Chapter 42

BENCH SCALE FLOTATION OF ALUNITE ORE WITH OLEIC ACID

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ABSTRACT

Alunite $[KAl_3(SO_4)_2(OH)_6]$ is a promising non-bauxitic aluminum resource, the domestic reserves of which are estimated to be 800 x 10^6 tons at 35 percent alunite. The major gangue mineral associated with alunite is quartz. The thermochemical process that has been developed for treatment of this ore does not provide for the initial removal of gangue prior to processing. In this regard, it may be desirable to concentrate the ore to gain significant technical and economic advantages.

Experimental results indicate the effectiveness of an oleic acid flotation technique and exemplify the problems encountered in fine particles processing. These results show that the quality of separation studied as a function of temperature, collector addition, and depressant addition is dependent on the degree of liberation achieved during size reduction. At coarse grinds the quality of separation was limited by the presence of locked particles while for finer grinds where more extensive liberation is achieved, better separations were obtained. The degree of liberation for each size interval was determined by an SEM/energy dispersive x-ray technique and found to be independent of the extent of size reduction, suggesting that liberation occurs by indiscriminate fracture and not by detachment. Up to 89 percent liberation was realized for a minus 400 mesh grind. Under these circumstances, the quality of separation appears not only to be limited by the remaining locked particles, but, also, by contamination of the alunite concentrate with fine quartz gangue (< 2μm). The best separation for the minus 400 mesh grind was a rougher recovery of 88 percent with a grade of 65% alunite at 50°C.

INTRODUCTION

Aluminum which is among the most abundant elements in the earth's crust is found in a number of fairly common minerals such as alunite, anorthosite, bauxite, clays and laterite. However, only bauxite is used as a feed material for the commercial production of aluminum. The United States, which produces one-third of the world's aluminum, imports ninety percent of its bauxite (1). Cost and supply of bauxite represent a potentially serious long term threat to the U.S. aluminum industry. The importance of the development of an alternate technology based on the use of non-bauxite domestic resources is therefore, apparent.

Several processes have been developed for the production of aluminum using domestic ores. Prominent among these are the nitric acid and hydrochloric acid-ion exchange processes for clay, the lime-soda sinter process for anorthosite and the modified Bayer process for alunite (1-7). The non-bauxitic resources are at present economically unattractive as compared to the imported bauxite due to several factors such as; lower alumina (25-35%) content compared to bauxite (> 50%), complex processes required to convert the ore to a suitable form (Al_2O_3 or $AlCl_3$) for aluminum production, and very high energy requirements (1). The rising cost of bauxite, its transportation cost, and the desire for self sufficiency may make alternate sources of alumina more attractive in the future. At least, ". . . the knowledge that a technically proven process exists will set a ceiling on bauxite prices and serve notice on bauxite producers that the U.S. could if necessary, satisfy its needs for alumina from its own sources." (2)

It was, therefore, not surprising that the National Materials Advisory Board in 1970, undertook a study of processes for extracting alumina from non-bauxitic sources (8). The study did not include alunite because it was completed before the large alunite deposits in Utah were discovered. These reserves which are estimated to be 800×10^6 tons at 35% alunite are more than adequate to sustain a viable industry of the size of 200,000 tons per year for 30 years. Extensive pilot plant studies of the thermochemical processing of alunite ore from southwest Utah have been completed by Alumet, but economic factors have delayed the construction of a full-scale plant.

Alunite Processing

The proposed thermochemical process for alunite ore would produce in addition to alumina, a potassium sulfate fertilizer and sulfuric acid (to be interfaced with a phosphate plant to produce triple super phosphate fertilizer) (9). The proposed process involves an initial dehydration followed by a reduction roast from which sulfuric acid would be produced. The reduction roast is followed by a water leach and crystallization from which potassium sulphate would be produced. The residue from the water leach would be feed material, synthetic bauxite, for a modified Bayer process producing alumina, the primary product. The flowsheet for the proposed alunite plant is shown in Figure 1.

The major drawback of the proposed process strategy is the relatively large amount of silica (about 50%) which will be processed and ultimately discarded. This large amount of silica necessitates the need for larger process equipment, reactors, filters, and higher energy requirements especially in the roasting steps. Further, it will increase the level of contamination in the final alumina product. Since this siliceous gangue will ultimately end up in the tailings pond, it seems unnecessary and wasteful to process this unwanted constituent of the alunite ore. Considerable advantage might be gained with regard to energy conservation, plant efficiency, and plant capacity if the siliceous gangue could be removed after crushing and grinding and before the roasting steps.

