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LUIGI D. PIANA

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
Mineral Dressing Option

MONTANA SCHOOL OF MINES
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ABSTRACT

A sample of gold-silver arsenical ore from the Herbert Gold Mine, Lincoln County, Montana, was subjected to a mineral dressing and marketing investigation to determine the treatment which will yield the highest dollar return per ton of ore processed.

The assay of the head sample was 0.25 oz Au and 19.5 oz Ag with abundant arsenic and antimony present; minerals identified by mineragraphic, selective iridescent filming, and X-ray analyses were arsenopyrite, marmatite, jamesonite, zinkenite, geocronite, proustite, argentite, and quartz.

Liberation studies demonstrated that a 20-minute batch grind of a 600-gram sample in a laboratory rod mill gave a total average liberation of 80 per cent. Gravity concentration and cyanidation of both raw ore and roasted concentrates were not considered feasible processes from either a metallurgical or an economic standpoint.

Both selective and bulk flotation tests were performed, but economic considerations indicate that a bulk concentrate assaying 1.10 oz Au, 105.5 oz Ag, 10.5 per cent Pb, and 9.2 per cent Zn shipped to the Bunker Hill smelter in Kellog, Idaho, will yield the greatest net total return, namely $17.82 per ton of ore treated f.o.b. smelter, with milling and transportation costs included.

Greater profits will be obtained by roasting the concentrates and shipping the calcine to the Bunker Hill smelter.

A particular scheme of operations for a mill to be erected is recommended, and after the mill is in operation a roasting plant could then be contemplated.

INTRODUCTION

General Considerations

The factors that influence the choice of a method for treatment of an ore comprise the technical and economic limitations and advantages, derived in detail and balanced according to the exigencies of the particular situation. If the precious metals in an ore are in such a quantity that the ore derives the greater amount of its value from their presence, the determination of the best recovery method in any particular case may be a critical technical job and, from an economic standpoint, a matter of opinion.

Gardner and Johnson[1] list, as major considerations in the choice of a type of mill, the following:

1. Nature of the valuable and valueless minerals.
2. Amenability of the valuable minerals to the available methods of treatment and the effect of the gangue minerals present.
3. Quantity of ore developed, and amount justifiably suspected.
4. Comparative treatment costs by the different methods applicable.
5. Comparative installation costs.
6. Comparative marketing costs.
7. Available methods of financing and their comparative costs.

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[1] References at the end of the paper.

The mining industry as a whole is not, with very few exceptions in particular periods of time, made up of eleemosynary organizations, but the economic problem of processing the minerals at a profit is an absolute dictator in the applicability of a chosen method.

The selection of a process for treating a precious metals ore is governed primarily by the characteristics of the ore under consideration. The form in which gold and silver occur, its mode of association with other minerals, the size and surface condition of the particles are important factors which influence materially the choice of procedure for recovering gold and silver. The presence of other valuable minerals as copper, lead, zinc, antimony, cobalt, etc., exerts a conspicuous influence not only on the primary process for the recovery of the precious metals, but also on the further processing of barren tailings. A knowledge of these characteristics may be obtained by microscopic and chemical methods combined with experimental testing.

Types of Ores and Processes

From the standpoint of a metallurgical treatment, the precious metal ores may be classified as follows:[2]

1. Ores containing free or native gold in quartz or quartz-adularia gangues. These ores seldom contain appreciable amounts of silver.

2. Ores in which the gold and silver are associated with the oxides of lead, copper, and iron.

3. Ores in which the gold and silver are associated with pyrite or other sulphides. The majority of precious metal ores fall into this class.

4. Ores which contain gold and silver with appreciable quantities of arsenic, antimony, and tellurium.

5. Ores in which the gold and silver are associated with the common sulfides minerals of copper, lead, and zinc, the precious metals being for the most part of secondary importance in the consideration of the most economical method of treatment.

6. Tailings (old or new).

The above subdivision must not be understood as rigidly delimiting fixed characteristics; but the ores, in their multiple variety, grade from one classification into another one, and may have several characteristics at the same time.

In applying concentration to the treatment of these various classes of precious metals ores, several processes and combination of processes may be applicable; but the economic status of a given operation usually limits the choice of processes. There are today four major methods of recovering precious metals from ore: amalgamation, gravity concentration, cyanidation, and flotation; each of these processes has a characteristic function, and the modern mill often combines two or more of these processes for the treatment of its ore.

