated oxidized pellets selling prices are as follows:

(1) Estimated average selling price of Scrap No. 1 for the first quarter of 1974, according to the prices of the first five days of October 1973 (Am. Met. Market) $62.90/GT

(2) $(\$62.90/\text{GT} \times \frac{(65\%\ \text{Fe})(80\%)}{(98\%\ \text{Fe})(94\%)} = \$35.51/\text{GT}$

(3) ($35.51/GT)-($15.00GT) = $20.51/GT

(4) ($20.51/GT)-($4.70/GT) = $15.81/GT

(5) $15.81/GT:65% Fe = $0.243/UGT

= $0.247/UMT

The calculation of metallized pellets selling price is as follows:

(1) Estimated average selling price of Scrap No. 1 for the first quarter of 1974, according to the prices of the first five days of October 1973 (Am. Met. Market): \$62.90/GT.

(2) $(\$62.90/GT) \times \frac{(90\% \ Fe)(92\%)}{(98\% \ Fe)(94\%)} = \$56.54/GT$

(3) $(\$56.54/GT) - (\$9.00/GT) = \$47.54/GT$

(4) $(\$47.54/GT) - (\$4.70/GT) = \$42.84/GT$

(5) $\$42.84/GT : 90\% \ Fe = \$0.476/UGT$

$= \$0.484/UMT$

Operating and Investment Costs

The operating requirements and projected operating costs for the HyL and Midrex processes are shown in Tables XIII and XV, respectively. These tables were quoted from the Metallurgical Plan and converted to U.S. dollars at an exchange rate of Bs 4.30 per dollar. Also included in these two tables are the fixed annual costs for the two processes. Tables XIV and XVI contain the estimated investment costs.

The quoted investment cost belongs to the production levels of 234,000, 504,000, 1,008,000 M.T. per year. Therefore, it was necessary to make diagrams in order to interpolate the fixed and investment costs for the 350,000 tons per year of production level which is the size of the eleven units to be constructed (See dotted lines in figs. 3 to 6).

They are:

Fixed costs for the HyL process (@ 350,000 tons/yr)
=$280,000/yr.

Fixed Costs for the Midrex process (@ 350,000 tons/yr)
=$217,000/yr.

Investment cost for the HyL process (@ 350,000 tons/yr)
=$22,000,000

Investment cost for the Midrex process (@ 350,000 tons/yr)
=$17,000,000

In the fixed annual costs it is estimated that there is no change in the number of employees* as the production level increases. Labor* and maintenance do increase. According to the authors of the cost estimates, the other fixed cost items (e.g. overhead) are included within the items listed in the tables. At this point it must also be noted that the Venezuelan mining laws do not consider depletion allowance.

The investment cost includes natural gas reforming plant, mineral reduction, control houses, instruments, compressors, raw material handling equipment, end product delivery equipment, and internal mineral transportation equipment.

In figure 5 two extra interpolations are made, but they are related to calculations contained in the following chapter.

*The term employees is used for salaried individuals, while labor is hourly workers.

TABLE XIII

Estimated Operating Costs for the HyL Process

Fixed Annual Costs ($/yr)

Prod. Level (tons/yr)	234,000	504,000	1,008,000
Employees	55,535	55,535	55,535
Labor	140,651	200,930	301,395
Maintenance	48,324	69,388	97,049
Total	244,510	325,853	453,979

Variable Costs ($/tons of product)

Item	Unit	Cost $/Unit	Requirement per MT Fe	Cost $/MT Fe
Oxidized Pellets	MT	16.08	1.396	22.42
Natural Gas	m^3x10^3	4.83	0.5886	2.84
Electric Power	Kwh	0.01	9.16	0.09
Water	m^3x10^3	5.81	0.00386	0.02
Materials	$/t. of prod.	-	-	1.49
Catalyst and Chemicals	$/t. of prod.	-	-	0.17
License	$/t. Fe met	1.02	0.824	0.84
Total				27.87

Source: The Metallurgical Plan

Figure 3

Estimated Fixed Costs for the HyL Process

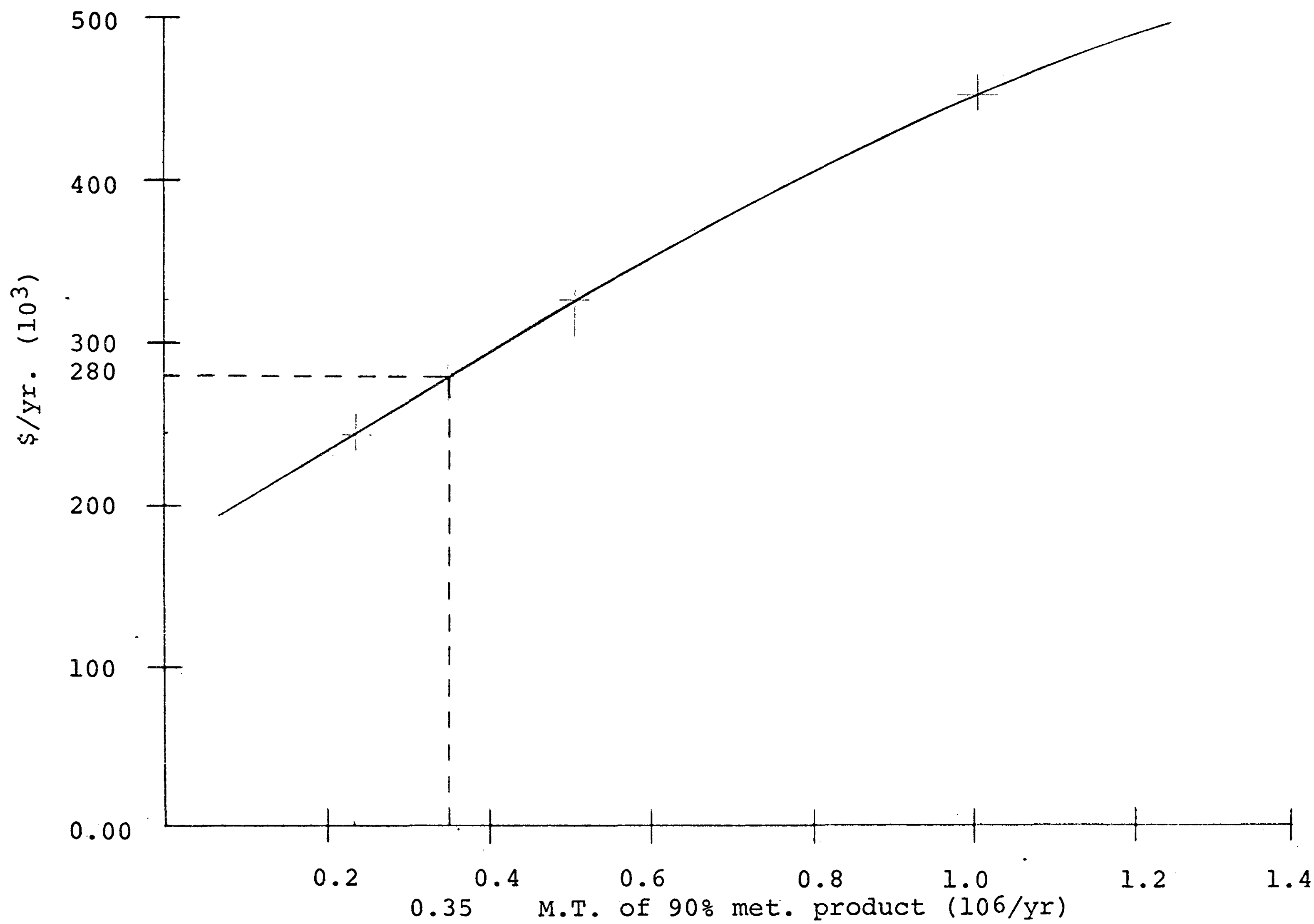


TABLE XIV

Estimated Investment Costs for the HyL Process
(In mill. U.S. dollars)

