A METALLURGICAL EVALUATION OF THE SNAKE RIVER IRON DEPOSIT

Prepared For

CREST EXPLORATION LIMITED

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J. E. Lawver

J. E. Lawver, E. Met., Sc. D., P. E.

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INTRODUCTION

Iron ore shipments during World War II reduced United State's reserves to the point that extensive world-wide exploration programs were instigated to assure the domestic industry adequate supplies of iron ore. The success of the exploration program is manifested by the known reserves today which are adequate to supply the world for at least 150 years, despite the fact that the requirements are expected to double within the next 30 years. Further, there exists almost unlimited amounts of iron-bearing material that can eventually be used as technology improves.[1] Thus, the question facing the steelmaking industry is not where to find iron ore but rather which sources should be developed. The answer, of course, is one of economics, and ore is purchased on a price-quality basis with due consideration given to the political stability of the country selling the ore.

The Snake River deposit is one of many recent and significant deposits that must be evaluated. This report is a critique of the metallurgical testwork carried out on Snake River material, and a presentation of the writer's views concerning the economic potential of beneficiating the deposit to produce salable products for today's iron ore market with emphases on products that could be used for the most likely market—Japan.

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(1) The Making, Shaping, and Treating of Steel. United States Steel Corporation, 8th Edition. "Economics and the Marketing of Iron Ores," pp. 177-179.

SUMMARY AND CONCLUSIONS

The Snake River Iron Formation (not yet a proven ore) is huge and easily mined with little or no stripping. The formation has the further advantage of lending itself to selective mining in terms of phosphorus control, although there are no sections sufficiently low in phosphorus to eliminate the necessity of phosphorus removal by beneficiation. However, this material is very difficult to concentrate due to the extreme fineness of the silica in the hematite layers and due to the relatively high phosphorus content in the iron formation. These difficulties, combined with the adverse consequences arising from the necessity to build a railroad to the ocean and the fact that there is no existing town or public power system, more than offset the mining advantages and probably preclude the immediate development of this deposit by conventional mineral dressing schemes directed to producing conventional furnace burdens such as sinter feed or pellets.

However, since the deposit exists in an area that is likely to have a cheap source of fuel such as gas or oil[2] the deposit should be considered in terms of producing a prereduced agglomerate for blast furnace burden or in producing a sponge iron product as a blast furnace supplement. Although the metallurgical testwork carried out on the samples is insufficient to either prove or disprove the feasibility of utilizing a direct reduction process, the data does show that:

(1) A partial concentrate can be made by simple gravity methods, and

(2) The process of direct reduction of the gravity concentrate followed by magnetic concentration will produce a high-grade sponge iron with an acceptable phosphorus content.[3]

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[2] Crest Exploration Engineers - Mr. L. Camp.

[3] Letter from J. S. Breitenstein, R-N Corp., March 23, 1964.

If Crest Exploration engineers are confident that cheap fuel will be available near the iron deposit, then the exploratory work carried out so far should be supplemented by a study of direct reduction processes applied to gravity concentrates. If this work is not carried out, one cannot define the economic potential of the deposit. The following table clearly shows that conventional blast furnace products are not apt to be economic, but that a prereduced product shows an economic potential.

Product	Capital Cost	Cost to Produce and Deliver the Concent rate to Japan	Estimated Selling Price of Product in Japan
Sinter feed produced by gravity 60% Fe, 9.5% SiO_2, 0.2% P	$109.0 x 10^6	$ 9.89	*$10.50
Pellets produced by gravity, flotation and leaching 63.6% Fe, 6.3% SiO_2, 0.07% P	271.5 x 10^6	15.13	15.90
Pellets produced by gravity and magnetic roasting 66% Fe, 6% SiO_2, 0.2% P	286.5 x 10^6	16.29	*15.85
R-N Sponge 94% Fe, 2.4% SiO_2, 0.1% P	264.0 x 10^6	27.47	40.00

*Based on Fe only. This product probably could not be sold at this price due to the high P content.

This conclusion is based on the fact that the most likely market is Japan — a country with an increasing demand for steel — but also a country that at the moment is enjoying a buyer's market as far as iron ore is concerned.**

**Recent discoveries of high-grade iron, low phosphorus deposits in India and Australia can provide both lump ore and sinter feed to the Japanese steelmakers in substantial quantities. Japan is also importing ore from Korea, China, Malaya, Philippine Islands, Canada, U.S.A., and South America as well as utilizing a small quantity of their own beach sands.

However, due to the high cost of metallurgical coke, * the Japanese iron and steelmakers will do almost anything to reduce their coke consumption. As a consequence, they are extremely interested in obtaining direct reduction products as a source of iron for steelmaking.

During the past decade there has been a sustained technical effort to bring direct reduction processes into commercial fruition. At present, the HyL direct gaseous reduction plant in Monterrey, Mexico, — Hojalata y Lamina — have started an 80-million-dollar expansion program to double their present production rate.[4] Two solid fuel direct reduction plants are planned in Canada and a gaseous fuel direct reduction plant is being planned in both Venezuela and India. The Japanese steelmakers have shown keen interest in all of the direct reduction processes because if these prove economic they will ameliorate their coke problem and provide increased steel production without building additional blast furnaces.[5]

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[4] The News, Mexico, D. F., February 23, 1965

[5] Private communication.

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*Coke costs between $25-$35 per ton in Japan, compared with about $14 in the U. S. A.

RECOMMENDATIONS FOR FUTURE METALLURGICAL TESTING

I have carefully reviewed the beneficiation test reports listed in the Addendum, and although the tests were not carried out to the point that one could actually design a plant, there was adequate data to make the order of magnitude cost estimates presented in the Summary and Conclusions. On the basis of economics alone, I strongly recommend that further testwork be carried out on direct reduction processes, and that no additional testwork be carried out along the lines of producing a conventional concentrate. There is little doubt that the procedures used to produce conventional concentrates could be improved, but it is most unlikely that the improvements would be so great that an economic process could be developed.

