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Beneficiation of Western Mesabi Range Oxidized Taconite

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By A. F. Colombo, H. D. Jacobs, and D. M. Hopstock



UNITED STATES DEPARTMENT OF THE INTERIOR

Cecil D. Andrus, Secretary

BUREAU OF MINES

Roger A. Markle, Director

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BENEFICIATION OF WESTERN MESABI RANGE OXIDIZED TACONITE

by

A. F. Colombo,¹ H. D. Jacobs,² and D. M. Hopstock²

ABSTRACT

In an effort to maintain an adequate supply of iron to meet national economic and strategic needs, the Bureau of Mines, U.S. Department of the Interior, is conducting research on methods for beneficiating the oxidized taconite resources located on the western Mesabi range in Minnesota, thus expanding domestic iron ore reserves. The selective flocculation and cationic flotation process was investigated in a pilot plant using an ore feed rate of 410 kg/hr (900 lb/hr) and 88 percent recycled water. Lime and a cationic flocculant were used to clarify the waste slurries treated in the water reclamation circuit. An oxidized taconite assaying approximately 36 percent Fe yielded concentrates averaging 63.5 percent Fe and 4.3 percent SiO₂ and having an average iron recovery of 84.4 percent. The estimated reagent cost per tonne of concentrate was approximately \$1.10. In an attempt to recover more of the iron lost in the flotation tailing, wet high-intensity magnetic (WHIM) separation was incorporated in the pilot plant circuit. The flotation concentrate grades obtained from beneficiation circuits with and without WHIM scavenging were similar, but when WHIM separation was used the overall iron recovery was increased by 2 to 9.5 percent, depending on the iron assay of the flotation tailing.

INTRODUCTION

With regard to domestic iron ore supply, the United States is facing two major problems. The more immediate problem is uncertainty in the availability of fuels required to maintain production of iron oxide pellets, whereas the long-range problem is a stagnant reserve base. The need to develop domestic resources is implicit in forecasts covering the period 1974 to 2000. Domestic production over this period is projected to increase from 50 to 70 million short tons of iron-in-ore per year, while primary demand is estimated as advancing from approximately 80 to 130 million short tons per year (3, 5).³

¹Research supervisor.

²Metallurgist.

All authors are with the Twin Cities Metallurgy Research Center, Bureau of Mines, Twin Cities, Minn.

³Underlined numbers in parentheses refer to items in the list of references at the end of this report.

These predictions are based on the utilization of Minnesota and Michigan magnetic taconites and of some Michigan oxidized taconites after our direct-shipping ores are depleted. The widening gap between domestic production and demand will add to an unfavorable balance of trade and intensify the worldwide competition for iron ores.

Although present domestic expansion is primarily concerned with materials amenable to low-intensity magnetic separation, there is a large amount of oxidized materials that do not respond to this method of concentration. The western Mesabi range contains about 10 billion long tons (lt) of oxidized materials assaying 30 to 40 percent Fe and offers the best prospect for overcoming the deficiency between domestic requirements and production at a cost near current levels. However, more information is needed on estimated tonnages, mineralogical associations, and the extent to which current beneficiation technology can economically extract the iron oxides from the oxidized materials.

As a result of research at the Bureau's Twin Cities Metallurgy Research Center, a process for beneficiating oxidized taconites by selective flocculation and flotation was developed (1). Research performed in cooperation with the Cleveland-Cliffs Iron Co. has led to the commercial application of the process to the Tilden ore body in Michigan. The Tilden mine is currently producing 4 million long tons of high-grade pellets annually (6) and is being expanded to 8 million long tons (4). Despite successful application of the selective flocculation and flotation process to the Tilden ore body, the process has not been utilized on the oxidized taconites that occur in the western Mesabi range.

In 1973, the Bureau of Mines started a major research effort to assure an adequate supply of iron to meet the needs of the Nation through augmenting the domestic iron ore reserve base. A 24-mile-long segment of the western Mesabi range between Keewatin and Grand Rapids, Minn., was divided into three blocks of materials. Three bulk samples, one from each block, were obtained, and their beneficiation characteristics are currently being evaluated by the selective flocculation and flotation process. The bulk sample discussed in this paper represents 1/2 to 1 billion long tons of crude material and is located in the middle of the 24-mile length of iron-bearing material being evaluated.

SAMPLE DESCRIPTION

A 600-tonne (t)⁴ (590-lt) bulk sample was obtained for pilot plant testing. The sample contained hematite, goethite, and trace amounts of magnetite, iron silicates, and iron carbonates. The hematite-to-goethite ratio was about 3 to 1. Quartz was the major gangue mineral. Liberation of the iron oxides from the gangue occurred at approximately 90 percent minus 44 micrometers (325 mesh). Table 1 shows a partial chemical analysis of the bulk sample.

