Pamphlet 81

October, 1948

STATE OF IDAHO
C. A. Robins, Governor

IDAHO BUREAU OF MINES AND GEOLOGY
A. W. Fahrenwald, Director

BENEFICIATION TESTS ON ORE FROM THE SUN VALLEY BARITE MINE, BLAINE COUNTY, IDAHO

By

Lewis S. Prater, Metallurgist

University of Idaho
Moscow, Idaho

TABLE OF CONTENTS

LIST OF ILLUSTRATIONS

BENEFICIATION TESTS ON ORE FROM THE
SUN VALLEY BARITE MINE, BLAINE COUNTY, IDAHO

INTRODUCTION

One of the difficulties in using barite that has been graded up by flotation for oil well drilling muds is to obtain a wettable product. Present practice is to calcine the barite concentrate to destroy the fatty acid film. This is costly. The tests outlined in this report were undertaken to determine if it were possible to obtain satisfactory beneficiation of this particular sample by flotation at a coarse size, followed by grinding to produce sufficient new wettable surface. Previous tests[1] made on another sample of ore from this property have indicated the feasibility of beneficiation by flotation.

A sample of approximately 300 pounds was received at the laboratory for testing purposes. It has been designated as Ore Lot #83. As received, the pieces ranged in size from 4 or 5 inches in diameter to fines, although the amount of fines was relatively small. In general appearance, the rock was a light gray color with iron stain evident on some of the pieces. Bedding planes could be occasionally observed showing some mineral differentiation, but most of the sample was of a uniform, fine-grained crystalline texture. The rock is soft and has a high grindability but does not form an excessive amount of slimes in ball milling.

The head sample assayed 84.8% $BaSO_4$. Its specific gravity was 4.07.

TESTING PROCEDURE AND DATA

Crushing and Preparation of Samples

The entire sample, as received, was crushed to approximately one inch in the laboratory jaw crusher. One half was reserved, and the other half was reduced to ¼ inch. Of this ¼-inch product, one half was again reserved and the remainder was crushed, dry, to minus 8 mesh. A head sample for assay and the samples required for testing were cut from this portion.

Sizing, Sink-Float Test

To determine the relationship between particle size and mineral liberation, a sizing, sink-float test was made on a sample of the minus 8 mesh rock. A screen analysis, using the $\sqrt{2}$ series through 200 mesh was made. Each of the size portions above 200 mesh was then fractionated into a float product and a sink product in acetylene tetrabromide having a specific gravity of 2.95. Chemical analysis and specific gravity determinations were then made on the sands in each sink product. The float products were weighed to obtain a weight balance and then discarded. The data obtained are tabulated below:

---

[1]Prater, L. S., Beneficiation Tests on Ore Lot #78. Idaho Bureau of Mines and Geology, Report on file.

Table 1

Screen Analysis and Heavy Density Separation on Ball Mill Feed

			Float Product at 2.95 Sp.G.		Sink Product at 2.95 Sp.G.			
Size (Mesh)	% Weight	Cumulative % Weight	% of Wgt. in Size Fraction	% of Wgt. in Entire Sample	% of Wgt. in Size Fraction	% of Wgt. in Entire Sample	Sp.G. of Sands	Assay of Sands % $BaSO_4$
+10	6.0	6.0	2.2	0.1	97.8	5.9	4.04	85.3
10/14	17.3	23.3	2.3	0.4	97.7	16.9	4.02	84.3
14/20	13.5	36.8	2.7	0.4	97.3	13.1	4.02	84.2
20/28	8.0	44.8	3.0	0.2	97.0	7.8	4.03	84.8
28/35	5.8	50.6	2.6	0.2	97.4	5.6	4.09	86.3
35/48	5.9	56.5	1.9	0.1	98.1	5.8	4.19	90.1
48/65	6.1	62.6	1.8	0.1	98.2	6.0	4.25	92.5
65/100	8.85	71.4	4.1	0.4	95.9	8.5	4.30	93.6
100/150	5.45	76.9	8.3	0.4	91.7	5.0	4.28	95.2
150/200	7.2	84.1	14.4	1.0	85.6	6.2	4.31	96.7
-200	15.9	100.0	---	---	---	---	---	---
Composite	100.0	---	---	3.3	---	80.8	---	---

