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Flotation Concentration of a Montana Lead-Zinc-Silver-Ore

Grover W. Gregson

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FLOTATION CONCENTRATION OF A MONTANA
LEAD-ZINC-SILVER-ORE

by
GROVER W. GREGSON

A Thesis

Submitted to the Department of Mineral Dressing
in Partial Fulfillment of the
Requirements for the Degree of
Bachelor of Science in Metallurgical Engineering

MONTANA SCHOOL OF MINES

Butte, Montana

May, 1948

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INTRODUCTION

This thesis is concerned with the selective flotation of a lead-zinc-silver ore from the Mike Horse Mine.

History and Location

The Mike Horse Mine, in the Huddelston mining district, is fifty-two miles northwest of Helena, Montana. The mine was discovered in 1898 by Joseph Heitmiller. There was only minor production from the date of discovery until 1915; the main drawback being lack of good road. There were a few tons of ore shipped in 1915 and again in 1917. There are no available figures regarding tonnages produced. In 1919 a small mill was built and 6,000 pounds of lead-silver concentrate were produced that year. There is no record of any production from 1919 to 1923. From 1923 to 1940, the mine was operated intermittently. Between 1923 and 1943, 1,120 tons of ore were treated, the net return from which was \$50,600, or an average yield of \$48 per ton, with treatment charges averaging about \$10 per ton. The gross yield was about \$61,800.

In 1940, the Mike Horse Mining and Milling Company was formed, and this company started development work on the Mike Horse vein and the Little Nell vein, which lies on the same property. A 250 ton mill was built on the site and

the first productive year for this company was 1943. There are no available production figures of the mine or mill from 1943 to present. It is presumed that the operation was successful. In July, 1945, the mine and mill were purchased by the Federal Mining and Refining Company, a subsidiary of the American Smelting and Refining Company.

Geology. The geology of the mine consists of belt formation, intruded by a gabbro sill which is cut by a trachyte porphyry dike. Mineralizing solution penetrated the above-mentioned formations and post mineralization movement crushed the ore bodies without causing much cross faulting. There seems to be two generations of mineralization, pyrite, galena, sphalerite, and silica making up the first generation. This generation replaces parts of altered wall rock. The second generation includes copper bearing sulphides, pyrite, and silica. At a still later period, the veins were reopened and a carbonate mixture averaging 85 per cent dolomite, 12 per cent rhodochrosite, and 3 per cent siderite came in, filling the open spaces and, in places, replacing the vein material. As the veins are found in more or less shoots, considerable development work must be done ahead of stoping operations. It has been found that the ore shoots in the gabbro sill do not differ in size or mineralogy from those in the trachyte or argillite, the gabbro, trachyte, and argillite making up the country rock.

In the ore shoots, the important valuable minerals are

galena, sphalerite, with minor amounts of bornite, chalcopyrite, and tetrahedrite. Where copper is present, the ratio is about $4\frac{1}{2}$ ounces of silver to 1 per cent of lead. According to the geologic reports, the gold assay averages 0.502 ounces per ton of ore, while the assay of the head sample of a portion of ore available for this thesis problem yielded only 0.005 ounces per ton.

All of the ore in the main body of the ore being mined at present is primary, but it is known that the ore near the surface is oxidized, and to what extent the ore body is oxidized, has yet to be determined.

Organization of the Problem

In order to approach the thesis problem most directly, available hand specimens of the ore were examined macroscopically and microscopically in order to determine the minerals present. A grab sample of the crushed ore product was taken and investigated with the aid of the binocular microscope to check minerals present.

The sample of ore available for investigation, a total of 68 pounds, had been previously crushed and ground to minus 20 mesh material. Liberation studies were carried out in both the laboratory pebble and rod mills. A screen analysis of each grind test was made and a briquette sample of each screen fraction was prepared. From these samples a study was made of the degree of liberation and the optimum grinding conditions.

Upon completion of the liberation studies, flotation tests were conducted. The general procedure for these tests being to selectively separate from the mineral crude, a lead concentrate and a zinc concentrate. From the previous mineralogical studies, the pyrite content of the mineral crude suggested possible difficulties that might be encountered in flotation. On the basis of reported operation experience with similar crudes it was thought likely that it would be difficult to prevent the pyrite from contaminating zinc concentrate.

Mineralogical Analysis

Macroscopic Analysis. Hand specimens of the ore examined showed a close relationship between the galena, the dominant valuable mineral, and sphalerite. Pyrite was disseminated throughout the vein material and country rock in about equal amounts. Also, minor amounts of chalcopyrite were present. The carbonate gangue material of the veins show a definite dolomite appearance, but the material lacks the hardness of dolomite. Minor amounts of quartzite were noted intermingled with the carbonate gangue material. The specific gravity of the ore was determined by selecting a range of specimens from wall rock through the vein material. The determined specific gravity was found to be 3.67.

Microscopic Analysis. A grab sample of the ore was taken and examined with the aid of the binocular microscope to determine the minerals present. The dominant mineral

noted was pyrite. Other minerals of prominence were galena and sphalerite, with minor amounts of chalcopyrite and bornite. Geologic reports^{1*} state that minor amounts of tetrahedrite were present, but none was noted in any of the samples analyzed. Gangue minerals noted were dolomite, quartzite, rhodochrosite, ankerite, and sericite. The sulphied minerals ore all interlocked. The interlocking between the galena and sphalerite is especially complex. Chalcopyrite and bornite interlocked with galena in all cases. Pyrite is interlocked with all minerals present.

Preparation of Material for Flotation

Liberation Studies. The portion of ore present was cut down by riffle sampler and a head sample of 2510 grams removed. 2044 grams of this was taken for screen analysis, (Plate I) and the remaining 466 grams split into three portions for assay.

After the head sample was removed, the remainder of the ore was split by the aid of riffle sampler into 600 gram samples for further liberation and flotation studies.

Grinding is usually the final step in liberation. If all the particles in a grind are composed of one grain of each mineral, they are said to be free, and liberation is perfect; however, grinding to complete liberation is not always economically or technically feasible, and a compromise must be reached by grinding to a point over which the extra cost of further grinding will not be warranted by the increase in metallurgical improvement.

^{1*} All references are contained in the bibliography.

Taggart² states that without any preferential grain boundary cleavage, grinding a relatively rich ore to the predominate grain size of the valuable mineral will free a product in which at least 80 per cent of the coarsest grains are free gangue, and well over 90 per cent of all grains are free gangue, although only about 70 per cent of all the valuable grains would be free. Preferential boundary cleavages increase these percentages to such an extent, in the usual ores, as to justify the rule to grind to the prevailing grain size of the valuable mineral.

Particle Size Reduction with Time of Grinding. To determine the grinding time necessary to obtain an optimum degree of liberation, a 600 gram sample of ore was diluted with 400 cubic centimeters of water. A 60 per cent solids pulp was chosen as it gave a consistency similar to molasses in appearance and feel, which, according to Taggart² is desired in most tumbling mills. The liberation tests were performed in both laboratory pebble and rod mills, in order to obtain a comparison between the two grinding actions. The pebble mill was charged with 4500 grams of medium-sized pebbles, filling the pebble mill to slightly over half capacity. The rod mill was charged with 8819 grams of rods of assorted sizes, filling the rod mill to about 30 per cent capacity.

The first pebble mill grind test was made over a period of 10 minutes. Further tests were made for 20 and 30 minute periods. Upon completion of the grinding period,

the pulp was removed from the mill and the pebbles and mill thoroughly washed. The pulp was washed over a 200 mesh screen in order to deslime it. The deslimed material was dried and screened through the 35, 48, 65, 100, 150, and 200 mesh screens on the Tyler Rotap Sieve Shaker for a 15-minute period. The minus 200 mesh material from the desliming of the pulp was dried and combined with the minus 200 mesh material of the screening operation. The weight of the screen fractions was taken and recorded and the cumulative weight per cent calculated. (See Table II)

The 10-minute pebble mill grind test yielded 46.25 per cent of minus 200 mesh material. The 20-minute grind test yielded 66.27 per cent of minus 200 mesh material. The 30-minute grind test yielded 86.04 per cent of minus 200 mesh material. As the 30-minute grind test produced such a large amount of slime material plus the fact that this slime material had a tendency to flocculate and form a clay-like material, it was decided to proceed no further with the pebble mill grind tests.