⁴Unless otherwise specified, the unit of weight used in this paper is the metric tonne or 2,205 pounds.

TABLE 1. - Bulk sample analysis,¹ percent

Fe	Fe ²⁺	SiO ₂	Al ₂ O ₃	TiO ₂	CaO	MgO	Mn	S	P	LOI, ² °C	
										400	1,000
36.1	0.5	44.5	0.64	0.26	0.41	0.26	0.08	<0.01	0.013	1.6	2.4

¹The K₂O and Na₂O contents of the bulk sample were less than 0.05 percent.

²LOI is loss on ignition.

SELECTIVE FLOCCULATION AND CATIONIC FLOTATION

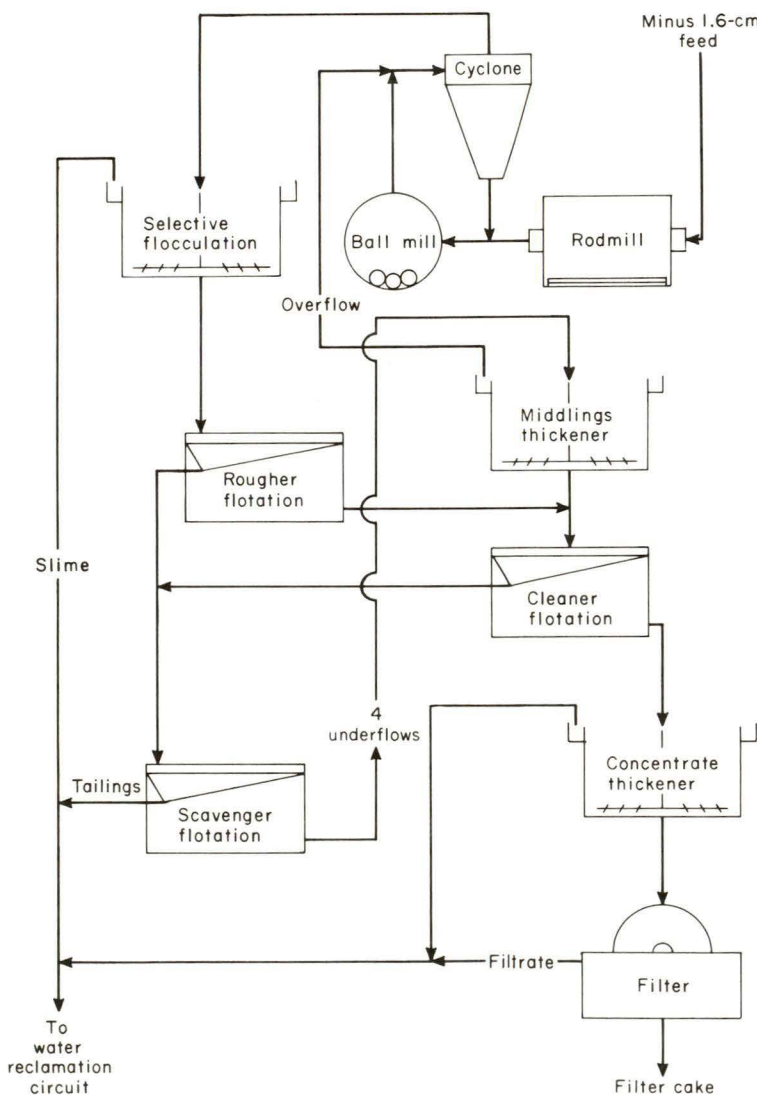
Pilot Plant Flowsheet and Equipment

FIGURE 1. - Selective flocculation and cationic flotation flowsheet.

Figure 1 is a general flowsheet of the unit operations used in the pilot plant research. About 410 kg/hr (900 lb/hr) of minus 1.6-centimeter (5/8-inch) ore were fed to a 60- by 120-centimeter (2- by 4-foot) rodmill containing a 712-kilogram (1,570-pound) rod charge ranging from 2.5 to 6.4 centimeters (1 to 2-1/2 inches) in diameter. Sodium hydroxide and sodium silicate were added to the rodmill feed to disperse the pulp and regulate the pH. The pulp, at about 64 percent solids, was ground to 42.6 percent minus 37 micrometers (400 mesh). The power consumption was 8.7 kw/hr/t (8.8 kw/hr/lt), and the rod wear was 925 grams per tonne (g/t) (2.07 lb/lt). Table 2 shows the screen analyses of the rodmill discharge and other products in the grinding circuit. The rodmill discharge was pumped to a 90- by 90-centimeter (3- by 3-foot) overflow-type ball mill which contained a seasoned charge of 912 kilograms (2,010 pounds) of balls ranging from 1.6 to 3.2 centimeters (5/8 to 1-1/4 inches) in diameter,

