FROTH FLOTATION
OF THE DAISY MINE FLUORITE ORES,
NYE COUNTY, NEVADA

A THESIS
SUBMITTED TO THE FACULTY OF THE UNIVERSITY
OF NEVADA IN PARTIAL FULFILLMENT OF THE REQUIREMENTS
FOR THE DEGREE OF MASTER OF SCIENCE

by

Donald K. McCready
Reno, Nevada
June, 1965
The writer wishes to express his sincere appreciation for the guidance of Professor Claude W. Hammond during the preparation of this paper, and to J. Irving Cramoitt Jr. and John Cramoitt for their help during the sampling of the Bogue step.

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Acknowledgements

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Abstract

Batch flotation tests were conducted with samples taken from the Daisy fluorite mine to determine some of the problems facing a future flotation operation. The 8 samples selected for study were those containing fluorite concentrations greater than 36 percent. The remaining 5 samples were rejected because of their low grades. These materials varied widely in their mineralogical composition as the sampling program was intended to accentuate the effects that grade and mineralogy would have upon flotation results.

These results, summarized in the data sheets of the Appendix, show that flotation of these ores to a 97 percent fluorite concentrate is possible through the use of a split product flowsheet. Fluorite, still unliberated after a 200 mesh grind and sericitic slimes may be removed to a steel grade product of 72 percent purity. The use of Quebracho, sodium meta silicate and starch as gangue depressants proved to be most effective when an oleic acid and tall oil collector combination was utilized. Mixtures of oleic acid and tall oil gave selective froths which allowed a more effective use of the depressants.

After three cleaning cycles and a slime separation, concentrate grades above 97 percent with less than 1 percent total insoluble matter were obtainable with recoveries ranging from 85 to 93 percent. Two samples containing large amounts of calcite and sericite gave the most difficulty with the final concentrate grades being greater than 96 percent but having high carbonate or insoluble residue assays. These
samples yielded recoveries greater than 94 percent. Dolomitic ores with fluorite present as minute inclusions were responsible for the lower recoveries.

Products obtained through the use of Amargosa River water from the Beatty area were identical to those produced through flotation in Reno water. The only noticeable flotation effect of the native water was that froths derived from Amargosa River water were slightly less stable, and small additional amounts of collector were needed.

**Introduction**

The beneficiation of fluorite ores to facilitate their use in modern industry is presently receiving an increasing amount of study. Steel and chemical plants are demanding high grade products which must meet strenuous physical specifications as well. With the advent of the modern oxygen steelmaking processes, fluorspar ores and concentrates must be sized so that their use in furnaces will conform to the optimum conditions of control and efficiency. Generally, size requirements are met by agglomerating the product after a fine grinding. Pelletizing of this material has met with the most universal acceptance and is preferred to a screened ore product, termed gravel spar, which is now readily available from Mexican sources. Plants now producing pelletized fluorspar products have found that flotation concentrates are readily adaptable to the agglomerating operation due to their fine grained nature. In this case, the expense of grinding an ore provides a means of upgrading the fluorspar content by flotation and at the same time provide the required finely sized feed for the pelletizing process.
Nevada has several fluorspar occurrences, but only one mine is presently shipping fluorite products. The Daisy mine is a unique occurrence in this country because here it is possible to mine a fluorspar vein selectively enough to produce a directly marketable ore. The main buyers for this material are found in the California steel industry. But introduction of the oxygen furnace has already begun, and as this process replaces open hearth production, all material fed to the furnaces, including the fluorspar fluxes, will have to meet size specifications. Selective mining has required that the operators of the Daisy mine leave a large tonnage of less than 70 percent fluorite ore in place where it is now available for milling to an acceptable steel grade product. California also supports a sizable acid grade fluorspar market, and the flotation process for the steel grade production might well provide a concentrate that could be supplied to these consumers as well.

With these facts in mind, this study was conducted to determine the possibility of beneficiating this ore by flotation into two grades of fluorite, each being acceptable for industrial usage. The results will hopefully be of use to the operators of the Daisy mine, and this writer trusts that this Nevada enterprise will maintain its market position despite heavy foreign competition.

**Property Description**

The Daisy mine is located in Fluorspar Canyon on the northern flank of Bare Mountain in the NWk, sec. 23, T. 12 S., R. 47 E. of Nye County, Nevada. This area, the Bullfrog District, boomed in the early 1900's as a gold camp. Beatty, Nevada lies about 5 miles to the west of the mine and is approximately 114 miles northwest of Las Vegas. Both the
town and the mine are shown on the U.S.G.S. topographic maps of the Death Valley, Bare Mountain and Bullfrog quadrangles.

The property was originally located by the Continental Fluorspar Company, under the direction of J. Irving Crowell. J. Irving Crowell Jr. took control of the claims in 1927, and the Daisy mine has maintained a fairly active production for the past 38 years. The total production as compiled by Cornwall and Kleinhampl for the period, 1919 to 1961, is close to 118,000 short tons. Ore grades in the past have ranged from 70 to 85 percent fluorite with an average of on the order of 75 percent. In recent years the mine has realized an increase in some shipping grades due to a decreasing concentration of insoluble material. Jack Crowell, one of the elder Crowell's two sons, has stated that on occasion, material mined on the lowest levels will contain fluorite having less than 1 percent insoluble residue and no more than 9 percent carbonates.

Geological Description

The following outline of the geology pertaining to the region associated with the Daisy mine is largely summarized from the work of Cornwall, H. G. and Kleinhampl, F. J., Geological Survey Professional Paper 454-J, 1964.

Most mineralization is observed to be closely related to the margins of a large collapse structure centered about 6 miles west of Beatty. This area is almost entirely covered by Tertiary volcanic flow rocks. Three precious metal deposits are located in the eastern rim of this caldera which is about 2.5 miles west of Beatty, and these can be seen in Fig. 1, page 5. However, the only substantial fluorspar deposit, the Crowell property, occurs in the pre-Tertiary rocks composing Bare Mountain.
Tertiary volcanics also appear to the north of Fluorspar Canyon, and a
down thrown section, most probably a graben like structure, expresses
its southern boundary in the canyon relief.

Pre-Tertiary rocks of Bare Mountain have been identified by U.S.
G.S. workers as being of Paleozoic age. These sediments consist of
mainly dolomites, limestones and quartzites. Various shales and silt-
stones are also to be found. The Daisy mine itself, is located in a
highly faulted part of a late Cambrian member of the Nopah formation.
This dolomite has in some extent been subject to a chemical interaction
with mineralizing hydrothermal solutions. Minute inclusions of dolomite
in the fluorite evidences either a replacement or co-precipitation tex-
ture, and this has had great significance in the flotation of several
samples taken in these wall rocks.

The main control over the ore deposition has been exercised by two
prominent trends of faulting. One set strikes roughly northeast and
dips vertically or steeply eastward. Ore shoots extend along and are
bounded laterally by faults of this type. But the distribution of the
ore seems to be more dependent on a second set of faults that strike
northwest and dip gently to the northeast. Both trends exhibit extreme
variations in strike and dip, and the ore shoots are invariably bounded
by the gouge zones of these faults. Cornwall and Kleinhampl state that
these gouge zones have apparently served to restrict the ore solutions
to definite channels where the fractured dolomite was almost completely
replaced by the flourine bearing solutions. Low grade ore of this type
is quite common, and its extent will demand a good deal of consideration
when beneficiation is considered.
Presently mined ore zones are composed mostly of very fine grained fluorite interbedded or seamed with gouge material. Very few sizable fluorite crystals can be seen in the high grade shoots. However, when the commonly found calcite veins are encountered, the fluorite assumes a comb structure. The color of the fluorite when found in calcitic areas will generally be white, yellow, or a very pale blue to purple. Many times, crystals of clear calcite and quartz are found in the vug fillings with the fluorite. Very infrequently, cinnabar is also discovered in the calcite vugs.

The U.S.G.S. investigators report that the fine gouge associated with the fluorite shoots is mainly composed of the clay mineral, montmorillonite. This investigator encountered considerable difficulty during the initial phases of this study because one of the gouge minerals was extremely floatable. Upon further study of the flotation concentrates and tailings with the X-ray Diffractometer, it was found that sericite was present in large amounts in most of the samples collected for this investigation. It is felt that much of the montmorillonite is actually the micaceous sericite.

Cornwall and Kleinhampl have also commented on the high level of uranium content. Values from 0.007 to 0.015 percent were encountered in the purple fluorite areas. The highest values were said to be associated with the clay inclusions in these zones.

Mining Methods

Selective mining of the Crowell property has required a very irregular approach. Since the heavy faulting in the deposit has made the
ore zones unpredictable in their location and extent, a technique of ex-
ploration where one drifts, raises, and winzes in ore as much as possible
has been developed. Consequently, the mine after years of operation, is a
veritable maze of workings as can be witnessed by the composite plan maps
in Fig. 2, page 9. Mining is presently being carried out in the 13th
level which is over 600 feet below the shaft collar. As the selective
mining concerns itself only with the high grade shoots, very little blast-
ing is needed. The ore in these areas is so soft and free running that
in most cases it can be removed with very little effort. The daily ton-
nage is now held at close to 15 tons using a three man underground crew.
This amount is easily provided for with a minimum of drilling and blasting.
If necessary, the Crowell's claim that the mine can increase its produc-
tion to a maximum of around 700 tons per month--the main limiting factor
being the single compartment Daisy shaft. The drilling, when required,
is done with a standard air leg drill, whereas in the mid 1940's, the
miners used hand augers. Once the material is dislodged, it is easily
slushed into short ore passes. The ore is hand trammed from the ore
chutes to the shaft station where it is transferred to single car cap-
acity, 24 by 36 inch hoisting buckets and is taken to the surface. Once
the ore is removed, the stopes are generally left unfilled. The dolomitic
wall rock is very competent, and stopes mined out in the early twenties
are still holding solidly. Very little backfilling with low grade waste
has been practiced. Material from headings not in metallurgical grade
fluorspar is hoisted to the surface. Then it is separated in the dumps,
with the barren rock being kept from material that in the future may be
beneficiated. This evidently has been a long standing practice because
the U. S. Bureau of Mines sampling program, conducted in 1945, showed that
MINK WORKINGS OF THE DAILY FLUORSPAR MINE, NYE COUNTY, NEVADA

Figure 2.
large portions of the Daisy mine dump averaged over 50 percent fluorite. Once at the surface, the ore is dumped into a coarse ore bin. Rock from the bin is screened at 1 inch, and the oversize is crushed in open circuit. The crusher product and the original minus 1 inch material are then bucket elevated into one of two IH dump trucks. The photographs below show the ore bin, crusher and surrounding surface installations.

Figure 3: Mine Surface Installations

The 4 mile downgrade haul to Highway 95 is made on a graded gravel road while the last mile northward to Beatty is made on the pavement of the U. S. Highway. J. Irving Crowell Jr. maintains a stockpile of fluorspar ore at his residence in town. The ore is stored here until a California buyer places an order. When this occurs, the Crowells load their 27 ton
IH semitrailer truck with a small tracked end loader, and the material is driven to the railhead at Las Vegas, 114 miles to the southeast, for the final transportation to the consumer.

The stockpile in Beatty lies adjacent to the old mill building that was erected in 1923. This was mainly a washing and sizing plant where the slime fraction of the ore was removed. Acceptable grades of metallurgical fluorspar were produced here, but this process tended to lose considerable fine grained fluorite in the slimes. Recoveries were usually in the 75 percent range. There never has been concern over the lack of water for a beneficiation plant because the Amargosa River runs beneath the surface, almost under the Crowell's mill building. A 300 gpm well close-by easily supplied the washing plant before it was shut down.

Figure 4: Mine Surface Installations
in the late 1940's. No later attempts at large scale beneficiation have been made for the reason that the U. S. Bureau of Mines investigators were unable to effectively or efficiently upgrade low value dump material or the stockpile ore in their 1945 study. Since metallurgical grade was available through selective mining techniques, there seemed little reason for the Crowell's to go to the expense of trying to beneficiate ore that is very difficult to handle.

Previous Studies

The Daisy mine, being a fairly large and unique fluorite occurrence, has interested both the U. S. Bureau of Mines and the U. S. Geological Survey. The U.S.G.S. report of 1964 has already been cited, and this writer now wishes to summarize the work done by the Bureau of Mines, for it provided the basis from which this study grew. In 1945 the Bureau conducted an extensive investigation of the Crowell property as a part of the wartime Government program of exploration of mineral deposits to ascertain their most effective and economical usage. This program included fairly comprehensive drilling, underground sampling, and mapping of the Daisy mine and its surroundings. Part of the work was also directed towards the beneficiation of the low grade and mine run ores. Metallurgical grades were already being produced, hence, a great deal of effort was spent in the attempt to make acid grade fluorspar. The results of these tests were then published in the Report of Investigations, 3954. A review of the difficulties encountered at this time is necessary because the same problems were faced in the present study.

At the time of the Bureau investigation, requirements for acid grade fluorspar stipulated that the product had to have at least 97.5 percent
fluorite with less than 1 percent silica. Presently, the specifications are less stringent, requiring a minimum of 97.0 percent fluorite and no more than 1 percent silica. The Bureau soon discovered that the fine replacement textures of many underground samples would render gravity methods of concentration completely impracticable and would greatly hinder flotation of the ores as well. The summary of a report, given by the Salt Lake City Laboratory, yields some quantitative feeling for the difficulties encountered during the flotation of a fairly typical low grade sample.

