Dr. Downes

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PILOT PLANT TESTS USING SILICA FLOTATION ON SAMPLES FROM STEEP ROCK IRON MINES LIMITED, ATIKOKAN, ONTARIO

by

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

MINERAL PROCESSING DIVISION

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SUMMARY OF RESULTS

Laboratory and pilot plant flotation tests were done on samples of 'crude' and combined 'crude' and 'direct' ores. Flotation laboratory tests gave concentrates assaying 62% Fe with less than 3% SiO_2 on deslimed feed. On undeslimed feed, concentrates over 60% Fe were produced and the recovery was 70.3% of the iron in the original feed compared to 60.5% using deslimed feed.

Approximately 100 tons of ore mixed in the ratio of 10 parts 'direct' to 7 parts 'crude' was treated in three pilot plant tests. After grinding and classification, the ore was deslimed in two stages of cyclones. About one-third of the ore was discarded as slimes before flotation. Altogether, 34 tons of flotation concentrate was produced grading over 60% Fe, and was shipped in drums for pelletizing tests. In Test 3, a concentrate was produced assaying 60.8% Fe with 4.4% SiO_2 at a recovery of 87.9% of the iron in the flotation feed.

Laboratory and pilot plant heavy-media separation tests were conducted on the 'crude' ore. A laboratory H.M.S. test was done on a sample of 'pyritic' ore.

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INTRODUCTION

The purpose of the investigation was to test a method of beneficiating Steep Rock ore by silica flotation on a pilot plant scale, and to prepare sufficient concentrate for pelletizing tests.

Shipment

Three types of iron ore were received from Steep Rock Iron Mines Limited, shipped under the supervision of Mr. R. J. Tremblay, Chief Metallurgist. A total of 116 tons of minus 4 in. 'direct' ore, 44 tons of minus 4 in. 'crude' ore, and 24 tons of 'pyritic' material were received at the Mines Branch in two gondola cars on July 16, 1962. Three drums containing 300 lb of minus 4 in. 'direct', minus 4 in. 'crude', and minus 4 in. 'pyritic', for preliminary tests were shipped express, and were received at the Mines Branch on July 18, 1962.

Sample Analysis

Unless otherwise shown, all iron analyses were done in this investigation, by the writers, using the 'Lerch' method for iron determinations by the stannous chloride-potassium dichromate procedure. Silica and insoluble analyses were done by the Analytical Chemistry Sub-division, Mineral Sciences Division, Mines Branch.

Outline of Investigation

In recent years, steel companies have been demanding higher grade, agglomerated concentrates for blast furnace feed. This has meant that direct shipping iron ore producers, like Steep Rock, have been forced to develop ways to beneficiate their ores in order that they may still have a market in the future.

There already exists at Steep Rock, concentrators using gravity separation methods, in which the lower grade 'crude' ore is upgraded to make a coarse and a fine concentrate. After crushing, the 'crude' ore is sized for H.M.S. and separation by jigs and spirals. Prior to the spiral treatment, the feed is sized in cyclones. The cyclone overflow, consisting of minus 150m material is rejected. This represents about 30% of the total concentrator feed weight.

A considerable amount of research has been conducted at the mine to discover the best way to concentrate this material. From this research, the Company has developed a cationic silica flotation process at a neutral pH which will produce an acceptable concentrate on deslimed material from the plant cyclone overflows. It is expected that after 1965 a large percentage of Steep Rock production will be concentrate in the form of pellets. Pellet feed would come most likely from ground gravity concentrates mixed with flotation concentrates. In order to find out enough information to build a pilot plant at the mine, the Mines Branch was asked to carry out this investigation. Another purpose of the investigation was to produce approximately 30 tons of flotation concentrate which could be shipped to various companies for pelletizing tests. Figure 1 shows the proposed new treatment scheme.

Preliminary flotation tests were conducted in the Mines Branch laboratory to investigate the Steep Rock process and to compare it to other processes. After these tests, a pilot plant investigation was started in which the ore was ground to minus 150m, was deslimed at 5 to 10 microns in cyclones, and the deslimed material was upgraded by the Steep Rock flotation procedure.

Besides flotation, the investigation included wet grinding a sample of 'direct' ore to minus 20m, H.M.S. tests on a sample of 'crude' ore and, in the laboratory, tests on a sample of 'pyritic' ore.

TEST PROCEDURE AND RESULTS

Laboratory Testing

Flotation

The 'crude' ore was the first ore to be investigated. The plus $\frac{1}{4}$ in. fraction of the sample, which was to be treated by sink-float method, was removed and the minus $\frac{1}{4}$ in. material was screened with the analysis shown in Table 1.

TABLE 1

Screen Analysis of Minus $\frac{1}{4}$ in. 'Crude' Ore

Mesh	Weight g	Weight %	Analysis % Sol Fe	Fe Units	Distn % Sol Fe
+4	17.0	3.0	41.6	1.2	2.7
-4+6	68.7	11.6	44.6	5.2	11.9
-6+8	67.7	11.4	42.3	4.8	10.9
-8+10	72.7	12.3	44.2	5.4	12.3
-10+14	90.7	15.3	44.9	6.9	15.8
-14+20	79.6	13.4	43.9	5.9	13.5
-20+28	54.6	9.2	43.3	4.0	9.1
-28+35	44.5	7.5	45.0	3.4	7.8
-35+48	10.4	1.8	44.6	0.8	1.8
-48+65	12.2	2.1	46.5	1.0	2.3
-65+100	11.2	1.9	46.3	0.9	2.1
-100+150	8.4	1.4	46.3	0.6	1.4
-150+200	6.5	1.1	43.8	0.5	1.1
-200+325	7.8	1.3	41.8	0.5	1.1
-325	41.0	6.7	39.6	2.7	6.2
	593.0	100.0		43.8	100.0

Previous testing on other iron ores has shown that for silica flotation, most of these ores should be reduced to minus 150m. To produce as little slimes as possible, stage grinding was adopted and the results for a minus 150m grind are shown in Table 2.

TABLE 2

Size Analysis of 'Crude' Ore Ground to minus 150m

Mesh	Weight g	Weight %	Analysis % Sol Fe	Fe Units	Distn % Sol Fe
+65m	0.6	0.1	29.4	0.1	0.2
-65+100m	1.0	0.2	34.1	0.2	0.5
-100+150m	8.5	1.3	41.4	0.5	1.2
-150+200m	107.2	16.4	44.2	7.3	17.9
-200m+56 μ	22.8	3.5	41.6	1.5	3.7
-56+40 μ	150.7	23.1	41.8	9.7	23.8
-40+28 μ	66.3	10.2	42.2	4.3	10.6
-28+20 μ	65.8	10.1	38.2	3.9	9.6
-20+14 μ	55.8	8.6	34.1	2.9	7.1
-14+10 μ	50.8	7.8	34.3	2.7	6.6
-10 μ	122.0	18.7	40.6	7.6	18.8
	651.5	100.0		40.7	100.0

Only a few flotation tests were performed on the 'crude' ore alone. The best results were obtained with an anionic silica float using 2 lb/ton of Dextrine 8072, 1.6 lb/ton of sodium hydroxide to produce a pH of 11.7, 0.75 lb/ton of calcium chloride and 2 lb/ton of Acintol FA-1. The test was done without desliming. The results of this test are shown in Table 3.