Precious metals ores, generally, have a much wider variation in their valuable components than base metals ores, and the physical characteristics of the valuable components are extremely important in working out the metallurgy. The mineral engineer's problem is to adopt those processes which will give the greatest dollar-yield-per-ton of ore treated. Since mines are a depleting asset, it is necessary to effect a maximum dollar yield with a minimum capital outlay.

Object of the Investigation

A gold-silver arsenical ore from the Herbert Gold Mine, Lincoln County, Montana, was subjected to a Mineral Dressing Investigation in order to determine the best method of treatment and the more advantageous procedure which will yield the highest dollar return per ton of ore treated.

Among all the possible methods of mineral beneficiation, only those which were deemed applicable by a medium sized mine operator were tested; the availability of necessary supplies such as fuel, water and electric power for a regular and efficient operation of a plant, and transportation facilities of both supplies and processed products were economic factors which were continuously remembered in applying new testing processes.

A study of smelter schedules for the most profitable sale of the products was pursued.

IDENTIFICATION OF MINERALS

Three hundred pounds of run of mine ore were received at the Mineral Dressing Laboratories of the Montana School of Mines, and the following procedure was followed for the preparation of the tests sample.

The ore was ground in a laboratory jaw crusher set at $\frac{1}{2}$ inch and screened through a 10-mesh screen. The oversize was ground in a set of laboratory rolls and screened again until all the ore was minus 10-mesh. A few pieces of ore were reserved for megascopic examination, for specific gravity measurements, and for polished section testing.

Megascopic Examination

Visual examination with the aid of a pocket lens revealed sulfide minerals imbedded in a massive quartzite gangue. The sulfides were characteristically homogeneous, and were classified as what is commonly called "feather ore", namely lead sulfosalts of various and variable composition.

Specific gravity measurements with a Jolly balance on several representative samples gave an average of 2.50.

Spectographic Analysis

In order to have a lead in the subsequent chemical assay of the ore, two qualitative spectrographic analyses were performed on pulverized representative sulfide samples to determine what elements were present in the ore. The elements determined were lead, zinc, iron, copper, arsenic, antimony, and sulfur.

Assay of the Ore

The ground ore was thoroughly mixed, coned and quartered, split with Jones riffles until a representative sample of about two pounds was obtained, which was further ground in a laboratory pulverizer to pass a 100-mesh screen. This product

Table I. Assay of the Untreated Ore.

Gold	0.25	oz/ton
Silver	19.5	oz/ton
Lead	2.2	per cent
Zinc	1.5	per cent
Copper	0.18	per cent
Iron	5.2	per cent
Arsenic	3.8	per cent
Antimony	1.3	per cent
Sulfur	4.4	per cent
Insoluble	73.1	per cent
Silica	65.8	per cent
Lime	1.0	per cent
Alumina	3.6	per cent

was parted: one half was stored as reject and one half was sent for assay *. The results are shown in Table I.

From these data, it is logical to conclude that the ore is highly siliceous, and that the base metals present will probably form mixed sulfides, antimonites and arsenites.

Microscopic Studies

Several representative samples of the ore, tabling concentrates, and flotation concentrates were mounted in lucite briquettes and polished for microscopic studies in order to determine the mineral species present in the ore.

This part of mineral investigation is one of the most important in the study of the possible methods of ore beneficiation, because the mineral engineer is concerned not so much with the elements, but with the mode of association and aggregation of the elements in complex compounds, which must be liberated one from the other, and separated in order to obtain profitable salable products.

Mineragraphic analyses with a metallurgical microscope and perpendicular illumination revealed the presence of three opaque minerals which were identified as arsenopyrite, zinc sulfide, and lead sulfosalts. Since microchemical analysis revealed the presence of abundant iron in the zinc sulfide, it was inferred that this mineral was marmatite, which, according to Dana [3], can contain as much as 26 per cent iron in a variety named christophite, iron which is present

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*C. J. Bartzen, Analyst, Montana Bureau of Mines and Geology, Montana School of Mines, Butte, Montana as an atomic replacement of the zinc in the lattice of the sphalerite.

The single determination of the lead sulfosalts was extremely difficult by mineragraphic and microchemical methods because of the following reasons:

1. The lead sulfosalts looked alike under perpendicular light.
2. Because of the above mentioned intermixing characteristics of the minerals, one drop of reagent covered different grains, setting up electrochemical secondary reaction.
3. Many lead sulfosalts can be both isotropic and anisotropic under polarized light.