Prod. Level (tons/yr)	234,000	504,000	1,008,000
Equip. and parts	9.30	16.51	27.51
C.I.F.	0.70	1.23	2.07
Installation and startup	1.16	2.07	3.44
Sub-total	11.16	19.81	33.02
Mechanical Services	0.93	1.65	2.74
Electric Services	0.23	0.42	0.70
Buildings and structures	1.16	2.07	3.44
Infrastructures	1.40	2.49	4.14
Engineering	0.70	1.23	2.07
Sub-total	4.42	7.86	13.09
Total	15.58	27.67	46.11

Source: The Metallurgical Plan

Figure 4.

Estimated Investment Costs for the HyL Process

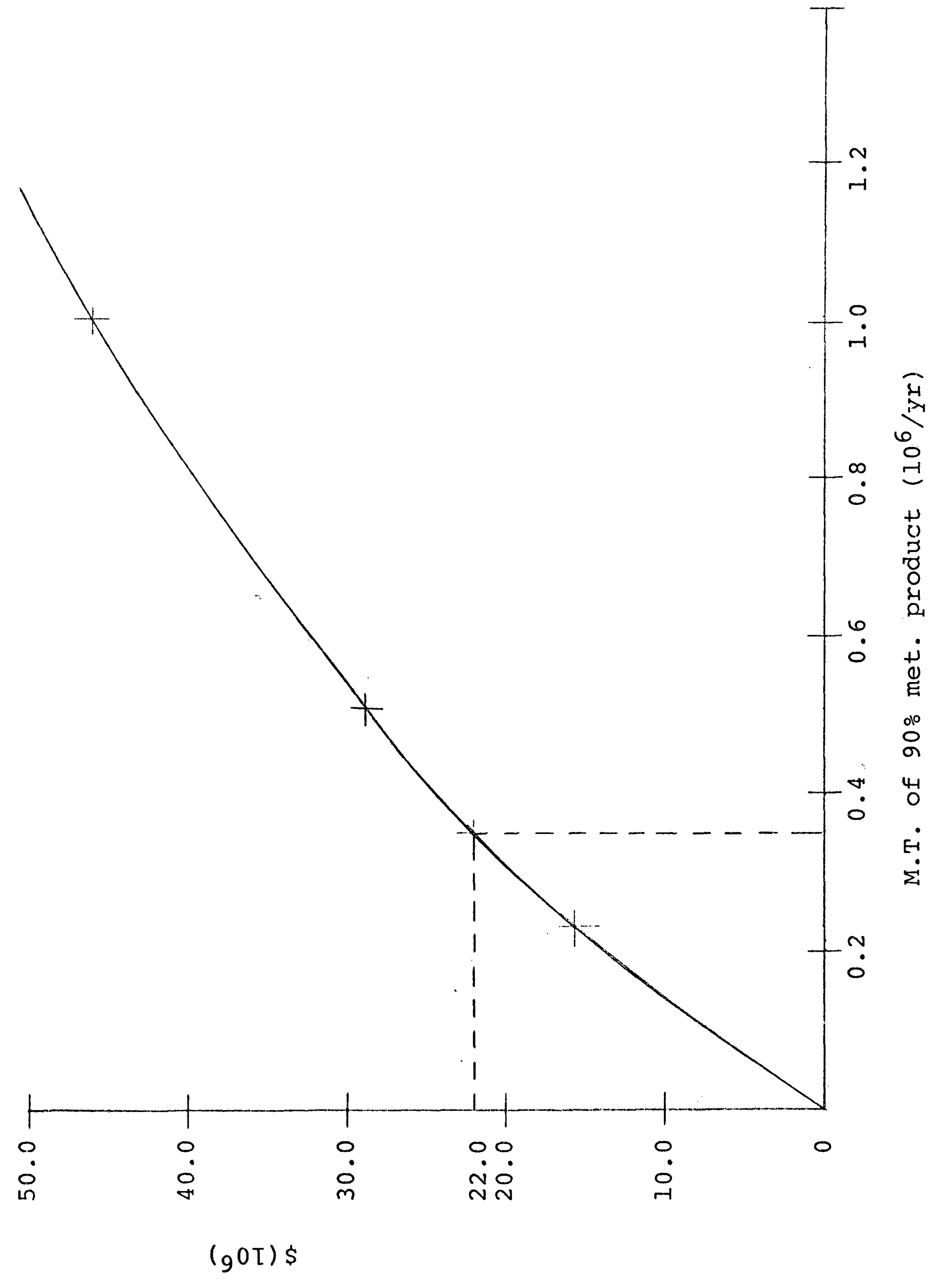


TABLE XV

Estimated Operating Costs for the Midrex Process

Fixed Annual Costs ($/yr)

Prod. Level (tons/yr)	234,000	504,000	1,008,000
Employees	55,535	55,535	55,535
Labor	80,372	120,558	160,744
Maintenance	48,324	69,388	97,049
Totals	184,231	245,480	313,328

Variable Costs ($/tons of products)

Items	Unit	Cost $/unit	Requirement per MT Fe	Cost $/MT Fe
Oxidized pellets	MT	16.08	1.396	22.42
Natural gas	m^3x10^3	4.83	0.352	1.70
Electric power	Kwh	0.01	140	0.81
Water	m^3x10^3	5.81	0.01	0.06
Materials	$/t. of prod.	-	-	1.49
Catalyst and Chemicals	$/t. of prod.	-	-	0.10
License	$/t. Fe met.	1.02	0.824	0.84
Total				27.42