"The head analysis of this metallurgical sample was:

<table>
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<tr>
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<th>CaF₂</th>
<th>CaCO₃</th>
<th>MgCO₃</th>
<th>SiO₂</th>
<th>Fe</th>
<th>Al₂O₃</th>
<th>S</th>
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<td>%</td>
<td>51.45</td>
<td>13.9</td>
<td>12.13</td>
<td>6.5</td>
<td>0.85</td>
<td>4.5</td>
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"By log washing and screening on 6 mesh, 47.7 percent of the lime and 8.9 percent of the CaF₂ was rejected in the plus 6 mesh fraction. The undersize was ground through 200 mesh, the fluorite was floated with oleic acid and quebracho, and the concentrate was cleaned six times. Seventy-six percent recovery of the fluorite was obtained in the rougher concentrate assaying 83.3 percent CaF₂, 5.64 percent CaCO₃, and 0.9 percent SiO₂. Cleaner concentrates assaying 86.0 and 93.5 percent CaF₂ were obtained with recoveries of 72.5 and 61.3 percent, respectively, of the fluorite. Concentrates met chemical specifications for metallurgical grade spar but would require briquetting or sintering to meet size specifications." (Geehan, p. 5.)

Work was also done on higher grades of fluorspar ore, some averaging up to 77 percent fluorite. While in these cases 97 to 98 percent material was produced, the recoveries rarely exceeded 60 percent. A final summary of the flotation studies on two samples assaying 72.2 and 77.7 percent states that:

"Neither of the Crowell samples were found to be readily amenable to the production of acid grade material under strict plus 97.5 percent CaF₂ specifications, and only low recoveries at this grade can be expected." (Geehan, p. 7.)
Evidently, if such ores are to be beneficiated by flotation, they will require fine grinding which in turn necessitates a very close control of the flotation process itself.

**Sampling**

Samples used in this investigation were collected during two visits to the Crowell property in January and July of 1964. With the mine workings being so complicated and no adequate maps being available at that time, it was decided that the sampling should be used to point out the effect of mineralogical differences in the orebody rather than to attempt to represent an average mine grade. A relatively few number of samples could give some idea of the mineralogical constitution at the various levels of the mine while at the same time, the differences in their behavior during flotation could be more easily related to their composition. Keeping this in mind, a total of 13 low grade samples were cut of material that obviously varied in its gangue constituents as well as the fluorite grade. Eight of the highest grade samples were selected from these for flotation study. Grades of below 30 percent fluorite were rejected because of the difficulty of material handling in the batch flotation testing itself and because one dolomitic sample yielded a tailing that, at best, contained 25 percent fluorite. Plant practice would reject this material, therefore, the 25 percent seemed to be a logical minimum head grade.

The ore taken for testing was cut from drifts, raises, and stope walls that were accessible from several of the more extensive mine levels. One large sample was taken from the mine dump. The suite shows most of the mineralogical combinations that could be taken for a mill feed. The
gangue mineralogical differences have indeed proven to be of prime consideration during flotation, and any milling operation dealing with these ores should be aware of them.

**Analytical Procedures**

The analytical procedure used to determine the amounts of carbonates, insoluble matter, and calcium fluoride is an adaptation of the Aluminum Chloride Method published in 1944 by John C. Detweiler in Denver Equipment Company's publication, Deco Trefoil. A complete step by step procedure is included in the Appendix. This method first determines total carbonate content by weight loss after treatment with dilute acetic acid. The residue from this step is boiled with aluminum chloride which dissolves the CaF$_2$. Aluminum chloride insoluble residue may be taken as an approximation of the silica content, or the true SiO$_2$ amount may be determined by the loss of weight after a hydrofluoric acid treatment. The approximation of the silica percentage was used in most cases for the analysis of heads and flotation products. But in several cases the insoluble matter was carried through the final acid treatment step, and the actual amount of silica was found to constitute slightly more than half of the aluminum chloride residue weight. Most of the flotation work was carried out to the extent that this residue was held below 1 percent, thus introducing an analytical safety factor.

The initial phases of the investigation showed that merely an analytical breakdown of the flotation results was not sufficient. A successful flotation practice demands an understanding of how the individual ore and gangue minerals are affected during the process. Towards this end, several techniques of mineral identification were employed.
Optical examination of the concentrates and tailings played a large part. A fairly accurate estimate of the recovery success was made by panning the tailing from the rougher flotation in a watch glass, concentrating the heavier and larger particles, and then examining the material under the binocular rock microscope. Not only could the recoveries be estimated before an extensive analytical treatment, but the identification of the minor minerals could be enhanced by working with a gangue concentrate after the fluorite removal.

A more sophisticated optical treatment was needed for concentrate study. To achieve a better idea of the extent of gangue inclusions due to the replacement origins of the ore, a technique using small amounts of the concentrate mounted in refractive oil on a thin section slide was employed. The high power of magnification available in the stage microscope coupled with the use of transmitted polarized light enabled one to easily identify the gangue minerals collectively and to determine their importance as included impurities. When a fluorite grain mount is viewed with transmitted polarized light, the CaF$_2$ appears as dark fragments while the gangue material readily shows up as lighter areas. Where included gangue material prevails, the impurities show as finely divided light patches within the darker fluorite grains. An example of this condition may be seen in the following microphotograph of a minus 200 mesh fraction of a cleaner concentrate.

Although the optical methods were very helpful during the replacement texture studies, it did not readily show what gangue minerals were present in the concentrates. Use of the X-ray Diffractometer filled this requirement. This machine directs a beam of X-rays to a flat surfaced mineral sample where the phenomenon of X-ray diffraction by the mineral crystal lattices occurs as a result of a coherent scattering
of the impinging photons by the electrons of the target mineral. The X-ray photons are simply bounced from the electrons of an atom with no change in their wavelength. Each crystalline material will produce a distinctive pattern as this scattering occurs when certain geometrical conditions are satisfied. Bragg’s Law is the statement of these conditions. When the diffracted X-radiation is recorded and automatically plotted, the result is a series of peaks characteristic to the mineral being irradiated. When an aggregate of minerals is examined, several sets of peaks are recorded, and a very crude estimate of the relative amounts of these minerals may be had by comparing their respective intensities shown by the peak heights. A following discussion of sample mineralogy includes two such Diffractometer scans. After a thorough under-
standing of the flotation effects that the gangue minerals exert, samples can be examined on the X-ray Diffractometer, and in a short time, the relative amounts of the several constituents can be estimated. This saves considerable time in optical study as well as time used for preliminary flotation testing. In the case of these ores, the identification of the very finely divided sericite allowed the proper steps to be taken so that this silicate mineral might be rejected to a tailing rather than be floated into a concentrate product.

Sample Description

The following list gives the analytical composition of the eight samples studied during this investigation.

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<tr>
<th>Sample</th>
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<th>% Insol.</th>
<th>% CaF₂</th>
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<td>35.92</td>
<td>9.84</td>
<td>54.24</td>
<td>Surface mine dump</td>
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In order to achieve an understanding of the flotation characteristics of these samples, the analytical data should be supplemented with estimates of the mineral assemblage constituting the carbonate and insoluble residue assays. The optical and X-ray methods used showed that the dominately occurring minerals of these ores were: fluorite, dolomite, calcite, quartz, and sericite. Flotation results have pointed to the importance of the carbonate composition in this process. For this reason several samples should be noted. Ore samples 2, 4, and 8 have dolomite as their predominate carbonate gangue mineral, while No. 5 was the
only material that contained calcite exclusively. Sample 6 contained a
great deal of calcite, but dolomite was present. The remaining test ores
possessed fairly even mixtures of the two carbonates. Insoluble materi-
ial is always dominated by quartz, and in most cases to a lesser degree
by sericite. Sericite usually is seen as small iron stained clusters of
fine grained mica. The Diffractometer scan on the following page shows
how this assemblage appears in the heads of sample 2. Sometimes large
flakes of biotite can be found in rougher tailings as well as the larger,
clear muscovite. The small particles of mica are easily floated into a
concentrate and can constitute the major amount of insoluble material
in these products. A scan of the acid grade concentrates of test DM-10,
shown on page 21, gives a typical example of how the gangue material ap-
pears and is identified. A listing of the Diffractometer settings and
the ASTM cards used for the mineral investigative work appears in the
Appendix. Once the peak locations are known, it becomes an easy matter
to use these charts to explain the quantitative results in a more de-
tailed manner.

The assemblage of minor minerals found in the Crowell ores include
magnetite and a variety of clay-like iron oxides. Cinnabar, while pre-
sent in small amounts, can almost always be found in rougher flotation
tailings. Barite, pyrite, and some clay minerals, perhaps montmoril-
lo-nite, can sometime be found after more extensive examination.

Flotation characteristics are greatly affected by the carbonate
minerals present. Dolomitic ores tend to contain a much greater amount
of intimately associated fluorite than the mixed or purely calcitic ma-
terial. This is apparently the result of the replacement textures that
predominate in the magnesium rich wall rocks. Where calcite is the dom-
inate carbonate gangue mineral, much difficulty is experienced in
DIFFRACTOMETER SCAN OF
SAMPLE No. 2.
(\(\frac{1}{2}\) of real size)

Figure 6.
DIFFRACTOMETER SCAN OF TEST DM-10 ACID
GRADE CONCENTRATE
(full scale)
efficiently depressing it. Ores of this type tend to be more easily ground than their dolomitic counterparts, and sliming becomes a problem. When a mixture of carbonates is encountered, this investigation has found that the severeness of these problems is greatly lessened. On the whole, ores containing both minerals in notable amounts, were found to be amenable to flotation. The desired grades could be obtained with respectable recoveries.

Quartz and sericite have proven to be less troublesome than the carbonates. Usually these minerals could be separated from the fluorite in the early cleaning stages of the test circuit. Some problems occurred due to the floatability of the micaceous material; these difficulties can be overcome through the development of the flowsheet and by the proper reagent additions. It is felt that the insoluble matter reaching the acid grade concentrates does so not as a result of being included within the fluorite grains, like the dolomite, but rather as a function of the effectiveness of the cleaning steps.

**Important Factors Affecting the Flotation Results**

**Sizing Requirements:**

The fine grinding that this ore requires has previously been mentioned by the U.S. Bureau of Mines investigators. At the time of the 1945 Government study, the ore samples from this property being treated by flotation were ground to 100 percent minus 200 mesh. In this work a grind of 80 to 90 percent minus 200 mesh has been considered adequate. This conclusion was reached after several high grade concentrates were classified in the Haultain Infrasizer and studied optically. The
Infrasizer is a pneumatic classifier manufactured in Toronto, Canada. This device utilizes a series of seven cones, each with an air stream passing through it, to size finely ground material effectively into the 10 micron range. Some estimate may be had of the liberation requirements, when sized concentrates are examined with the polarizing microscope. This investigator felt that a 200 mesh grind would be sufficient for these ores if the unliberated fluorite grains and inclusions could be diverted to a steel grade product. This would become a necessity in plant practice since the material could never be completely liberated and would certainly contaminate the acid grade products. The curve on the following page represents an average plot of the cumulative weight percent of a flotation concentrate retained at the corresponding particle sizes. When dealing with the dolomitic ores, a fine grind is essential not only to effect the desired grades but also to allow reasonable recoveries. Dolomite is readily depressed in a rougher float, and any associated fluorite is effectively kept from the froth as well. Experience has shown that finer grinding produces excessive amounts of slime while little additional liberation is attained. The microphotographs below, being of fluorite grains magnified 800 times, show the minuteness of the inclusions. The photograph was taken using regular transmitted light while the lower print shows the sub-micron inclusions in the same field viewed with polarized light. Some samples, such as No. 8, had dolomite inclusions to the extent that they could be easily seen under the binocular rock scope.

Sliming Effects:

Slimes produced during the fine grinding of the Daisy mine ores have a great effect upon froth characteristics. An excessive amount of fine
A Typical Weight-Particle Size Plot of a Flotation Concentrate
Figure 9: Fluorite Concentrate Grains Viewed With a Nonpolarized Light Source

Figure 10: Fluorite Concentrate Grains Viewed With a Polarized Light Source
material in the rougher flotation causes the froth to be watery and very uncontrollable. Reagent dosage must be increased and a much less effective collection of the fluorite is experienced. This study has utilized stage addition of the collectors in the rougher, and in all cases the slime fraction of the pulp has been the first to be carried by the froth. Once in the concentrate, this material is very difficult to separate from the fluorite in later cleaning steps. Several attempts were made to remove these micron sized particles before flotation, but recoveries have suffered as a result. The nature of this problem is explained by a polarized microphotograph of a slime fraction taken from a test DM-10 second cleaner concentrate. This print, appearing on page 27, shows the fluorite grains laced with gangue inclusions. If this material is rejected to a tailing product in the rougher states, considerable fluorite loss is experienced. However, this fraction can be removed in a cleaning step and may be placed in the steel grade product where specifications are more tolerant. In this manner recoveries can be maintained.

Flowsheet Development:

Much of the test work previously performed on fluorite ores was intended to produce either a steel grade product or one meeting acid grade requirements. If one attempted to float the Daisy mine ores on a plant scale to an acid grade purity, he would soon find that the circulating loads in the cleaning cells would become very large. Much material containing fluorite with locked gangue would continue to build up in this part of the circuit, until it would be promoted to the concentrate or would be rejected in a rougher tailing. Such a condition would not be conducive to either a high grade of concentrate or an acceptable recovery.
Figure 11: Microphotograph of a Concentrate Slime Fraction

Instead, it seems reasonable to use a circuit where this semi-locked fluorite can be removed as a cleaner tailing to the steel grade product, while the more completely liberated fluorite is repeatedly cleaned until acid grade requirements are met. This study has found that it is advisable to return the first cleaner tailing to the rougher circuit because of its low fluorite content and high insoluble residue assay. The remaining cleaner tailings may then be removed directly to the lower grade industrial product. This system has shown promise in its ability to produce both a chemical grade concentrate and also a product that meets the 72 percent fluorite requirement of the industrial users. When the first cleaner tailings are re-circulated to the rougher circuit, the insoluble matter is always below the 5 percent silica minimums specified by the
steel companies. However, the carbonate gangue minerals sometimes exceed their allowable limits. If the gangue slime fraction that is so difficult to remove by repeated flotation is separated from the larger fluorite concentrates in a hydroclassifier, the fines may be included in these industrial grades also. The underflow from such a classifier may then be re-cleaned by repeated flotation until chemical specifications are met. In the laboratory such a flowsheet has been simulated by placing the second cleaner concentrates in a large bell jar where the remaining froth is broken by a fine water spray. The pulp is diluted and agitated for a short time and then allowed to settle. Larger grains of fluorite and sometimes gangue minerals rapidly sink to the bottom leaving the slimes suspended, where they are readily removed by decantation to the steel grade concentrate. The flowsheet as it was used in this investigation appears below. It is to be understood that the concentrates will be involved in additional operations after this process such as filtering.

