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Flotation of Marquette Range Nonmagnetic Taconite Using Innovative Procedures

UNITED STATES DEPARTMENT OF THE INTERIOR

Report of Investigations 7826

Flotation of Marquette Range Nonmagnetic Taconite Using Innovative Procedures

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UNITED STATES DEPARTMENT OF THE INTERIOR
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FLOTATION OF MARQUETTE RANGE NONMAGNETIC TACONITE USING INNOVATIVE PROCEDURES

by

D. W. Frommer,[1] P. A. Wasson,[2] and D. L. Veith[3]

ABSTRACT

Processing of nonmagnetic taconites by selective flocculation-desliming and anionic flotation of calcium-activated silica has been investigated using innovative procedures as a means of improving operating costs. Areas chosen for cost-cutting evaluation were grinding, fine screening, and water conservation and reclamation.

The investigation was performed on Marquette range nonmagnetic taconites in the Bureau of Mines Twin Cities Metallurgy Research Center flotation pilot plant, at feed rates ranging between 900 and 1,100 lb/hr.

With fully integrated water reclamation, flotation concentrates containing 65.8 percent Fe and 3.4 percent SiO_2 were made in the best operating period, with accompanying weight and iron recoveries of 40 and 74 percent, respectively. During this same period, the estimated cost of both flotation and water reclamation reagents was only \$0.80 per ton of concentrate (1968 pricing basis). This cost is about 15 percent lower than obtained previously at this Center for flotation reagents only.

The investigation demonstrated additional potential savings in operating costs of approximately \$0.96 per ton of concentrate through autogenous grinding as a substitute for ballmilling. Fine screening of intermediate concentrates was employed advantageously, but the frequency of blinding may be a factor militating against use of the screen in this flotation system.

INTRODUCTION

During the decade of 1960 to 1970, the Bureau of Mines engaged in extensive research to develop the potential of nonmagnetic taconites of the Lake Superior region. Investigations were undertaken to determine the extent and

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[1]Supervisory metallurgist.
[2]Metallurgist (now retired)
[3]Metallurgist.

character of some of the more significant reserves,[4] and considerable effort was devoted to studying and improving a flotation process employing anionic flotation of calcium activated silica. These investigations were conducted on both bench and pilot plant scale, both independently and with the support of individual iron mining companies. Starting with demonstrations[5] of the process viability on a variety of ores, the scope of the investigations was extended to include the development of pragmatic methods of exercising control over the flotation process,[6] the conceptual development of a selective flocculation-desliming step as a precondition to flotation,[7] and finally to the development of a compatible system of water reclamation.[8]

In spite of the cited advances in technology, the cost of flotation processing on nonmagnetic taconites is thought to place these materials at an economic disadvantage to the more cheaply processed magnetic taconites. Hays[9] has indicated that production costs for pellets from magnetic taconites would be about $10.20 per ton, whereas the costs from nonmagnetic sources processed by selective flocculation-desliming flotation might be $0.78 to $0.88 per ton higher.

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[4] Heising, L. F., and D. W. Frommer. Lake Superior Iron Resources: Preliminary Samples and Metallurgical Evaluation of Selected Michigan-Wisconsin Iron Formations. BuMines RI 6895, 1967, 35 pp.
Marovelli, R. L., D. W. Frommer, F. W. Wessel, L. F. Heising, P. A. Wasson, and R. E. Lubker. Lake Superior Iron Resources: Preliminary Sampling and Metallurgical Evaluation of Mesabi Range Nonmagnetic Taconites. BuMines RI 5670, 1961, 35 pp.

[5] Frommer, D. W., M. M. Fine, and L. Bonicatto. Anionic Flotation of Silica From Western Mesabi and Menominee Range Iron Ores. BuMines RI 6399, 1964, 25 pp.
Wasson, P. A., R. T. Sorensen, and D. W. Frommer. Anionic Flotation of Silica From Goethitic Iron-Bearing Material of Cuyuna Range, Minn. BuMines RI 6199, 1963, 11 pp.

[6] Colombo, A. F. Effect of Tapioca Flour on the Anionic Flotation of Gangue From Iron Ores. BuMines RI 7618, 1972, 18 pp.
Colombo, A. F., R. T. Sorensen, and D. W. Frommer. Calcium Ion Measurements Provide Insights to Anionic Flotation of Silica. Trans. Soc. Min. Eng., AIME, June 1965, pp. 100-109.

[7] Frommer, D. W. USBM-CCI Cooperative Research on Flotation of Nonmagnetic Taconites of Marquette Range. Blast Furnace and Steel Plant, v. 57, No. 5, May 1969, pp. 380-388.
Frommer, D. W., and A. F. Colombo. Process for Improved Flotation Treatment of Iron Ores by Selective Flocculation. U.S. Pat. 3,292,780, Dec. 20, 1966.

[8] Frommer, D. W. Aspects of Water Reuse in Experimental Flotation of Nonmagnetic Taconite. Trans. Soc. Eng., AIME, v. 247, December 1970, pp. 281-287.

[9] Hays, Ronald M. Economic Evaluation of Processing Low-Grade Oxidized Iron Ores. Blast Furnace and Steel Plant, v. 57, No. 8, August 1969, pp. 652-658.

This report describes processing studies with cost reduction as the objective. These cost reductions were sought through changes in grinding and classification, in flotation circuitry, and in water conservation and reclamation procedures. Because of complexities in the circuit, the time needed to establish circuit equilibrium, and because of unavoidable ore variations, the results obtained are considered to be primarily of directional value. The more promising modifications would warrant confirmatory studies over longer operating periods.

ACKNOWLEDGMENTS

The investigation was conducted on ore samples from the Tilden mine near Ishpeming, Mich. The assistance given by the Cleveland-Cliffs Iron Co. in support of the cooperative effort through contributions of ore, partial funding, and manpower is gratefully acknowledged.

RAW MATERIALS

Two ore samples were used, both from the Tilden east pit. Both were similar in that they were essentially mixtures of hematite-martite, quartz, and lesser amounts of relict magnetite, and goethite. Less than 10 percent of the iron oxides occurred as goethite. As has been the case with other samples from the same locality, the size distribution of flotation concentrates obtained from those ores must be 90 to 95 percent passing 400 mesh to insure a high-quality product.

During the short-term investigation reported on pp. 3-20 a stockpile sample was used; this sample contained about 36 percent Fe and 46 percent SiO_2. A sample designated as being from the lower bench was used during the continuous investigation described on pp. 20-28. The only discernible differences between the two samples were that the lower bench materials were about 0.5 percent lower in iron content than the stockpile sample, and that the stockpile sample apparently contained more goethite.

SHORT-TERM INVESTIGATION OF INNOVATIVE PROCEDURES

Cost Comparisons--Pebble Mill Versus Conventional Ball Mill Secondary Grind

In previous studies,[10] grinding of the ore was accomplished with a rod mill-ball mill combination. A typical condition employed a Marquette range hematitic-goethitic-martitic jasper fed at a rate of 900 lb/hr. The feed was ground at 80 percent solids in a 2-ft by 4-ft rodmill, loaded with 1,500 lb of steel rods. Power consumption in the rodmill averaged about 12.5 hp-hr/ton of feed.[11] The feed size was minus 3/4 inch, with about 25 percent coarser than 1/2 inch. The rodmill product was finer than 10 mesh, and about 50-55 percent of the weight was distributed to the minus 200-mesh sizes.

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[10] First work cited in footnote 7.

[11] Whenever "ton" is used in this report, a long ton of 2,240 lb is meant.

A final grind of 93 percent minus 400 mesh was obtained as a cyclone overflow after grinding at 62 percent solids in a 3-ft by 3-ft ball mill loaded with 2,400 lb of graduated steel balls. This mill drew an average of 22 hp-hr/ton of new feed. Thus, a datum of 34.5 hp-hr/ton was established as an average power requirement for both rod and ball milling. Such determinations as were made indicated a rod consumption of 2.1 lb/ton of feed. This amount of rod wear, at the rate of 0.22 lb per kW-hr, does not appear excessive. Although a measured ball consumption of 4.4 lb/ton first appears somewhat high, normal wear rate of 0.15 to 0.25 lb/kW-hr has been reported.[12] Using the latter, high value for ball wear rate gives a slightly lowered value for ball consumption of 4.1 lb/ton.

Metal consumption of 2.1 and 4.1 lb/ton for rods and balls, respectively, has been assumed in the following discussion, and appears to be realistic when the character of the ore and the fine grinding employed are considered. The contribution of such grinding to the cost of producing concentrates can be partially measured, employing both the above and the following assumptions:

Power costs = $0.010 per kW-hr = $0.0075 per hp-hr[13]

Costs of grinding steel = $0.125/lb(balls), $0.0925/lb(rods).

Recovery in concentrate, wt-pct = 42.5

The rodmill-ballmill column of table 1 shows that reduction in metal wear would be particularly beneficial to process economics, and suggests that autogenous grinding might be profitably applied with the objective in mind. One outstanding example of both primary and secondary autogenous grinding in the iron ore industry is that of the Empire mine.[14] In the case of the Empire installation, primary autogenous grinding substitutes for crushing and rodmilling and provides pebbles used in the secondary autogenous grinding stage.

[12]Faucher, J. A. R. Comminution by Conventional and Autogenous Grinding. Can. Min. J., June 1969, pp. 69-73.
[13]Work cited in footnote 9.
[14]Bjorne, Calvin, and Lee Erickson. Autogenous Grinding at the Empire Mine. Proc. 29th Ann. Mining Symp. and 41st Ann. Meeting of Minn. Sec. AIME., Jan. 15-17, 1968, University of Minnesota, 1968, pp. 199-204

TABLE 1. - Some estimated comparative costs of grinding in rodmill-ball mill versus rodmill-pebble mill circuits

Item of cost	Cost per ton of concentrates	
	Rodmill-ball mill	Rodmill-pebble mill
Grinding power:		
Rodmill	$0.22	$0.22
Secondary milling	.39	.63
Total	.61	.85
Metal wear:		
Rods	.46	.46
Balls	1.20	-
Total	1.66	.46
Grand total	2.27	1.31

Although the Bureau lacked facilities for primary autogenous grinding, it was possible to compare rodmill-secondary autogenous grinding with the more conventional rodmill-ball mill combination.