Source: The Metallurgical Plan

Figure 5

Estimated Fixed Costs for the Midrex Process

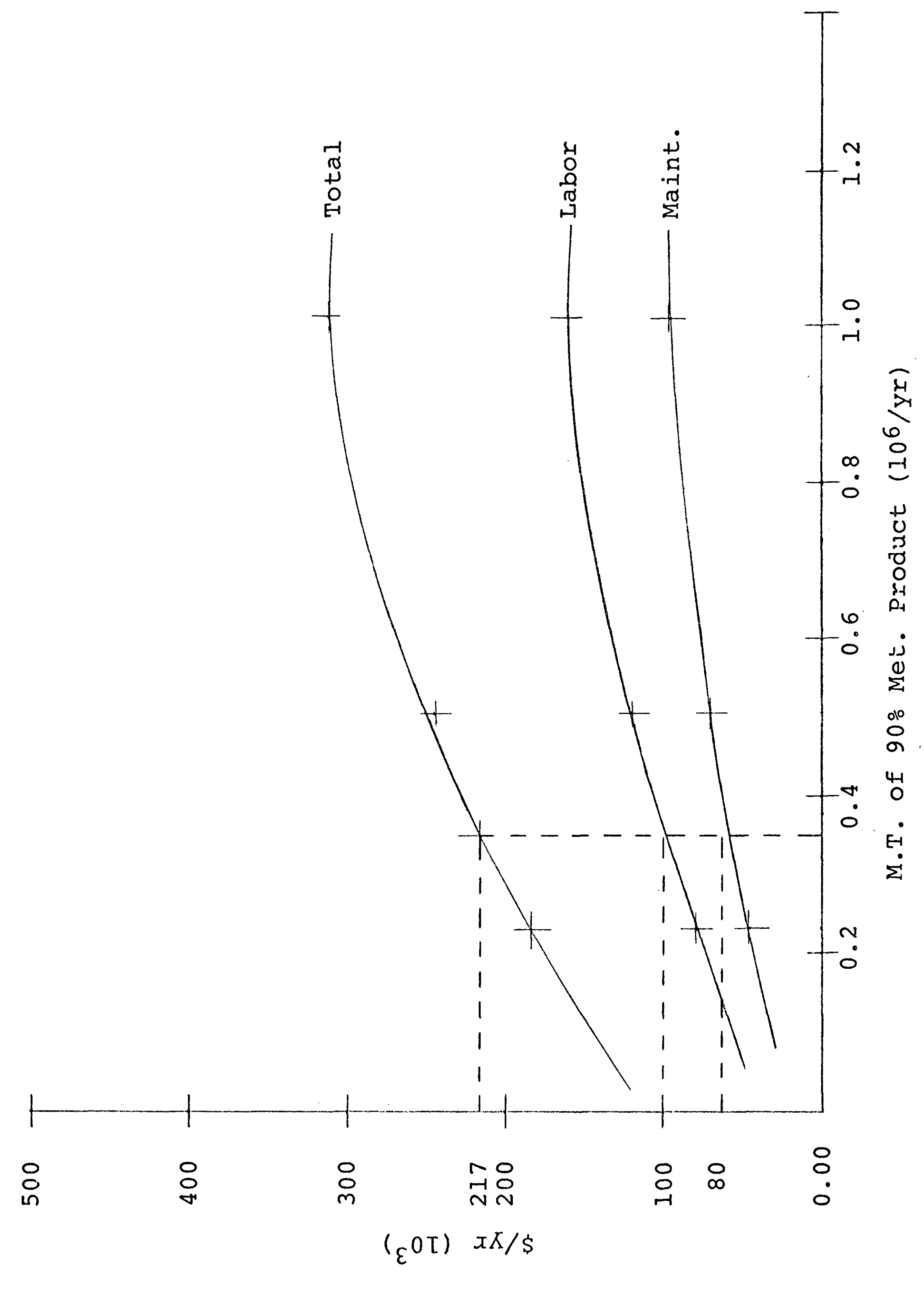


TABLE XVI

Estimated Investment Costs for the Midrex Process
(in million U.S. dollars)

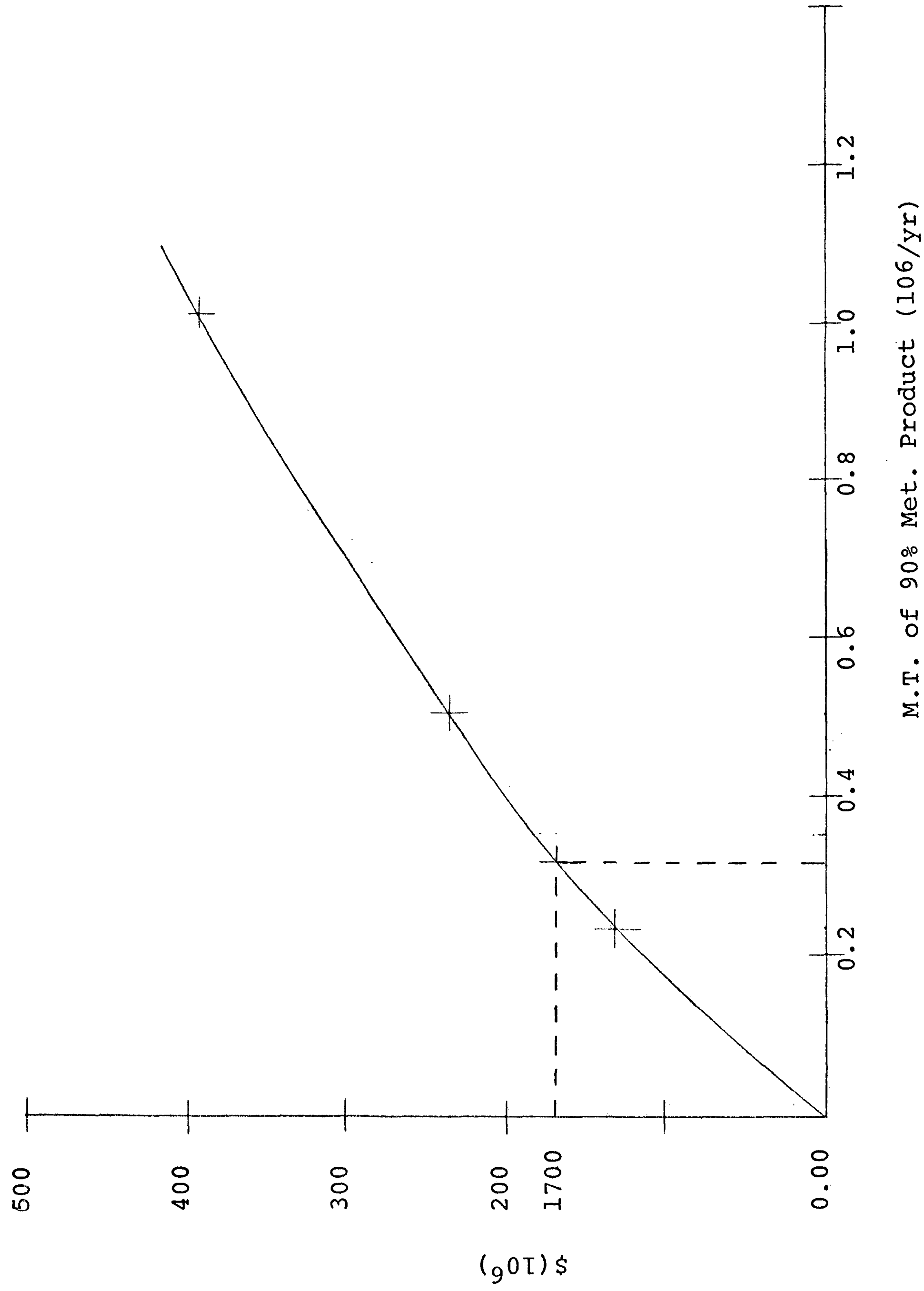
Prod. Level (tons/yr)	234,000	504,000	1,008,000
Equipment and parts (59.68)	7.95	14.12	23.53
C.I.F. (4.50)	0.60	1.07	1.77
Installation and startup (7.51)	1.00	1.77	2.93
Sub-total (71.70)	9.55	16.96	28.23
Mechanical services (5.93)	0.79	1.42	2.35
Electrical services (1.43)	0.19	0.35	0.58
Buildings & structures (7.51)	1.00	1.77	2.93
Infrastructures (8.93)	1.19	2.12	3.53
Engineering (4.50)	0.60	1.07	1.77
Sub-total (28.30)	3.77	6.73	11.16
Total	13.32	23.69	39.39

Source: The Metallurgical Plan

Note: Numbers in parenthesis indicate percent of total.
See Table XVIII for 350,000 MT/yr.

