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PILOT PLANT CONCENTRATION OF A TACONITE IRON ORE FROM KUKATUSH MINING CORPORATION BY SILICA FLOTATION

by

P. D. R. MALTBY & L. L. SIROIS

MINERAL PROCESSING DIVISION

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SUMMARY OF RESULTS

Laboratory and small scale pilot plant flotation tests were carried out on a sample of cobbed taconite concentrate from ore of Kukatush Mining Corporation, Kukatush, Ontario. This sample was sent from Lakefield Research of Canada after the crude ore had been ground to 10 mesh in a Cascade mill and cobbed on a magnetic separator. Recovery was approximately 88% of the iron in the crude ore.

In a pilot plant test, a combined flotation concentrate assaying 66.9% Fe and 5.8% SiO_2 was produced with a recovery of 79.4% of the iron in the original crude ore. Ratio of concentration from the crude ore was 2.40:1. The concentrate was 92% minus 325 mesh and a filter cake moisture of 9.6% was obtained using a test filter leaf.

The cost of flotation reagents was $0.65/ton of concentrate. By one stage of hydroseparation, it was possible to upgrade the concentrate to 67.9% Fe with 4.3% SiO_2.

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*Scientific Officers, Mineral Processing Division, Mines Branch, Department of Mines and Technical Surveys, Ottawa, Canada.

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INTRODUCTION

The purpose of this investigation was to further investigate, at small pilot plant scale, the process developed by bench scale testing for concentration of taconite iron ores by silica flotation instead of magnetic separation. The objective was to produce a concentrate suitable for making premium grade pellets and to prove the feasibility of the process.

During 1960 and 1961 several taconite iron ores were processed in the Metallic Minerals pilot plant by conventional rod and ball milling, followed by three-stage magnetic separation. Subsequent feasibility studies and research work on autogenous grinding proved the desirability of autogenous or semi-autogenous grinding to reduce the high costs of grinding. Pilot plant tests were conducted using semi-autogenous grinding at the Mines Branch, and fully autogenous grinding at Lakefield Research of Canada Limited. The economies expected were proven but it was found that to make a pellet grade concentrate, an extremely fine grind of 96-98% minus 325 mesh was necessary. This gave rise to the expensive and nearly impossible problem of filtering the concentrate to obtain a product dry enough to pelletize. Plant capitalization for the additional magnetic separation and filtering would be high and would more than offset the savings in autogenous grinding.

In addition to these difficulties in making grade, recoveries were not high enough on the taconite ores which contained appreciable non-magnetic iron in addition to magnetite. Therefore, a research investigation was initiated to find a more feasible approach to making a pellet-grade concentrate, preferably at a coarser size, thereby saving in both grinding and filtering costs. Among several alternatives, the flotation of silica was proposed as this process was being investigated both as a final upgrading stage in magnetic concentration plants, and as a concentrating process for off-grade hematite ores.

The first laboratory scale tests were encouraging and, as the investigation proceeded, it was apparent that flotation of silica gave better grades of concentrate at higher recoveries than the conventional process. Since the laboratory tests were so successful, it was decided that a pilot plant continuous test should be made. Through the cooperation of Mr. J. C. Dumbrille, President of Kukatush Mining Corporation, a shipment of minus 10 mesh cobbed concentrate from Kukatush ore was obtained. The processing of this ore sample by silica flotation is the subject of this report. The details of the laboratory test work will be reported as a separate investigation.

The previous investigations have been reported in Industrial Confidential reports either by the Mines Branch or Lakefield Research of Canada.

Shipment

The material received at the Mines Branch was cobbed concentrate from a test run at Lakefield Research of Canada Limited. The original ore was mixed in a ratio of 1:1 type 'E' ore to type 'F' ore and was said to be representative of the orebody at Kukatush, Ontario. Arrangements for shipping the ore to the Mines Branch were made by Mr. J. C. Dumbrille, President, Kukatush Mining Corporation (1960) Limited. One drum of cobbed concentrate was received at the Mines Branch on June 29, 1962, weighing 400 lb. A further nine drums were received on September 12, 1962, with a net weight of 6807 lb. Six green drums, weighing 5058 lb net, designated S-1 concentrate Test 1-3, and three black drums weighing 1749 lb net, designated S-1 concentrate Test 17 - 1, 2 and 3, made up the shipment. The concentrate in the black drums was of slightly higher grade than the concentrate in the green drums.