Free gold and silver were not observed in either the ore specimens or in the concentrates, this fact indicating that the gold is present as a chemical compound or as an atomic substitute of base metals, whereas the silver may be combined with sulfosalts.

Selective Iridescent Filming

In order to be relatively sure of the presence of different lead sulfosalts and to determine the existence of silver minerals, an investigation with selective iridescent filming was performed.

This method, discovered by Leo, applied quantitatively and developed theoretically by Gaudin[4], McGlashan[5] and coworkers, consists in forming on the polished surface a film of transparent substance, whose thickness is controlled by the time of contact of the polished surface with the filming solution. The light striking perpendicularly is reflected from the top, from the bottom, and from within the film; since the various reflections are out of step with reference to each other, they interfere and give rise to characteristic colors determinative of the particular mineral surface, and of the thickness of the film. In other words, two different mineral surfaces can have the same color for different times of contacts with the filming solution, but they will have different colors for the same time of contact.

Painstaking experiments were performed by Gaudin and co-workers in order to prepare determinative tables of minerals according to solution used and time of contact; but the work, now just in the initial stage, requires the collaboration and integration of the results of a great number of investigators.

At present this method is particularly helpful in distinguishing between minerals which react in the same way to etch tests, in determining exsolution, and in studies of metallurgical products, like mill tailings, for particle counting.

<u>Filming Solutions.</u> Different solutions were prepared for silver minerals and for lead solfosalts. Also the silver minerals are stained differentially when filming the lead sulfosalts, but their differentiation is not clear.

The standard silver filming solution was prepared by mixing one part of 2 per cent dissolved iodine in methyl alcohol and one part of concentrated sulfuric acid. In the present work, hydrochloric acid was used in place of sulfuric acid in order to shorten appreciably the time of contact of the minerals with the staining solution.[5] The standard sulfides staining solution was prepared by dissolving 8.2g grams of Cr_2O_3

in 50 cc of 12 N hydrochloric acid, and diluting the solution with H_2O to 250 cc.

Procedure. A particular section of a well-cleaned and polished mineral surface was photographed with the use of a petrographic microscope and a Leica camera. Then the surface was brought into contact with the staining solution for a determined time, washed well with water, rinsed with a stream of acetone, and quickly dried with a jet of air. The same place was again photographed with the same instruments.

Photomicrographs. Some of the most characteristic photomicrographs of filmed and unfilmed minerals are here reproduced. It must be emphasized that under the microscope a filmed mineral appears with striking and characteristic colors of the first order, and the mineral differentiation is easily performed; in a black and white photomicrograph this sharp cut differentiation is not easily observed because the characteristic colors are represented only in some shades of gray.

A polished section reproduced in Fig. 1 and 2 representing lead sulfosalts in marmatite was filmed with standard silver solution for 10 seconds, and a reticular-like formation of silver sulfides in the lead minerals resulted in bright blue colors (gray in picture). It is interesting to note that also in the marmatite, silver sulfides were revealed in deep purple colors (black in the photograph).

An area reproduced in Fig. 3 and 4 and representing marmatite and

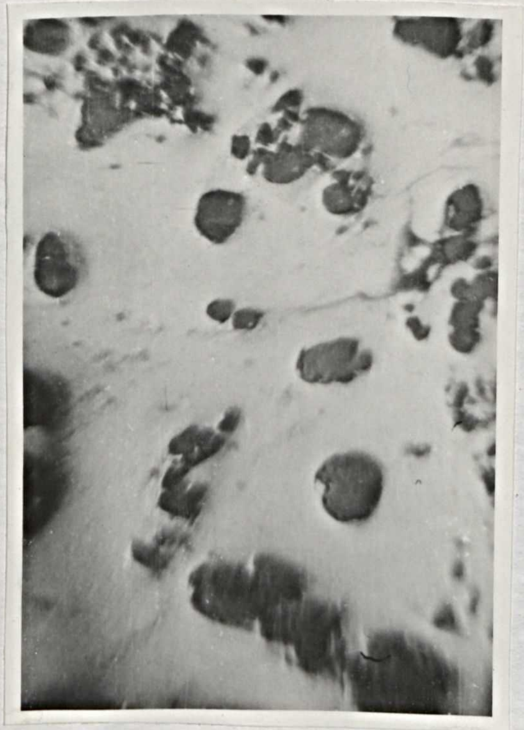
Fig. 1.- Lead sulfosalts (white) replacing marmatite (dark); not filmed. X 200.