and was ground to 54.8 percent minus 37 micrometers. Ball wear averaged 2.2 kg/t (4.94 lb/lt), and the power consumption was 16.6 kwhr/t (16.9 kwhr/lt). The ball mill discharge was diluted with reclaimed water and middlings thickener overflow and then classified in a 7.6-centimeter (3-inch) hydrocyclone. The cyclone underflow, averaging 44.3 weight-percent (calculated on an incoming feed basis) and 44.9 percent minus 37 micrometers, was routed to the ball mill for regrinding. The cyclone overflow, averaging 90.9 percent minus 37 micrometers, was conditioned with the selective flocculant (causticized pearl starch) and then deslimed in a 90- by 90-centimeter (3- by 3-foot) thickener. The thickener overflow or slime product, which was regulated at pH 11.0, was routed to the water reclamation section. The rise rate of the slime in the thickener averaged 4.25 cm/min (1.7 in/min). The underflow was pumped to a two-cell No. 20 Fagergren⁵ rougher flotation unit, which had a total volume of 135 liters (4.8 ft³). Cationic collector, a mixture of 8- and 10-carbon chain alkyl oxypropylamine (MG-98A), was added to the rougher feedbox. The rougher froth was scavenged in four flotation steps consisting of two four-cell, one three-cell, and one two-cell No. 18 Fagergren units. The volume per cell was 42.5 liters (1.5 ft³). The fourth froth was the final tailing. Causticized pearl starch was added to the first scavenger froth to depress the iron oxides. The four scavenger underflows were thickened in a 150- by 90-centimeter middlings thickener because excessive spray water was required to keep the froths moving in the scavenging circuit. The thickener overflow was used to dilute the hydrocyclone feed. The underflow was combined with the rougher concentrate and cleaned in a 285-liter (10-ft³) bank of three No. 28 Fagergren flotation cells. A small amount of cationic collector was added to the cleaner cells. The cleaner underflow was the final concentrate, which was thickened and filtered. The froth was routed to the first scavenger flotation step.

TABLE 2. - Average screen analyses of grinding circuit products

Sieve size ¹		Cumulative weight-percent retained			
Micrometer	Mesh	Rodmill discharge	Ball mill discharge	Cyclone underflow	Cyclone overflow
297	48	2.1	-	-	-
210	65	6.5	1.0	1.3	-
149	100	16.8	2.8	3.4	-
105	150	28.3	6.2	7.5	1.7
74	200	43.2	19.3	22.6	2.4
53	270	50.9	32.5	38.8	3.8
44	325	52.6	35.7	44.2	7.2
37	400	57.4	45.2	55.1	9.1

¹ Tyler standard screen sieves.

Metallurgical Results

The data from a 5-day continuous pilot plant campaign is discussed. The pilot plant streams were sampled for chemical analysis twice during every 8-hour shift. The first set of samples (three cuts over a 1/2-hour period)

⁵ Reference to specific brand names is made for identification only and does not imply endorsement by the Bureau of Mines.

was taken 4 hours before the second set of samples (five cuts over a 1-hour period). The pilot plant streams were usually sampled at 2 and 6 hours into the shift. The first sample set was a check on the second set, which was used to determine the metallurgical response obtained during the 8-hour shift period. Timed-weight samples for material balance calculations were taken between the two sampling periods. The metallurgical data obtained from the three- and five-cut sets of samples were similar. The average metallurgical results obtained over seven consecutive 8-hour shifts are shown in table 3. Table 4 gives the reagent suite, function, addition point, and dosages used. Concentrates averaging 48.0 percent of the feed weight, 63.5 percent Fe, and 4.3 percent SiO₂ with an accompanying iron recovery of 84.4 percent were obtained. The slime product averaged 19.8 percent Fe and 12.0 weight-percent. The flotation tailing assayed 8.0 percent Fe and contained 9.1 percent of the iron in the feed. Table 5 shows the partial chemical analysis of the iron oxide concentrate. The average Blaine surface area of the concentrates was 2,850 cm²/g. The concentrates gave filter cake moistures ranging from 12 to 14 percent.

TABLE 3. - Selective flocculation and flotation metallurgical results,¹ percent

Product	Analysis			Fe distribution
	Weight	Fe	SiO ₂	
Feed.....	100.0	36.1	44.5	100.0
Concentrate.....	48.0	63.5	4.3	84.4
Slime-tailing.....	52.0	10.7	-	15.6
Slime.....	12.0	19.8	-	6.5
Flotation tailing.....	40.0	8.0	-	9.1

¹Average data for samples taken over seven consecutive 8-hour shifts.