Figure 12: The Laboratory Flowsheet
drying and then pelletizing treatments. On the laboratory batch scale, it is felt that this flowsheet has contributed considerably towards the successful flotation of several of the Crowell ore samples.

Collectors, Frothing Agents and Froth Control:

Fatty acids have found the most use as fluorite collectors in this country. Most of the time oleic acid is used extensively as is seen in the U.S.B.M. summary previously mentioned. However, several companies are now marketing promoting agents for non-metallic flotation that are combinations of oleic, linoleic acids, and small amounts of rosin acids. American Cyanamid Company includes this type of promoter in its 700 series, while the Arizona Chemical Company manufactures the same reagent under the FA designation of tall oil fatty acids. A number of these collectors were studied in the early stages of this examination. Pure oleic acid has a tendency to be very unselective in its collective properties. Froth modification with high order aliphatic alcohols helps to a degree, but when a flotation pulp is rich in calcite, much of this material is very efficiently collected and along with the fluorite remains in the cleaner concentrates. The oleic acid promoter evidently contacts the Ca anion during agitation, and the similarity between CaF$_2$ and the CaCO$_3$ of calcite probably accounts for this difficulty. When a mixture of Mg and Ca occurs in the carbonate, as it does in dolomite, the promoter avidity for the Ca ion in the gangue mineral is almost entirely absent. But the problem of calcite promotion is usually replaced with the inclusion difficulties inherent in the dolomitic ores.

Use of a refined tall oil in place of oleic acid will result in a much more selective flotation. However, recoveries have been found to be
lower when oleic acid is not used. Aerofloat 723 special was employed during the tests conducted on sample 1. This reagent gave good froth control after a technique of stage addition of the promoter to the rougher float was developed. Use of Aerofroth 73 provided the required selectivity. A froth produced by this combination was well controlled, but had to be maintained through additional dosages of collector in the cleaning stages. When a request was sent to American Cyanamid Company for additional 723 special, the inquiry was forwarded to the Arizona Chemical Company plant. This organization now manufactures this reagent as Acintol FA-1 special tall oil fatty acid. Additional test work with the new collector gave results which were inferior to those obtained through the use of its older Cyanamid counterpart. Perhaps the initial success of the original sample of 723 special was due to its mellowing upon a laboratory shelf for several years. Nevertheless, this posed a problem until it was discovered that a mixture of red oleic acid and an FA series tall oil could be used to give the desired results. In these cases the mixture was added to the rougher float in 3 or 4 doses. The mixture of collectors can be changed easily to give the most controllable and long lasting froth. Usually, the amount of Aerofroth 73 could be lessened. Some additional dosages of collector is often needed in the final flotation cleanings to maintain the froth, in this case specifically Aerofroth 73. The fluorite at this point is quite pure and contains very little slime. As has been mentioned before, slimes tend to give a lighter, more brittle froth and when they are absent, the fluorite froth becomes very tough and heavy and is consequently quite unselective. Small additions of 73 at this stage overcomes the heaviness, and the desirable texture may be maintained. Two photographs illustrate the differences between a heavy froth and one of the desired appearance. The
Figure 13: A Rougher Froth Texture After a Single Addition of Oleic Acid

Figure 14: A Rougher Froth Texture After Stage Additions of an Oleic Acid and Tall Oil Mixture
print in Figure 13 shows the heavy froth that results when 1.6 pounds per ton of oleic acid is added at once, while the print in Figure 14 illustrates how the froth texture appears if the same amount of an oleic acid and tall oil mixture is added in three stages. If the stage addition is not used in the rougher circuit, the tough froth may persist into the cleaning circuit where it hinders the concentrate cleaning. The print below shows how the froth will appear in the second cleaner if the stage addition is practiced.

Figure 15: A Second Cleaner Froth Texture Resulting from the Use of the Oleic Acid and Tall Oil Additions

Depressants and Their Effectiveness:

Gradual addition of depressants is also thought to be an important factor in effecting the removal of undesired gangue minerals. A number
of varied reagents for this use have been investigated. Cyanamid's S-3223 was found to be a good depressant for carbonate minerals, but it apparently is an anionic active reagent because fluorite is sometimes de-activated as well as the gangue. Palcotan is a very strong carbonate depressant. Very small amounts of this Redwood bark extract could efficiently hold down dolomite and calcite. In the later cleaning stages, especially when the flotation is carried out in Beatty water, the froth is very difficult to maintain. Considerable amounts of additional collector and frother are required to re-establish an effective froth. When this is done, there is a possibility that a good deal of calcite is also promoted by the increased collector concentrations. Very careful control of the Palcotan dosage must be practiced so that this condition does not occur. Orzan S, also a Redwood extract, is much less severe in its depressing action. This material has little froth depressing characteristics and seems to do an adequate job in the rougher and first cleaner circuit. However, unless the ore has little calcite content, the final cleaning stages are ineffective in their removal of the final few percent of carbonates. When a high silicate bearing ore is dealt with, the Orzan is effective enough. The Appendix contains several examples where Orzan S proved to be sufficient for the depression of the carbonate compliment of sample 3. The third bark extract, Quebracho, seemed to be the most effective carbonate depressant studied. This product affected the froth quality to a lesser extent than did Palcotan and possessed the stronger reagent's depressing ability to a large degree. This seemed to be the best reagent for batch test control and it has been used extensively in this investigation. Yet, Palcotan may prove effective on a mill scale. The additional
depressive strength available in this reagent and its lower cost make it attractive from an economic point of view.

Some experimental work was conducted using hydrofluoric acid as a silica depressant, but Metso Granular sodium meta silicate has been found to be the most effective. This material is certainly easier to handle than HF and also serves as a dispersing agent during flotation. Its use has been justified by the results obtained with some of the highly silicious ores that were investigated. The use of sodium silicate as a depressant seems effective only for the removal of quartz. Sericite is sometimes readily floated despite the concentration of this reagent. After the problem of high silica estimates had been traced to the sericite, additions of starch to the pulp was initiated. Aero Depressant 610, especially recommended by American Cyanamid Company for the removal of micaceous material, was also studied, but in the case of the Crowell ores the starch additions seemed to be just as effective as the more sophisticated Reagent 610. An extensive use of the Cyanamid reagent would probably be ruled out in this case due to its higher cost. A sericite depressant is required when dealing with most ores from this property; effective froth control and a flowsheet designed to remove fine material from the final flotation concentrate are only occasionally sufficient in themselves. The first ore sample was successfully treated without the use of such a reagent. However, the oleic acid and tall oil mixture is usually severe in its sericite promotion. The success of the tests performed upon sample 1 are most probably the result of the use of the more selective original Aerofloat 723 special. Cost savings possible through the use of the crude red oleic with the cheaper tall oils can easily provide the few cents per pound required for the starch
additions. During the flotation of the sericite rich ores, its use is important if the insoluble residue specifications are to be met.

In summary, the most successful suite of reagents used during this investigation are: a combination of oleic acid and refined tall oil for promotion and froth control, Quebracho for the depression of carbonate gangue minerals, sodium meta silicate for slime dispersion and quartz depression, and commercial quality starch for the removal of the sericitic micas. Stage additions of these reagents are seemingly most effective, giving the greatest control over the froth texture, grade, and recovery.

Flotation Cell, Temperature and Water Effects:

Three makes of flotation cells were available for this work. The Booth cell provided a very violent agitation during flotation which made froth control almost impossible. Agitation is very important for the maximum recovery of the fluorite, but the most successful work was carried out in the Fagergren Laboratory batch cell. This machine provides sufficient agitation during the rougher flotation to achieve respectable recoveries, while at the same time it serves as a very controllable cleaning cell. Perhaps on a plant scale the Denver flotation cell would be a more effective cleaner. This device has a baffled quiet zone specifically designed to allow gangue minerals to drop out of the froth. In a rougher flotation of these ores, the quiet zone lowers the recoveries. Evidently the dolomitic inclusions greatly hinder fluorite promotion because of the inactivity of the carbonate. This material rises into the froth only with considerable difficulty, and the Denver cell does not manage to keep dolomitic fluorite well enough in the froth.
so that it may be removed from the cell as a concentrate. Despite this, the machine performed adequately as a cleaner. The Fagergren was preferred in the batch test cleaning because more material can be promoted into the concentrate product at each successive cleaning stage. When ores containing only 40 percent fluorite were treated, pulp density in the last cleaner became a concern. The Denver machine dropped out more fluorite in each cleaning and several times the solids concentration fell so low that it prevented the formation of a froth in the third cleaner. To overcome this problem, higher densities were used in the rougher flotation and the Fagergren cell was used to produce more concentrate in the first cleaning steps. Use of the higher rougher densities, perhaps as high as 30 percent solids, and the promotion of more fluorite as the concentrate is cleaned yields an interesting result in that the first cleaner tailings generally contain less CaF$_2$ than do the heads. Usually, cleaner tailings can be expected to have more value than the original feed. This material, as witnessed by several test analyses appearing in the Appendix, carries a large amount of insoluble residue. For this reason, it is best returned to the rougher circuit as a circulating load. With this material removed from the cleaning stages, the insoluble requirements of the concentrates are more easily met.

A number of commercial operations engaged in the upgrading of fluorite ores have used heated pulps to more efficiently promote the fluorite values. This was tried in the present investigation where the pulps were heated to nearly 100°C prior to the rougher flotation. After several such tests it was decided that the hot pulp did indeed promote fluorite, but it also overcame the effectiveness of some of the carbonate depressants as well. More effective and selective flotation seemed possible through the use of cold pulps, with the promotion of the fluorite depending upon oleic acid or tall oil additions.
Some previous discussion of the effects of the Amargosa River water has already been made with respect to its effect upon froth character. The water for this examination was taken from an open ditch approximately one mile south of the mouth of Fluorspar Canyon. Part of the underground river comes to the surface at this point and flows southward where some is used for irrigation. The water has a fluorine content of 5 to 7 parts per million. This amount of fluorine renders the water unusable for human consumption, and all of the drinking water for Beatty comes from uncontaminated springs and wells. When a comparison is made between the flotation effects of Reno city water and that of the Amargosa River, little difference is noticed. The water, being hard in nature, seems to render the froth a little less stable, but identical recoveries and grades are obtainable. Results of several such comparisons may even indicate that depressant effectiveness is increased but with added expense for small additional amounts of collector that must be used to maintain the final cleaner froth. The proximity of the river, being directly beneath the old mill building, is indeed a unique situation in this state, and its favorable use in flotation seems to indicate that a successful flotation operation could be had.

Conclusions

It is hoped that the results of this investigation have shown that the Crowell ores may be successfully upgraded through flotation. An attempt has been made to use crude reagents that may be obtained cheaply. The quality of products made in a dual concentrate flowsheet will depend upon the ability of the operators to control conditions within the flotation process. Froth control, effective depressant usage, and slime removal
all constitute important parts of this process. In order that these controls may be maximized, it is apparent that the mineralogical fluctuation of the mill feed must be held at a minimum. The completely different flotation characteristics of calcite and dolomite can cause a great deal of difficulty. Because these gangue minerals are to be found in relatively pure concentrations within the orebody itself, the greatest care must be practiced during mining. The blending of the mine run ores will probably be the only method of stabilizing a mill feed, and such a program is admittedly very difficult to carry out in a shallow epithermal deposit. However, if the high grade ores, now shipped directly to the consumer, are used to bring the lower grades up to an overall CaF$_2$ content of 50 or 60 percent, the flotation problem would be greatly simplified.

Also, particular attention will have to be paid to the mineralogy of these ores, as the constituents will decide the final outcome of the beneficiation attempt.

Some idea of the reagent cost as well as some pertinent data referring to market specifications and possible concentrate payments may be found in the Appendix. The extremely competitive nature of the fluorite market on the West Coast places severe demands upon the process cost. It is felt that the reagent use and dosage amounts correspond to these requirements. Promoter use is generally below 1.7 pounds per ton of ore fed to the circuit and this seems to be in agreement with present mill usages as well as Government investigations involving fluorspar ores from other parts of the country (Thom, p. 343). Depressant usage is seen to be slightly greater than many practices, but again, the use of crude reagents may help to compensate for this shortcoming. The abandonment of pH control through the use of sodium carbonate will also be of economic interest.
Operators of Kaiser Steel Company's fluorspar flotation operation in Fallon, Nevada found it necessary to use large amounts of soda ash to establish the desired alkalinity. The natural water from the Beatty area is evidently sufficient in itself to allow flotation, thus, saving as much as ten cents per ton of mill feed. Water conditions for a flotation operation are extremely favorable. Future industrial usage of the Amargosa River most likely will not effect or be hindered by the human population.