In this regard a research program has been initiated to develop a flotation technique for the separation of alunite from the siliceous

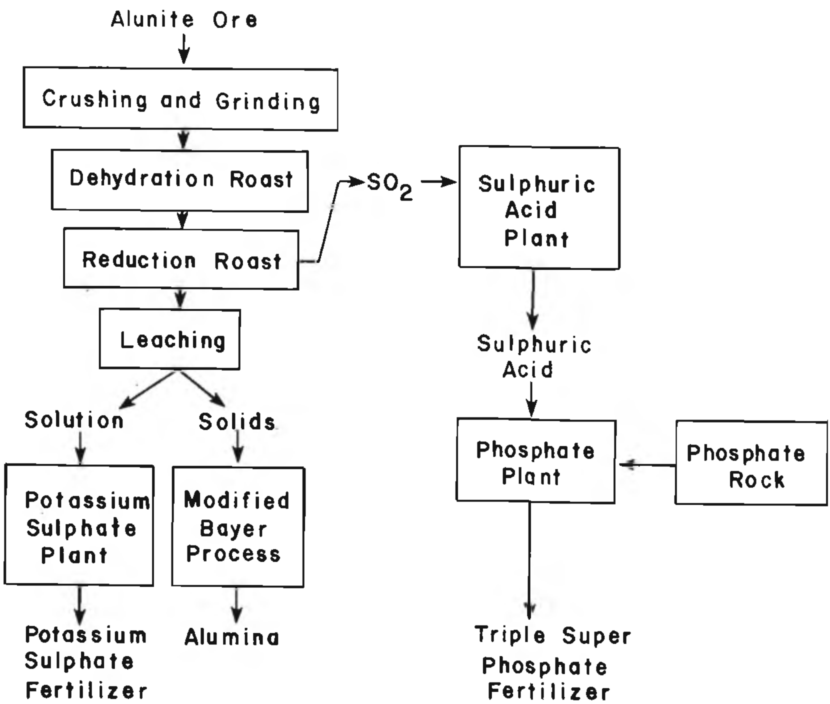


Figure 1. General flowsheet for the proposed thermochemical alunite plant.

gangue which consists primarily of quartz.

Flotation of Alunite

Alunite can be classified as a semisoluble salt mineral (10) and may be represented by the formula $KAl_3(SO_4)_2(OH)_6$ which does not completely describe the complex structure of this mineral. Although alunite has a complex structure, its flotation behavior is expected to be similar to that of other salt-like minerals. However, very little research has been done regarding the flotation of alunite. An investigation (11) by U.S.B.M. in 1942 indicated that complete separation of alunite from silica at 100 mesh would be difficult due to the presence of significant amounts of locked particles. The maximum grade achieved was 73% alunite at an unsatisfactory recovery of 40%. The quality of separation at smaller particle sizes with better liberation was not studied. Only one study of fundamental nature has been reported (12). Optimum flotation of alunite with oleic acid was reported to be achieved between pH 6.2 and 9.6. Other results from the study of alunite surface chemistry at the University of Utah have provided further insight into the nature of oleic acid adsorption on alunite. These results are being reported in a separate communication (13). The available evidence suggests that oleate is chemisorbed by alunite.

In addition, bench scale, fine particle flotation of alunite ore with oleic acid has been investigated as a function of temperature, collector addition and depressant addition in terms of the extent of liberation and transport of fine gangue into the concentrate. The results of this latter research program are presented in this contribution.

EXPERIMENTAL TECHNIQUES

The bench scale flotation experiments were designed to establish conditions under which satisfactory separation of alunite from the siliceous gangue constituent could be achieved (14). In addition, the results were analysed in terms of the extent of liberation and other phenomenological effects to explain the quality of separation achieved under a given set of operating conditions. Alunite ore used in these experiments was from the Wah-Wah Mountains in Southwestern Utah near Milford, the composition of which is given in Table I.

Flotation Experiments

The following experimental procedure was used for the bench scale flotation experiments:

1. Samples were prepared by crushing the ore to minus 10 mesh and the required quantities of material were collected by coning and quartering. The material was further split into

Table I. Composition of Alunite Ore from Southwestern Utah.

Chemical		Mineralogical	
Element	Wt.%	Mineral	Wt.%
Al	9.66	Alunite	45
K	4.20	Quartz	55
Na	0.21		
S	7.58		
Si	22.50		
Fe	0.80		

300-1000 gm samples using a Tyler 16 to 1 splitter and riffle.

2. Grinding was done in a 10 inch laboratory tumbling mill. 1000 gms of ore were wet screened on the desired mesh and the coarse fraction was wet ground at 60% solids for 10 minutes. This was followed by screening and returning the oversize to the mill for another 10 minute regrind. Three stage grinding was usually enough to obtain about 80-90 percent product passing a 400 mesh screen.

3. Desliming, when necessary, was done at approximately 11 microns using the Warman Cyclosizer Model M-2. The slimes were collected in a 55 gallon drum.

4. Conditioning of the 15% solids slurry was done for 20 minutes at 600 rpm. The reagents were added in the following order: 1)Aerosol GPG 2)sodium silicate 3)potassium oleate with 1 to 2 minutes between additions. The desired temperature and pH were maintained by heating with hot plate and by appropriate additions of KOH and/or HCl when necessary.