TABLE 3

Anionic Silica Float of Undeslimed 'Crude' Ore

Products	Weight g	Weight %	Analysis % Sol Fe	Fe Units	Distn % Sol Fe
Conc	280	62.2	55.0	34.2	79.0
Midds	74	16.5	34.1	5.6	13.0
Tails	96	21.3	16.3	3.5	8.0
	450	100.0		43.3	100.0

Since it was planned to run the pilot plant test on a 10:7 mixture of 'direct' to 'crude' ore, efforts were then concentrated in that direction.

The plus 3/8 in. fraction of the 'direct' ore was removed and the minus 3/8 in. portion was reduced to minus 150m by stage grinding. It was then mixed in a ratio of 10:7 with the 'crude' ore, also reduced to a minus 150m size. The composite had the structure shown in Table 4.

TABLE 4

Analysis of 10:7 'Direct' to 'Crude' Ore Composite

Mesh	Weight g	Weight %	Analysis % Sol Fe	Fe Units	Distn % Sol Fe
+100m	3.1	1.0	55.4	0.6	1.1
-100+150m	9.4	3.2	55.7	1.8	3.3
-150+200m	89.0	29.9	56.7	17.0	30.9
-200m+56 μ	19.4	6.5	60.2	3.9	7.1
-56+40 μ	30.6	10.3	56.3	5.8	10.5
-40+28 μ	24.4	8.2	55.1	4.5	8.2
-28+20 μ	19.0	6.4	53.3	3.4	6.2
-20+14 μ	18.0	6.1	50.9	3.1	5.6
-14+10 μ	15.2	5.1	48.6	2.5	4.5
-10 μ	69.0	23.3	53.1	12.4	22.6
	297.1	100.0		55.0	100.0

It had been determined by the Research Department of Steep Rock Iron Mines Limited on work done on their concentrator cyclone overflow, that Dextrine WW82 was the most effective depressant for the iron and that Rosin Amine D Acetate (RADA) and a mixture of Pine Oil, Aerosol OT 100 and water (POA) were the most satisfactory collector and frother for their ores. Testing with other depressants, done at the Mines Branch laboratories, verified this result. This combination of reagents was thus adopted for the ensuing testing.

Tests were performed on the composite without any desliming. The results of Test 12, a two-stage cleaning operation, are shown in Table 5.

TABLE 5

Cationic Silica Float on Composite without Desliming

Products	Weight g	Weight %	Analysis % Sol Fe	Fe Units	Distn % Sol Fe
Rougher tails	257.0	25.1	44.9	11.3	20.5
1st cleaner tails	71.4	7.0	51.1	3.6	6.5
2nd " "	82.7	8.1	49.4	4.0	7.3
Fe conc *	612.0	59.8	60.4	36.1	65.7
	1023.1	100.0		55.0	100.0

* Fe conc contained 97.6% plus 5 microns or 99.6% plus 4 microns.

Flotation pH was natural at 7.5.

The total amounts of reagents used were: 1.9 lb/ton of Dextrine, 0.7 lb/ton of RADA, and 0.3 lb/ton of POA mixture for a total flotation time of 10 min.

In a three-stage cleaning operation, Test 19, the results shown in Table 6 were obtained with a 12 min total flotation time.

TABLE 6

Cationic Silica Float on Composite Without Desliming

Products	Weight g	Weight %	Analysis % Sol Fe	Fe Units	Distn % Sol Fe
Rougher tails	136.8	13.3	42.6	5.7	10.2
1st cleaner tails	115.2	11.2	50.5	5.7	10.2
2nd " "	58.0	5.6	52.6	2.9	5.2
3rd " "	46.4	4.5	51.0	2.3	4.1
Fe conc *	672.0	65.4	60.3	39.4	70.3
	1028.4	100.0		56.0	100.0

* Fe conc contained 96% plus 5 microns or 99.6% plus 4 microns.

Reagents used were Dextrine 1.5 lb/ton, RADA 0.5 lb/ton, POA 0.4 lb/ton; pH 7.5. The insoluble assay was 5.04% and SiO_2 was 3.84%.

For the same conditions but with only half the Dextrine, 0.7 lb/ton, a higher grade of concentrate (61.9%) could be obtained but at a lower recovery (53.2%); the insolubles were 4.54% or 2.12% SiO_2.

Different types of desliming operations were then attempted -- settling and decantation, desliming pail, hydroseparator. None of these methods produced satisfactory results, as the slimes were flocculated and no reasonable separation could be obtained.

A Dorr-Oliver M-30 cyclone was then used with variations in both the pressure and the feed density. The pressure was varied from 10 psi to 30 psi and the feed density from 5% to 30% solids.

The objective was to attain a flotation feed containing plus 5 micron particles and another one containing plus 10 micron particles to compare their flotation characteristics.

The best combination to arrive at a plus 5 micron flotation feed was obtained with a 15% solids cyclone feed by operating the cyclone at 15 psi. The first overflow was repassed in the cyclone and the second underflow was blended with the first one. The second overflow went to tailings. The results are shown in Table 7.

TABLE 7

Flotation Feed Produced at 15 psi by Cycloning

Products	Weight g	Weight %	Analysis % Sol Fe	Fe Units	Distn % Sol Fe
1st Underflow	2358	69.7	54.8	38.2	70.7
2nd "	334	9.9	50.7	5.0	9.3
2nd Overflow	688	20.4	53.1	10.8	20.0
Feed	3380	100.0		54.0	100.0

The grain-size analysis was done by a sedimentation process using ASTM standard D 422-54 T and showed that 90.0% of the flotation feed was made up of plus 5 micron particles.

Flotation test No. 43 was performed on this feed using 2.0 lb/ton of Dextrine, 0.6 lb/ton of RADA and 0.6 lb/ton of POA to produce the results shown in Table 8.

TABLE 8

Flotation Results Obtained on Plus 5 micron Feed

Product	Weight g	Weight %	Analysis % Sol Fe	Fe Units	Distn % Sol Fe
Rougher tails	60.0	13.4	31.6	4.2	7.7
1st cleaner tails	26.2	5.9	41.8	2.5	4.6
2nd " "	23.2	5.2	42.6	2.2	4.0
3rd " "	57.8	12.9	53.2	6.9	12.6
Fe conc	280.0	62.6	62.0	38.8	71.1 *
	447.2	100.0		54.6	100.0

* 56.9% of cyclone feed

Fe conc contained 99.7% plus 5 microns.