TABLE 4. - Reagent suite

Reagent	Function	Addition point	Addition	
			Grams/t of feed	Lb/1t of feed
SELECTIVE FLOCCULATION AND FLOTATION				
Sodium hydroxide..	Dispersant-pH modifier	Rodmill.....	750	1.68
Sodium silicate...	Dispersant.....do.....	765	1.72
C-165 pearl starch ¹	Selective flocculant..	Conditioner before desliming thickener.	250	.56
Do. ¹	Depressant.....	First scavenger flotation froth.	125	.28
Arosurf MG-98A ² ...	Collector-frother.....	Rougher flotation feed.	105	.235
Do. ²do.....	Cleaner flotation feed.	27	.06
WATER RECLAMATION				
Lime.....	Coagulant.....	Slime-tailing thickener	1,550	3.47
Superfloc 330 ³	Flocculant.....do.....	30	.067
Do. ³do.....	Concentrate thickener..	15	.033

¹A product of Grain Processing Corp., Muscatine, Iowa.

²A product of Ashland Chemical Co., Columbus, Ohio.

³A product of American Cyanamid Co., Wayne, N.J.

TABLE 5. - Concentrate analysis, percent

Fe	Fe ²⁺	SiO ₂	Al ₂ O ₃	TiO ₂	MgO	CaO	S	P	Mn	K ₂ O	Na ₂ O	LOI, ¹ ° C	
												400	1,000
63.5	0.9	4.3	0.40	0.27	0.14	0.46	0.034	0.037	0.17	<0.05	<0.1	2.8	3.4

¹LOI is loss on ignition.

WET HIGH-INTENSITY MAGNETIC SCAVENGING OF FLOTATION TAILINGS

Equipment and Operating Conditions

The purpose of adding the wet high-intensity magnetic (WHIM) separation step was to increase the iron recovery by scavenging the iron units from the flotation tailing. Figure 2 shows the WHIM separation circuit that was