Competition from Mexican sources will pose a serious threat to domestic fluorspar producers in the future. Presently, it is rumored that these interests are beginning to market acid grade concentrates in this country as well as sized steel grade material. Pelletizing plants are being constructed, and this may place the foreign deposits in such a position that they may be able to under sell the small domestic miner. However, the nearness of the Nevada deposits to the California market may make deliveries from this area more reliable and therefore more favorably received by the consumers. Regardless of the present conditions, the future will undoubtedly see the use of the lower grade ores of the Daisy mine. Flotation will most likely be the answer to the beneficiation of this material, and this writer hopes that this investigation will prove useful when that time comes.
ANALYSIS OF PHOSPHATE

Determination of Carbonate, Insoluble, and Calcium Phosphate

Alumina Chloride Method. (Adapted, with modifications, from a method published by John G. Davison in Data Trafics, August, 1964.)

Weigh 0.5000 g. of ore, pulverized to pass 100-mesh screen, into 150 ml. beaker. Add 50 ml. of 10-percent acetic acid and heat slowly, boiling for 15 minutes. Filter through 11 cm. Whatman textured "ash-
lace" paper to aid the retention of final divided material. Place beaker thoroughly and each five times with hot water. Place paper containing washed residue in a tared No. 0 platinum crucible, dry thoroughly on a cabinet or over a burner, using an electrically actuated transfer to a hot plate between (1000°C.) and turn off all combustion material, leaving some of melting point upon to induce the access to the ample of sufficient air to produce rapid and complete combustion of the paper and pulp. Remove to a desiccator to cool. Weigh and record net weight as "insoluble insoluble residue".

Brush residue back into the 150 ml. beaker. Add 15 g. ethylene diamine triacetic acid (ADA), 20 ml. and boil gently for 20 minutes. Filter through a 11 cm. Whatman paper to a crucible of the same grade of paper in the same size. Rinse beaker thoroughly with 20 ml. ethanol.- Rinse paper containing washed residue to the crucible used to boiling with weighing the "metallic chloride insoluble" and identity as before. Return to a desiccator and cool. weights and record the weights using occasional care as "alumina chloride insoluble residue".

Calculate percent yield by subtracting the weight of the "alumina chloride insoluble residue" from the weight of the "insoluble insoluble residue" and multiplying by 200 ( pretext 2.000 g. sample is treated). When concentrations are adjusted, 0.456 percent should be added as the difference from the carbonate base. This is no cavitated by the acetic acid solution.

Calculate percent yield by subtracting the weight of the "calcium and insoluble residue" from the 0.500 g. and multiplying by 200. 0.456 percent should be subtracted when concentrations are made with.

The weight of the "calcium chloride insoluble residue" gives a rough approximation of the "insoluble insoluble residue" weight as 500 to estimate percent insoluble. When 0.500 g. sample is used, 200-gram percent insoluble. 200 gm. (1) Insoluble mixed alumina chloride insoluble insoluble residue with a small burned platinum crucible and weighed. Add from 1 to 3 ml. (according to the size of the residue of high chloride acid (1:42) and a few drops of concentrated sulfuric acid (1:3). Evaporate to dryness and ignite in a hot muffin furnace (1000°C.). Cool in a desiccator and weigh. Take the weight to the nearest 0.001 g. reasonably of 200 to obtain percent (see
Analysis of Fluorspar

Determination of Carbonates, Insoluble, and Calcium Fluoride

Aluminum Chloride Method. (Adapted, with modifications, from a method published by John C. Detweiler in Deco Trefoil, November, 1944.)

Weigh 0.5000 g. of ore, pulverized to pass 100-mesh screen, into 150 ml. beaker. Add 15 ml. of 10-percent acetic acid and heat below boiling for 15 minutes. Filter through an 11 cm. medium textured "ashless" paper to aid the retention of finely divided material. Police beaker thoroughly and wash five times with hot water. Place paper containing the washed residue in a tared No. 0 porcelain crucible, dry thoroughly on a hotplate or over a burner, using an asbestos pad. Transfer to a hot muffle furnace (1000 C.) and burn off all carbonaceous material, leaving door of muffle partly open to insure the access to the sample of sufficient air to promote rapid and complete combustion of the paper and pulp. Remove to a desiccator to cool. Weigh and record net weight as "acetic acid insoluble residue".

Brush residue back into the 150 ml. beaker. Add 10 g. aluminum chloride (AlCl₃·6H₂O) and 50 ml hot water. Place over moderate heat and boil gently for 30 minutes. Filter through an 11 cm. paper using paper pulp, on the same grade of paper as was used in the first filtration. Police beaker thoroughly and wash ten times with hot water. Transfer paper containing washed residue to the crucible used in burning and weighing the "acetic acid insoluble", dry, and ignite as before. Remove to a desiccator and cool. Weigh and record net weight (using original tare) as "aluminum chloride insoluble residue".

Calculate percent CaF₂ by subtracting the weight of the "aluminum chloride insoluble residue" from the weight of the "acetic acid insoluble residue" and multiplying by 200 (when 0.5000 g. sample is taken). When concentrates are assayed, 0.45 percent should be added to the CaF₂ and subtracted from the carbonates since this amount is dissolved by the acetic acid solution.

Calculate percent total carbonates by subtracting the weight of the "acetic acid insoluble residue" from the 0.5000 g. and multiplying by 200. 0.45 percent should be subtracted when concentrates are dealt with.

The weight of the "aluminum chloride insoluble residue" gives a rough approximation of the silica; multiply this weight by 200 to obtain percent insoluble (when 0.5000 g. sample is used). To obtain percent SiO₂ (if desired) brush "aluminum chloride insoluble residue into a small tared platinum crucible and weigh. Add from 1 to 5 ml. (according to the size of the residue) of hydrofluoric acid (HF, 48%) and a few drops of concentrated sulfuric acid (H₂SO₄ sp. gr. 1.84). Evaporate to dryness and ignite in a hot muffle furnace (1000 C.). Cool in a desiccator and weigh. Loss in weight is SiO₂; multiply by 200 to obtain percent (for
0.5000 g. sample is taken). In most instances, the weight of the residue from the volatilization of the SiO₂ may be taken as R₂O₃ (oxides of iron, aluminum, manganese, titanium, etc.); if it is desired to report percent R₂O₃, multiply the weight of this residue by 200. If R₂O₃ is not to be determined, the tare-weighing of the platinum crucible may be omitted.

### ASTM Card References

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### X-Ray Machine Settings

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### Product Specifications and Payments

Taken from E&MJ Quotations

**Acid Grade Fluorspar** - A minimum of 97.0 percent CaF₂ and a maximum of 1 percent SiO₂. The remainder may be carbonates.

Payments as high as $49.00 per ton, f.o.b Illinois, carloads have been made. West Coast prices are probably considerably lower.

**Steel Grade Fluorspar** - A minimum of 72.5 percent CaF₂, and a maximum of 5 percent SiO₂. The remainder may be carbonates. 2.5 percent CaF₂ subtracted for every percent SiO₂.

Domestic payments are listed as $35 to $38, f.o.b. Illinois. Mexican
payments are placed at $26 to $28 at Border, all rail, duty paid. West Coast prices are again considerably lower.

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<td>Acintol FA-1 special</td>
<td>11</td>
</tr>
<tr>
<td>Acintol Heads 1112</td>
<td>5</td>
</tr>
<tr>
<td>Soda Ash</td>
<td>3</td>
</tr>
<tr>
<td>Oleic Red Oil</td>
<td>14</td>
</tr>
<tr>
<td>Palcotan (sulfonate)</td>
<td>9</td>
</tr>
<tr>
<td>Sodium silicate</td>
<td>3</td>
</tr>
<tr>
<td>Ligro or Indusoil tall oil</td>
<td>2</td>
</tr>
<tr>
<td>Starch</td>
<td>4</td>
</tr>
</tbody>
</table>

A typical reagent cost per ton of feed for a test of this series would amount to approximately 37¢.
Test No.  DM-8

Sample No.  
Sample weight  1
Grind (lab. mill)  600 gms.
Sizing  10 min. @ 50% solids.
Water used  83.9% minus 200 mesh.

Test Conditions: (flotation preceded by a deslime)

<table>
<thead>
<tr>
<th></th>
<th>temp.</th>
<th>pH</th>
<th>cond. time</th>
<th>float time</th>
<th>machine</th>
<th>aeration</th>
</tr>
</thead>
<tbody>
<tr>
<td>Rougher</td>
<td>27 C</td>
<td>7</td>
<td>8 min.</td>
<td>10 min.</td>
<td>Fag. sub.</td>
<td></td>
</tr>
<tr>
<td>Cleaner</td>
<td>&quot;</td>
<td>6.8</td>
<td>1</td>
<td>4</td>
<td>&quot;</td>
<td>&quot;</td>
</tr>
<tr>
<td>Deslime</td>
<td>&quot;</td>
<td>6.8</td>
<td>1</td>
<td>-</td>
<td>Denver</td>
<td>-</td>
</tr>
</tbody>
</table>

Reagents (lb./ton):

<table>
<thead>
<tr>
<th></th>
<th>depressants</th>
<th>collector</th>
<th>frother</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Que.</td>
<td>Na silicate</td>
<td>Aerofloat</td>
</tr>
<tr>
<td>Rougher</td>
<td>0.40</td>
<td>0.50</td>
<td>0.80</td>
</tr>
<tr>
<td>Cleaner 1</td>
<td>0.20</td>
<td>0.40</td>
<td>0.30</td>
</tr>
<tr>
<td>Cleaner 2</td>
<td>0.20</td>
<td>0.20</td>
<td>-</td>
</tr>
<tr>
<td>Cleaner 3</td>
<td>-</td>
<td>0.10</td>
<td>-</td>
</tr>
<tr>
<td>Cleaner 4</td>
<td>-</td>
<td>-</td>
<td>-</td>
</tr>
</tbody>
</table>

Analysis:

<table>
<thead>
<tr>
<th></th>
<th>wt.</th>
<th>% carbonates</th>
<th>% insol.</th>
<th>% Si</th>
<th>% fluorite</th>
</tr>
</thead>
<tbody>
<tr>
<td>Rougher tails</td>
<td>233 gms.</td>
<td>43.02</td>
<td>42.50</td>
<td></td>
<td>14.48</td>
</tr>
<tr>
<td>Slime tails</td>
<td>56</td>
<td>37.20</td>
<td>44.42</td>
<td></td>
<td>18.38</td>
</tr>
<tr>
<td>Cleaner tails</td>
<td>1</td>
<td>13.35</td>
<td>6.80</td>
<td></td>
<td>79.85</td>
</tr>
<tr>
<td>Cleaner conc.</td>
<td>123</td>
<td>2.00</td>
<td>0.72</td>
<td></td>
<td>97.28</td>
</tr>
<tr>
<td>Slime</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

Recovery:

<p>| | | |</p>
<table>
<thead>
<tr>
<th></th>
<th></th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>Acid grade product</td>
<td>32.2%</td>
<td></td>
</tr>
<tr>
<td>Steel grade product</td>
<td></td>
<td>88.1%</td>
</tr>
<tr>
<td>Total recovery</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>
Sample No. | DM-10
--- | ---
Sample weight | 600 gms.
Grind (lab. mill) | 12 min. @ 50% solids
Sizing | 84% minus 200 mesh
Water used | Reno

**Test Conditions:**

<table>
<thead>
<tr>
<th>temp</th>
<th>pH</th>
<th>cond. time</th>
<th>float time</th>
<th>machine</th>
<th>aeration</th>
</tr>
</thead>
<tbody>
<tr>
<td>Rougher</td>
<td>26 C</td>
<td>7</td>
<td>10 min.</td>
<td>12 min.</td>
<td>Fag.</td>
</tr>
<tr>
<td>Cleaner</td>
<td>&quot;</td>
<td>7</td>
<td>2</td>
<td>5</td>
<td>&quot;</td>
</tr>
<tr>
<td>Deslime</td>
<td>&quot;</td>
<td>6.7</td>
<td>1</td>
<td>-</td>
<td>Bell jar</td>
</tr>
</tbody>
</table>

**Reagents (lb./ton):**

<table>
<thead>
<tr>
<th>depressants</th>
<th>collector</th>
<th>frother</th>
</tr>
</thead>
<tbody>
<tr>
<td>Que.</td>
<td>Na silicate</td>
<td>Aerofloat 723 sp.</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Rougher</th>
<th>Cleaner 1</th>
<th>Cleaner 2</th>
<th>Cleaner 3</th>
</tr>
</thead>
<tbody>
<tr>
<td>Rougher</td>
<td>0.50</td>
<td>0.25</td>
<td>-</td>
</tr>
<tr>
<td>Cleaner</td>
<td>0.50</td>
<td>0.25</td>
<td>0.12</td>
</tr>
</tbody>
</table>

**Analysis:**

<table>
<thead>
<tr>
<th>wt.</th>
<th>% carbonates</th>
<th>% insol.</th>
<th>% Si</th>
<th>% fluorite</th>
</tr>
</thead>
<tbody>
<tr>
<td>Rougher tails</td>
<td>179 gms.</td>
<td>29.40</td>
<td>58.00</td>
<td>12.60</td>
</tr>
<tr>
<td>Slime tails</td>
<td>30</td>
<td>48.06</td>
<td>33.26</td>
<td>18.68</td>
</tr>
<tr>
<td>Cleaner tails</td>
<td>230</td>
<td>26.46</td>
<td>6.04</td>
<td>67.50</td>
</tr>
<tr>
<td>Cleaner conc. Slime</td>
<td>119</td>
<td>1.78</td>
<td>0.80</td>
<td>97.42</td>
</tr>
<tr>
<td>Rougher conc. Slime</td>
<td>17</td>
<td>20.70</td>
<td>45.88</td>
<td>33.42</td>
</tr>
</tbody>
</table>

**Recovery:**

| Acid grade product | 31.2% |
| Steel grade product | 23.4% |
| Total recovery | 92.4% |
Test No. DM-11

Sample No. _______1________
Sample weight 600 gms.
Grind (lab. mill) 12 min. @ 50% solids
Sizing 86.5% minus 200 mesh
Water used Reno

Test Conditions: (cleaner #2 conc. deslimed)