5. Flotation was carred out in a 2 liter Galigher flotation cell at the desired temperature (± 2°C) for 5 minutes. The air flow rate was maintained at 5000 cc/min. The concentrate would represent the froth product from the first stage. Subsequent stages were carried out by turning off the air, reconditioning for 5 minutes in the cell itself with fresh additions of reagents followed by flotation. The first five stages were sometimes lumped together and called the rougher concentrate, while concentrates from stages number 6 through 10 were called the scavenger concentrate. The rougher concentrate was sometimes put through a cleaning stage to obtain a cleaner concentrate and cleaner tails. The processing flowsheet is depicted in Figure 2. The samples were analysed by a weight loss technique (14). Frother was added in a few experiments, but its use was discontinued subsequently when it was found to be of no advantage. Standard conditions for flotation experiments are given in Table II.

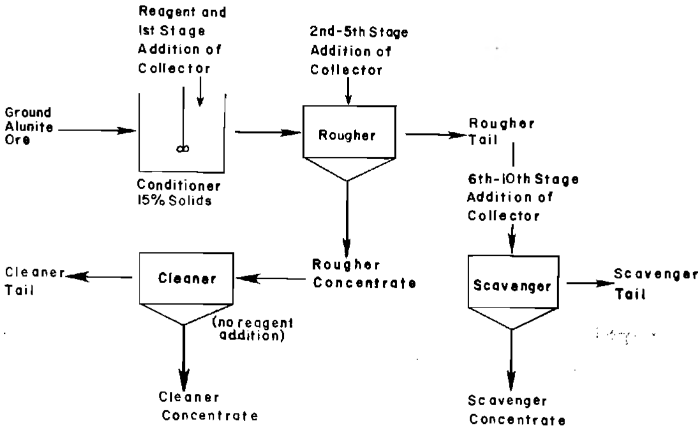


Figure 2. Standard experimental scheme for the bench scale flotation of alunite ore.

Table II. Standard Conditions for Bench Scale Flotation Experiments.

Condition	Remarks
Reagents:	Oleic Acid, 0.2 pounds per ton per stage
	Sodium Silicate, 4 pounds per ton
	Aerosol GPG, 1 drop
pH:	8
Grind Size:	-400 mesh
Conditioning Time:	20 minutes initially and 5 minutes between stages
Flotation Time:	5 minutes/stage
Temperature:	50°C
Percent Solids:	15 percent
Air Flow Rate:	5000 cc/minute
Impeller Rotation:	800 r.p.m.

Liberation Study

The samples were analysed using the Hitachi series 500 scanning electron microscope with an x-ray elemental analysis attachment (KEVEX). The following procedure was used to determine the degree of liberation.

1. Crushed feed (+ 48 mesh) was stage ground to obtain -48 mesh undersize.

2. Using screens and cyclosizer various sized fractions with average sizes between 12 microns and 48 mesh were obtained.

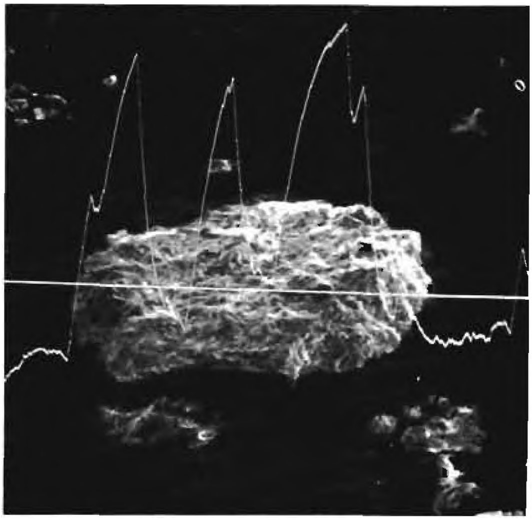
3. A particular size fraction was then mounted on the sample stage using double stick cellophane tape or aqueous colloidal graphite by collecting a random sample on the stage surface.

4. The samples were then coated with a very thin layer of carbon to make the particles conductive.

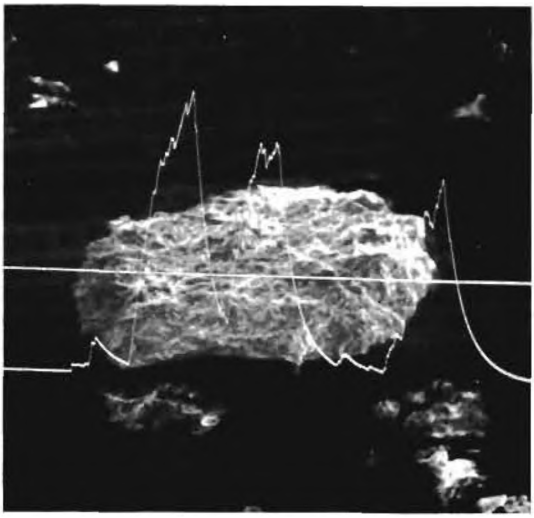
5. It was possible to determine the presence of alunite or quartz in the particle with the help of the KEVEX attachment. Silicon elemental analysis was indicative of quartz while sulfur analysis was taken to indicate the presence of alunite. Generally two line scans were taken for each particle at different sections. If both scans indicated a single phase the particle was considered liberated. If a distinct second phase was present, regardless of the relative amount, the particle was designated as locked. The relative number of x-ray counts due to quartz and alunite was also taken into account to determine liberation. The particle was scanned extensively if a large number of counts were registered even though no distinct peaks for the second phase were observed (such particles were very few 5 to 10% of the total number). The preferred technique for the identification of a liberated particle that evolved over the course of this investigation involved a normalization with respect to the intensity observed for "pure" samples of alunite and quartz. A typical SEM photograph of a locked particle is shown in Figure 3. The straight line represents the position of the analysis scan and the other trace represents the intensity of a given phase. It can be seen that where one phase ends the other phase begins, revealing the particle's locked nature.