Sample Analysis

All the iron assays in this investigation were done by the writers using the 'Lerch' method for iron determinations by the stannous chloride-potassium dichromate procedure. Silica assays were done by the Analytical Chemistry Sub-division, Mineral Sciences Division, Mines Branch. Magnetic iron assays were done by Davis tube separation with iron analysis of the products.

Outline of Investigation

In two previous Mines Branch reports*, tests were done on types 'E' and 'F' Kukatush ore separately. Whereas an acceptable concentrate was produced on 'E' ore using a standard taconite flowsheet, the results on 'F' ore showed that the highest grade of concentrate produced was only 64.39% Fe at a grind of 98.2% minus 325m. Further testing was done at Lakefield Research of Canada Limited, on a 44 ton sample made up of 50% 'E' ore and 50% 'F' ore. Autogenous grinding, by Cascade and pebble milling, was used in conjunction with magnetic separation. Their report stated that the best results obtained gave a concentrate assaying 65.62% Fe with 8.70% SiO_2 at a recovery of 76.3% of the iron in the original feed. The concentrate was approximately 98% minus 325m and ratio of concentration from crude feed was 2.87:1.

Laboratory tests meanwhile at the Mines Branch had shown that concentrates assaying better than 67% Fe could be obtained at a grind of about 90% minus 325m using silica flotation. Accordingly, a drum of cobbed concentrate was shipped from Lakefield for further testing. This cobbed concentrate was obtained by crushing the crude ore wet in a Cascade mill,

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*Mines Branch Investigation Reports IR 61-85 and IR 61-103.

and cobbing the mill discharge at about 10m on a magnetic wet drum separator.

After the results from different flotation tests had been evaluated, a flowsheet was set up, similar to the one shown in Figure 1, and three tests were made using a dry feed rate of 250 lb/hr for each test.

MINERALOGY

The mineralogy of both 'E' and 'F' ores has been examined.** Generally, 'E' ore consists of magnetite-rich bands, chert-rich bands, and jasper bands which will liberate to give an acceptable concentrate at about 90% minus 325m with good recovery. The 'F' ore consists of magnetite-rich and chert-rich layers. The magnetite-rich layers consist in some cases of magnetite in a chert-minnesotaite matrix, or magnetite in a chert-stilpnomelane matrix. This ore requires much finer grinding than the 'E' ore for an acceptable concentrate to be produced.

TEST PROCEDURE AND RESULTS

Small Scale Tests

All laboratory testing was done on the preliminary 400 lb sample. A 20 lb head sample was cut, screened, and the fractions assayed. The results are shown in Table 1.

TABLE 1

Composition of Preliminary Sample

Mesh	Weight %	Analysis % Sol Fe	Distn % Sol Fe
+35	39.5	39.9	34.6
-35 +48	7.4	41.9	6.8
-48 +65	6.9	41.7	6.3
-65 +100	6.3	43.3	6.0
-100 +150	5.2	44.7	5.1
-150 +200	4.8	48.1	5.0
-200 +325	6.8	52.6	7.9
-325	23.1	55.8	28.3
Feed*	100.0	45.6	100.0

*calculated

**From Internal Reports MS 61-66 and MS 61-84 of the Mineral Sciences Division, Mines Branch, by Dr. W. Petruk, Mineralogy Section.

In previous work on silica flotation it had been discovered that very few silica particles coarser than 150m would float. Therefore, any efficient flotation scheme would work only on material ground finer than this size. Three different reagent combinations were tried before one was selected to be used in subsequent tests.

In order to compare reagent conbimations, some of the minus 200m material in the feed sample was screened out, and three 1000 g lots were prepared. Reagent additions were governed by results obtained in previous investigations. The flotation results are shown in Table 2.

TABLE 2

Flotation Results on Minus 200m Feed Fraction

Product	Test No. 1			Test No. 2			Test No. 3		
	% Wt	% Fe	Distn % Fe	% Wt	% Fe	Distn % Fe	% Wt	% Fe	Distn % Fe
Rougher tailing	19.7	19.6	7.3	18.8	23.7	8.5	24.3	18.05	8.9
Cleaner "	21.8	50.6	21.0	11.8	35.8	8.1	27.9	48.5	27.2
Concentrate	58.5	64.5	71.7	69.4	63.0	83.4	47.8	66.5	63.9
Feed*	100.0	52.6	100.0	100.0	52.4	100.0	100.0	49.9	100.0

*calculated

In Test No. 1, the pH was raised to 8.5, and a total of 1 lb/ton Dextrine WW82 and 0.6 lb/ton Armac C was used.