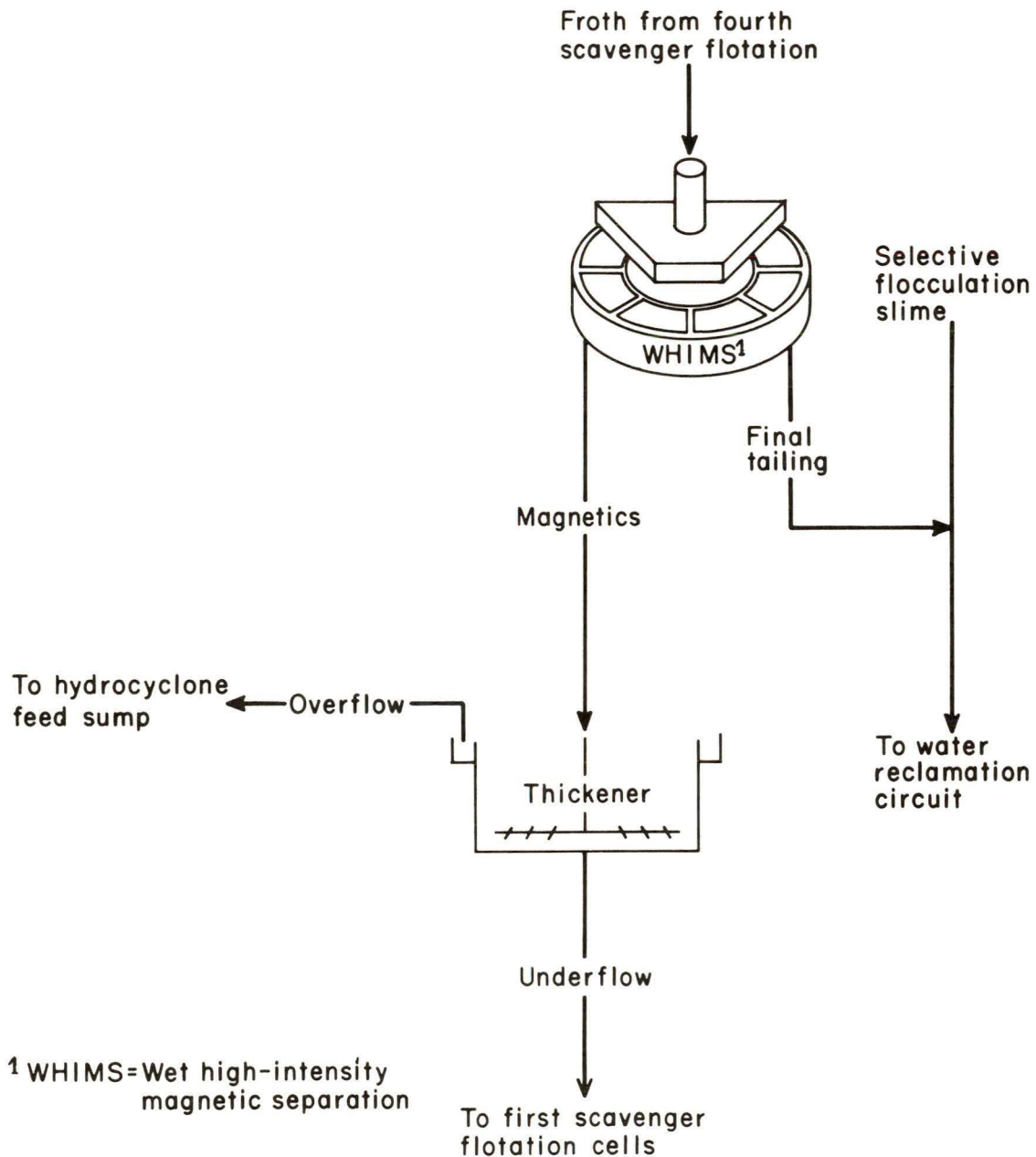


FIGURE 2. - Simplified flowsheet showing wet high-intensity magnetic scavenging of the flotation tailing.

incorporated into the selective flocculation and flotation flowsheet. The froth tailing from the fourth scavenger flotation cells was fed to the WHIM separator, and the weakly paramagnetic iron oxide particles were collected. Since the magnetic concentrate was very dilute, a 120- by 30-centimeter (4- by 1-foot) thickener was used to increase the solids content of the concentrate before it was recirculated to the flotation circuit. The thickener underflow was returned to the first scavenger flotation cells. The thickener overflow was used to dilute the feed to the classifying hydrocyclone. The nonmagnetic tailing from the WHIM separator, rather than the fourth scavenger flotation froth, was the final tailing product and was pumped to the slime-tailing thickener in the water reclamation circuit.

The WHIM separator used to scavenge the flotation tailing was the Mark I High-Gradient Carousel separator, a prototype of a continuous separator designed by Magnetic Engineering Associates (Sala Magnetics) of Cambridge, Mass. Kelland and Maxwell (2) have described the application of the separator to the beneficiation of oxidized taconites. The separator is illustrated in figure 3. The outer diameter of the matrix holder is 46 centimeters (18.1 inches). Each of the 12 matrix compartments has an area of 59 cm² (9.1 in²) and is packed with matrix to a depth of 9 centimeters (3.54 inches). The matrix was a coarse grade of ferritic stainless steel wool (15 percent Cr). The filaments, which were irregular in size and shape, were between 110 and 150 micrometers in diameter and had a packing density of 0.17 g/cm³ (2.2 volume-percent). The

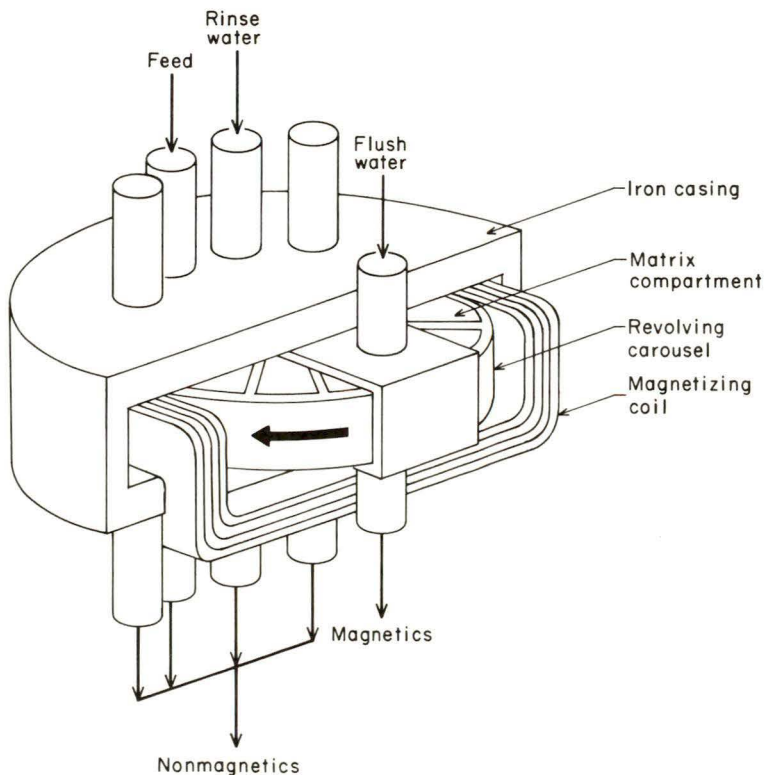


FIGURE 3. - Mark I High-Gradient Carousel separator.