The pertinent experimental data used in the rodmill-pebble mill column of table 1 was generated in a 3-ft by 8-ft peripheral discharge rodmill that had been converted to a lifter-type grate discharge pebble mill. Pebbles consisted of the coarse, minus 2-1/2-in--plus 1-in fraction from the Tilden ore, similar in other characteristics to the sample being treated. The complete grinding circuit consisted of a rodmill in series with the pebble mill, which was in closed circuit with a 3-in cyclone. The initial pebble load was 2,400 lb, and with a pebble addition of 50 lb/hr, the power consumption by the pebble mill was stabilized at 35.5 hp-hr/ton. The 50-lb/hr pebble addition rate was considered the minimum needed to maintain sufficient media volume in the mill for efficient grinding.

The pebble mill, at a feed rate of 900 lb/hr, produced a grind of 88 to 90 percent minus 400 mesh, which was only slightly coarser than that obtained from the ball mill, with a net input of 22 hp-hr/ton. Even with this grind, the power input of 35.5 hp-hr/ton to the pebble mill was over 50 percent higher than the average expected from the ball mill. However, despite the added $0.24 in power cost assigned to the pebble mill operation, the operating costs in producing a ton of concentrates can be lowered by an estimated $0.96 through elimination of ball wear, as shown in table 1. Furthermore, the pebble mill product responded adequately to the selective flocculation and flotation processes when using a circuit similar to that previously described.[15]

[15] Page 381 of first work cited in footnote 7.

Other Potential Cost-Saving Measures

Circuit Changes

In the next phase of the investigation, the circuit was altered radically to incorporate several changes which were thought to have potential for improving processing economics. These measures are listed below, along with the rationale for their employment:

(1) Coarse grinding--Coarse grinding of feed to flotation was expected to reduce power costs and to yield concentrates of lower surface area than might normally be expected. The expectation was that liberated silica could be floated at these coarser sizes, but also that the coarse fractions of the concentrates would be subgrade.

(2) Screening of concentrates--The treatment proposed to overcome the deficiency in quality of rougher concentrates was to pass these initial concentrates over a 325-mesh (0.004-in opening) rapped screen panel to remove the oversize particles.[16] This oversize product was reground and routed to the head of flotation circuit. To further insure final concentrates of acceptable quality, an additional increment of fatty acid was added to the screen undersize, and the product subjected to a scavenging flotation stage. The scavenger froth, being of intermediate quality, was also routed back to the head of the flotation circuit. The proportion of total fatty acid added to each of the two distribution points (pre-flotation conditioning and screen undersize) was varied considerably in an attempt to optimize the flotation circuit.

(3) Reduced water volume to desliming vessel--Two avenues to reduce the water volume used in the 5-ft diameter by 4-ft desliming vessel were explored. By installation of a 1-ft by 15-ft spiral screw classifier as a substitute for the hydraulic cyclone classifier, the desired size separation could be made with a higher range of overflow solids. As an example, the spiral classifier overflow was taken off at 20 to 25 percent solids, compared with the operating norm of 12 to 15 percent solids obtained in the cyclone overflow. At a more advanced stage of the investigation, one-half of the desliming vessel overflow was routed back to join the classifier overflow prior to entry to the desliming vessel. This return flow was expected to increase the rise rate in desliming vessel without materially increasing fresh water consumption.

General Test Conditions and Results

Test runs 1 through 3 used a 3-ft by 5-ft rodmill (drawing about 1 hp-hr above previously described levels), a 18-in by 30-in regrind ball mill, and a 3-ft by 3-ft ball mill. The power to the regrind mill loaded with 500 lb of balls was not measured, nor was it included in the reported totals, but it was estimated to be about 3 hp-hr. The 3-ft by 3-ft mill had a 2,000-lb ball loading. In runs 1 and 2, water reclamation and reuse were practiced, using

[16] Frommer, D. W. An Improved Method for Treating Iron Ores by Flotation and Screening. Invention Report MIN-1652, Mar. 13, 1970.

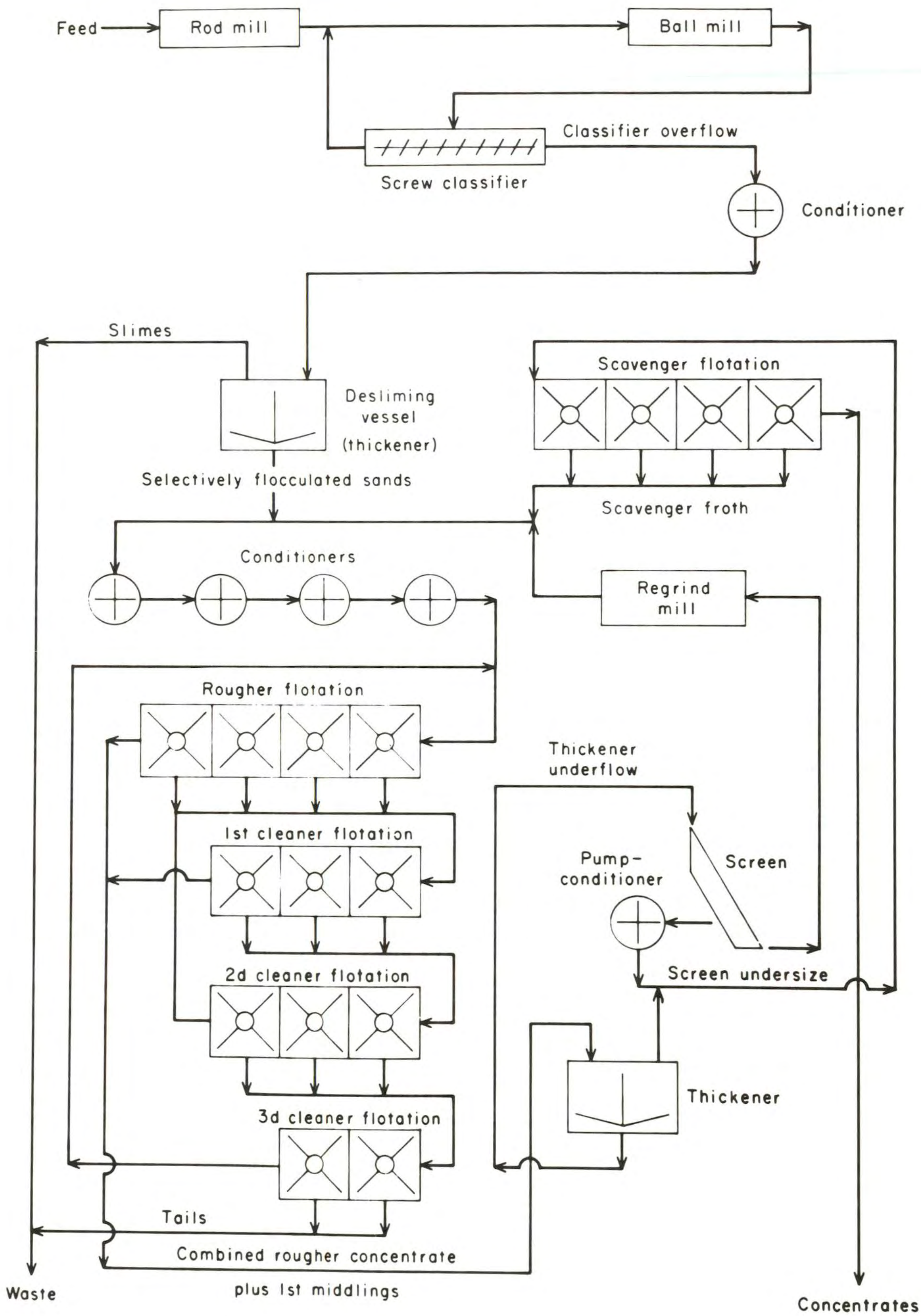

FIGURE 1. - Diagrammatic representation of typical flowsheet, test 2.

procedures described in subsequent sections of this report. In test 3, fresh water from Minneapolis municipal supplies was used; some fresh water was also used as makeup water in runs 1 and 2.

Experimental data generated under these varied test conditions are given in tables 2-3 and 4-5. A pilot plant flowsheet, as used in these tests (but excluding the water treatment circuit) is given in figure 1.

TABLE 2. - Metallurgical results: Rodmill-ball mill grinding, with fine screening of intermediate concentrates, using reclaimed water

Product	Test 1			Test 2		
	Wt-pct	Iron, pct		Wt-pct	Iron, pct	
		Analysis	Distribution		Analysis	Distribution
Screen oversize	31.8	43.8	38.8	24.6	43.1	29.7
Screen undersize	79.9	55.4	123.3	80.1	56.7	127.7
Total screen feed	111.7	52.1	162.1	104.7	53.5	157.4
Scavenger froth	42.2	46.6	54.3	43.5	48.2	59.0
Slime	8.1	17.8	4.0	9.1	19.7	5.1
Flotation tails	54.2	17.9	27.0	54.3	17.2	26.2
Final concentrate[1]	37.7	65.7	69.0	36.6	66.8	68.7
Total	100.0	-	100.0	100.0	-	100.0

[1] SiO_2 in final concentrate: Test 1, 3.5 pct; test 2, 3.7 pct.

TABLE 3. - Operating data for tests 1 and 2 (See table 2)

	Test 1	Test 2
Feed....pct iron	35.9	35.6
Feed rate....lb/hr	1100	1100
Grinding power....hp-hr/ton	28.8	26.4
Classifier overflow....pct minus 400 mesh	82	78
Ca^{2+} in conditioned flotation feed....ppm	19	15
pH of conditioned flotation feed	11.8	11.8
Reagent additions, lb/ton:		
Sodium hydroxide	3.1	3.1
Sodium tripolyphosphate	0.28	0.14
Sodium silicate	-	0.28
Corn starch	1.4	1.4
Calcium chloride	1.67	1.4
Fatty acid:		
Rougher flotation	0.45	0.45
Scavenger flotation	0.78	0.78
Frother	0.006	0.006

TABLE 4. - Metallurgical results: Rodmill-ball mill grinding with fine screening of intermediate concentrates, using municipal water

Product	Wt-pct	Iron, pct	
		Analysis	Distribution
Screen oversize	9.2	46.8	12.1
Screen undersize	51.1	62.6	89.9
Total screen feed	60.3	60.2	102.0
Scavenger froth	7.5	45.8	9.7
Slime	5.0	18.4	2.6
Flotation tails	51.4	11.9	17.2
Final concentrate[1]	43.6	65.4	80.2
Total	100.0	-	100.0

[1] SiO_2 in final concentrate, 3.6 pct.

