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Beneficiation of a Western Mesabi Nonmagnetic Taconite After Reduction Roasting With Lignite

By Roy E. Peterson and John E. Moy

UNITED STATES DEPARTMENT OF THE INTERIOR

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UNIT OF MEASURE ABBREVIATIONS USED IN THIS REPORT

Btu/h	British thermal unit per hour	in	inch
Btu/ton	British thermal unit per ton	kW•h/ton	kilowatt hour per ton
°C	degree Celsius	lb	pound
cm	centimeter	lb/h	pound per hour
ft	foot	lb/ton	pound per ton
ft^3	cubic foot	min	minute
g	gram	µm	micrometer
G	gauss	Oe	oersted
gal/min	gallon per minute	pct	percent
h	hour	ppm	part per million
hp•h/ton	horsepower hour per ton	rpm	revolution per minute
Hz	hertz	T	tesla

BENEFICIATION OF A WESTERN MESABI NONMAGNETIC TACONITE AFTER REDUCTION ROASTING WITH LIGNITE

By Roy E. Peterson[1] and John E. Moy[1]

ABSTRACT

The Bureau of Mines has been developing and evaluating methods for beneficiating the large deposits of nonmagnetic (oxidized) taconite located on the western Mesabi Range. In this segment of the research, a sample representing about 0.5 billion tons of nonmagnetic taconite from near Grand Rapids, Minn., was beneficiated using a reduction roasting, magnetic separation, flotation process. The sample contained 32.1 pct iron and 49.1 pct silica. Tests were conducted both on a bench scale and with a 1,000-lb/h pilot plant. In the bench-scale research a concentrate containing 69.0 pct iron and 4.5 pct silica was produced with an iron recovery of 82.1 pct. In the pilot plant, reduction roasting was conducted in a 35-ft kiln using lignite as the reductant and major source of fuel. Dust from the kiln exhaust was pelletized and recycled to the kiln. The roasted taconite was beneficiated by magnetic separation and flotation to produce a product containing 67.9 pct iron, 5.5 pct silica, and 69.7 pct of the iron in the kiln feed. The cyclone separation in the grinding circuit was sharpened by demagnetizing the cyclone feed in a 400-Hz, 1,000-Oe magnetic field.

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INTRODUCTION

This report gives the results of the last of a group of six investigations that the Bureau of Mines Twin Cities Research Center has completed on the beneficiation of nonmagnetic taconites (oxidized taconites)[2] located on the western Mesabi Range in Minnesota. This effort was part of the Bureau's objective to help assure the Nation of an adequate, continuing iron ore supply. Today, most of the domestic iron-bearing materials are derived from the magnetic taconites, which are beneficiated by magnetic separation. When these deposits become depleted, it will be necessary to use the nonmagnetic taconites. On Minnesota's western Mesabi Range alone, these include an estimated 10 billion tons[3] of material assaying 30 to 40 pct Fe.

The deposits investigated in these Bureau studies lie in a 24-mile-long segment of the Biwabek iron formation within the western Mesabi Range and between Keewatin and Grand Rapids, Minn. This area was divided into three areas, designated as WMR-1, WMR-2, and WMR-3, as shown in figure 1. A sample was taken from each area and subjected to two alternative technologies: selective flocculation and cationic flotation (1)[4] and reduction roasting, magnetic separation, and flotation. The test objectives were to optimize both technologies and to obtain data on concentrating the taconites to a product containing about 5 pct SiO_2.

The results using selective flocculation and cationic flotation have already been reported on the WMR-1 (6), WMR-2 (5), and WMR-3 (4) samples. Results using reduction roasting, magnetic separation, and flotation have been reported for the samples of WMR-1 (9) and WMR-2 (8), and this report describes the results of the WMR-3 investigation. In this process the ore is roasted in a moderately reducing atmosphere to convert the nonmagnetic iron oxide, Fe_2O_3, to the magnetic oxide (magnetite), Fe_3O_4. Lignite, the solid reductant source in the Bureau tests, reacts to generate CO and H_2, which are the gaseous reductants. Lignite is also pulverized to fuel a burner to supply heat for the roasting. After the iron oxide is converted to the magnetic form (Fe_3O_4), it is concentrated by magnetic separation and flotation.

[2]Throughout this report, "nonmagnetic taconite" is synonymous with "oxidized taconite" and refers to taconite in which the iron oxides are not recovered by a drum-type separator having a maximum field intensity of 3,000 G (0.3 T) at its surface.

[3]Throughout this report, tons refers to long tons (2,240 pounds).

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

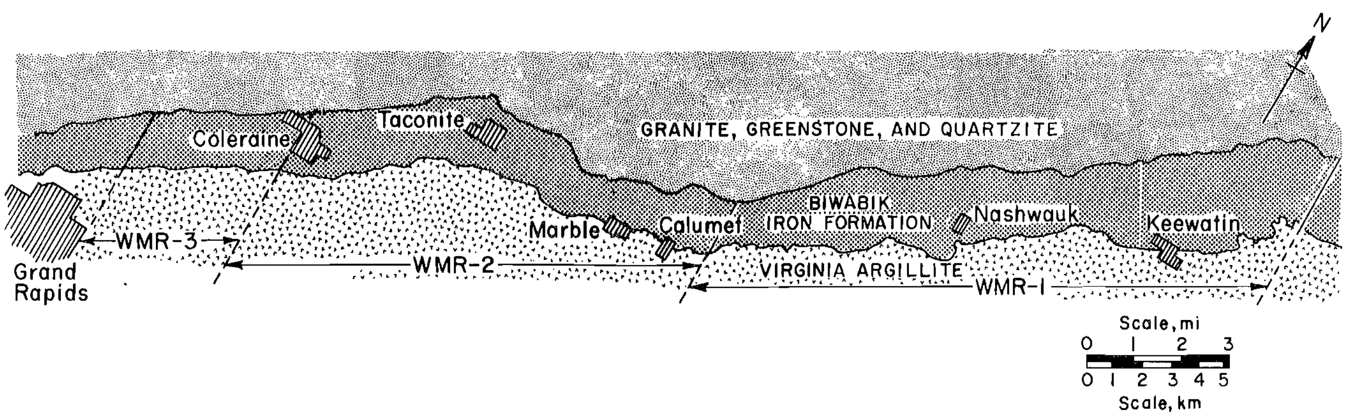

FIGURE 1. - Western Mesabi Range sample areas. Samples were taken from the Biwabik iron formation (darkly shaded area), which is bounded by deposits of granite, greenstone, and quartzite to the north, and Virginia Argillite to the south.

RAW MATERIALS

TACONITE

An 800-ton sample was obtained for this research from the WMR-3 area. This area between Grand Rapids and Coleraine, Minn., contains about 0.5 billion tons of nonmagnetic taconite. The sample was crushed to minus 5/8 in and blended by repeated bedding. A portion was removed from this stockpile and crushed to minus 3/8 in (table 1). It contained 32.1 pct Fe, 1.2 pct Fe^{++}, and 49.1 pct SiO_2 (table 2). Petrographic studies indicated that the only major iron minerals were hematite and goethite, which occurred in a ratio of 2.1:1. The minor iron minerals (magnetite, iron silicate, and carbonates) totaled less than 3 pct of the sample. The gangue was substantially all quartz.

TABLE 1. - Size and iron analysis of taconite sample, percent

Sieve size[1]	Weight	Iron	Iron distribution
Plus 3/8 in	1.7	37.9	2.0
Minus 3/8 in plus 3 mesh	9.8	32.1	9.8
Minus 3 plus 4 mesh	16.4	33.2	17.0
Minus 4 plus 10 mesh	32.2	32.4	32.6
Minus 10 plus 28 mesh	16.3	31.5	16.0
Minus 28 plus 65 mesh	12.2	31.5	12.0
Minus 65 plus 100 mesh	1.3	33.9	1.4
Minus 100 plus 200 mesh	2.3	34.7	2.5
Minus 200 plus 325 mesh	1.6	30.7	1.5
Minus 325 plus 500 mesh	1.6	24.8	1.2
Minus 500 mesh	4.6	27.8	4.0
Total or composite	100.0	32.1	100.0

[1]Standard Tyler sieve size.

