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OTTAWA

MINES BRANCH INVESTIGATION REPORT IR 63-93

PILOT PLANT CONCENTRATION OF MAGNETITE-ILMENITE FROM ROMAINE RIVER VALLEY, P. Q., FOR QUEBEC IRON AND TITANIUM CORPORATION

by

P. D. R. MALTBY & W. S. JENKINS

MINERAL PROCESSING DIVISION

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PILOT PLANT CONCENTRATION OF MAGNETITE-ILMENITE FROM ROMAINE RIVER VALLEY, P.Q. FOR QUEBEC IRON

by

P. D. R. Maltby* and W. S. Jenkins**

SUMMARY OF RESULTS

The investigation demonstrated that production of high grade magnetite and ilmenite concentrates meeting the specifications of Quebec Iron and Titanium Corporation was feasible by conventional magnetic concentration of the magnetite and an ilmenite concentration process employing gravity and high intensity magnetic concentration followed by apatite flotation to reduce the phosphorus content of the final ilmenite concentrate to less than 0.05% P_2O_5.

From a shipment of 53 tons, approximately 5 tons each of magnetite and ilmenite concentrates were produced and shipped to Sorel, P.Q. The magnetite concentrate contained 10.7% of the original feed weight and assayed 69.26% Fe, 0.43% TiO_2, and 0.011% P_2O_5. A screen test showed it to be 63.8% minus 325 mesh. The ilmenite concentrate contained 11.0% of the original feed weight and assayed 38.9% TiO_2. Due to production difficulties, the phosphorus content of the shipment exceeded 0.09% P_2O_5, but it is believed this could have been reduced by flotation if it had been detected before shipment.

Apatite flotation concentrates assaying 36.7% P_2O_5 were produced from the tailing. A 389 lb shipment of recleaned apatite concentrate was sent to Sorel.

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INTRODUCTION

Purpose of the Investigation

The purpose of the investigation was to prove, in a pilot plant, the feasibility of a process to recover a magnetite concentrate and an ilmenite concentrate to meet required specifications. Work was also done to recover an apatite concentrate from the ilmenite tailing products.

The company required 5 tons of ilmenite concentrate containing at least 36% TiO_2 with less than 0.05% P_2O_5 for smelting tests at Sorel, P.Q.

Shipment

A shipment of 53 tons was received at the Mines Branch on April 22, 1963, labelled 'Positive Anomaly Ore'. Previous shipments had been received earlier from the same area for small scale tests from Mr. J. M. Noy, Research Director, Quebec Iron and Titanium Corporation, Sorel, Quebec. The 'ore' was made up mostly of rounded lumps 6 in. or more in diameter.

Location of the Property

The shipments originated from a property in the Romaine River Valley, Saguenay County, Quebec.

Previous Testwork

Much testwork had been done on this material and the investigation was described in Mines Branch Investigation Report IR 62-86, by W. S. Jenkins. Previous testwork had also shown that a magnetite concentrate could be produced to meet specifications, and that apatite recovery from the ilmenite tailing products was possible.

It was on the basis of the results obtained in these investigations that the pilot plant investigation was done.

Sample Analysis

All chemical analyses in this investigation were made by the Analytical Chemistry Sub-Division, Mineral Sciences Division, Mines Branch, Ottawa.

TEST PROCEDURE AND RESULTS

Preliminary Treatment - Mill Run 1

From the results of the laboratory investigation, the magnetite was best separated from the ilmenite at 35m, using a wet magnetic cobbing separator. In the pilot plant test, the crude ore was first crushed to ½ in. and ground to 30m in a rod mill at a rate of about 2000 lb/hr. The rod mill was operated in closed circuit with a 30m vibrating screen. The screen undersize product was then pumped to a single drum, low intensity, wet magnetic separator. The magnetic concentrate from this cobbing drum was ground to 150m in a ball mill in closed circuit with a vibrating screen, and the ground product was cleaned using a low intensity, counter-current-wash, 3-drum, wet magnetic separator. The tailing was discarded to waste and the concentrate was sampled and stored in drums.