TABLE 5. - Operating data for rodmill-ball mill grinding tests using municipal water (See table 4)

Feed....pct iron..	35.6
Feed rate....lb/hr..	1100
Grinding power....hp-hr/ton..	27.4
Classifier overflow.....pct minus 400 mesh..	85
Ca^{2+} in conditioned flotation feed.....ppm..	7
pH of conditioned flotation feed	11.7
Reagent additions, lb/ton:	
Sodium hydroxide	5.6
Sodium tripolyphosphate	0.14
Sodium silicate	.28
Tapioca flour	1.40
Calcium chloride	0.84
Fatty acid:	
Rougher float	1.12
Scavenger float	0.28
Frother	0.006

Discussion of Results

The experimental results show that concentrates of excellent quality could be made from relatively coarse feeds to primary flotation. The combination of fine screening and scavenger flotation of rougher concentrates, or of rougher concentrates and first middlings, raised the grade of these products from marginal acceptability or below to the 65 to 66 percent Fe content reported. Abbreviated flowsheets are shown in figures 2, 3, and 4 with, respectively, a superimposed partial material balance, screen analyses, and reagent use for test 2.

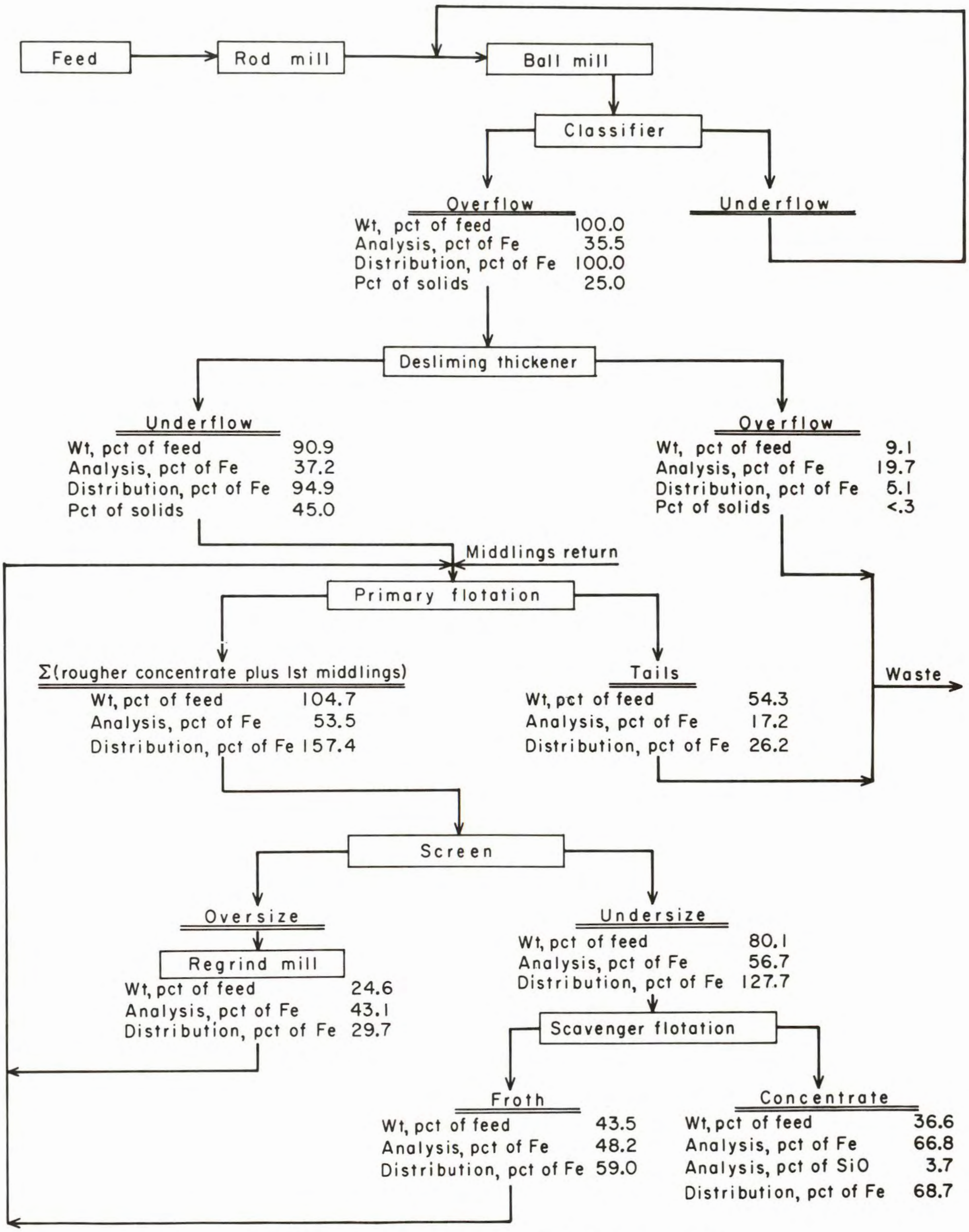

FIGURE 2. - Partial materials balance, test 2.

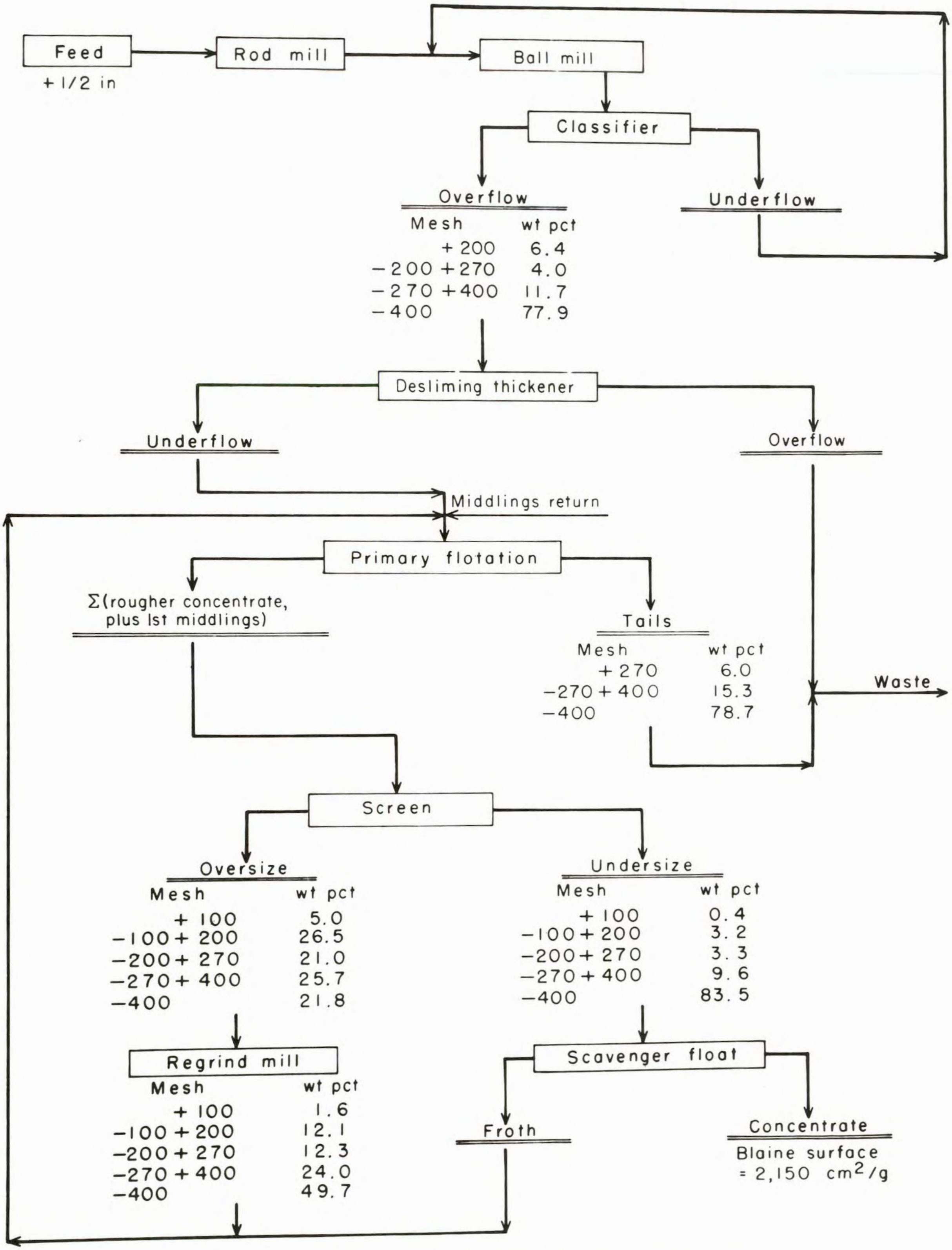

FIGURE 3. - Screen sizing of select process streams, test 2.

