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DEPARTMENT OF MINES AND TECHNICAL SURVEYS

OTTAWA

MINES BRANCH INVESTIGATION REPORT IR 62-24

**FLOTATION OF LEAN YELLOW ORE  
FROM IRON ORE COMPANY OF CANADA,  
SCHEFFERVILLE, QUE.**

*IR 62-24 IR 62-24*

CENTRAL TECHNICAL  
FEB 27 1963  
GEOLOGICAL FILES

by

LOUIS L. SIROIS

MINERAL PROCESSING DIVISION

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Mines Branch Investigation Report IR 62-24

FLOTATION OF LEAN YELLOW ORE FROM IRON ORE COMPANY OF CANADA  
SCHEFFERVILLE, QUE.

by

Louis L. Sirois\*

SUMMARY OF RESULTS

Flotation tests were made on samples of Lean Yellow iron ores from the Iron Ore Company of Canada, Schefferville, Que. containing 51% Fe, mainly as limonite and goethite, and 14% silica.

Desliming of the ore was not practical as 60% of the material could be rejected as slimes. The flotation feed had to be conditioned at a pH of 11, at a high percent solids, to disperse the slimes and clean the silica particles which were floated with a cationic collector, Armac C.

Grades of 58% Fe and 4% silica were obtained with recoveries of 95%.

Reagent consumptions of 0.6 lb/ton of Armac C, 0.5 lb/ton of Dextrine and 3 lb/ton of NaOH were required to obtain an acceptable concentrate.

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## INTRODUCTION

### Shipment

On October 26, 1961, three samples of Lean Yellow ores were received at the Mines Branch Laboratories, Ottawa, from Mr. H.E. Neal, Research Supervisor, Ore Testing and Research Dept., Iron Ore Company of Canada, Schefferville, Que.

The three samples consisted of 100 lb of T-14758 ore, 100 lb of T-14744 ore and 800 lb of T-14700 ore, all minus 4 mesh material.

### Object of Investigation

The object of the present investigation was to develop a suitable flotation procedure for these Lean Yellow ores consisting mainly of quartz, goethite and limonite, to produce an iron concentrate with iron-silica ratio of at least 9:1.

Flotation of the silica portion of the ore by a cationic process was aimed at since quartz was the lesser constituent and the major portion of the gangue.

### Sample Analysis

All routine analyses in connection with this investigation were done by Mr. Jules Desjardins of the Iron Ore Company of Canada, on loan to the Mineral Processing Division for the duration of the investigation.

Credit is given to the Analytical Chemistry Subdivision, Mineral Sciences Division, Mines Branch, Ottawa, for 98 special chemical analyses related to this study. (Internal Reports MS-61-823, MS-AC-62-262 and MS-AC-62-643). (See Acknowledgements).

## CHARACTERISTICS OF THE ORE

### Texture and Mineralogy

The greatest portion of this investigation was conducted on the T-14700 sample. The following mineralogical description of this ore is taken from the report of microscopic examination by Mr. I.S. Zajac of the Iron Ore Company of Canada:

This report is based on the microscopic examination, in natural state and in polished section, of the minus 4 mesh sample of this T-14700 sample. The mineralogical composition, given below, is expressed in percent by weight. It was derived by calculation from the percent by volume composition obtained with a point-counter, assuming the specific gravity of quartz as 2.7, of goethite as 4.3, of limonite as 4.0 and of hematite as 5.2.

The T-14700 sample of Lean Yellow ore is yellow, fine-grained, fairly soft, friable and porous. Microscopic examination shows it to be composed mainly of goethite, limonite and quartz, with small amounts of hematite and manganese oxides:

Quartz	12%
Goethite	52%
Limonite	33%
Hematite	1%
Mn Oxides	Less than 1%

Chemical analysis of this Lean Yellow ore shows 0.7% of  $Al_2O_3$  which suggests the presence of aluminosilicates, most probably about 2% of kaolinite. It is doubtful if any other silicates remain in the ore.

Quartz is the only gangue mineral which can be distinguished under the microscope. It occurs dominantly as roughly equidimensional grains irregularly distributed throughout the ore or as clusters of grains finely cemented by goethite. In some of the coarse particles, the clusters of quartz grains are in the form of stringers and poorly defined bands. These goethite cemented quartz clusters, with the exception of botryoidal goethite, are the hardest and the least porous parts of the ore. Ten to thirty percent of the quartz in the ore is loosely bound or entirely free and commonly intimately mixed with limonite.

The grain size of the quartz varies from 0.3 mm to 0.002 mm but most of it is within 0.03 mm to 0.005 mm.

Goethite occurs as hard, dense to fairly porous aggregates of very fine-grained haphazardly intergrown grains. A small amount of goethite derived from vugs and open fractures is botryoidal, comparatively hard and invariably free of silica. A few feathery, partly radiating or net-like clusters are also present.

The coarse particles in the minus 4 mesh material are partially to completely surrounded by limonite coatings some of which contain 30% to 60% of quartz.

Limonite is the yellow powdery material which at first glance appears to make up most of the ore. Most of it is undoubtedly extremely fine-grained goethite with or without adsorbed water. This material is very fine-grained, soft, porous and has a tendency to adhere to other particles, particularly those with rough surfaces. The adhesive tendency of limonite is illustrated by the abundance of limonitic coatings which are so extensive that it is difficult to distinguish the other minerals in the ore with the unaided eye.

Hematite is present as very small disseminated particles in goethite. Practically all of it is in the form of martite formed by alteration of pre-existing magnetite. The small amount of hematite is typical of most yellow ore.

The manganese oxides pyrolusite and/or psilomelane occur as fine-grained, dense fracture fillings in the ore.

Screen and Chemical Analysis

The following screen and chemical analysis, which was supplied by the Iron Ore Company of Canada Laboratories, was obtained from the head sample of the T-14700 ore, as received.

Table 1

Screen and Chemical Analysis of T-14700 Head Sample

Mesh	% Wt	% Cum	% Fe	% SiO <sub>2</sub>
+ 8	17.1	17.1	56.57	7.58
14	19.8	36.9	55.60	7.54
28	12.7	49.6	55.11	8.51
48	8.5	58.1	54.30	9.93
65	4.1	62.2	53.65	11.49
100	3.3	65.5	53.65	12.12
150	2.5	68.0	53.17	11.51
200	2.2	70.2	52.27	14.35
325	3.0	73.2	50.25	17.35
- 325	<u>26.8</u>	100.0	<u>43.60</u>	<u>28.72</u>
	100.0		51.94	14.40

Mn 0.4 %  
 Al<sub>2</sub>O<sub>3</sub> 0.7 %  
 L.O.I. 10.39 %

OUTLINE OF INVESTIGATION

From the characteristics of the ore, it can be seen that the major problem arising out of this investigation is the large amount of slimes present.

Fine slime particles are detrimental to flotation by consuming reagents and by coating the larger mineral particles so that the possibility of contact between a bubble and the mineral surface is reduced.

Three different grinds were tried to produce as little slimes as possible and still reduce the size of the quartz particles. There was complete liberation of quartz from iron minerals at minus 100 mesh but it was found that the quartz particles were too coarse to float at this size and should be reduced to minus 150 mesh for effective flotation.

Floating at a high pH was the only method found effective to obtain complete dispersion of the slimes and produce clean quartz surfaces capable of reacting with cationic collectors. It was impossible to deslime the ore without large iron losses.

Conditioning of the ore, prior to flotation, was done in the Agitair cell itself for periods ranging from 16 to 17 minutes at pulp densities varying from 30% to 60% solids. Conditioning speed was 1500 rpm.