The plus 10 micron flotation feed was obtained with a 10% solids pulp through the cyclone operating at 28 psi. The results are shown in Table 9.

TABLE 9

Flotation Feed Produced at 28 psi by Cycloning

Product	Weight g	Weight %	Analysis % Sol Fe	Fe Units	Distn % Sol Fe
1st Underflow	908	57.6	53.1	30.6	57.4
2nd "	322	20.4	55.3	11.3	21.2
2nd Overflow	346	22.0	51.9	11.4	21.4
Feed	1576	100.0		53.3	100.0

The same conditions were used for the plus 10 micron flotation as for the plus 5 micron and gave the following results from Test 42 shown in Table 10.

TABLE 10

Flotation Results Obtained on Plus 10 micron Feed

Product	Weight g	Weight %	Analysis % Sol Fe	Fe Units	Distn % Sol Fe
Rougher tails	57.5	14.1	26.8	3.8	7.0
1st cleaner tails	21.4	5.2	40.9	2.1	3.9
2nd " "	26.2	6.4	45.1	2.9	5.4
3rd " "	29.0	7.1	52.6	3.7	6.8
Fe conc	275.0	67.2	62.0	41.7	76.9 *
	409.1	100.0		54.2	100.0

* 60.5% of cyclone feed

The grain-size analysis showed that 86.5% of the flotation feed was made up of plus 10 micron particles.

H.M.S. Laboratory Testing

A series of bucket tests were done, using a ground ferrosilicon medium supplied by Steep Rock, on 'crude' and 'pyritic' ore samples. The material in each test ranged from ¼ in. to 1 in. in size. Test procedure was identical for each sample. A bucket of medium was adjusted to the desired density and was agitated. Approximately 1500 g of sample was put into the bucket and was allowed to settle for 10 sec. After the settling period had elapsed, float was removed from the surface, washed on a screen, dried and weighed. The sink product was also washed, dried and weighed. The results of tests on the 'crude' ore are shown in Table 11, and on the 'pyritic' ore in Table 12.

TABLE 11

H.M.S. Bucket Tests on 'Crude' Ore

Medium Sp. Gr.	Product	Weight %	Analysis % Sol Fe	Distn % Sol Fe
2.95	Sink Float	65.7 34.3	56.9 21.2	83.7 16.3
	Feed*	100.0	44.7	100.0
3.10	Sink Float	50.4 49.6	50.65 23.6	71.6 28.4
	Feed*	100.0	41.3	100.0
3.20	Sink Float	50.8 49.2	57.7 25.15	70.3 29.7
	Feed*	100.0	41.7	100.0
3.40	Sink Float	39.4 60.6	59.95 38.9	50.1 49.9
	Feed*	100.0	47.2	100.0

* calculated

TABLE 12

H.M.S. Bucket Tests on 'Pyritic' Ore

Medium Sp. Gr.	Product	Weight %	Analysis % Sol Fe	Insol	S	Distn % Sol Fe
3.25	Sink Float	64.3 35.7	60.4 45.7	6.02	0.36	70.4 29.6
	Feed*	100.0	55.2			100.0
3.00	Sink Float	93.6 6.4	58.6 35.4	7.54	0.50	96.0 4.0
	Feed*	100.0	56.7			100.0
3.10	Sink Float	88.5 11.5	58.5 35.6	7.44	0.17 0.13	92.7 7.3
	Feed*	100.0	55.9			100.0

* calculated

The tests on the 'pyritic' ore were done on a sample cut from the 300-lb drum of 'pyritic' ore. The sulphur content of this sample appeared to be lower than the sample shipped in the car, and so these results should only be regarded as preliminary.

Summary of Results

On a similar feed, flotation tests were done successfully with and without desliming. An acceptable concentrate was produced in each case. Without desliming, a concentrate assaying 60.3% Fe with 3.84% SiO_2 was obtained containing 70.3% of the iron in the original feed. With desliming at 10 microns, a concentrate assaying 62% Fe was obtained containing 60.5% of the iron in the original undeslimed feed. Reagent additions in each case were comparable.

A sizing analysis of each concentrate showed that, in the case of the tests with undeslimed feed, the bulk of the minus 5 micron material had been removed in the flotation froth. As a result, there was little difference in structure between the concentrates. (Compare Tables 5 and 8).

In spite of the apparent increased recovery using undeslimed feed, it appeared that higher concentrate grade could be obtained on deslimed feed. With slimes present, the flotation froth would be that much harder to break down especially if a scavenger flotation circuit was installed to recover additional iron from this froth. For the pilot plant tests, it was decided to deslime before flotation in two stages of cycloning. However, as about 25% of the iron was discarded in the desliming operation, flotation on undeslimed material should not be ruled out without doing a continuous pilot plant test.

PILOT PLANT TESTS

A test programme for treatment of the three samples of ore was drawn up before testing began and is shown in Figure 2. Due to other commitments and the deadline laid down for pelletizing tests, no work was done in the pilot plant on the 'pyritic' ore. The 'direct' and 'crude' ore samples were screened dry as shown at 3/8 in. and 1/4 in. respectively. A sample of the plus 3/8 in. 'direct' ore was ground wet in a rod mill to 20m. The plus 1/4 in. fraction of 'crude' ore was treated by H.M.S., feed weight amounting to nearly 15 tons. The screen undersize of both samples was blended in the desired proportions as flotation feed.

H.M.S. Tests on 'Crude' Ore

The H.M.S. feed was prepared by screening the 'crude' ore on 1/4 in. The screen oversize was then screened on a 1 in. screen, and any oversize was crushed until it passed the 1 in. screen. This minus 1 in. plus 1/4 in. material was washed and was allowed to dry. The total weight was just under 15 tons.

A 24 in. O.C.C. type 152 heavy-media separator was set up with washing screens. Ferrosilicon media was made up to a density of 3.2 and feed was introduced at a rate of about 1000 lb/hr. Great difficulty was experienced in preventing the separator from stalling, and a sink product assaying only 51% Fe was made. As the media installed was fresh, no degradation had taken place, with the result that it settled rapidly. The media was finally ground to minus 100m. Better results were obtained with the ground media and a sink grade of 58% Fe was obtained. By this time, however, the original plans had been changed, and it was decided to blend the fines from the 'direct' and 'crude' ores as flotation feed and to disregard the screen oversize. The amount of flotation concentrate produced would be sufficient for pelletizing tests.