Direct reduction tests should be carried out on both raw ore and on gravity concentrates even though the testwork to date strongly indicates that the most likely flowsheet will be to reduce gravity concentrates.

The first step in designing a program of this nature is to select a fuel because the type of fuel used defines, at least to a large extent, the type of direct reduction process that can be used. I assume that Crest Exploration would consider using natural gas; thus, processes such as Esso or HyL should be studied.

The next step is to choose the end products. There is a definite market for sponge iron in Japanese foundries and open hearths, where the physical form of the sponge is much less important than the chemical purity. This market, however, is limited, and since a direct reduction plant at the Snake River Deposit would have to be large to be economic, it would be desirable to also produce a blast furnace burden. In this case, the physical properties of the product are extremely important.

Unfortunately, I believe that the only direct reduction briquets that have ever been tested in a blast furnace were tested by United States

Steel Corporation in their own experimental furnace and the results are not available to the public. Even if (as I would guess) the tests using a small experimental furnace were successful, the Japanese would not be apt to enter into a contract until direct reduction briquets have been tested using full-scale blast furnaces. This problem may soon be solved if United States Steel produces briquets in Venezuela, but it means that Crest Exploration has no alternative other than to study the production of sponge iron from their ore and simply wait to see if briquets prove feasible in the blast furnace.

In view of the size of the deposit and the funds that have already been spent to obtain a partial answer, I feel that it is a good risk for Crest Exploration to continue the study of utilizing their ore by direct reduction. The effort should be made at an attenuated rate by working with a company or companies that will realize a financial advantage if their particular direct reduction process is used. For example, arrangements could be made to carry out testwork with Esso in Toronto and/or the M. W. Kellogg Company (HyL) in New York City. The testwork would, of course, have to be coordinated by a Crest Exploration engineer familiar with the overall problem.

The most important fact to determine <u>early</u> in the direct reduction and post beneficiation tests is the phosphorus level in the final product. The data by R-N is not consistent; in one case — Breitenstein's letter of March 23 — the final product produced from the jig concentrate had an acceptable phosphorus level, while a more recent test — letter from Hains Engineering, Feb. 24, 1965 — showed that much of the phosphorus stayed with the concentrate after grinding through 0.06 mm (about 270 mesh). The phosphorus liberation size is quite fine — about 500 mesh — and further grinding <u>might</u> improve the product. However, from a theoretical view there is no reason to expect phosphorus removal through the conditions of direct reduction, and one is inclined to cast doubt on the analysis of the first test. Thus, the very first test program should be to confirm that a reasonable phosphorus level (0.1-0.13) can be obtained in a concentrate containing about 90 percent metallic iron. If the phosphorus specification cannot be obtained, the project should be dropped; if it can, then the project should be fully evaluated from an economic standpoint.

THE METALLURGICAL CHARACTERISTICS OF THE SNAKE RIVER DEPOSIT AND A BRIEF CRITIQUE OF THE METALLURGICAL TESTS

The geology and mineralogy of the Snake River Deposit is accurately and adequately described in the research reports 1, 2, 3, 4, and 10, listed in the Addendum. From a metallurgical standpoint the deposit is characterized by:

(a) Liberation of coarse bands of gangue at about 4 mesh will allow a partial concentration by simple gravity methods (see Addendum - Flow Sheet 1).

(b) The hematite-rich bands contain extremely fine silica which means that a low silica concentrate can be obtained only after grinding to about 80 percent minus 325 mesh.

(c) The phosphorus minerals are associated with both the gangue and the hematite-rich layers. Complete liberation of the phosphorus minerals requires grinding to about 500 mesh.

(d) The deposit is remarkably uniform in composition with the exception of the percent phosphorus which decreases with depth.

Considerable insight to the metallurgical characteristics of an ore can be obtained by carefully roasting a sample to magnetite and then determining its separation characteristics using a Davis tube. Dr. N. F. Schulz (Mines Experiment Station) determined the separation characteristics of both crude ore and jig concentrates using this procedure. Typical separation curves showing theoretical grade vs recovery are given on page 9. A complete report is given in the Addendum.