Three products were normally assayed, the iron concentrate from the rougher float, the silica concentrate and the iron middling from the scavenger float. Since the iron middling was usually acceptable as part of the final concentrate, it was combined with the iron concentrate from the rougher float to give a final grade and recovery.

Many different reagents were tried to obtain the best results and included alkalis to raise the pH, collectors for silica, and depressants to inhibit iron losses. Although staging of reagents appeared to be beneficial to flotation at the beginning, single additions proved just as successful once a process had been worked out.

Ottawa water was used for most grinding and flotation tests. Other tests were performed with distilled and also with Schefferville water to determine their effect on flotation. In no case was the water detrimental to the flotation.

Effort was directed to decrease the collector consumption. The solution from a standard rougher float was collected and used as make-up water for a successive rougher float. Several tests were performed in this manner to try and recover any collector left in the solutions. Assuming that the iron slimes consumed the collector, the slimes present in the solution after conditioning with an alkali and a depressant were removed and the remainder of the ore then treated with the collector. A third method consisted in raising the normal density of the flotation pulp from 30% to 40% solids. Only the density control was successful.

Due to extreme difficulties in filtering the different products of flotation, caused by the high degree of dispersion of the pulp, tests were carried out with different reagents and filter aids to promote a more rapid filtration.

#### DETAILS OF INVESTIGATION

##### Grinding

The three different grinds performed on the T-14700 sample of Lean Yellow ore, with their infrasizer and chemical analyses are reported in Table 2.



Table 2

Infrasizer and Chemical Analyses for 12 Minute, 18 Minute  
and Minus 150-Mesh Stage Grinds

	12 Minute Grind					18 Minute Grind					- 150 M Stage Grind				
	% Wt	% Fe	Fe Dist'n	% Insol	Insol Dist'n	% Wt	% Fe	Fe Dist'n	% Insol	Insol Dist'n	% Wt	% Fe	Fe Dist'n	% Insol	Insol Dist'n
100 m	5.58	55.78	6.28	7.60	2.37	1.37	56.02	1.51	7.73	0.68					
150	7.84	55.18	8.75	7.80	3.45	6.38	55.86	7.04	8.29	3.19	7.27	55.05	7.85	8.75	4.09
200	8.51	55.08	9.47	8.28	3.96	6.63	55.61	7.26	8.65	3.49	11.40	55.13	12.32	8.65	6.32
56 ←	5.62	56.17	6.38	6.16	1.98	4.52	57.30	5.10	5.69	1.59	7.92	56.66	8.81	6.34	3.19
40	9.37	54.02	10.22	9.64	5.09	11.44	55.21	12.43	9.18	6.44	10.09	54.72	10.83	9.74	6.26
28	7.39	51.23	7.66	14.92	6.22	9.12	52.72	9.46	13.50	7.54	8.84	52.23	9.07	13.75	7.78
20	5.90	45.82	5.45	23.92	7.97	8.27	48.61	7.91	19.89	10.06	7.60	48.03	7.16	20.35	9.90
14	6.36	43.27	5.55	28.48	10.23	7.90	45.88	7.13	24.44	11.83	7.86	45.29	6.99	24.92	12.52
10	6.83	42.83	5.92	28.64	11.08	6.98	44.99	6.18	25.78	11.04	7.76	44.49	6.77	26.09	12.90
-10	36.60	46.41	34.32	23.04	47.65	37.39	48.93	35.98	19.25	44.14	31.26	49.24	30.20	18.56	37.04
Total	100.00	49.51	100.00	17.69	100.00	100.00	50.83	100.00	16.31	100.00	100.00	50.96	100.00	15.66	100.00

A sedimentation analysis done on the minus 10 micron portion of the 12 minute grind sample indicated that 9.4% by weight of this sample was minus 2 microns.

These analyses point out the extreme fineness of the ore with the resulting problems of slimes, high reagent consumption and filtration.

There was complete liberation of the quartz from the iron minerals at minus 100 mesh, but it was found that the quartz particles were too coarse to float at this size and should be reduced to minus 150 mesh for effective flotation.

The ore was ground wet in a 1000 g porcelain mill with steel balls for periods of 12 and 18 minutes for the first two grinds. Each batch was then split with a wet cutter to provide 500 g samples which were floated in an Agitair cell.

The third grind was done by stage-grinding 25 lb of the ore to minus 150 mesh. This was accomplished by using a 2000 gm steel mill, grinding for 6 minute periods and screening wet on a 150 mesh Sweco screen. This material was again split with a wet cutter into 500 g batches. All samples were kept under water to prevent any surface alteration or agglomeration.

This stage grinding had the beneficial effects of lowering the amount of slimes by over 5% from the 12 minute grind and over 6% from the 18 minute grind while decreasing the amount of coarse quartz particles.

Table 3 reports three of the better floats, one for each grind, and establishes the relationship between grind and grade of iron concentrate. The reduction in size of the quartz particles permitted a better grade of concentrate to be produced.

Table 3

Relationship of Grinds and Grades of Iron Concentrates

Grind	Test No.	Fe Concentrate		
		% Fe	% Insol	% Recovery
12 minutes	FY-21	56.37	6.33	96.83
18 minutes	FY-32	57.57	4.72	97.61
-150 m stage	FY-36	58.15	3.62	95.84

These tests were performed under the same conditions and using the same reagents. Details are contained in Appendix I.

Desliming of the flotation feed, after grinding, was attempted. Up to 60% by weight of the material could be rejected. The reject products assayed between 46% and 52% Fe and 25% and 15% Insol. Since no high iron or low silica portions could be obtained, within reasonable limits, the project was abandoned and the samples floated with their slime content.

#### pH

Since the slimes had to be present during flotation, they had to be dispersed. The only method found to accomplish this was to float at a high pH. It was necessary to operate at a pH of 11 or over.

Below a pH of 11, the slimes stabilized the froth and made it rubbery and unmanageable. The grade became erratic due to slime coating on the quartz particles.

In tests FY-38, FY-39 and FY-40, no cleaner float could be performed as the froth was too rubbery. The froth was completely removed during the rougher float to determine if grade could be obtained. Figure 1 illustrates the variation of recovery with pH and Table 4 reports the grades obtained. Details of these tests are reported in Appendix I.

Table 4  
Variation of Grade and Recovery with pH

Test No.	pH	Fe Concentrate		
		% Fe	% Insol	% Recovery
FY-41	2.0	-	-	-
FY-39	6.9	56.26	6.50	79.40
FY-40	8.3	57.06	5.14	82.10
FY-38	10.2	56.26	7.06	86.20
FY-36	11.2	58.15	3.62	95.84
FY-51	11.4	58.03	3.88	95.31
FY-49	11.7	57.65	4.30	95.69

Flotation became exceedingly difficult as the pH was lowered and became almost impossible at pH 2. At pH's higher than 11, recovery remained fairly constant at 95% with a good grade product.

A discussion of this problem can be found in "Discussion of Results" on page 22.