Wet Grinding of 'Direct' Ore

A test was run on the plus 3/8 in. 'direct' shipping ore to find out if it was amenable to wet grinding to 20m. A 15-ton sample was cut out, and the plus 1/2 in. material was crushed to 3/8 in. The crushed ore was fed to a Marcy 36 in. x 61 in. rod mill driven by a 25 hp motor. The rod charge was 3000 lb and the dry feed rate was just over 2 tons/hr. The rod mill discharge was lifted by bucket elevator to a 20m vibrating screen and the oversize returned to the rod mill. A screen analysis was done on the products and the results are shown in Table 13.

TABLE 13

Results of Wet Grinding Test on 3/8 in. 'Direct' Ore

Mesh	Rod Mill Feed		Rod Mill Discharge		Screen o'size		Screen u'size	
	Wt %	% Sol Fe*	Wt %	% Sol Fe*	Wt %	% Sol Fe*	Wt %	% Sol Fe*
+3/8 in.	0.3							
-3/8 +3 m	15.3	58.10						
-3 +4	15.3	58.0						
-4 +6	14.9	57.7						
-6 +8	10.7	58.2			1.3			
-8+10	9.0	57.8			3.3			
-10+14	6.0	57.6	0.7		10.9	55.36		
-14+20	5.3	57.6	2.8	57.60	45.7	56.20	0.4	
-20+28	4.0	58.0	6.5	58.00	8.2	57.80	3.1	55.96
-28+35	3.8	57.9	15.2	57.80	5.9	57.42	7.9	56.14
-35+48	2.8	57.4	13.3	57.30	3.8	57.36	9.8	57.04
-48+65	2.4	57.6	12.5	58.00	4.5	57.48	12.8	58.10
-65+100	2.8	57.7	12.0	57.96	3.5	57.24	13.5	58.12
-100+150	1.8	58.9	7.7	59.90	2.4	57.40	10.0	57.90
-150+200	1.3	58.0	6.3	57.68	2.3	57.80	7.6	58.40
-200+325	1.2	57.1	4.2	57.60	1.9	56.60	5.9	57.88
-325	3.1	55.6	18.8	55.62	6.3	54.10	29.0	56.18
Total	100.0		100.0		100.0		100.0	

* From Internal Report MS-AC-62-959. Analyst - E. Mark.

As soon as sufficient material had been treated, various ways were tried to dewater the product. None was successful. Slimes, either present before or produced by grinding, acted on the filter cloth like clay and blinded it completely in a short period. Flocculants added to the material in laboratory tests did not help. A sample of fines from the overflow of the classifier, used at one stage to try to separate the ground material, was left to stand in a pan filter under 25 in. of vacuum for 5 hrs. Very little filtering took place. The test was stopped after about 5 tons of feed had been used. It appeared that dry grinding would be the only way to produce a product that could be handled successfully.

Pilot Plant Flotation Tests

From the results of tests done at Steep Rock, it had been decided to adopt the following requirements as closely as possible in the pilot plant testing:

1. Grinding - The flotation feed should be all minus 150m, approximately 60% minus 325m, with as little minus 5μ material as possible.

2. Desliming - Cyclone conditions required to produce suitable flotation feed had to be known. The feed should not contain more than 10% minus 9μ or 4% minus 3μ, and the cyclone overflow should have as little plus 9μ material as possible for optimum iron recovery.

3. Froth handling - Experience on this type of ore had shown that an extremely stable froth was formed which was hard to break down in launders and pump boxes. Accordingly, attention was given to the type of sprays that would be required to cause as little dilution as possible.

After several changes, the final flowsheet is shown in Figure 3, p. 25.

Test 1

Test 1 consisted mainly of trying different equipment to obtain the best results. On the first day, two rake classifiers were used to classify the rod mill discharge. It was intended to grind the ore in a Marcy 36 in. x 61 in. rod mill driven by a 25 hp motor. The rod mill discharge would be fed to No. 1 classifier, the o'flow from this going to No. 2 classifier. Both classifier sand products would be pumped to the grate discharge ball mill for grinding, and the ball mill would be in closed circuit with No. 1 classifier. However, it was found that the ore would not classify. The slimes in the ore appeared to act as a heavy liquid and kept the sands in suspension.

On the second day the classifiers were replaced by a 65m DSM screen which returned an oversize product to the ball mill. The rod mill discharge was fed straight to the ball mill. The screen undersize was pumped to a P50 cyclone, the cyclone underflow being returned for regrinding in the ball mill. This arrangement was better but it was thought that better grinding would be obtained by a ball mill alone with a larger circulating load to help prevent overgrinding.

Meanwhile, preliminary flotation results had been obtained using a bank of 12 Denver No. 7 Sub A cells. No scavenger circuit was installed, the froth being discarded as tailing. From the primary P50 cyclone, the overflow was pumped for desliming to a second P50 cyclone. The overflow from this cyclone was pumped to a bank of four M30 cyclones. The overflow from the M30 cyclones was discarded as slimes, and the underflow was combined with the underflow from the second P50 cyclone as flotation feed. Cyclone feed pressures were 20 psi. Dry rod mill feed rate was 2200 lb/hr. The results of screen tests are shown in Table 14. Altogether, 15 tons of feed were ground before the circuit was considered to be in equilibrium for sampling.

TABLE 14

Results of Screen Tests on Test 1

Mesh	Rod Mill		Ball Mill	DSM Screen		No.1 Cyclone		No.2 Cyclone		No. 3 Cyclone
	Feed	Discharge	Discharge	O'size	U'size	O'flow	U'flow	O'flow	U'flow	U'flow
+3	6.7	-	-	-	-	-	-	-	-	-
-3 +4	12.7	-	-	-	-	-	-	-	-	-
-4 +6	11.7	-	-	-	-	-	-	-	-	-
-6 +8	13.5	-	-	-	-	-	-	-	-	-
-8 +10	8.6	-	-	-	-	-	-	-	-	-
-10+14	10.1	-	-	-	-	-	-	-	-	-
-14+20	10.4	-	-	-	-	-	-	-	-	-
-20+28	7.7	-	-	-	-	-	-	-	-	-
-28+35	6.3	-	-	-	-	-	-	-	-	-
-35+48	3.8	-	-	-	-	-	-	-	-	-
-48+65	1.9	18.5	-	18.4	-	-	0.8	-	-	-
-65+100	1.7	18.4	2.2	22.0	7.8	0.7	2.4	-	-	-
-100+150	1.1	12.8	6.2	13.8	11.8	2.4	6.0	0.4	2.0	-
-150+200	0.8	8.0	12.0	7.8	12.0	3.5	10.0	2.2	5.6	-
-200+325	0.8	7.8	21.4	7.2	12.2	8.0	19.8	8.0	26.4	0.2
-325	2.2	34.5	58.2	30.8	56.2	85.4	61.0	89.4	66.0	99.8
Total	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0

Micron sizing was done on the slime product: 90% of the slimes was minus 10 μ. The flotation feed contained about 20% minus 10 μ. Approximately 40% of the rod mill feed was discarded to waste as slimes. The best grade of flotation concentrate obtained was 58.9% Fe. No accurate records of reagent consumptions were kept.