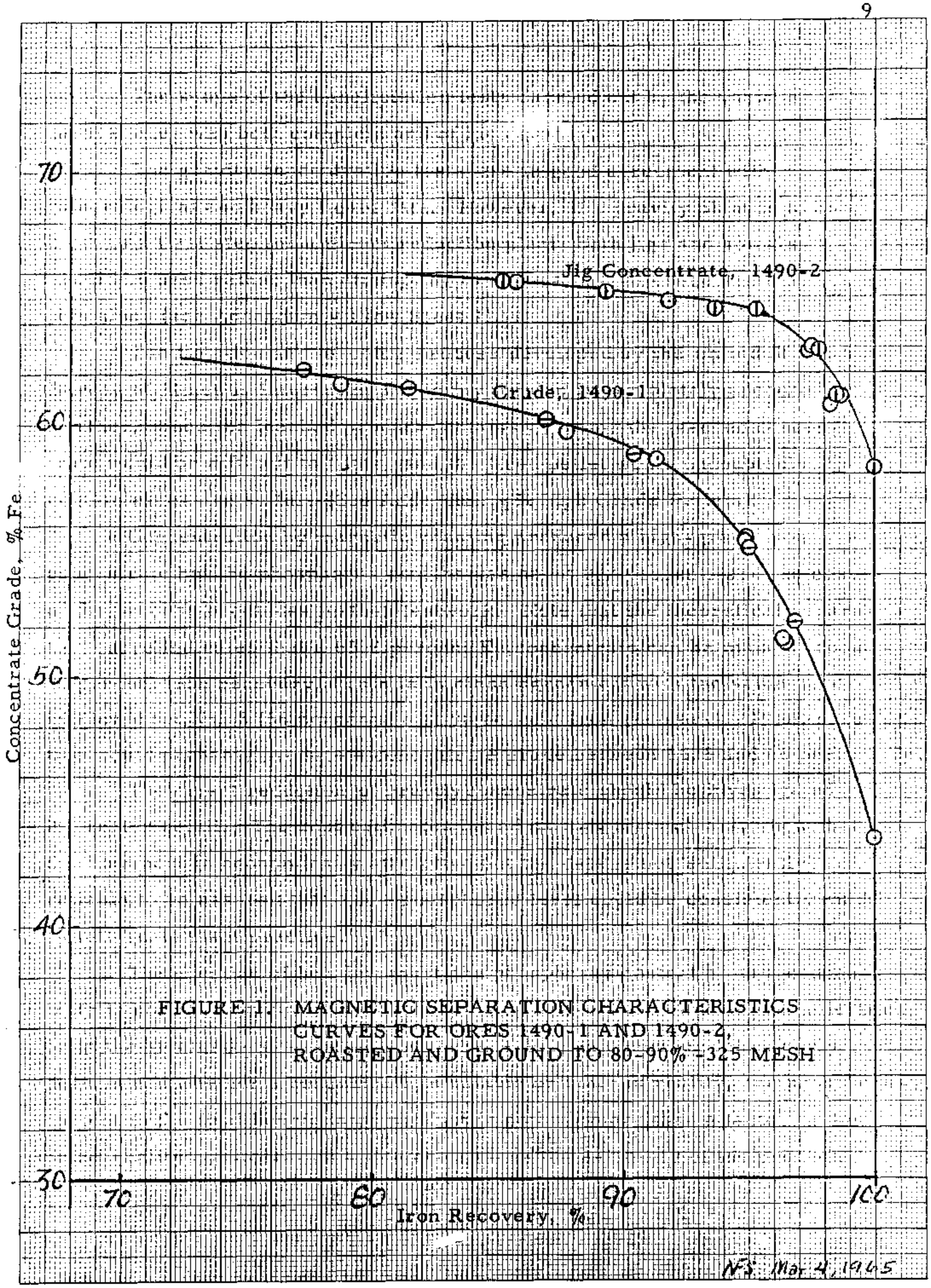


FIGURE 1. MAGNETIC SEPARATION CHARACTERISTICS CURVES FOR ORES 1490-1 AND 1490-2, ROASTED AND GROUND TO 80-90% -325 MESH

The scope of the beneficiation tests carried out by the various laboratories was both complete and of high quality.

The gravity concentration tests are in good agreement with mineralogical and heavy liquid data. However, if a gravity plant is to be built as a first step for a direct reduction plant, both jigs and spirals should be used rather than attempting to make a partial concentrate by jigs alone (see Addendum - Flow Sheet 1).

The flotation and leaching process developed by the United States Bureau of Mines is interesting but in my opinion is completely uneconomic. Further, since the leached product contains gypsum, the leached concentrate would be difficult to filter, and the fired pellet would undoubtedly contain excessive sulfur. (See Addendum - B. High-Grade Pellets).

Concentration by magnetic roasting produced a high-grade iron concentrate but the phosphorus content was about 0.2 percent, and the cost of the process precludes further testwork. Note that the theoretical separation curve indicates that a 65.2 percent Fe concentrate can be made at about 90 percent iron recovery. This is in close agreement with the value used in Flowsheet No. 3 (66 percent Fe at 90 percent iron recovery). Plant practice would probably produce a concentrate grade between 64.5 and 66 percent Fe.

The direct reduction of gravity concentrate should be investigated in greater detail. The cost estimates based on R-N tests clearly show that the flowsheet is economically attractive but the data is not sufficiently reliable to be assured that a low phosphorus level could be consistently made by full-scale plant practice.

The question of how low must the phosphorus content be in a concentrate is not really defined. Phosphorus may be removed either by beneficiation or in the steelmaking process, and there is no magic number that clearly determines whether a concentrate is salable or not. In reality, the phosphorus content in the final beneficiated product is nothing more than a factor in the price-quality consideration which determines the salability of the product.

ADDENDUM

BENEFICIATION FLOWSHEETS BASED ON METALLURGICAL TESTS

One way of evaluating the Snake River Deposit — at least as far as its economic potential is concerned — is to estimate the capital costs, operating costs, and sales value of possible products using flowsheets derived from the metallurgical test data. These cost estimates are not accurate engineering estimates because none of the flowsheets shown have been tested at large-scale-pilot-plant level. The estimates are based on known costs of metallurgical plants beneficiating other ores; however, I believe that these estimates are sufficiently accurate to show the relative merits of producing,

(a) Sinter Feed,

(b) High-Grade Pellets produced by gravity concentration followed by flotation and leaching,

(c) High-Grade Pellets produced from gravity concentration followed by magnetic roasting, and

(d) Gravity concentration followed by direct reduction,, post beneficiation and agglomeration.

The cost estimates for corporate overhead, mining, transportation, housing, developing the mine site, erecting the oxygen plant, cost of stockpiles, port loading, working capital, and process development are the values used by Crest Exploration Engineers. Thus, my capital estimates are shown in two parts:

(a) Process capital estimate, which is my estimate of the process equipment, and

(b) Cost of supporting facilities which includes all other capital and is the value given by Crest Engineers.

All estimates are based on producing five million tons of concentrates per year, with the exception of the direct reduction process which is based on five million tons of gravity concentrates to the direct reduction plant.

POSSIBLE PRODUCTS

A. Sinter Feed (Flow Sheet 1)

The Snake River Deposit lends itself to gravity concentration after comminution through about 4 mesh. The concentrate produced should contain about 60 percent Fe, 9.5 percent SiO_2, and 0.2 percent P. This product would be of questionable market value (even when mixed with high-grade sinter feed) because of its high phosphorus content. However, the market value of an iron ore depends entirely on the price/quality ratio and even a 0.2 phosphorus concentrate will have a market if the selling price is low enough. In fact, a considerable tonnage of high phosphorus ore is currently being sold to the Japanese. This is illustrated in the following table of the chemical analysis of ores imported to Japan.

The following analysis shows that a gravity plant could deliver a concentrate to Japan for approximately $9.89 per ton. If the capital and operating cost estimates are correct, this would be a break-even proposition.