<u>Figure 6</u>

Estimated Investment Costs for the Midrex Process



ECONOMIC SELECTION BETWEEN THE HyL AND MIDREX PROCESSES

At this point we can make a preliminary selection between the two processes, using the economic parameters studied so far. From the tables and diagrams of the previous section, we can observe that the Midrex method has several accounting advantages. The following differences are in favor of the Midrex process: \$63,000/yr in fixed costs, \$5,000,000 in investment costs, and \$157,500 (\$0.45/ton) in variable costs. Table XVII shows a summary of these economic variables. The economic life of the project is assumed to be 13 years. For these types of projects, the Venezuelan government requires a 58 percent income tax rate. The depreciation is calculated on a straight line basis as a percent per year on net investment.

Lines 8 through 12 in the screening program consist of the calculations necessary to determine the capital recovery rate (line 13), which can be translated directly into a percentage ROI by reference to the graph in the book by Park, W.R., p. 65. The graph is entered on the vertical axis at the computed capital recovery rate, and the ROI is read on the

TABLE XVII

Screening between the HyL and Midrex
[350,000 M.T. (yr)]

Line	Item	Units	Source	HyL	Midrex
1	Average yr/sales	$/yr	Input	15,246,000	15,246,000
2	Direct prod. costs	$/yr	Input	7,700,000	5,950,000
3	Ind. & overhead costs	$/yr	Input	280,000	217,000
4	Net investment	$	Input	22,000,000	17,000,000
5	Economic project life	years	Input	13	13
6	Income tax rate	%	Input	58	58
7	Average depreciation	$/yr	4/5	1,692,308	1,307,692
8	Total deductions	$/yr	2+3+7	9,672,308	7,474,692
9	Net profits before tax	$	1-8	5,573,692	7,771,308
10	Income taxes	$	6x9	3,232,741	4,507,359
11	Net profit after tax	$	9-10	2,340,951	3,263,949
12	Net cash flow	$	11+7	4,033,259	4,571,641
13	Capital recov. rate	Ratio	12/4	0.183	0.269
14	ROI	%	Graph*	15.5	25.5

* Park, W.R., p. 65.

Source: Line 1: Price (43.56) x Prod. Level.
Line 2: Tables XIII and XV.
Line 3: Figures 3 and 5
Line 4: Figures 4 and 6

horizontal scale for the appropriate project life. A difference of 10 percent in ROI also favors Midrex. Table XVIII shows a detailed estimate of the fixed and investment costs that will be used in the economic analysis of the Midrex process that follows in the next section (see Figure 6 and Table XVI).

Economic Analysis of the Midrex Direct Reduction Process

The selection of the Midrex process, based on the rate of investment (ROI) calculations done in the previous section is an approximate method to economically evaluate a project. The ROI obtained reinforces the advantages in costs (fixed, variable, and investment) of the Midrex process. It seems to be the more economic method under the conditions assumed by the group that made the calculations in the Metallurgical Plan.

In this chapter a more accurate rate will be obtained by making a detailed financial and discounted cash flow rate of return analysis.

The Financial Analysis: The financial model used (Park, W.R., pp. 238-246) is shown in Table XIX through XIV (Schedules A-F). In every analysis of this type several assumptions must be made. The actual conditions of the project will be known only when it is underway, when the assumed conditions on this preinvestment analysis must be adjusted.

TABLE XVIII

Estimated Fixed Costs for the Midrex Process
(350,000 ton/yr capacity; U.S. Dollars)*

Costs	
Employees	55,535
Labor	100,000
Maintenance	61,465

Estimated Investment Costs for the Midrex Process
(350,000 M.T./yr; in mill U.S. Dollars)**

Costs	
Equip and parts	10.15
C.I.F.	0.77
Inst. and startup	1.28
Sub-total	12.19
Mechanical Serv.	1.01
Electrical Serv.	0.24
Building and structures	1.28
Infrastructures	1.52
Engineering	0.77
Sub-total	4.81
Total ($\$x10^6$)	17.00

Sources:* Figure 5
** Table XVI and figure 6

Schedule A (Table XIX) shows that after the plant construction and startup (3 years) continuous operation for 13 years will follow. For the first and second year of operation the production level is assumed to be at 70 and 90 percent, respectively, and at 100 percent thereafter. The product's selling price, initially $0.484/1% Fe/MT, increases $0.01 during the sixth, nineth, and fourteenth years since the start of the project. Selling costs are estimated to be 10.0 percent of net sales.

Schedule B (Table XX) shows the manufacturing costs. All unit costs, except license, increase 10 percent during the nineth and fourteenth years. The period costs (or time-related charges) stay at $217,000 per year throughout the project's operating life.

Schedule C (Table XXI) shows the capital requirements by years for the project. Year One represents the start of the building construction and equipment assemblage (10 percent of both), also includes 20 percent of the engineering research and development work; Year Two includes most of the actual plant construction, along with additional engineering work; and Year Three sees the completion of the plant, followed by startup and initial production. Working capital requirements in the example are estimated at 10 percent of total manufacturing cost for inventories and at 10 percent of net sales for accounts receivable. The capital requirements for years four to fifteen represent only the additional working capital required each year to keep up with increases in sales and manufacturing costs.

Schedule D (Table XXII) shows the depreciation charges for the project. Buildings are depreciated over a 15-year period, and equipment over 10 years. Depreciation charges are computed on a straight-line basis. The rate applied to buildings is 7 percent, and to equipment 10 percent. Buildings have a salvage value of $746,667 at the end of the project.

Schedules E and F (Tables XXIII and XXIV) bring together the data from the previous schedules to provide for the computation of net profit after taxes (in schedule E) and cash flow (in schedule F). Income taxes are estimated at 58 percent of the net profit before taxes. In schedule F, in year 16, the inflow shown represents the return of working capital (year 15), and the undepreciated balance of the building account, upon completion of the project. All other figures are self explanatory.