Fig. 2.- Same section represented in Fig. 1 filmed for 10 seconds with standard silver solution. X 200.

lead sulfosalts was filmed with standard sulfides solution for 10 seconds and bright purple and brown colors differentiated the various lead sulfosalts (different shades of gray in picture).

An area reproduced in Fig. 5 and 6 with arsenopyrite, lead sulfosalts, and marmatite was filmed with standard sulfides solution for 15 seconds; one of the best results was obtained, but the black and white photomicrograph does not distinguish very sharply with various shades of dark gray, between the brilliant purple, blue, and brown colors which differentiated the various lead sulfosalts. Arsenopyrite was unaffected.

<u>Conclusion from Mineral Filming.</u> Selective filming of zinc and



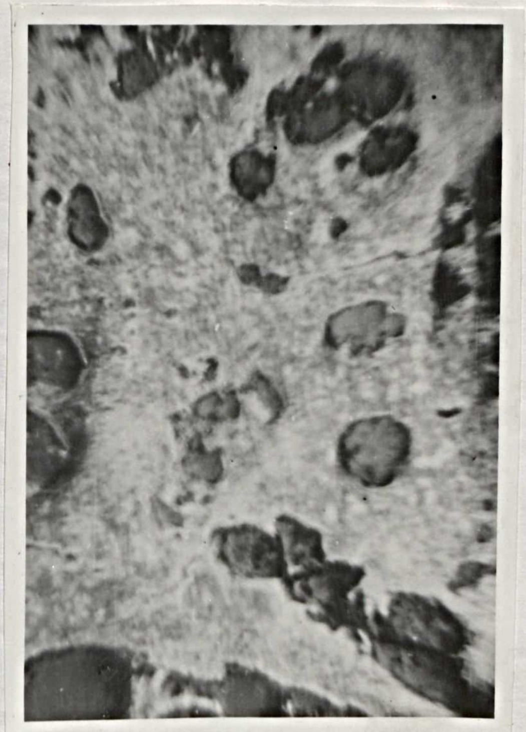


Fig. 3.- Lead sulfosalts (white) replacing marmatite (dark); not filmed. X 200.

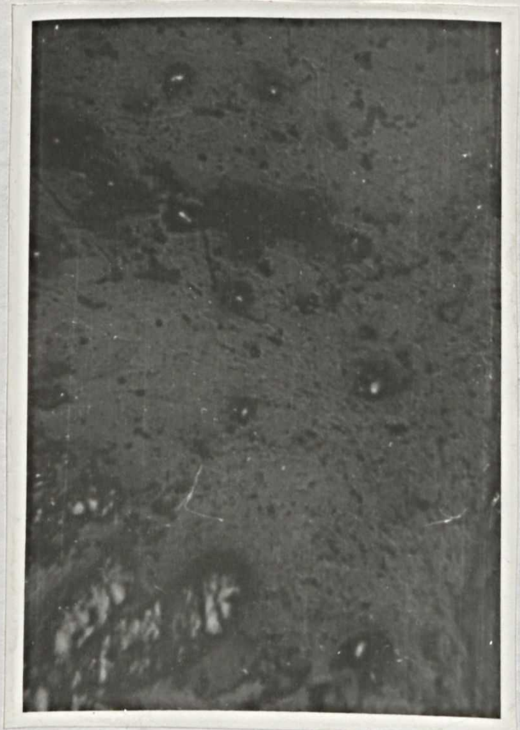
Fig. 4.- Same section represented in Fig. 3 filmed for 10 seconds with standard sulfides solution. X 200.

lead minerals revealed the following points:

1. Silver minerals are present as several different sulfosalts in both lead and zinc minerals.

2. Lead sulfosalts grade one into the other without definite boundary demarkation. This well established fact precludes the selective separation and concentration of wanted and unwanted lead minerals.

3. With reference to the method of selective iridescent filming, the following silver minerals were tentatively identified: proustite (Ag_3AsS_3), stephanite (Ag_5SbS_4), pyrargyrite (Ag_3SbS_3), miargyrite ($AgSbS_2$), argentite (Ag_2S).