Particle Size Measurement

A number of particle size measurements were done, not only for the liberation study, but also to analyse flotation data as well. For these measurements a representative sample was analysed by the Leeds and Northrup Microtrac. In these measurements it was suspected that some aggregation of the particles



A. Traverse of locked particle for sulfur (alunite) by energy dispersion analysis showing regions of alunite predominance.



B. Traverse of locked particle for silicon (quartz) by energy dispersion analysis shownina regions of quartz predominance.

Figure 3. SEM photographs of a 48 x 65 mesh particle illustrating that acceptable liberation is not achieved at this particle size and that further size reduction will be necessary in order to achieve a quality separation.

might be occurring so dispersion was enhanced by the use of an ultrasonic probe and by washing the samples in ether to remove any surface chemicals present. Other particle size measurements were made using sieves and the cyclosizer. The size given by the cyclosizer was used as a reference point and measurements by other methods were adjusted to it.

EXPERIMENTAL RESULTS AND DISCUSSION

The experimental results to be discussed represent a portion of the first stage of research in a comprehensive study of the flotation separation of the mineral alunite from its siliceous gangue constituents. Bench scale flotation tests and a liberation study were conducted on an alunite ore from the Wah-Wah Mountains in southwestern Utah. This ore can be considered to be a binary mixture containing approximately 45% alunite and 55% quartz.

The quality of separation is assessed by the grade (product purity) and recovery (component distribution) achieved. In addition to determining these parameters, the quality of separation is measured by the coefficient of separation, a single parameter characterization, which has not been used extensively but provides an excellent assessment of the effectiveness of the separation. As can be shown from mathematical arguments for a binary system, the coefficient of separation is calculated as follows,

$$CS = R_1^a + R_2^b - 1 \tag{1}$$

where

CS - coefficient of separation, expressed as a fraction

R_1^a - fractional recovery of component a in product 1

R_2^b - fractional recovery of component b in product 2

Consideration of the derivation of this expression reveals the physical significance of the coefficient of separation; the fraction of the feed material which undergoes an ideal, or perfect, separation.

Bench Scale Flotation

Oleic acid was selected as collector for this study based on the flotation response of the pure mineral alunite with various collectors (13,14). The standard conditions used in bench scale flotation were already presented in Table II.

Particle Size. Particle size was thought to be an important factor due to the apparent small liberation size of the ore, later confirmed in the liberation study. In this regard, a series of experiments were completed using various grinds, where the top sizes were

150, 200, 270 and 400 mesh. The standard conditions for flotation were used. These results are presented in Figure 4 in terms of cumulative grade versus recovery and cumulative coefficient of separation versus recovery for the various size distributions. All of the data are plotted on the one figure to emphasize the similarity of the results. As one can see in Figure 4, the data essentially fall on the same general grade-recovery curve that starts at about an 80 percent grade and falls to a 60 percent grade at 95 percent recovery. A closer look, however, reveals that the coarser grinds do not attain very significant recoveries, while the finest grind (-400 mesh) extends the grade-recovery line to high recoveries. This is illustrated by the -150 mesh grind where 5 stages of flotation only brought the recovery to 40 percent while the -400 mesh grind yielded over 90 percent recovery in 5 stages. Furthermore, only at the finest grind does a maximum occur in the coefficient of separation, indicating that finer grinding improves the overall quality of the separation. A not so obvious fact shown in Figure 4 is that reagent economy is significantly improved by fine grinding. Figure 5 plots the data from these tests as recovery versus pounds per ton of collector used. It can be readily seen that the amount of collector used to achieve any given recovery is less for the finer grind.

Temperature. It has beem demonstrated by microscale flotation testing, that increased temperature improves the alunite flotation response with oleic acid (13,14). To quantify the effect of temperature five standard bench scale tests were run at temperatures varying from 20° to 80°C. These data are presented in Figure 6 with the cumulative grade, and coefficients of separation plotted against the cumulative recovery. Again, one can see the now familiar -400 mesh grade-recovery curve that results regardless of the variables imposed. There seems to be some indication that the low temperature of 20°C leads to poorer results but all of the other temperatures form the same general curve. Here the higher temperatures tend to extend the line to higher recoveries. By plotting this data as cumulative recovery versus pounds per ton of collector, the effect of temperature on the response becomes evident (Figure 7). The most dramatic improvement in collector economy occurs in going from 35° to 50°C, but going to higher temperatures does not yield much improvement. Thus 50°C is an appropriate temperature at which to carry out flotation.