<table>
<thead>
<tr>
<th></th>
<th>temp.</th>
<th>pH</th>
<th>cond. time</th>
<th>float time</th>
<th>machine</th>
<th>aeration</th>
</tr>
</thead>
<tbody>
<tr>
<td>Rougher</td>
<td>27 C</td>
<td>6.7</td>
<td>10 min.</td>
<td>10 mon.</td>
<td>Fag.</td>
<td>sub.</td>
</tr>
<tr>
<td>Cleaner</td>
<td>&quot;</td>
<td>6.6</td>
<td>3</td>
<td>4</td>
<td>&quot;</td>
<td>&quot;</td>
</tr>
<tr>
<td>Deslime</td>
<td>&quot;</td>
<td>6.5</td>
<td>1</td>
<td>-</td>
<td>Bell jar</td>
<td>-</td>
</tr>
</tbody>
</table>

Reagents (lb./ton):

<table>
<thead>
<tr>
<th></th>
<th>depressants</th>
<th>collector</th>
<th>frother</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Que.</td>
<td>Na silicate</td>
<td>Aerofloat</td>
</tr>
<tr>
<td>Rougher</td>
<td>0.50</td>
<td>0.50</td>
<td>0.50</td>
</tr>
<tr>
<td>Cleaner</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>1</td>
<td>-</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td>2</td>
<td>0.17</td>
<td>0.17</td>
<td>0.24</td>
</tr>
<tr>
<td>3</td>
<td>-</td>
<td>0.06</td>
<td>0.12</td>
</tr>
</tbody>
</table>

Analysis:

<table>
<thead>
<tr>
<th></th>
<th>wt.</th>
<th>% carbonates</th>
<th>% insol.</th>
<th>% Si</th>
<th>% fluorite</th>
</tr>
</thead>
<tbody>
<tr>
<td>Rougher tails</td>
<td>188 gms.</td>
<td>21.84</td>
<td>58.88</td>
<td></td>
<td>19.28</td>
</tr>
<tr>
<td>Cleaner tails 2</td>
<td>87</td>
<td>19.64</td>
<td>2.64</td>
<td>77.72</td>
<td></td>
</tr>
<tr>
<td>Cleaner conc. Slime</td>
<td>151</td>
<td>1.62</td>
<td>0.82</td>
<td>97.56</td>
<td></td>
</tr>
</tbody>
</table>

Recovery:

<table>
<thead>
<tr>
<th></th>
<th>Acid grade product</th>
<th>Steel grade product</th>
<th>Total recovery</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>25.2%</td>
<td></td>
<td>90.3%</td>
</tr>
</tbody>
</table>
## Test No. DM-14

<table>
<thead>
<tr>
<th>Sample No.</th>
<th>Test Conditions:</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Sample weight</td>
</tr>
<tr>
<td></td>
<td>Grind (lab. mill)</td>
</tr>
<tr>
<td></td>
<td>Sizing</td>
</tr>
<tr>
<td></td>
<td>Water used</td>
</tr>
<tr>
<td></td>
<td></td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Rougher</th>
<th>Test Conditions:</th>
<th>Cleaner</th>
<th>Deslime</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>temp. pH time float time machine aeration</td>
<td></td>
<td></td>
</tr>
<tr>
<td>25 C</td>
<td>6.7 10 min. 13 min. Fag. std.</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Deslime</th>
<th></th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Rougher</th>
<th>Cleaner</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
</tr>
</tbody>
</table>

### Reagents (lb./ton):

<table>
<thead>
<tr>
<th>Rougher</th>
<th>Cleaner</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
</tr>
</tbody>
</table>

### Analysis:

<table>
<thead>
<tr>
<th>Rougher tails</th>
<th>Cleaner tails</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>wt.</th>
<th>% carbonates</th>
<th>% insol.</th>
<th>% Si</th>
<th>% fluorite</th>
</tr>
</thead>
<tbody>
<tr>
<td>292 gms.</td>
<td>7.52</td>
<td>82.60</td>
<td>9.88</td>
<td></td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Rougher conc.</th>
<th>Cleaner conc.</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>wt.</th>
<th>% carbonates</th>
<th>% insol.</th>
<th>% Si</th>
<th>% fluorite</th>
</tr>
</thead>
<tbody>
<tr>
<td>208</td>
<td>2.78</td>
<td>21.88</td>
<td>75.34</td>
<td></td>
</tr>
</tbody>
</table>

### Recovery:

<table>
<thead>
<tr>
<th>Acid grade product</th>
<th>Steel grade product</th>
<th>Total recovery</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td>86.1%</td>
</tr>
</tbody>
</table>
Test No. DM-96

Sample No.
Sample weight
Grind (lab. mill)
Sizing
Water used

Test Conditions:

<table>
<thead>
<tr>
<th>temp.</th>
<th>pH</th>
<th>cond. time</th>
<th>float time</th>
<th>machine</th>
<th>aeration</th>
</tr>
</thead>
<tbody>
<tr>
<td>Rougher</td>
<td>26°C</td>
<td>7</td>
<td>12 min.</td>
<td>12 min.</td>
<td>Fag.</td>
</tr>
<tr>
<td>Cleaner</td>
<td>&quot;</td>
<td>7.2</td>
<td>1</td>
<td>3</td>
<td>&quot;</td>
</tr>
<tr>
<td>Deslime</td>
<td>&quot;</td>
<td>7.2</td>
<td>1</td>
<td>-</td>
<td>Bell jar</td>
</tr>
</tbody>
</table>

Reagents (lb./ton):

<table>
<thead>
<tr>
<th>Starch</th>
<th>depressants</th>
<th>Na silicate</th>
<th>collector</th>
<th>frother</th>
</tr>
</thead>
<tbody>
<tr>
<td>Rougher</td>
<td>0.5</td>
<td>0.2</td>
<td>0.5</td>
<td>Oleic Tall</td>
</tr>
<tr>
<td>Cleaner</td>
<td>0.5</td>
<td>0.1</td>
<td>0.5</td>
<td>-</td>
</tr>
<tr>
<td>1</td>
<td>0.5</td>
<td>0.2</td>
<td>0.3</td>
<td>-</td>
</tr>
<tr>
<td>2</td>
<td>0.2</td>
<td>0.1</td>
<td>0.2</td>
<td>-</td>
</tr>
<tr>
<td>3</td>
<td>0.1</td>
<td>0.2</td>
<td>0.2</td>
<td>-</td>
</tr>
</tbody>
</table>

Analysis:

<table>
<thead>
<tr>
<th>Wt. % carbonates</th>
<th>% insol.</th>
<th>% Si</th>
<th>% fluorite</th>
</tr>
</thead>
<tbody>
<tr>
<td>Rougher tails</td>
<td>34.32</td>
<td>62.84</td>
<td>2.84</td>
</tr>
<tr>
<td>Cleaner tails</td>
<td>15.05</td>
<td>6.90</td>
<td>2.14</td>
</tr>
<tr>
<td>2, slimes and 3</td>
<td>15.05</td>
<td>6.90</td>
<td>2.14</td>
</tr>
<tr>
<td>Cleaner conc. Slime</td>
<td>0.71</td>
<td>1.92</td>
<td>1.04</td>
</tr>
<tr>
<td>Acid grade product</td>
<td>12.9%</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Steel grade product</td>
<td>30.2%</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Total recovery</td>
<td>95.9%</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>
Test No. DM-70

Sample No.  
Sample weight  
Grind (lab. mill)  
Sizing  
Water used

Test Conditions:

<table>
<thead>
<tr>
<th></th>
<th>temp.</th>
<th>pH</th>
<th>cond.</th>
<th>float</th>
<th>time</th>
<th>time</th>
<th>machine</th>
<th>aeration</th>
</tr>
</thead>
<tbody>
<tr>
<td>Rougher</td>
<td>25 C</td>
<td>6.8</td>
<td>10 min.</td>
<td>12 min.</td>
<td></td>
<td></td>
<td>Fag.</td>
<td>Std.</td>
</tr>
<tr>
<td>Cleaner</td>
<td>&quot;</td>
<td>6.5</td>
<td>2</td>
<td>5</td>
<td></td>
<td></td>
<td>&quot;</td>
<td>&quot;</td>
</tr>
<tr>
<td>Deslime</td>
<td>&quot;</td>
<td>6.5</td>
<td>1</td>
<td>-</td>
<td></td>
<td></td>
<td>Bell jar</td>
<td>-</td>
</tr>
</tbody>
</table>

Reagents (lb./ton):

<table>
<thead>
<tr>
<th></th>
<th>Aero dep.</th>
<th>depressants</th>
<th>collector</th>
<th>frother</th>
<th>Aerofroth</th>
</tr>
</thead>
<tbody>
<tr>
<td>Rougher</td>
<td>0.2</td>
<td>0.3</td>
<td>0.5</td>
<td>1.2</td>
<td>0.7</td>
</tr>
<tr>
<td>Cleaner 1</td>
<td>0.2</td>
<td>0.1</td>
<td>0.3</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td>Cleaner 2</td>
<td>0.2</td>
<td>-</td>
<td>0.2</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td>Cleaner 3</td>
<td>0.1</td>
<td>0.2</td>
<td>0.2</td>
<td>-</td>
<td>-</td>
</tr>
</tbody>
</table>

Analysis:

<table>
<thead>
<tr>
<th></th>
<th>wt.</th>
<th>% carbonates</th>
<th>% insol.</th>
<th>% Si</th>
<th>% fluorite</th>
</tr>
</thead>
<tbody>
<tr>
<td>Rougher tails</td>
<td>170 gms.</td>
<td>13.88</td>
<td>72.37</td>
<td>13.78</td>
<td></td>
</tr>
<tr>
<td>Cleaner tails</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Cleaner conc.</td>
<td>70</td>
<td>2.14</td>
<td>0.64</td>
<td></td>
<td>97.22</td>
</tr>
<tr>
<td>Slime</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

Recovery:

<table>
<thead>
<tr>
<th></th>
<th>Acid grade product</th>
<th>16.6%</th>
</tr>
</thead>
<tbody>
<tr>
<td>Steel grade product</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Total recovery</td>
<td></td>
<td>94.3%</td>
</tr>
</tbody>
</table>
Test No. DM-71

Sample No.
Sample weight 600 gms.
Grind (lab. mill) 12 min. @ 60% solids
Sizing 88% minus 200 mesh
Water used Beatty

Test Conditions:

<table>
<thead>
<tr>
<th></th>
<th>temp.</th>
<th>pH</th>
<th>cond.</th>
<th>float</th>
<th>machine</th>
<th>aeration</th>
</tr>
</thead>
<tbody>
<tr>
<td>Rougher</td>
<td>25 C</td>
<td>7</td>
<td>10 min.</td>
<td>13 min.</td>
<td>Fag.</td>
<td>std.</td>
</tr>
<tr>
<td>Cleaner</td>
<td>&quot;</td>
<td>7</td>
<td>3</td>
<td>4</td>
<td>&quot;</td>
<td>&quot;</td>
</tr>
<tr>
<td>Deslime</td>
<td>&quot;</td>
<td>7.1</td>
<td>1</td>
<td>-</td>
<td>Bell jar</td>
<td>-</td>
</tr>
</tbody>
</table>

Reagents (lb./ton):

<table>
<thead>
<tr>
<th></th>
<th>Aero dep.</th>
<th>depressants</th>
<th>collector</th>
<th>frother</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>610</td>
<td>Que</td>
<td>Na silicate</td>
<td>Oleic Tall</td>
</tr>
<tr>
<td>Rougher</td>
<td></td>
<td>0.2</td>
<td>0.3</td>
<td>0.5</td>
</tr>
<tr>
<td>Cleaner</td>
<td></td>
<td>0.2</td>
<td>0.1</td>
<td>0.3</td>
</tr>
<tr>
<td></td>
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<td>0.2</td>
<td>0.2</td>
</tr>
<tr>
<td></td>
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Analysis:

<table>
<thead>
<tr>
<th></th>
<th>wt.</th>
<th>% carbonates</th>
<th>% insol.</th>
<th>% Si</th>
<th>% fluorite</th>
</tr>
</thead>
<tbody>
<tr>
<td>Rougher tails</td>
<td>185 gms.</td>
<td>13.40</td>
<td>71.32</td>
<td>15.28</td>
<td></td>
</tr>
<tr>
<td>Cleaner tails</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>2, slimes and 3</td>
<td>286</td>
<td>3.34</td>
<td>2.08</td>
<td>94.58</td>
<td></td>
</tr>
<tr>
<td>Cleaner conc. Slime</td>
<td>43</td>
<td>2.16</td>
<td>0.72</td>
<td>97.12</td>
<td></td>
</tr>
</tbody>
</table>

Recovery:

<p>| | |</p>
<table>
<thead>
<tr>
<th></th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>Acid grade product</td>
<td>10.2%</td>
</tr>
<tr>
<td>Steel grade product</td>
<td>66.0%</td>
</tr>
<tr>
<td>Total recovery</td>
<td>93.1%</td>
</tr>
</tbody>
</table>
**Test No. DM-72**

Sample No. 3
Sample weight 600 gms.
Grind (lab. mill) 10 min. @ 60% solids
Sizing 83.2% minus 200 mesh
Water used Reno

**Test Conditions:**

<table>
<thead>
<tr>
<th></th>
<th>temp.</th>
<th>pH</th>
<th>cond. time</th>
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<th>machine</th>
<th>aeration</th>
</tr>
</thead>
<tbody>
<tr>
<td>Rougher</td>
<td>24 C</td>
<td>7.1</td>
<td>8 min.</td>
<td>11 min.</td>
<td>Fag.</td>
<td>Std.</td>
</tr>
<tr>
<td>Cleaner</td>
<td>&quot;</td>
<td>7</td>
<td>1</td>
<td>4</td>
<td>&quot;</td>
<td>&quot;</td>
</tr>
<tr>
<td>Deslime</td>
<td>&quot;</td>
<td>6.8</td>
<td>1</td>
<td>-</td>
<td>Bell jar</td>
<td>-</td>
</tr>
</tbody>
</table>