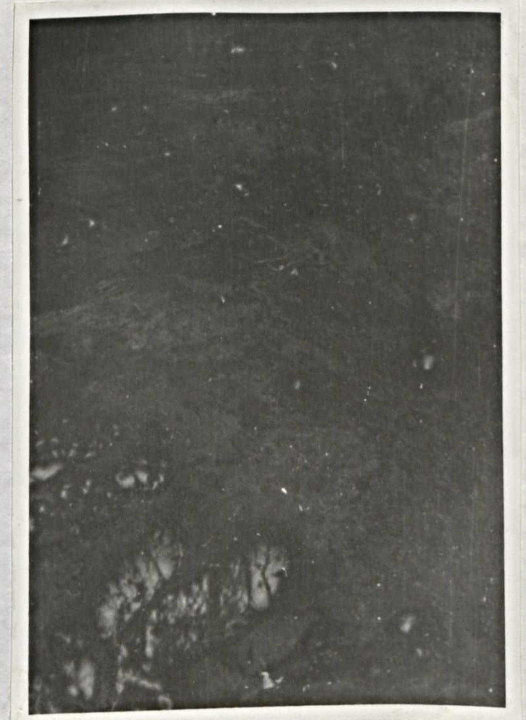


Fig. 5.- Arsenopyrite (white) and marmatite (dark) replaced by lead minrals (gray); not filmed. X 200.

Fig. 6.- Same section represented in Fig. 5 filmed for 15 seconds with standard sulfide solution. X 200.

4. The identification of the lead sulfosalts presented more difficulties, but it is safe to assume that the lead minerals which occur with greater frequency in this ore are jamesonite ($Pb_4FeSb_6S_{14}$), geocronite ($Pb_5(Sb, As)_2S_8$), dufrenoisite ($Pb_2Ag_2S_5$), and zinkenite ($Pb_6Sb_{15}S_{27}$).

X-Ray Analysis

All methods of crystal analysis by X-Rays depend upon the diffraction of X-ray waves from the planes of atoms in a crystalline substance. From the standpoint of the diffracting medium, there are three methods of X-ray analysis:

1. Laue's method, in which a beam of X-rays is passed through a thin oriented section of a crystal.

2. The spectrometer method, developed by the Braggs[6], in which X-rays are diffracted from planes of atoms parallel to the smooth surface of an oriented crystal.

3. The powder method, developed by Debye and Scherr, in which the rays pass through a small sample of crystalline powder.

The third method was the one employed in this investigation. The objective was to determine, as accurately as possible the different mineral species present in the ore, and to use this information to corroborate the microscopic analysis.

In the powder method[7], a sample of fine crystalline powder is placed in the path of a ribbon-like beam of monochromatic X-ray. The powder causes diffraction of the rays which are registered on a photographic film curved to conform with the arc of a circle with the sample at the center. The film shows a series of parallel lines of varying intensities, and the pattern produced is characteristic of the crystalline material under investigation.

Procedure

In order to obtain a high-purity material, two products, the zinc and lead clearer-concentrates from differential flotation Test No. 5, were sieved and the -150 + 200 fractions subjected to a sink and float separation in methylene iodine of specific gravity 3.2. The float products were discarded and the sink products, after being washed with methyl alcohol and rinsed with acetone, were mixed with varying quantities of corn starch, and introduced into lucite cylinders 15 mm long and 1 mm in diameter for X-ray analysis.

The apparatus used was operated at 15 milliampers and 45 kv. The powder sample was continuously rotated by mechanical means during the exposures of different lengths of time in the circular camera of 7 cm radius.

Copper and cobalt targets were used, which gave respectively a Ka radiation of 1.5412 A°, with an iron oxide filter, and a Ka radiation of 1.7892 A°, with a nickel oxide filter.

Knowing the radius of the camera, the distance of the lines from the center of the film and the wave length of the radiation used, the first, second, and third orders of the distance between the atomic planes of the crystalline substance under investigation were easily calculated from the Bragg's equation:

$$d = \frac{\lambda}{2 \sin \theta}$$

By comparing the results obtained for several distances with standard tables, mineral identification was attempted.

X-Ray Results

Zinc Minerals. After two unsuccessful attempts, a satisfactory diffraction pattern was obtained using the Cu K radiation and an exposure

Table II. X-Ray Data on Zinc Minerals.

Intensity	d(calc.)	d(from tables*)	Mineral	Formula
strong	3.346	3.35	Quartz	SiO_2
strong	3.119	3.12	Sphalerite	ZnS
faint	2.431	2.43	Argentite	Ag_2S
strong	1.914	1.91	Sphalerite	ZnS
faint	1.821	1.81	Quartz	SiO_2
faint	1.633	1.63	Sphalerite	ZnS

* Card File of X-ray diffraction data published by the American Society for Testing Materials, Philadelphia, Pa., 1944.

of one hour. Table II presents the results obtained; only double lines were used in the calculations. The data indicated that quartz and zinc sulfides were undoubtedly present. The presence of argentite was given by only one faint double line; and, considering the small percentage of silver present, the determination of argentite lines is not an easy achievement.