The procedure for the rod mill grind tests was exactly the same as that outlined for the pebble mill tests. The results (Table II) show that the 10-minute grind test yielded 55.91 per cent of minus 200 mesh material, and the 20-minute grind test yielded 93.15 per cent of minus 200 mesh material. The rod mill tests were carried no further because of the excessive amount of slime material produced in the 20-minute grind test.

Briquetting and Grain Counting. Each screen fraction of the various grind tests were sampled and a two gram portion removed wherever possible. This sample was mixed with an equal volume of leucite. This mixture was poured into the briquetting cylinder of the briquetting machine and twelve grams of leucite added. The cylinder was placed in the machine and a pressure of 100 pounds applied. Heat was applied until the temperature reached 150°C. During this period, the pressure was maintained at 100 pounds. At this point the pressure was raised to 4,000 pounds, the heating element was removed, and the cooling plates set in place. When the temperature of the cylinder had been lowered to 80°C., the cooling plates were removed, and the pressure gradually released. After the pressure was entirely released, the cylinder was set in place under the jacking pin and the briquette removed from the cylinder with the aid of pressure.

When the briquette had been removed from the cylinder, it was first placed on the grinding wheel to remove the excess layer of leucite that had formed over the face of the specimen. The specimen was next ground with a mixture of 400 carborundum and water on a glass plate. This was done to remove the deep scratches formed by the grinding wheel. After this operation, the samples were ground and polished in the Graton-Vandawilt Polishing Machine, Model G. This grinding and polishing was supposed to be carried out in four stages: (1) Coarse grinding. (2) Fine grinding. (3) First polish. (4) Finish polish. Only the first three stages

could be carried out, however, as there was no available supply of abrasive for the finishing polish.

A cast iron lap wheel with abrasive No. K-1222, and kerosene as the lubricant, was used for the coarse grind. Each setting in the polishing machine was comprised of six briquettes. The briquettes are set in brass holders which are held in place on the lap wheel by a cast iron holder set on a guide post. The lap wheel was coated with a paste of K-1222 and kerosene, the briquettes set in place, and the machine set in motion, the lap wheel revolving clockwise, and the briquettes revolving in counter-clockwise direction. Kerosene was applied to the lap as needed through the aid of a drip regulator. The first two sets of briquettes were checked every five minutes as to the progress of the grinding and it was found that an average of forty minutes was sufficient to produce a good surface free of the deep scratches formed by grinding with the carborundum. The other five sets of briquettes were ground for thirty minutes before checking their surfaces and were ground for fifteen minutes more and removed. As they were removed from the machine, all briquettes were first washed in kerosene, rinsed with water, then washed in a soap solution, and rinsed with water. This procedure was done to remove any traces of abrasive that might adhere to the surface of the briquette.

After all briquettes were given a satisfactory coarse grind, they were subjected to the fine grinding. The cast iron lap wheel was coated with a paste of abrasive No. K-1225

and instrument oil. The change from kerosene to instrument oil as lubricant was made because the instrument oil is a slightly heavier base oil and thus has better lubricating properties than the kerosene. During the grinding period, the briquettes were checked every fifteen minutes as to their surface. The average overall grinding time for this stage was forty-five minutes. The washing procedure was the same as that outlined above in the coarse grinding procedure.

For the first polishing stage a lead faced lap wheel was used in place of the cast iron lap wheel. A paste of abrasive No. K-1232 and instrument oil was worked into the face of the lap wheel so that all grooves and scars were filled. This was done with the aid of two steel facing blocks. It was found during the grinding of the first set of briquettes that it was impossible to use the lubricant regulator to limit the amount of instrument oil to the lap wheel. It was set and re-set, in an attempt to get a satisfactory flow of oil, until the point was reached where just one more notch would shut the flow off entirely, but still the flow was too much. It was finally decided to add the oil to the wheel as needed, with the aid of a dropper. It is very necessary in this polishing stage to have just enough lubricant on the lap wheel to keep the briquettes from sticking to the lap wheel. An over supply of lubricant would destroy the effectiveness of the abrasive. Each set of briquettes was ground for forty minutes and removed and washed and the lap wheel cleaned and new abrasive added.

This procedure was repeated and after checking the surfaces, a further grinding period of one hour was given. The surfaces, at the end of this period, were carefully checked and found to be satisfactory for grain counting and calculation of the degree of liberation.

Determination of Degree of Liberation. Quantitative determination of degree of locking is obtained in one of two ways, depending upon whether the important minerals are transparent or opaque.

If the important minerals are transparent, they are viewed with a polarizing microscope as loose grains sprinkled on a slide with an oil of suitable index of refraction. If the minerals are opaque, they are viewed with a metallographic microscope equipped to use polarized light. As the mineral grains in this case are opaque and are mounted in leucite briquettes, the latter procedure was used.

Identification of opaque grains involves the use of three criteria: color, relief, and behavior under polarized light.

Mineral particles are classified into free particles, binary particles, and ternary products. Under heading of free particles, are recorded the particles which appear free; under heading of binary particles, are recorded those which appear to consist of two mineral species only, and under heading of ternary particles, those which appear to be composed of three or more mineral species are recorded.

From the results obtained, it is possible to calculate,

not only an ultimate chemical analysis, but also, and this is the important point, the degree of association of minerals to form locked particles and the extent to which each one of the minerals contributes to each particular type of locked particle.

Although a free particle cannot be cut so as to appear to be locked, a locked particle may be cut so as to appear to be free. It follows that the observations made under the microscope yield a result that is overstating the liberation of the material under consideration. To correct this error, inherent in the method of examining polished surfaces of briquetted mill products, it is necessary to increase the number of particles actually observed by a locking factor. According to Gaudin, as a first approximation, adoption of a single locking factor of about 1.3 is reasonable.

In order to conserve time, it was decided not to consider the rod mill grind test because of the excessive amount of slimes produced. The 30-minute pebble grind test was disregarded for the same reason. This left the 10 and 20-minute pebble mill grind for examination as to the degree of liberation. The results are recorded in Plate III.

The results of the liberation studies show that the degree of liberation for the 10-minute grind was 76.6 without applying the locking factor, and 71.6 when the locking factor was applied. The results show that the degree of liberation of the galena was 63.4 without applying the

locking factor; for the sphalerite it is 63.6, and by applying the locking factor, the degree of liberation for the galena was 57.1, and for the sphalerite, it was 57.5. The results for the 20-minute grind give a degree of liberation of 82.2 without applying the locking factor, and, by applying the locking factor, the degree of liberation is 18.0. The degree of liberation for the galena in the former case is 74.8, and for the sphalerite, is 51.5 and, in the latter case, for the galena it is 69.6, and for the sphalerite, it is 44.2.

A thorough examination of the degree of liberation studies show that liberation of the sphalerite is not as complete as it might be, in that the degree of liberation is in the general neighborhood of 50. This means that 50 per cent of the sphalerite is combined with other minerals. This will affect the floatability of the sphalerite to a certain extent, and will contribute to low grade concentrate. The results of the galena show a degree of liberation of about 70, which is about as good as can be expected in an ore of this type. It is to be noted that the pyrite has a high degree of liberation, about 80. This will tend to increase the tendency of the pyrite to float, in that the concentration of pyrite in this ore is exceptionally high.

Flotation

Froth Flotation. Flotation is a method of wet concentration of ores in which the separation of mineral from gangue is effected by causing the mineral to float to the

surface of a liquid pulp, while the gangue remains submerged.

The essential condition for froth flotation is that a particle of mineral, already completely wetted by water, should become attached to an air bubble large enough to buoy it to the surface. In practice, the particles are so small that many of them become attached to each of the air bubbles, which are described as being mineralized, or "armoured" by the particles. Often a bubble becomes so highly mineralized that it sinks to the bottom. If the system is shaken, however, the bubbles may lose some particles and rise slowly to the surface.

The basis on which the separation of the various mineral species is accomplished is the alteration of the mineral particles by suitable reagents. Certain of these reagents selectively change the surfaces of the minerals in a pulp, making the desired mineral particles water repellent and effect selective adherence of these minerals to the froth bubbles.