Grinding for Flotation

The sink-float test tabulated above indicates that little or no mineral liberation is accomplished above 48 mesh. On the basis of these data, flotation tests were made on pulps ground as follows:

Grind #1 - all through 48 mesh
Grind #2 - " " 100 mesh
Grind #3 - " " 200 mesh
Grind #4 - " " 325 mesh
Grind #5 - " " 100 mesh (confirmatory tests)

In preparing these ground products the minus 8 mesh ball mill feed was washed on the controlling screen, and the oversize was batch ground in the laboratory ball mill. Short grinding times, followed by screening, were used to prevent, as much as possible, any overgrinding. For the finer sized products the small pebble mill and finally a hand mortar were used to complete the grind. In each grind, 2000 grams of ore were prepared and then split into four 500-gram samples for testing purposes. The screen analyses of these ground products are tabulated below.

Table II

Screen Analyses of Ground Pulps

Size (Mesh)	Grind #1 (48 mesh)			Grind #2 (100 mesh)			Grind #3 (200 mesh)			Grind #5 (100 mesh)		
	% Wgt.	Cum.% Wgt.	Sp.G. of Sand	% Wgt.	Cum.% Wgt.	Sp.G. of Sand	% Wgt.	Cum.% Wgt.	Sp.G. of Sand	% Wgt.	Cum.% Wgt.	Sp.G. of Sand
48/65	5.05	5.05	4.27	----	----	----	----	----	----	----	----	----
65/100	10.9	15.9	4.20	----	----	----	----	----	----	----	----	----
100/150	9.05	25.0	4.10	10.7	10.7	4.12	----	----	----	10.3	10.3	4.15
150/200	16.2	41.2	3.98	19.0	29.7	4.02	----	----	----	19.2	29.5	4.07
200/325	16.0	57.2	3.88	18.9	48.6	4.00	20.6	20.6	3.95	19.0	48.5	3.99
-325	42.8	100.0	3.95	51.4	100.0	3.99	79.4	100.0	4.05	51.5	100.0	4.14
Comp.	100.0	---	4.00	100.0	---	4.00	100.0	---	4.03	100.0	---	4.10

Note: None of the products were sized below 325 mesh.

All the flotation tests on each grind were completed immediately following the grind, before proceeding with the parallel tests on the succeeding grind. No ground pulp stood more than half a day before the flotation tests were made on it. In this report, however, the tests are arranged to compare the parallel tests on the different grinds. All grinding and flotation tests were made with zeolite-softened water.

Screen Analyses of the Flotation Products

The following three tests are reported to indicate the size distribution of the products in flotation. Reagent additions to the flotation cell were as follows:

Test No.	Pulp	Sodium Silicate	Oleic Acid	Methyl amyl Alcohol
3	$\frac{1}{4}$ of Grind #1 (48 mesh)	0.12 lb./ton	0.45 lb./ton	0.10 lb./ton
6	$\frac{1}{4}$ of Grind #2 (100 mesh)	0.12 lb./ton	0.45 lb./ton	0.05 lb./ton
12	$\frac{1}{4}$ of Grind #3 (200 mesh)	0.12 lb./ton	0.60 lb./ton	0.10 lb./ton

In each test the pulp was allowed to condition 3 or 4 minutes with sodium silicate before adding the other reagents. Oleic Acid was added in stages of 0.15 lb./ton and methyl amyl alcohol in stages as required. In test No. 3 and test No. 12, the concentrate was returned to the cell for cleaning with no additional reagents added. In test No. 6, only the rougher concentrate and tailing were prepared. Wet-dry screen analyses and specific gravity determinations were then made on all the products. No $BaSO_4$ assays were run on these tests.