The fact should be emphasized that sphalerite and marmatite have the same diffraction pattern, which indicates that the atomic substitution of iron to zinc does not distort the crystal lattice pattern of sphalerite.

Lead Minerals. Satisfactory diffraction patterns for lead minerals were not obtained. Five attempts were made with both Cu K and Co K radiations and with different exposure times, but none was quite successful.

Table III. X-Ray Data on Lead Minerals.

Intensity	d(calc.)	d(from tables*)	Mineral	Formula
strong	3.346	3.35	Quartz	SiO_2
strong	1.980	1.98	Zinkenite	$Pb_6Sb_{15}S_{27}$
medium	1.920	1.92	Sphalerite	ZnS
medium	1.815	1.82 1.81	Zinkenite Quartz	$Pb_6Sb_{15}S_{27}$ SiO_2
faint	1.544	1.54	Quartz	SiO_2
faint	1.376	1.37	Quartz	SiO_2

Table III exhibits the results obtained from double lines; results from single lines or from very weak lines were discarded as not reliable. The results indicated that quartz is undoubtedly present, and sphalerite was probable because of imperfect separation. The only lead mineral determined is zinkenite, the presence of which was given by only a strong double line and by a medium line which can belong either to zinkenite or to quartz. It is certainly a positive fact that other lead sulfosalts, especially arsenides, were present. Better manipulation procedures, different dilution with corn starch, different intensities and apertures of X-ray beams might yield more determinative results.

LIBERATION STUDIES

The photomicrographs shown in Fig. 1, 2, 3, 4, 5, and 6, clearly illustrate the fact that complete liberation of the lead sulfosalts, marmatite, and arsenopyrite from each other and from the quartz would be obtained only at a very small particle size.

Two 600-gram batches of minus 10-mesh ore were ground in a laboratory rod mill with a charge of 13 Kg of rods at 60 rpm. One batch was run for 10 minutes and the other one for 20 minutes; the pulps were wet and dry screened, and the results are reported in Table IV.

Table IV. Sieve Analyses of 600-grams minus 10-mesh Batches of Ore Ground in a Laboratory Rod Mill for 10 and 20 minutes.

Mesh	10' Grind		20' Grind	
	% Weight	Cumulative Weight %	% Weight	Cumulative Weight %
+ 35	0.14	0.14		
+ 48	3.43	3.57		
+ 65	19.28	22.85	0.28	0.28
+ 100	19.29	42.14	2.70	2.98
+ 150	11.28	53.42	16.76	19.74
+ 200	8.71	62.13	18.04	37.78
- 200	37.85		62.22	

These grinding tests are compared on a percentage-wise basis; the 10-minute grind gave a 3.5 per cent plus 48-mesh product with 37.8 per cent minus 200-mesh; the 20-minute grind gave a 3.0 per cent plus

100-mesh product with 62.2 per cent minus 200-mesh. With consideration to the high weight of rods used, these tests accentuate the considerable hardness of the ore.

A representative sample of each screened size was mounted in lucite, polished and subjected to a microscopic particle counting for calculation of the degree of liberation. Only bulk liberation of the sulfide minerals from the quartz was calculated because of the already explained difficulty in the mineral differentiation. These results plotted in Table V verify the deduction drawn from the interpretation of the photomicrographs: namely, great difficulty of liberation because of the extremely fine character of the sulfide inclusions.

Table V. Computation of the Degree of Liberation.

Size	Per cent of free particles		% Degree of Liberation	
	Quartz	Sulfides	Quartz	Sulfides
- 65 + 100	62.68	4.78	69.58	48.18
- 100 + 150	57.14	7.79	65.34	62.02
- 150 + 200	68.70	8.78	79.08	67.18
- 200 (deslimed)	56.29	29.47	84.93	87.40

In order to calculate the total degree of liberation obtained after a certain grinding time, the following procedure was observed: the degree of liberation of each size was multiplied by the corresponding weight per cent of the size and the product were added.

For the 20-minute grind the results are reported in Table VI; absolute exactness is not claimed for the following reasons:

a. Constant error of the operator.

b. Error caused by the finite numbers of particles counted.

c. Particles counted as free quartz could possibly have had a core of sulfides, and vice versa.

d. Error caused by the desliming of the 200-mesh size.