Chemical Composition of Imported Iron Ores

表 1・12 輸入鉄鉱石品位 (%)

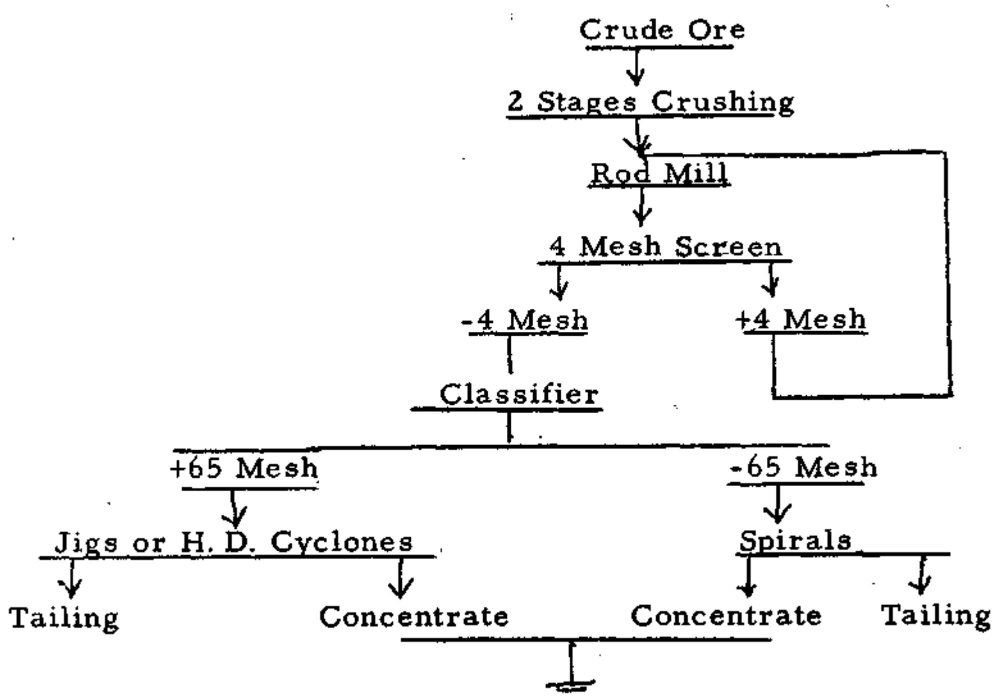
	銘柄	C.W	Fe	Mn	SiO_2	S	P	Cu	Al_2O_3	CaO	MgO	FeO	Fe_2O_3	
朝鮮 Korea	忠州	0.41	55.97	0.05	19.57	0.084	0.015	0.005	0.60	0.27	0.19	7.36	70.44	
	蔚珍	0.79	58.43	0.34	9.18	0.059	0.017	0.012	1.01	6.50	0.25	18.85	62.50	
	金谷	0.28	49.77	0.04	19.01	0.008	0.010	0.006	0.44	0.12	0.15	4.84	64.89	
	殷協	0.72	56.74	0.19	10.15	0.040	0.132	0.005	1.17	3.55	3.47	23.93	54.47	
	茂山(粉)	0.62	57.55	0.03	17.00	0.005	0.054	0.007	0.62	0.68	0.59	23.54	56.12	
	大莊	0.55	54.55	0.24	12.66	0.058	0.023	0.277	0.43	8.21	0.37	18.14	57.83	
	御来	0.46	54.93	0.03	21.07	0.017	0.011	0.005	0.25	0.05	0.08	8.68	68.84	
	南部	1.35	37.60	0.09	42.63	0.007	0.024	0.014	1.23	0.03	0.14	1.58	52.47	
	厚美	0.78	58.65	0.43	7.49	0.045	0.019	0.012	2.15	3.46	3.14	23.16	58.12	As0.20
	開慶	1.30	46.75	0.01	29.69	0.003	0.026	0.004	1.73	0.03	0.09	0.51	66.35	
中国 China	香港(粉)	0.81	56.29	0.49	8.99	0.063	0.011	0.006	0.96	2.52	6.08	23.29	54.30	Sn0.06
	田独	2.39	63.28	0.55	4.75	0.019	0.023	0.010	1.97	0.23	0.11	6.78	82.84	
	石碌	1.86	57.86	0.22	11.83	0.04	0.03	0.006	2.84	0.24	0.10	2.21		TiO_2 0.13
	大冶	4.18	57.72	0.14	8.89	0.274	0.199	0.273	2.38	0.62	0.60	4.64	77.44	
マラヤ Malaya	テマンガン	9.88	57.50	2.41	2.70	0.010	0.083	0.018	1.01	0.03	0.10	0.13	82.08	
	ズングン(水洗)	5.34	61.86	0.07	3.13	0.034	0.031	0.018	3.44	0.03	0.07	3.45	84.66	Sn0.02
	〃 〃 (粉)	4.65	55.70	0.08	10.52	0.044	0.029	0.021	4.43	0.07	0.08	1.62	78.02	Sn0.03
	エンダオ	8.79	50.57	1.47	5.58	0.094	0.055	0.033	8.05					
	スリメダン	1.90	61.50	0.11	6.36	0.016	0.176	0.007	2.45	0.09	0.23	0.79	86.73	TiO_2 0.33
	ケダー	3.71	54.66	0.22	12.79	0.041	0.031	0.005	5.44	0.03	0.11	6.05	69.86	TiO_2 0.30
	イポー	4.66	58.30	1.89	1.55	0.008	0.050	0.007	6.41	0.08	0.14	0.19	83.77	
	ケママン	9.26	45.30	6.02	6.75	0.069	0.109	0.036	5.95	0.18	0.15		64.76	
比島 Philippine Islands	ララップ	1.92	56.69	0.11	9.55	0.775	0.102	0.052	2.72	2.06	1.57	20.42	57.95	
	〃 〃 (粉)	2.20	55.96	0.10	9.81	4.375	0.146	0.135	3.07	0.91	1.21	21.35	51.75	
	サマール	3.75	49.86	0.33	17.95	0.190	0.065	0.016	4.95	0.80	0.67	9.74	60.35	
	マリンズケ	1.32	60.25	0.13	7.73	0.585	0.026	0.088	1.28	3.25	0.49	20.24	63.27	
	パラカレ	2.03	64.08	0.07	4.99	0.011	0.036	0.014	1.17	0.18	0.17	6.56	84.33	
	マティー	2.62	58.81	0.22	7.47	0.932	0.052	0.150	2.35	1.83	0.83	10.74	70.61	
	スリガオ	13.79	47.76	0.96	1.08	0.21	0.027		8.71			Cr_2O_3 4.09		Ni0.75
印度 India	カランパダ	2.43	65.18	0.39	1.99	0.004	0.040	0.006	1.86	0.04	0.10	2.99	89.97	
	ジャガヤペット	1.66	64.45	0.06	3.56	0.005	0.054	0.005	2.14	0.10	0.07	0.66	91.41	
	オリッサ	0.92	67.44	0.11	1.09	0.006	0.024	0.006	1.23	0.02	0.06	1.04	95.24	
	ナユデュペタ	3.60	63.25	0.11	4.41	0.015	0.120	0.007	0.92	0.07	0.07	2.05	88.15	
	ベラリー	1.36	65.66	0.08	2.70	0.006	0.049	0.096	1.62	0.04	0.05	0.79	93.00	
	ロハーラ	1.30	64.73	0.06	4.03	0.004	0.057	0.006	1.71	0.04	0.07	0.76	91.73	
	ラジャスタン	0.84	63.80	0.05	7.66	0.009	0.062	0.006	0.50	0.08	0.09	6.67	83.82	
	トムカ	1.45	66.55	0.13	1.73	0.006	0.042	0.004	1.34	0.03	0.07	1.54	93.44	
	レデイ	1.63	61.25	0.08	1.63	0.019	0.057	0.007	3.64	0.03	0.06	0.90	86.57	
	ゴア	3.55	61.90	0.94	2.81	0.019	0.034	0.005	3.75	0.03	0.05	2.87	85.54	
米国 USA	ユタ	2.75	54.08	0.06	11.11	0.078	0.236	0.015	2.84	1.53	2.03	4.47	72.58	
	ネバタ	2.13	58.27	0.10	6.25	0.257	0.076	0.025	0.79	1.35	5.88	20.24	60.78	
	ハイザー	1.42	58.71	0.09	7.57	0.012	0.216	0.007	1.82	2.47	1.73	16.97	65.26	TiO_2 0.17
	イーグルマウンテン	2.99	54.59	0.05	9.79	0.299	0.082	0.047	1.46	1.28	4.94	8.91	69.87	
カナダ Canada	テキサダ(粉)	0.82	63.71	0.12	3.44	2.179	0.015	0.096	0.86	2.20	1.00	27.19	58.27	
	ハインターバー(粉)	0.56	56.34	0.26	9.16	0.102	0.033	0.009	1.51	8.01	0.41	22.72	55.30	TiO_2 0.13
	オツチノー	0.75	58.02	0.12	8.40	0.079	0.020	0.017	2.32	4.73	1.63	24.06	56.35	
南米 South America	ブラジル	0.46	68.72	0.03	0.51	0.004	0.023	0.004	0.68	0.03	0.05	0.64	97.49	
	ベネズエラ	4.97	63.45	0.02	0.78	0.024	0.112	0.006	1.50	0.03	0.04	0.83	90.51	
	ペルー	1.90	64.45	0.03	3.91	0.104	0.040	0.046	0.47	0.33	0.71	3.04	88.46	
	チリー	1.63	65.90	0.09	2.48	0.048	0.013		0.59	0.48	0.34	5.79	87.74	