There are a number of factors on which flotation is dependent. Some of these are definitely known; many others are still a matter of scientific conjecture and problems for research. It is not within the scope of this thesis, however, to deal with such matters, but rather to apply known flotation fundamentals and reagents to bring about the selective separation of the minerals in the ore.

Experimental Factors in the Separation of Minerals. No two mineral crudes react toward flotation in the same manner.

Each must be determined experimentally according to the constituent minerals within the sample. Gaudin³ states that success in flotation is predicated on establishing a water-repellent or air-adhering surface on some minerals, whereas others are prevented from acquiring it.

The specific problem of this thesis is the separation of galena and sphalerite. The general procedure is to carry the flotation out in two steps:

1. Float the galena while depressing the sphalerite and pyrite.
2. Float the sphalerite while inhibiting the pyrite.

For the successful fulfillment of the problem, a judicious selection of reagents is demanded. Four general methods have been proposed to separate galena from sphalerite. These methods are based on the use of: (a) cyanide with or without zinc sulfate; (b) sodium sulphide; (c) a soluble sulfite or sulfurous acid with or without zinc sulfate; and (d) a chromate or dichromate with or without copper sulfate. Of these methods, the first three aim at permitting the galena to float while inhibiting the sphalerite. The last method is aimed at inhibiting the galena while floating the sphalerite. The cyanide-zinc sulfate method is most widely used. This system was used in this thesis as it has the effect of depressing the pyrite, of which there is an overabundance, as well as the sphalerite.

The separation of galena from sphalerite by floating sphalerite away from galena is more difficult than the usual

separation in other direction, but it is not impossible. Although there seems to be little need at present for a method to float sphalerite ahead of galena, experimental data suggests the possibility of cleaning lead and zinc concentrates by reversing the order of flotation used in roughing.

The problem of separating galena from sphalerite is largely a structural problem because grinding is not always conducted to insure liberation of the sulphides from each other to the most economical point. It was noted in the ore used in this thesis that there was a tendency for both sphalerite and galena to be interlocked with the carbonate material. This leads to a somewhat low grade concentrate. Even if perfect liberation is attained and reagents are used properly, it must be kept in mind that it is not possible to separate galena and sphalerite as perfectly as it would be to separate galena from an equivalent amount of silicate gangue.

The prevention of iron reporting in the lead concentrate is relatively unimportant as iron is needed for lead smelting and is credited in many lead-smelter settlements, however, in the case of the Mike Horse ore, it was found that it was important to try to inhibit the pyrite as much as possible in order to obtain a high grade concentrate.

The elimination of iron from zinc concentrates is desired for the following reasons:

1. If iron is present in a large amount in zinc concentrate, it may result in the formation of ferrite (zinc oxide-iron oxide compound) which will reduce

the amount of zinc extracted as the ferrite is insoluble.

2. Iron in zinc concentrate increases the buld of the sludge from the leaching tanks, and thereby dilutes the valuable constituents (largely lead, gold and silver) contained in the sludge.
3. Zinc concentrates being products of relatively low value, dilution even with a harmless constituent, is objectionable in that it increases the freight-charge burden to be carried by the recoverable zinc in the concentrate.

It is theoretically possible to make a perfect separation between sphalerite and pyrite. In the case of the Mike Horse ore this separation presents a difficult problem in that pyrite rather resists depression and readily floats.

Flotation Reagents

Collecting Agents. Collecting agents exert their action by adhering to the surface of certain mineral particles, either in their natural state or in some altered chemical form, forming there a non-polar coating that repels water but attaches itself readily to gas bubbles or to non-polar mediums. The use of collecting agents having different reactivities with different minerals is of fundamental importance in selective flotation.

Collecting agents may be classed in two varieties: those that form definite compounds by metathesis with the

surface of the mineral, and those that do not seem to do so. The xanthates are of the former type, and are among the most commonly used collecting agents. It was decided to use potassium ethyl xanthate as a collecting agent for the galena in the ore, as there is much evidence from previous experiments on flotation of galena as to its adaptability. It has been shown that in the case of galena there is metathesis between oxidized compounds on the mineral surface and the dissolved xanthate. The inference is drawn that the same mechanism accounts for the action of xanthates on other sulphides, but evidence is lacking.

The evidence that galena is floated by the formation of coatings of lead xanthate in place of oxidized patches is as follows:

1. Galena oxidizes readily until a thin skin of basic lead sulfate and lead carbonate covers it more or less completely.
2. Lead sulfate, basic lead sulfate, and lead carbonate react with alkali xanthates, forming lead xanthates, which are much less soluble than the lead sulfates or lead carbonates.
3. Slightly oxidized galena abstracts xanthate ions from solution and yields sulfate, hydroxyl and carbonate ions to an approximately equivalent amount.
4. Slightly oxidized galena treated with an alkali xanthate and leached with a solvent for lead xanthate, yields lead xanthate.

The collecting agents used for the sphalerite was also potassium ethyl xanthate for the first few tests. Kopper's reagent B-140 was supplemented. Very little is known about this reagent as the company is very reluctant to release any information regarding its composition.

Frothing Agents. Froth flotation,³ which consists of buoying solids heavier than water to the surface of the pulp by means of gas bubbles utilizes the force of surface tension to attach the solid particles to the gas bubbles and to maintain the floating mass of bubbles as a more or less persistent froth at the surface of the pulp for a sufficient length of time to allow its removal. Liquids foam only if composed of several substances which, if pure, have different surface tensions. Upon examining Gibbs' equation, it is noted that it takes a large amount of a substance raising the surface of a liquid to materially change the surface tension, but that small amounts of a substance lowering the surface tension may be sufficient to bring about a drop in surface tension equivalent to the rise obtained otherwise. This explains (on economic grounds) why all frothing agents used in practice are substances that lower the surface tension sufficiently, even if used in very small amounts. Almost all organic substances lower the surface tension of water, sometimes amazingly in regard to the amount added.

The extent to which substances that lower the surface tension of water affect the surface tension of water is closely related to their structure. Frothing agents having

structural formulas are characterized by the presence of two constituents having opposite properties: One part of the molecule is polar and the other is non-polar; that is, one part repels water, while the other hydrates. Frothing properties are related to the composition and structure of both parts of the molecule. As a result of the addition of a frother, the gas bubbles formed under the surface of a liquid are more or less completely lined with a monomolecular sheath of frother molecules which allows each bubble with its lining to come in contact with other bubbles without coalescing and this forms a froth. In a measure, the effect produced by addition of a frother is proportional to the amount of frother. Past a certain point, the effect of further additions of frother is less than that of preceding additions; eventually, the addition of further amounts results in a decrease of the frothing action, and finally in the total absence of frothing.

In practice, the most widely used frothers are pine oil and cresylic acid. Pine oil was used as the frothing agent for the flotation tests in this thesis. It was noted that Kooper's reagent B-140 also had frothing properties, as well as collecting properties.

Activating Agents. Activating agents³ are those which, through their effect upon the surface of otherwise non-floatable or poorly floatable minerals, make them amenable to flotation with the usual collecting agents.

As sphalerite in its natural state is not very amenable

to flotation, copper sulphate was added to the pulp as the activating agent. The copper sulfate changes the surface of sphalerite to covellite which is readily floated through the use of a collecting agent.

Depressing Agents. Depressing agents is the term reserved for reagents that form upon the mineral surface a coating which is less floatable than the natural mineral surface.

Cyanide acts as a depressing agent for pyrite, but the exact way in which it operates is not certain, and is open to study and research. It has been reported by students of opaque minerals that there is no action on pyrite that is manifested by a change in color or by etching even upon treatment with 20 per cent KCN solution.

pH Regulating Agents. pH regulating agents are used to effect hydrogen ion concentration of the pulp. Different ores give best results at different acidities or alkalinities. For each flotation operation, there is a definite pH range for good results.

For this investigation it was found necessary to have a pH range of 10.5 to 11 to float the lead and keep the pyrite depressed. This range was maintained by the use of sodium hydroxide. For the flotation of sphalerite the best operating pH range was found to be 8 to 8.5. This pH range was obtained by addition of hydrochloric acid. Sulfuric acid has a tendency to clean the pyrite surface, thus making it more adaptable to flotation.