TABLE XIX

Schedule A: Sales and Income Forecast
(100% Equity Capital, Metric Tons and U.S.$)

Year	Units Sold	Per Unit	Net Sales	Selling Costs	Net Income From Sales Per Unit	Net Income From Sales Annual
4	245,000	43.56	10,672,200	1,067,220	39.20	9,604,980
5	315,000	43.45	13,721,400	1,372,140	39.20	12,349,260
6	350,000	44.46	15,561,000	1,556,100	40.01	14,004,900
7	350,000	44.46	15,561,000	1,556,100	40.01	14,004,900
8	350,000	44.46	15,561,000	1,556,100	40.01	14,004,900
9	350,000	45.36	15,876,000	1,567,600	40.82	14,288,400
10	350,000	45.36	15,876,000	1,567,600	40.82	14,288,400
11	350,000	45.36	15,876,000	1,587,600	40,82	14,288,400
12	350,000	45.36	15,876,000	1,587,600	40.82	14,288,400
13	350,000	45.36	15,876,000	1,587,600	40.82	14,288,400
14	350,000	46.26	16,191,000	1,619,100	41.63	14,571,900
15	350,000	46.26	16,191,000	1,619,100	41.63	14,571,900

TABLE XX

Schedule B: Manufacturing Costs

		Unit Costs				Period Costs			Total Operat. Costs	
Year	Units Produced	Raw Matrls.	Power & Water	Chem. & Lic.	Total	Fixed Overhead	Main-tenance	Total	Per Ton	Annual
4	245,000	24.12	0.87	2.43	27.42	155,535	61,465	217,000	28.31	6,934,900
5	315,000	24.12	0.87	2.43	27.42				28.11	8,854,300
6	350,000	24.12	0.87	2.43	27.42				28.04	9,814,000
7	350,000	24.12	0.87	2.43	27.42				28.04	9,814,000
8	350,000	24.12	0.87	2.43	27.42				28.04	9,814,000
9	350,000	26.53	0.96	2.59	30.08	same	same	same	30.70	10,745,000
10	350,000	26.53	0.96	2.59	30.08				30.70	10,745,000
11	350,000	26.53	0.96	2.59	30.08				30.70	10,745,000
12	350,000	26.53	0.96	2.59	30.08				30.70	10,745,000
13	350,000	26.53	0.96	2.59	30.08				30.70	10,745,000
14	350,000	29.19	1,05	2.76	33.00				33.62	11,767,000
15	350,000	29.19	1.05	2.76	33.00				33.62	11,767,000

Note: Includes an allowance for equipment replacement.

TABLE XXI

Schedule C: Capital Requirements

Year	Depreciable Invest. Buildings	Depreciable Invest. Equipment	Expensed Investment Engineering R&D	Expensed Investment Startup	Working Capital Receivables	Working Capital Inventory	Total Capital Requirements
1	250,000	1,092,000	404,000	0	0	0	1,776,000
2	2,240,000	8,736,000	1,212,000	0	0	0	12,188,000
3	280,000	1,092,000	404,000	1,280,000	0	0	3,056,000
4					1,067,220	693,490	1,760,710
5					1,372,140	885,430	496,860
6					1,556,100	981,400	279,930
7					1,556,100	981,400	0
8					1,556,100	981,400	0
9					1,587,600	1,074,500	124,600
10					1,587,600	1,074,500	0
11					1,587,600	1,074,500	0
12					1,587,600	1,074,500	0
13					1,587,600	1,074,500	0
14					1,619,100	1,176,700	133,700
15					1,619,100	1,176,700	0
Total	2,800,000	10,920,000	2,020,000	1,280,000	18,283,860	12,249,020	19,815,800

TABLE XXII

Schedule D: Depreciation Allowances

Year	Buildings (15 yrs) Balance	Buildings (15 yrs) Amount	Equipment (10 yrs) Balance	Equipment (10 yrs) Amount	Total Balance	Total Amount
4	2,800,000	186,667	10,920,000	1,092,000		1,278,667
5	2,613,333	186,667	9,828,000	1,092,000		1,278,667
6	2,426,667	186,667	8,736,000	1,092,000		1,278,667
7	2,240,000	186,667	7,644,000	1,092,000		1,278,667
8	2,053,333	186,667	6,552,000	1,092,000		1,278,667
9	1,866,667	186,667	5,460,000	1,092,000		1,278,667
10	1,680,000	186,667	4,368,000	1,092,000		1,278,667
11	1,493,333	186,667	3,276,000	1,092,000		1,278,667
12	1,306,667	186,667	2,184,000	1,092,000		1,278,667
13	1,120,000	186,667	1,092,000	1,092,000		1,278,667
14	933,333	186,667	0	-		186,667
15	746,667	186,667	0	-	746,667	186,667

TABLE XXIII

Schedule E: Income Statement

Year	Net Income from Sales	Operating Costs	Net Operating Income	Depre-ciation	Expensed Investment	Net Profit Before Taxes	Income Taxes	Net Profit After Taxes
1	0	0	0	0	404,000	(404,000)	-	(404,000)
2	0	0	0	0	1,212,000	(1,212,000)	-	(1,212,000)
3	0	0	0	0	1,684,000	(1,684,000)	-	(1,684,000)
4	9,604,000	6,934,900	2,669,100	1,278,667	0	1,390,433	806,451	583,982
5	12,349,260	8,854,300	3,494,960	1,278,662	0	2,216,293	1,285,445	930,843
6	14,004,900	9,814,000	4,190,900	1,278,667	0	2,912,233	1,689,095	1,223,138
7	14,004,900	9,814,000	4,190,900	1,278,667	0	2,912,233	1,689,095	1,223,138
8	14,004,900	9,814,000	4,190,900	1,278,667	0	2,912,233	1,689,095	1,223,138
9	14,288,400	10,745,000	3,543,400	1,278,667	0	2,264,733	1,313,545	951,188
10	14,288,400	10,745,000	3,543,400	1,278,667	0	2,264,733	1,313,545	951,188
11	14,288,400	10,745,000	3,543,400	1,278,667	0	2,264,733	1,313,545	951,188
12	14,288,400	10,745,000	3,543,400	1,278,667	0	2,264,733	1,313,545	951,188
13	14,288,400	10,745,000	3,543,400	1,278,667	0	2,264,733	1,313,545	951,188
14	14,571,900	11,767,000	2,804,900	186,667	0	2,618,233	1,518,575	1,099,658
15	14,571,900	11,767,000	2,804,000	136,667	0	2,618,233	1,518,575	1,099,658

TABLE XXIV

Schedule F: Cash Flow

Year	Net Profit After Taxes	Depreciation	Net Cash Inflow	Net Cash Outflow	Net Cash Flow	Cumulative Cash Flow
1	(404,000)	0	0	1,776,000	(1,776,000)	(1,776,000)
2	(1,212,000)	0	0	12,188,000	(12,188,000)	(13,964,000)
3	(1,684,000)	0	0	3,056,000	(3,056,000	(17,020,000)
4	583,982	1,278,667	1,862,649	1,760,710	101,939	(16,918,061)
5	930,848	1,278,667	2,209,515	496,860	1,712,655	(15,205,406)
6	1,223,138	1,278,667	2,501,805	279,930	2,221,875	(12,983,531)
7	1,223,138	1,278,667	2,501,805	0	2,501,805	(10,481,726)
8	1,223,138	1,278,667	2,501,805	0	2,501,805	(7,979,921)
9	951,188	1,278,667	2,229,855	124,600	2,105,255	(5,874,666)
10	951,188	1,278,667	2,229,855	0	2,229,885	(3,644,811)
11	951,188	1,278,667	2,229,855	0	2,229,855	(1,414,956)
12	951,188	1,278,667	2,229,855	0	2,229,855	814,899
13	951,188	1,278,667	2,229,855	0	2,229,855	3,044,754
14	1,099,658	186,667	2,378,325	133,700	2,244,625	5,289,379
15	1,099,658	186,667	2,378,325	0	2,378,325	7,667,704
16	0	0	3,542,467	0	3,542,467	11,210,171