Although the same reagents were used, the above tabulation indicates some variation in amounts. These were added as appeared necessary in the progress of each test. The tests are to indicate only the size distribution in the products and not for direct comparison of grade and recovery with given amounts of collector and conditioning reagents.

Table III

Size Distribution of Flotation Products

Test No.	Size (Mesh)	Concentrate		Middling		Tailing		Composite	
		Weight (Grams)	Dist. %	Weight (Grams)	Dist. %	Weight (Grams)	Dist. %	Weight (Grams)	Calc. Head % Wgt.
3 (48) (Mesh)	+65	---	0.0	0.5	1.7	28.3	98.3	28.8	5.4
	65/100	---	0.0	3.0	5.7	49.5	94.3	52.5	9.9
	100/150	0.2	0.5	6.5	16.1	33.7	83.4	40.4	7.6
	150/200	6.1	7.7	22.1	28.0	50.7	64.3	78.9	14.8
	200/325	21.2	27.9	27.6	36.4	27.1	35.7	75.9	14.3
	-325	138.8	54.4	77.7	30.4	38.8	15.2	255.3	48.0
	Total	166.3	31.3	137.4	25.9	228.1	42.8	531.8	100.0
6 (100) (Mesh)	100/150	3.2	7.2	---	---	41.3	92.8	44.5	9.4
	150/200	12.2	13.8	---	---	76.2	86.2	88.4	18.6
	200/325	27.6	31.6	---	---	59.6	68.4	87.2	18.4
	-325	184.0	72.3	---	---	70.7	27.7	254.7	53.6
	Total	227.0	47.8	---	---	247.8	52.2	474.8	100.0
12 (200) (Mesh)	+325	9.8	11.1	51.7	58.7	26.6	30.2	88.1	18.3
	-325	203.3	51.8	136.6	34.8	52.4	13.4	392.3	81.7
	Total	213.1	44.3	188.3	39.2	79.0	16.5	480.4	100.0

Table IV

Specific Gravities of Products

Size (Mesh)	Test #3 (48 mesh grind)			Test #6 (100 mesh grind)		Test #12 (200 mesh grind)		
	Conc.	Middling	Tailing	Conc.	Tailing	Conc.	Middling	Tailing
65	----	)	4.27	----	----	----	----	----
65/100	----	) 4.39	4.23	----	----	----	----	----
100/150	4.40	)	4.08	)	4.09	----	----	----
150/200	4.40	4.40	3.83	) 4.45	3.97	----	----	----
200/325	4.45	4.40	3.34	4.46	3.82	4.49	4.43	3.12
-325	4.42	4.26	3.03	4.40	3.51	4.42	4.29	2.91
Composite	4.43	4.32	3.76	4.42	3.81	4.43	4.33	2.98
Calc. Head	4.08			4.07		4.06		

The above tables need little explanation. They prove quite conclusively that the small size particles are floated selectively from the coarse. Also, the difference in gravity between the coarse particles in the concentrates and the coarse particles in the tailings is small. Little beneficiation is accomplished on these. The difference in gravity between the finer size particles in the respective products is noticeably greater.

Relationship Between Recovery and Grade of Concentrate

Three parallel tests were made to determine the effect of increasing the recovery on the grade of the concentrate at 100 mesh, 200 mesh, and 325 mesh. Identical procedures were followed in all three tests. The pulp was conditioned three or four minutes with 0.12 lb. of sodium silicate per ton. Oleic acid and methyl amyl alcohol were then added in stages with the concentrate which was collected after each addition kept separate. Reagent additions were as follows.

	Oleic Acid lb./ton	Methyl amyl Alcohol - lb./ton
Conc. #1	0.15	0.05
Conc. #2	0.15	0.025
Conc. #3	0.15	0.025
Conc. #4	0.30	0.025
Total	0.75	0.125

The metallurgical results of these tests are given in Table V.