The magnetic cobber tailing was treated by gravity separation for ilmenite recovery. The cobber tailing was first pumped to a cone for de-watering, and then fed at 25% solids to a Humphreys spiral where a rougher concentrate was made. The spiral was adjusted so that a middling product was returned to the feed tank. Screen tests were done on some of the products with the results shown in Table 2. Some ilmenite lost in the spiral tailings was collected by passing the tailings over a Wilfley table and then re-tabling the table middling product on a second table. All three gravity concentrates were combined to form a rougher ilmenite concentrate.

The results of the first two days run using 2000 lb of rods and a feed rate of 2000 lb/hr are shown in Table 1.

TABLE 1

Results of Mill Run 1 (Preliminary)

Product	Weight		Analysis % *			Distn %
	lb/hr	%	Fe	TiO_2	P_2O_5	TiO_2
R.M. feed	2041	100.0	14.84	6.74		100.0
Cobber conc	276	13.5	61.31	1.95		3.7
Regrind mag conc	237	11.6	69.10	0.49	0.026	1.0
" " tail	39	1.9	14.99	9.69		2.7
Spiral conc	406	19.9		23.61	2.98	67.3
" tail	1359	66.6		3.05		29.0
No.1 table conc	19	0.9		35.03	1.67	4.6
" " tail	1160	56.9		1.65		13.4
No.2 " conc	17	0.8		33.81	2.15	3.9
" " tail	163	8.0		6.29		7.1

*From Internal Report MS-AC-63-416.

TABLE 2

Results of Screen Tests on Mill Run 1

Size	Rod Mill		Cobber Concentrate	Spiral Concentrate	Table Concentrate	
	Feed	Discharge			No. 1	No. 2
+14m	53.3	-	-	-	-	-
-14+20m	5.9	-	-	-	-	-
-20+28m	6.0	1.4	0.2	-	-	-
-28+35m	6.9	2.8	2.5	1.0	0.4	2.0
-35+48m	6.0	8.6	7.4	4.0	5.5	9.0
-48+65m	5.1	14.6	14.3	10.9	16.2	20.2
-65+100m	4.5	18.3	17.6	19.9	25.0	17.5
-100+150m	4.0	14.3	14.3	21.0	21.2	11.5
-150+200m	2.7	10.7	11.3	18.0	11.9	12.2
-200	5.6	29.3	32.4	25.2	19.8	27.6
Total	100.0	100.0	100.0	100.0	100.0	100.0

Mill Run 2

During the next three days of the run, the weight of rods in the rod mill was reduced to as low as 1250 lb. This was done to try to obtain as coarse a grind through 30m as possible, and to reduce the amount of fine ilmenite that would otherwise be lost in the gravity tailing. With 1250 lb of rods, the circulating load was too high for the bucket elevator handling the rod mill discharge. Best results were obtained using 1500 lb of rods. The feed rate was kept at 2000 lb/hr. Various adjustments were made to the spiral with increased wash water to obtain the best grade and recovery of ilmenite. The average results obtained over the three days' run are shown in Table 3.

TABLE 3

Results of Mill Run 2

Product	Weight		Analysis % ✱			Distn %	
	lb/hr	%	Fe	TiO_2	P_2O_5	TiO_2	P_2O_5
R.M. feed	2060	100.0	15.13	6.95	3.76	100.0	100.0
Cobber conc	268	13.0	59.88	2.47	0.36	4.6	1.4
Regrind mag conc	220	10.7	69.26	0.43	0.011	0.7	-
" " tail	48	2.3	15.37	12.13	2.01	3.9	1.4
Cone overflow	127	6.2		5.66	5.96	5.0	10.1
Spiral conc	544	26.4		19.18	4.54	73.2	32.9
" tail	1121	54.4		2.19	3.74	17.2	55.6
No.1 table conc	7	0.3		34.1		1.4	
" " tail	941	45.7		1.33	2.91	9.0	
No.2 " conc	4	0.2		25.32		0.7	
" " tail	169	8.2		5.11	6.65	6.1	
Combined R conc	555	26.9		19.37	4.52	75.3	33.2

✱ From Internal Report MS-AC-63-425

The rod mill circulating load was 240 lb/hr. The results shown in Table 3 were considered to be as good as could be obtained under the circumstances, and the remainder of the ore was treated during the next 6 days. A screen test on the final magnetite concentrate showed it to be 63.8% minus 325m.