In Test No. 2, a fatty acid collector was used at a high pH with $CaCl_2$ as silica activator. The pulp was conditioned for 2 minutes with 0.75 lb/ton Dextrine WW82, after which the pH was raised to 11.3 with NaOH. A total of 0.6 lb/ton $CaCl_2$ and 1.3 lb/ton Acintol FA1 tall oil were added.

In Test No. 3, flotation was carried out at a neutral pH. The pulp was conditioned with 1 lb/ton Dextrine WW82, and was floated using 0.65 lb/ton RADA* and 0.30 lb/ton POA**.

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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.

From the results of these and other tests, the method used in Test No. 3 appeared as promising as any, and it was decided to concentrate on this exclusively. The reagent combination used in this method was also the cheapest.

In order to get good recoveries, tests were carried out to recover iron lost in the middlings. It appeared from microscopic examination that this fraction contained a large amount of minnesotaite and stilpnomelane which required finer grinding for liberation of the iron. Accordingly, a test was carried out in which the middling fraction was reground before refloating. 2000 g of crude feed was ground for 40 minutes in a ball mill to 86% minus 325m and all passing 150m. The pulp was conditioned in an Agitair cell for 5 minutes with 1.6 lb/ton Dextrine WW82. A total of 0.5 lb/ton RADA and 0.25 lb/ton POA was added to the rougher float which lasted 8 minutes. Middlings were then floated for 5 minutes with an additional 0.20 lb/ton RADA and 0.10 lb/ton POA. The results of this test are shown in Table 3.

TABLE 3

Flotation Results on Crude Feed

Product	Weight %	Analysis % Sol Fe	Distn % Sol Fe
Rougher tailing	35.9	22.11	16.5
Middling	19.8	54.5	22.6
Concentrate	44.3	65.95	60.9
Feed*	100.0	48.0	100.0

*calculated

The middling product was filtered and ground for 15 minutes, to 97% minus 325m. It was then floated for 5 minutes using 0.2 lb/ton Dextrine WW82, 0.15 lb/ton RADA and 0.07 lb/ton POA, based on the original feed.

The results of this float are shown in Table 4.

TABLE 4

Results of Flotation of Reground Middlings

Product	Weight % of crude feed	Analysis % Sol Fe	Distn % of crude feed Sol Fe
Regrind midd conc	12.5	62.71	16.4
" " tail	7.3	40.52	6.2
Feed*	19.8	54.5	22.6

*calculated

The combined concentrate gave an iron recovery of 77.3% at a grade of 65.5% Fe. This new concentrate was treated in a hydroseparator at an upflow rate of 60 ft/hr for the results shown in Table 5.

TABLE 5

Results of Hydroseparation of Combined Concentrates

Product	Weight % of crude feed	Analysis %		Distn % of crude feed Sol Fe
		Sol Fe	SiO_2	
Hydroseparator spigot	2.7	44.7		2.6
Hydroseparator o'flow	54.1	66.5	6.8	74.7
Feed*	56.8	65.5		77.3

*calculated

The results of a screen test on the hydroseparator spigot are shown in Table 6.

TABLE 6

Results of Screen Test on Hydroseparator Spigot

Mesh	Weight %	Cum Weight % Retained
+150	0.1	0.1
+200	4.4	4.5
+325	11.8	16.3
-325	83.7	
Total	100.0	

Before it was finally decided to beneficiate the larger sample using the above method in the pilot plant, some tests were done at pH from 4.5 to 5.5 using no Dextrine. The theory is that at this pH range, the charge on the iron and silica particles are opposite. Silica has a negative sign and thus should be collected by cationic reagents selectively.

After a 40 min grind, a 1000 g sample of crude feed, all passing 150m, was pulped for 5 minutes at pH 5 using 0.3 lb/ton H_2SO_4. A rougher float for 10 min used 0.4 lb/ton Armac C. A further 0.3 lb/ton Armac C was added and a middling was floated for 5 min. Iron depression was good in the first float, but iron floated too readily in the middling float. In order to get a reasonable iron recovery, a lower grade of concentrate was made. The results of this test are shown in Table 7.