Laboratory Procedure

Head Assay. Before any flotation tests were started, an assay was made of the head sample. This assay was as follows:

Lead - 4.4 per cent
Zinc - 3.2 per cent
Copper - 0.18 per cent
Silver - 1.702 per ton
Gold - 0.00502 per ton

Upon the basis of this assay, it was decided not to bother with a pyrite separation, as there was insufficient gold value in the pyrite to warrant its attempted recovery.

Flotation Tests. Flotation tests were conducted with 600 gram samples of minus 20 mesh ore. The ore was diluted with 400 cubic centimeters of distilled water, making a 60 per cent pulp. The pH of the distilled water averaged 7 for the series of tests. The ore sample, the water, and the various reagents; namely, sodium hydroxide, sodium cyanide, and zinc sulphate, were placed in a laboratory pebble mill with 4,500 grams of pebbles and ground for a 20-minute period. At the termination of the grinding period, the mill was emptied and thoroughly washed with distilled water. The pebbles were emptied out on a coarse screen and thoroughly washed with distilled water. The pulp from this washing operation was transferred to Flagergren Laboratory flotation cell. The pulp density of the Flagergren cell was maintained at about 23 per cent. This called for an addition of 1,700

cubic centimeters of distilled water to the original pulp from the mill. After a few trial tests, it was found possible to wash the mill and the pebbles thoroughly with approximately the amount of distilled water necessary to bring the pulp density of the cell to meet the standard requirements.

The first operation after the pulp had transferred to the flotation cell was the removal of a sampler of the solution and testing with the Beckman pH machine to obtain the pH of the original solution. As soon as the pH was recorded, the sample was returned to the cell and the conditioning agent, potassium ethyl xanthate, added. After the pulp had been conditioned for a definite period, the frothing agent, pine oil, was added and the galena laden froth removed. The next operation was to modify the pH by addition of an acid in order to have the pulp within the pH range necessary to obtain optimum recovery of the sphalerite. This was followed by the addition of an activating agent, copper sulfate, for sphalerite.

The pulp was allowed to condition for a definite period of time, and then a conditioning agent, potassium ethyle xanthate for the first series of tests, was added, later supplanting it by Kooper's reagent B-140. The pulp was again allowed to condition for a definite period of time and then the air was admitted and the sphalerite laden pulp was removed. As soon as results warranted it, the lead concentrate was submitted to a cleaning operation. The tails

from this operation were added to the zinc concentrate. This was done in order to obtain a pulp that would not be so dilute as to make flotation almost infeasible.

All concentrates and tails were first filtered, then dried and weighed. In order to keep operating expenses to a minimum, it was deemed necessary to assay only those products from operations that from observation seem to present the best results obtainable.

Discussion of Flotations Tests

Test No. 1. This test was a bulk flotation test on the ore to determine its behavior in the flotation machine. Reagents were added as shown in the accompanying data sheets. Lime, two pounds per ton, sodium cyanide, one pound per ton, and zinc sulphate, one pound per ton, were added to the pebble mill charge. The pH of the pulp was 10. Potassium ethyl xanthate, 0.2 of a pound per ton, was added and the pulp allowed to condition for 3 minutes. At the end of this period, one drop of pine oil was added with an eye dropper and the valve was opened and the galena laden froth removed. The froth was very friable and the bubbles very small. This evidence showed that there was too great a quantity of frother added. The concentrate was heavy in galena and pyrite, showing the pyrite was hard to depress, while the galena floated rather easily. The frothing period was continued for 3 minutes and then the air valve was closed. Copper sulfate, 0.15 pound per ton, was added and conditioning continued for 3 minutes. At the end of this period, the air

valve was opened and the sphalerite laden froth removed. An excellent froth was obtained with no additional frother being added. The concentrate proved to have a fair amount of sphalerite with considerable amounts of pyrite and carbonate coming off with the froth. The amount of the various reagents added, the conditioning times, the weight of the various products, and their assay are recorded on the accompanying sheets.

Test No. 2. In light of the floatibility of the pyrite in the previous test, it was decided to increase the lime to try to raise the pH in order to check the effect of a change in pH. The lime was increased to 4 pounds per ton and the other reagents were retained in the same amounts as in the previous test. The same grinding procedure as noted above was followed. The pH of the pulp was 10.0. The conditioning time for the potassium ethyl xanthate was again 3 minutes and then 2 drops of pine oil were added with the aid of a hypodermic syringe. The froth was again very friable and heavy. It was also fairly heavy with galena and pyrite. The frothing time was held to 3 minutes. The conditioning time for the copper sulfate and for the potassium ethyl xanthate was three minutes each. The froth was fairly good, in that the bubbles were of about the right size, but were not too persistent. The frothing time was continued for about 2 minutes. The froth was laden with sphalerite, also with pyrite, with some carbonate being carried over. The pH at the end of the test was 9.5. The products were filtered,

dried, weighed, and recorded. No assays were made.

Test No. 3. In light of results of the previous test, it was decided to increase the amount of lime to 6 pounds per ton and increase the potassium ethyl xanthate used as a collecting agent for the lead to 0.3 pound per ton, while decreasing the amount used for a collecting agent on the sphalerite to 0.1 pound per ton. The conditioning time for the galena was maintained at 3 minutes. The conditioning time for the copper sulfate was held at 3 minutes, and conditioning time for the potassium ethyl xanthate was also held at 3 minutes. The sphalerite froth was rather light. The concentrate was fairly heavy with sphalerite and pyrite. The amount of carbonate was rather limited. The frothing period was held for 2 minutes. The pH at the end of the sphalerite froth was 8.9. The products were filtered, dried, and weighed. No assays were made of this test.

Test No. 4. For this test the amount of lime was raised to 7 pounds per ton to try to raise the pH to 11 or over. The amount of potassium ethyl xanthate for collecting the galena was reduced to 0.1 pound per ton. All other reagents were maintained in the same amounts. The pH for the lead float was 10.2. The conditioning period for the galena was 3 minutes. One drop of pine oil was added. A good froth was obtained and the concentrate was heavy in lead and the amount of pyrite was not nearly as much as in the previous tests. The conditioning times for copper sulfate and potassium ethyl xanthate were maintained at 3 minutes each. The froth was rather light for the sphalerite

float. The concentrate contained sphalerite and pyrite with some carbonate present. The products were all filtered, dried, and weighed. No assays were made.

Test No. 5. For this test, the amount of lime was increased to 8 pounds per ton, the sodium cyanide was increased to 1.5 pounds, zinc sulfate was increased to 1.5 pounds, and all others reagents were maintained in the same quantity as previously. The noted pH of the pulp before the lead float was 10.2. An exceptionally good lead froth was obtained and the pyrite content seemed to be rather low in comparison to the other tests. One drop of pine oil was added for the sphalerite froth to try to obtain a more persistent froth than had been obtained in the last tests. The froth was an improvement over that of the previous test. The concentrate seemed to be fairly heavy, also pyrite. The carbonate was fairly heavy also. The pH at the end of the sphalerite flotation was found to be 8.85. The products were filtered, dried, and weighed. Assays were made of all products. The lead concentrate proved to be low in lead, with considerable sphalerite coming over with it, and the same for iron. The zinc concentrate proved to be very low in zinc and rather high in lead and iron. The tails were rather clean of lead, but contained altogether too much zinc.

Test No. 6. In all the above tests, the increasing of the lime content from 2 pounds to 8 pounds per ton had little if any effect in changing pH. Evidence seemed to point to a buffer action being set up by the ore. It was decided to

change to sodium hydroxide for a pH modifying agent. It was also decided to cut the amount of zinc sulfate to 0.5 pound and to reduce the amount of sodium cyanide to one pound per ton. It was also decided to try Kooper's reagent B-140 as a collecting reagent for sphalerite. Potassium ethyl xanthate was retained as a collecting agent for galena, pine oil was retained as frothing agent for the lead, and copper sulfate was retained as activating agent for the sphalerite. The pH for the galena float was 10.7. The conditioning time for the lead was maintained at 3 minutes. One drop of pine oil was added, and a fairly heavy froth was obtained. The froth was heavy with galena and pyrite. 0.6 pound of copper sulfate was added and the conditioning time was maintained at 3 minutes. 0.2 pound of B-140 was added and conditioned for 2 minutes. A very heavy froth was obtained carrying over considerable pyrite and carbonate, thus cutting down the concentration of sphalerite considerably. The pH at the end of the flotation period was 9.2. The products were filtered, dried, and weighed. No assays were made.