<u>Discounted Cash Flow Rate of Return Analysis</u>: The profitability index (or DCFROR) of the project is calculated as shown in Table XXV and figure 7. The present worth of the net cash flow is calculated at 6.5 and 6.0 percent discount rates. In order to avoid a trial and error calculation, the following equation has been deduced by straight triangles from figure 7.

$$\frac{b-a}{y+z} = \frac{x-a}{y}$$

from which,

$$x = a + \frac{y}{y+z}(b-a)$$

and substituting,

$$x = 6 + \frac{270{,}935}{270{,}935+293{,}531}(0.5) = 6.24\%$$

This interest rate represents the highest rate at which the required capital can be borrowed and paid off from the cash earnings generated by the project over its economic life. The balance of $4855.00 can be regarded as small as compared with the yearly cash flows.

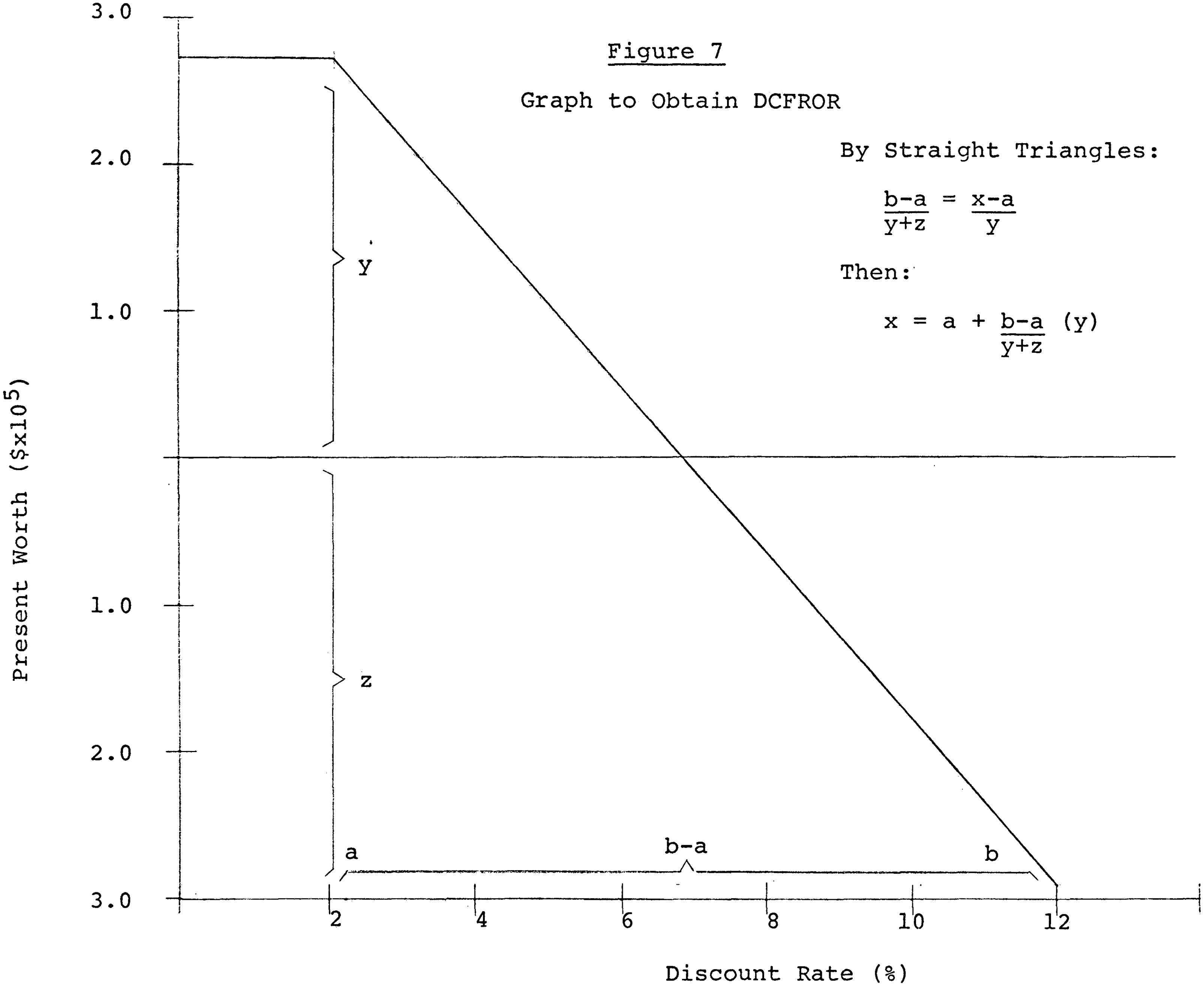
TABLE XXV

Discounted Cash Flow Rate of Return

Year	Discounted at 6.5%	Discounted at 6.24%	Discounted at 6.0%
1	(1,667,606)	(1,671,687)	(1,675,472)
2	(10,745,663)	(10,798,323)	(10,847,277)
3	(2,529,907)	(2,548,527)	(2,565,877)
4	79,240	80,018	80,745
5	1,250,034	1,265,405	1,279,796
6	1,522,727	1,545,223	1,566,334
7	1,609,927	1,637,710	1,663,843
8	1,511,669	1,541,519	1,569,663
9	1,194,423	1,220,990	1,246,097
10	1,187,902	1,217,296	1,245,139
11	1,115,401	1,145,798	1,174,660
12	1,047,325	1,078,499	1,108,170
13	983,403	1,015,154	1,045,443
14	929,500	961,858	992,800
15	924,756	959,291	992,392
16	1,293,338	1,344,921	1,394,479
Balance	-293,531	-4,855	270,935

Figure 7

Graph to Obtain DCFROR



By Straight Triangles:

$$\frac{b-a}{y+z} = \frac{x-a}{y}$$

Then:

$$x = a + \frac{b-a}{y+z}(y)$$

The Production Theory Analysis: The financial and rate of return analyses consider the development of the project through time. The production analysis determines the efficiency by which the input factors (natural gas, chemicals, and so on) are being used within the process. This analysis results also in an optimum economic level of maximum profit, according to the existing market conditions or selling price of the end product.

Only two data points are available for the calculations. One is at 90° and the other at 70° of metallization. Moreover, the 70° of metallization data corresponds to a different model of plant than the one at 90°. Although not a realistic data, it is the only information available to develop the model that shows the relationships among the costs of production, market selling prices, profits, and the technical efficiency attained by the factory under study here. Table XXVI shows the variable costs for an operation at 70° metallization.

The following column references are addressed to the Table XXVII, entitled production analysis.

The natural gas consumption was estimated from tables XIII, XV, and XXVI, column 1.