TABLE 7

Flotation Results on Crude Feed at Acid pH

Product	Weight %	Analysis % Sol Fe	Distn % Sol Fe
Rougher tailing	28.7	18.48	11.1
Middling	30.6	54.9	35.0
Concentrate	40.7	63.45	53.9
Feed*	100.0	47.9	100.0

*calculated

Other collectors including RADA and Armoflote P were also tried at acid pHs, but the method did not show any real advantage over the previous results. Due to other commitments, there was not time to complete any more laboratory testing before the pilot plant run was started.

Pilot Plant Tests

The duration of the pilot plant tests was limited by the small amount of feed available - about 3.5 tons. It was decided to grind the crude feed in a ball mill at a rate of 250 lb/hr, passing the discharge through a Dorr-Oliver M30 cyclone after removing coarse material on a 48m vibrating screen. The cyclone spigot would be returned for regrinding, and the cyclone overflow product would be thickened before flotation. The only cells available for a run of this size were two banks of five Galigher Agitair 8 in.cells. Conditioning was done in a separate conditioner before flotation, retention time being calculated as 10 min at 30% solids. Flotation retention time at a feed rate of 4 lb/min at 30% solids for 10 cells would be approximately 20 min. The final flowsheet is shown in Figure 1.

Test 1

During the first 2 days of the tests, no provision was made for regrinding the middling product. Reagents were made up to the following dilutions: Dextrine WW82 2%, RADA 1%, and POA 1%. After 5 hours the circuit appeared to be well settled and the middling product was returned to the thickener ahead of flotation. Time samples were taken, and screen tests showed the grind to be on the coarse side. Accordingly, before the start of Test 2, an additional 150 lb of balls was added for a total charge of 750 lb. Reagent additions were Dextrine, 1.8 lb/ton; RADA, 0.9 lb/ton, split between the rougher and middling circuits, and POA, 0.45 lb/ton, also split between the two circuits. The results of time samples are shown in Table 8. Total running time was 10 hr.

TABLE 8

Results of Test 1

Product	Rate lb/hr	Solids %	Analysis % Sol Fe
Ball mill discharge	920	66	52.54
Cyclone spigot	620	76	56.4
Cyclone overflow	300	15	47.7
Flotation feed	308	30	50.6
Rougher conc	228	35	58.91
Rougher tail	80	22	37.90
Middling	120	5	52.23
Concentrate	100	30	65.60

Cyclone feed density was maintained at 30% solids, pressure was 35 psi. Screen tests were done on various products with the results shown in Table 9.

TABLE 9

Results of Screen Tests on Test 1

Mesh	Ball Mill Discharge	Cyclone		Middling	Concentrate
		Spigot	Overflow		
+100	5.0	5.2	0.8	0.1	1.0
-100 +150	4.4	5.4	1.5	0.8	1.6
-150 +200	7.6	10.0	2.8	2.0	2.8
-200 +325	24.6	24.6	10.7	13.0	9.6
-325	58.4	54.8	84.2	84.1	85.0
Total	100.0	100.0	100.0	100.0	100.0

No metallurgical balance was drawn up for this test due to the unbalanced circuit.

Test 2

For the next two days, the middlings were returned to the thickener before flotation after regrinding in a separate mill. Conditions otherwise were the same as before. Due to the low density of the middlings in the regrind mill (26% solids), not enough grinding was achieved. It was found hard to control the circuit with the middling return load and thus ensure that the product was up to grade. The results are shown in Table 10.

TABLE 10

Results of Test 2

Product	Weight %	Analysis %		Distn %
		Sol Fe	Mag Fe	Sol Fe
Crude feed	100.0	47.62	45.4	100.0
Ball mill discharge	297.2	-	-	-
Cyclone spigot	197.2	57.46	-	-
Cyclone overflow	100.0	48.1	-	100.0
Flotation feed	182.5	50.36	-	191.1
Rougher conc	134.9	57.85	-	162.3
Rougher tail	47.6	29.14	22.4	28.8
Middling	82.5	53.1	-	91.1
Final conc	52.4	65.33	-	71.2

The reagent consumption was:-

Dextrine WW82 - 2 lb/ton added to the conditioner with a retention time of 10 minutes;

RADA - 0.8 lb/ton. Half added to the flotation feed and the remainder added in two stages in the middling circuit, one at the feed end and one half-way down;

POA - 0.4 lb/ton added in the same way as RADA.

Duration of the run was 8 hr.

The results of screen tests are shown in Table 11.