Test No. 7. For this test the amount of sodium hydroxide was set at 2.5 pounds and the zinc sulfate was set at one pound per ton. All other reagents were maintained in the same quantities. Conditioning times were also held the same as in the above test. The froth for the sphalerite was again too heavy carrying over considerable pyrite and carbonate. All products were filtered, dried, and weighed. No assays were made.

Test No. 8. For this test the amount of sodium hydroxide was set at 3 pounds per ton. The amount of zinc sulfate was set at 0.5 pound per ton. The reason the zinc sulfate has been varied from test to test was to see if any noticeable effect could be obtained in the depressing of the pyrite, but as the variation seemed to make little if any change, it was decided to cut it back to the lowest possible point and leave it there for the time being and concentrate on trying to effect a change in the pH to above 11 in order to see if any improvement could be obtained in depressing the pyrite in that range. The amount of B-140 added for this test was reduced to 0.1 pound. This was done in order to try to improve the froth for the sphalerite, as it had been too heavy in the last two tests and carrying over too much pyrite and carbonate. The lead concentrate, on examination, was noted to be about the same as in all previous tests. The pH at the start of the flotation was 10.8. The froth for the sphalerite floating was still too heavy, but not as heavy as in the previous tests. All products were treated as outlined above and no assays were made.

Test No. 9. For this test the sodium hydroxide content was increased to 4 pounds per ton. All other reagents were maintained in the same amounts as outlined in the previous test. An exceptionally good galena concentrate was obtained, although the pyrite content was observed to be about the same as noted in above tests. The pH for the lead flotation was 10.8. The froth for the sphalerite flotation was not

as heavy as in previous tests, but still carried considerable pyrite and carbonate. The products were all filtered, dried, weighed, and assayed. The assay showed the lead concentrate to be slightly improved as far as the lead content was concerned, but still carrying too much zinc and iron. The zinc concentrate was much lower in lead content than the previous assay showed, but the zinc content was very low with the iron remaining about the same as in previous tests assayed. The tails proved to be fairly clean of lead, but still too high in zinc. At this point, it was noted that the two big problems presented by this ore are the finding of a method of depressing the pyrite and more important, a method of floating the sphalerite which seems to be in two places, the lead concentrate and the tails, rather than the zinc concentrate. The reasons for this odd behavior are hoped to be found before this thesis is completed.

Test No. 10. For this test the sodium hydroxide content was increased to 6 pounds per ton and the amount of B-140 was decreased to 0.05 pounds per ton. The pH of the pulp taken just before the galena float was pulled was 11.5. A good lead froth was obtained and on examination the pyrite content was noted to be slightly less than that of previous tests. A very good froth was obtained for the sphalerite float; however, the pyrite and carbonate content was still too high. The pH for the sphalerite test was 10.0. Four drops of sulfuric acid were added to the pulp after the lead float was pulled to modify the pH of pulp to a range for

sphalerite flotation. After the addition, it was found that this acid has a cleaning effect on the surface of the pyrite which makes it amenable to flotation. It was decided to use hydrochloric acid as a pH modifier in all future tests.

Test No. 11. The results of the previous test showed that a fair lead roughing concentrate was being obtained and that the main contaminating mineral in the zinc float was the carbonate mixture, also that most of the sphalerite was floating over with the galena. On the basis of the above observations, it was decided to try a cleaning operation on the roughing concentrates of both galena and sphalerite. For this operation all reagents were retained in the same quantities as in the previous test. The pH for the lead rougher float was 11.2. Five drops of hydrochloric acid were added as modifying agent and the pH for the sphalerite flotation was 9.8. A separate cleaning operation was made on both the lead and sphalerite rougher concentrates. A fair froth was obtained on the lead concentrate and a good lead cleaner concentrate was obtained and the pyrite content could hardly be detected by visual observation. A fair froth was obtained on the sphalerite cleaner flotation. The pyrite content was noticeably lower and the carbonate content was also noticeably lower. The flotation product was cut to minimum due to the fact that the pulp was too dilute to obtain a good flotation product. For previous cleaning operation, it was decided to clean the lead rougher concentrate and add the tail from this operation to the zinc rougher concentrate in order that a pulp density that is more amenable to a cleaning operation

could be obtained.

Test No. 12. For this test all reagents were retained in the same amount as in the two previous tests, except that five drops of hydrochloric acid were added. The pH for the lead rougher concentrate was 11.2 and the pH for the zinc rougher flotation was 9.8. The pH for the lead cleaner flotation was 10.6 and for the zinc cleaner float, was 9.4. A good lead cleaner concentrate, rather free from pyrite, was obtained. The froth for the zinc cleaner flotation was rather poor, in that it was not too persistent. It was noted that a noticeable amount of galena came over with the sphalerite froth. This shows that the lead rougher concentrate is not being satisfactorily cleaned of galena.

Test No. 13. The amount of reagents used for this test was the same as in the two previous tests, except that six drops of hydrochloric acid were added to modify the pH for the sphalerite rougher float. During the lead cleaner flotation period, it was noted that there was a tendency for a break in froth to form about 90° from the center of the tip on both sides of the cell. This break seemed to be due to a suction action, and there was a tendency for galena particles to collect at these two points. The galena collecting at these two points was being retained and was coming over with the sphalerite froth. There seems to be no feasible explanation for this phenomenon, as it did not occur with either the lead or zinc rougher froths, and the tendency was only slightly noticeable in the zinc cleaner froth.

Assays were made on the products and it was noted that a lead concentrate was almost up to a shipping grade and the zinc and iron content had been reduced considerably but was still not up to the standards required by most custom smelters. The zinc cleaner concentrate contained altogether too much lead, but even if this lead content were reduced to nil the grade of the concentrate, it still would not be anywhere near a shipping grade. The sphalerite is not being sufficiently cleaned from the tails and too much is remaining with the lead concentrate. These results point out that a major part of this problem lies in trying to make the sphalerite react in a manner that will enable flotation. To try to effect the sphalerite in some manner, so that it will be amenable to flotation, will be the major interest of all future tests and effort must be put forth to improve the grade of the lead concentrate and to cut the amount of iron to a minimum in both the lead and zinc concentrates.

Test No. 14. For this test eight drops of hydrochloric acid were added to the pulp after the galena float was pulled to modify the pH. The pH for the lead rougher froth was 11.5 and the pH for the zinc rougher froth was 9.2. .25 pound per ton of sodium cyanide was added to the pulp and a conditioning time of 2 minutes set for the pulp. Sodium cyanide is commonly used as a depressing agent for pyrite so it was decided to make this addition in order to try to get the lead concentrate as free from iron as possible. The pH for the lead cleaner froth was 10.6. .025 pound of

B-140 was added to pulp for the zinc flotation in order to obtain a more persistent froth and, more important, to try to improve the collection of sphalerite, thus improving the grade of the zinc concentrate by cleaning the tails to a better degree than in previous tests. The pH for the zinc float was 9.6. The same difficulty was met with the lead cleaner froth as was noted in the previous test and no amount of manipulation could correct this difficulty.

Test No. 15. The reagents and their amounts were the same as those used in the previous test. The pH for the lead rougher froth was 11.4. The froth was exceptionally heavy. At such a high pH range it has been noted that the froth is very hard to control. There is a tendency to overfroth at the slightest provocation. The same tendency of the lead cleaner froth to form a break after about 2 minutes of the frothing period was noted. The lead cleaning frothing period was increased to 5 minutes, but that had no effect on picking up the lead that had been dropped by this break in the froth. This lead was first to float over with the sphalerite froth.

Test No. 16. The only factor changed for this test was that ten drops of hydrochloric acid were added to the sphalerite pulp. This lowered the pH to 8.2. The observation showed this test to be no different than the previous test in the results obtained.