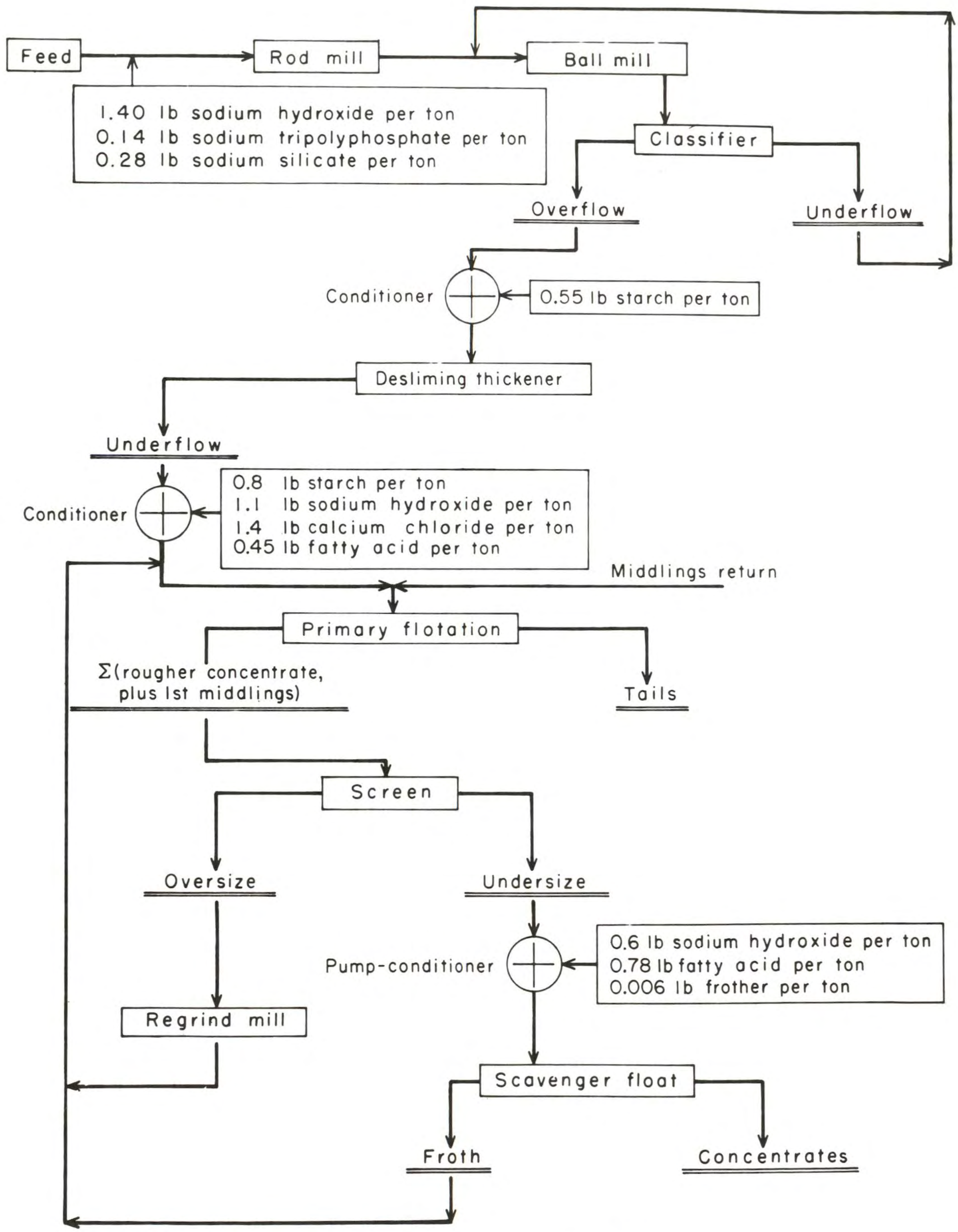

FIGURE 4. - Reagent use, test 2.

The intent of distributive variations in fatty acid to the rougher and scavenger flotation sections was to produce a tailings of low iron content from the rougher section, and by intensive flotation in the scavenger to make a high-grade concentrate and a medium-grade scavenger froth. The latter would then be recirculated to the head of the conditioning section. It was thought that the residual collector coatings on minerals in the scavenger froth would then be desorbed and redistributed when these products came in contact with fresh feed reagentized with low levels of collector.

In tests 1 and 2, the fatty acid was distributed so that 0.45 lb/ton was added to the rougher flotation stage and 0.78 lb/ton to the scavenger flotation cells. In still other tests, not reported here, the fatty acid distributed was the reverse of that given above. It would appear that adding more of the fatty acid to the scavenger stage had the expected effect of increasing the circulatory load originating from that source. Otherwise, no substantial difference in metallurgy, as measured by iron content of concentrates and tailings, was evident as a consequence of the reversed fatty acid distributions.

These investigations illustrated that high-quality concentrates could be made from relatively coarse feeds with a minimum of attention to flotation control, as evidenced by the nearly identical metallurgical results obtained in tests 1 and 2, despite some minor differences in the reagent suite and pulp Ca^{2+} level. However, the iron content of 17 to 18 percent in the tailings was considered excessive in all tests using the reclaimed water. Screen analyses made on the tailings showed that most of the contained iron was in the minus 400-mesh fractions, as illustrated in the example listed in table 6. With the distribution as given in table 6, the problem of high iron content in tailings can be attributed largely to deficiencies in flotation selectivity rather than to inadequate liberation. Lowered selectivity, not liberation, is the causative factor, because tailings with less than 12 percent Fe were made in test 3, which used fresh municipal water.

TABLE 6. - Distribution of iron content in flotation tailings of test 2

Size, mesh	Wt-pct	Iron, pct	
		Analysis	Distribution
Minus 100 plus 270.......	6.0	15.5	5.5
Minus 270 plus 400.......	15.3	21.4	19.3
Minus 400................	78.7	16.1	75.2
Composite...........	100.0	16.9	100.0

It will be noted that not only does test 3 employ a different water source, but also that the reagent usage differs from those of tables 2-3. However, the examples given in both tables 2-3 and 4-5 represent a close approach to "best" reagent usage compatible with their water supply. Thus, the fact that numerous tests with reclaimed water did not yield tailings of comparable low iron content to those obtained with fresh water is strongly indicative of the influence of water quality on flotation selectivity. Since exclusive use of fresh water is impractical, the higher iron contents in tailings produced with recycled water are considered to be representative of the more realistic condition.

Several additional factors need to be considered in evaluating the flowsheet as described. The first of these is that a substantial circulating load can be generated within the flotation circuit with the screen oversize, scavenger froth, and middling products being returned to advanced sections of the circuit. The amount of material being returned from each of these operations places restraints on the desirability of grinding as coarse as permitted by the flotation function.

Large circulating loads can be self-perpetuating by exceeding the capacity of the fine screen and/or the regrind mill.

A second factor to be considered is that screen blinding may be a problem. Part of this problem may be accentuated by intermittent use, resulting in alternate wetting and drying, but, based on subsequent comparison with a flotation system using cationic collectors, the chemicals used in anionic silica flotation are thought to contribute to excessive[17] screen blinding. In any event, cementation of ore particles in the screen slots occurred, and could require such frequent cleaning so as to render impractical the use of the fine screens.

Initially, it was thought that the coarse grinding given to the primary feed to flotation would produce concentrates of relatively low surface areas. This hypothesis appeared to be validated in tests typified by test 2, giving a surface area of approximately 2,200 cm^2/g. However, these tests were characterized by only moderate iron recoveries; in duplicate tests similar to test 3 yielding iron recoveries of about 80 percent, the surface areas were in the range of 3,000 to 3,200 cm^2/g. These latter test results reporting large surface areas are believed to be due to added recovery of fine iron formerly being lost to the tailings.

Development of Water Reclamation Methods

Studies to reclaim water in a manner compatible to reuse in the anionic system for flotation of silica began in 1967. These first efforts were confined to batch studies on slime samples collected from pilot plant selective flocculation-desliming. A number of starches and synthetic flocculants were examined, as well as inorganic multivalent electrolytes.

These studies established that water clarification could be accomplished by controlled additions of lime. The calcium ion or calcium-hydroxy complex formed from dissolution of the lime could be expected to be electropositive; it thereby neutralizes the highly electronegative surfaces fo the particulate matter, and thus achieves flocculation. Water quality with respect to calcium hardness was kept within specified limits by an initial treatment with lime to

[17] Keller, Leon D. Fine Screening, the Current State of the Art. Skillings' Min. Rev., v. 60, No. 19, May 8, 1971, pp. 8-11, 23.

accomplish most, but not all, of the flocculation. A finishing treatment to further reduce turbidity was accomplished in bench-scale tests with one of several synthetic flocculants. In this manner, clarified effluents with ≤16 ppm Ca^{2+} were obtained. This 16 ppm Ca^{2+} content was chosen as goal in water treatment because the value corresponded with typical Minneapolis municipal water used successfully in prior investigations.

A multistage treatment was adapted to the flotation pilot plant that included recovery of water from tailings and concentrates, as well as from the slimes. Results of these investigations over a two-week around-the-clock campaign have been previously published[18] and figure 5 provides a simplified representation of the flowsheet used. It is important to note that lime additions to the concentrate thickener flocculated the concentrates and provided overflows of high calcium content. This overflow joined the flotation tailings in a second thickener and reacted with residual fatty acid contained in the tailings to form insoluble calcium soaps. Effluent from the tailings thickener passed through froth skimmers (flotation cells) to remove relict froth and foam for discard. These partially clarified effluents, plus additional lime, were joined with the slimes in a third thickener where slime flocculation occurred. The total of combined and partially clarified effluents were allowed to settle in a fourth thickener while in contact with Na_2CO_3 added to aid in lowering soluble calcium through precipitation as $CaCO_3$. Polymer X-150[19] a cationic flocculant also was added to reduce residual turbidity. The average amounts of chemicals used for water reclamation, in lb/ton of ore introduced as fresh feed, were as follows: (1) Hydrated lime, 4.7; (2) soda ash, 1.5; and (3) Polymer X-150, 0.012. The net result was a finished, reclaimed water containing ≤16 ppm Ca^{2+}, having turbidities in the range of 300 to 800 ppm, and a pH of ≥11.5. Use of the reclaimed water allowed for reduction in amounts of flotation chemicals having a value approximately equal to the cost of chemicals needed for water treatment.

These investigations with reclaimed water indicated the need for adjustments in the amounts of flotation reagents over those previously established for fresh water use, but otherwise demonstrated the compatibility of reclaimed water in the flotation process. An estimated 85 percent of total water requirements was readily reclaimed without prolonged impoundment.

Following the first continuous flotation campaign using reclaimed water, the next undertaking was designed to provide the additional 15 percent of makeup water needed. It was assumed that the only available supply would be in the "hard water" category. Obviously, such water could be softened at some expense before use, but it was believed that its hardness could be used to

[18] Frommer, D. W. Aspects of Water Reuse in Experimental Flotation of Nonmagnetic Taconites. Trans. AIME, v. 247, December 1970, pp. 281-287.

[19] Use of trade names here and elsewhere is to facilitate understanding and does not imply endorsement by the Bureau of Mines.