**Reagents (lb./ton):**

<table>
<thead>
<tr>
<th></th>
<th>depressants</th>
<th>collector</th>
<th>frother</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Starch</td>
<td>Orzan S</td>
<td>Na silicate</td>
</tr>
<tr>
<td>Rougher</td>
<td>0.2</td>
<td>0.5</td>
<td>0.5</td>
</tr>
<tr>
<td>Cleaner</td>
<td>0.2</td>
<td>0.1</td>
<td>0.3</td>
</tr>
<tr>
<td>1</td>
<td>0.2</td>
<td>-</td>
<td>0.2</td>
</tr>
<tr>
<td>2</td>
<td>0.2</td>
<td>0.2</td>
<td>0.2</td>
</tr>
<tr>
<td>3</td>
<td>0.2</td>
<td>-</td>
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**Analysis:**

<table>
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<tr>
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<th>% carbonates</th>
<th>% insol.</th>
<th>% Si</th>
<th>% fluorite</th>
</tr>
</thead>
<tbody>
<tr>
<td>Rougher tails</td>
<td>165 gms.</td>
<td>9.46</td>
<td>71.20</td>
<td>-</td>
<td>19.34</td>
</tr>
<tr>
<td>Cleaner tails 2, slime and 3</td>
<td>184</td>
<td>3.18</td>
<td>3.52</td>
<td>93.30</td>
<td></td>
</tr>
<tr>
<td>Cleaner conc. Slime</td>
<td>180</td>
<td>1.56</td>
<td>0.86</td>
<td>97.58</td>
<td></td>
</tr>
</tbody>
</table>

**Recovery:**

Acid grade product 41.9%
Steel grade product 41.8%
Total recovery 92.2%
Test No. DM-74

Sample No.  3
Sample weight 600 gms.
Grind (lab. mill) 12 min. @ 60% solids
Sizing 87.2% minus 200 mesh
Water used Beatty

Test Conditions:

<table>
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<tr>
<th></th>
<th>temp.</th>
<th>pH</th>
<th>cond. time</th>
<th>float time</th>
<th>machine</th>
<th>aeration</th>
</tr>
</thead>
<tbody>
<tr>
<td>Rougher</td>
<td>26 C</td>
<td>7</td>
<td>10 min.</td>
<td>12 min.</td>
<td>Fag</td>
<td>Std.</td>
</tr>
<tr>
<td>Cleaner</td>
<td>&quot;</td>
<td>7.1</td>
<td>2</td>
<td>5</td>
<td>&quot;</td>
<td>&quot;</td>
</tr>
<tr>
<td>Deslime</td>
<td>&quot;</td>
<td>7.1</td>
<td>1</td>
<td>-</td>
<td>Bell jar</td>
<td>-</td>
</tr>
</tbody>
</table>

Reagents (lb./ton):

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<thead>
<tr>
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<th>depressants</th>
<th>collector</th>
<th>frother</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Starch</td>
<td>Orzan S</td>
<td>Na silicate</td>
</tr>
<tr>
<td>Rougher</td>
<td>0.2</td>
<td>0.5</td>
<td>0.5</td>
</tr>
<tr>
<td>Cleaner</td>
<td>0.2</td>
<td>0.1</td>
<td>0.3</td>
</tr>
<tr>
<td>1</td>
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<td>0.2</td>
<td>0.2</td>
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<tr>
<td>3</td>
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<td>0.2</td>
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Analysis:

<table>
<thead>
<tr>
<th></th>
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<th>% insol.</th>
<th>% Si</th>
<th>% fluorite</th>
</tr>
</thead>
<tbody>
<tr>
<td>Rougher tails</td>
<td>172 gms.</td>
<td>9.88</td>
<td>76.06</td>
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<td>14.06</td>
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<tr>
<td>Cleaner tails 2, slimes and 3</td>
<td>166</td>
<td>5.00</td>
<td>4.44</td>
<td>2.70</td>
<td>90.56</td>
</tr>
<tr>
<td>Cleaner conc. Slime</td>
<td>201</td>
<td>1.18</td>
<td>1.16</td>
<td>0.62</td>
<td>97.66</td>
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Recovery:

<p>| | |</p>
<table>
<thead>
<tr>
<th></th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>Acid grade product</td>
<td>47.5%</td>
</tr>
<tr>
<td>Steel grade product</td>
<td>36.4%</td>
</tr>
<tr>
<td>Total recovery</td>
<td>94.1%</td>
</tr>
</tbody>
</table>
Test No. DM-90

Sample No.
Sample weight
Grind (lab. mill)
Sizing
Water used

Test Conditions:

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<th></th>
<th>temp.</th>
<th>pH</th>
<th>cond. time</th>
<th>float time</th>
<th>machine</th>
<th>aeration</th>
</tr>
</thead>
<tbody>
<tr>
<td>Rougher</td>
<td>27 c</td>
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<td>8 min.</td>
<td>10 min.</td>
<td>Fag.</td>
<td>Std.</td>
</tr>
<tr>
<td>Cleaner</td>
<td>&quot;</td>
<td>7.2</td>
<td>2</td>
<td>4</td>
<td>&quot;</td>
<td>&quot;</td>
</tr>
<tr>
<td>Deslime</td>
<td>&quot;</td>
<td>7.2</td>
<td>1</td>
<td>-</td>
<td>Bell jar</td>
<td>-</td>
</tr>
</tbody>
</table>

Reagents (lb./ton):

<table>
<thead>
<tr>
<th></th>
<th>depressants</th>
<th>collector</th>
<th>frother</th>
<th>Aerofroth</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Starch</td>
<td>Que.</td>
<td>Na silicate</td>
<td>Oleic Tall</td>
</tr>
<tr>
<td>Rougher</td>
<td>0.3</td>
<td>0.2</td>
<td>0.3</td>
<td>1.0</td>
</tr>
<tr>
<td>Cleaner</td>
<td>1</td>
<td>0.2</td>
<td>-</td>
<td>0.2</td>
</tr>
<tr>
<td>2</td>
<td>0.2</td>
<td></td>
<td>0.2</td>
<td>-</td>
</tr>
<tr>
<td>3</td>
<td>0.2</td>
<td>0.1</td>
<td>0.2</td>
<td>0.1</td>
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</tbody>
</table>

Analysis:

<table>
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<tr>
<th></th>
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<th>% insol.</th>
<th>% Si</th>
<th>% fluorite</th>
</tr>
</thead>
<tbody>
<tr>
<td>Rougher tails</td>
<td>204 gms.</td>
<td>32.12</td>
<td>47.22</td>
<td></td>
<td>12.50</td>
</tr>
<tr>
<td>Cleaner tails 1</td>
<td>86</td>
<td>38.12</td>
<td>17.20</td>
<td></td>
<td>44.68</td>
</tr>
<tr>
<td>2, slimes and 3</td>
<td>218</td>
<td>14.12</td>
<td>1.74</td>
<td>1.01</td>
<td>84.20</td>
</tr>
<tr>
<td>Cleaner conc.</td>
<td>92</td>
<td>1.78</td>
<td>0.98</td>
<td>0.61</td>
<td>97.24</td>
</tr>
<tr>
<td>Slime</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

Recovery:

<p>| | |</p>
<table>
<thead>
<tr>
<th></th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>Acid grade product</td>
<td>26.1%</td>
</tr>
<tr>
<td>Steel grade product</td>
<td>55.0%</td>
</tr>
<tr>
<td>Total recovery</td>
<td>92.3%</td>
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</table>
**Test No. DM-42**

Sample No.  
Sample weight  
Grind (lab. mill)  
Sizing  
Water used

Test Conditions:

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<th></th>
<th>temp.</th>
<th>pH</th>
<th>time</th>
<th>float time</th>
<th>machine</th>
<th>aeration</th>
</tr>
</thead>
<tbody>
<tr>
<td>Rougher</td>
<td>25 C</td>
<td>7</td>
<td>8 min.</td>
<td>13 min.</td>
<td>Fag.</td>
<td>sub.</td>
</tr>
<tr>
<td>Cleaner</td>
<td></td>
<td>6.8</td>
<td>1</td>
<td>5</td>
<td></td>
<td>sub.</td>
</tr>
<tr>
<td>Deslime</td>
<td></td>
<td>6.6</td>
<td>1</td>
<td></td>
<td>Bell jar</td>
<td></td>
</tr>
</tbody>
</table>

Reagents (lb./ton):

<table>
<thead>
<tr>
<th></th>
<th>Starch</th>
<th>depressants</th>
<th>collector</th>
<th>frother</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>S-3223</td>
<td>Na silicate</td>
<td>723 reg.</td>
<td>73</td>
</tr>
<tr>
<td>Rougher</td>
<td>0.1</td>
<td>0.10</td>
<td>1.6</td>
<td>0.08</td>
</tr>
<tr>
<td>Cleaner</td>
<td>1</td>
<td>0.18</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td></td>
<td>2</td>
<td>0.20</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td></td>
<td>3</td>
<td>0.20</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td></td>
<td>4</td>
<td>0.20</td>
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<td>-</td>
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</tbody>
</table>

Analysis:

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<tr>
<th></th>
<th>wt.</th>
<th>% carbonates</th>
<th>% insol.</th>
<th>% Si</th>
<th>% fluorite</th>
</tr>
</thead>
<tbody>
<tr>
<td>Rougher tails</td>
<td>90 gms.</td>
<td>25.32</td>
<td>56.04</td>
<td></td>
<td>18.64</td>
</tr>
<tr>
<td>Cleaner tails</td>
<td>290</td>
<td>53.16</td>
<td>2.54</td>
<td>44.30</td>
<td></td>
</tr>
<tr>
<td>2,3,slimes and 4</td>
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<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Cleaner conc. Slime</td>
<td>122</td>
<td>53.82</td>
<td>2.04</td>
<td>44.20</td>
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</tbody>
</table>

Recovery:

<table>
<thead>
<tr>
<th></th>
<th>Acid grade product</th>
<th>Steel grade product</th>
<th>Total recovery</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td></td>
<td>93.1%</td>
</tr>
</tbody>
</table>
Test No. DM-63

Sample No. 5
Sample weight 600 gms.
Grind (lab. mill) 12 min. @ 60% solids
Sizing 93.4% minus 200 mesh
Water used Reno

Test Conditions:

<table>
<thead>
<tr>
<th></th>
<th>temp.</th>
<th>pH</th>
<th>cond. time</th>
<th>float time</th>
<th>machine</th>
<th>aeration</th>
</tr>
</thead>
<tbody>
<tr>
<td>Rougher</td>
<td>25 C</td>
<td>7</td>
<td>10 min.</td>
<td>15 min.</td>
<td>Fag.</td>
<td>sub.</td>
</tr>
<tr>
<td>Cleaner</td>
<td></td>
<td>6.8</td>
<td>2</td>
<td>4</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Deslime</td>
<td></td>
<td>6.6</td>
<td>1</td>
<td>-</td>
<td>Bell jar</td>
<td></td>
</tr>
</tbody>
</table>

Reagents (lb./ton):

<table>
<thead>
<tr>
<th></th>
<th>depressants</th>
<th>collector</th>
<th>frother</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Starch</td>
<td>Que.</td>
<td>Na silicate</td>
</tr>
<tr>
<td>Rougher</td>
<td>0.2</td>
<td>0.2</td>
<td>0.2</td>
</tr>
<tr>
<td>Cleaner</td>
<td>0.2</td>
<td>0.2</td>
<td>0.2</td>
</tr>
<tr>
<td>1</td>
<td>0.2</td>
<td>0.4</td>
<td>0.2</td>
</tr>
<tr>
<td>2</td>
<td>0.2</td>
<td>0.4</td>
<td>0.2</td>
</tr>
<tr>
<td>3</td>
<td>0.2</td>
<td>0.2</td>
<td>0.2</td>
</tr>
</tbody>
</table>

Analysis:

<table>
<thead>
<tr>
<th></th>
<th>wt.</th>
<th>% carbonates</th>
<th>% insol.</th>
<th>% Si</th>
<th>% fluorite</th>
</tr>
</thead>
<tbody>
<tr>
<td>Rougher tails</td>
<td>93 gms.</td>
<td>25.56</td>
<td>56.80</td>
<td></td>
<td>17.64</td>
</tr>
<tr>
<td>Cleaner tails</td>
<td>336</td>
<td>53.72</td>
<td>2.22</td>
<td>44.06</td>
<td></td>
</tr>
<tr>
<td>2, slimes and 3</td>
<td>336</td>
<td>53.72</td>
<td>2.22</td>
<td>44.06</td>
<td></td>
</tr>
<tr>
<td>Cleaner conc. Slime</td>
<td>75</td>
<td>6.16</td>
<td>1.32</td>
<td>92.52</td>
<td></td>
</tr>
</tbody>
</table>

Recovery:

Acid grade product
Steel grade product
Total recovery 97.4%
Test No. DM-83

Sample No. 5
Sample weight 700 gms.
Grind (lab. mill) 10 min. @ 50% solids
Sizing 84.2% minus 200 mesh
Water used Reno

Test Conditions:

<table>
<thead>
<tr>
<th></th>
<th>temp.</th>
<th>pH</th>
<th>cond. time</th>
<th>float time</th>
<th>machine</th>
<th>aeration</th>
</tr>
</thead>
<tbody>
<tr>
<td>Rougher</td>
<td>26 C</td>
<td>7</td>
<td>8 min.</td>
<td>13 min.</td>
<td>Fag.</td>
<td>sub.</td>
</tr>
<tr>
<td>Cleaner</td>
<td>&quot;</td>
<td>6.8</td>
<td>1</td>
<td>3</td>
<td>&quot;</td>
<td>&quot;</td>
</tr>
<tr>
<td>Deslize</td>
<td>&quot;</td>
<td>6.8</td>
<td>1</td>
<td></td>
<td>Bell jar</td>
<td></td>
</tr>
</tbody>
</table>