Test No. 17. This test was run in the same manner as the previous test, with the same results being obtained. The products were assayed to see if any answer to the problems

obtained could be found by analyzing these results. The results showed that the lead was being cleaned within economic limits from the tailings, but that 30.8 of the zinc concentrate being lead, this cut the grade of the lead concentrate to a point that was too low for a shipping grade. Most custom smelters have set a minimum of 70 per cent, with a penalty applied to any grade below this. It was also noted that the tailings obtained contained 1.7 per cent zinc. It seems that no amount of manipulation can entice this last sphalerite to float. It was also noted that the lead concentrate contained 8.3 per cent zinc. As to why this zinc floats over with the lead while a major portion refuses to float at all cannot be understood at this time.

Four more similar tests were made, but no difference in results were noted, so it was decided to discard the results.

As noted previously, the sample of the ore had been crashed to a minus 20 mesh over two years previous. Gaudin, Wark, and Taggart state that for flotation tests and commercial flotation the ore should be as fresh as possible, that is, it should be brought from the mine to the laboratory or mill as soon as possible in order to obtain indicative flotation results. On the basis of this statement and the fact that no exact conclusion could be drawn from the tests made, it was decided to forego any further test and to try to obtain a fresh sample from the mine. As a new sample of ore could not be obtained within the time allotted for completion of this problem, no further test could be made.

Flotation Summary

From the preliminary flotation tests, the following observations were made:

1. Maintaining a strong alkaline circuit produced no satisfactory degree of selectivity as the sphalerite and pyrite floated over rather readily with the lead rougher concentrate.
2. The pyrite was depressed to a more satisfactory degree in the pH range of 10 to 11.5 than from 8 to 10.
3. That galena floated rather readily in the pH range of 8 to 11.5. The froth, however, was difficult to control in the range of 10.5 to 11.
4. The selective flotation of sphalerite proved to be a difficult problem and is open to much study and research. The amount floating over with the lead seemed to remain fairly constant throughout and resisted all efforts to depress it in order that it could be floated in its proper place. The sphalerite remaining in the tailings resisted, in much the same manner, all efforts to collect it for flotation.
5. Kooper's reagent proved to be slightly more effective than potassium ethyl xanthe as a collecting agent for the sphalerite than did the potassium ethyl xanthate. It contains a frothing property as well as acting as a collecting agent, which is an additional advantage

6. Delicate control of the amount of B-140 must be taken or an over frothing action is readily obtained.
7. The pH range for the optimum recovery of lead showed to be 10.5 to 9. The pyrite was found to be more difficult to depress in this latter range, however.

Recommendations for Future Work

To aid future work on the problem of separation of lead and zinc from galena-sphalerite-pyrite ore by froth flotation, the following recommendations are made:

1. It should be ascertained that the ore sample is as fresh as possible.
2. A study of pH of the ore pulp by grinding with distilled water and no reagents would eliminate, to a certain extent, the trial and error method of selecting the amount of reagent necessary to attain a certain pH range for the flotation of the various products.
3. The pulling of a bulk concentrate of lead and zinc by adding copper sulfate and conditioning for 5 to 10 minutes and then adding Kooper's reagent B-140 and conditioning for 5 to 8 minutes and then floating a bulk concentrate, might be attempted. The pH for this flotation period should be 10 to 11. To this bulk concentrate, add sodium cyanide and zinc sulfate as depressing agents for the sphalerite and pyrite and condition for 8 to 10 minutes. At the completion of the conditioning period, add potassium ethyl

xanthate as conditioning agent for the galena and condition for 3 to 5 minutes. After this conditioning period, it may or may not be necessary to add a frothing agent in order to float the lead, as the B-140 has very good frothing properties and addition of additional frother might cause over frothing or loss of frothing action. After removal of the lead concentrate, it would be necessary to add a modifying agent in order to lower the pH range to between 8 and 9. The addition of copper sulfate as an activating agent would follow with a conditioning time of 3 to 5 minutes, following this with an addition of B-140 and conditioning for 3 to 5 minutes, and then pulling the sphalerite concentrate. The tails could be discarded or cleaned and the product obtained added to the bulk concentrate. This operation would depend on lead and zinc content of this product. An operation such as this would eliminate the carrying of a circulating load of pyrite, which is an undesirable product in an ore of this type, as is carried in the method outlined in this thesis.

4. The copper sulfate, acting as an activating agent for the sphalerite, should be given a sufficiently long period of conditioning in order to assure the activation of all the sphalerite. Also, the conditioning time for the collecting agent should be lengthened to a point where the operation can be

ascertained that all of the sphalerite particles have been acted upon. The length of these periods can only be found by trial and error, this is where the art enters into a mineral dressing operation.

5. Flotation tests using xanthates with a long chain might prove enlightening for the collection of sphalerite.

Conclusions

In the writer's opinion, the following facts have been ascertained from this thesis:

1. That the lead can be readily floated within a pH range of 11 to 11.5, and the pyrite is depressed more readily in this range than in a range from 8 to 10. Careful control is, however, necessary in order to obtain a good frothing condition.
2. That a higher degree of recovery of sphalerite could be obtained than was obtained in this study if sufficiently longer periods of conditioning for the activating and conditioning are used. Careful control must be maintained over the amount of reagents used and the pH range over which the collection of the sphalerite is effected.

BIBLIOGRAPHY

1. Thomas K. Shea, "Geology and Mines of the Upper Blackfoot Valley," Montana
2. Taggart, A. F. "Handbook of Mineral Dressing" - 1947
3. Gaudin, A. M. "Flotation" - 1932
4. Gaudin, A. M. "Principles of Mineral Dressing" - 1939
5. Wark, I. W. "Principles of Flotation" - 1938
6. The Dow Chemical Company, "Flotation Fundamentals"
7. U. S. Geological Survey Bulletin 842

A P P E N D I X

TABLE I

SCREEN ANALYSIS - HEAD SAMPLE

| Mesh | Weight (grams) | Per Cent Weight | Cumulative Weight Per Cent |
|------|-------------------|--------------------|-------------------------------|
| -20 | 372.8 | 18.28 | 18.28 |
| -28 | 393.5 | 19.29 | 37.57 |
| -35 | 307.2 | 15.07 | 52.64 |
| -48 | 242.2 | 11.87 | 64.51 |
| -65 | 185.9 | 9.12 | 73.63 |
| -100 | 133.0 | 6.51 | 80.14 |
| -150 | 95.9 | 4.71 | 84.85 |
| -200 | <u>308.9</u> | <u>15.15</u> | 100.00 |
| | 2039.4 | 100.00 | |

TABLE II
PARTICLE SIZE REDUCTION
WITH TIME OF GRINDING

PEBBLE MILL

Ten-Minute Grind

| Mesh | Weight (grams) | Per Cent | Cumulative Weight Per Cent |
|------|-------------------|--------------|-------------------------------|
| -28 | 14.9 | 2.48 | 2.48 |
| -35 | 25.2 | 4.21 | 6.69 |
| -48 | 56.2 | 9.48 | 16.17 |
| -65 | 89.6 | 14.95 | 31.12 |
| -100 | 75.7 | 12.63 | 43.75 |
| -150 | 60.0 | 10.00 | 53.75 |
| -200 | <u>277.0</u> | <u>46.25</u> | 100.00 |
| | 599.2 | 100.00 | |

Twenty-Minute Grind

| Mesh | Weight (grams) | Per Cent | Cumulative Weight Per Cent |
|------|-------------------|--------------|-------------------------------|
| -28 | 2.0 | 0.33 | 0.33 |
| -35 | 2.0 | 0.33 | 0.66 |
| -48 | 6.4 | 1.07 | 1.73 |
| -65 | 29.0 | 4.84 | 6.57 |
| -100 | 74.4 | 12.41 | 18.98 |
| -150 | 88.5 | 14.75 | 33.73 |
| -200 | <u>397.5</u> | <u>66.27</u> | 100.00 |
| | 599.8 | 100.00 | |