magnetic field is parallel to the direction of slurry flow in the feed inlet region. The magnet was operated at 0.6 tesla (6 kilogauss) and required a current of 1,100 amperes at 13.2 volts, or 14.5 kilowatts of electric power. Water is used to wash the nonmagnetics through the matrix. Mineral particles retained in the matrix are carried by the rotation of the matrix holder into a shielded, low-field region and are flushed out.

During the flotation tailing scavenging tests, the matrix holder rotation rate was set at 4 rev/min. The tailing slurry was fed into the second inlet port. Rinse water at a rate of 15 liters/min (4 gal/min) was added to the third inlet port. A flow of

30 liters/min (8 gal/min) of water was used to flush the magnetic concentrate out of the matrix. All flow velocities through the matrix were controlled by gravity and the resistance of the bed.

Metallurgical Results

In a preliminary study, the possibility of incorporating WHIM scavenging in the selective flocculation and flotation flowsheet was explored. The results of an 8-hour pilot plant test are shown in table 6. The magnetic separator recovered 46.9 percent of the Fe in the flotation tailing and 25.1 percent of the weight. The final tailing assay was decreased from 12.7 to 9 percent Fe. Recycling the magnetic concentrate to the flotation circuit increased the overall recovery from 72 percent in the baseline test to 81.5 percent. The concentrate grade was 63.5 percent Fe in both tests.

TABLE 6. - Preliminary metallurgical results for WHIM scavenging of the flotation tailing, percent

Product	Fe	Based on flotation tailing		Based on rodmill feed	
		Weight	Fe distribution	Weight	Fe distribution
Magnetic concentrate	23.7	25.1	46.9	13.8	9.0
Nonmagnetic tailing.	9.0	74.9	53.1	41.1	10.2
Flotation tailing...	12.7	100.0	100.0	54.9	19.2

During the 5-day continuous campaign, the pilot plant was operated for eight consecutive 8-hour shifts without WHIM scavenging, for six consecutive shifts with scavenging, and for a final shift without scavenging. The average metallurgical results for the WHIM scavenging tests are given in tables 7-8. About 51.2 percent of the Fe and 19.3 percent of the weight in the flotation tailing were recovered. The WHIM concentrates at 1.9 percent solids were too dilute to recycle to the flotation circuit and had to be thickened. The thickener underflow contained 5.8 percent solids, assayed 17.2 percent Fe, and contained 80 percent of the solids in the magnetic concentrate. The thickener overflow assayed 22.2 percent Fe. A large circulating load due to recycling the magnetic concentrates to the flotation circuit did not develop. The weight of the flotation tailing increased by about 5 percent.⁶

TABLE 7. - Metallurgical results for WHIM scavenging of the flotation tailing,¹ percent

Product	Fe	Based on flotation tailing		Based on rodmill feed	
		Weight	Fe distribution	Weight	Fe distribution
Magnetic concentrate	18.2	19.3	51.2	8.1	4.1
Nonmagnetic tailing.	4.1	80.7	48.8	33.8	4.0
Flotation tailing...	6.9	100.0	100.0	41.9	8.1

¹Average of six consecutive 8-hour shifts.

⁶From table 8, percent increase in weight = $\frac{(41.9 - 40.0)}{40.0} \times 100 \approx 5$ percent.

TABLE 8. - Comparison of metallurgical data with and without WHIM scavenging, percent

Product	Without WHIM scavenging ¹	With WHIM scavenging ²
Feed: Fe.....	36.1±0.8	35.5±1.0
Concentrate:		
Fe.....	63.5±1.1	63.6±1.0
SiO ₂	4.3±1.1	4.4± .9
Weight.....	48.0±4.2	48.3±2.0
Fe distribution.....	84.4±4.6	86.5±1.9
Slime:		
Fe.....	19.8±2.6	18.9±2.4
Weight.....	12.0±2.8	17.9±2.4
Fe distribution.....	6.5±1.2	9.5±1.8
Flotation tailing:		
Fe.....	8.0±3.2	³ 6.9±1.1
Weight.....	40.0±3.2	³ 41.9±1.6
Fe distribution.....	9.1±4.3	³ 8.1±1.7
WHIM tailing:		
Fe.....	-	4.1± .6
Weight.....	-	33.8± .9
Fe distribution.....	-	4.0± .6

¹Average data for seven consecutive 8-hour shifts and 95-percent confidence intervals.