The best spiral operating conditions were with a feed density of 25% solids, and a middling return load of about 28% of the feed weight. Only 3 concentrate ports were used - the top, the middle and the second from the bottom. The middling product was collected off all the middling ports at varying openings. Wash water rate was kept as high as possible.

Concentrate Cleaning

The combined rougher concentrate from Mill Run 2 was cleaned twice by two stages of spirals, followed in each case by table cleanup of the spiral tailings. After the second cleaning step, it was apparent that any increased grade obtainable by gravity separation would be accompanied by a high tailing loss. This appeared to be due to the presence of an iron silicate gangue (pyroxene) which had a relatively high specific gravity of 3.7 compared to 4.9 for an ilmenite concentrate. As before, the cleaner spiral worked best with a high wash water rate and approximately the same feed conditions and settings. The results of the two cleaning operations are shown in Table 4 from averaged results.

TABLE 4

Results of Final Gravity Ilmenite Concentration

Stage	Product	Weight	Analysis % **		Distn % of orig feed	
		% of orig feed	TiO_2	P_2O_5	TiO_2	P_2O_5
First Cleaner	Spiral conc	17.5	25.1	3.71	64.2	17.8
	" tail	9.4	8.11	6.12	11.1	15.4
	Table conc	1.4	26.85	3.11	5.6	1.1
	" tail	8.0	4.88	6.50	5.5	14.3
	Feed*	26.9	19.37	4.52	75.3	33.2
Second Cleaner	Spiral conc	15.1	28.5	3.05	63.0	12.4
	" tail	3.8	12.04	6.32	6.8	6.5
	Table conc	0.6	31.12	1.94	2.7	0.3
	" tail	3.2	8.21	7.19	4.1	6.2
	Feed*	18.9	25.3	3.66	69.8	18.9

* The feed was made up of the combined spiral and table concentrates from the previous stage. Feed rate was 1500 lb/hr.

** From Internal Reports MS-AC-63-517 and MS-AC-63-523.

Further Ilmenite Concentration

The final spiral and table concentrates were combined and dried. Test results had shown that good results were obtained by dry high intensity magnetic separation using a Wetherill cross-belt separator. Accordingly, the majority of the gravity concentrates were passed over the Wetherill separator.

Another alternative was to use a Jones high intensity wet magnetic separator which would mean that the gravity concentrate would not first need to be dried. Therefore, a 1000 lb sample was retained for testing with a Jones separator after the removal of a small amount of magnetics on the first belt of the Wetherill separator. This was done to avoid plugging the magnet plates later with any magnetite remaining in the gravity concentrates.

The results obtained by dry high intensity magnetic separation were good, and the averaged results are shown in Table 5. A Wetherill separator was used with 3 magnetic cross-belts. Feed rate was 300 lb/hr. Magnetite was removed on the first belt at an intensity of 1.5 amp. An ilmenite concentrate was removed on the second belt at an intensity of 7 amp. A silicate

middling product was removed on the third belt at a setting of 10 amp. The Wetherill tailing contained a high proportion of apatite with some silica.

TABLE 5

Results of Wetherill Separation

Product	Weight	Analysis % *		Distn % of orig feed	
	% of orig feed	TiO_2	P_2O_5	TiO_2	P_2O_5
Magnetics	0.2	25.91	-	0.7	-
Ilmenite conc	11.3	38.93	0.38	64.4	1.1
Middling	2.0	1.86	0.34	0.5	0.2
Tailing	2.2	0.20	19.31	0.1	11.4
Feed	15.7	28.7	3.04	65.7	12.7

* From Internal Report MS-AC-63-525.

A small sample of the Wetherill ilmenite concentrate was repassed at different amperage settings to find out if more apatite could be rejected. No improvement was noted in any of the tests. With a loss of another 1% of the ilmenite in the feed, the cleaner ilmenite concentrate assayed 39.45% TiO_2. Two small scale tests, using a Jones separator, were also run on a sample of the Wetherill concentrate. At 30m there was no improvement in grade. At 150m the Jones concentrate contained 0.061% P_2O_5 with a weight recovery of 74.4%. The middling product contained 0.082% P_2O_5 with a further weight recovery of 11.4%.