Table V

Metallurgical Results of Stage Flotation Tests

Test No.	Product	Weight (Grams)	% Weight	Sp.G. of Sands	Assay % $BaSO_4$	$BaSO_4$ Dist. %	Cum. Recov. %
18 (100 mesh) (grind #5)	Conc. #1	119.6	23.5	4.36	97.3	26.9	26.9
	Conc. #2	78.2	15.4	4.36	96.3	17.4	44.3
	Conc. #3	72.8	14.3	4.33	94.9	15.9	60.2
	Conc. #4	165.5	32.6	4.18	89.8	34.3	94.5
	Tailing	72.5	14.2	3.06	32.9	5.5	----
	Composite	508.6	100.0	4.06	85.2	100.0	----
11 (200 mesh) (grind #3)	Conc. #1	181.7	37.3	4.49	97.3	42.9	42.9
	Conc. #2	70.3	14.5	4.47	96.6	16.5	59.4
	Conc. #3	98.2	20.3	4.45	96.2	23.0	82.4
	Conc. #4	59.7	12.3	4.32	92.5	13.4	95.8
	Tailing	75.6	15.6	2.99	23.2	4.2	----
	Composite	485.5	100.0	4.12	84.7	100.0	----
14 (325 mesh) (grind #4)	Conc. #1	220.4	46.5	4.40	96.8	53.0	53.0
	Conc. #2	115.8	24.4	4.35	95.2	27.3	80.3
	Conc. #3	58.5	12.4	4.27	93.1	13.6	93.9
	Conc. #4	27.0	5.7	3.74	72.1	4.8	98.7
	Tailing	51.9	11.0	2.79	10.0	1.3	----
	Composite	473.6	100.0	4.07	84.9	100.0	----

From the above tables, composite concentrate grades have been calculated giving the grades of successive products that might be expected with increasing recovery. The following figures graphically present these data.