²Average data for six consecutive 8-hour shifts and 95-percent confidence intervals.

³Feed to WHIM separator.

The metallurgical results with and without magnetic scavenging in the selective flocculation and flotation flowsheet are compared in table 8. The average Fe loss in the slime was 9.5 percent, compared with 6.5 percent for the circuit not including WHIM scavenging. The difference was probably caused by the magnetic concentrate thickener overflow, which was used to dilute the hydrocyclone feed. The thickener overflow contained the finest particles in the magnetic concentrate and a higher level of residual amine in the aqueous phase than the dilution water from the reclamation circuit. Magnetic scavenging decreased the loss of iron in the flotation tailings from an average of 9.1 to 4.0 percent. The net increase in iron recovery was from 84.4 to 86.5 percent. The average concentrate grade of 63.6 percent Fe and 4.4 percent SiO₂ was similar to the grade obtained with no recycling of the magnetic concentrate.

The metallurgical results and separation efficiency obtained with the WHIM separator should be improved with a current state-of-the-art separator design. The Mark I Carousel separator has two design limitations: (1) The magnetic shielding of the section of the separator in which the magnetics are flushed out is not adequate, and (2) since the matrix compartments are unsealed, the feed slurry, rinse water, and flush water have flow velocities which are determined by gravity and the resistance of the bed. With a 0.6-tesla (6-kilogauss) field in the feed section, there is enough field

leakage in the magnetic flush section to impede matrix cleaning. With inadequate cleaning of the matrix, many of the most active collection sites in the matrix become occupied, and the magnetics in the new feed are collected with decreased effectiveness on less active sites. A feed slurry and rinse water velocity in the range of 1 to 5 cm/sec (0.4 to 2 in/sec) are desirable, but a flush water velocity in excess of 20 cm/sec (7.9 in/sec) is desirable to improve cleaning of the steel wool matrix. The Mark I separator was operated at theoretical feed, rinse, and flush flow velocities of 7.4, 5.3, and 10.6 cm/sec (2.9, 2.1, and 4.2 in/sec), respectively.

WATER RECLAMATION

Figure 4 is the flowsheet of the water reclamation system used to treat the waste streams. The combined slime and flotation tailing gave a 9.1-percent-solids slurry which was flocculated with hydrated lime and Superfloc 330 (table 3) before being fed to the 1.8- by 1.8-meter (6- by 6-foot) thickener.

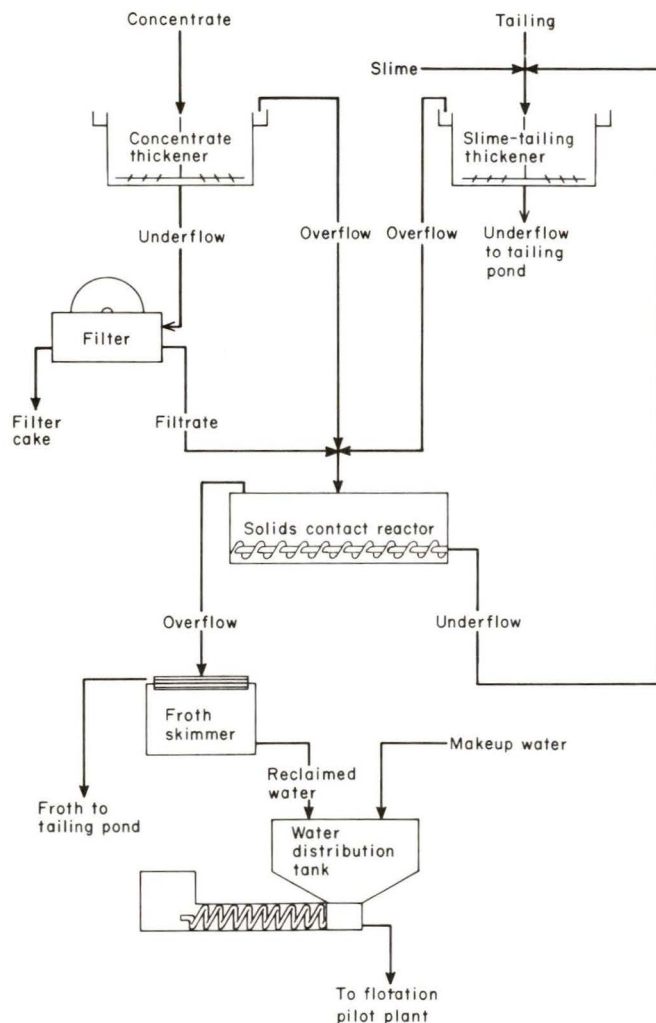


FIGURE 4. - Flowsheet of water reclamation system.