A continuous test of three hours was done using a Jones separator. The maximum feed rate obtained was 350 lb/hr, at an intensity of 7 amp and a feed density of 20% solids. The middling product contained considerably more ilmenite than the Wetherill middling product and was, therefore, repassed on the Jones separator, at 7 amp, for additional ilmenite recovery. The results of this test are shown in Tables 6 and 7.

TABLE 6

Upgrading of Ilmenite Gravity Concentrate Using a Jones Separator

Product	Weight	Analysis % **		Distn % of orig feed
	% of orig feed	TiO_2	P_2O_5	TiO_2
7 amp conc	8.3	37.05	0.52	46.5
" midd	5.1	21.92		17.0
" tail	2.1	4.74		1.5
Feed*	15.5	27.72		65.0

*calculated **From Internal Report MS-AC-63-548.

TABLE 7

Retreatment of Middling Product Using a Jones Separator

Product	Weight	Analysis % **		Distn % of orig feed
	% of orig feed	TiO_2	P_2O_5	TiO_2
7 amp conc	2.5	33.0	0.61	13.5
" midd	1.8	11.02		3.4
" tail	0.8	0.79		0.1
Feed*	5.1	20.2		17.0

*calculated **From Internal Report MS-AC-63-548.

The two Jones concentrates were dried and combined with the Wetheril1 concentrate giving a weight of approximately 11,000 lb.

Final Removal of Apatite From Ilmenite Concentrate

It was required to lower the apatite content of the Wetherill concentrate to 0.05% P_2O_5 to meet the final specifications. Much work had been done on small scale tests to try to accomplish this. In some tests the P_2O_5 content had been lowered to 0.05% but the ilmenite loss was high, amounting to about 20% of the ilmenite in the flotation feed. In all these tests the Wetherill concentrate had been ground first to 150m, and apatite was floated, at a pH of 9.5 with soda ash, using oleic acid as a collector. However, the problem remained of obtaining a more selective apatite float.

A small continuous circuit was set up to treat the Wetherill concentrate at a feed rate of 120 lb/hr. The feed was first ground in a ball mill in closed circuit with a 100m screen. The screen undersize was pumped to a conditioner and from there flowed to a bank of cells. The preliminary results were disappointing. Various combinations of reagents were tried until it was decided to use a dextrine, WW 82, to see if it would depress ilmenite and improve the selectivity of the apatite flotation. Results improved immediately and once the optimum rate of 1.0 lb of dextrine/ton had been established, a very clean apatite froth was made with a recovery of practically 100% of the ilmenite in the flotation tailing.

Larger units were then used for a feed rate of 800 lb/hr. The ore was first ground to 100m, conditioned, and floated in 4 Denver No. 7 cells. A 30 in. x 18 in. ball mill was used with 1700 lb of steel balls. Dextrine and soda ash were added in the required amounts to the ball mill feed. Sodium oleate and a frother, a mixture of pine oil and Aerosol OT 100, were added to the conditioner and the third rougher cell. Sodium oleate was made up as a 5% solution by neutralizing oleic acid No. 4 stoichiometrically with a 10% NaOH solution. Reagent additions are shown in Table 9.

The apatite froth was cleaned in 3 Denver No. 5 cells and the middling product was returned to the conditioner at the head end of the circuit. Flotation feed density was kept at 30% solids. In the preliminary small scale test where dextrine was first used, the ilmenite concentrate assayed 0.05% P_2O_5. In the larger scale test it was harder to obtain a good apatite froth and some apatite floated on top of the cone used for storage before filtering. It was found easy to remove this small amount of apatite by allowing the overflow to run to waste. The averaged results of apatite flotation are shown in Table 8.

TABLE 8

Results of Apatite Flotation From Ilmenite Concentrate

Product	Weight %	Weight % of orig feed	Analysis % *** TiO_2	Analysis % *** P_2O_5	Distn % P_2O_5
Apatite Cl froth	1.4	0.2		29.18	86.1
Middling	-	-		2.33	-
Ilmenite conc	98.6	11.1	38.9	0.067	13.9
Cone o'flow	0.3	0.1		0.65	0.2
Ilmenite filter cake**	98.3	11.0	38.9	0.052	13.7
Feed*	100.0	11.3		0.48	100.0

*calculated

** Ilmenite recovery from the crude ore was 63.3%.

*** From Internal Reports MS-AC-63-848; MS-AC-63-863; MS-AC-63-577.

TABLE 9

Reagent Additions for Apatite Flotation

Reagent	Amount lb/ton flotation feed	Cost $ per ton flotation feed
Dextrine WW 82	1.0	0.18
Soda ash	2.0	0.06
Oleic acid (as sodium oleate)	0.35	0.05
Frother (PoA)	0.15	0.01

The results of two screen tests are shown in Table 10.