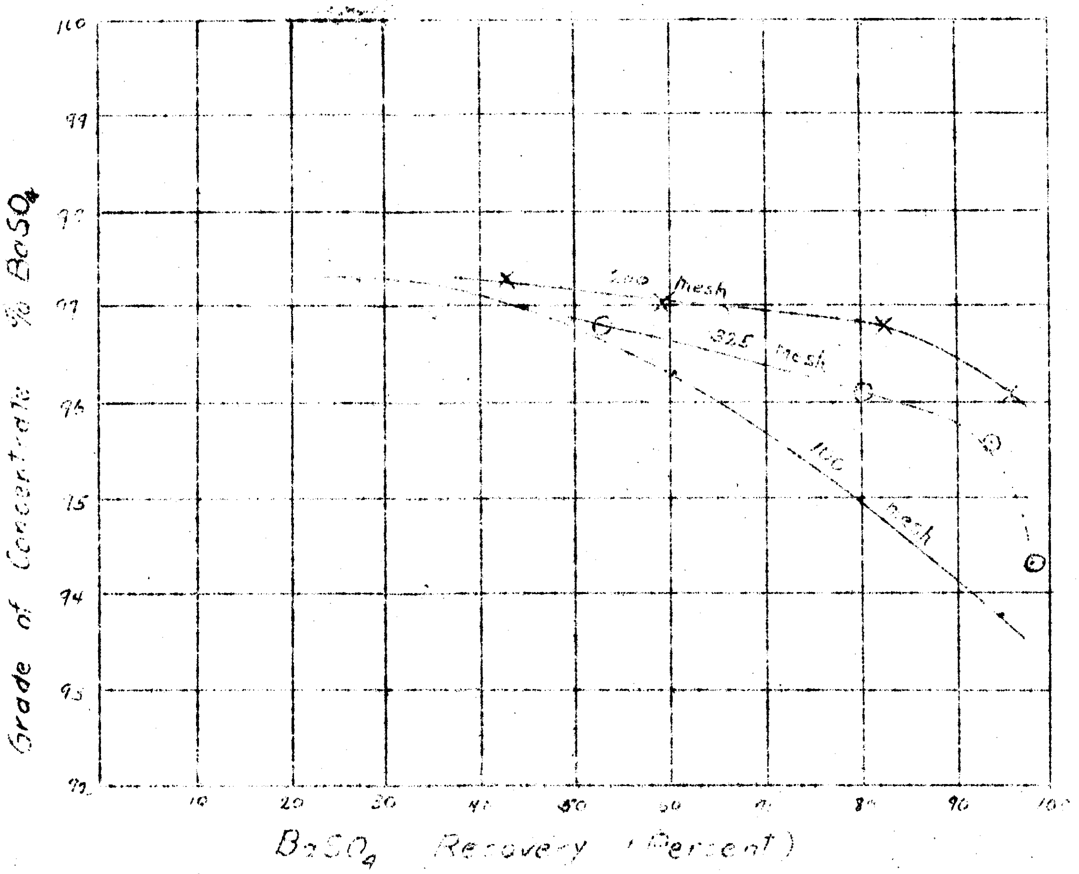

FIG I – RELATIONSHIP OF RECOVERY & GRADE OF CONCENTRATE

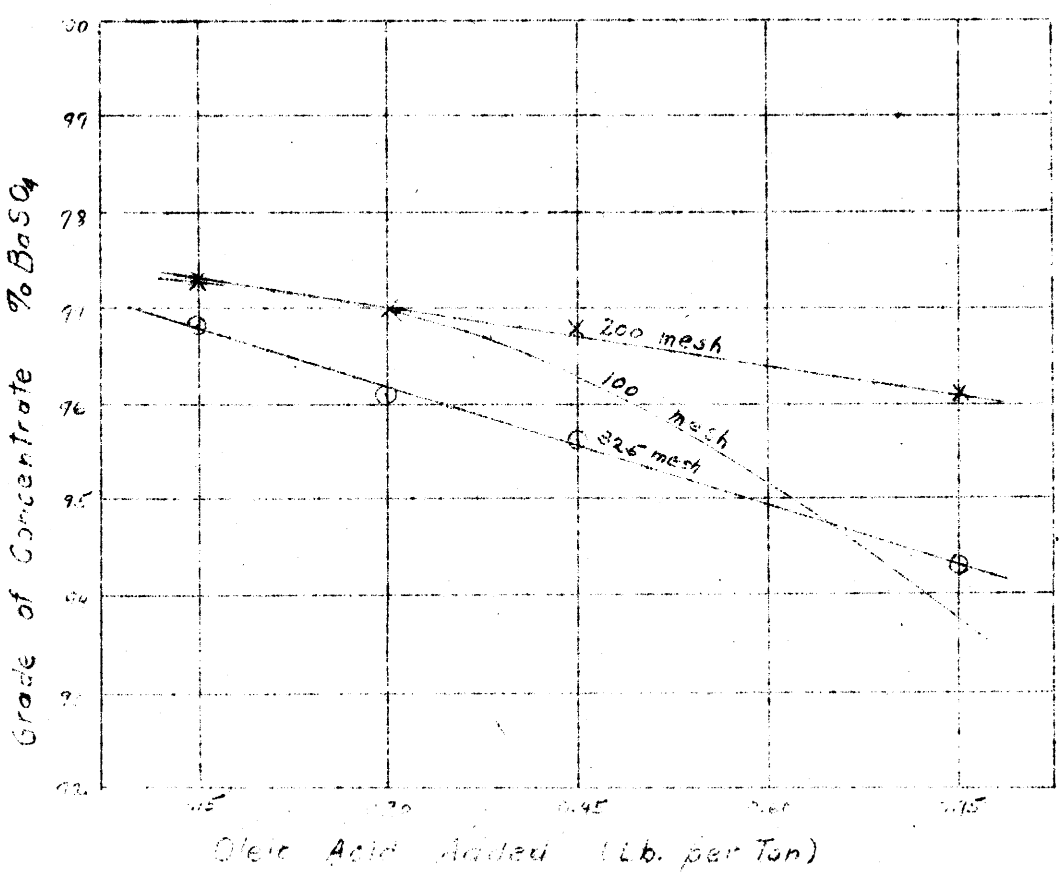

FIG. II EFFECT OF COLLECTOR ADDITIONS ON CONCENTRATE GRADE

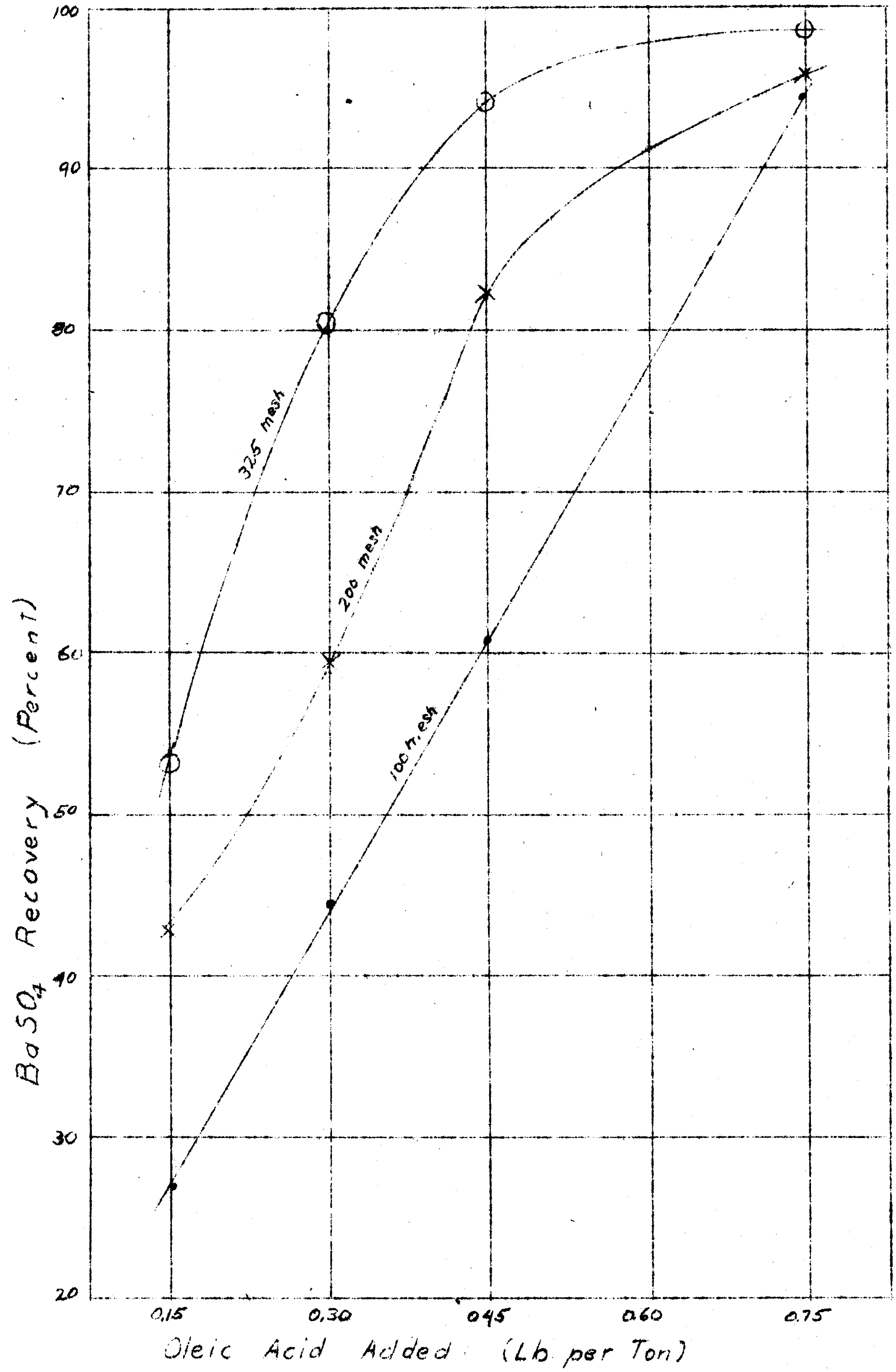

FIG. III · EFFECT OF COLLECTOR ADDITIONS ON RECOVERY

Cleaner Concentrates

Several tests were made in which the concentrate was returned to the cell for cleaning. Although the same reagents were used in all tests, the amount of oleic acid collector varied from 0.45 lb. to 0.60 lb. per ton in the rougher circuit. For this reason the results are not strictly comparable on a quantitative basis, but they clearly indicate what may be expected at various grinds. In the first tests, no additional reagents were added when the concentrate was returned to the flotation cell for cleaning. As a result it was found that a large bulk of fairly high grade middling was produced, lowering the barite recovery in the concentrate. For obvious reasons, this situation would undoubtedly be less serious in a continuous circuit. One test was, therefore, made in which additional collector was added to the cleaner circuit. This raised the recovery in the concentrate without appreciably lowering