TABLE 11

Results of Screen Tests on Test 2

Mesh	Ball Mill Discharge	Cyclone		Middling Ball Mill Discharge	Final Concentrate
		Spigot	O'flow		
+65	2.2	1.6	-	-	-
-65+100	3.0	3.2	-	-	-
-100+150	4.0	4.8	1.0	-	1.2
-150+200	6.8	8.2	1.4	1.6	2.0
-200+325	24.6	25.8	6.0	7.6	8.0
-325	59.4	56.4	91.6	90.8	88.8
Total	100.0	100.0	100.0	100.0	100.0

Test 3

The balance of feed left was sufficient for a 9 hr run, and consisted of the higher grade material shipped in 3 black drums. The results of cobbing tests showed that this feed contained 88.7% of the iron in the crude ore and 63.0% of the original feed weight. Grade was 49.1% Fe.

Laboratory tests on tailing from the previous test showed that additional iron could be recovered from the rougher tailing as well as from the middling. From previous experience, it also appeared that a separate regrind on the middling and tailing products followed by flotation in a separate circuit would yield better grades and recoveries. Due to the lack of small flotation cells, this could not be done in conjunction with the rest of the test. Neither did time allow the middling and rougher tailing products to be stockpiled and treated later. Accordingly, the circuit was run as before but the middling was not returned to the head of the circuit but collected for subsequent testing in the laboratory.

The regrind mill was still operated and middling flotation tried after regrinding in two 12 in. Fagergren cells. However, control and results were poor as there was not enough feed to build up a stable froth. In this test it was found much easier to control the rougher tailing grade, and the middling circuit. The flotation feed density was varied between 30 and 40% solids and it seemed that 35% solids gave good results. Densities and circulating loads in the grinding circuit remained as in previous tests. The results obtained in the test are shown in Table 12.

TABLE 12

Results of Test 3

Product	Weight %	Analysis %			Distn %
		Sol Fe	Mag Fe	SiO_2	Sol Fe
Flotation feed	100.0	49.31	47.1	-	100.0
Rougher conc	65.7	62.7	-	-	83.5
Rougher tail	34.3	23.73	17.3	-	16.5
Middling	22.1	53.11	-	-	23.8
Concentrate	43.6	67.39	-	4.91	59.7

A sample of the concentrate was upgraded by hydroseparator at an upflow rate of 60 ft/hr to 68.12% Fe with 4.64% SiO_2. Recovery was 54.6% of the iron in the flotation feed. The grade of the hydroseparator overflow was 60.2% Fe.

A representative sample of combined rougher tailing and middling was ground in a laboratory ball mill for 10 min and was refloated for 6 min.

The flotation concentrate was upgraded in the hydroseparator. Results are shown in Table 13.

TABLE 13

Results of Refloating Ground Middling and Tailing

Product	Weight % of flotn feed	Analysis %			Distn % of flotn feed Sol Fe
		Sol Fe	Mag Fe	SiO_2	
Regrind flotn conc	22.6	65.12	-	-	29.8
Hydrosep. conc**	19.4	67.22	-	5.04	26.4
" o'flow	3.2	52.47	44.9	-	3.4
Final tail	33.8	15.25	7.14	-	10.5
Feed*	56.4	35.2	-		40.3

*calculated **upflow rate was 60 ft/hr.

A screen test showed the hydroseparator concentrate to be 97.2% minus 325m.

With no hydroseparation the combined concentrates assayed 66.9% Fe with 5.8% SiO_2. Recovery was 89.5% of the iron in the flotation feed or 79.4% of the iron in the crude ore. Ratio of concentration from the crude ore was 2.40:1. With hydroseparation on just the regrind concentrate, a grade of 67.3% Fe with a recovery of 86.1% was obtained. With full hydroseparation on both concentrates, grade was 67.9% Fe and 4.8% SiO_2, with 81% recovery. Ratio of concentration from crude ore was 2.69:1.

The results of screen tests are shown in Table 14.

TABLE 14

Results of Screen Tests on Test 3

Mesh	Crude Feed	Ball Mill Discharge	Cyclone Spigot	Cyclone Overflow	Middling	Final Conc
+10	0.7	-	-	-	-	-
-10 +14	0.8	-	-	-	-	-
-14 +20	0.5	-	-	-	-	-
-20 +28	3.2	-	-	-	-	-
-28 +35	11.4	-	-	-	-	-
-35 +48	11.1	-	-	-	-	-
-48 +65	11.3	2.0	1.0	-	-	-
-65 +100	11.0	2.8	3.0	-	-	-
-100 +150	8.0	3.6	4.6	0.6	0.8	2.0
-150 +200	6.0	6.4	7.8	1.0	1.6	2.8
-200 +325	8.0	20.2	22.2	4.4	5.6	9.2
-325	28.0	65.0	61.4	94.0	92.0	86.0
Total	100.0	100.0	100.0	100.0	100.0	100.0

The amounts of reagents used are shown in Table 15.