Sodium Silicate. It was believed that sodium silicate would affect flotation through dispersion of fine particles and improve the depression of quartz. However, it was found that it actually had no significant effect on the system at pH 8 other than nonselective depression of both quartz and alunite. Recent research by Bunte (15) shows that a similar response is observed at pH 10 as well. Experimental results are presented in the quality of separation plot presented in Figure 8.

Frother. MIBC, Dow froth 250, Aerofroth 65 and pine oil were

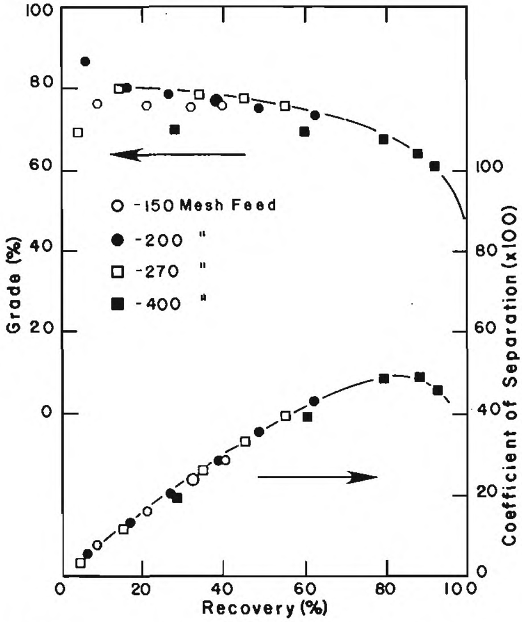


Figure 4. Cumulative grade and coefficient of separation vs. recovery for concentrates from successive stages of flotation of various feed sizes.

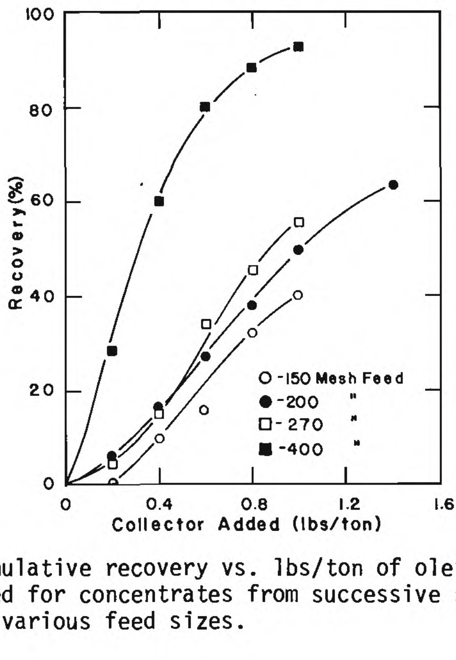


Figure 5. Cumulative recovery vs. lbs/ton of oleic acid collector used for concentrates from successive stages of flotation of various feed sizes.

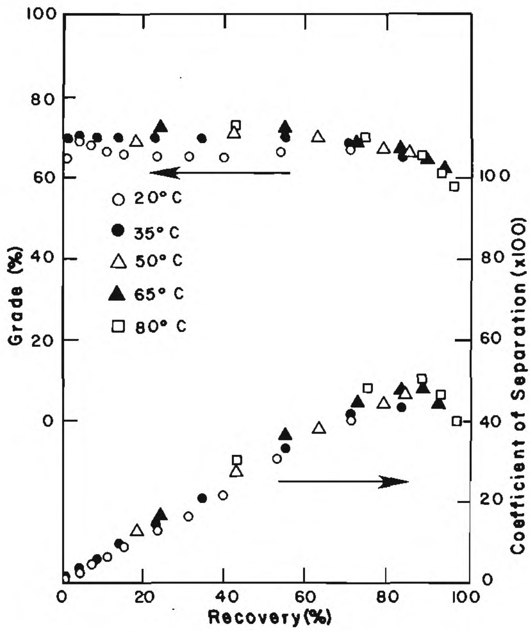


Figure 6. Cumulative grade and coefficient of separation vs. recovery for concentrates from successive stages of flotation of -400 mesh feed at various temperatures.

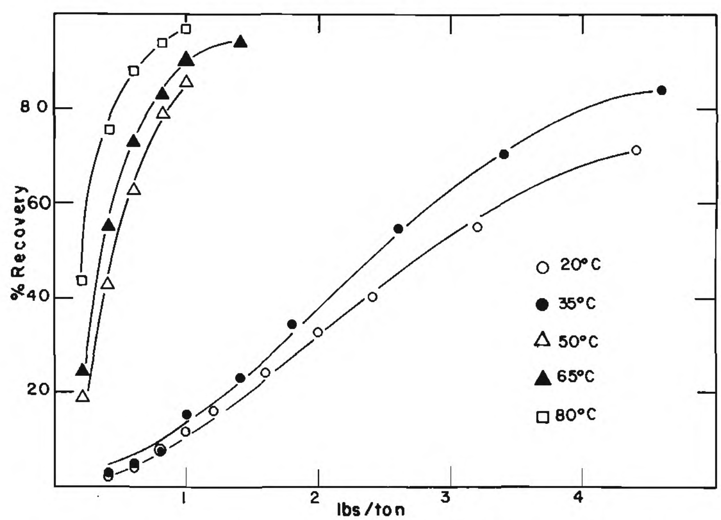


Figure 7. Cumulative recovery vs. lbs/ton oleic acid collector added for concentrates from successive stages of flotation of -400 mesh feed at various temperatures.