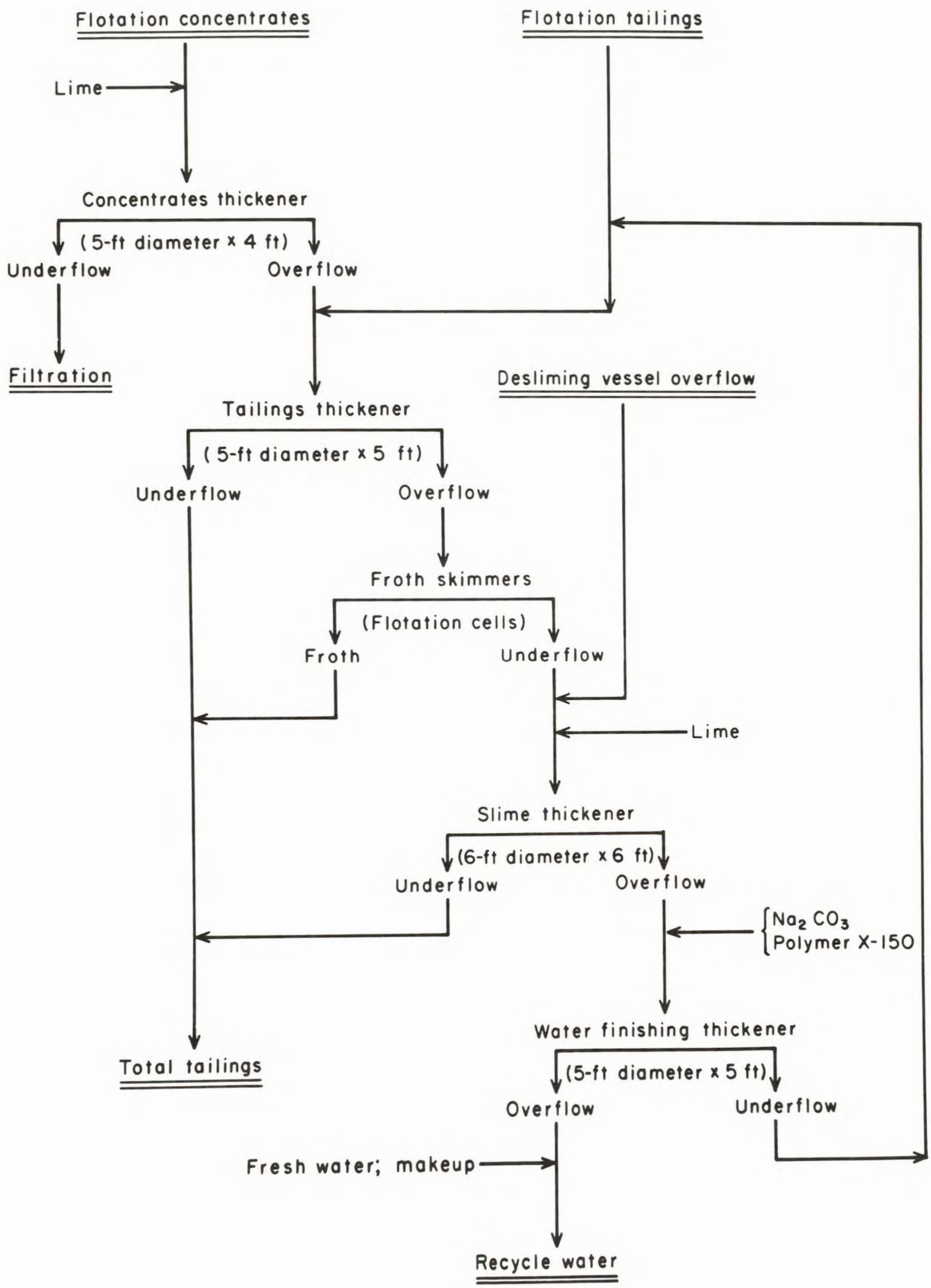

FIGURE 5. - Initial water reclamation flowsheet.

advantage in reclamation of the process streams. Spring water having the following approximate composition was available to test this hypothesis:

Total hardness-----472 ppm

Total alkalinity---292 ppm, as $CaCO_3$

Calcium hardness---318 ppm, as $CaCO_3$

Magnesium hardness-154 ppm, as $CaCO_3$

Calcium (Ca^{2+})-----127 ppm

pH----------------7.3

The spring water was added at 3 gal/min to meet the requirement for makeup water during a short test period of several weeks' duration.

Concurrent with the start of investigations using spring water, the water reclamation circuit was simplified by eliminating the froth skimmer in favor of a hydraulic cyclone, as shown in figure 6. Also, a solids-contact (sludge-blanket) flocculator[20] of at least 3 times the volume was substituted for the 5-ft by 5-ft finishing thickener, shown in figure 4. As far as could be determined, no particular benefit resulted from use of the solids-contact reactor that would not have been obtained with a conventional thickener of equivalent volume.

In the modified flowsheet, the spring water was added to the flotation tailings and slimes. The inorganic salts contained in the spring water were sufficient to provide incipient flocculation. Mixing of the spring water with the overflow from the concentrate-thickener, where only 1.2 lb of lime per ton of flotation feed was added, as shown in figure 6, resulted in a final effluent or finished water that contained 11 to 12 ppm Ca^{2+}, 700 ppm turbidity, but with a pH of 10. Additions of sodium carbonate and Polymer X-150 remained at the same levels as in the previous investigation. Although no attempt was made to use this reclaimed water in the flotation circuit, its quality with respect to Ca^{2+} and turbidity appeared to be equal to that obtained by previous procedure. Furthermore, these tests demonstrated a practical method of utilizing the ore process to extract calcium hardness from the makeup water.

After discontinuance of spring water in the tests, the physical arrangement was retained; however, after reversion to Minneapolis water for makeup, it was necessary to increase lime requirements to a total of 3.1 lb/ton of feed. It was also determined that the Na_2CO_3 and Polymer X-150 could be eliminated without an adverse effect on water quality.

Two additional changes were made and incorporated into the final flowsheet employing most of the previously discussed innovative procedures in a

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[20] Nordell, Eskel. Water Treatment for Industrial and Other Uses. Reinhold Publishing Corp., New York, 1961, pp. 495-501

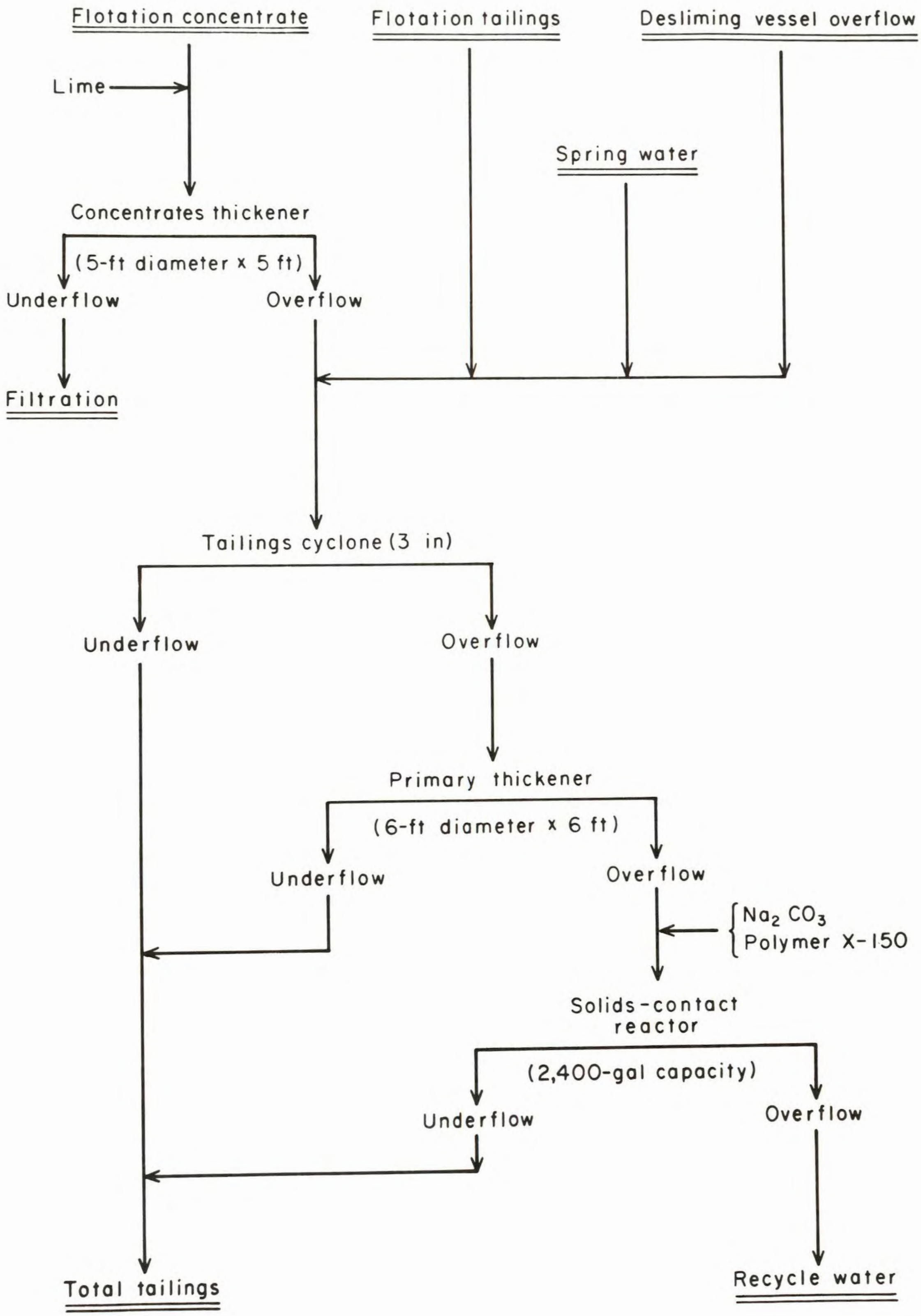

FIGURE 6. - Revised water reclamation circuit using spring water.