TABLE 2. - Chemical analysis of nonmagnetic taconite feed sample

Component	pct	Component	ppm
Carbon	0.24	Antimony	<1
Iron:		Arsenic	<5
Total	32.1	Cadmium	.14
Ferrous	1.2	Chromium	30
Manganese	.08	Cobalt	<5
Phosphorus	<.010	Copper	<20
Sulfur	.014	Fluorine	<10
LOI[1] at 400° C	.8	Lead	<.01
LOI[1] at 1,000° C	2.1	Mercury	<.1
Al_2O_3	.43	Nickel	<50
CaO	.62	Selenium	<5
K_2O	.07	Silver	<5
MgO	.21	Zinc	50
Na_2O	.09		
SiO_2	49.1		
TiO_2	<.2		
V_2O_5	<.002		

[1]Loss on ignition.

LIGNITE

A North Dakota lignite was used as the burner fuel and as the reductant. The "as received" lignite was crushed to minus 2.5-in and had the proximate analysis and heat value reported in table 3. Lignite for the reductant was screened to remove the minus 0.25-in fraction to minimize the amount of lignite blown out of the kiln. The minus 0.25-in material was combined with unscreened lignite and used as feed to the pulverized-burner system.

TABLE 3. - Analysis and heating value of lignite, as-received basis

Proximate analysis:	
Moisture....................pct..	33.4
Volatile matter............pct..	29.1
Fixed carbon...............pct..	31.1
Ash........................pct..	6.4
Heating value...........Btu/lb..	7,260
Ultimate analysis:	
Ash........................pct..	6.4
Carbon.....................pct..	42.8
Hydrogen...................pct..	6.6
Nitrogen...................pct..	0.6
Oxygen.....................pct..	43.0
Sulfur.....................pct..	0.6

BENCH-SCALE ROASTING AND BENEFICIATION

EQUIPMENT AND PROCEDURES

A portion of the 800-ton taconite sample was roasted for the bench-scale beneficiation tests in an 18-cm-ID drum used in previous investigations (10). The drum had four lifter bars, was externally heated in an electric furnace operated with a programmable temperature controller, and was rotated at 5 rpm. Forty grams of minus 3- plus 14-mesh lignite was placed on top of a 2,000-g sample charge before the roasting cycle was begun. The sample charge and lignite were heated to 800° C to maintain a rapid reaction rate between the carbon in the lignite and CO_2 in the atmosphere to generate sufficient CO to reduce the ferric oxides. A simulated mixture of natural gas combustion products, composed of 70.6 pct N_2, 19.7 pct H_2O, and 9.7 pct CO_2, was passed through the drum during the roasting test. At the end of the test, the sample was cooled in a nitrogen atmosphere to 100° C.

Beneficiation (fig. 2 and table 4) consisted of three stages of wet magnetic separation and two stages of flotation with intermediate grinding. All steps were performed batchwise, using a Sala[5] magnetic separator, an 18- by 24-cm rodmill, and a 600-g Wemco laboratory flotation cell. The magnetic separator had a 20.3-cm-diameter drum with a maximum field strength of 1,300 G at its surface.

The roasted taconite was crushed to minus 20 mesh (fig. 3) for the cobber magnetic separation, and the magnetic fractions from the cobber and rougher separations were ground in the rodmill to improve liberation for the respective subsequent separations. By rejecting gangue directly at each successive separation, grinding costs are kept low compared to processes where all the ore is initially ground to the fineness ultimately needed to achieve the required grade. In the last and finest grind, the scavenger flotation feed, consisting only of 11.1 pct of the cobber feed weight, was ground to 87 pct minus 325 mesh.

The cobber, rougher, and finisher nonmagnetic products, combined with the scavenger flotation tailing,[6] constituted the total tailings. The final concentrate was the combined primary and scavenger concentrates.

BENCH-SCALE RESULTS

A series of bench-scale tests were performed to develop a flowsheet and a data base for evaluating the subsequent pilot plant results. Initially the taconite

[5]Reference to specific products does not imply endorsement by the Bureau of Mines.

[6]In this report, "tailing" is used to refer to the residue from one step in the beneficiation process, and "tailings" is the combined residue from various steps.

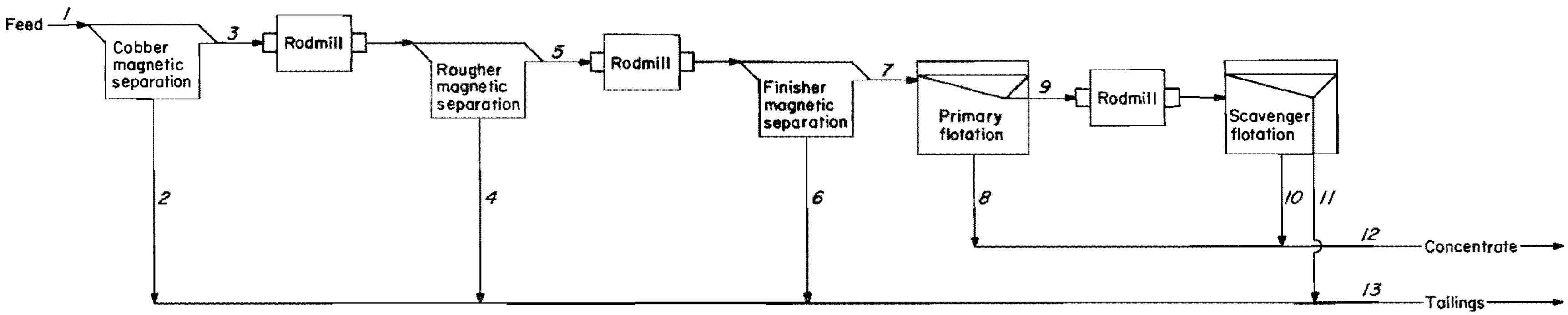

FIGURE 2. - Flowsheet for bench-scale beneficiation tests.

TABLE 4. - Material balance for bench-scale beneficiation flowsheet shown in figure 2, percent

Analysis	Streams												
	1-Feed	2-Cobber tailing	3-Cobber concentrate[1]	4-Rougher tailing	5-Rougher concentrate[1]	6-Finisher tailing	7-Finisher concentrate	8-Primary flotation concentrate	9-Primary flotation tailing[1]	10-Scavenger flotation concentrate	11-Scavenger flotation tailing	12-Combined concentrate[1]	13-Combined tailings[1]
Weight distribution	100.0	33.0	67.0	15.6	51.4	5.2	46.2	35.1	11.1	4.7	6.4	39.8	60.2
Total iron...	33.5	5.7	47.2	6.0	59.7	11.8	65.0	69.3	51.6	66.6	40.5	69.0	10.0
Ferrous iron.	12.9	2.6	NA	2.7	NA	4.4	24.8	NA	NA	NA	NA	NA	NA
Total iron distribution	100.0	5.6	94.4	2.8	91.6	1.8	89.8	72.7	17.1	9.4	7.7	82.1	17.9

NA Not available.
[1]Not sampled. Values calculated by product formulas.