TABLE 10

Results of Screen Tests

Size	Flotation Feed Weight %	Ilmenite Concentrate Weight %
+100m	0.2	0.6
-100+150m	9.3	10.2
-150+200m	25.6	24.7
-200+325m	23.4	22.4
-325m	41.5	42.1
Total	100.0	100.0

Very poor results were obtained when filtering the ilmenite flotation concentrate as produced. However, the addition of sulphuric acid to reduce the pH to 6.0, followed by the addition of 1 to 2 lb of lime/ton, resulted in excellent filtering and a cake moisture of about 8%.

On the completion of this part of the investigation, about 5 tons each of magnetite and ilmenite concentrates, that met the required specifications, were shipped to the Sorel smelter.

Apatite Recovery From Tailing Produots

The Company desired to investigate the feasibility of recovering a high grade apatite concentrate from a composite sample of the various tailing products. Laboratory tests had shown that, after grinding to 100m, concentrates containing up to 40% P_2O_5 were obtainable using an oleic acid float. A 10 ton composite sample was, therefore, made up of 75% rougher gravity tailing, 22.5% cleaner and re-cleaner gravity tailing, and 2.5% Wetherill tailing. The sample was ground at a rate of about 600 lb/hr in a ball mill to 100m. A rougher concentrate was floated in 4 Denver No. 7 cells, and the froth was cleaned and re-cleaned in 3 Denver No. 5 cells. The middling products were returned to the feed end of each circuit. The reagents used in this test were 2 lb of soda ash/ton for a pH of 9.0 and 0.9 lb of oleic acid No. 4/ton added to the conditioner and the third rougher cell feed. Better results were obtained using oleic acid instead of sodium oleate. The results are shown in Table 11.

TABLE 11

Results of Apatite Flotation at 100m

Product	Weight		Analysis %**	Distn %
	lb/hr	%	P_2O_5	P_2O_5
Re-ol froth	45	8.0	36.80	70.2
Tailing	520	92.0	1.35	29.8
Feed*	565	100.0	4.18	100.0

*calculated ** From Internal Report MS-AC-63-880.

In the re-cleaning stage about 0.5 lb of sodium silicate/ton was added to help to depress silicates. The amount of apatite represented in the composite tailing sample was about 87% of the apatite in the crude feed. Most of the apatite not available for recovery was lost in the cone overflow prior to the rougher spiral treatment stage.

Coarse Apatite Flotation

Since the cost of grinding the tailing products from 30 to 100m would be high compared to the value of recoverable apatite, it was decided to try to recover apatite from the 30m tailing with no prior grinding. Feed was pumped to a conditioner at about 700 lb/hr and a rougher apatite froth was floated. The froth was cleaned and the cleaner froth was ground to 100m and was refloated separately in order to bring it up to grade specifications. The pH was maintained at 8.5 with 5.0 lb of soda ash/ton. Larger amounts of oleic acid were required than before, half in the conditioner and half in the feed to the third rougher cell, for a total of 2.5 lb/ton. A total of 0.8 lb of sodium silicate/ton was added, and 0.025 lb of PoA frother/ton was used. The flotation results are shown in Table 12.

TABLE 12

Results of Apatite Flotation at 30m

Product	Weight		Analysis % **	Distn %
	lb/hr	%	P_2O_5	P_2O_5
Recl froth***	39	5.6	36.72	43.4
Tailing	657	94.4	2.85	56.6
Feed*	696	100.0	4.75	100.0

*calculated ** From Internal Report MS-AC-63-900.

*** ground to minus 100m.

The best grade of apatite concentrate at 30m was 33.36% P_2O_5. The results of screen tests are shown in Table 13.