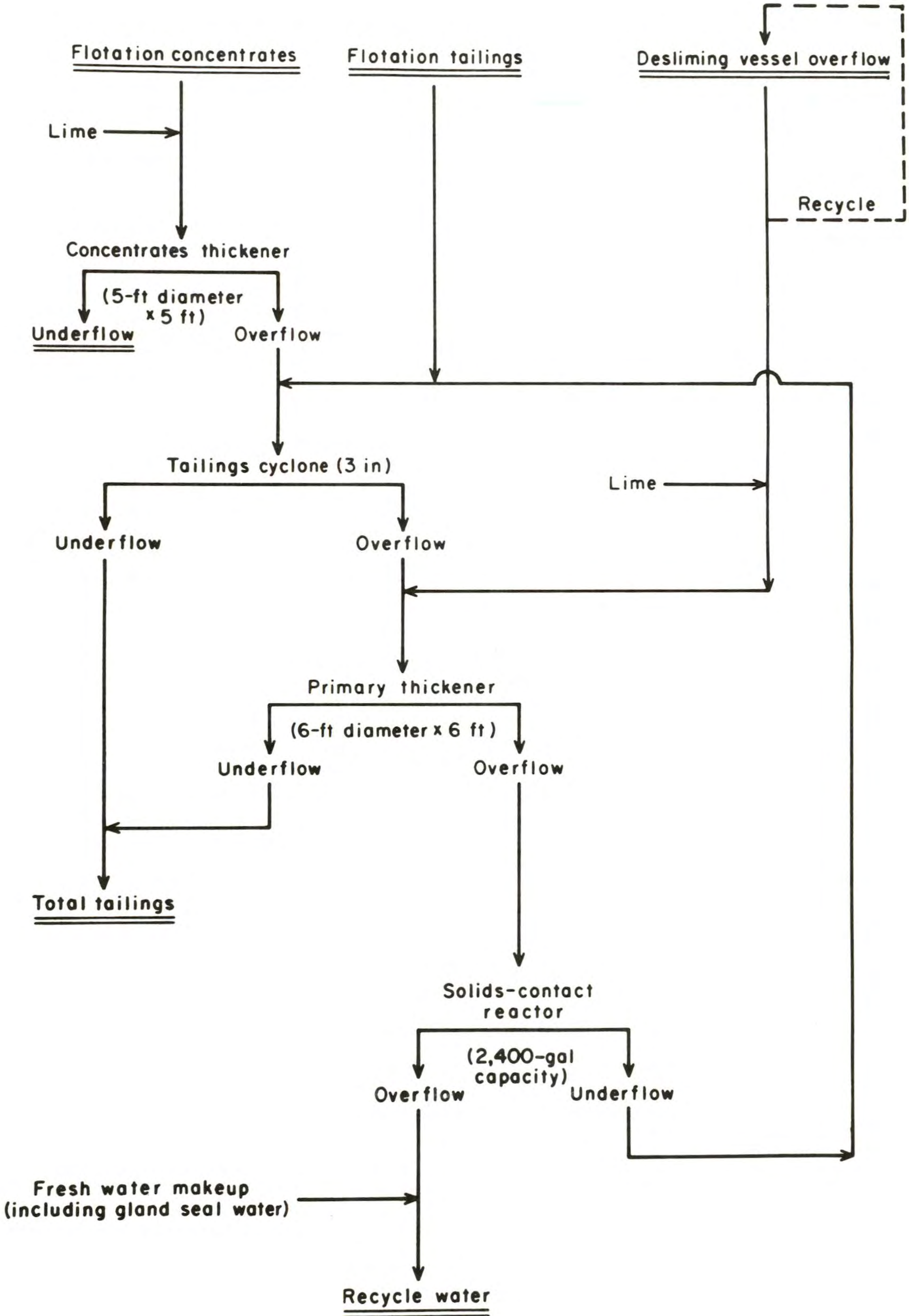

FIGURE 7. - Final water reclamation flowsheet.

second continuous around-the-clock campaign, using reclaimed water. These changes were--(1) one-half of desliming vessel overflow was continuously recirculated, as shown in figure 7, and (2) lime was added to the screen oversize.

Recirculation of a portion of the slime overflow allowed for a flexibility in overflow rates that hitherto had been achieved by water additions. By this substitution, there was the expectation of accomplishing the objective without the necessity of reclaiming an equivalent amount of off-process water, thereby registering savings in both process and water treatments chemicals. While theoretically an optimum rise rate could be obtained by close sizing of the desliming vessel, different settling rates from floccs from the various ore lots indicate that an oversize desliming vessel would be a more prudent choice, with the overflow rate being controlled by water additions. However, even the new rise rate of ~0.90 in/min was still substantially less than the 4 in/min established in prior investigations.

The lime added to the screen oversize progressed through the regrind mill to the flotation conditioners, where it replaced the calcium chloride previously used to activate silica for flotation. The use of lime satisfied the need for Ca^{2+} and provided alkalinity to the pulp, thereby permitting a reduction in the amounts of more expensive NaOH needed to establish the required pH.

CONTINUOUS INVESTIGATION USING INNOVATIVE PROCEDURES

Flotation Procedures, Results, and Discussion

The circuit employed was essentially as previously described and as shown in figure 1. Deviations from that flowsheet consisted of the closed-loop recycling of one-half of the desliming vessel overflow as shown in figure 7.

Grinding power consumption by the rodmill was found to average 13.1 hp-hr/ton, while that of the 3-ft by 3-ft ball mill averaged 14.1 hp-hr/ton through test 6B. Thereafter, an addition of 200 lb of balls to the existing 2,000-lb ball load resulted in an average power draw of 15.9 hp-hr/ton. The regrinding ball mill continued with a 500-lb load of balls and an unmeasured power consumption.

Reagent additions were made at the required points, and these conditions are summarized in table 7. A new ore sample, designated as being from the lower bench of the same mine, was introduced at this stage of the investigation. As previously noted, there was little discernible difference between this and the previous sample.

During the course of 13 operating shifts, as the influence of reclaimed water and of circuit equilibrium became evident, the general trend was towards stabilization of NaOH and fatty acid at levels substantially below that required at the outset.

TABLE 7. - Reagent additions:[1] Flotation and water reclamation (continuous tests)

(Pounds per long ton)

Point of addition..	Sodium hydroxide		Sodium silicate[2]	Trisodium polyphosphate	Tapioca flour		Fatty acid		Hydrated lime		
										Water reclamation	
	Desliming	Rougher flotation	Desliming	Desliming	Desliming	Rougher flotation	Rougher flotation[3]	Scavenger flotation[4]	Rougher flotation	Slimes thickener	Concentrates thickener
Test 4:											
A........	2.2	1.1	0.28	0.28	0.28	1.12	0.90	0.28	0.56	0.88	3.0
B........	1.68	.56	.28	.28	.28	1.12	.90	.28	1.34	.46	3.0
C........	.84	.56	.28	.28	.28	1.12	.90	.28	1.00	.40	3.17
Test 5:											
A........	.84	.56	.28	.28	.28	1.12	.90	.28	1.34	.56	3.09
B........	.84	.56	.28	.28	.28	1.12	.90	.28	1.12	.53	3.07
C........	.84	.84	.28	.14	.28	1.12	.73	.28	1.23	.48	2.91
Test 6:											
A........	.84	.84	.28	.14	.28	1.12	.73	.28	1.23	.22	2.49
B........	.84	.84	.28	.14	.28	1.12	.73	.28	.78	.27	2.41
Test 7:											
A........	.84	.84	.28	.14	.56	1.12	.73	.28	1.12	.19	2.41
B........	.84	.84	.28	.14	.56	1.12	.73	.28	1.34	.27	2.41
C........	.84	.84	.28	.14	.56	1.12	.56	.28	1.12	.16	2.64
Test 8:											
A........	1.12	.84	.28	.14	.56	1.12	.56	.28	1.23	.35	2.41
B........	1.12	.84	.28	.14	.56	1.12	.56	.28	1.12	.38	2.49

[1]Reagent levels are based on a feed rate of 1,100 lb/hr, uncorrected for moisture contents of 2 to 3 percent.
[2]Sodium silicate: 41° Baume, $\simeq$ 8.9 percent Na_2O, 29 percent SiO_2.
[3]Tall oil derivative composed of nearly equal parts of oleic and lineolic acid.
[4]Fatty acid addition: 0.0056 lb/ton of #250 Dowfroth also added.

The metallurgical results obtained during the effective operating shifts are given in table 8. It should be noted that test 4A was made on the startup shift, and stable conditions had not yet been obtained. During the early part of test 6C, a rapid buildup in amount of screen feed was noted, necessitating a shutdown to add 200 lb of balls to the ball mill, and to clean the screens so as to restore their slot openings. Screen cleaning was accomplished by dissolving the cemented material that was obstructing the openings with hydrochloric acid.

TABLE 8. - Metallurgical results (continuous tests)

	Final concentrate				Tailings			Slime		
	Wt-pct	Iron, pct		SiO_2, pct	Wt-pct	Iron, pct		Wt-pct	Iron, pct	
		Analysis	Distribution			Analysis	Distribution		Analysis	Distribution
4B...	33.6	67.2	63.6	1.8	53.4	17.8	26.8	13.0	26.1	9.6
4C...	36.0	66.8	66.0	1.7	53.3	17.4	25.5	10.7	28.9	8.5
5A...	32.6	66.8	61.3	2.2	57.5	19.3	31.3	9.9	26.4	7.4
5B...	27.5	67.1	51.8	2.0	60.1	23.1	39.0	12.4	26.4	9.2
5C...	33.6	66.7	62.8	2.8	62.4	19.8	34.5	4.0	23.7	2.7
6A...	36.3	66.3	68.3	2.8	59.2	17.1	28.8	4.5	22.6	2.9
6B...	32.5	66.2	60.9	3.0	53.1	19.6	29.6	14.4	23.4	9.5
7A...	40.2	65.8	73.7	3.4	55.9	15.4	24.0	3.9	21.0	2.3
7B...	37.9	65.4	71.4	3.9	53.4	14.8	22.8	8.7	23.2	5.8
7C...	41.5	64.2	75.3	6.0	56.2	14.6	23.1	2.3	24.5	1.6
8A...	40.8	62.8	72.8	7.9	49.3	13.6	19.1	9.9	28.8	8.1
8B...	36.3	65.3	68.2	4.3	54.5	15.8	24.7	9.2	26.9	7.1

[1]No sample taken for test 4A--startup shift.

Throughout the campaign, the quality of the slimes discarded in the selective flocculation step was poor; the slimes were of variable weight and generally of high iron content. These conditions persisted in spite of attempts at improvement through lowered dispersant and increased tapioca flour additions. This relatively poor performance in the selective flocculation stage was probably related to both the attempts to economize on water and physical equipment, and upon the characteristics of the ore itself. The effect of closed-loop recycling of one-half of the desliming thickener overflow could not be quantitatively assessed, except that no obvious benefits from this mode of treatment were observed.