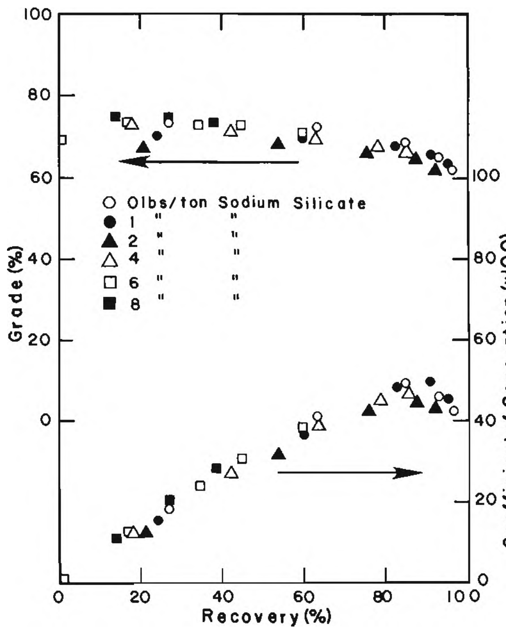


Figure 8. Cumulative grade and coefficient of separation vs. recovery for concentrates from successive stages of flotation of -400 mesh feed at various sodium silicate concentrations.

tried as frothers. As mentioned earlier frother was not required since sufficient froth formed even without a frother. Frother type had no affect on grade or recovery.

Percent Solids. Similarly, variation in the percent solids from 10 to 30 percent did not significantly influence the flotation response.

Desliming. The most significant improvement in the grade of the concentrate occurred when the feed was deslimed at a cut size of 11 microns. The average amount of -11 micron slimes present in a typical -400 mesh grind was 40 percent. A series of flotation tests was done for grinds from -48 to -400 mesh to investigate the effect of desliming. The experimental data is plotted in Figure 9.

The standard flotation procedure adopted in the bench scale flotation tests generally used -400 mesh ore and at this grind one could expect about 90 percent overall liberation. As expected, desliming of the feed material at 11 microns resulted in a lower degree liberation of the deslimed feed, about 80 percent. Nevertheless a

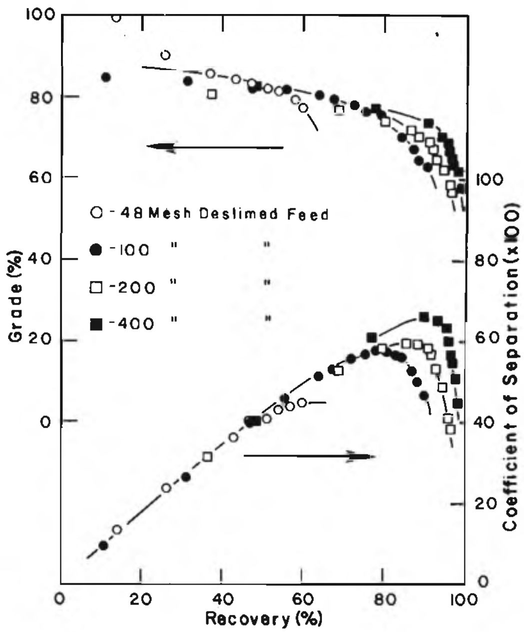


Figure 9. Cumulative grade and coefficient of separation vs. recovery for concentrates from successive stages of flotation of various deslimed feed sizes.

significant improvement in the quality of separation from the deslimed feed was achieved with a 10 to 15 percent improvement in grade for any given recovery. On the basis of these results it appears that the quality of the separation for the standard -400 mesh grind is limited by the presence of slimes and to a lesser extent by the presence of locked particles.

Cleaning of the rougher concentrate was done with no further reagent additions and the results are shown in Table III. This table also gives a good comparison between the results obtained with the -400 mesh feed and the -400 mesh deslimed feed. Note, for example, that a cleaner concentrate containing 95% alunite was produced using deslimed feed.

Liberation Study

The liberation study was carried out to determine whether or not locked particles were present in the feed and whether their presence might significantly affect the grade of the concentrate. The results of the liberation study are presented in Figure 10, where the percent of liberated particles found in any one size fraction is

Table III. Results Obtained from Cleaning of the Rougher Concentrate.