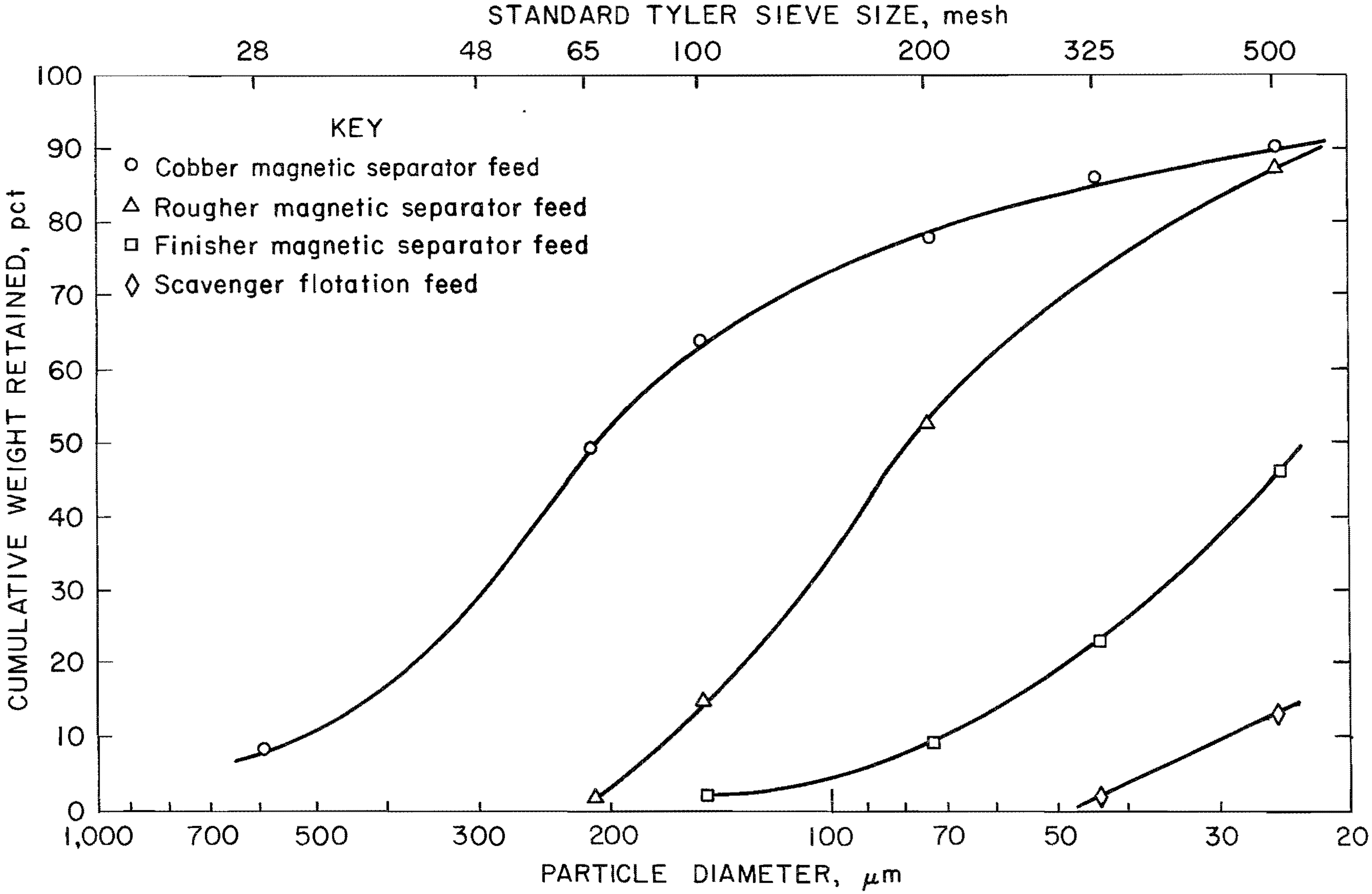

FIGURE 3. - Size analyses for bench-scale tests.

was roasted to an Fe^{2+}/Fe ratio of 0.40 (ratio for Fe_3O_4 is 0.33). This material was concentrated in the magnetic separation circuit to produce a concentrate containing 65.0 pct Fe and 13.4 pct SiO_2 (fig. 2 and table 4). In the subsequent flotation separations, this material was upgraded to yield a final concentrate assaying 69.0 pct Fe and 4.5 pct SiO_2. The usage of an amine collector (Arosurf MG-98A) was 0.14 lb/ton; the consumption of an alcohol frother (Dowfroth 250) was 0.017 lb/ton.[7] The Fe recovery was 82.1 pct, with the Fe loss being 10.2 pct in the magnetic separation system and 7.7 pct in the flotation circuit.

The observations from petrographic examination of the process products, when evaluated with reference to the size distributions of the feeds shown in figure 3, indicated that incomplete liberation was only a secondary impediment to better recovery. In the cobber tailing the typical locked iron was minus 200-mesh magnetite. However, 75 pct of the iron in this tailing was liberated but was in the nonmagnetic, hematitic form. This indicates that incomplete roasting to magnetite was the major deterrent to better recovery at the magnetic separator. High iron liberation, 90 pct, was also evident in the scavenger flotation tailing. Improvements in the sharpness of flotation selectivity would be the best way to increase iron recovery at this point in the process.

PILOT PLANT ROASTING

EQUIPMENT AND PROCEDURES

The flowsheet for the reduction roasting of taconite in a 35-ft-long rotary kiln is shown in figure 4. The material balance is tabulated in table 5, and the equipment is briefly described in table 6. The feed, consisting of taconite, lignite, and pelletized recycle dust, was fed into the cold end of the kiln. A relatively high kiln rotational speed of 2.5 rpm was maintained to prevent backflow and spillage at this point.

The kiln was lined from the feed end to the midpoint with fire clay bricks, and from there to the discharge or hot end with 70-pct-alumina bricks. A 4.5-in annular dam at the discharge end maintained the desired depth of material. A coal burner and a premix natural gas burner, both inserted in the discharge end, provided the needed heat (7, 9). The gas burner preheated the kiln, ignited and maintained ignition of the pulverized lignite from the coal burner, and regulated the kiln temperature. An automatic temperature control system modulated the gas flow to sustain the temperature near 820° C at a thermocouple location 4.5 ft from the hot end of the kiln. A hammermill, in closed circuit with a classifier using preheated air for drying, ground the lignite for the coal burner. The pulverized lignite was 48 pct minus 200 mesh and contained 13 pct moisture.

An exhaust fan kept the kiln under a slight negative pressure, thus preventing fugitive dust losses. A cyclone dust collector removed most of the dust entrained in the exhaust. Most of the cyclone dust was pelletized, using only water as the binder, and recycled to the kiln. Additional particulates and water-soluble gases were removed from the cyclone exhaust by a Venturi-type wet scrubber.

A seal, consisting of spring-loaded graphite blocks mounted around the discharge end of the kiln, prevented the influx of air past the area where the rotating shell intersected the stationary burner housing. To prevent the hot product from oxidizing in air, it was quenched by terminating the discharge chute under the water surface in the pool of a screw classifier. The classifier overflow, containing undesirable slimes, was rejected to the tailings pond, while the classifier sands were stockpiled in a covered concrete bin for subsequent beneficiation tests.

[7]Reagent rates are based on long tons of cobber feed.