FLOW SHEET 1 — GRAVITY CONCENTRATION

Basis: Feed - 41.0% Fe, 0.3% P
Average grade assuming all waste included. Zones 4, 5, 7.

It is assumed that comparable grades and recoveries can be made by using either jigging or heavy density cyclones. In either case the -65 mesh should be treated with spirals.

Flow Sheet



MATERIAL BALANCE — FLOW SHEET 1

MATERIAL BALANCE – Basis: 100 Ton Feed
STEP 1 – GRAVITY CONCENTRATION

		Analysis		Units		Distribution	
Product	Tons	% Fe	% P	Fe	P	% Fe	% P
Feed	100.00	41.0	0.300	41.000	0.300	100.00	100.00
Gravity Conc.	45.71	60.0	0.186	27.426	0.085	66.89	28.34
Total Tails	54.29	25.0	0.396	13.574	0.215	33.11	71.66
Total	100.00			41.000	0.300	100.00	100.00

45.71 Tons

COST ESTIMATE — FLOW SHEET 1

Taxes and Royalties are not shown
Ratio of Concentration = 2.2:1

Item	Crude	Concentrate
*Mining and Transportation	$ .40	$ .88
Crushing and Rod Milling	.22	.48
Gravity Concentration	.08	.18
Drying		.40
*Rail Freight		3.60
*Port Loading		.13
*Ocean Freight (with back haul)		2.43
Amortization of Capital		1.09
Corporate Overhead		.70
Cost at Japan		$9.89
**Product Value at Japan		$10.50

Capital Estimate

Beneficiation Plant and Tailings Disposal	$\$27.5 \times 10^6$
*Supporting Facilities	81.5×10^6
Total	$\$109.0 \times 10^6$

Amortization $\dfrac{\$109.0 \times 10^6}{20 \text{ years} \times 5 \times 10^6 \text{ tons}} = \$1.09$ per ton

*Estimate made by Crest Engineers.