TABLE 13

Results of Screen Tests on Apatite Float Products

Size	30m Float		100m Float	
	Feed	C1 Froth	Feed	C1 Froth
+48m	30.1	1.0	-	-
-48+65m	18.0	5.1	-	-
-65+100m	16.4	16.7	-	-
-100+150m	11.0	23.0	5.2	1.0
-150+200m	9.1	22.8	19.4	8.5
-200+325m	8.4	19.5	25.4	12.4
-325m	7.0	11.0	50.0	78.1
Total	100.0	100.0	100.0	100.0

The lower apatite recovery with the coarse float appeared to be due to flotation of only the finer apatite particles. From the screen test results on the 30m cleaner froth, it can be seen that very little plus 65m apatite floated.

DISCUSSION OF RESULTS

The results showed that good quality magnetite, ilmenite and apatite concentrates could be produced from the sample submitted. The main problem in producing an ilmenite concentrate economically, containing less than 0.05% P_2O_5, was solved by a final flotation step in which the addition of an ilmenite depressant was critical. It is believed that the methods used for ilmenite recovery are practical and of relatively low cost. The only major decision to be made in the treatment scheme is the choice between wet or dry high intensity magnetic separation. The use of the Jones separator was investigated and the results obtained were not as good as those obtained with dry separation. However, this may be more than offset by the cost of drying the feed before dry magnetic separation. The better dry magnetic separation results appeared to be due to the more successful rejection of siliceous gangue particles containing fine magnetite inclusions.

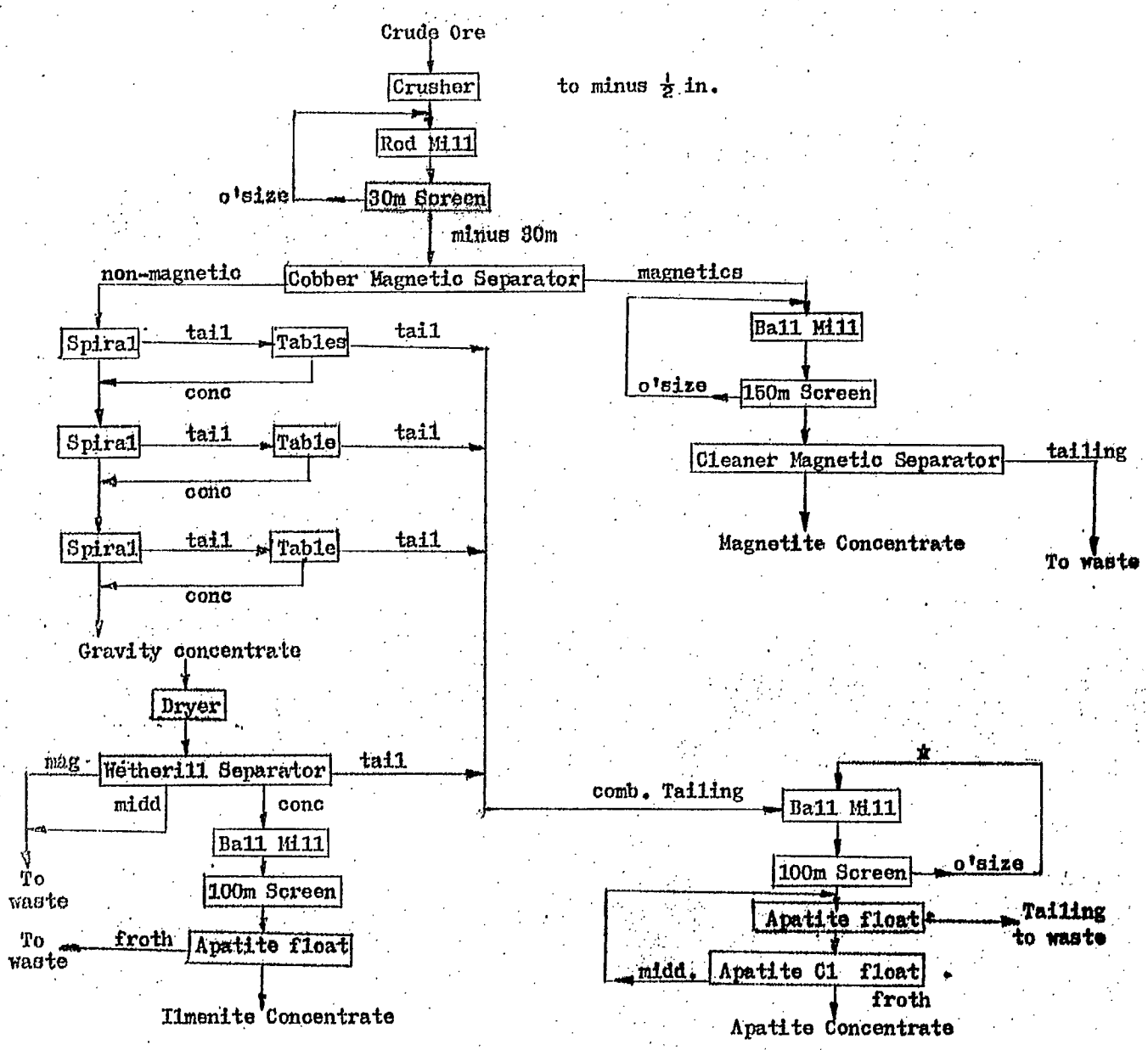
Apatite could be recovered from a composite tailing sample either by grinding first to about 100m, or else by floating at 30m, followed by grinding and cleaning the coarse apatite concentrate. Fine flotation