Table VI. Computation of the Total Degree of Liberation after 20-minute Grind in a Rod Mill.

Size	% Weight	Degree of Liberation		Size Total	
		Quartz	Sulfides	Quartz	Sulfides
- 65 + 100	2.98	.8958	.4818	2.07	1.43
- 100 + 150	16.76	.6534	.6202	10.94	10.39
- 150 + 200	18.04	.7908	.6718	14.24	12.10
- 200 (deslimed)	62.22	.8493	.8740	52.80	54.38
Total Liberation				80.05	78.20

At any rate, relying upon the fact that some of the errors could have been equivalent in opposite directions, the results of 20-minutes grinding-time were considered satisfactory to obtain a pulp for subjection to flotation for the recovery of a rougher concentrate. If a cleaner concentrate should report a too great amount of quartz floated in locked or binary particles, a regrind of the rougher concentrate for more complete liberation could then be taken into consideration.

GRAVITY CONCENTRATION

Deduction and considerations drawn from the study of the mineralogical components of the ore and from the mode of liberation was unfavorable to gravity concentration for the following reasons:

1. No great difference in specific gravity between sulfides and gangue.

2. Complete liberation requires extremely fine grinding: small particles do not respond well to gravity concentration but are lost in the tailings.

At any rate tabling and jigging tests were performed in order to verify in practice what theoretical considerations suggested.

Tabling

Three batches of 600-grams each of minus 100-mesh ore were ground in a laboratory rod mill for 20 minutes, dried and charged with a vibratory feeder to a laboratory Wilfley table using small amount of water, small angle of tilt, and low feeding rate. Because of the smallness of the particles, the tabling operation produced wide variations of mineral bands on the concentrating end of the deck, and these bands were very sensitive to minute variations in adjustment of the table.

A 3.5 per cent concentrate was obtained assaying 1.62 ounces of gold, 81.4 ounces of silver, 6.8 per cent of lead, and 6.9 per cent of zinc. The operation was declared unsuccessful in separation because of the high assay of the tailing which, being 36.6 per cent by weight, had a content of 0.22 ounces of gold, 21.6 ounces of silver, 1.1 per cent of lead, and 1.1 per cent of zinc.

Jigging

The same procedure was followed for the preparation of a feed to a Denver laboratory mechanical jig. Difficulties were encountered in preparing a constant bed, and the introduction of a coarser feed in part solved the problem.

A 7.7 per cent concentrate was obtained, assaying 0.90 ounces of gold, 52.1 ounces of silver, 5.8 per cent lead, and 5.0 per cent zinc. Again, the operation was considered unsuccessful because of the high grade tailings produced assaying 0.18 ounces of gold, 15.2 ounces of silver, 1.8 per cent lead, and 1.3 per cent zinc.

Complete tabulation of results of gravity concentration are reported in the appendix.

FLOTATION

Gold and silver in the free state respond quite readily to the common flotation reagents. If the precious metals are present either as sulfosalts or are entrapped in other base metal sulfides, the flotation of the base metal minerals will also separate gold and silver from the gangue.

From a theoretical point of view the best results on this ore should be obtained by differential flotation yielding a high grade lead concentrate and a high grade zinc concentrated to be shipped to the respective reduction plants. Since the zinc plant pays for only 80 per cent of gold and silver, it would be well to have most of the precious metals in the lead concentrate; the iron should be depressed in the gangue together with arsenic and antimony. The conclusion from the mineralogical studies indicated that the lead is chemically tied with arsenic, antimony and sulfur, whereas the zinc is associated with considerable iron. In reference to the lead and zinc minerals it must be remembered that flotation is not an operation that destroys the chemical bond. Actually, the unitary cell structure will persist even with the very finest comminution. Therefore, it seems quite a remote possibility that high grade base metals will be obtained in the concentrates. Even with a successful depression of arsenopyrite, if barren of precious metals, considerable amount of iron will still be present in jamesonite and marmatite. The iron present will reduce the ratio of concentration, will be a lost weight in transportation, and will be penalized in a zinc plant at 30 cents per unit. On the other hand, a high grade zinc concentrate is a necessity because of the additional penalty in a zinc plant if the assay of the zinc is lower than 45 per cent.

Flotation Testing Procedure

All the flotation tests were performed as batch operations; a 600-gram representative sample of the minus 10-mesh ore was ground with the necessary reagents for 20 minutes in a laboratory rod mill to duplicate the results of Table IV. The pulp was transferred to a 600-gram Fargergren laboratory flotation machine at an initial pulp density of 23 per cent, and the conditioning, modifying and collecting agents were added. After conditioning, a frother was added, and the mineral bearing froth was removed. Subsequently, cleaning and recleaning operations were carried out as deemed necessary to each test.