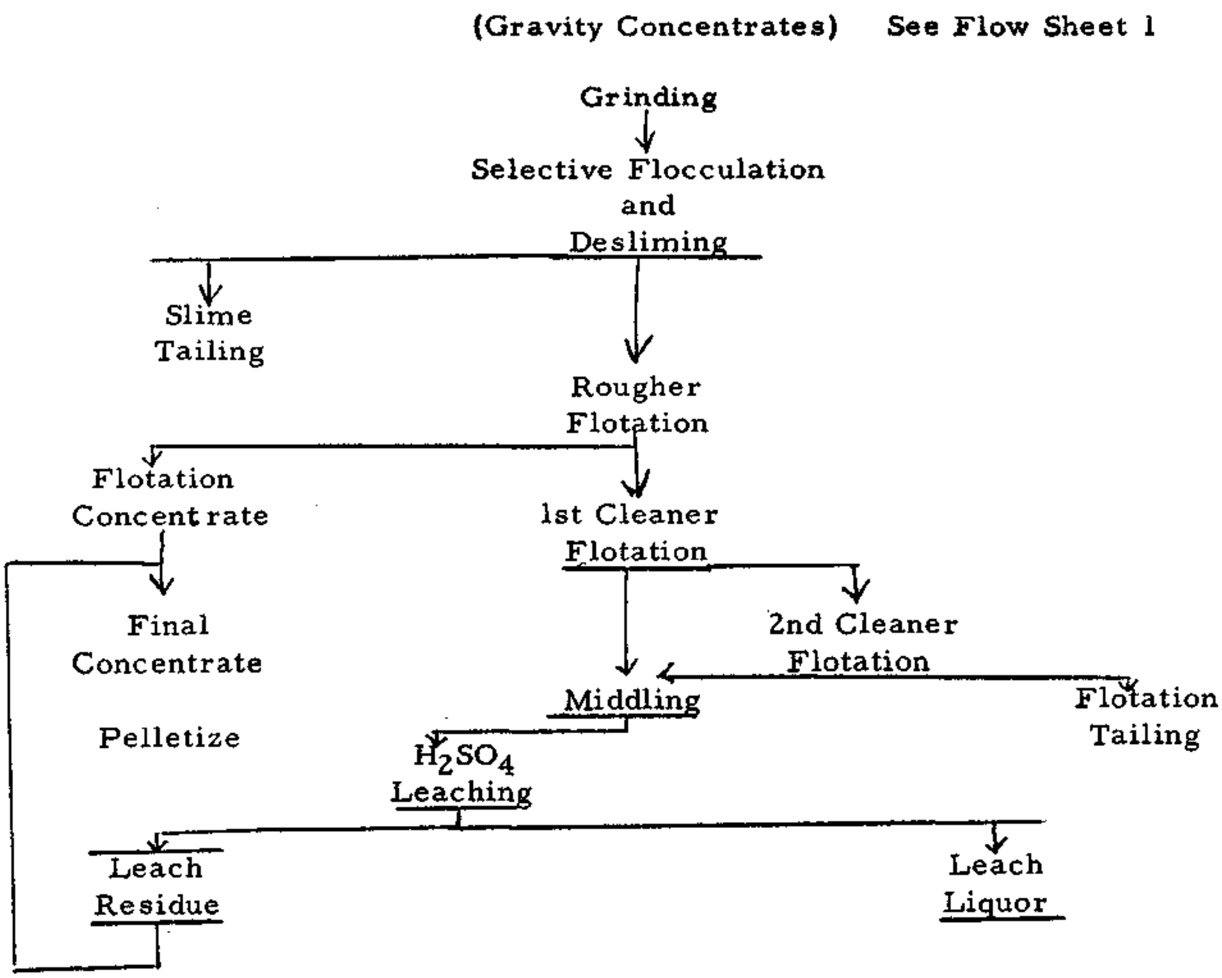
**Based on Fe content but with high P the product has a reduced value.

B. High Grade Pellets

The work done by the United States Bureau of Mines on jig concentrates shows that a 63.60 percent Fe, 6.3 percent SiO_2, and 0.07 percent P concentrate can be produced by froth flotation and acid leaching of flotation middlings. This circuit is clearly not practical. The following analysis shows that the cost of producing and delivering a pellet to Japan would be about \$15.13 and that the likely market value would only be \$15.90. Further, the leached concentrate would probably be very difficult to filter, might not ball easily due to the contained gypsum, and there is no assurance that the contained sulfur could be adequately lowered by washing to meet sulfur requirements. Calcium sulfate is not removed during the agglomeration step. It is rather unfortunate that filtering and agglomeration tests were not carried out by the United States Bureau of Mines. However, the process does not appear economic even when assuming all favorable conditions.

FLOW SHEET 2

COMBINATION GRAVITY, FLOTATION, AND LEACHING



MATERIAL BALANCE — FLOW SHEET 2

GRAVITY CONCENTRATION FOLLOWED BY FLOTATION AND LEACHING

Basis: 100 Ton Feed

STEP 1 — GRAVITY CONCENTRATION

Product	Tons	Analysis % Fe	Analysis % P	Units Fe	Units P	Distribution % Fe	Distribution % P
Feed	100.00	41.0	0.300	41.000	0.300	100.00	100.00
Gravity Conc.	45.71	60.0	0.186	27.426	0.085	66.89	28.43
Total Tails	54.29	25.0	0.396	13.574	0.215	33.11	71.66
Total	100.00			41.000	0.300	100.00	100.00

45.71 Tons

STEP 2 — FLOTATION AND LEACHING

Product	Tons	Analysis % Fe	Analysis % P	Units Fe	Units P	Distribution % Fe	Distribution % P
Float Conc.	24.00	64.8	0.098	15.55	0.024	37.9	8.0
Leached Middling	13.58	61.5	0.022	8.35	0.003	20.4	1.0
Float Tails	5.62	44.1	0.420	2.48	0.024	6.1	8.0
Slime Tails	2.33	42.6	0.660	0.99	0.014	2.4	4.7
Leach Solution	0.18	-	10.500	-	0.019	-	6.3
Total	45.71			27.37	0.084	66.8	28.0
Combined Conc.	37.58	63.60	0.07	23.90	0.027	58.3	

37.58 Tons at 63.6% Fe
6.3% SiO_2
0.07% P

To Pellet Plant

COST ESTIMATE — FLOW SHEET 2

Taxes and Royalties are not shown

$$\text{Total ratio of concentration} = \frac{100}{37.58} = 2.66$$

$$\text{Flotation and leaching ratio of concentration} = \frac{45.71}{37.58} = 1.22$$