Feed Size	Product	Grade % Alunite	Recovery % Alunite	Coefficient of Separation, %
-400 mesh	Rougher Conc.	69.4	80.9	50.1
	Rougher Tail	19.2	19.1	
	Cleaner Conc.	86.8	49.9	43.4
	Cleaner Tail	52.5	31.0	
	Scavenger Conc.	39.4	10.8	
	Scavenger Tail	11.6	8.3	
-400 mesh Deslimed	Rougher Conc.	83.6	89.3	74.2
	Rougher Tail	10.6	10.7	
	Cleaner Conc.	95.0	58.7	56.1
	Cleaner Tail	68.0	30.6	
	Scavenger Conc.	48.3	3.3	
	Scavenger Tail	7.2	7.4	

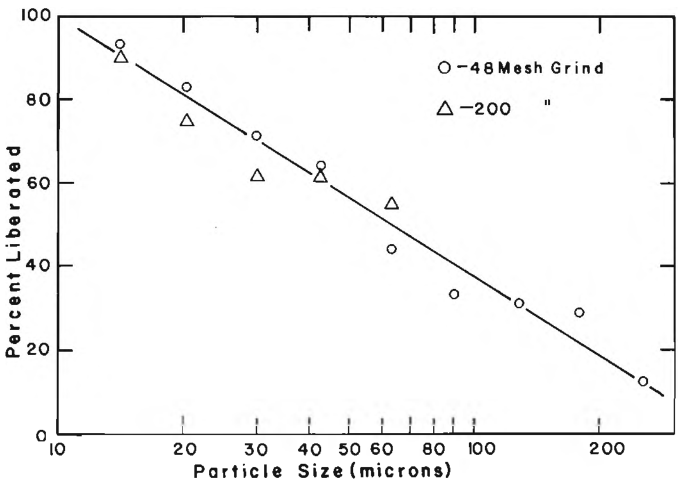


Figure 10. Liberation curve for two different size grinds of southwestern Utah alunite ore.

presented as a function of the geometric mean size of that interval. The results for two different grinds, -48 and -200 mesh, are shown, and the data indicate that complete liberation occurs at approximately 10 microns. Also, note that the extent of liberation in any given size fraction is independent of the fineness of grind. This is indicated by the fact that the results of the -48 and -200 mesh grind are coincident. Actually, one can visually observe a distinct change in the character of the particles below 16 microns. Here virtually all of the particles appear as single crystals rather than a group of cyrstals and are thus liberated. In this regard, complete liberation can be considered to occur somewhere between 10 and 16 microns. Clearly, liberation occurs by size reduction rather than by a detachment process.

The liberation study indicates that feed particles in the 400 mesh size region are only 60 percent liberated. By using the liberation curve and the size distribution of the -400 mesh feed the percent liberation for this grind can be calculated. Such an excercise is shown in Table IV for a regular grind as well as for a deslimed -400 mesh grind. Summation of the amount liberated over the entire distribution results in the percentage of overall liberation in that particular feed size distribution. From these computations it is seen that grinding to -400 mesh results in approximately 89 percent overall liberation and 80 percent overall liberation for a -400 mesh deslimed grind. The deslimed feed has a lower liberation because all of the material removed in desliming is liberated, leaving behind a larger proportion of locked particles. These figures can also be used as an indication as to what a maximum possible grade-recovery point might be. For instance, with a regular grind a maximum recovery of 89 percent would be possible at a 100 percent grade. Looking at some of the concentrate products from a standard flotation test (see Table V, in this case the cleaner concentrate was pulled more extensively than it was for the results reported in Table III) one can see that the locked particles distribute themselves among the products fairly evenly. The cleaner concentrate, the cleaner tail, and the scavenger tail all show 5 to 7 percent locked particles. The one exception is the scavenger concentrate (stages 6-10) where the second pound per ton of oleic acid was added, and 13 percent locked particles were observed. Because this product represents only 3 percent of the alunite recovery it is relatively insignificant. It may, however, still indicate that with increasing collector, larger proportion of locked particles are floated. It should be noted that the percentages of locked particles observed is 7 to 15 percent lower than expected from the results of the liberation study. This may be due to a less accurate counting procedure. Here, the samples were observed as is, and not broken down into small size fractions and this changed the number of particles counted from 400 to 100 per sample.

Because the locked particles tend to distribute themselves among the products uniformly, and because better results are obtained for

Table IV. Calculation Showing the Total Percentage of Liberated Particles.

Normalized Geometric Means of Size Interval	Weight % of Size Interval i	% Liberated of Size Fraction i as Determined From Figure 10	% of Feed of Size i that is Liberated (Columns 2 x 3)
	-400 Mesh Grind		
87	0.23	40	0.09
61	0.00	52	0.00
43	3.55	60	2.30
30	10.83	71	7.69
22	18.34	78	14.31
15	18.65	88	16.41
11	14.02	97	13.60
7.6	12.17	100	12.17
5.4	7.75	100	7.75
3.8	6.64	100	6.64
2.7	4.46	100	4.46
1.5	3.35	100	3.35
			88.77% Liberated
	Deslimed -400 Mesh Grind		
87	0.03	40	0.01
61	0.10	52	0.05
43	8.74	60	5.24
30	21.30	71	15.12
22	29.13	78	22.72
15	24.17	88	21.18
11	12.32	97	11.95
7.6	4.12	100	4.12
5.4	0.17	100	0.17
3.8	0.00	100	0.00
2.7	0.00	100	0.00
1.5	0.00	100	0.00
			80.56%

Table V. Liberation Observed in the Products of a Standard Deslimed -400 Mesh Flotation Experiment.