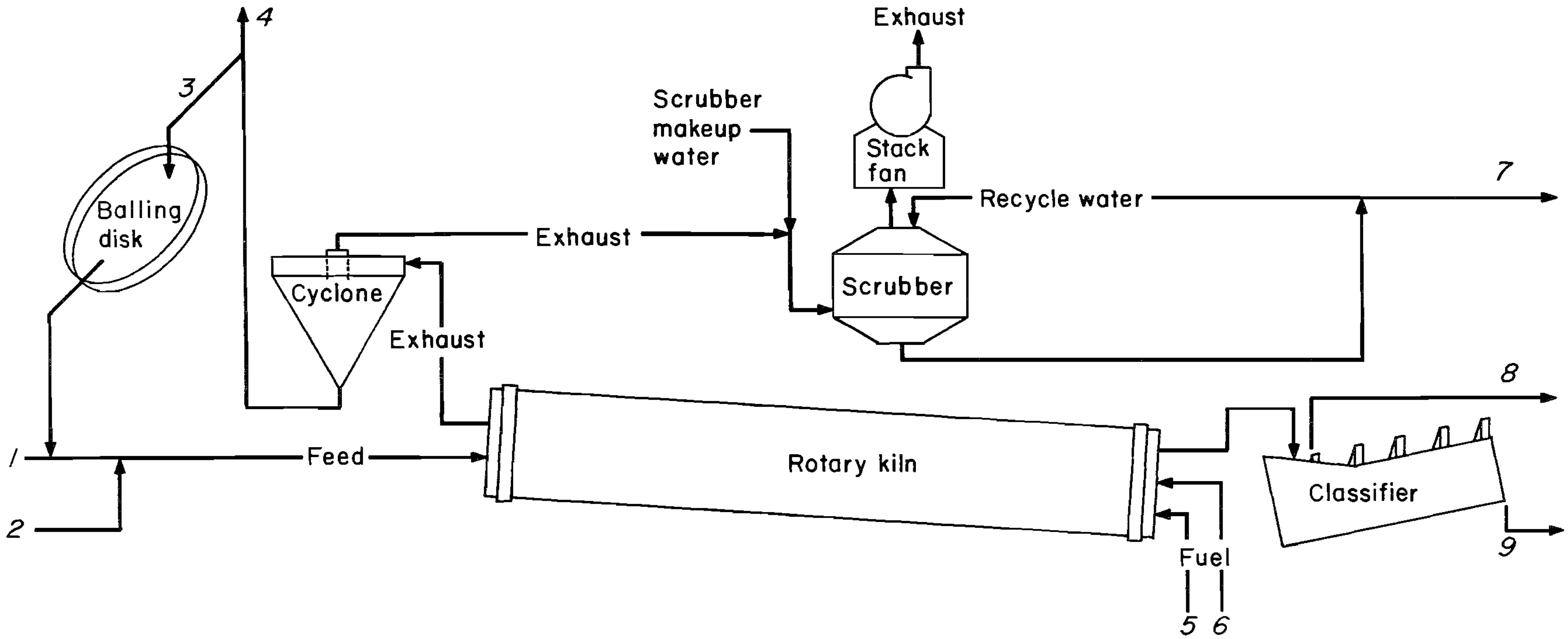

FIGURE 4. - Reduction roasting flowsheet.

TABLE 5. - Material balance for reduction roasting flowsheet shown in figure 4[1]

Analysis	Streams								
	1-Taconite	2-Lignite	3-Recycle dust	4-Excess dust	5-Natural gas	6-Pulverized lignite	7-Scrubber sludge	8-Classifier slimes	9-Roasted taconite
Solids rate........lb/h..	1,089	36	144	3	NAp	96	17	18	1,023
Iron...............pct..	32.1	NAp	26.7	26.7	NAp	NAp	30.8	22.0	33.2
Fuel..........MM Btu/h..	NAp	0.26	NAp	NAp	0.55	0.70	NAp	NAp	NAp

NAp Not applicable.
[1]Additional material balance data are shown in table 7.

TABLE 6. - Description of roasting equipment

Equipment	Description
Balling disk...........	5-ft diameter
Cyclone dust collector.	3-ft diameter
Rotary kiln............	34-in ID by 35 ft long
Wet scrubber...........	Stainless steel, Venturi type with 75-hp fan
Screw classifier.......	2 by 11 ft

Samples of the roasted product were taken from the classifier every 2 h. As soon as a sample was dried, a quick approximate measurement was made of its conversion to magnetite with a Ramsey coil magnetics detector. Then a portion of the sample was ground to minus 200 mesh and magnetically separated with a Davis tube (11). The Davis tube feed and products were analyzed for total and ferrous iron. These analytical data were then used to calculate the total to ferrous iron ratios and the Davis tube iron recoveries. With this information, the engineers in charge could regulate the roasting conditions to adjust the extent of the reduction as necessary.

ROASTING RESULTS

In an around-the-clock test conducted in the reduction roasting pilot plant, oxidized iron in the taconite was converted to magnetite to render it amenable to magnetic separation. Test results reported herein are averages from the 119-h steady state operation. The taconite was fed at an average rate of 1,089 lb/h with 36 lb/h of lignite as the reductant (tables 5 and 7). In addition, 96 lb/h of pulverized lignite was consumed by the lignite burner. Relating the fuel to feed rate, the total fuel consumption (including reductant) was 3.1 MM Btu/ton of taconite feed. Of this, 64 pct was supplied by the lignite used as the reductant and burner fuel, with the rest derived from the gas burner.

The temperature profile in figure 5 shows the approximate retention time to a particular point in the kiln. Since the temperature curve projects upwards toward the burner end past the last and hottest thermocouple location, it seems likely that the actual peak temperature was 40° C higher than the 820° C measured at this thermocouple. The time scale on the

TABLE 7. - Processing rates and distribution of materials

Material	Rate,[1] lb/h	Analysis, pct		Distribution, pct	
		Fe	Fe^{2+}	Weight	Fe
Input:					
Taconite..................	[2]1,089	32.1	1.2	99.3	99.7
Ash from lignite[3]........	8	11.6	NA	.7	.3
Total.................	1,097	NAp	NAp	100.0	100.0
Output:					
Roasted taconite.........	1,023	33.2	10.7	93.3	97.1
Classifier overflow......	18	22.0	5.9	1.6	1.1
Scrubber sludge..........	17	30.8	1.7	1.5	1.5
Cyclone dust.............	3	26.7	2.1	.3	.2
LOI from ore.............	22	NAp	NAp	2.0	.0
O_2 loss from reduction...	14	NAp	NAp	1.3	.0
Total.................	1,097	NAp	NAp	100.0	100.0

NA Not available. NAp Not applicable.

[1]Moisture-free basis.

[2]Weight obtained by difference between the total product weight and the calculated ash weight.

[3]Feed consumption of lignite was 132 lb/h of which 36 lb/h also served as reductant.

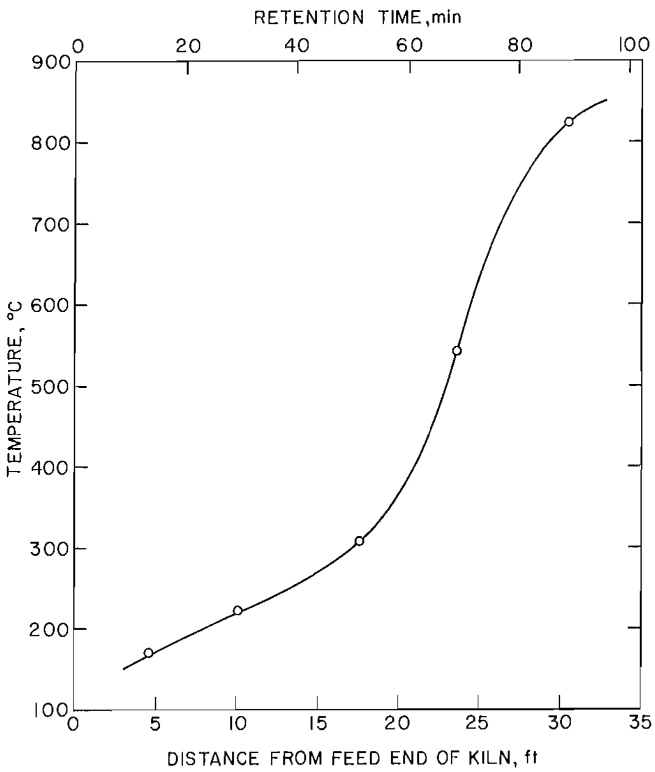

FIGURE 5. - Roasting temperature profile and retention time.