As in the developmental tests, little difficulty was experienced in producing a final concentrate of good quality, although iron losses to the tailings remained relatively high. Tailings of somewhat lower iron content were noted after the extra grinding balls were added prior to test 7A. However, an examination of typical product screen analyses, as listed in table 9, fails to provide sufficient indication that a substantially finer grind and hence better liberation was achieved in tests subsequent to 6B. Iron distribution to the tailings paralleled the screen analyses, with the preponderant amount being contained in the minus 400-mesh fraction, after the pattern shown in table 6. Petrographic examinations showed that while most of the iron contained in the tailings was liberated, there was also a moderate amount of locking that persisted in sizes coarser than 10 micrometers.

TABLE 9. - Operating data: Screening and scavenger flotation

(Percent[1])

Product	Test											
	4B	4C	5A	5B	5C	6A	6B	7A	7B	7C	8A	8B
Screen feed:												
Wt-pct.........	40	40	68	45	70	76	59	52	70	82	153	73
Iron:												
Analysis.....	66.2	66.4	61.9	64.3	58.4	59.0	60.0	62.6	57.7	51.0	45.9	54.2
Distribution.	74	74	119	80	115	128	100	90	115	118	200	114
Screen oversize:												
Wt-pct.........	3	2	23	7	11	17	11	4	5	8	33	5
Iron:												
Analysis.....	59.4	61.8	54.5	56.0	45.3	51.1	52.5	53.8	44.9	36.6	36.5	43.9
Distribution.	5	3	35	12	13	24	16	5	6	8	34	7
Screen undersize:												
Wt-pct.........	36	39	45	37	60	[2]60	48	48	65	74	120	68
Iron:												
Analysis.....	66.8	66.6	65.6	65.9	60.7	61.2	61.7	63.2	57.6	52.6	48.5	55.0
Distribution.	68	71	84	69	102	104	84	85	109	110	165	108
Screen feed thickener overflow:												
Wt-pct.........	-	2	1	1	<1	6	13	<1	2	1	2	1
Iron:												
Analysis.....	-	60.2	57.2	56.5	54.5	53.9	55.9	56.5	35.5	54.9	41.3	51.2
Distribution.	-	3	5	1	<1	10	20	<1	2	2	2	2
Scavenger flotation feed:												
Wt-pct.........	36	40	46	38	60	66	61	49	68	75	121	69
Iron:												
Analysis.....	66.8	66.3	65.3	65.7	60.6	60.5	60.4	63.1	57.0	52.7	48.4	55.0
Distribution.	68	73	86	70	102	114	105	86	111	112	167	110
Scavenger flotation froth:												
Wt-pct.........	3	4	14	9	27	30	29	8	30	39	[2]80	33
Iron:												
Analysis.....	61.8	62.7	62.0	61.4	53.0	53.5	53.9	50.1	46.2	33.8	41.2	43.7
Distribution.	5	8	24	16	40	46	44	11	40	37	94	42
Scavenger flotation concentrate:												
Wt-pct.........	34	36	32	29	34	36	32	40	38	41	41	36
Iron:												
Analysis.....	67.2	66.8	66.8	67.1	66.7	66.3	66.2	65.8	65.4	64.2	62.8	65.3
Distribution.	64	66	61	54	63	68	61	74	71	74	73	68
SiO_2--analysis...	1.8	1.7	2.2	2.0	2.8	2.8	3.0	3.4	3.9	6.0	7.9	4.3

[1]Weight and distribution values reported to nearest whole percent.

[2]Sample lost in processing--estimated values.

The only concentrates that were not considered acceptable were obtained in tests 7C and 8A. Two probable explanations are that (1) these tests were deficient in fatty acid collector--except that tests 8B produced an acceptable concentrate with a like amount of fatty acid; or that (2) there was a temporary introduction of ore having characteristics dissimilar to the bulk blend used--. Whether attributable to cause or effect, the feed to the fine screens in these poorer tests was characterized by both higher weight and lowered iron content when compared to more satisfactory tests. Corroboratory data are to be found in table 9.

Data presented in table 10 give some typical surface areas of the concentrates. While the lowest recoveries generally are associated with lowest surface areas, the relationship between recovery and surface area is not well defined. Moreover, there does not appear to be any significant differences in screen analyses between tests reporting significantly different surface areas. Furthermore, it would appear that at equivalent recoveries, the lower bench sample produced concentrates of higher surface area that did the developmental ore. These data, however, are contradictory, and their acceptance as part of a normal pattern should be approached with reservation.

TABLE 10. - Typical flotation final concentrate surface area measurements

Test	Surface area cm^2/g		Fe recovery, percent
	Fisher[1]	Blaine[2]	
5...	2,780	-	51.8
6...	2,870	-	60.9
7A..	3,300	3,000	73.7
7B..	2,590	2,370	71.4
8B..	3,370	3,040	68.2

[1]Fisher sub-sieve sizer surface area.
[2]Blaine method.

Some previous testwork had indicated that lime instead of calcium chloride could be used to provide calcium for activation of silica in the flotation operation. This tenative conclusion was confirmed in the continuous investigation with the $Ca(OH)_2$ (hydrated lime) being added to the screen oversize. The lime and screen oversize next passed through the regrind mill before being joined with incoming fresh feed at the head of the conditioning sequence. The use of lime was considered to be a successful modification contributing to an increased pH and a reduction in the amount of more expensive NaOH needed. However, some difficulty was experienced in maintaining an even rate of lime feeding. Because of this, it was not uncommon for the Ca^{2+} content of the conditioned pulp to show as much as a 10-ppm variation in a single shift.

Table 11 gives the record of calcium determinations made on conditioned flotation feed during each of the operating shifts. From this record, it would appear that precise calcium control during rougher flotation is not as important in the present circuit as had been thought necessary for antecedent circuits. Possibly, benefit may have been derived from elimation of the chloride ion associated with the calcium chloride used previously. A more likely explanation is that use of the scavenger flotation step makes calcium control less critical than in the system producing a finished concentrate in the first, or rougher, stage.

TABLE 11. - Record of Ca^{2+} content of conditioned flotation pulps[1] (ppm)

Test	Time of sample (hours from start of shift)				
	0	2	[2]4	6	8
4A..........	-	-	2	2	2
4B..........	3	9	11	6	13
4C..........	13	3	9	8	6
5A..........	5	6	8	11	8
5B..........	9	4	10	<1	13
5C..........	8	13	13	12	7
6A..........	8	11	6	9	9
6B..........	8	10	8	13	6
7A..........	6	7	7	9	11
7B..........	12	7	7	21	15
7C..........	15	16	12	8	10
8A..........	19	3	14	16	9
8B..........	8	6	15	11	18

[1]Determination made on conditioned pulp on entry to rougher flotation stage.
[2]Product samples usually taken over 30 min. during this period.

Water Reclamation

The water reclamation and recycle system used throughout the continuous test series has been depicted in figure 7. The amounts of lime used in water reclamation are given in table 7, and an evaluation of select process elements and streams with respect to pH, turbidity, and calcium are given in table 12. Equipment description was given previously.

The basic water treatment system begins with the concentrate thickener. An average of 2.7 lb lime per ton of crude ore was added to the thickener for flocculation of the concentrates and to produce an overflow of low turbidity and high calcium content. The overflow was combined with the flotation tailing and the solids-contact reactor underflow to provide the feed to the tailings cyclone. About 33 percent of the total tailings were removed as the cyclone underflow, and the overflow ($\geq$66 percent of the tailings weight) was combined with the desliming vessel thickener overflow for clarification in the slimes (primary) thickener.

TABLE 12. - Compilation of selected test parameters

Test	Parameter and Location of Measurement											
	pH						Turbidity, ppm			Ca^{2+}, ppm[1]		
	Desliming vessel overflow	Rougher flotation feed	Scavenger flotation feed	Concentrates thickener overflow	Slimes-thickener overflow	Solids-contact reactor overflow	Concentrates thickener overflow	Slimes-thickener overflow	Solids-contact reactor overflow	Concentrates thickener overflow	Slimes-thickener overflow	Solids-contact reactor overflow
4B...	11.5	11.7	11.3	11.9	11.5	11.4	790	590	210	104	14.7	14.3
4C...	11.1	11.6	11.4	11.8	11.6	11.3	310	370	210	142	17.6	14.7
5A...	11.1	11.6	11.3	12.0	11.5	11.5	60	400	120	204	19.0	11.3
5B...	11.1	11.6	11.5	12.2	11.4	11.4	60	1,080	210	273	12.9	13.8
5C...	10.9	11.6	11.4	12.2	11.3	11.3	10	650	230	303	12.8	13.4
6A...	10.9	11.7	11.5	12.1	11.4	11.4	20	230	50	274	19.8	15.6
6B...	11.1	11.9	11.5	12.2	11.4	11.5	200	420	140	226	16.4	14.3
7A...	10.7	11.6	11.4	12.0	11.3	11.2	120	340	120	150	14.6	11.4
7B...	10.6	11.6	11.4	12.1	11.4	11.4	170	590	180	145	12.2	12.2
7C...	10.8	11.9	11.5	12.3	11.4	11.5	20	360	160	228	11.9	13.8
8A...	10.8	11.7	11.5	12.1	11.2	11.3	20	230	50	256	12.7	12.7
8B...	10.8	11.6	11.4	12.3	11.3	11.3	30	310	60	274	11.4	10.1

[1]See table 11 for Ca^{2+} contents of rougher flotation feed.

Use of the tailings cyclone was particularly beneficial in that very little froth was transferred from the tailings to the cyclone overflow. Such froth formerly would build up in the circuit and hinder effective operation; with the tailings cyclone operating, it was possible to dispense with froth skimmers.

An average of 0.4 lb lime per ton of crude ore also was added to the slimes to effect a substantial removal of the solids in the slimes thickener. The overflow then was pumped to the solids-contact reactor for further clarification before recycling to the flotation system. Average values of select parameters important to the evaluation of the process streams for the test series are given in table 13.