Test 2

From the experience gained in Test 1, several changes were made for Test 2. The rod mill was taken out of the circuit. Feed to the ball mill was kept at 2200 lb/hr, and 500 lb of balls was added bringing the total charge to 3000 lb. With a larger circulating load it was hoped that less overgrinding would occur. Changes were made in classification. The P50 cyclones were replaced by 3 in. Dorrclones to allow for adjustment of the feed, apex, and vortex openings. The ball mill discharge was passed over a DSM 65m screen. The screen oversize was returned to the mill. The screen undersize was pumped at 30% solids to the No. 1 Dorrclone. The overflow went to the desliming circuit, while the underflow was fed to the No. 2 Dorrclone. The second cyclone overflow also went to desliming, while the underflow was returned to the ball mill.

The desliming circuit consisted of a 3 in. Dorrclone and after many changes, a P50 cyclone and four M30 cyclones in parallel. The No. 3 Dorrclone took the overflows from Nos. 1 and 2 Dorrclones as feed. The overflow was deslimed further in the second stage, while the underflows from the three cyclones made up the flotation feed. The overflow from the P50 and M30 cyclones was discarded as slimes. Careful control of dilution water was exercised so that the cyclones were working at the optimum pressures and feed densities. A week was spent in attaining the best classification results. Once satisfactory conditions had been obtained, grinding and classification were left alone and efforts were concentrated on improving flotation procedures. The final operating conditions and cyclone settings are shown in Table 15.

TABLE 15

Final Conditions and Cyclone Settings

Product	Conditions	3 in. Dorrclones			Combined P50 and M30 Cyclones
		No. 1	No. 2	No. 3	
Feed	lb/hr	3600	1950	2360	730
Feed volume	igpm	20.0	16.1	26.3	12.6
Feed density	% solids	34	25	20	12
Overflow density	% solids	22	12	12	9
Underflow density	% solids	75	74	66	21
Feed opening	sq in.	0.19	0.125	0.375	Standard
Vortex opening	sq in.	0.44	0.79	0.60	Standard
Apex opening	sq in.	0.11	0.05	0.20	Standard
Feed pressure	psi	22	10	20	5

The P50 cyclone dimensions are: feed 0.23 sq in., vortex 0.42 sq in., and apex 0.19 sq in. The dimensions of each M30 cyclone are: feed 0.045 sq in., vortex 0.15 sq in., and apex 0.028 sq in.

In Test 1 considerable trouble had been experienced in controlling the air on the Denver cells, with the result that froth control was poor. It was decided to use a bank of six Fagergren 12 in. cells with individual drives fitted with a standard feed and discharge box. The impeller speeds were measured, and found to be the same -- 1180 rpm. No provisions were made for a scavenger circuit to clean the flotation froth. The primary concern was to get concentrate grade of 60% Fe or better and to find out the correct reagent additions. Flotation control was much better using these cells, but the froth produced was hard to control once out of the cell. Homemade sprays were made and by using compressed air and water, some success was achieved. Altogether, 26 tons of feed was treated in Test 2. Timed samples were taken and the results are shown in Table 16.

TABLE 16

Results of Test 2

Product	Weight lb/hr	Weight %	Analysis % Sol Fe	Distn % Sol Fe
B. M. feed	2258	100.0	55.2	100.0
B.M. discharge	4352	192.7	56.6	197.6
DSM o'size	970	42.9	57.3	44.5
No. 1 Dorr o'flow	1816	80.4	55.4	80.7
No. 2 Dorr o'flow	442	19.6	54.4	19.3
No. 2 Dorr spigot	1124	49.8	58.8	53.1
No. 3 Dorr spigot	1046	46.3	58.0	48.6
Slime cyclone spigot	350	15.5	57.2	16.1
Slimes to waste	862	38.2	51.0	35.3
Flotation feed	1396	61.8	57.8	64.7
Conc	895	39.6	60.9	43.7
Tail	501	22.2	52.4	21.0

The concentrate was pumped to a settling cone and filtered. Filter cake moisture was 12.9%.

The reagent consumptions in this test were calculated to be: Dextrine WW82, 2.4 lb/ton; RADA, 0.65 lb/ton, and POA, 0.65 lb/ton. The results of screen tests are shown in Table 17.

TABLE 17

Results of Screen Tests on Test 2 Products

Mesh	Ball Mill		DSM	No.1 Dorrclone		No.2 Dorrclone		No. 3 Dorrclone
	Feed	Discharge	o'size	o'flow	u'flow	o'flow	u'flow	u'flow
+3	3.4	-	-	-	-	-	-	-
-3+4	12.2	-	-	-	-	-	-	-
-4+6	13.7	-	-	-	-	-	-	-
-6+8	14.6	-	-	-	-	-	-	-
-8+10	10.0	-	-	-	-	-	-	-
-10+14	9.5	-	-	-	-	-	-	-
-14+20	9.7	-	-	-	-	-	-	-
-20+28	6.5	-	-	-	-	-	-	-
-28+35	6.6	4.5	13.6	-	-	-	-	-
-35+48	3.9	2.4	8.8	-	-	-	-	-
-48+65	2.4	4.6	13.3	-	5.4	1.2	4.8	0.7
-65+100	2.2	9.4	17.2	0.5	14.1	6.8	14.8	3.0
-100+150	1.2	12.0	12.0	1.8	25.2	14.4	26.0	7.4
-150+200	0.9	13.0	8.2	4.3	24.0	19.4	28.5	12.0
-200+325	1.0	14.2	7.6	12.2	19.2	23.2	20.2	26.2
-325	2.2	39.9	19.3	81.2	12.1	35.0	5.7	50.7
Total	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0

The DSM screen was a 12 in. wide, 60° sieve bend type. Wooden inserts were placed either side of the screen so that the width was reduced to 6.5 in.

Test 3

Using the same flowsheet as Test 2, efforts were made to improve the recovery in Test 3 by cleaning the rougher flotation froth in a scavenger circuit. The main problem encountered was to break the froth down with water sprays so that the material could be pumped to the scavenger circuit. Home-made sprays were made using a mixture of compressed air and water, but even with three of these sprays fitted over the pump box feeding the dewatering cyclones, some froth was lost. It was calculated that the amount of spray water added was 9.6 gal/min, so that from a froth containing 15% solids leaving the cells, the density was reduced to about 3% solids which was far too dilute for scavenger flotation. Unfortunately, satisfactory sprays could not be acquired in time for this test.