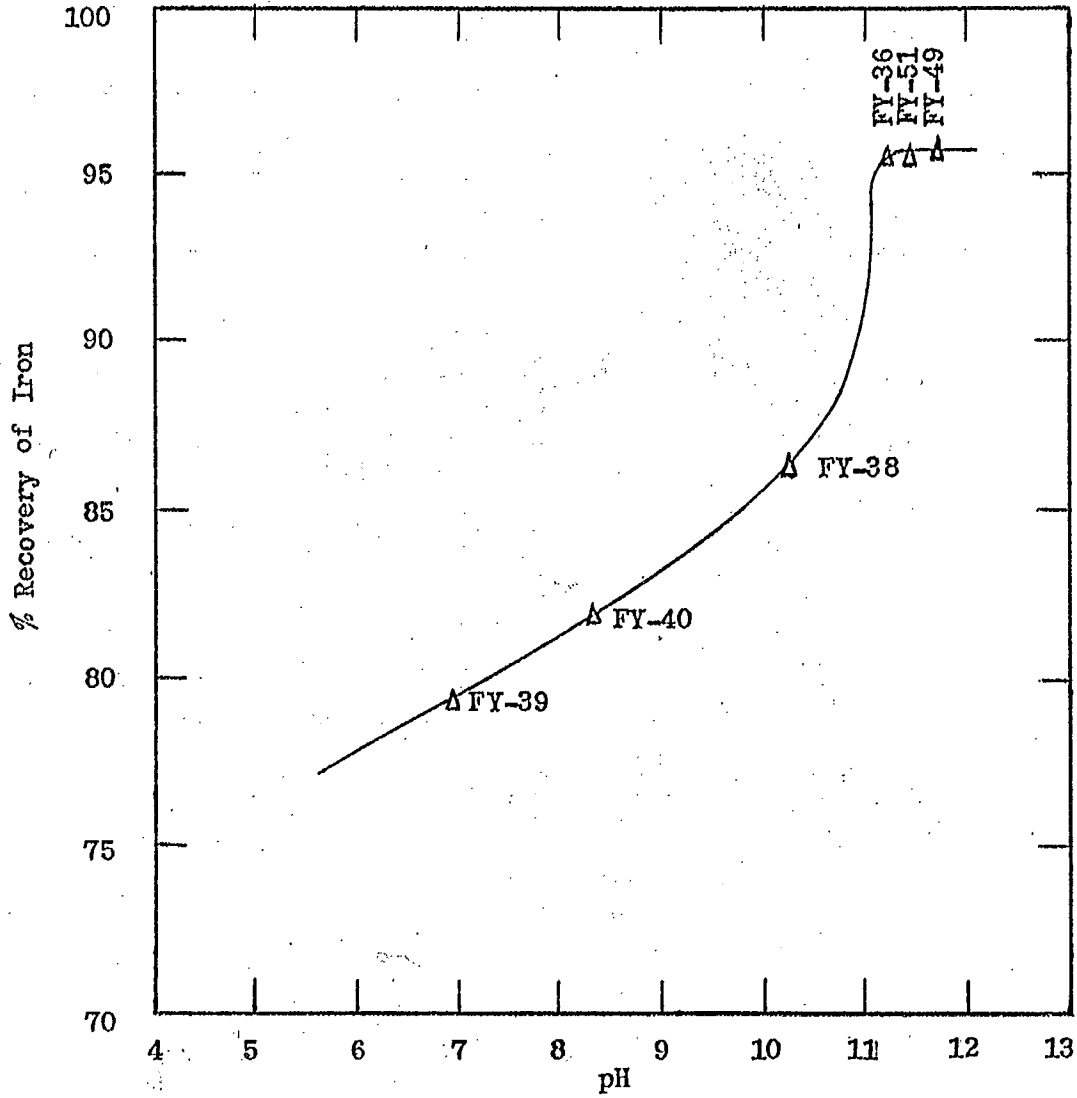


Figure 1 - pH vs Recovery of Iron

### Conditioning

Conditioning of the ore, prior to flotation, was done in the Agitair cell itself for periods ranging from 16 to 17 minutes at pulp densities varying from 30% to 60% solids. Conditioning speed was 1500 rpm.

The ore was conditioned at 55% to 60% solids with an alkali for the first 10 minutes, at a pH above 11, to disperse the slimes. Next, an iron depressant was added and conditioning continued for another 5 minutes. The silica collector was then introduced into the cell for a further conditioning of 1 to 2 minutes. The density of the pulp was then lowered to 30% with make-up water, the air turned on and the silica floated.

Reagent consumption decreased when the pulp was conditioned at a high per cent solids. The maximum density which could be obtained in the Agitair cell and still permit rotation of the impeller without splashing was between 55% and 60% depending on the cell used.

### Cationic Collectors

The following cationic collectors were tried:

Armac C - a primary coco amine acetate - Armour  
Armac T - a primary tallow amine acetate - Armour  
Armac 18-D - a distilled stearylamine acetate - Armour  
Armoflote P - mono-amines dissolved in solvent - Armour  
Armacflote P - mono-amines dissolved in solvent - Armour  
DP-243 - laurylamine hydrochloride - Dupont Chemical Co.  
Duomeen T - tallow N - alkyltrimethylene diamine - Armour  
Amine 220 - an imadazoline - Carbide and Carbon Chemical Co.  
Aeromine 3037 - partial acetate salt - American Cyanamide  
Amine D Acetate - Acetate salt of Pine resin amine - Hercules Powder Company  
Rosin Amine D Acetate - primary amine acetate - Hercules Powder Company

These collectors were either used singly or in conjunction with each other.

Armac C, Armoflote P, Armacflote P and DP 243 were found to be the best collectors giving both acceptable grades and high recoveries. DP 243 was dropped because it is no longer produced on a commercial basis.

In an effort to lower the consumption of these high priced collectors, cheaper ones were used in conjunction with them but no combination appeared to produce desirable results.

Table 5 illustrates the grades and recoveries which can be expected from some of the collectors used either alone or in conjunction with each other.

Table 5

Collectors, Grades and Recoveries

Collectors	Test No.	Fe Concentrate		
		% Fe	% Insol	% Recovery
Armac C	FY-51	58.03	3.88	95.31
DP 243	FY-16	56.82	5.90	93.80
Rosin Amine D Acetate and Armac C	FY-27	54.95	8.61	98.66
Armac 18-D	FY-15	53.79	10.51	97.95
Rosin Amine D Acetate and Duomeen T	FY-28	52.00	13.85	98.69
Armac T	FY-17	51.11	15.42	99.34
Rosin Amine D Acetate and Amine 220	FY-18	51.00	15.46	98.38
Amine D Acetate	FY-19	50.92	15.84	91.94
Armacflote P	FY-75	57.50	4.13	96.07
Armoflote P	FY-74	57.70	4.45	96.69

Since Armac C most consistently produced an acceptable grade, most of the test work was done with this collector. Figure 2 illustrates the effects of Armac C consumption on grade and recovery. Table 6 reproduces the data in table form. Details of these tests are tabulated in Appendix I.

Table 6

Effects of Armac C on Grades and Recoveries

Test No.	Consumption lb Armac C/ton	Fe Concentrate		
		% Fe	% Insol	% Recovery
FY-36	1.0	58.15	3.62	95.84
FY-51	0.8	58.03	3.88	95.31
FY-47	0.7	58.01	4.50	95.42
FY-38	0.5	56.26	7.06	86.20