Item	Crude	Concentrate
*Mining and Transportation	$ .40	$1.06
Crushing and Rod Milling	.22	.59
Gravity Concentration	.08	.21
Grinding - $.60/ton gravity conc.		.73
Reagents:		
H_2SO_4 (80 lb/ton middling)** (24% of feed) (at 1.2 ct/lb)		.28
NaOH (6.5 lb) (6.84 ct/lb) (1.2)		.53
Tapioca (1.5 lb) (5.0 ct/lb) (1.2)		.09
Fa-2 (0.75 lb) (9.0 ct/lb) (1.2)		.08
***825 (1.5 lb) (6 ct/lb) (1.2)		.11
$Na_4P_2O_7$ (0.5 lb) (5 ct/lb) (1.2)		.03
Flotation and Leaching Labor, Power, Maintenance		.15
Agglomeration		1.60
*Rail Freight		3.60
*Port Loading		.13
*Ocean Freight (with back haul)		2.43
Corporate Overhead		.80
Amortization of Capital		2.71
Total		$15.13

Capital Estimate

Beneficiation and Agglomeration Plant, $37/ton year	= $185.0 x 10^6
*Supporting Facilities	= 86.5 x 10^6
Total	$271.5 x 10^6

Amortization of Capital = $2.71

*Estimate made by Crest Engineers.
**There is no reason to expect that the H_2SO_4 contained in the product could be reclaimed. Therefore, use 80 lb rather than the consumption value of 46 lb.
***AP 825 costs 15 ct/lb. Price used is for similar reagents.

C. High-Grade Pellets Produced from Gravity Concentration Followed by Magnetic Roasting

Magnetic roasting tests performed on jig concentrates produced a magnetic concentrate containing about 66 percent Fe, 0.2 percent P, and less than 6 percent SiO_2 (not analyzed). No effort was made to lower the phosphorus content on the final product by froth flotation because of the failure to lower phosphorus by flotation of artificial magnetic concentrates produced from the raw ore. Although it is possible that a flotation step could be added to reduce the phosphorus, it is probably not worth the experimental costs because the overall economics of roasting do not appear favorable.

The following analysis shows that it would cost about $16.29 to deliver a product to Japan that would have a sales value of about $16.50 even without a phosphorus penalty.

FLOW SHEET 3

GRAVITY CONCENTRATION FOLLOWED BY MAGNETIC ROASTING

Ore

↓

4 Mesh Grind

↓

Gravity Concentration

↓ Concentrate ↓ Tails

Concentrate

↓

Dry

↓

Magnetic Roast

↓

Fine Grinding

↓

Magnetic Separation

↓ Concentrate ↓ Tails

Concentrate

↓

Pelletize

MATERIAL BALANCE — FLOW SHEETS 1, 2, and 3

Basis Feed: 41% Fe - 0.3% P
Total Tails: 25% Fe
Gravity Concentrate: 60% Fe
Magnetic Roast: 1% Loss on Ignition - Complete Conversion
Concentrate from Magnetic Separation: 66% Fe
Fe Recovery from Magnetic Separation: 91% Fe

STEP 1 — GRAVITY CONCENTRATION

Basis: 100 Ton Feed

		Analysis		Units		Distribution	
Product	Tons	% Fe	% P	Fe	P	% Fe	% P
Feed	100.00	41.0	0.300	41.000	0.300	100.00	100.00
Gravity Conc.	45.71	60.0	0.186	27.426	0.085	66.89	28.34
Total Tails	54.29	25.0	0.396	13.574	0.215	33.11*	71.66
Total	100.00			41.000	0.300	100.00	100.00

45.71 tons

STEP 2 — MAGNETIC CONCENTRATION

Product	Tons	% Fe	Units Fe	Distrib., % Fe
Gravity Conc.	45.71	60.0	27.426	66.89
Roast to				
Magnetite	44.19	62.06	27.426	
Loss on Ignition	43.75	62.69	27.426	
2 1/2 % Dust Loss	1.09	62.69	0.686	1.66*
Magnetic Separation Feed	42.66	62.69	26.740	65.23
Magnetic Concentrate	36.87	66.00	24.333	59.35*
Magnetic Tails	5.79	41.57	2.407	5.87*
Total				100.00*

COST ESTIMATE — FLOW SHEET 3

Taxes and Royalties are not shown

$$\text{Total ratio of concentration} = \frac{100}{36.87} = 2.71$$

$$\text{Magnetic ratio of concentration} = \frac{45.71}{36.87} = 1.24$$

Item	Crude	Concentrate
*Mining and Transportation	$ .40	$1.08
Crushing and Rod Milling	.22	.60
Gravity Concentration	.08	.22
Drying (1.24 x .40)		.50
Grinding (1.24 x .60)		.74
**Roasting and Concentration (1.24 x .90)		1.17
Agglomeration		1.60
*Rail Freight		3.60
*Port Loading		.13
*Ocean Freight (with back haul)		2.43
Corporate Overhead		1.36
Amortization of Capital		2.86
Total		$16.29

Capital Estimate

Beneficiation Plant, $40/ton year	=	$\$200.0 \times 10^6$
*Supporting Facilities	=	86.5×10^6
Total		$\$286.5 \times 10^6$

*Estimate made by Crest Engineers.

**This item could be reduced if cheaper fuel were at the plant site. Based on 17 ct/million Btu.

D. Gravity Concentration followed by Direct Reduction, Post Beneficiation and Agglomeration

The following cost estimate was made on the basis of a pilot plant test to produce a jig concentrate and one test by R-N Corporation. The reduction data used is the information furnished in Mr. Breitenstein's letter of March 23, 1964. Since we have no plant information on the actual cost of grinding, magnetic concentration, and agglomeration of the reduced product, my cost estimate is, at best, an educated guess and will have to be modified after further testwork is carried out. However, in all probability, my cost estimate is a little high for both capital and operating costs. Yet, the process still appears economically attractive and certainly warrants further testing.