In order to raise the froth density, two cyclones and a thickener were used. The froth was pumped by a 3 in. x 3 in. SRL pump to two P50 cyclones in parallel. Feed pressure was 10 to 15 psi. The underflow density was raised to 9% solids and pumped to a 4 ft x 4 ft thickener. Both the cyclone and thickener overflows were discarded to tailing. The thickener underflow averaged 23% solids, and the pulp was floated in six Fagergren 12 in. cells at this density. Before the scavenger float, the pulp was conditioned for 5 minutes with Dextrine WW82 in a 12 in. x 18 in. Denver conditioner. The flowsheet for Test 3 is shown in Figure 3.

Altogether, 60 tons of ore were treated in Test 3. The averaged results are shown in Table 18.

TABLE 18

Results of Test 3

Product	Weight lb/hr	Weight %	Anal % Sol Fe	SiO_2	Insol	Distn % Sol Fe
B.M. feed	2400	100.0	57.0	-	-	100.0
B.M. discharge	4573	190.5	57.3	-	-	191.5
Screen o'size	1156	48.2	56.6	-	-	47.9
No. 1 cyclone o'flow	1557	64.9	57.2	-	-	65.1
No. 2 cyclone o'flow	843	35.1	56.6	-	-	34.9
No. 2 cyclone spigot	1017	42.3	58.8	-	-	43.6
No. 3 cyclone spigot	1340	55.8	58.3	-	-	57.1
Slime cyclone spigot	272	11.4	57.8	-	-	11.5
Slimes to waste	788	32.8	54.5	-	-	31.4
Flotation feed	1612	67.2	58.2	-	-	68.6
Flot concentrate	885	36.9	60.9	4.32	6.32	39.4
Scavenger cyclone o'flow	62	2.6	43.2	-	-	2.0
Thickener o'flow	18	0.7	40.3	-	-	0.5
Thickener u'flow	647	27.0	56.4	-	-	26.7
Scavenger conc *	470	18.2	60.8	4.54	6.60	19.4
Scavenger tail	177	8.8	47.4	-	-	7.3

The recovery was 87.9% of the iron in the flotation feed.

* A considerable amount of feed to the scavenger circuit was lost due to froth spillage on the floor from the pump box. These results are based on the total calculated froth product from flotation, assuming no losses.

The reagent consumptions are shown in Table 19.

TABLE 19

Reagent Consumptions in Test 3

Reagent	Reagent Soln Strength %	Rougher lb/ton		Scavenger lb/ton	Total lb/ton flot feed
		1	2	1	
Dextrine WW82	5	2.4	-	1.8	4.2
RADA	1	0.40	0.25	0.20	0.85
POA	1	0.40	0.25	0.20	0.85

The results of micron sizing by sedimentation are shown in Table 20.

TABLE 20

Micron Sizing Results on Test 3 Products

Microns	Desliming Cyclones		Flotation Products	
	U'flow	O'flow (to waste)	Feed	Concentrate
+33	2.1	-	46.1	59.7
-33 +24	8.3	-	14.1	16.0
-24 +15	16.6	2.1	15.7	14.4
-15 +8	28.2	16.6	10.8	5.9
-8 +5	13.3	8.3	3.3	1.6
-5	31.5	73.0	10.0	2.4
Total	100.0	100.0	100.0	100.0

During Test 3, the cell retention times, froth densities, and assays were measured with the results shown in Table 21.

TABLE 21

Measurement of Cell Conditions in Test 3

	Units	Feed	Rougher Cells						Feed	Scavenger Cells					
Cell No.			1	2	3	4	5	6		1	2	3	4	5	6
Water	igpm	1.45	-	-	1.41	-	0.70	-		-	-	-	-	-	-
Froth density	% solids	32.0	22.5	16.5	16.0	14.8	13.3	15.5	23.0	22.7	22.0	22.4	24.1	24.9	28.9
Froth analysis	% Fe	-	43.8	48.3	51.9	53.7	54.9	58.2	-	48.2	51.6	55.0	55.9	57.4	55.3
Retention time	min	-	1.33	1.67	1.67	2	2	3	-	3.33	4.0	4.67	6.0	6.5	7.0

Total retention time in the rougher cells was 11.67 min and in the scavenger cells 31.5 min. In actual practice, it is not thought necessary to have more than 15 min in the scavenger float. The pulp density of the rougher cell discharge was 42% solids and for the scavenger cell discharge 12% solids.

Altogether, 100 drums of concentrate assaying over 60% Fe were shipped for pelletizing. The total dry weight of the concentrate was about 34 tons. After the flotation concentrates had been thickened in a settling cone, they were pumped at 58% solids to a two-disc 4 ft diameter filter. No trouble was experienced in filtering and a cake over 0.5 in. thick was formed. The filter rotated at 0.4 rpm. The filtering rate was 1.8 tons/sq ft of filtering area/24 hr calculated from the production of 110 lb/min of concentrate at 12.9% moisture.

The settling rates were calculated for the scavenger flotation feed and the flotation concentrate. The settling rate for the scavenger feed from 9% to 25% solids was 11.1 ft/hr. The settling rate for the concentrate to 58% solids was 4.4 ft/hr.

CONCLUSIONS

The results obtained in this investigation show that further pilot plant work is needed to decide on the best way to concentrate this ore. As far as it went, the investigation was successful. It was desired to produce sufficient flotation concentrate for pelletizing tests, and to discover any difficulties that would be encountered in a continuous operation.

The ore slimed readily when ground with the result that about one-third was lost when desliming in cyclones at 5 to 10 microns. It is possible that grinding may not be done prior to flotation of the fines at Steep Rock, but, if the ore is to be deslimed before flotation, a considerable amount of iron will be lost. Flotation laboratory tests showed that an acceptable concentrate could be made on undeslimed flotation feed. However, this might not be practical due to higher reagent consumptions, filtering problems, lower concentrate grades, and froth handling problems.

In the pilot plant flotation tests the chief problem was froth control. Far too much water was used with the result that the rougher froth had to be thickened before scavenger flotation. With properly designed sprays, this problem should be largely overcome. It is thought that a different flotation flowsheet should be tried using 'pumper' cells to move the froth back for cleaning, eliminating a separate circuit with the additional equipment involved. The arrangement visualized is shown in Figure 4.