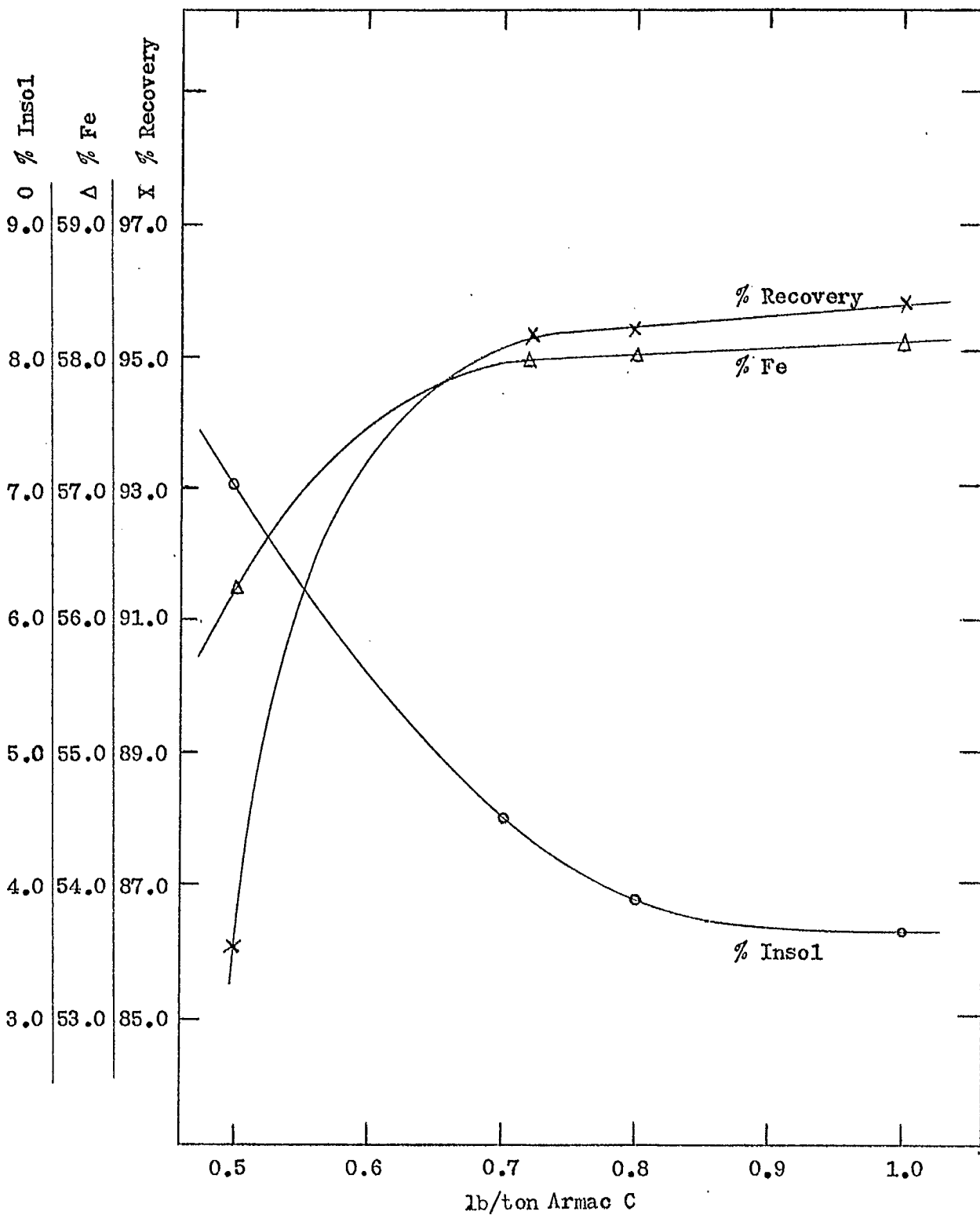


Figure 2 - Effects of Armac C on Grades and Recoveries.

Depressants

A few iron depressants were tried and it was found that they did not have a great effect on the silica flotation. Table 7 lists the four which were tried with the grind used on the sample. (The grind is listed to place the grade in its proper perspective and Table 3 should be referred to).

Yellow Dextrine and Dextrine 8072 were the depressants used most consistently and the amount needed to obtain the most efficient flotation, can be observed on Figure 3 and Table 8. Details of the tests can be found in Appendix I.

Table 7

Depressants, Grades and Recoveries

Depressant	Test No.	Grind	Fe Concentrate		
			% Fe	% Insol	% Recovery
Yellow Dextrine	FY-23	18 min.	57.08	5.26	96.50
Dextrine 8072	FY-51	-150 m	58.03	3.88	95.31
Lignosol DXD	FY-52	-150 m	57.76	4.53	95.11
Pearl Starch	FY-16	12 min.	56.82	5.90	93.80

Table 8

Effects of Dextrine on Recoveries and Grades

Test No.	Consumption lb/ton Dextrine	Fe Concentrate		
		% Fe	% Insol	% Recovery
FY-46	0.0	57.68	4.73	96.02
FY-51	0.5	58.03	3.88	95.31
FY-48	0.9	58.01	3.94	95.40
FY-36	6.0	58.15	3.62	95.84



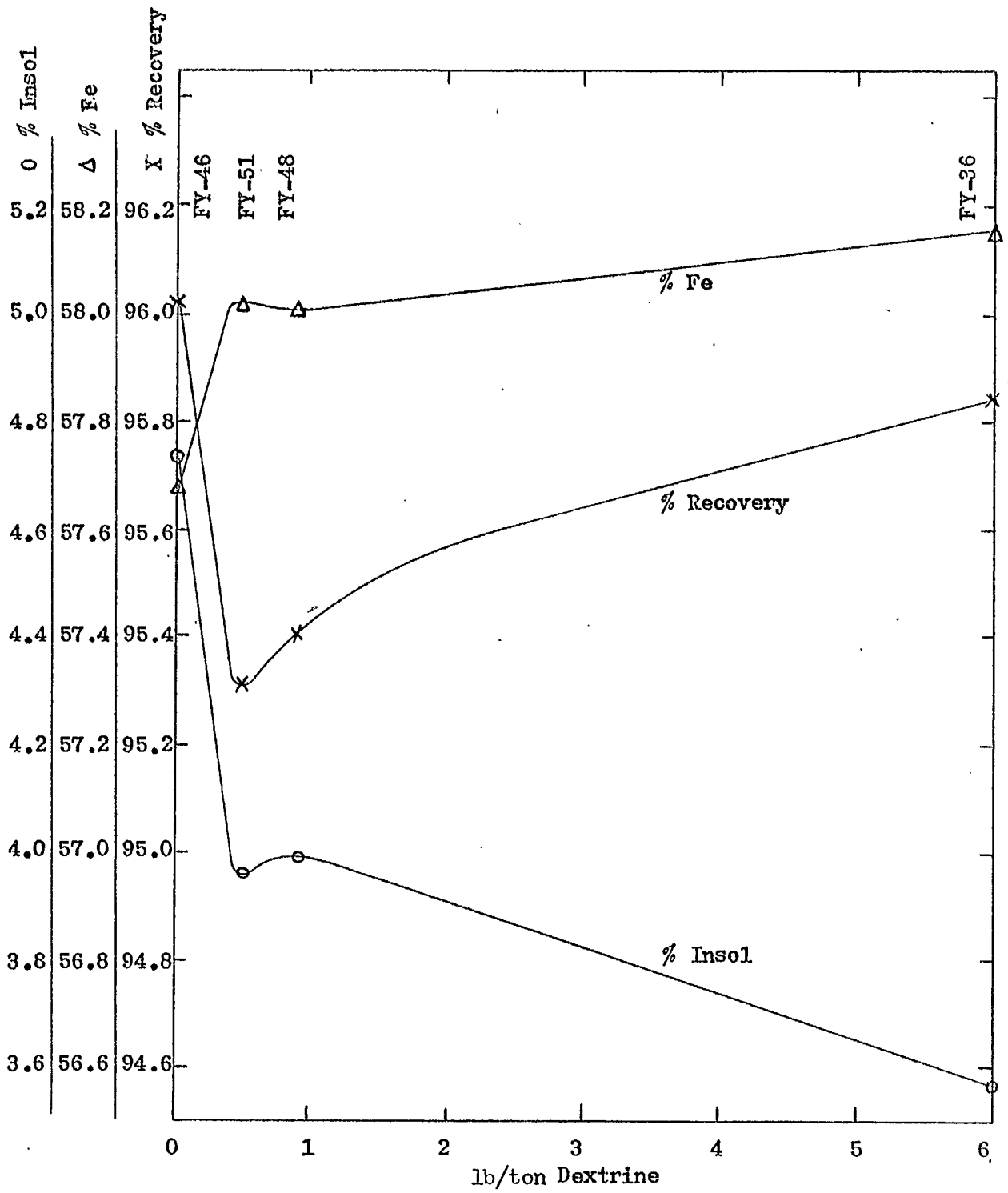


Figure 3 - Effects of Dextrine on Grades and Recoveries.