TABLE II

PARTICLE SIZE REDUCTION
WITH TIME OF GRINDING

PEBBLE MILL

Thirty-Minute Grind

| Mesh | Weight (grams) | Per Cent | Cumulative Weight Per Cent |
|------|-------------------|--------------|-------------------------------|
| -28 | 0.8 | 0.13 | 0.13 |
| -35 | 0.5 | 0.08 | 0.21 |
| -48 | 0.7 | 0.12 | 0.33 |
| -65 | 3.2 | 0.54 | 0.87 |
| -100 | 19.1 | 3.19 | 4.06 |
| -150 | 59.2 | 9.90 | 13.96 |
| -200 | <u>514.5</u> | <u>86.04</u> | 100.00 |
| | 598.0 | 100.00 | |

TABLE III

PARTICLE SIZE REDUCTION
WITH TIME OF GRINDING

ROD MILL

Ten-Minute Grind

| Mesh | Weight | Per Cent | Cumulative Weight Per Cent |
|------|--------------|--------------|-------------------------------|
| -28 | 0.2 | 0.03 | 0.03 |
| -35 | 0.3 | 0.05 | 0.08 |
| -48 | 2.4 | 0.41 | 0.49 |
| -65 | 52.7 | 8.85 | 9.34 |
| -100 | 119.6 | 20.07 | 29.41 |
| -150 | 87.5 | 14.68 | 44.09 |
| -200 | <u>333.9</u> | <u>55.91</u> | 100.00 |
| | 596.6 | 100.00 | |

Twenty-Minute Grind

| Mesh | Weight | Per Cent | Cumulative Weight Per Cent |
|------|--------------|--------------|-------------------------------|
| -28 | 0.00 | 0.00 | 0.00 |
| -35 | 0.1 | 0.01 | 0.01 |
| -48 | 0.1 | 0.01 | 0.02 |
| -65 | 0.1 | 0.01 | 0.03 |
| -100 | 6.1 | 1.02 | 1.05 |
| -150 | 34.8 | 5.80 | 6.85 |
| -200 | <u>558.9</u> | <u>93.15</u> | 100.00 |
| | 600.0 | 100.00 | |

Test NO. 1

Mineral Dressing Results

| Product | Per cent Weight | Assays | | | Per cent Distribution | | |
|----------|-----------------|--------|------|------|-----------------------|------|------|
| | | Pb | Zn | Ag | Pb | Zn | Ag |
| Pb Conc. | 15.0 | 39.7 | 15.0 | 15.4 | 90.9 | 63.0 | 86.5 |
| Zn Conc. | 1.4 | 36.6 | 4.8 | 14.9 | 7.8 | 1.8 | 7.8 |
| Tails | 83.6 | 0.1 | 1.5 | 0.2 | 1.3 | 35.2 | 6.7 |
| Feed | 100.0 | 4.4 | 3.2 | 1.7 | 100.0 | 100 | 100 |

Test No. 5

Mineral Dressing Results

| Product | Per cent Weight | Assays | | | Per Cent Distribution | | |
|---------|-----------------|--------|-----|----------|-----------------------|-------|-----------|
| | | Pb | Zn | Ag Fe | Pb | Zn | Ag Fe |
| Pb Conc | 11.3 | 31.0 | 9.7 | 12.0 8.0 | 55.6 | 44.4 | 79.2 7.1 |
| Zn Conc | 1.3 | 13.9 | 6.4 | 7.2 9.0 | 2.8 | 3.4 | 5.5 0.9 |
| Tails | 87.4 | 0.2 | 1.5 | 0.3 13.3 | 17.5 | 52.6 | 15.3 92.0 |
| Feed | 100.0 | 4.4 | 3.2 | 1.7 13.3 | 100.0 | 100.0 | 100 100 |

Test No. 9

Mineral Dressing Results

| Product | Per cent Weight | Assays | | | Per cent Distribution | | |
|---------|-----------------|--------|-----|-----|-----------------------|------|-------|
| | | Pb | Zn | Fe | Pb | Zn | Fe |
| Pb Conc | 20.6 | 46.1 | 7.0 | 6.0 | 38.4 | 48.2 | 83.3 |
| Zn Conc | 7.5 | 14.7 | 5.4 | 3.0 | 10.2 | 13.5 | 15.1 |
| Tails | 71.9 | 0.2 | 1.6 | 0.3 | 1.4 | 38.3 | 1.682 |
| Feed | 100.0 | 4.4 | 3.2 | 1.7 | 100 | 100 | 100 |

Roughing Operation

| Point of Addition | Conditions | | Reagents - pounds per ton | | | | | | | |
|-------------------|------------|-------|---------------------------|------|------|-------------------|-----|---------|-------------------|-------|
| | Time | % Sol | pH | NaOH | NaCN | ZnSO ₄ | Z-3 | PineOil | CuSO ₄ | B-140 |
| Grind | 20 | 60 | | 6.0 | 1.0 | 1.0 | | | | |
| Condition | 3 | 23 | 11.5 | | | | 0.1 | | | |
| Flotation | 3 | 23 | 11.5 | | | | | 1 drop | | |
| Condition | 2 | 16 | 10.0* | | | | | | 0.6 | |
| Condition | 2 | 16 | 10.0 | | | | | | | 0.05 |
| Flotation | 2 | 16 | 10.0 | | | | | | | |

* 4 drops of H₂SO₄ added

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Roughing Operation

| Point of Addition | Conditions | | Reagents - pounds per ton | | | | | | |
|-------------------|------------|-------|---------------------------|------|------|-------------------|-------------|-------|-------------------|
| | Time | % Sol | pH | NaOH | NaCN | ZnSO ₄ | Z-3 PineOil | B-140 | CuSO ₄ |
| Grind | 20 | 60 | | 6.0 | 1.0 | 0.5 | | | |
| Condition | 3 | 23 | 11.1 | | | | 0.1 | | |
| Flotation | 3 | 23 | 11.1 | | | | 1 drop | | |
| Condition | 2 | 20 | 10.0* | | | | | | 0.6 |
| Condition | 2 | 20 | 10.0 | | | | | 0.05 | |
| Flotation | 2 | 20 | 10.0 | | | | | | |

* 4 drops of HCl added

Cleaning Operation

| Point of Addition | Conditions | | Reagents - pounds per ton | | | | | | |
|-------------------|------------|-------|---------------------------|------|------|-------------------|-------------|-------|-------------------|
| | Time | % Sol | pH | NaOH | NaCN | ZnSO ₄ | Z-3 PineOil | B-140 | CuSO ₄ |
| Flotation | 3 | 5.5 | 10.9 | | | | | | |
| Flotation | 2 | 2.0 | 9.8 | | | | | | |

Test No. 12

Roughing Operation

| Point of Addition | Conditions | | Reagents - pounds per ton | | | | | | |
|-------------------|------------|-------|---------------------------|------|-------------------|-----|---------|-------------------|-------|
| | Time | % Sol | NaOH | NaCN | ZnSO ₄ | Z-3 | PineOil | CuSO ₄ | B-140 |
| Grind | 20 | 60 | 6.0 | 1.0 | 0.5 | | | | |
| Condition | 3 | 23 | | | | 0.1 | | | |
| Flotation | 3 | 23 | | | | | 1 drop | | |
| Condition | 2 | 20 | | | | | | 0.5 | |
| Condition | 2 | 20 | | | | | | | 0.05 |
| Flotation | 2 | 20 | | | | | | | |

* 5 Drops HCl added

Cleaning Operation

| Point of Addition | Conditions | | Reagents - pounds per ton | | | | | | |
|-------------------|------------|-------|---------------------------|------|-------------------|-----|---------|-------------------|-------|
| | Time | % Sol | NaOH | NaCN | ZnSO ₄ | Z-3 | PineOil | CuSO ₄ | B-140 |
| Flotation | 3 | 6.0 | | | | | | | |
| Flotation | 3 | 4.0 | | | | | | | |

Roughing Operation

| Point of Addition | Conditions | | Reagents - pounds per ton | | | | | | |
|-------------------|------------|-------|---------------------------|------|------|-------------------|-------------|-------------------|-------|
| | Time | % Sol | pH | NaOH | NaCN | ZnSO ₄ | Z-3 PineOil | CuSO ₄ | B-140 |
| Grind | 20 | 60 | | 6.0 | 1.0 | 0.5 | 0.1 | | |
| Condition | 3 | 23 | 11.7 | | | | | | |
| Flotation | 3 | 23 | 11.7 | | | | 1 drop | | |
| Condition | 3 | 19 | 10.2* | | | | | 0.5 | |
| Condition | 3 | 19 | 10.2 | | | | | | 0.05 |
| Flotation | 2 | 19 | 10.2 | | | | | | |