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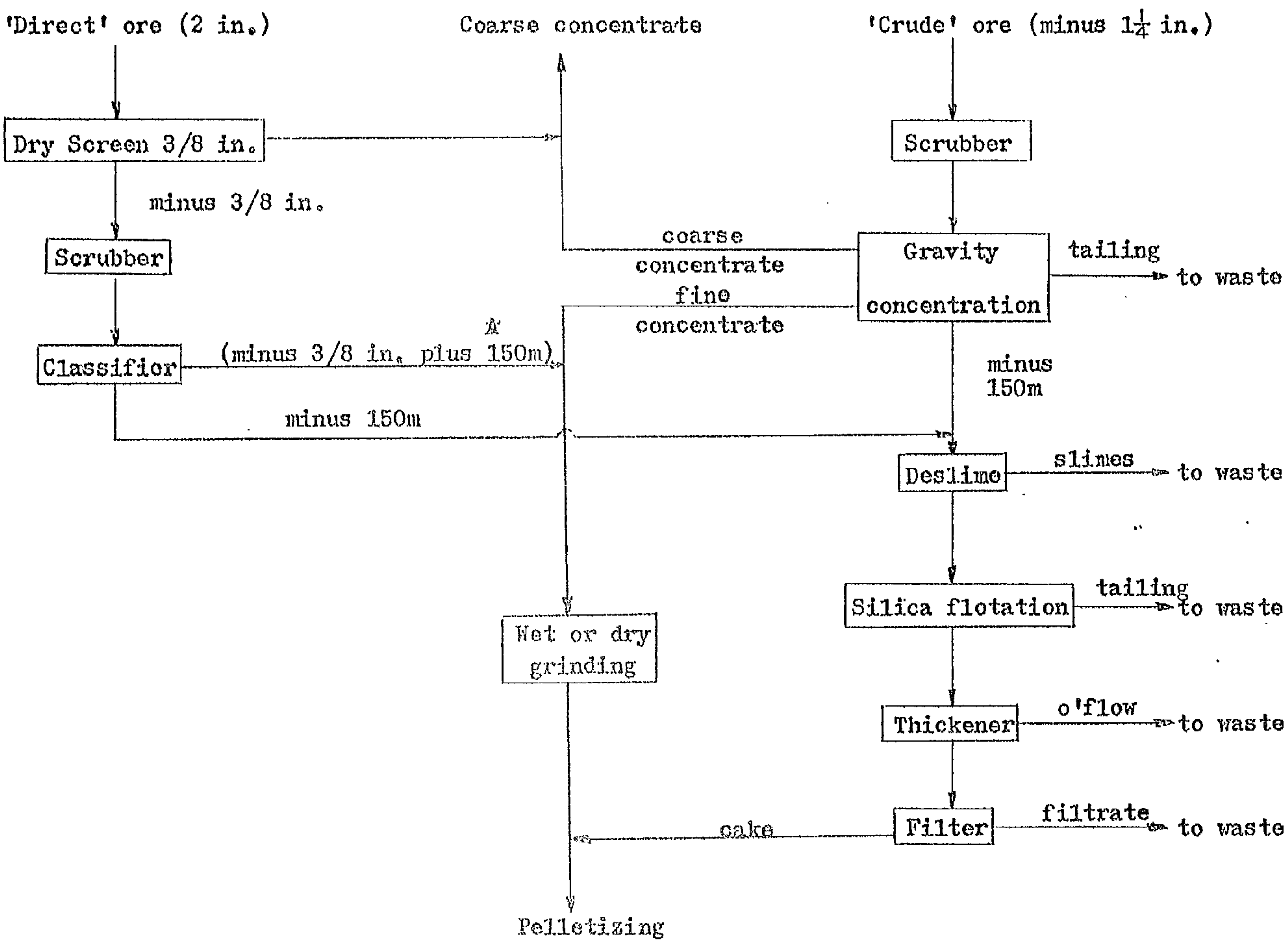


☆ This fraction could be upgraded by gravity methods.