A 0.5 lb/ton addition of Dextrine in the conditioning stage seems to be the most beneficial amount to be used as the increase in grade and recovery obtained by using more Dextrine does not seem to warrant the expense.

Guartec was also tried, instead of Dextrine, in the conditioning stage of the flotation to flocculate the slimes. It was tried first, after the pH was increased to 11 and the pulp conditioned for 10 minutes. In a second test, it was added to the pulp, at the natural pH of the ore, for a 5 minute conditioning period; the pH was then increased to 11 for further conditioning. Neither test showed any improvement over the use of Dextrine. The two tests were FY-65 and FY-66 and are listed in Table 9 and Appendix I.

Table 9

Effects of Guartec as a Depressant

Test No.	lb/ton Guartec	Fe Concentrate		
		% Fe	% Insol	% Recovery
FY-65	0.2	56.45	5.54	96.16
FY-66	0.2	55.81	7.71	97.95

Water

Ottawa water and water sent to the Mineral Processing Laboratories from Schefferville by the Iron Ore Company of Canada analyzed as follows:

Table 10

Analysis of Ottawa and Schefferville Water

	Ottawa	Schefferville
Avg pH	8.3	7.5
Ca	15 ppm	8.3 ppm
Mg	2.3 ppm	5.9 ppm
Al	2.0 ppm	0.0 ppm
Total Hardness	47	45
Conductance	111	96.6

The Ottawa water was used for the major portion of the investigation. Testing was done with the Schefferville water to determine if the local water had any effects on the cationic flotation process developed with the Ottawa water. Two tests were performed with Schefferville water, one at

28°C and the other at 12°C, to see if temperature also had an effect on the silica flotation. Results are reported in Table 11.

Table 11  
Comparison of Effects of Ottawa, Schefferville and  
Distilled Water on Grades and Recoveries of Silica Float

Test No.	Water	Temp °C	Ca Content ppm	Fe Concentrate		
				% Fe	% Insol	% Recovery
FY-51	Ottawa	21°	15	58.03	3.88	95.31
FY-67	Schefferville	28°	8	57.33	4.55	96.41
FY-69	Schefferville	12°	8	58.10	4.00	95.71
FY-54	Distilled	-	0	58.01	4.19	96.23
FY-55	Distilled	-	20	58.05	4.15	96.18
FY-56	Distilled	-	60	58.09	4.17	96.22

Other tests were performed with distilled water to observe if the metallic ions contained in ordinary tap water had any effects on collector adsorption.

Further tests were done with distilled water in which Ca ions were introduced. These were attempted as Ca is adsorbed on silica, changing its surface charge from negative to positive and therefore obstructing the cationic float.

As observed in Table 11, flotation with Schefferville and distilled water gave similar results to those obtained with the Ottawa tap water. The low temperature of the water did not seem to harm the process.

For the calcium additions shown also in Table 11, the grade and recovery remain constant showing no adsorption due to the Ca.

In no case, therefore, was the water detrimental to the flotation.

#### Efforts to Decrease Collector Consumption

According to the literature, silica can ordinarily be floated with 0.1 to 0.2 lb/ton of a cationic collector. The silica flotation of the T-14700 sample of Lean Yellow ore required 0.8 lb/ton. The high consumption of the cationic collector, Armac C in this instance, could be the result of three factors: (1) adsorption on the iron slimes, (2) large residual amounts left in the float solutions, (3) unusually large adsorption on the silica due to its extreme fineness and its resulting large surface area. A series of tests was instituted to consider these factors.

Slime Rejection. In this test, 2000 g of ore was conditioned at 60% solids at a pH of 11.7 for 10 minutes. The cell was then filled with 4000 cc of water and conditioned for another 5 minutes before the pulp was allowed to settle for 30 minutes; 4000 cc of the supernatant solution was siphoned. The Dextrine and the Armac C were added to the pulp which was conditioned for 2 minutes at a pH of 11.5. Make-up water was then returned to the cell and the flotation was performed at a pH of 11.2. Table 12 indicates the results obtained. Only 0.4 lb/ton of Armac C was added in this test as compared to the usual 0.8 lb/ton.

Table 12  
Results of Desliming Test FY-64

Product	% Wt	Fe Concentrate		
		% Fe	% Insol	% Recovery
Conc.	64.28	57.22	5.58	72.23
Midds	7.84	38.74	35.29	3.55
Tails	4.67	6.78	88.07	1.04
Slimes	<u>23.21</u>	<u>50.85</u>	<u>14.40</u>	<u>23.18</u>
Total	100.00	50.92	15.48	100.00

Over 23% of the ore was removed as slimes giving assays similar to those of the ore; 50.85% Fe for the slimes as compared to 50.92% Fe for the ore. These slimes cannot be rejected in the tails or incorporated in the iron concentrate.

The iron concentrate itself contained 5.58% Insol, which meant that there was a deficiency of collector and that, since some of the slimes had been rejected, they could not adsorb the greater portion of the collector.

The low Fe assay for the middlings from the scavenger float was due, most likely to the low pulp density in the cell during the flotation.

Recirculation of Flotation Solutions. After a rougher float in which 0.88 lb/ton of Armac C was used, 4000 cc of solution was siphoned from the cell and used as make-up water for the second rougher float; 4000 cc of solution from the second rougher float was also used for make-up water for the third rougher float and so on for the fourth rougher float. Clear water was added to the cell for all the scavenger floats.

For the second, third and fourth floats, 0.64 lb/ton of Armac C was used instead of the original 0.88 lb/ton. The solutions siphoned from the rougher floats contained 4% solids at a pH of 11.4.

Table 13  
Grades and Recoveries from Floats Done with  
Recirculated Solutions

Test No.	Float	Fe Concentrate		
		% Fe	% Insol	% Recovery
FY-60	Standard	57.70	4.64	91.23
FY-61	Cycle 1	57.57	4.65	94.47
FY-62	Cycle 2	57.04	5.31	95.48
FY-63	Cycle 3	56.85	5.62	97.20

Results obtained from the four floats, illustrated in Figure 4, are tabulated in Table 13 and indicate a decrease in the silica floated. This would seem to indicate that most of the Armac C is not present in the make-up solutions causing a lack of Armac during the flotation and that the fineness of the silica is of major importance. The minus 150 m grind reduces 37% of the silica in the ore to minus 10 microns (Table 2). This results in a large surface area, a consequently large adsorption of the collector on the silica particles and therefore a high consumption of Armac C.

The slimes in the make-up solutions could also be the reason for this decrease in grade. Appendix I shows the details of these tests.

Flotation at Higher % Solids. Since recirculation of solution and desliming did not reduce the amount of collector needed to remove the silica particles, flotation at a higher % solids was attempted. All tests had been done at 30% solids and two tests FY-71 and FY-72 were done at 40% solids, all other conditions remaining the same. Table 14 illustrates the potentials of floating at a higher % solids.