* 6 drops of HCl added

Cleaning Operation

| Point of Addition | Conditions | | Reagents - pounds per ton | | | | | | |
|-------------------|------------|-------|---------------------------|------|------|-------------------|-------------|-------------------|-------|
| | Time | % Sol | pH | NaOH | NaCN | ZnSO ₄ | Z-3 PineOil | CuSO ₄ | B-140 |
| Flotation | 3 | 6.0 | 10.6 | | | | | | |
| Flotation | 3 | 5.0 | 9.6 | | | | | | |

Test No, 13

Mineral Dressing Results

| Product | Per cent Weight | Assays | | | Per cent distribution | | | | |
|---------|-----------------|--------|-----|-------|-----------------------|------|-------|------|------|
| | | Pb | Zn | Ag Fe | Pb | Zn | Ag Fe | | |
| Pb Conc | 2.54 | 63.0 | 5.2 | 19.6 | 2.8 | 59.7 | 5.4 | 50.4 | 0.6 |
| Zn Conc | 5.24 | 28.2 | 9.5 | 11.2 | 6.0 | 36.7 | 20.6 | 55.9 | 2.8 |
| Mids | 15.08 | 1.7 | 1.7 | 0.6 | 8.3 | 6.4 | 10.6 | 5.4 | 11.2 |
| Tails | 77.14 | 0.9 | 2.0 | 0.6 | 12.3 | 17.2 | 63.4 | 28.3 | 85.4 |
| Feed | 100.00 | 4.4 | 3.2 | 1.7 | 13.3 | 100 | 100 | 100 | 100 |

Roughing Operation

| Point of Addition | Conditions | | | Reagents - pounds per ton | | | | | |
|-------------------|------------|-------|------|---------------------------|------|-------------------|-------------|-------------------|-------|
| | Time | % Sol | pH | NaOH | NaCN | ZnSO ₄ | Z-3 PineOil | CuSO ₄ | B-140 |
| Grind | 20 | 60 | | 6.0 | 1.0 | 0.5 | | | |
| Condition | 3 | 23 | 11.5 | | | | 0.1 | | |
| Flotation | 3 | 23 | 11.5 | | | | 1 drop | | |
| Condition | 2 | 19 | 9.2* | | | | | 0.6 | |
| Condition | 2 | 19 | 9.2 | | | | | | 0.05 |
| Flotation | 2 | 19 | 9.2 | | | | | | |

* 8 drops HCl added

Cleaning Operation

| Point of Addition | Conditions | | | Reagents - pounds per ton | |
|-------------------|------------|-------|------|---------------------------|-------|
| | Time | % Sol | pH | NaCN | B-140 |
| Condition | 3 | 6.0 | 10.6 | 0.25 | |
| Flotation | 3 | 6.0 | 10.6 | | |
| Condition | 2 | 5.0 | 9.6 | | 0.025 |
| Flotation | 2 | 5.0 | 9.6 | | |

Roughing Operation

| Point of Addition | Conditions | | Reagents - pounds per ton | | | | | | | |
|-------------------|------------|-------|---------------------------|------|------|-------------------|------|---------|-------------------|-------|
| | Time | % Sol | pH | NaOH | NaCN | ZnSO ₄ | Z-3 | PineOil | CuSO ₄ | B-140 |
| Grind | 20 | 60 | | 6.0 | 1.0 | 0.5 | | | | |
| Condition | 3 | 23 | 11.4 | | | | 0.05 | | | |
| Flotation | 3 | 23 | 11.4 | | | | | 1 drop | | |
| Condition | 2 | 20 | 9.2* | | | | | | 0.6 | |
| Condition | 2 | 20 | 9.2 | | | | | | | 0.05 |
| Flotation | 3 | 20 | 9.2 | | | | | | | |

* 8 drops of HCl added

Cleaning Operation

| Point of Addition | Conditions | | Reagents - pounds per ton | | |
|-------------------|------------|-------|---------------------------|------|-------|
| | Time | % Sol | pH | NaCN | B-140 |
| Condition | 3 | 4.0 | 10.5 | 0.25 | |
| Flotation | 3 | 4.0 | 10.5 | | |
| Condition | 2 | 3.0 | 9.2 | | 0.025 |
| Flotation | 2 | 3.0 | 9.2 | | |

Roughing Operation

| Point of Additions | Conditions | | Reagents - pounds per ton | | | | | | |
|--------------------|------------|-------|---------------------------|------|------|-------------------|-------------|-------------------|-------|
| | Time | % Sol | pH | NaOH | NaCN | ZnSO ₄ | Z-3 PineOil | CuSO ₄ | B-140 |
| Grind | 20 | 60 | | 6.0 | 1.0 | 0.5 | | | |
| Condition | 3 | 23 | 11.5 | | | | 0.05 | | |
| Flotation | 3 | 23 | 11.5 | | | | 1 drop | | |
| Condition | 2 | 20 | 8.2* | | | | | 0.6 | |
| Condition | 2 | 20 | 8.2 | | | | | | 0.05 |
| Flotation | 2 | 20 | 8.2 | | | | | | |

* 10 drops of HCl added

Cleaning Operation

| Point of Additions | Conditions | | Reagents - pounds per ton | | |
|--------------------|------------|-------|---------------------------|------|-------|
| | Time | % Sol | pH | NaCN | B-140 |
| Condition | 3 | 5.0 | 10.3 | 0.25 | |
| Flotation | 3 | 5.0 | 10.3 | | |
| Conditions | 2 | 4.5 | 8.0 | | 0.025 |
| Flotation | 2 | 4.5 | 8.0 | | |

Roughing Operation

| Point of Addition | Conditions | | Conditions and Reagents | | | | | | |
|-------------------|------------|-------|-------------------------|------|------|-------------------|-------------|-------------------|-------|
| | Time | % Sol | pH | NaOH | NaCN | ZnSO ₄ | Z-3 PineOil | CuSO ₄ | B-140 |
| Grind | 20 | 60 | | 6.0 | 1.0 | 0.5 | | | |
| Condition | 3 | 23 | 11.4 | | | | 0.05 | | |
| Flotation | 3 | 23 | 11.4 | | | | 1 drop | | |
| Condition | 2 | 21 | 8.6* | | | | | 0.6 | |
| Condition | 2 | 21 | 8.6 | | | | | | 0.05 |
| Flotation | 2 | 21 | 8.6 | | | | | | |

* 10 drops HCl added

Cleaning Operation

| Point of Addition | Conditions | | Conditions and Reagents | |
|-------------------|------------|-------|-------------------------|---------------------------|
| | Time | % Sol | pH | Reagents - pounds per ton |
| Condition | 3 | 3.5 | 10.3 | NaCN 0.25 B-140 |
| Flotation | 3 | 3.5 | 10.3 | |
| Condition | 2 | 3.5 | 9.6 | 0.025 |
| Flotation | 2 | 3.5 | 9.6 | |

Test No. 17

Mineral Dressing Results

| Products | Per cent Weight | Assays | | | Per cent Distribution | | | | |
|----------|-----------------|--------|------|-------|-----------------------|------|-------|------|------|
| | | Pb | Zn | Ag Fe | Pb | Zn | Ag Fe | | |
| Pb Conc | 3.92 | 58.0 | 8.3 | 19.2 | 3.8 | 54.7 | 14.7 | 1.7 | 1.3 |
| Zn Conc | 2.22 | 30.8 | 13.4 | 13.4 | 7.3 | 19.5 | 13.6 | 16.6 | 1.4 |
| Mids | 9.86 | 1.4 | 1.6 | 0.8 | 7.7 | 3.8 | 7.1 | 4.4 | 6.8 |
| Tails | 84.00 | 0.5 | 1.7 | 0.8 | 12.0 | 12.0 | 64.6 | 37.3 | 90.5 |
| Feed | 100.00 | 4.4 | 3.2 | 1.7 | 13.3 | 100 | 100 | 100 | 100 |

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