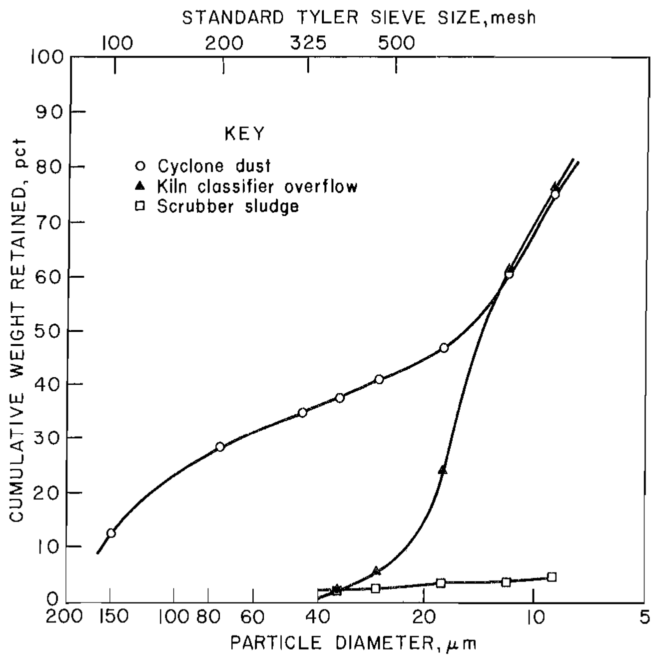

FIGURE 6. - Size analyses of fine-particulate outputs from reduction roasting plant.

figure indicates the calculated length of time that the charge has been in the kiln. For instance, when the charge reached the 700° C point, it was 26 ft into the kiln and had been heating up for 76 min. Since the total retention time was 102 min, the time it was retained at temperatures above 700° C is the difference, 26 min.

Samples of the fine-particulate outputs from the roasting process (cyclone dust, scrubber sludge, and classifier slimes) were sized in a Warman cyclosizer, and the results are plotted in figure 6. Dust recovered at the cyclone collector from the kiln exhaust was mostly hematite and therefore nonmagnetic. Its unreduced state indicated that it became entrained at the feed end of the kiln, probably as the ore tumbled in. During the first part of the roasting run preceding the 119-h data averaging period, the dust was not pelletized and recycled, and the cyclone dust collection rate was 111 lb/h. After pelletizing and recycling was initiated the rate rose to 147 lb/h owing to the attrition of the pellets, but the increased volume of dust was easily handled. Through the use of the dust pelletization circuit, it was possible to successfully roast the cyclone dust.

The scrubber sludge particulate was very fine, 95 pct minus 9 µm. It was mostly nonmagnetic with an Fe^{2+}/Fe ratio of only 0.06. Because of the small quantity and the expected difficulty of beneficiating this material, it was discarded as a tailing product.

The slimes removed from the roasted product in the overflow of the quenching classifier contained 92 pct minus 25-µm particulate. It was advantageous to remove this fraction because of its threat to the quality of the final concentrate. Though concentrating a 15-g sample in a Davis tube magnetic separator did yield a substantial iron recovery (69.6 pct), the concentrate contained excessive silica (36.1 pct).

The water in the classifier overflow had a pH of 11.4, high in comparison to the pH of 2.6 in the scrubber water (table 8). The scrubber water becomes acid because of the sulfur oxides derived from burning the coal; the classifier water becomes basic from basic constituents in

TABLE 8. - Chemical analysis of filtrates from slurry effluents from roasting plant, milligrams per liter

	Scrubber sludge[1]	Classifier slimes[2]		Scrubber sludge[1]	Classifier slimes[2]
Aluminum	73	2.0	Manganese	3.8	<.05
Antimony	<1	<1	Mercury	.002	.002
Arsenic	NA	<.01	Nickel	.28	<.1
Cadmium	<.01	<.01	Selenium	<.002	<.002
Chromium	<.1	<.1	Silver	<.01	<.01
Copper	.49	.07	Zinc	2.0	<.05
Iron	63.8	.4	Ca^{2+}	280	320
Lead	NA	.22	SO^{2-}	1,400	200
Magnesium	47.4	.3			

NA Not available. [1]Flowrate 2 gal/min; pH 2.6. [2]Flowrate 5.3 gal/min; pH 11.4.

TABLE 9. - Size and iron analysis of roasted taconite

Sieve size[1]	Weight, pct	Iron, pct	Iron distribution, pct	Fe^{2+}/Fe
Plus 3/8 in	1.0	16.7	0.5	0.34
Minus 3/8 in plus 3 mesh	8.9	31.4	8.4	.21
Minus 3 mesh plus 4 mesh	15.3	33.7	15.5	.26
Minus 4 mesh plus 10 mesh	28.7	33.7	29.1	.33
Minus 10 mesh plus 28 mesh	16.6	33.7	16.9	.39
Minus 28 mesh plus 65 mesh	12.4	35.1	13.1	.27
Minus 65 mesh plus 100 mesh	2.6	37.2	2.9	.37
Minus 100 mesh plus 200 mesh	4.5	36.2	4.8	.35
Minus 200 mesh plus 325 mesh	3.4	31.4	3.3	.33
Minus 325 mesh plus 500 mesh	3.0	27.3	2.4	.31
Minus 500 mesh	3.6	28.0	3.1	.30
Total or composite	100.0	33.2	100.0	0.31

[1]Standard Tyler sieve size.

the lignite ash and taconite. It may be feasible to use the overflow to neutralize the scrubber water.

The roasted taconite product contained 97.1 pct of the iron contained in the kiln feed. It analyzed 33.2 pct Fe and 10.7 pct Fe^{2+} and had an Fe^{2+}/Fe ratio of 0.32. In Davis tube product evaluation, the average iron recovery was 86.5 pct in a magnetic fraction containing 64.5 pct iron and 9.5 pct silica.

Chemical analyses were performed on size fractions (table 9) so that the relationship of particle size to the analyses and extent of reduction could be studied. The fraction coarser than 4 mesh, except the small amount of plus 3/8-in product, was less reduced than the composite. This is to be expected because the reduction proceeds topochemically (2) from the particle surface. Thus the larger particles would tend to be less completely reacted near their center. The minus 325-mesh product was slightly underreduced. This result may have been caused by reoxidation as the product was discharged through the oxidizing atmosphere in the burner housing and discharge chute, and accentuated in the fine fractions because of their relatively large surface area.

PILOT PLANT BENEFICIATION

EQUIPMENT AND PROCEDURES

The beneficiation flowsheet (fig. 7 and tables 10 and 11) includes two stages of magnetic separation and three stages of flotation. A brief description of the equipment is given in table 12. The roasted taconite is fed to a rodmill at

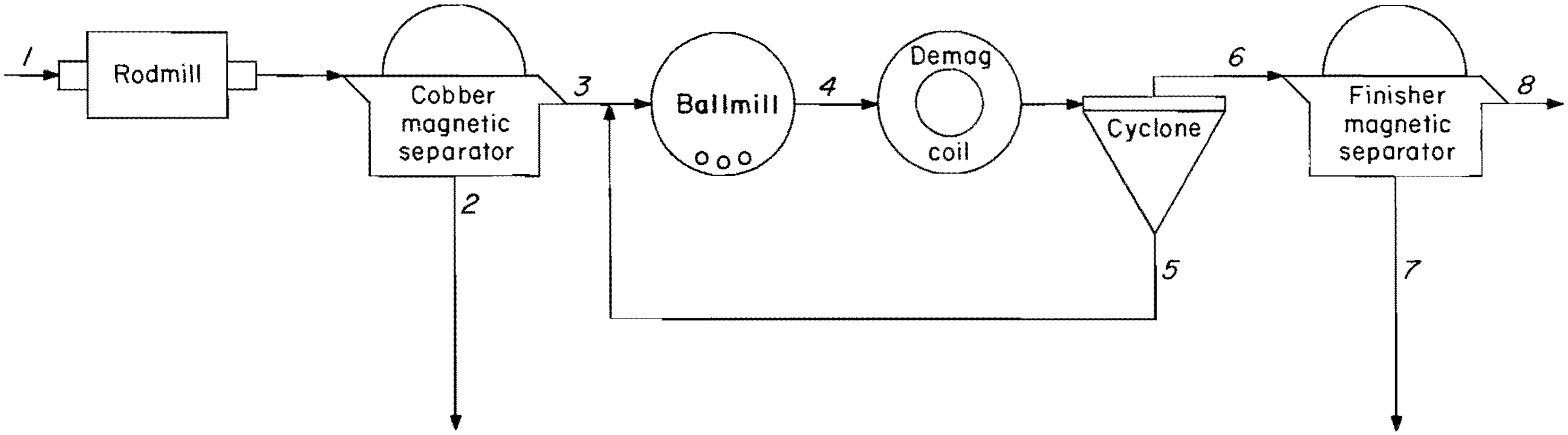

FIGURE 7. - Flowsheet for pilot plant beneficiation tests.