its grade and at the same time lowered both the weight and grade of the middling, proving that some of the middling in the previous tests was in reality free mineral that was not floated.

Table VI

Summary of Cleaner Tests

Test No. and Grind	Cleaner Concentrate Product		Remarks
	Assay % $BaSO_4$	$BaSO_4$ Recovery %	
Test #4-Grind #1 (48 mesh)	96.0	26.8	These tests made with no additional collector added to the cleaner circuit.
Test #7-Grind #2 (100 mesh)	98.3	23.8	
Test #12-Grind #3 (200 mesh)	98.0	51.0	
Test #16-Grind #4 (325 mesh)	97.9	57.0	
Test #20-Grind #5 (100 mesh)	97.1	86.7	Additional collector added to cleaner circuit.

Flotation of Impurities with Cationic Reagents

Since the preceding tests indicate that fine grinding will be required to obtain satisfactory metallurgical results with soap flotation, the possibility of producing a coarse concentrate and destroying the water repellent surface by grinding appears remote. Preliminary tests have, therefore, been made to test the possibility of floating the quartz and other siliceous impurities from the barite, with cationic reagents. The separation has been found to be less selective, but more complete testing would be required to determine the best alternatives following this procedure.

Three tests are reported below on 100 mesh, 200 mesh, and 325 mesh feed. The reagent additions for each test were as follows:

Sodium silicate	=	0.12 lb./ton
DP 243	=	0.25 lb./ton
Methyl amyl alcohol	=	0.07 lb./ton

On the 100 mesh feed and 325 mesh feed, the collector was added in four stages with the concentrate collected by each addition kept separate. A single concentrate and tailing was made from the 200 mesh feed.

Table VII

Metallurgical Results of Cationic Flotation

Test No. and Grind	Product	Weight (Grams)	% Weight	Sp.G. of Sands	Assay % $BaSO_4$	$BaSO_4$ Dist. %
19 (Grind #5) (100 mesh)	Conc. #1	35.1	7.1	3.46	57.8	4.8
	Conc. #2	27.6	5.6	3.54	61.8	4.0
	Conc. #3	30.7	6.2	3.65	67.4	4.9
	Conc. #4	41.1	8.3	3.72	69.5	6.8
	Tailing	361.3	72.8	4.27	92.9	79.5
	Composite	495.8	100.0	4.06	85.0	100.0
13 (Grind #3) (200 mesh)	Conc.	132.5	24.4	3.70	67.3	19.2
	Tailing	411.2	75.6	4.24	91.1	80.8
	Composite	543.7	100.0	4.09	85.1	100.0
15 (Grind #4) (325 mesh)	Conc. #1	59.5	12.5	3.62	65.0	9.6
	Conc. #2	64.3	13.6	3.71	67.5	10.7
	Conc. #3	67.5	14.2	3.88	76.8	12.8
	Conc. #4	81.5	17.2	4.17	88.1	17.8
	Tailing	202.0	42.5	4.43	98.1	49.1
	Composite	474.8	100.0	4.08	84.9	100.0