Reagents (lb./ton):

<table>
<thead>
<tr>
<th></th>
<th>Starch</th>
<th>Pal.</th>
<th>Na silicate</th>
<th>Oleic Tall</th>
<th>frother Aerofroth</th>
</tr>
</thead>
<tbody>
<tr>
<td>Rougher</td>
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<td>0.9</td>
</tr>
<tr>
<td>Cleaner</td>
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<tr>
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<td>0.3</td>
<td>0.2</td>
<td></td>
<td>-</td>
</tr>
<tr>
<td>2</td>
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<td>0.2</td>
<td>0.2</td>
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</tr>
<tr>
<td>3</td>
<td>0.2</td>
<td>0.1</td>
<td>0.2</td>
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</table>

Analysis:

<table>
<thead>
<tr>
<th></th>
<th>wt.</th>
<th>% carbonates</th>
<th>% insol.</th>
<th>% Si</th>
<th>% fluorite</th>
</tr>
</thead>
<tbody>
<tr>
<td>Rougher tails</td>
<td>233 gms.</td>
<td>53.92</td>
<td>32.64</td>
<td></td>
<td>13.44</td>
</tr>
<tr>
<td>Cleaner tails</td>
<td>244</td>
<td>33.38</td>
<td>1.80</td>
<td></td>
<td>64.82</td>
</tr>
<tr>
<td>2, slimes and 3</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Cleaner conc. Slime</td>
<td>37</td>
<td>9.64</td>
<td>0.92</td>
<td></td>
<td>89.44</td>
</tr>
</tbody>
</table>

Recovery:

<table>
<thead>
<tr>
<th></th>
<th>Acid grade product</th>
<th>9.6%</th>
</tr>
</thead>
<tbody>
<tr>
<td>Steel grade product</td>
<td>45.9%</td>
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</tr>
<tr>
<td>Total recovery</td>
<td>90.9%</td>
<td></td>
</tr>
</tbody>
</table>
### Test Conditions:

<table>
<thead>
<tr>
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<th>cond. time</th>
<th>float time</th>
<th>machine</th>
<th>aeration</th>
</tr>
</thead>
<tbody>
<tr>
<td>Rougher</td>
<td>26 C</td>
<td>7</td>
<td>8 min.</td>
<td>13 min.</td>
<td>Fag.</td>
<td>std.</td>
</tr>
<tr>
<td>Cleaner</td>
<td>&quot;</td>
<td>7.2</td>
<td>1</td>
<td>3</td>
<td>&quot;</td>
<td>&quot;</td>
</tr>
<tr>
<td>Deslime</td>
<td>&quot;</td>
<td>7.2</td>
<td>1</td>
<td>-</td>
<td>Bell jar</td>
<td>-</td>
</tr>
</tbody>
</table>

### Reagents (lb./ton):

<table>
<thead>
<tr>
<th></th>
<th>depressants</th>
<th>collector</th>
<th>frother</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Starch</td>
<td>Que.</td>
<td>Na silicate</td>
</tr>
<tr>
<td>Rougher</td>
<td>0.3</td>
<td>0.2</td>
<td>0.3</td>
</tr>
<tr>
<td>Cleaner</td>
<td>1</td>
<td>0.2</td>
<td>0.3</td>
</tr>
<tr>
<td></td>
<td>2</td>
<td>0.2</td>
<td>0.3</td>
</tr>
<tr>
<td></td>
<td>3</td>
<td>0.2</td>
<td>0.1</td>
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### Analysis:

<table>
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<tr>
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<th>% insol.</th>
<th>% Si</th>
<th>% fluorite</th>
</tr>
</thead>
<tbody>
<tr>
<td>Rougher tails</td>
<td>106 gms.</td>
<td>39.96</td>
<td>47.74</td>
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<td>12.30</td>
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<td>Cleaner tails</td>
<td>196</td>
<td>78.62</td>
<td>7.76</td>
<td></td>
<td>13.62</td>
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<td></td>
<td>1</td>
<td>78.62</td>
<td>7.76</td>
<td></td>
<td>13.62</td>
</tr>
<tr>
<td></td>
<td>2, slimes and 3</td>
<td>36.34</td>
<td>1.52</td>
<td>1.31</td>
<td>62.14</td>
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<tr>
<td>Cleaner conc.Slime</td>
<td>56</td>
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<td>0.45</td>
<td>96.13</td>
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</table>

### Recovery:

<table>
<thead>
<tr>
<th></th>
<th>Acid grade product</th>
<th>Steel grade product</th>
<th>Total recovery</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>22.2%</td>
<td>61.3%</td>
<td>94.7%</td>
</tr>
</tbody>
</table>
### Test No. DM-87

**Sample No.**
**Sample weight** 700 gms.
**Grind (lab. mill)** 12 min. @ 60% solids
**Sizing** 85% minus 200 mesh
**Water used** Reno

#### Test Conditions:

<table>
<thead>
<tr>
<th></th>
<th>temp.</th>
<th>pH</th>
<th>cond. time</th>
<th>float time</th>
<th>machine</th>
<th>aeration</th>
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</thead>
<tbody>
<tr>
<td>Rougher</td>
<td>27 C</td>
<td>7</td>
<td>10 min.</td>
<td>14 min.</td>
<td>Fag.</td>
<td>std.</td>
</tr>
<tr>
<td>Cleaner</td>
<td></td>
<td>6.8</td>
<td>2</td>
<td>4</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Deslime</td>
<td></td>
<td>6.6</td>
<td>1</td>
<td></td>
<td>Bell jar</td>
<td></td>
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#### Reagents (lb./ton):

<table>
<thead>
<tr>
<th></th>
<th>depressants</th>
<th>collector</th>
<th>frother</th>
</tr>
</thead>
<tbody>
<tr>
<td>Starch</td>
<td>Orzan S</td>
<td>Na silicate</td>
<td>Oleic Tall</td>
</tr>
<tr>
<td>Rougher</td>
<td>0.2</td>
<td>0.3</td>
<td>0.4</td>
</tr>
<tr>
<td>Cleaner</td>
<td>1</td>
<td>0.2</td>
<td>0.2</td>
</tr>
<tr>
<td>2</td>
<td>0.2</td>
<td>0.2</td>
<td>0.2</td>
</tr>
<tr>
<td>3</td>
<td>0.2</td>
<td>0.2</td>
<td>0.2</td>
</tr>
<tr>
<td></td>
<td>0.3</td>
<td>0.4</td>
<td>0.2</td>
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#### Analysis:

<table>
<thead>
<tr>
<th></th>
<th>wt.</th>
<th>% carbonates</th>
<th>% insol.</th>
<th>% Si</th>
<th>% fluorite</th>
</tr>
</thead>
<tbody>
<tr>
<td>Rougher tails</td>
<td>216 gms.</td>
<td>26.12</td>
<td>59.90</td>
<td></td>
<td>13.92</td>
</tr>
<tr>
<td>Cleaner tails</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>2, slimes and 3</td>
<td>281</td>
<td>19.76</td>
<td>2.30</td>
<td></td>
<td>77.94</td>
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<tr>
<td>Cleaner conc.</td>
<td>81</td>
<td>7.15</td>
<td>0.97</td>
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<td>91.88</td>
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<tr>
<td>Slime</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

#### Recovery:

- Acid grade product: 18.5%
- Steel grade product: 54.4%
- Total recovery: 92.5%
Test No. DM-91

<table>
<thead>
<tr>
<th>Sample No.</th>
<th>6</th>
</tr>
</thead>
<tbody>
<tr>
<td>Sample weight</td>
<td>600 gms.</td>
</tr>
<tr>
<td>Grind (lab. mill)</td>
<td>10 min. @ 50% solids</td>
</tr>
<tr>
<td>Sizing</td>
<td>79.8% minus 200 mesh</td>
</tr>
<tr>
<td>Water used</td>
<td>Reno</td>
</tr>
</tbody>
</table>

Test Conditions:

<table>
<thead>
<tr>
<th></th>
<th>temp.</th>
<th>pH</th>
<th>cond. time</th>
<th>float time</th>
<th>machine</th>
<th>aeration</th>
</tr>
</thead>
<tbody>
<tr>
<td>Rougher</td>
<td>25 C</td>
<td>7.1</td>
<td>8 min.</td>
<td>13 min.</td>
<td>Fag.</td>
<td>std.</td>
</tr>
<tr>
<td>Cleaner</td>
<td>&quot;</td>
<td>7.1</td>
<td>2</td>
<td>4</td>
<td>&quot;</td>
<td>&quot;</td>
</tr>
<tr>
<td>Deslime</td>
<td>&quot;</td>
<td>7.2</td>
<td>1</td>
<td>-</td>
<td>Bell jar</td>
<td>-</td>
</tr>
</tbody>
</table>

Reagents (lb./ton):

<table>
<thead>
<tr>
<th></th>
<th>depressants</th>
<th>collector</th>
<th>frother</th>
<th>73</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Starch</td>
<td>Qua.</td>
<td>Na silicate</td>
<td>Oleic Tall</td>
</tr>
<tr>
<td>Rougher</td>
<td>0.4</td>
<td>0.2</td>
<td>0.4</td>
<td>0.7</td>
</tr>
<tr>
<td>Cleaner 1</td>
<td>0.2</td>
<td>-</td>
<td>0.2</td>
<td>-</td>
</tr>
<tr>
<td>2</td>
<td>0.2</td>
<td>0.2</td>
<td>0.2</td>
<td>-</td>
</tr>
<tr>
<td>3</td>
<td>0.2</td>
<td>0.1</td>
<td>0.2</td>
<td>0.1</td>
</tr>
</tbody>
</table>

Analysis:

<table>
<thead>
<tr>
<th></th>
<th>wt.</th>
<th>% carbonates</th>
<th>% insol.</th>
<th>% Si</th>
<th>% fluorite</th>
</tr>
</thead>
<tbody>
<tr>
<td>Rougher tails</td>
<td>201 gms.</td>
<td>32.60</td>
<td>43.06</td>
<td>15.40</td>
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</tr>
<tr>
<td>Cleaner tails 1</td>
<td>63</td>
<td>35.34</td>
<td>15.22</td>
<td>49.44</td>
<td></td>
</tr>
<tr>
<td>2, slimes and 3</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Cleaner conc. Slime</td>
<td>235</td>
<td>20.08</td>
<td>1.28</td>
<td>0.98</td>
<td>78.64</td>
</tr>
<tr>
<td>101</td>
<td>2.18</td>
<td>0.76</td>
<td>0.49</td>
<td>97.16</td>
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</table>

Recovery:

<p>| | | | | | |</p>
<table>
<thead>
<tr>
<th></th>
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<th></th>
<th></th>
<th></th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>Acid grade product</td>
<td>28.5%</td>
<td>Steel grade product</td>
<td>53.5%</td>
<td>Total recovery</td>
<td>91.0%</td>
</tr>
</tbody>
</table>
Test No. DM-76

Sample No. 7
Sample weight 600 gms.
Grind (lab. mill) 12 min. @ 60% solids
Sizing 87% minus 200 mesh
Water used Beatty

Test Conditions:

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<tr>
<th></th>
<th>temp.</th>
<th>pH</th>
<th>cond. time</th>
<th>float time</th>
<th>machine</th>
<th>aeration</th>
</tr>
</thead>
<tbody>
<tr>
<td>Rougher</td>
<td>26 C</td>
<td>7</td>
<td>10 min.</td>
<td>14 min.</td>
<td>Fag.</td>
<td>std.</td>
</tr>
<tr>
<td>Cleaner</td>
<td></td>
<td>7.2</td>
<td>1</td>
<td>4</td>
<td>&quot;</td>
<td>&quot;</td>
</tr>
<tr>
<td>Deslime</td>
<td></td>
<td>7.1</td>
<td>1</td>
<td>-</td>
<td>Bell jar</td>
<td>-</td>
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</tbody>
</table>

Reagents (lb./ton):

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<tr>
<th></th>
<th>Aero dep.</th>
<th>depressants</th>
<th>collector</th>
<th>frother</th>
<th>Aerofloat</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>610</td>
<td>Organ S</td>
<td>Na silicate</td>
<td>Oleic Tall</td>
<td>73</td>
</tr>
<tr>
<td>Rougher</td>
<td></td>
<td>0.3</td>
<td>0.5</td>
<td>1.2</td>
<td>-</td>
</tr>
<tr>
<td>Cleaner</td>
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<td>0.1</td>
<td>0.3</td>
<td>-</td>
</tr>
<tr>
<td>1</td>
<td></td>
<td>0.2</td>
<td>0.2</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td>2</td>
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<td>0.2</td>
<td>0.2</td>
<td>-</td>
<td>-</td>
</tr>
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<td>3</td>
<td></td>
<td>0.2</td>
<td>0.2</td>
<td>-</td>
<td>0.3</td>
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Analysis:

<table>
<thead>
<tr>
<th>wt.</th>
<th>% carbonates</th>
<th>% insol.</th>
<th>% Si</th>
<th>% fluorite</th>
</tr>
</thead>
<tbody>
<tr>
<td>Rougher tails</td>
<td>120 gms.</td>
<td>37.94</td>
<td>42.22</td>
<td>19.84</td>
</tr>
<tr>
<td>Cleaner tails</td>
<td>266</td>
<td>31.92</td>
<td>2.28</td>
<td>65.80</td>
</tr>
<tr>
<td></td>
<td>2, slimes and 3</td>
<td>3.20</td>
<td>1.00</td>
<td>95.80</td>
</tr>
<tr>
<td>Cleaner conc. Slime</td>
<td>71</td>
<td></td>
<td>0.44</td>
<td></td>
</tr>
</tbody>
</table>