TABLE 10. - Material balance for pilot plant flowsheet shown in figure 7, streams 1-8, percent

Analysis	Streams							
	1-Rodmill feed	2-Cobber tailing	3-Cobber concentrate	4-Ball mill discharge	5-Cyclone underflow	6-Cyclone overflow	7-Finisher tailing	8-Finisher concentrate
Weight distribution....	100.0	43.8	56.2	285.4	229.2	56.2	9.6	46.6
Total iron............	33.2	8.1	52.8	58.3	59.7	52.8	11.5	61.3
Ferrous iron..........	10.7	2.4	17.2	18.7	19.1	17.2	2.6	20.2
Silica................	NA	NA	25.5	NA	NA	NA	NA	14.6
Total iron distribution	100.0	10.7	89.3	501.5	412.2	89.3	3.3	86.0

NA Not available.

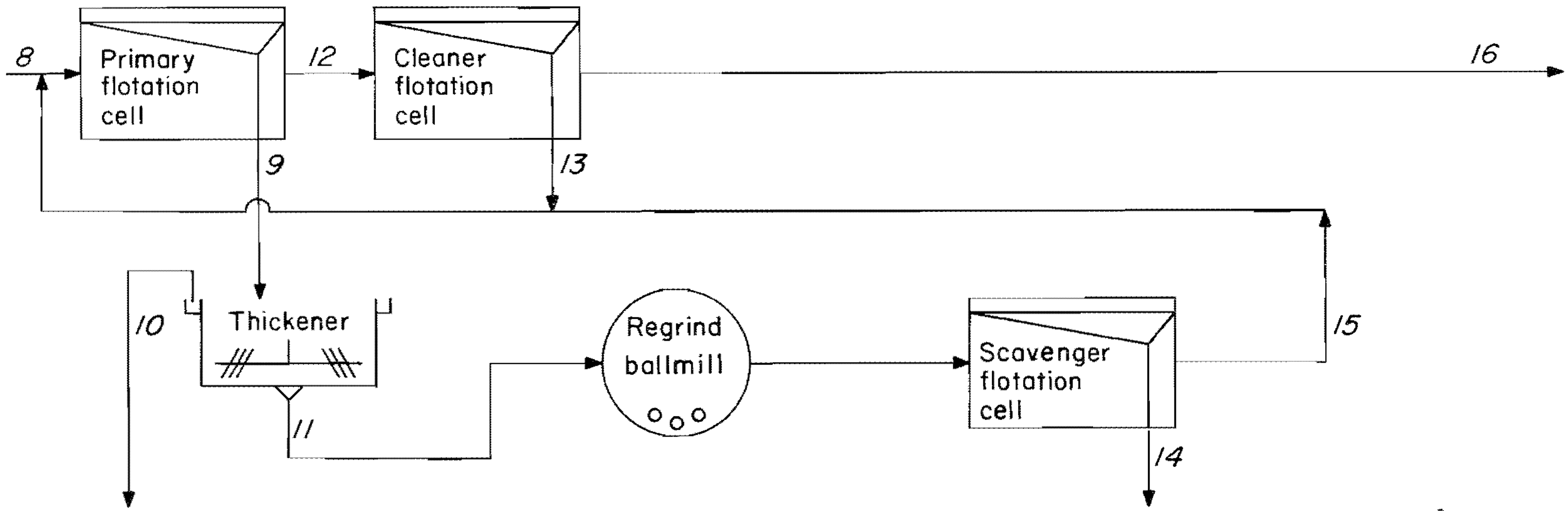

FIGURE 7. - Flowsheet for pilot plant beneficiation tests—Continued.

TABLE 11. - Material balance for pilot plant flowsheet shown in figure 7, streams 9-16, percent

Analysis	Streams							
	9-Primary flotation tailing	10-Thickener overflow	11-Thickener underflow	12-Primary flotation concentrate	13-Cleaner flotation tailing	14-Scavenger flotation tailing	15-Scavenger flotation concentrate	16-Cleaner flotation concentrate
Weight distribution	28.6	0.9	27.7	41.8	6.7	10.6	17.1	35.1
Total iron...	55.1	26.2	56.1	66.6	60.1	42.3	64.6	67.9
Ferrous iron.	18.4	8.5	18.7	22.0	19.9	14.0	21.6	22.4
Silica.......	NA	NA	NA	NA	NA	NA	9.8	5.5
Total iron distribution	47.6	.7	46.9	83.9	12.2	13.5	33.4	71.8

NA Not available.

TABLE 12. - Description of beneficiation equipment

Equipment	Description
Rodmill	2- by 4-ft mill, 690-lb rod charge.
Cobber magnetic separator	30- by 18-in concurrent drum. Maximum surface field strength 1,400 G.
Ballmill	3- by 3-ft mill, overflow type; 2,000-lb ball charge.
Demagnetizing coil	2-3/4-in ID; operated at 400 Hz and 1,000 Oe.
Cyclone	3-in diameter.
Finisher magnetic separator	30- by 18-in semiconcurrent drum. Maximum surface field strength 1,400 G.
Rougher flotation cells	Bank of 2 cells, 3 ft^3.
Cleaner flotation cell	Single cell, 1.5 ft^3.
Thickener (regrind feed)	3-ft diameter.
Regrind ballmill	3- by 3-ft mill, 500-lb ball charge.
Scavenger flotation cell	Single cell, 1.5 ft^3.

an average rate of 995 lb/h (dry basis) with enough water to produce a slurry containing 67.4 pct solids.

After being ground to about 71 pct minus 48 mesh (fig. 8), the slurry was concentrated with the cobber magnetic separator. The nonmagnetic product was dewatered in a screw classifier to yield an overflow, which was routed to the water reclamation system, and a coarse tailing. Concentrate from the cobber was ground in a ballmill, demagnetized, and classified in a cyclone. Demagnetization was employed to prevent magnetic flocculation, which might cause fines to accompany the coarse fraction into the cyclone underflow. The cyclone underflow was returned to the ballmill for regrinding, and the overflow, at 78 pct minus 500 mesh, was passed through the finisher magnetic separator.

To reduce the gangue content below that attainable with magnetic separation, the finisher concentrate was upgraded in a flotation section consisting of a primary flotation unit operating in closed circuit with scavenger and cleaner cells. A cationic collector, Arosurf MG-98A amine, was added to each stage to float the gangue. To improve liberation in the scavenger feed, it was thickened and reground to 82 pct minus 500 mesh.

The cobber tailing classifier overflow, the finisher tailing, and the scavenger tailing were flocculated with 0.09 lb/ton of a cationic flocculant, Superfloc 330, and pumped to a thickener in the water reclamation system (fig. 9). The thickener underflow was pumped to a tailings pond, and the overflow was discharged into a sludge-blanket clarifier. Filtrate obtained from the filtration of the cleaner concentrate was also pumped to the clarifier. Slurry from the clarifier was returned to the thickener, while the clarified overflow water was recycled to the pilot plant.

Minneapolis city water, source of the plant makeup water, contained 17 ppm Ca^{2+}. If unchecked, the Ca^{2+} concentration would rise further from basic constituents dissolved from the ore and lignite ash. Since past experiences have indicated that excessive concentrations of Ca^{2+} adversely affect flotation selectivity, Na_2CO_3 was added at a rate of 0.81 lb/ton to the rodmill to restrict the Ca^{2+} level to 15 ppm by precipitating $CaCO_3$.