The iron ore used by the direct reduction plants comes in the forms of oxidized pellets, with an average degree of metallization of 65 or an approximate iron content of

TABLE XXVI

Variable Costs for the Midrex Process

(70% metallization and 350,000 MT/yr)

Items	Unit	Cost $/Unit	Requirement Per MT Fe	Costs $/MT Fe
Oxidized pellets	MT	16.08	1.327	21.34
Natural Gas	m^3x10^3	4.83	0.251	1.21
Electric Power	Kwh	0.058	100.0	0.58
Water	m^3x10^3	5.81	0.010	0.058
Materials	$/t. of prod.			1.49
Catalysis & Chemicals	$/t. of prod.			0.07
License	$/t. Fe met.	1.02	0.609	0.62
Total				25.37

Source: The Metallurgical Plan.

TABLE XXVII

Production Analysis

(1) X Natural Gas m^3	(2) Y Lab. maint. em. $	(3) q Production M.T.	(4) VC $	(5) FC (2)x($1) $	(6) STC (4)+(5) $	(7) AP_{VC} (3)÷(4) MT/$	(8) AVC (4)÷(3) $/MT	(9) MP Δ(3)÷Δ(4) MT/$	(10) SMC Δ(6)÷Δ(3) $/MT
-	217,000	262,000	5,161,400	217,000	5,378,400	0.0507	19.70		
-	217,000	267,000	5,372,040	217,000	5,589,040	0.0497	20.12	0.0237	42.13
-	217,000	272,000	5,589,600	217,000	5,806,600	0.0486	20.55	0.0229	43.51
-	217,000	277,000	5,814,230	217,000	6,031,230	0.0476	20.99	0.0222	44.93
-	217,000	282,000	6,046,080	217,000	6,263,080	0.0466	21.44	0.0216	46.37
-	217,000	287,000	6,282,430	217,000	6,499,430	0.0456	21.89	0.0212	47.27
-	217,000	292,000	6,529,120	217,000	6,746,120	0.0447	22.36	0.0202	49.34
-	217,000	297,000	6,783,480	217,000	7,000,480	0.0438	22.84	0.0197	50.87
-	217,000	302,000	7,045,660	217,000	7,262,660	0.0429	23.33	0.0191	52.44
-	217,000	307,000	7,312,740	217,000	7,529,740	0.0420	23.82	0.0187	53.42
-	217,000	312,000	7,590,960	217,000	7,807,960	0.0412	24.33	0.0180	55.64
227	217,000	317,000	7,877,450	217,000	8,094,450	0.0403	24.95	0.0174	57.30
251	217,000	322,000	8,172,360	217,000	8,389,360	0.0394	25.38	0.0169	58.98
270	217,000	327,000	8,475,840	217,000	8,692,840	0.0386	25.92	0.0164	60.70
290	217,000	332,000	8,788,040	217,000	9,005,040	0.0378	26.47	0.0160	62.44
324	217,000	337,000	9,112,480	217,000	9,329,480	0.0370	27.04	0.0155	64.89
366	217,000	342,000	9,442,620	217,000	9,659,620	0.0363	27.61	0.0151	66.03
439	217,000	347,000	9,785,400	217,000	10,002,400	0.0355	28.20	0.0146	68.56
-	217,000	349,500	9k960,750	217,000	10,177,750	0.0351	28.50	0.0028	70.14

TABLE XXVII (continued)

Production Analysis

(11) AFC (5)÷(3)	(12) SAC (5)÷(3)	(13) TR (P=$44.46)	(14) Π (P=$44.46)	(15) MΠ (P=$44.46)	(16) TR (P=$68.56)	(17) Π (P=$68.56)	(18) MΠ (P=$68.56)	(19) Total Fe (%)	(20) Deg. Metal (%)
$/MT	#/MT	$	$	$	$	$	$		
0.828	20.53	11,648,520	6,270,120		17,962,720	12,584,320		74.86	4.93
				2.33			26.43		
0.813	20.93	11,870,820	6,281,780		18,305,520	12,716,480		76.29	10.37
				0.95			25.05		
0.798	21.35	12,093,120	6,386,520		18,648,320	12,841,720		77.71	15.76
				-0.47			23.63		
0.783	21.77	12.315,420	6,284,190		18,991,120	12,959,890		79.14	21.20
				-1.91			22.19		
0.770	22.21	12,537,720	6,274,640		19,333,920	13,070,840		80.57	26.64
				-2.81			21.29		
0.756	22.65	12,760,020	6,260,590		19,676,720	13,177,290		82.00	32.07
				-4.88			19.22		
0.743	23.10	12,982,320	6,236,200		20,019,520	13,273,400		83.43	37.51
				-6.41			17.69		
0.731	23.57	13,204,620	6,204,140		20,362,320	13,361,840		84.86	42.94
				-7.98			16.12		
0.719	24.05	13,426,920	6,164,260		20,705,120	13,442,460		86.29	48.38
				-8.96			15.14		
0.707	24.53	13,649,220	6,119,480		21,047,920	13,518,180		87.71	53.78
				-11.18			12.92		
0.696	25.03	13,871,520	6,053,560		21,390,720	13,582,760		89.14	59.21
				-12.84			11.26		
9.685	25.54	14,093,820	5,999,370		21,733,520	13,639,070		90.57	64.65
				-14.52			9,58		
0.674	26.05	14,316,120	5,926,760		22,076,320	13,686,960		92.00	70.08
				-16.24			7.86		
0.664	26.58	14,538,420	5,845,580		22,419,120	13,726,280		93.43	75.52
				-17.98			6.12		
0.654	27.12	14,760,720	5,755,680		22,761,920	13,756,880		94.86	80.96
				-20.43			3.67		
0.644	27.68	14,983,020	5,653,540		23,104,720	13,775,240		96,29	86.39
				-21.57			2.03		
0.635	28.25	15,205,320	5,545,700		23,447,520	13,787,900		97.71	91.79
				-24.10			0.00		
0.625	28.83	15,427,620	5,425,220		23,790,320	13,787,920		99.14	97.22
				-25.68			-1.58		
0.621	29.12	15,538,770	5,361,020		23,961,720	13,783,970		99.86	99.96

91 percent. Therefore all the calculations below 64.65 degrees of metallization (90.57 Fe total) were estimated, (columns 19 and 20). The relationship between degree of metallization and total iron content was assumed to follow a straight line relationship. The following straight line equation was obtained with a coefficient of correlation of 0.9997. The values for the equation were taken from Table VIII:

$$y = 3.80\,x - 279.62$$

where x represents the total iron content and y corresponds to the degree of metallization attained by the end product. An exponential relationship was assumed between the degree of metallization and the variable costs. The following exponential equation was obtained with a correlation coefficient of 1.0000. Values from Table XV and XXVI are:

$$y = 19.33e^{0.0039x}$$

where x represents the degree of metallization and y the average variable cost, (column 8).

In order to construct the general model, a short run analytical period in which at least one factor of production cannot be varied was assumed.

Columns 2 through 6 are self explanatory. The intervals in column 3 were chosen to be constant, so as to make the comparison realistic on a per unit basis.

Column 7 is the average productivity of the variable factors, or how many iron units are produced by each dollar of the variable inputs. The average variable costs (8) increases as the average productivity decreases. An analogous relation exists between the marginal productivity (9) and the marginal costs (10).