Product	Locked Particles %	Grade %	Recovery %
Cleaner Concentrate	5	81.4	83.3
Cleaner Tail	7	49.7	6.4
Scavenger Concentrate	13	44.0	3.0
Scavenger Tail	6	6.8	6.2

deslimed feed, although the overall liberation caused by desliming decreases, it would appear that the presence of locked particles is not a major factor in limiting the quality of separation. In a -400 mesh grind the locked particles are only 10 percent of the total. The other possible factor leading to dilution of the concentrate, besides locked particles, is that of the gangue slime particles being transported into the froth phase and removed as concentrate (see Figure 11). This is probably the major contribution to the inefficiency of the separation for two reasons. First, it has been demonstrated that desliming will improve the grade-recovery curve, and secondly, desliming improves the curve despite the fact that about 80 rather than 89 percent by weight of the +11 micron material is liberated.

Two methods of slime transport can be identified and these are transport by slime coating and by mechanical entrapment. The case for transport by slime coating is not likely for two reasons. First, if the slime coatings adhered to the surface it would seem to be difficult to achieve the high grades realized in the cleaner concentrate, and this is not the case since cleaning improves the grade. Second, the use of sodium silicate in various concentrations in the system would be expected to prevent slime coating from ocurring.

The method of transport by mechanical entrapment and/or by convective transfer also appears tenuous. Increasing the froth volume would lead to an increase of transport by a convective process simply because of the increase in the flow of water into the concentrate. The increased amount of froth itself should deter drainage and help entrap particles indiscriminately. This, however, was not observed in experiments dealing with frothers (14). Also, increased percent solids would lead to increased entrapment of fine particles between larger particles as they rise to form the froth, lowering the concentrate grade and this, too, was not evident.

It should be mentioned that diagnostic tests provide no evidence to support the idea that quartz is being activated. It appears that

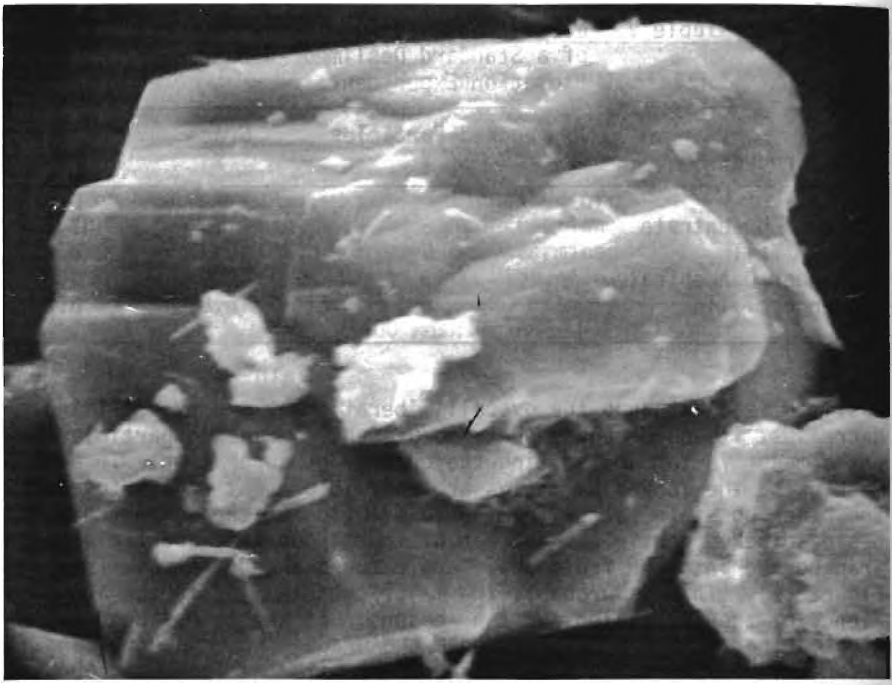


Figure 11. SEM photograph of a 20 micron alunite particle and associated quartz slimes from a concentrate product.

dilution of the concentrate is occuring by indiscriminate entrapment of fine particles, since desliming significantly improves the quality of separation. Because variations in factors that lead to the transport of fine particles into the froth, namely percent solids and froth volume did not alter the flotation response, some other factor must be taken into consideration in order to explain these results.

At this time, it is not possible to assess the exact mechanism by which the slimes affect the quality of separation. But it is evident that the presence of slimes, and not the activation of quartz or the presence of locked particles, is the main reason for the dilution of the concentrate grade.

CONCLUSIONS

1. Alunite responds very well to oleic acid flotation and higher temperatures improve the metallurgical results significantly. A minus 400 mesh grind was required to achieve 89 percent liberation and under these circumstances the best separation resulted in a rougher recovery of 88 percent with a grade of 65 percent alunite at 50°C.

2. Examination of the products by SEM and energy dispersive x-ray analysis indicated that the contamination of the concentrate occurred by indiscriminate entrapment of fine quartz particles (< 2μm).

3. The quality of separation achieved by oleic acid flotation was found to be insensitive to sodium silicate addition at pH 8.0.

4. Desliming of the feed resulted in an improved concentrate grade of up to 95% alunite at a recovery of 58% in the cleaned product.

5. Liberation of particles in a particular size fraction during size reduction is independent of the extent of grinding.

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