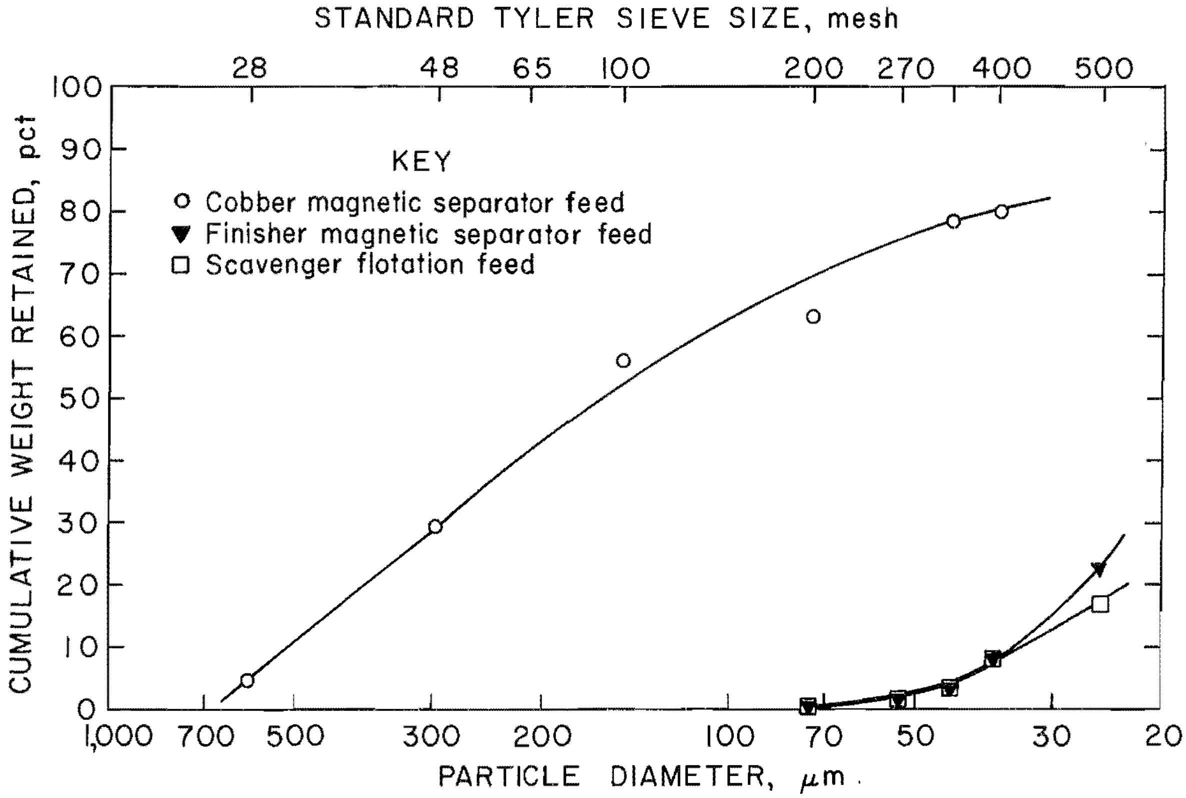

FIGURE 8. - Size analyses for pilot plant beneficiation tests.

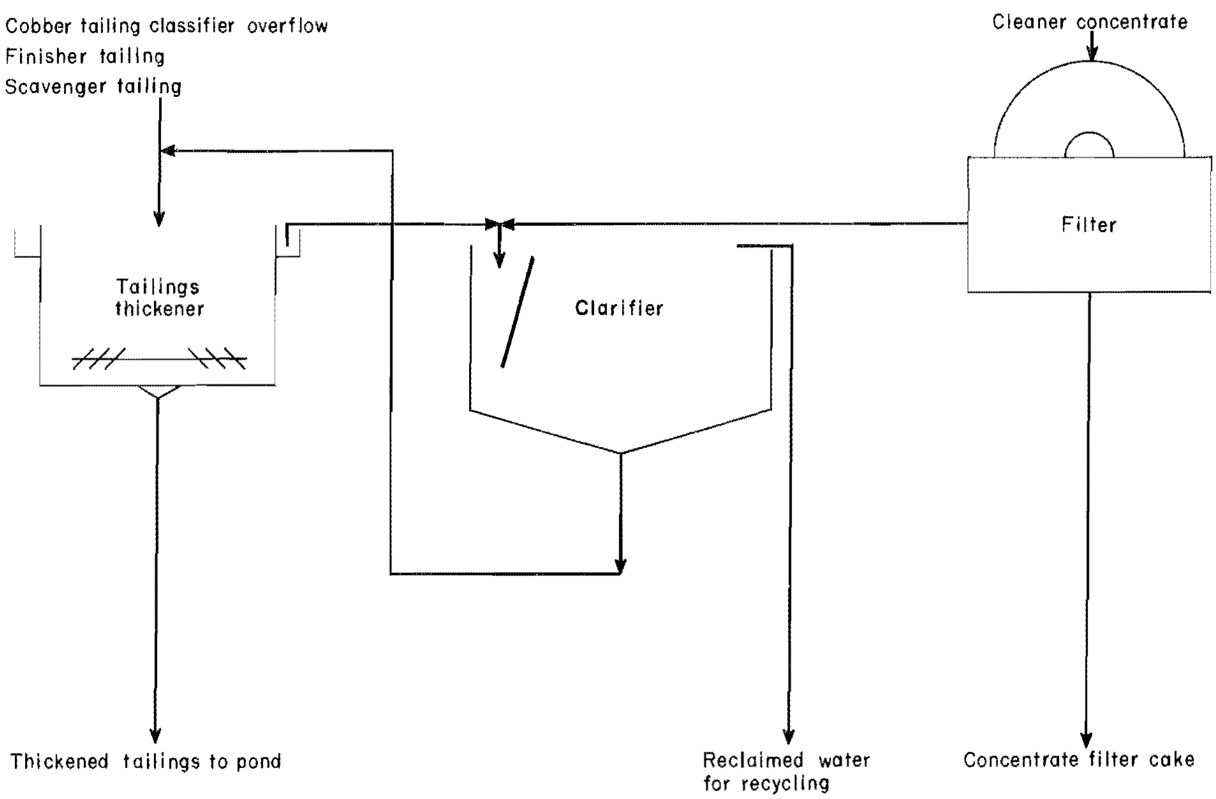

FIGURE 9. - Water reclamation flowsheet.

BENEFICIATION RESULTS

Before the sustained effort to obtain operating data, a number of 5-h tests were conducted to establish good operating parameters. One of the objectives was to determine the best conditions for running the coil that demagnetized the cyclone feed. Demagnetization prevents magnetic flocculation, a condition that may cause fine magnetite particles to report with the cyclone coarse product. A demagnetizing field with a frequency of 60 Hz, which is often used for natural magnetite, is not effective for artificial magnetitie slurries (3). To achieve adequate demagnetization, the Bureau obtained a coil that would provide a field strength up to 1,000 Oe and a frequency range of 400 to 800 Hz. The preliminary tests indicated that its effectiveness was not highly sensitive to variation within this frequency range, but was highly responsive to the strength of the field and gave the best demagnetization at the maximum field strength.

In the subsequent pilot plant test, the coil was operated at 1,000 Oe and 400 Hz. Demagnetizing did diminish the proportion of the minus 22-µm fraction in the cyclone underflow from 20 to 15 pct, but the coil power requirement was high, 20 kW•h/ton (based on rodmill feed rate). The effect of the better classification on grinding efficiency was not evaluated.

After the preliminary tests, a 5-day continuous run was conducted to obtain operating and metallurgical data under a prolonged operation. A set of samples was accumulated during 1 h of each 4-h period. The flow rate and analytical data obtained from each set of samples were entered in a computer program which constructed a material balance. The program achieves a balance for a system having too few unknowns by adjusting values using a least squares approach (12). Adjustments are made in accordance with standard deviation estimates to minimize the sum of squares of the changes that must be made to achieve the balance. The material balance (tables 10 and 11) is the average of results from a 24-h period of steady-state operation.