TABLE 15

Reagent Quantities in Test 3

Reagent	Rougher lb/ton	Middling lb/ton	Refloat lb/ton	Cost c/ton
Dextrine WW82	1.8	-	0.5	21
RADA	0.35	0.35	0.1	20
POA	0.20	0.15	0.05	3

Total cost of reagents was $0.44/ton of flotation feed.

A filtering test was done on the combined concentrate at 24 in. Hg using a test filter leaf. Cake thickness and moisture were 0.5 in. and 9.6% respectively.

Operation of Hydroseparator

From the results it was noted that hydroseparation losses on the flotation concentrates were considerable. On a combined concentrate, the total iron loss was 8.5% of the iron in the flotation feed. Some of this loss may be due either to the fact that the feed was not magnetized first or else that the upflow rate was too high. Some Davis tube tests were done on the hydroseparator overflow products to determine the magnetic iron content. The Davis tube concentrates assayed about 63% Fe and contained about 89% of the iron in the hydroseparator overflow. In practice, the hydroseparator overflow could be passed through a magnetic separator, and the concentrate combined with the original hydroseparator spigot product. Doing this on both flotation concentrates, the final recovery would be 88.5% of the iron in the flotation feed at a grade of 67.47% Fe.

Discussion

The advantages of treating this ore by flotation over other known methods are:

1. Concentrates can be produced at a higher grade containing less silica with better iron recoveries. These concentrates can be easily dewatered with one filtering stage to under 10% moisture.

2. The process is flexible. After installation of a concentrator, improvements in reagents and techniques can be incorporated with little or no additional capital cost. Changes could also be made easily to treat differences in ore structure.

3. Grinding costs should be lower. With the ore tested, a relatively coarse concentrate of finished grade could be first recovered. The refractory ore could then be reground separately for additional iron recovery in open circuit. In brief, the flotation process effects a classification of the iron minerals.

4. It should be possible to produce flotation concentrates of finished grade using autogeneous grinding. This could not be done using an autogeneous grind with magnetic separation even on a fine-grained relatively pure magnetite-silica ore.

5. There may be savings in capital cost on a concentrator using flotation.

CONCLUSIONS

The results in this investigation have shown that a sample of cobbed concentrate could be upgraded by continuous flotation to produce a concentrate to meet premium grade pellet specifications.

The best flotation method appeared to be as follows. The cobbed concentrate was ground to 90% minus 325m containing as little material on 150m as possible. The ground product was conditioned with Dextrine WW82, followed by flotation of silica and refractory material to give a primary concentrate assaying better than 67% iron. The float product was then reground to about 97% minus 325m in open circuit and was refloated in a separate circuit. The combined concentrates can be upgraded by hydroseparators depending on the final grade desired. Filtration tests on the combined concentrate showed that a cake moisture of under 10% could be made. It should be possible to grind the ore prior to flotation in a pebble mill, using pebbles made from the crude ore. However, it would probably be more practical to do the float regrind in a ball mill.

The reagent cost in Test 3 was $0.65/ton of concentrate or about $0.27/ton of crude ore. A considerable part of this cost is compensated as the amount of grinding in the flotation process is less than that required to produce concentrates by magnetic separation. It is anticipated that no filter aids would be needed to dewater the flotation concentrates before pelletizing. A further saving in grinding costs might be realized by removing part of the minus 150m material already in the cobbed concentrate, and passing it by the primary grinding mill. It is intended to try this at the Mines Branch using a 3 in. Dorrclone in a closed circuit.

With the amount of time available, it was not possible to investigate reagent additions to any great extent. However, it is believed that some reduction in the amount of Dextrine WW82 might be achieved. As the middling and tailing products are both returned for regrinding, it is possible that Dextrine need only be added to the middling or regrind floats for magnetite depression. It is proposed to investigate other reagents to improve the economics or metallurgy of the process. A large scale pilot plant test at Lakefield Research using autogeneous grinding and rock milling in conjunction with flotation is also proposed.