FLOW SHEET 4

GRAVITY CONCENTRATION FOLLOWED BY DIRECT REDUCTION, POST BENEFICIATION AND AGGLOMERATION

Ore

↓

Gravity Concentration

↓

Drying

↓

Direct Reduction and Concentration

↓

Agglomeration

ORDER OF MAGNITUDE OF COST ESTIMATE FOR FLOW SHEET 4

Taxes and Royalties are not shown
Basis: 5x106 Ton/Year'Gravity Conc. and $3x10^6$ Ton/Year Reduced Product

Item	Cost per Ton Agglomerate
*Mining and Transportation	$1.47
Crushing and Rod Milling	.80
Gravity Concentration	.30
Drying	.67
Amortization of Capital for Producing Gravity Concentrates	1.82
Amortization of Capital for Direct Reduction Plant	2.58
Production Cost of R-N Process (based on estimates by R-N)	11.31
*Port Loading	.13
*Rail Freight	4.77
*Ocean Freight	2.26
*Corporate Overhead	1.36
Total	$27.47

The sales value of product would be about $40 per ton.

Order of magnitude cost estimate to produce 5×10^6 tons of gravity concentrates at 60% Fe = 3×10^6 tons of agglomerate at 94% Fe (R-N Test, March 23, 1964)

Capital Estimate for Reduction Plant

Direct Reduction and Agglomeration Plant	$35/ton year =	$\$105 \times 10^6$
Post Beneficiation	$15/ton year =	$,45 \times 10^6$
*Additional Supporting Facilities		5×10^6
Total		$\$155 \times 10^6$

Amortization of Capital for Direct Reduction, Post Beneficiation and Agglomeration:

$$\frac{\$155 \times 10^6}{20 \text{ years} \times 3 \times 10^6 \text{ tons}} = \$2.58$$

(Continued on next page)

ORDER OF MAGNITUDE OF COST ESTIMATE FOR FLOW SHEET 4 (Continued)

Total Capital

Beneficiation Plant and Tailings Disposal	$ 27.5 x 10^6
Direct Reduction Plant	105.0 x 10^6
Post Beneficiation Plant	45.0 x 10^6
*Supporting Facilities	86.5 x 10^6
	$264.0 x 10^6

Return on Investment:

$$\frac{(\$40 - 27.47) \times 3 \times 10^6 \text{ tons} \times 100}{\$264 \times 10^6} = 14.24\% \text{ before taxes.}$$

*Estimate made by Crest Engineers.

California Standard Company
Research Contract No. S64003

MAGNETIC ROASTING AND SEPARATION TESTS ON CREST ORE (No. 1490)

Materials

Ore 1490-1 - Crude ore, 43.4% Fe, 0.32% P
Ore 1490-2 - Jig concentrate, 58.2% Fe, 0.17% P

Treatment

Ore samples were pulverized to minus 65 mesh, agglomerated with water, and roasted in a fixed bed to completion, (Fe++/Fe) = 0.33, at 650°C in a gas mixture composed of 10% H_2, 20% CO_2, 70% N_2, and saturated with water vapor at 60°C.

The roasted products were pulverized as indicated by the percent minus 325 mesh.

Test Data

Davis tube tests were run to establish the magnetic separation characteristic curves shown in Figure 1. The test data are given in Table 1. This method of evaluating a magnetite ore is described by Schulz, AIME Trans. SME, vol. 229, pp. 211-216, June 1964.

Discussion

Grinding the roasted products more finely than 80 percent minus 325 mesh did not significantly improve the magnetic separation characteristics.

The maximum ultimate separation efficiency occurred at about 60% Fe and 87% iron recovery for the roasted crude ore and at about 64% Fe and 96% iron recovery for the roasted jig concentrate.

For both ore samples, only about one-third of the phosphorus content was rejected, and that very early in the magnetic concentration process, indicating that about two-thirds of the phosphorus in this material is closely associated with the iron oxide minerals.

N F Schulz

N. F. Schulz
Research Associate

March 4, 1965

TABLE 1. DAVIS TUBE TEST DATA

Test No.	% Wt Rec	Conc, % Fe	% Fe Rec	% P
	Crude, 1490-1, 44% -325 Mesh			
6977	83.49	50.38	96.96	
6978	76.09	53.59	94.00	
6979	65.53	57.18	86.38	
7001	62.04	58.64	83.88	
7002	56.81	59.52	77.96	0.23
	Crude, 1490-1, 83% -325 Mesh			0.32
6980	80.60	52.14	96.88	
6981	74.99	55.03	95.12	
6982	66.76	58.79	90.48	
7003	62.73	60.16	87.01	
7004	57.55	61.44	81.52	
7005	53.96	62.15	77.33	0.21
	Crude, 1490-1, 89% -325 Mesh			
6983	81.77	51.20	96.53	
7006	81.16	51.53	96.43	
6984	74.51	55.53	95.40	
7007	74.36	55.36	94.92	
6985	67.70	58.56	91.41	
7008	63.80	59.68	87.79	
7009	55.47	61.60	78.79	0.20

TABLE 1. DAVIS TUBE TEST DATA (Cont'd)

Test No.	% Wt Rec	Conc, % Fe	% Re Rec	% P
		Jig Concentrate, 51% -325 Mesh		
6992	94.06	61.45	99.24	
6993	91.16	62.80	98.29	
6994	84.32	64.03	92.70	
7010	84.73	64.16	93.34	
7011	78.59	64.48	87.01	0.132
		Jig Concentrate, 80% -325 Mesh		0.17
6995	93.85	61.12	98.49	
6996	90.54	62.89	97.77	
6997	86.09	64.48	95.31	0.12
7012	84.59	64.49	93.67	
7013	79.80	65.20	89.34	
7014	75.66	65.63	85.25	0.124
		Jig Concentrate, 90% -325 Mesh		
7015	94.23	60.72	98.24	
6998	94.08	61.12	98.73	
7016	90.20	62.88	97.38	
6999	90.21	63.04	97.64	
7000	85.55	64.48	94.72	0.119
7017	82.54	64.81	91.85	
7018	76.18	65.60	85.80	0.124

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