During this test, the rodmill was fed roasted ore at an average rate of 995 lb/h. The largest consumer of grinding power was the ballmill, requiring 15.7 kW•h/ton (table 13), while the smallest was the regrind mill, using 0.5 kW•h/ton to grind the scavenger feed. A petrographic examination of the scavenger tailing indicated that the iron minerals, mostly magnetite, were 70 pct liberated. The silica in the cleaner concentrate was 52 pct liberated. From these observations it is concluded that the fineness of the grind (fig. 8) was adequate to permit much better iron recovery if the flotation selectivity were to be improved.

TABLE 13. - Grinding mill power requirements

Mill	Power requirement[1]	
	kW•h/ton	hp•h/ton
Rodmill.........	5.6	7.5
Ballmill........	15.7	21.0
Regrind mill....	.5	.7
Total......	21.8	29.2

[1]Based on tons of beneficiation plant feed.

The total water usage in the pilot plant was 29.8 gal/min, 88.2 pct of which was reclaimed water. The supply water, reclaimed plus makeup water, had a pH of 9.4, a Ca^{2+} content of 15.2 ppm, and a turbidity of 103 ppm silica equivalents (Hellige turbidimeter).

The reagent feed rates are shown in table 14. The total collector consumption was 0.22 lb/ton. Beneficiation iron recovery was 71.8 pct, with the losses divided about equally between the magnetic-separation and flotation circuits. After adjusting for the 2.9-pct loss in the roasting plant, the overall recovery was 69.7 pct. The final concentrate contained 67.9 pct Fe and 5.5 pct SiO_2 (table 15).

TABLE 14. - Beneficiation reagent suite

Reagent and function	Addition point	Amount added, lb/ton of beneficiation plant feed
Arosurf MG-98A,[1] collector.....	Primary flotation.............	0.07
Do..........................	Scavenger flotation...........	.09
Do..........................	Cleaner flotation.............	.06
Sodium carbonate, soluble calcium control.	Rodmill.......................	.81
Superfloc 330,[2] flocculant.....	Water reclamation thickener...	.09

[1]Ashland Chemical Co., Columbus, OH.
[2]American Cyanamid Co., Wayne, NJ.

TABLE 15. - Chemical analysis of process products

	Kiln waste products			Beneficiation tailings		Product--cleaner concentrate
	Cyclone dust	Classifier overflow solids	Scrubber sludge	Classifier sands[1]	Tailings thickener solids	
			PCT			
Carbon...........	0.97	0.64	2.1	0.50	0.35	<0.1
Iron:						
Total..........	26.7	22.0	30.8	8.7	25.3	67.9
Ferrous........	2.1	5.9	1.7	1.9	7.8	22.4
Manganese........	.07	.10	<.10	.05	.09	.12
Phosphorus.......	.1	.06	.07	<.01	<.01	.05
Sulfur...........	.14	.12	.31	.027	.035	.015
Al_2O_3............	1.10	1.9	2.3	.94	.41	.34
CaO..............	1.36	2.6	.81	.65	.67	.26
K_2O.............	.12	.13	.17	.16	.05	<.05
MgO..............	.45	1.30	.37	.25	.29	.11
Na_2O............	.44	.33	.50	.25	.09	.33
SiO_2.............	54.5	60.3	46.4	83.8	61.5	5.5
TiO_2.............	.65	<.2	<.2	<.2	<.2	<.5
V_2O_5............	NA	.005	.008	<.005	<.005	<.05
			PPM			
Antimony.........	NA	<1	<1	<10	<10	<5
Arsenic..........	NA	80	NA	13	16	<10
Cadmium..........	NA	<100	<100	.15	.18	<10
Chromium.........	NA	10	20	60	<30	<20
Cobalt...........	NA	<5	<5	<50	<50	<10
Copper...........	NA	35	65	<20	<20	<10
Fluorine.........	NA	130	120	<1,000	<1,000	<1,000
Lead.............	NA	10	NA	<.01	<.01	<50
Mercury..........	NA	.67	.69	<.5	<.5	<.5
Nickel...........	NA	10	10	<50	<50	<50
Selenium.........	NA	2	3	<10	<10	<.01
Silver...........	NA	<100	<100	<5	<5	<50
Zinc.............	NA	<.1	<.1	<50	<50	<50

NA Not available.
[1]Coarse fraction of cobber tailing. The slimes were pumped to the tailings thickener.

SUMMARY AND CONCLUSIONS

Pilot plant processing of material from the WMR-3 sample area on the western Mesabi Range demonstrated that an acceptable iron concentrate, containing 67.9 pct iron and 5.5 pct silica, could be produced with a 69.7 pct iron recovery. The samples from the WMR-1 and WMR-2 areas had responded with iron recoveries of 69.2 and 82.4 pct, respectively (table 16). Relative to the other two samples, the WMR-3 sample was of lower grade and required more collector. Especially compared with the WMR-2 sample, this material may be considered less desirable from a metallurgical viewpoint.

While substantial iron losses occurred in both the magnetic separation and flotation circuits, laboratory investigations indicated a potential for improvement through further research. Petrographic studies of the tailings showed that liberation was sufficient to allow substantial improvement of iron recovery. In addition, in bench-scale beneficiation tests, a much higher overall iron recovery of 82.1 pct was obtained while making a concentrate containing 69.0 pct iron and 4.5 pct silica. To improve the magnetic separation recovery, more complete conversion to magnetite is needed. To improve flotation recovery, flotation selectivity must be sharpened.

The environmental acceptability of the roasting process was enhanced by operating the kiln under a negative pressure to control fugitive dust emissions and by cleaning exhaust with a cyclone collector and a Venturi scrubber. The coarse fraction of the exhaust dust was recovered in the cyclone, pelletized, and subsequently processed through the kiln. The very fine particulate in the roasted product, which does not respond selectively to

TABLE 16. - Comparison of results from processing samples from three areas of the western Mesabi Range

	WMR-1	WMR-2	WMR-3
Feed:			
Estimated resource...billion tons..	2.0	0.75	0.5
Iron...pct..	35.4	36.8	32.1
Roasted ore:			
Iron loss in roasting and desliming...pct..	3.1	4.0	2.9
Iron...pct..	37.6	40.6	33.2
Ferrous-total iron ratio...	0.33	0.32	0.31
Lignite used as reductant...pct of ore..	2.6	3.4	2.9
Finisher magnetic concentrate, pct:			
Iron...	[1]62.9	65.9	61.3
Silica...	[1]12.4	8.0	14.6
Iron loss in magnetic separator tailings...	13.6	9.7	14.0
Flotation concentrate:			
Iron...pct..	67.9	68.1	67.9
Silica...pct..	4.9	4.6	5.5
Iron-silica ratio...	13.9	14.8	12.3
Iron recovery--beneficiation[2]...pct..	71.4	85.8	71.8
Iron recovery--roasting and beneficiation...pct..	69.2	82.4	69.7
Reagent usage,[2] lb/ton:			
Arosurf MG-98A (collector)...	0.09	0.11	0.22
Sodium carbonate (calcium control)...	0	0	0.81
Superfloc 330 (flocculant)...	0.07	0.06	0.09

[1]Data not comparable with those from other tests because flotation middlings were recycled through the magnetic separation circuit in this test.

[2]Based on beneficiation plant feed.

magnetic separation, was removed in the overflow of a screw classifier.

In the beneficiation plant the cyclone separation was sharpened to decrease the amount of fines being recycled to the ballmill. This was accomplished by demagnetizing the cyclone feed with a 400-Hz, 1,000-Oe magnetic field. Demagnetization prevented magnetic flocculation of the fines, which would influence them toward discharging with the coarse product. Though the demagnetizing system used required an excessive amount of electric energy, it diminished the amount of minus 22-μm fines in the coarse product from 20 to 15 pct.

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