Table 14  
Collector Consumption for Floats at 30% and 40% Solids

Test No.	Grind	Armac C lb/ton	% Solids	Fe Concentrate		
				% Fe	% Insol	% Recovery
FY-71	18 min	0.6	40	57.96	4.36	94.43
FY-78	-150 m	0.6	30	57.10	4.75	97.16
FY-72	18 min	0.8	40	57.97	4.11	94.49
FY-31	18 min	0.8	30	57.19	5.25	98.00

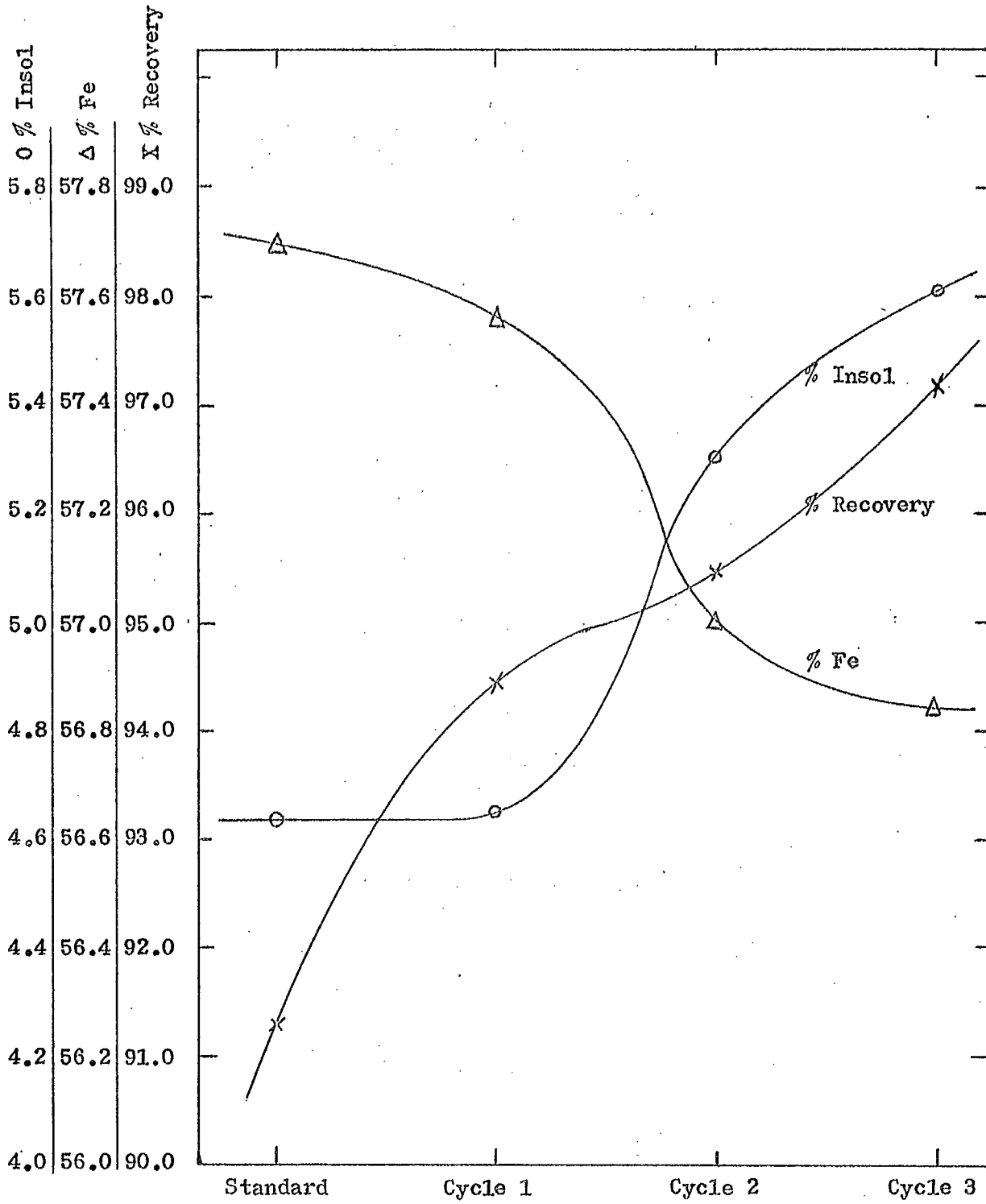


Figure 4 - Curves Representing Grades and Recoveries from Floats Done with Recirculated Solutions.

Three of these tests were performed on 18 minute ground material and results would be expected to be higher had they been performed on minus 150 m material.

Floating at a higher per cent solids, therefore, seems to reduce the amount of collector required although there is a decrease in recovery. Another beneficial side effect is a slight drop in the amounts of NaOH needed to raise the pH above 11.

### Filtration

Because of the high pH needed to disperse the slimes to perform effective flotation, filtration is hindered. Following is a report on laboratory filtering tests performed by Mr. D.E. Picket, Head, Ferrous Section, Mineral Processing Division, Mines Branch.

These laboratory filtering tests were performed on the iron concentrate of a T-14700 sample of Lean Yellow ore floated with Armac C at a pH of 11.5.

1. (a) The concentrate did not settle to give a clear supernatant solution given any length of time, even up to four days.
  - (b) In this condition, no filter cake could be picked up.
  - (c) After adding 1.1 lb/ton of CaO to a pulp sample, by top loading the test disk a 1/16 inch cake was produced in 13 minutes. After a 5 minute drying time, the cake contained 15% moisture. This was the driest cake produced with this sample, but filtering time and capacity are impractical.
  - (d) Addition of 2 lb/ton CaO plus 2 lb/ton H<sub>2</sub>SO<sub>4</sub> produced a thickened pulp with about 50% H<sub>2</sub>O. At 23 in. vacuum only 1/16 in. cake could be picked up in 1 minute and after 3 minute drying, it contained 21.2% moisture.
  - (e) Adding 2 lb/ton CaO and 0.1 lb/ton Separan, resulted in rapid settling to 60% solids. This thickened pulp filtered rapidly so that a 3/16 in. cake could be picked up in 30 seconds. However the cake cracked badly and the lowest moisture obtained was 25.5%.
  - (f) Slapping of a cake formed as in (e) closed the cracks and consolidated the cake. Best moisture obtained was 18.5% with a 3/16 in. cake. Average of 6 tests was 19.5% moisture.
  - (g) Top loading the disk with this thickened pulp produced a cake 3/8 in. thick with 17.4% moisture. This cake was slapped to close the cracks. Total time was 9 minutes.
2. By method (f) approximately 500 g of filter cake was produced with a moisture content (drier) of 19.5% H<sub>2</sub>O. This was tested by "moisture teller" which gave a result of 16.5% H<sub>2</sub>O. The filter cake was pelletized to 3/8 in. pellets in a pelletizing tire. Some drying occurred in the process so that the final pellets contained 13.0% H<sub>2</sub>O by "moisture teller".

Flotation of T-14744 and T-14758 Samples of Lean Yellow Ore

Two other samples of Lean Yellow Ore were also floated to determine their flotation characteristics and compare them with the T-14700 sample.

The T-14744 sample had a head assay of 53.84% Fe and 12.91% Insol. Two tests, FY-42 and FY-43 were performed on this sample and the details are reported in Appendix I.

Table 15

Grades and Recoveries of T-14744 Sample

Test No.	Armac C lb/ton	Fe Concentrate		
		% Fe	% Insol	% Recovery
FY-42	1.0	58.75	3.15	95.92
FY-43	0.8	58.38	3.59	96.81

The T-14758 sample assayed 50.94% Fe and 15.60% Insol. Two tests, FY-44 and FY-45 were also performed on this sample and the details are shown in Appendix I.

Table 16

Grades and Recoveries of T-14758 Sample

Test No.	Armac C lb/ton	Fe Concentrate		
		% Fe	% Insol	% Recovery
FY-44	1.0	57.59	4.64	97.20
FY-45	0.7	57.90	4.74	96.74

These two samples of Lean Yellow ore react to flotation in the same manner as the T-14700 sample and can, therefore, be treated with it.

Flotation of T-14770 Sample of Lean Blue Ore

The process developed for the Lean Yellow ore was tried on a sample of Lean Blue ore with a head assay of 55.45% Fe and 15.47% Insol and gave the following results.