The thickener underflow was controlled at about 50 percent solids to maximize water recovery and was routed to the tailings pond. The overflow was pumped to a solids contact reactor (SCR).⁷ The concentrate thickener overflow and the discharge from the filtrate pump were routed to the SCR. The SCR underflow, at about 10 percent solids, was pumped to the slime-tailing thickener. The SCR overflow was treated in a two-cell flotation unit (froth skimmer) to remove excess froth, whereas the underflow or reclaimed water was pumped to the distribution tank, in which makeup water was added to compensate for the water lost in the pilot plant product. An average of 67 liters/min (17.7 gal/min), or 88 percent of the total water usage of 76 liters/min (20.1 gal/min),

⁷Solids contact reactor (SCR) is a rectangular compartmented water treatment tank, which employs the principles of sludge-blanket clarification.

was recovered and recycled. About 6 liters/min (1.6 gal/min) were lost in the slime-tailing thickener underflow, less than 1 liter/min in the filter cake, and the remainder in the skimmer froth. The recycled water had an average pH of 11.0, turbidity of 480 ppm SiO₂ equivalent, calcium ion content of 7.6 ppm, and amine analysis of 0.6 ppm.

SUMMARY AND CONCLUSIONS

Pilot plant research using the selective flocculation and flotation process on a bulk sample that represents 1/2 to 1 billion tonnes of oxidized taconite gave excellent metallurgical results. The concentrates averaged 63.5 percent Fe and 4.3 percent SiO₂ with an accompanying average iron recovery of 84.4 percent.

The cost of the reagents used in the pilot plant evaluation are estimated in table 9 and represent October 1977 price quotes f.o.b. for bulk quantities at the manufacturer's plant. The estimate shows that the beneficiation and water reclamation costs are 42.7 and 10 cents, respectively, or a total of 52.7 cents/t (53.5 cents/lt) of crude ore. Assuming a 48-percent weight recovery, the f.o.b. reagent cost per tonne of concentrate is approximately \$1.10 (\$1.11/lt). The reagent usages indicate that all the reagents except sodium hydroxide are at a minimum level consistent with good metallurgical results. Since only 88 percent of the pilot plant water usage was reclaimed, the caustic addition could probably be decreased because about 90 to 95 percent of the water usage would be recycled in a commercial plant.

TABLE 9. - Estimated costs of reagents used in selective flocculation and flotation

Reagent	Reagent cost ¹		Amount used		Cost	
	Cents/kg	Cents/lb	Grams/t	Lb/lt	Cents/t of feed	Cents/lb of feed
SELECTIVE FLOCCULATION AND FLOTATION						
Sodium hydroxide.....	² 17.6	8.0	³ 760	1.70	13.4	13.6
Sodium silicate.....	7.6	3.45	765	1.72	5.8	5.9
Pearl starch.....	17.6	8.0	375	.84	6.6	6.7
Arosurf MG-98A.....	127.9	58.0	132	.295	16.9	17.1
Subtotal.....	-	--	-	-	42.7	43.3
WATER RECLAMATION						
Lime.....	3.5	1.6	1,550	3.47	5.4	5.6
Superfloc 330.....	101.4	46.0	45	.10	4.6	4.6
Subtotal.....	-	-	-	-	10.0	10.2
Total.....	-	-	-	-	52.7	53.5

¹Reagent costs are f.o.b. quotes, October 1977, for bulk quantities.

²Cost per dry kilogram of solids in a 50-percent NaOH solution.

³The value includes 10 g/t (0.02 lb/lt) used to prepare the starch solution.

WHIM separation was not applied to the primary concentration of oxidized taconites, but was used as an adjunct beneficiation step to scavenge the iron oxides from the cationic flotation tailing. The use of WHIM separation as a froth tailing scavenger indicated that the flotation concentrate grades

obtained with and without WHIM scavenging were similar, and that the iron lost in the final tailing can be decreased by approximately one-half. The net gain in recovery for the tests reported ranged from 2 to 9.5 percent, depending on the iron assay of the flotation tailing. If the iron recovery in the selective flocculation and flotation process is increased by only 2 percent, the added expense of WHIM separation may not be justified. The addition of a WHIM separation step will also increase the amount of water treated in the water reclamation circuit. However, when the iron recovery in the flotation phase of the selective flocculation and flotation process varies because of changes in ore characteristics, WHIM scavenging may be an effective method for leveling off the iron recovery.

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