Recovery:

- Acid grade product 23.8%
- Steel grade product 3.8%
- Total recovery 91.7%
Test No. DM-84

Sample No. 
Sample weight 7
Grind (lab. mill) 700 gms. 
Sizing 12 min. @ 60% solids 
Water used 85% minus 200 mesh 

Test Conditions:

<table>
<thead>
<tr>
<th>temp.</th>
<th>pH</th>
<th>cond.</th>
<th>time</th>
<th>float</th>
<th>time</th>
<th>machine</th>
<th>aeration</th>
</tr>
</thead>
<tbody>
<tr>
<td>Rougher</td>
<td>25 C-</td>
<td>7</td>
<td>10 min.</td>
<td>14 min.</td>
<td>Fag.</td>
<td>std.</td>
<td></td>
</tr>
<tr>
<td>Cleaner</td>
<td>&quot;</td>
<td>6.8</td>
<td>2</td>
<td>4</td>
<td>&quot;</td>
<td>&quot;</td>
<td></td>
</tr>
<tr>
<td>Deslime</td>
<td>&quot;</td>
<td>6.8</td>
<td>1</td>
<td>-</td>
<td>Bell jar</td>
<td>&quot;</td>
<td></td>
</tr>
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Reagents (lb./ton):

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<tr>
<th>depressants</th>
<th>collector</th>
<th>frother</th>
</tr>
</thead>
<tbody>
<tr>
<td>Starch</td>
<td>Pal.</td>
<td>Na silicate</td>
</tr>
<tr>
<td>Rougher</td>
<td>Cleaner</td>
<td>1</td>
</tr>
<tr>
<td>0.2</td>
<td>0.1</td>
<td>0.4</td>
</tr>
<tr>
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</tr>
<tr>
<td>0.2</td>
<td>0.1</td>
<td>0.2</td>
</tr>
</tbody>
</table>

Analysis:

<table>
<thead>
<tr>
<th>wt.</th>
<th>% carbonates</th>
<th>% insol.</th>
<th>% Si</th>
<th>% fluorite</th>
</tr>
</thead>
<tbody>
<tr>
<td>Rougher tails</td>
<td>174 gms.</td>
<td>46.12</td>
<td>37.84</td>
<td>16.04</td>
</tr>
<tr>
<td>Cleaner tails</td>
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<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>2, slimes</td>
<td>207</td>
<td>17.58</td>
<td>2.22</td>
<td>80.20</td>
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<tr>
<td>and 3</td>
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<td></td>
</tr>
<tr>
<td>Cleaner conc.</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Slime</td>
<td>57</td>
<td>2.38</td>
<td>0.54</td>
<td>97.08</td>
</tr>
</tbody>
</table>

Recovery:

| Acid grade product | 19.5% |
| Steel grade product | 58.4% |
| Total recovery | 90.2% |
Test No. DM-86

Sample No.
Sample weight
Grind (lab. mill)
Sizing
Water used

Test Conditions:

<table>
<thead>
<tr>
<th></th>
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<th>pH</th>
<th>cond. time</th>
<th>float time</th>
<th>machine</th>
<th>aeration</th>
</tr>
</thead>
<tbody>
<tr>
<td>Rougher</td>
<td>26 C</td>
<td>7</td>
<td>8 min.</td>
<td>12 min.</td>
<td>Fag.</td>
<td>std.</td>
</tr>
<tr>
<td>Cleaner</td>
<td>&quot;</td>
<td>7.2</td>
<td>1</td>
<td>4</td>
<td>&quot;</td>
<td>&quot;</td>
</tr>
<tr>
<td>Deslime</td>
<td>&quot;</td>
<td>7.2</td>
<td>1</td>
<td>-</td>
<td>Bell jar</td>
<td>-</td>
</tr>
</tbody>
</table>

Reagents (lb./ton):

<table>
<thead>
<tr>
<th></th>
<th>depressants</th>
<th>collector</th>
<th>frother Aerofroth</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Starch</td>
<td>Orzan S</td>
<td>Na Silicate</td>
</tr>
<tr>
<td>Rougher</td>
<td>0.2</td>
<td>0.3</td>
<td>0.4</td>
</tr>
<tr>
<td>Cleaner</td>
<td>1 0.2</td>
<td>0.1</td>
<td>0.3</td>
</tr>
<tr>
<td></td>
<td>2 0.2</td>
<td>0.2</td>
<td>0.2</td>
</tr>
<tr>
<td></td>
<td>3 0.2</td>
<td>0.2</td>
<td>0.2</td>
</tr>
</tbody>
</table>

Analysis:

<table>
<thead>
<tr>
<th></th>
<th>wt.</th>
<th>% carbonates</th>
<th>% insol.</th>
<th>% Si</th>
<th>% fluorite</th>
</tr>
</thead>
<tbody>
<tr>
<td>Rougher tails</td>
<td>175 gms.</td>
<td>41.34</td>
<td>39.74</td>
<td></td>
<td>18.92</td>
</tr>
<tr>
<td>Cleaner tails</td>
<td>208</td>
<td>25.12</td>
<td>1.58</td>
<td></td>
<td>73.30</td>
</tr>
<tr>
<td>and 3</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Cleaner conc. Slime</td>
<td>104</td>
<td>4.25</td>
<td>1.04</td>
<td></td>
<td>94.71</td>
</tr>
</tbody>
</table>

Recovery:

Acid grade product 35.2%
Steel grade product 53.6%
Total recovery 88.3%
Test No. DM-36

Sample No. 8
Sample weight 500 gms.
Grind (lab. mill) 10 min. @ 50% solids
Sizing 82.3% minus 200 mesh
Water used Reno

Test Conditions:

<table>
<thead>
<tr>
<th></th>
<th>temp.</th>
<th>pH</th>
<th>cond.</th>
<th>float</th>
<th>time</th>
<th>machine</th>
<th>aeration</th>
</tr>
</thead>
<tbody>
<tr>
<td>Rougher</td>
<td>27 C</td>
<td>7</td>
<td>8 min.</td>
<td>10 min.</td>
<td>10 min.</td>
<td>Fag.</td>
<td>std.</td>
</tr>
<tr>
<td>Cleaner</td>
<td>&quot;</td>
<td>7</td>
<td>2</td>
<td>4</td>
<td>&quot;</td>
<td>&quot;</td>
<td>&quot;</td>
</tr>
<tr>
<td>Deslime</td>
<td>6.7</td>
<td>1</td>
<td>-</td>
<td>Bell jar</td>
<td>-</td>
<td>&quot;</td>
<td>&quot;</td>
</tr>
</tbody>
</table>

Reagents (lb./ton):

<table>
<thead>
<tr>
<th></th>
<th>depressants</th>
<th>collector</th>
<th>frother</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>S-3223</td>
<td>Na silicate</td>
<td>Aerofloat</td>
</tr>
<tr>
<td>Rougher</td>
<td>0.4</td>
<td>0.3</td>
<td>1.3</td>
</tr>
<tr>
<td>Cleaner</td>
<td>1</td>
<td>0.1</td>
<td>0.2</td>
</tr>
<tr>
<td></td>
<td>2</td>
<td>0.2</td>
<td>0.2</td>
</tr>
<tr>
<td></td>
<td>3</td>
<td>0.2</td>
<td>0.1</td>
</tr>
</tbody>
</table>

Analysis:

<table>
<thead>
<tr>
<th></th>
<th>wt.</th>
<th>% carbonates</th>
<th>% insol.</th>
<th>% Si</th>
<th>% fluorite</th>
</tr>
</thead>
<tbody>
<tr>
<td>Rougher tails</td>
<td>139 gms.</td>
<td>46.00</td>
<td>25.38</td>
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<td>28.62</td>
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<td>Cleaner tails</td>
<td>182</td>
<td>36.42</td>
<td>4.36</td>
<td>3.76</td>
<td>59.22</td>
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<td></td>
<td>78</td>
<td>3.14</td>
<td>3.20</td>
<td>1.20</td>
<td>93.66</td>
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</tbody>
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Recovery:

<table>
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<tr>
<th></th>
<th>Acid grade product</th>
<th>27.1%</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Steel grade product</td>
<td>39.8%</td>
</tr>
<tr>
<td></td>
<td>Total recovery</td>
<td>85.2%</td>
</tr>
</tbody>
</table>
**Test No. DM-81**

Sample No.
Sample weight
Grind (lab. mill)
Sizing
Water used

Test Conditions:

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<thead>
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<th>temp.</th>
<th>pH</th>
<th>cond.</th>
<th>float</th>
<th>machine</th>
<th>aeration</th>
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</thead>
<tbody>
<tr>
<td>Rougher</td>
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<td>7</td>
<td>8 min.</td>
<td>12 min.</td>
<td>Fag.</td>
<td>std.</td>
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<tr>
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<td>&quot;</td>
<td>6.7</td>
<td>1</td>
<td>5</td>
<td>&quot;</td>
<td>&quot;</td>
</tr>
<tr>
<td>Deslime</td>
<td>&quot;</td>
<td>6.7</td>
<td>1</td>
<td></td>
<td>Bell jar</td>
<td></td>
</tr>
</tbody>
</table>

Reagents (lb./ton):

<table>
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<tr>
<th></th>
<th>depressants</th>
<th>collector</th>
<th>frother</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Starch</td>
<td>Orzan S</td>
<td>Na silicate</td>
</tr>
<tr>
<td>Rougher</td>
<td>0.2</td>
<td>0.3</td>
<td>0.4</td>
</tr>
<tr>
<td>Cleaner</td>
<td>0.2</td>
<td>0.2</td>
<td>0.2</td>
</tr>
<tr>
<td>1</td>
<td>0.2</td>
<td>0.2</td>
<td>0.2</td>
</tr>
<tr>
<td>2</td>
<td>0.2</td>
<td>0.2</td>
<td>0.2</td>
</tr>
<tr>
<td>3</td>
<td>0.1</td>
<td>0.2</td>
<td>0.1</td>
</tr>
</tbody>
</table>

Analysis:

<table>
<thead>
<tr>
<th></th>
<th>wt.</th>
<th>% carbonates</th>
<th>% insol.</th>
<th>% Si</th>
<th>% fluorite</th>
</tr>
</thead>
<tbody>
<tr>
<td>Rougher tails</td>
<td>171 gms.</td>
<td>38.04</td>
<td>34.50</td>
<td></td>
<td>27.46</td>
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<tr>
<td>Cleaner tails 2, slimes and 3</td>
<td>242</td>
<td>33.18</td>
<td>2.84</td>
<td>63.98</td>
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</tr>
<tr>
<td>Cleaner conc. Slime</td>
<td>95</td>
<td>3.86</td>
<td>0.94</td>
<td>95.20</td>
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</tr>
</tbody>
</table>

Recovery:

- Acid grade product: 23.8%
- Steel grade product: 40.8%
- Total recovery: 87.7%
Test No. DM-82

Sample No.  
Sample weight  
Grind (lab. mill)  
Sizing  
Water used  

Test Conditions:

<table>
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<th></th>
<th>temp.</th>
<th>pH</th>
<th>time</th>
<th>float</th>
<th>machine</th>
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<tbody>
<tr>
<td>Rougher</td>
<td>27 C</td>
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<td>8 min.</td>
<td>14 min.</td>
<td>Fag.</td>
<td>std.</td>
</tr>
<tr>
<td>Cleaner</td>
<td></td>
<td>7.2</td>
<td>1</td>
<td>5</td>
<td>&quot;</td>
<td>&quot;</td>
</tr>
<tr>
<td>Deslime</td>
<td></td>
<td>7.2</td>
<td>1</td>
<td>-</td>
<td>Bell jar</td>
<td>-</td>
</tr>
</tbody>
</table>

Reagents (lb./ton):

<table>
<thead>
<tr>
<th></th>
<th>depressants</th>
<th>collector</th>
<th>frother</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Starch</td>
<td>Pal.</td>
<td>Oleic</td>
</tr>
<tr>
<td>Rougher</td>
<td>0.2</td>
<td>0.1</td>
<td>1.0</td>
</tr>
<tr>
<td>Cleaner</td>
<td>0.2</td>
<td>0.05</td>
<td>0.2</td>
</tr>
<tr>
<td>1</td>
<td>0.2</td>
<td>0.1</td>
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</tr>
<tr>
<td>2</td>
<td>0.2</td>
<td>0.2</td>
<td>-</td>
</tr>
<tr>
<td>3</td>
<td>0.2</td>
<td>0.1</td>
<td>0.1</td>
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</tbody>
</table>

Analysis:

<table>
<thead>
<tr>
<th></th>
<th>wt.</th>
<th>% carbonates</th>
<th>% insol.</th>
<th>% Si</th>
<th>% fluorite</th>
</tr>
</thead>
<tbody>
<tr>
<td>Rougher tails</td>
<td>220 gms.</td>
<td>48.20</td>
<td>26.74</td>
<td></td>
<td>25.06</td>
</tr>
<tr>
<td>Cleaner tails</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>2, slimes and 3</td>
<td>264</td>
<td>26.72</td>
<td>1.24</td>
<td>0.41</td>
<td>72.04</td>
</tr>
<tr>
<td>Cleaner conc. Slime</td>
<td>41</td>
<td>2.38</td>
<td>0.60</td>
<td>0.30</td>
<td>97.02</td>
</tr>
</tbody>
</table>

Recovery:

<p>| | |</p>
<table>
<thead>
<tr>
<th></th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>Acid grade product</td>
<td>10.5%</td>
</tr>
<tr>
<td>Steel grade product</td>
<td>50.0%</td>
</tr>
<tr>
<td>Total recovery</td>
<td>85.0%</td>
</tr>
</tbody>
</table>
Bibliography