Therefore, what we have here is a case of decreasing returns to the variable inputs. The reason for it is because the input factors are the only variable here at an assumed fixed price per unit of output. So that when the average productivity falls, the cost per unit of output (AVC) must increase. Likewise, when the marginal productivity of inputs decline, the extra costs per unit of output (SMC) must rise.

Fixed costs being constant, AFC (column 11) continuously declines as output is increased.

Column (12) represents the cost per unit of output, and it equals AVC + AFC.

Columns 13 through 18 show the method of obtaining the price levels at which the final end product must be sold for profit maximization. Columns 13, 14, and 15 are based on the going market price of $44.46/1%/Fe. Columns 16, 17, and 18 are based on the profit maximization price of $68.56/1%/Fe.

Columns 13 and 16 of total revenues represent relevant price times quantity. This will appear graphically as a straight line from the origin with slope (MR) equal to price.

Columns 14 and 17 show the profits (TR-STC) obtained at the two mentioned prices. The maximum profits correspond to those points where P $\equiv$ MC $\equiv$ MR. That is, where the slopes of the total costs and profit curves are equal. This can be clearly seen by the tendency of the values and also at the zero values obtained in columns 15 and 18, respectively. Beyond the zero marginal profit the slope of the curve becomes negative, indicating that the maximum point has already been reached.

CONCLUSIONS

The required raw materials for direct reduction are plentiful and easily available in Venezuela. The three iron ore mines that are presently under exploitation have proved reserves of 1182 million tons. At the present rate of extraction of 25 million ton/year, these reserves will last for 48 more years. Several tests had been run in different direct reduction plants under different processing conditions. The results have proven the suitability of the Venezuelan ore for this type of processing. However, these tests were run with the ore now available (see Table V), which is of prime quality. The variations in the chemical, mineralogical, and physical characteristics of the mineral must be closely controlled. This control will determine what magnitude of variations in the properties can the process under consideration tolerate without totally failing to produce quality product.

The HyL and Midrex processes use an average of 400 cubic meters of gas per ton of product. The plans of the Venezuelan government call for an increased capacity to 3.8 million tons per year of reduced iron. Thus, the higher capacity will utilize approximately 5 percent of the available natural gas with no apparent future use. The gas available also has the advantage of not being associated with oil production, since it will come from the Barcelona region where 83.0 percent of the gas produced is non-associated. If we omit the cost of the iron pellets, natural gas represents 60 percent of the variable costs of production. The economics of the entire process is thus essentially determined by the reducing gas. Therefore, a strict quality control must be applied with the reforming natural gas continually. This control will assure stoichiometric ratios of oxidation agents and hydrocarbons to be converted so that no wasting occurs due to improper mixing of reagents.

The two competing processes considered here, the HyL and Midrex, show noticeable differences in their historical background and present status of operation. The HyL process has a well established scaling-up trend since 1955. These plants that are not in operation, with a combined capacity of 705,000 MT per year are all operating at above-capacity levels. HyL plants under construction presently represent a capacity of 645,000 MT per year and they will increase overall total to 1,350,000 MT per year. The pilot plant and scaling up consideration for the Midrex method are less easily available. The plants currently operating under the Midrex patent have a combined capacity of 1,200,000 MT per year. No statement was made, however, of their actual production or duration of operating. Midrex plants under construction or in the stage of testing have an added capacity of 2,000,000 MT per year, and another plant of unknown capacity is under construction at Finland. The overall Midrex capacity will total at least 3,200,000 MT per year or approximately 150 percent above HyL capacity.

The pilot plant operating conditions must be known to determine the confidence level when evaluating the performance of the plant proposals at industrial production levels. Not all the problems are noticeable in a small-scale pilot plant, unless it is run for a sufficiently long duration on a continuous basis.

The final end product is to be utilized in the electric arc furnace. The economic utilization of the electric furnace is highly sensitive to the chemical and physical characteristics of the metallized products. Both direct reduction methods produce a solid product and also do not reduce the gangue content. The contaminant elements such as phosphorous sulphur, copper, etc., continue to be present even after the product goes through the furnace. The difference of the end product quality exists in the oxygen residual and the carbon content. A low oxygen residual will require less carbon for reduction and thus decrease the electrical energy consumption.

The Midrex direct reduction process has a higher average degree of reduction, or conversely, a lower degree of oxidation average. The carbon that must be added for steelmaking is shown on Table X and p. 31 for the Hyl and Midrex processes, respectively. The product quality is more consistent and better controlled with the Midrex method. In fact, due to the varying degrees of reduction in the HyL method, the product must first go through a blending procedure in order to achieve the required uniformity. This is due to the stationary status of the material during reduction with HyL, as compared to the continuously downward movement of the charge with the Midrex process. However, relatively few publications are available on the behavior of the Midrex product in steelmaking as compared with the several published reports of the HyL product (e.g. the ones in Table XI).

The basic chemical and thermodynamic principles are essentially the same for both methods. Both methods use reformed natural gas converted to hydrogen for the reduction process. Both methods have provisions for the recycling of the hydrogen, because it is not possible to utilize all the hydrogen introduced in the reactor the first time through.

The HyL method operates between 1600 and 1900° F, and the Midrex method operates above 1000° F (not specified). The discharge temperatures are 150° and 120° F for the HyL and Midrex, respectively. A complete reducing cycle within an HyL reactor requires 12 hours as opposed to 6.5 hours with the Midrex method. Processes in which the reactions and transfer rates are slow will always cost more to build and to operate than those in which the reaction and transfer rates can be speeded up. These increased expenses are reflected in the investment and operating figures shown for HyL in Tables XIII and XIV.

The screening program in Table XVIII shows a definite economic advantage of the Midrex over the HyL process. The two methods' direct and indirect costs, and the net investment are compared on an equal basis of yearly annual sales, project life, and income tax rates. The calculation of items seven through twelve result in a higher capital recovery rate and rate of return on investment for the Midrex process.

The financial, rate of return, and profit maximization analyses result in a very low rate of return and a relatively low operation level with respect to the maximum profit conditions.

The average variable costs per unit of output (AVC) increase due to the decreasing productivity of the inputs.

Likewise, the rate of change of total cost with respect to changes in output (SMC) is also increasing due to the decreasing marginal productivity of the inputs. Where the marginal productivity of the inputs equals zero total productivity of the inputs is at a maximum.

After this point, further utilization of the inputs factors becomes redundant. In our case, this point is between 97.22 and 99.96 degrees of metallization. This will be the optimum technical position, although practically it cannot ever be achieved. The exponential equation used in the text will still give theoretically valid values at 100° of metallization. With more data available, the actual shape of the curve could be better determined.

The economic optimum at an assumed selling price of \$68.56 is between the 91.79 and 97.22 levels of metallization. According to the difference between the profits at these two levels of metallization, which is only \$20.00, it is safe to assume that profit maximum is at the midpoint, or at 94.5° of metallization.

The going market price shows a hypothetical optimum of below a 10 percent degree of metallization. Below the 65 percent level, the behavior of the input factors and the quality of the end products is estimated, and they are presented here in order to compare the two situations.

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