Figure 1. - Proposed New Steep Rock Flowsheet

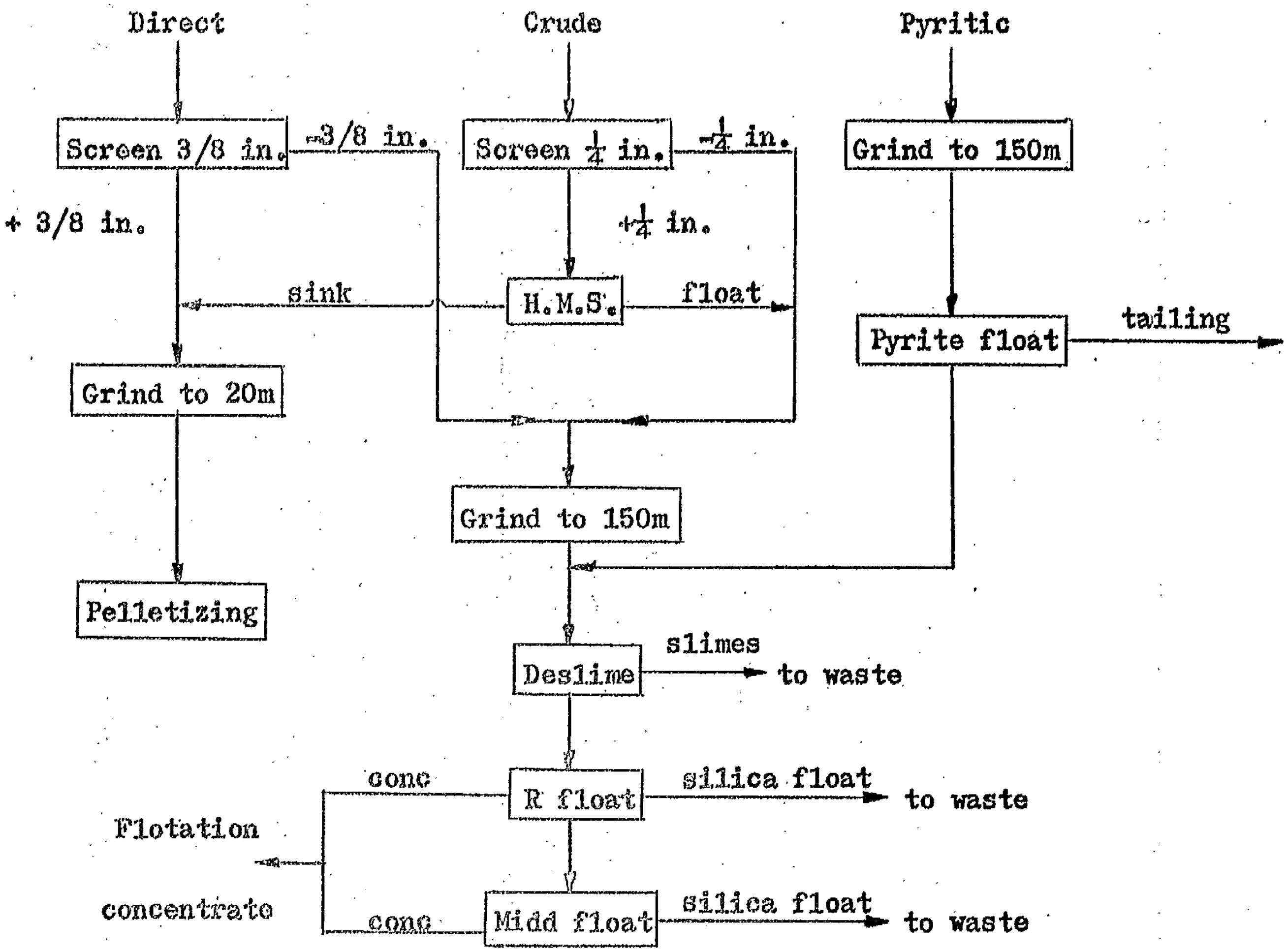
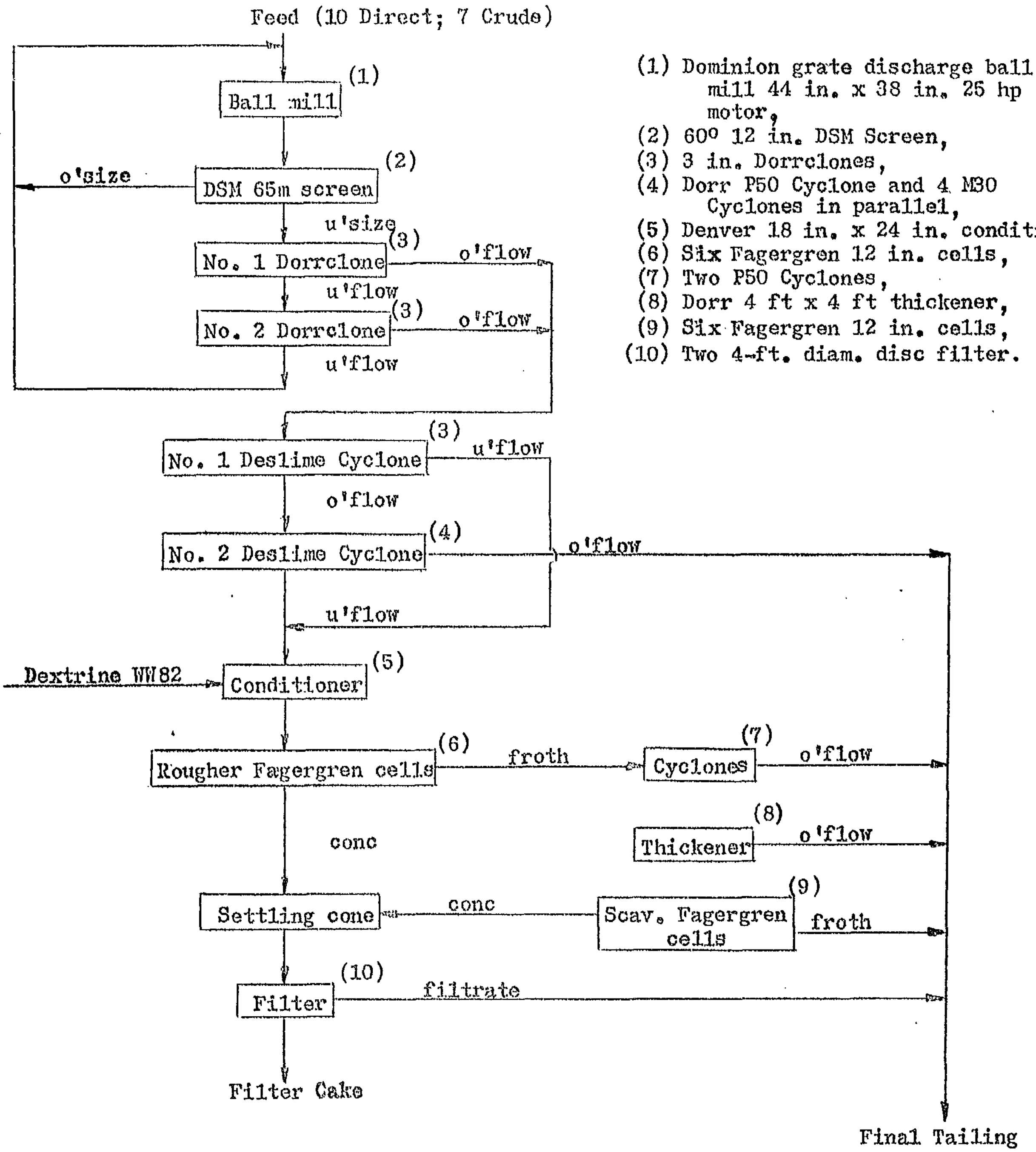


Figure 2.- Treatment Scheme Proposed for Tests at the Mines Branch



(1) Dominion grate discharge ball mill 44 in. x 38 in. 25 hp motor,
(2) 60° 12 in. DSM Screen,
(3) 3 in. Dorrclones,
(4) Dorr P50 Cyclone and 4 M30 Cyclones in parallel,
(5) Denver 18 in. x 24 in. conditioner,
(6) Six Fagergren 12 in. cells,
(7) Two P50 Cyclones,
(8) Dorr 4 ft x 4 ft thickener,
(9) Six Fagergren 12 in. cells,
(10) Two 4-ft. diam. disc filter.

Figure 3. - Flowsheet of Pilot Plant Test

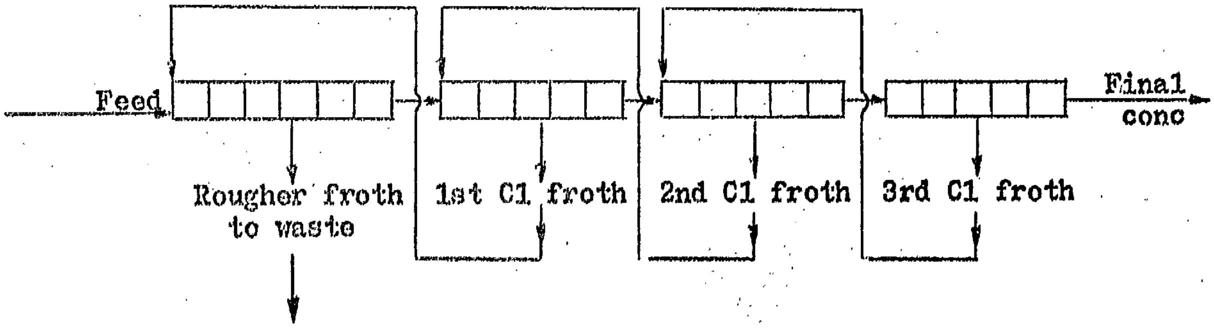


Figure 4. - Suggested Flotation Scheme