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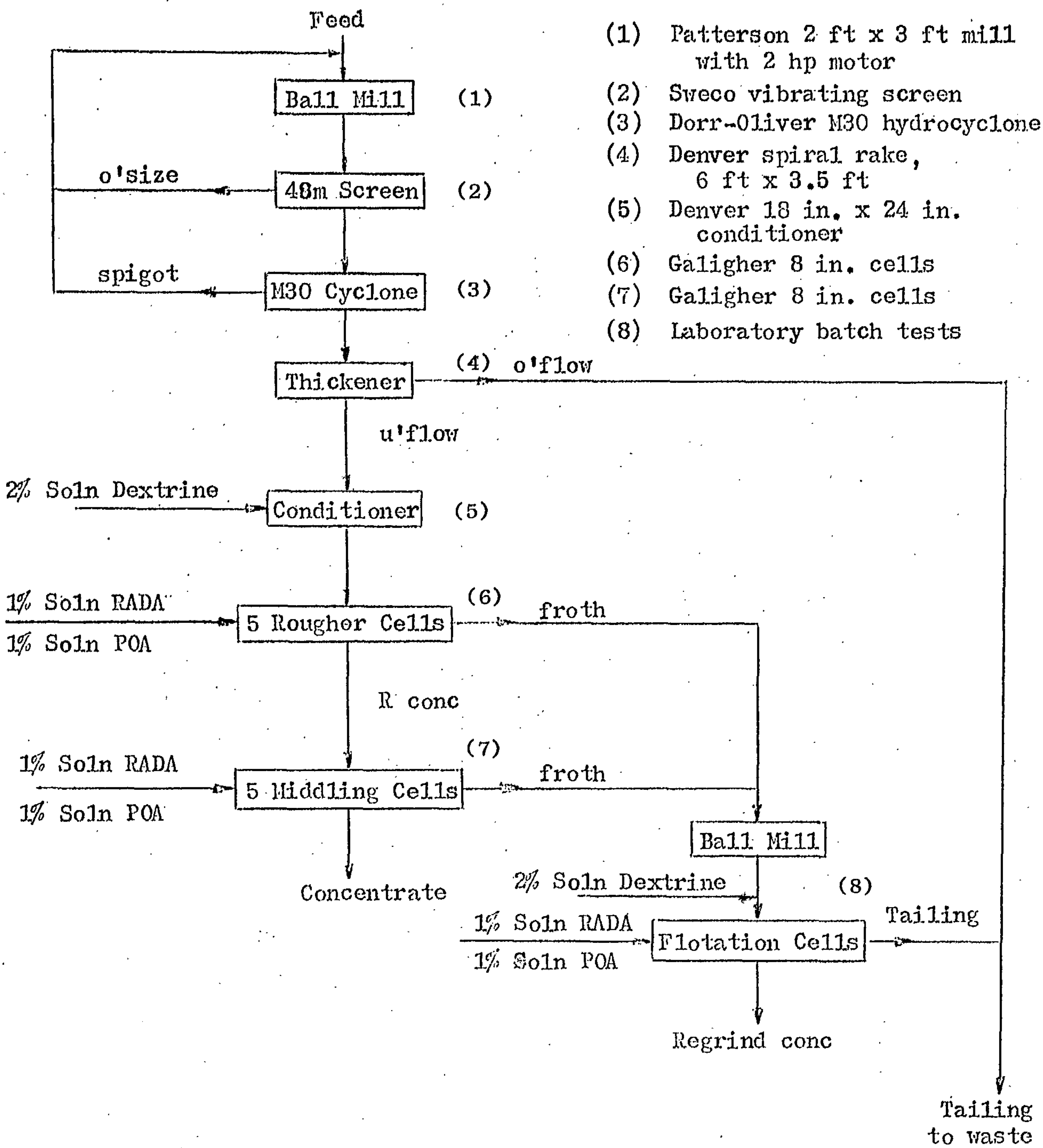
FDM:EBM



(1) Patterson 2 ft x 3 ft mill with 2 hp motor
(2) Sweco vibrating screen
(3) Dorr-Oliver M30 hydrocyclone
(4) Denver spiral rake, 6 ft x 3.5 ft
(5) Denver 18 in. x 24 in. conditioner
(6) Galigher 8 in. cells
(7) Galigher 8 in. cells
(8) Laboratory batch tests

Figure 1. - Flowsheet of Pilot Plant Test 3