resulted in a better apatite recovery. However, the value of the apatite concentrate produced would determine the more economical method of recovery. The loss in apatite recovery using the 30m float might be outweighed by the cost of grinding all the tailing sample before apatite flotation at 100m.

The results of a flotation test shown in the Appendix were not very encouraging, due to poor ilmenite depression and inability to float all the silicates in a cationic silica float. Therefore, flotation was abandoned in favour of gravity separation followed by high intensity magnetic separation.

CONCLUSIONS

A suggested treatment scheme, based on the results of the pilot plant investigation, is shown in Figure 2.



* Alternative: Coarse 30m apatite float followed by grinding and cleaning the rougher apatite froth.

Figure 1.- Pilot Plant Flowsheet

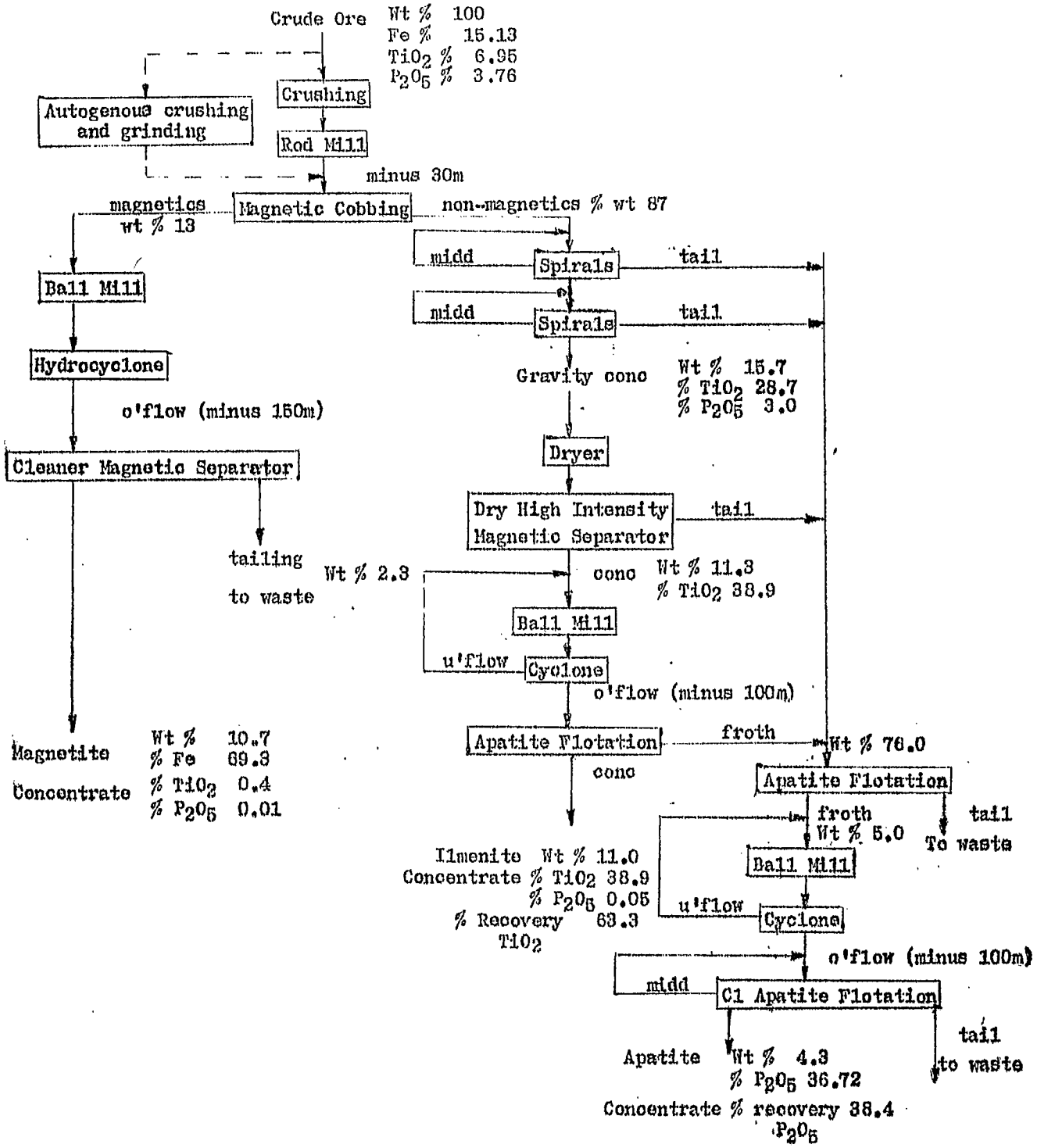


Figure 2.- Recommended Plant Flowsheet

APPENDIX 1

Flotation Test on 30m Cobber Non-Magnetics to Recover Ilmenite

Before the start of the pilot plant investigation, a flotation test was done to discover if an ilmenite concentrate could be recovered by Jones separator after removal of the gangue minerals by cationic flotation. Previous tests, using oleic acid as a collector, on the flotation of ilmenite had been relatively unsuccessful due to poor concentrate grade and recovery.

A 2000 g sample of 30m cobber non-magnetic tailing was ground to 150m and pulped in a cell with 1.0 lb of dextrine WW 82 per ton. After 5 minutes, 0.3 lb each of RADA* and PoA** per ton were added. A froth was floated for 5 minutes - mainly silica and silicates. Another 0.2 lb of RADA and PoA per ton were added and another froth floated. Finally, a third froth was floated with an additional 0.1 lb of RADA and PoA per ton.