Table 17  
Grade and Recovery of T-14770 Sample

Test No.	Armac C lb/ton	Fe Concentrate		
		% Fe	% Insol	% Recovery
FB-9	0.7	64.14	2.10	94.40

RESULTS

Although there is complete liberation of the constituents of the Lean Yellow ore at minus 100 mesh, it was found necessary to grind to minus 150 mesh to obtain acceptable grades. Coarser than 150 mesh, the quartz particles were too coarse and could not be properly floated. Stage grinding was found effective to reduce the amount of slimes produced.

The large amount of slimes present made flotation extremely difficult without the use of a dispersant. Raising the pH above 11 thoroughly dispersed the slimes and produced clean silica surfaces. For any pH below 11, the slimes stabilized the froth and rendered it very difficult to manage. Acceptable results could still be obtained, but they were erratic with a large drop in recovery. An average of 4 lb/ton of sodium hydroxide was required to obtain this pH of 11 when floating at 30% solids. For an increase in density to 40% solids of the flotation pulp, the amount of sodium hydroxide needed can be cut down to 3 lb/ton.

Of all the cationic collectors which were tried, Armac C, Armac-flote P and Armoflote P were the ones which produced the most desirable results. For a collector consumption of 0.6 lb/ton of Armac C an iron concentrate of 58% Fe, 4.3% Insol, with a 95% recovery can be expected. This grade and recovery for a consumption of 0.6 lb/ton of collector was obtained only when floating at 40% solids.

Over 37% of the silica is in the minus 10 micron range and its extreme fineness and large surface area seem responsible for this large consumption of collector.

Of the different methods tried to lower the consumption of reagents, recirculation of solution or desliming or floating at a higher per cent solids, increasing the density from 30% to 40% solids was the best way of obtaining beneficial results. Collector consumption was reduced from 0.8 lb/ton to 0.6 lb/ton while sodium hydroxide was lowered from 4 lb/ton to 3 lb/ton. A drop in recovery can be expected.

A few iron depressants were used and it was found that they did not have a great effect on the silica flotation. Dextrine 8072 and Yellow Dextrine were used most consistently; consumptions of 0.5 lb/ton were normal.

This silica float is rapid and a period of one minute conditioning with the collector is all that is necessary. A longer conditioning time is required with the alkali and the depressant to allow for scrubbing of the silica particles. A period of 15 minutes was found ample.

Tests were done with Ottawa and Schefferville water and also with distilled water containing none or large amounts of calcium. Water had no influence on the grade or recovery of the iron concentrate. Low temperature water also had no effect on the floats.

Because of the high dispersion needed for effective flotation, filtration was hindered. Addition of 2 lb/ton CaO and 0.1 lb/ton Separan to the rougher iron concentrate produced rapid settling to 60% solids. This thickened pulp then filtered rapidly. Slapping the filter cake reduced the moisture to 16.5% H<sub>2</sub>O and it was pelletized to 3/8 in. pellets containing 13% moisture.

All pertinent tests are tabulated in Appendix I. The complete analysis of the iron concentrate of test FY-35 gave the following results:

Sol. Fe	57.53%
SiO <sub>2</sub>	3.68%
Al <sub>2</sub> O <sub>3</sub>	0.75%
CaO	0.14%
MgO	0.67%
TiO <sub>2</sub>	0.10%
S	0.009%
P	0.010%
Mn	0.63%
H <sub>2</sub> O	11.11%
Na <sub>2</sub> O	0.04%
K <sub>2</sub> O	0.01%
C	0.29%
CO <sub>2</sub>	0.27%

#### DISCUSSION OF RESULTS

Colloids will flocculate even though the particles carry the same charge, but when two dispersed colloids of opposite charge are mixed, flocculation ensues even more rapidly. Analogous phenomena seem responsible for slime coating on minerals in flotation systems.

Whether a suspension is flocculated or dispersed depends upon whether the repulsive forces or the attractive forces between particles are dominant. Since Van der Waals' attractive forces are always acting between particles, these forces must be counteracted by the repulsive forces arising from the interaction of similarly charged double layers if a stable suspension is to be maintained. By reducing the thickness of the electrical double layer, particles can approach each other sufficiently close for the attractive forces to predominate, and the suspension is then flocculated.

However, in suspensions containing two different solids which are charged oppositely to each other, the attractive forces will be electrostatic in addition to Van der Waals' forces and flocculation will be greatest for systems involving oppositely charged particles.

An essential criterion for this type of flocculation is that the solids must be different materials since the sign of charge on the solid surface is determined by the concentration of potential-determining ions in solution, and the potential-determining ions are unique for each solid. Thus every goethite particle in a particular suspension carries the same surface charge but a quartz or a limonite particle in the same suspension may be charged oppositely to the goethite.

Because of the long-range nature of attractive electrostatic forces, slime coatings should be heaviest under conditions wherein the slime and floatable mineral are oppositely charged.

Lighter slime coatings may result when the shorter-range Van der Waals' forces are able to overcome the repulsive forces of double layer interaction, such as the coating of goethite with limonite slimes.

Slime coatings are absent when the slime and mineral particles are charged alike and are surrounded by thick double layers. Since the properties of the electrical double layers of these oxides depend on the pH of the solution, pH will be the most important variable. Because the zero point of charge for silica, iron oxide and alumina occurs at pH 3.7, 8.0 and 9.5 respectively, the relative charge on the slime and minerals can be changed over wide ranges.

Silica, iron oxides and alumina are negatively charged at pH values greater than 3.7, 8 and 9.5 respectively and are positively charged at pH values lower than these values. The slime coating would be expected to be heaviest under conditions in which the slime is uncharged or is charged oppositely to the minerals. Thus the densest coating of iron oxide slime on quartz should occur in solutions near neutral pH and the coating density should decrease as the pH becomes acid or basic.

The results obtained from the multiple tests performed during this investigation bear out this reasoning.

No flotation could be obtained at the very low pH and this may be due, amongst many reasons, to the fact that the collector may break down at those low pH's, that the isoelectric points shift in the presence of reagents or that the isoelectric point of quartz is not 3.7 but 1 as advocated by certain authors.

One interesting question to settle would be to determine if the isoelectric points of the different minerals vary in the presence of reagents and if so how much and which way.

The extreme fineness of the silica seems to account, to a certain extent, for the large consumption of collector.

### CONCLUSIONS

Even though the Lean Yellow ores contain a large amount of slimes which cannot be discarded for easier concentration, these ores can be scrubbed and the fine material dispersed to present clean surfaces for collector adsorption.

The types and amounts of iron depressants are not critical. Scrubbing at a high pH of 11 or over was the best method found to prepare the surfaces for an effective flotation.

Experience has shown that although Armac C appears to be one of the best collectors for floating silica in batch operations, cheaper collectors may be substituted in a continuous process and therefore cut the cost of this expensive reagent.

The process as developed is very simple. Over 75% by weight of the ore is produced as an iron concentrate in the rougher flotation. Only one scavenging operation is required to recover an additional 8% by weight of the iron mineral which was entrained with the silica froth. There is no recirculation of middling products for retreatment, so the operation would be easy to control. An overall recovery of 95% can be expected with a concentrate containing about 4% silica and 58% iron.

This means an iron concentrate with an iron-silica ratio of 14.5:1 which is higher than the minimum 9:1 ratio specified by the Iron Ore Company of Canada.

This process could be tried on pilot-plant scale to determine its true possibilities.

### ACKNOWLEDGEMENTS

The complete analysis of the iron concentrate of test FY-35 was done by Mr. F.W. Brethour, Mr. D.J. Charette, Miss E.M. Penner, Mr. J. Hole, Miss E. Kranck. (Internal Report MS-AC-62-262).