TABLE 13. - Average values for select water reclamation parameters

Process stream	pH	Turbidity, ppm	Ca^{2+}, ppm
Slimes-thickener overflow......................	10.9	450	14.7
Solids-contact reactor overflow (finished water)	11.4	150	13.1
Concentrates-thickener overflow................	12.1	150	215

During test 7, the necessary samples and measurements were taken for calculation, and the resulting ore and water balance is given in table 14. The water flows also are summarized in figure 8. The total internal flow of about 11 gpm (gallons per minute) is considerably below 18 to 20 gpm of the earlier investigation,[21] and represents a substantial savings in water use and handling. The diminished use of water, while using much of the same equipment, allowed for longer retention times and may account for the low turbidity values contained in table 13. In any event, the water reclamation system performed exceptionally well, and the product was a recycled water with pH, turbidity, and Ca^{2+} content compatible with all aspects of the ore treatment system.

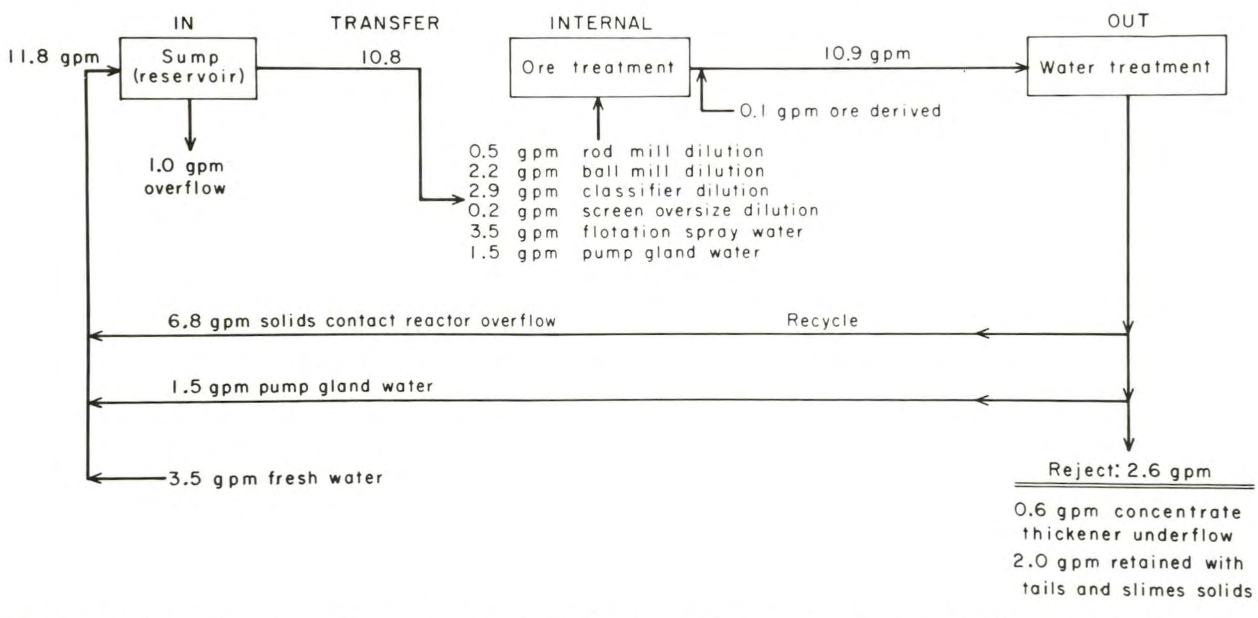

FIGURE 8. - Diagrammatic representation of water balance: feed rate, 1,100 pounds per hour.

[21]Work cited in footnote 18.

TABLE 14. - Ore and water balance

Product	Wt-pct	Solids, pct	Iron, pct: Analysis	Iron, pct: Distribution	Water gpm
Feed[1]	100.0	97.4	35.4	100.0	0.1
Rodmill dilution	-	-	-	-	.5
Rodmill feed	100.0	78.4	35.4	100.0	.6
Classifier underflow	180.0	77.7	52.1	264.8	1.1
Ball mill dilution	-	-	-	-	2.2
Ball mill feed	280.0	60.6	46.1	364.8	3.9
Classifier dilution	-	-	-	-	2.9
Classifier overflow	100.0	27.2	35.4	100.0	5.7
Desliming vessel feed	102.7	18.7	35.1	101.8	9.5
Desliming vessel underflow	97.3	52.0	35.7	98.2	1.9
Desliming vessel overflow	5.4	1.5	24.5	3.6	7.6
One-half desliming vessel overflow	2.7	1.5	24.5	1.8	3.8
Reground screen oversize	8.2	38.5	36.6	8.5	.3
Rougher flotation feed	139.3	44.4	36.5	143.5	3.7
Rougher flotation concentrates	83.3	37.6	53.8	44.4	3.4
Cleaner flotation middling		22.0	49.7	75.9	
Flotation spray dilution	-	-	-	-	3.5
Flotation tailings	56.0	23.6	14.6	23.2	3.9
Screen feed thickener feed	83.3	34.5	51.1	120.3	3.4
Screen feed thickener overflow	1.4	2.4	54.9	2.2	1.2
Screen feed thickener underflow	81.9	44.7	51.0	118.1	2.2
Screen undersize	73.7	43.2	52.6	109.6	2.1
Screen oversize	8.2	66.1	36.6	8.5	.1
Regrind mill dilution	-	-	-	-	.2
Regrind mill feed	8.2	38.5	36.6	8.5	.3
Scavenger flotation feed	75.1	32.8	52.7	111.8	3.3
Scavenger flotation froth	33.8	31.9	38.6	36.8	1.5
Scavenger flotation concentrates	41.3	33.6	64.2	75.0	1.8
Concentrates thickener underflow	41.3	60.4	64.2	75.0	.6
Concentrates thickener overflow	-	-	-	-	1.2
Tailings cyclone feed	57.4	15.9	-	-	6.5
Tailings cyclone underflow	18.9	46.8	-	-	.5
Tailings cyclone overflow	38.5	12.0	-	-	6.0
Slimes thickener feed	41.2	8.2	-	-	9.8
Slimes thickener underflow	39.8	35.0	-	-	1.5
Slimes thickener overflow	1.4	.4	-	-	8.3
Solids-contact reactor underflow	1.4	2.0	-	-	1.5
Solids-contact reactor overflow	-	-	-	-	6.8
Gland seal dilution	-	-	-	-	1.5
Makeup dilution	-	-	-	-	3.5
Waste water	-	-	-	-	1.0
Total tailings	58.7	38.1	15.1	25.0	2.0

[1]Feed rate at 1100 lb/hr.

Reagent Costs

The continuous test series yielded the lowest reagent costs ever achieved in tests conducted at the Twin Cities Metallurgy Research Center using the calcium activation system for flotation of silica. Compared to a 1968 estimate of $0.95 per ton of concentrates, for flotation reagents alone, the present system, in test 7A, required an equivalent cost of under $0.80 for both flotation and water reclamation chemicals (table 15). The same test yielded concentrates of 40.5 percent weight, of 65.8 percent Fe and 3.4 percent SiO_2 contents, and recovered 74.1 percent of the iron in the feed. Somewhat lower costs were achieved in tests 7C and 8A, but the concentrates were not acceptable.

TABLE 15. - Estimated reagent cost per ton of concentrates for individual processes[1]

Test	Desliming	Rougher flotation	Scavenger flotation	Water reclamation	Total
4B....	$0.325	$0.572	$0.089	$0.103	$1.089
4C....	.215	.525	.083	.099	.922
5A....	.238	.591	.092	.112	1.033
5B....	.269	.660	.104	.126	1.159
5C....	.198	.550	.089	.101	.938
6A....	.182	.508	.082	.075	.847
6B....	.255	.555	.092	.083	.985
7A....	.205	.452	.074	.064	.795
7B....	.219	.490	.079	.071	.859
7C....	.201	.401	.072	.068	.742
8A....	.229	.409	.073	.068	.779
8B....	.259	.459	.082	.080	.880

[1]Cost basis; cents/lb reagent (1968 estimates):
Sodium hydroxide........... 3.82
Sodium tripolyphosphate.... 7.80
Sodium silicate............ 2.30
Tapioca flour.............. 6.00
Lime....................... 1.00
Fatty acid.................10.00
Frother....................30.00

It should be noted that these costs were calculated using 1968 price estimates for the reagents. Reagent costs have responded to inflationary pressures since the initial calculation so that costs of table 15 are in a true relationship to earlier work, but may not correspond to present pricing.

SUMMARY

This investigation was undertaken both to explore use of innovative procedures to complement a flotation system employing anionic flotation of calcium activated silica from iron ore, and as a means of accomplishing cost reductions. For reasons indicated in the body of the report, the data should be interpreted as being relative and directional, rather than being absolute in a quantitative sense. With these restrictions governing, the following observations are made:

(1) The flotation system is operative with a primary feed as coarse as 80 percent minus 400 mesh, as compared to >90 percent minus 40 mesh employed in some previous investigations. Even considering the necessity of some regrinding, the reduction in grinding power accomplished as a consequence of coarse primary grinding amounts to 10 to 20 percent of that obtained in previous investigations. However, since the flowsheet depends upon the fine screening of intermediate concentrates, the failure of these screens to function properly over sustained periods without blockage is an operating handicap. Screen blockages appear to be associated with the fatty acid system, because a similar use of screens in a cationic system during a subsequent investigation did not result in blockages.

(2) Autogenous pebble mill grinding can be substituted for ball milling without apparent effect on the desliming or flotation processes. Despite higher power requirements for the pebble mill grinding substantial operating savings are possible through lowered metal (ball) consumption.

(3) The use of a scavenger flotation treatment to advance submarginal concentrates to acceptability allows for a minimum of operating control and a lessened requirement to maintain pulp calcium ion concentrations within prescribed limits. However, experiments in which various proportions of the total fatty acid required were distributed between the rougher and scavenger flotation stages indicate a preference for adding the larger portion to the rougher. In contrary situations, where the larger portion of fatty acid was added to the scavenger stage, large circulating loads accumulated to overload the fine screen and the regrinding mill.

(4) A water reclamation system compatible with the anionic method for flotation of calcium activated silica was developed and improved. Water clarification was accomplished by controlled additions of lime.

A few brief tests were made that indicated that hard water used for makeup could contribute to water clarification with an accompanying reduction in lime requirements.

These investigations showed that the cost of flotation chemicals could be reduced by amounts equal to or exceeding the cost of chemicals used in water reclamation.

(5) Modification in the desliming circuit to include variations in entrance solids in chemical additions and rise rates to include partial recirculation of overflows had little detectable beneficial effect. Any benefits obtained by these modifications were probably indirect, and related to water consumption and water reclamation.

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