The second and third froths were then cleaned with no additional reagents. The tailing was returned to the remaining tailing from the original floats for removal of apatite using soda ash and Aerofloat 710. A total of 0.5 lb of Aerofloat 710 per ton was added, after the pulp had been adjusted to a pH of 8.5 using soda ash. The apatite froth was cleaned once, and the remaining rougher and cleaner tailing products passed through a Jones separator twice at 7 amperes. The results are shown in Table 14.

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* Rosin Amine D. Acetate 70%, Hercules Powder Co.

** Frother mix (by weight) 50% pine oil, 2.5% Aerosol OT 100, 47.5% water.

TABLE 14

Flotation of 30m Non-Magnetics Followed by Jones Separation

Product	Weight %	Analysis % **		Distn %
		TiO_2	P_2O_5	TiO_2
No.1 silica froth	55.3	4.04	3.35	31.7
No.2 and 3 silica cl froth	5.3	8.21		6.3
Apatite cl froth	4.7	1.03	37.73	0.7
7 amp cl conc	11.1	30.16	0.055	47.7
7 " " midd	3.8	10.32		5.5
7 " " tail	0.7	5.09		0.6
7 " R midd	12.4	3.58		6.2
7 " R tail	6.7	1.36		1.3
Feed*	100.0	7.03		100.0

*calculated ** From Internal Report MS-AC-63-413.

From the results of this test, flotation did not seem to hold out much promise. Depression of fine ilmenite in the gangue flotation stage was not very good, leading to a high ilmenite loss. It was not found possible to float a considerable amount of the silicate gangue, especially at the coarser sizes. It was, therefore, decided to abandon flotation in favour of gravity separation followed by high intensity magnetic separation.

PDRM:WSJ:EBM