CONCLUSIONS

The heavy density separation tests indicate poor mineral liberation coarser than 48 mesh. Grinding to 150 or 200 mesh would appear necessary to obtain any appreciable increase in weight in the material in the float discard and to obtain a satisfactory gravity in the sink product.

The three tests in which screen analyses were made on the products bring out some significant data. In all cases the small size particles float in preference to the coarse. Reference to Test #3, Table III, shows only 1.7% of the plus 65 mesh particles in the feed going into the combined concentrate and middling with 98.3% going into the tailings. In comparison, 84.8% of the minus 325 mesh particles is in the combined concentrate and middling, and only 15.2% is in the tailings. This pattern is carried, to a remarkable degree, through all three tests at different grinds.

Control on these tests was on the basis of specific gravities. The spread between the gravity of the coarse particles in the concentrate and the gravity of the coarse particles in the tailing is small compared to the spread between the minus 325 mesh fractions in the various products. In other words, beneficiation is much poorer on the coarse particles than on the fine.

The results of the tests in which the concentrate was removed in stages confirm the previous data. The surprising thing has been the increased recovery per unit of collector on the finely ground products (see Figure III). This is not readily explained.

It can be noted in Table V that on Test 14 (-325 mesh feed) and to a lesser degree on Test 11 (-200 mesh feed) most of the floatable mineral was collected in the first two concentrates. In Test 18, however, on minus 100 mesh feed a large bulk was floated in Concentrate 4. The grade of this concentrate fraction dropped noticeably below that of the preceding products to a figure only slightly higher than the head. No screen analyses were made, but visual examination of the products indicated that Concentrate 4 was much more sandy than the others. This again would seem to indicate that the fine, liberated mineral is floated first and following addition of sufficient collector the coarser middling particles start to float.

In Figure I it can be noted that on the minus 325 mesh feed the grade of the concentrate does not drop appreciably until the recovery exceeds 90%; for 200 mesh feed the grade begins to fall when the recovery passes 80%; and on 100 mesh feed the grade starts to drop at 50-60% recovery, indicating again the points at which locked middlings start to come over. Reference to these same curves also indicates slightly better metallurgical results for 200 mesh than for 325 mesh feed. This is probably the result of slimes being mechanically included in the concentrate on minus 325 mesh feed. However, the cleaner tests indicated that the cleaned concentrate from minus 325 mesh feed could be made equivalent and that the overall recovery from minus 325 mesh was somewhat higher than in any other tests. It is doubtful, though, if the advantage to be gained from this small increase in recovery would be economic unless it were otherwise necessary to grind to minus 325.

The tests reported on the use of cationic reagents to remove the siliceous impurities are only preliminary. Undoubtedly with further refinements, it would be possible to improve the selectivity, although probably not to the degree attained with soap flotation. Flotation of the impurities appears to be the most likely method of obtaining a wettable product without calcination. Coarse particles can be floated but beneficiation is poor.

In most tests, both specific gravity determinations and barite assays were made on the products. Exact correlation between percent barite and specific gravity was not always possible, but fairly close checks were obtained. Wood alcohol was found to be a satisfactory medium for measuring the specific gravities of oleic acid filmed concentrate.

ACKNOWLEDGEMENTS

The writer wishes to acknowledge the work of Mr. C. R. Kurtak, Chemist for the Idaho Bureau of Mines and Geology, who made all the chemical analyses for barite percentages reported herein.