The control analyses on 16 samples to check the routine method were done by Mr. D.J. Charette. (Internal Report MS-61-823).

Routine analyses on 18 other samples were performed by Mr. J. Hole. (Internal Report MS-AC-62-693).

APPENDIX I

TEST NO.	COLLECTOR	CONS. LB/T	ACTIVATOR	CONS. LB/T	pH	DEPRESSANT	CONS. LB/T	CONDITIONING		GRIND	Fe CONCENTRATE			SAMPLE	REMARKS
								% SOLIDS	TIME		% Fe	% INSOL	% RECOVERY		
FY-15	Armec 18-D	1.2	NaOH	2.0	12.0	Yellow Dextrine	2.0	30	17 min	12 min	53.79	10.51	97.95	T-14700	
-16	DF-2L3	1.2	NaOH	2.0	12.1	Pearl Starch	3.2	30	17 "	12 "	56.82	5.90	93.80	"	
-17	Armec-T	6.0	NaOH	8.0	11.7	Yellow Dextrine	7.0	30	17 "	12 "	51.11	15.42	99.34	"	
-18	Rosin Amine D AC	1.2													
	Amine 220	2.0	NaOH	2.0	12.1	" "	7.0	30	17 "	12 "	51.00	15.46	98.38	"	
-19	Amine D Acetate	1.2	NaOH	2.0	11.9	" "	5.0	30	17 "	12 "	50.92	15.84	91.94	"	
-21	Armec C	2.0	NaOH	2.0	11.7	" "	6.5	30	17 "	12 "	54.37	6.33	96.83	"	
-23	Armec C	2.0	NaOH	2.0	11.7	" "	6.5	30	17 "	18 min	57.08	5.26	96.50	"	
-27	Rosin Amine D AC	0.6													
	Armec C	0.6	NaOH	4.0	11.8	" "	7.0	50	17 "	18 "	54.95	2.61	98.66	"	
-28	Rosin Amine D AC	0.6													
	Buomeen T	0.2	NaOH	6.0	11.9	" "	6.8	50	17 "	18 "	52.00	13.85	98.69	"	
-32	Armec C	1.1	NaOH	6.0	11.9	Dextrine 8072	5.7	50	17 "	18 "	57.57	4.72	97.61	"	
-36	Armec C	1.0	NaOH	3.5	11.2	" "	6.0	55	17 "	-150 mesh	58.15	3.62	95.84	"	
-38	Armec C	0.5	NaOH	2.0	10.2	" "	5.0	55	17 "	" "	56.26	7.06	96.20	"	
-39	Armec C	0.5	NaOH	-	6.9	" "	5.0	55	17 "	" "	56.26	6.50	79.40	"	
-40	Armec C	0.5	NaOH	0.4	8.3	" "	2.0	55	17 "	" "	57.06	5.14	82.10	"	
-41	Armec C	1.0	H <sub>2</sub> SO <sub>4</sub>	4.0	2.0	" "	5.0	55	17 "	" "	-	-	-	"	
-42	Armec C	1.0	NaOH	3.1	12.3	Yellow Dextrine	6.5	55	17 "	18 min	58.75	3.15	95.92	T-14744	
-43	Armec C	0.8	NaOH	3.0	11.6	" "	6.2	55	17 "	" "	58.38	3.59	96.81	T-14744	
-44	Armec C	1.0	NaOH	4.0	11.9	" "	6.8	55	17 "	" "	57.59	4.64	97.20	T-14758	
-45	Armec C	0.7	NaOH	3.5	11.5	" "	6.2	55	17 "	" "	57.90	4.74	96.74	T-14758	
-46	Armec C	1.0	NaOH	4.0	11.4	-	-	55	17 "	-150 mesh	57.62	4.73	96.02	T-14700	
-47	Armec C	0.7	NaOH	3.6	11.6	Dextrine 8072	0.9	55	16 "	" "	58.01	4.50	95.42	"	
-48	Armec C	0.7	NaOH	3.6	11.1	" "	0.9	55	16 "	" "	58.01	3.94	95.40	"	
-49	Armec C	0.7	NaOH	4.4	11.7	" "	0.5	55	16 "	" "	57.65	4.30	95.69	"	
-51	Armec C	0.8	NaOH	4.0	11.4	" "	0.5	55	16 "	" "	58.03	3.88	95.31	"	
-52	Armec C	0.8	NaOH	4.0	11.4	Lignosol DXD	0.5	55	16 "	" "	57.76	4.53	95.11	"	
-54	Armec C	0.9	NaOH	4.0	11.8	Yellow Dextrine	0.5	55	16 "	18 min	58.01	4.19	96.23	"	Distilled water
-55	Armec C	0.9	NaOH	4.0	11.8	" "	0.5	55	16 "	" "	58.05	4.15	96.18	"	Distilled water + 20 ppm Ca
-56	Armec C	0.9	NaOH	4.0	11.9	" "	0.5	55	16 "	" "	58.09	4.17	96.22	"	Distilled water + 50 ppm Ca
-60	Armec C	0.8	NaOH	4.0	11.9	" "	0.5	55	17 "	-150 mesh	57.70	4.64	91.23	"	
-61	Armec C	0.6	NaOH	3.5	11.7	" "	0.5	55	17 "	" "	57.57	4.65	94.47	"	Recirculated solution
-62	Armec C	0.6	NaOH	3.5	11.7	" "	0.5	55	17 "	" "	57.04	5.31	95.48	"	"
-63	Armec C	0.6	NaOH	3.5	11.8	" "	0.5	55	17 "	" "	56.85	5.62	97.20	"	"
-64	Armec C	0.4	NaOH	3.5	11.7	" "	0.5	55	17 "	" "	57.22	5.58	72.23	"	Slime rejection
-65	Armec C	0.7	NaOH	4.0	11.8	Quartec	0.2	55	17 "	" "	56.45	5.54	96.16	"	
-66	Armec C	0.7	NaOH	4.0	11.7	Quartec	0.2	55	17 "	" "	55.21	7.71	97.95	"	
-67	Armec C	0.8	NaOH	4.0	11.8	Yellow Dextrine	0.5	55	16 "	" "	57.33	4.55	96.41	"	Schefferville water 28°C
-69	Armec C	0.8	NaOH	4.0	11.6	" "	0.5	55	16 "	" "	58.10	4.00	95.71	"	Schefferville water 12°C
-71	Armec C	0.6	NaOH	3.0	11.7	" "	0.5	55	16 "	18 min	57.96	4.36	94.43	"	Float at 40% solids
-72	Armec C	0.8	NaOH	3.0	11.8	" "	0.5	55	16 "	" "	57.97	4.11	94.49	"	Float at 40% solids
-73	Armec C	0.3	NaOH	4.0	11.5	" "	0.5	55	16 "	-150 mesh	57.85	3.56	92.93	"	
-74	Armoflote P	0.8	NaOH	4.0	11.6	" "	0.5	55	16 "	" "	57.70	4.45	96.65	"	
-75	Armoflote P	0.8	NaOH	4.0	12.0	" "	0.5	55	16 "	" "	57.50	4.13	96.07	"	
-76	Armoflote P	0.6	NaOH	4.0	11.9	" "	0.5	55	16 "	" "	57.23	5.13	97.85	"	
-77	Armoflote P	0.6	NaOH	4.0	11.8	" "	0.5	55	16 "	" "	56.78	5.77	97.59	"	
-78	Armec C	0.6	NaOH	4.0	11.6	" "	0.5	55	16 "	" "	57.10	4.75	97.16	"	
FB-9	Armec C	0.7	NaOH	2.8	12.0	" "	0.5	55	17 "	-100 mesh	64.14	2.10	94.40	T-14770	Lean Blue ore