Each single flotation test is reported in the Appendix at the end of the paper in a series of tables which show assays, per cent distribution, reagents consumed, and brief discussions of particular tests.

Flotation Results

From a general point of view, the ore can be very easily floated with the usual flotation reagents used for sulfide ores; both xantates and areofloats are effective collective agents for the lead and zinc sulfosalts, whereas the use of Minerec A (dixanthogen) is critical for the complete recovery of silver minerals. Zinc sulfate and copper sulfate proved effective in the respective depression and activation of zinc minerals, and sodium silicate was particularly helpful in the cleaner circuits to depress the slimy silicates necessarily present because of the fine grind to which the ore was subjected for mineral liberation. Lime was used to adjust the pH and to depress the arsenopyrite, and pine oil was the frother which gave the best results.

No difficulty was encountered in obtaining final tails as low as 0.03 ounces of gold, 1.1 ounces of silver, 0.1 per cent lead, and 0.1 per cent zinc, (see Flotation Test No. 6). Much more difficult was the problem of obtaining high grade clean concentrates for the already explained reasons: namely, presence of great quantities of arsenic and antimony intimately associated with the lead minerals, and abundant iron in both lead and zinc minerals. At first it was reasonably thought that, if arsenopyrite was depressed in the tailings, concentrates might have a lower content of iron and arsenic; but it was not possible to depress the arsenopyrite to any reasonable degree either with lime (see Flotation Tests No. 9, 10, 11, and 12), or with cationic reagents (see Flotation Tests No. 14, 15, and 16).

In the early phase of the flotation experiments it was thought that probably a differential flotation would have brought the best results from an engineering point of view. Flotation Test No. 6 is a typical example with a ratio of concentration of 6.25 : 1. The lead cleaner concentrate assayed 0.87 ounces of gold 115.1 ounces of silver, 10.5 per cent lead, and 3.6 per cent zinc; the zinc cleaner concentrates assayed 0.54 ounces of gold, 12.9 ounces of silver, 0.7 per cent lead, and 23.7 per cent zinc. If we consider the fact that in a continuous flotation operation the lead and zinc cleaner tails are recirculated and their values not lost, the distribution of both precious and base metals was over 90 per cent.

As will be explained later, because the low gross revenue of the concentrate from a zinc plant caused by the low zinc assay, the differential flotation from an economic point of view is not as profitable as the bulk flotation. Flotation Test No. 13 is a typical example in which, in order to obtain a high grade concentrate with low silica content, the rougher concentrate was further ground for 12 minutes in a laboratory rod mill for more complete liberation prior to cleaning. The cleaner concentrate assayed 1.10 ounces of gold, 105.5 ounces of silver, 10.5 per cent lead, 9.2 per cent zinc, 15.0 per cent iron, and 3.2 per cent silica with a ratio of concentration of 8.04 : 1.

Flotation Test No. 17 represents a series of twelve single flotation tests performed with the general conditions of Test No. 13, in order to obtain a substantial amount of concentrate to be subjected to roasting and cyanidation.

Amine flotation of the sulfides minerals was tried using Laurylaminehydrochloride (DP 243) as collector following the experiments of Kellog and Vasques-Rosas [9]; the pH was maintained at above 11.0 to depress the quartz and very incouraging results were obtained from the grade of concentrate and recoveries. Especially important is the economic factor derived from using exceedingly small amounts of Laurylaminehydrochloride, no Minerec, and probably no activator agent for the zinc minerals (see Flotation Tests No. 14, 15 and 16).

ROASTING AND CYANIDATION

General Considerations

Since free gold and silver are readily soluble in cyanide solution, this method of treatment is particularly effective both on low-grade ores and on high-grade concentrates. Cyanide solutions must contact the gold to insure recovery, and the porosity of the ore treated and the fineness of grinding required are important governing factors. The use of lime in cyanidation is necessary to protect the cyanide, to precipitate undesirable substances, to settle slimes, and to safeguard the health of the workers.

Several substances in the ores are harmful during cyanidation; among them are those that combine with cyanide to form cyanides, and carbon, which precipitates gold from the pregnant solution. Arsenic and antimony are particularly detrimental if present in sufficient quantities, because they increase the cyanide consumption so much that the treatment will be uneconomical. These and other difficulties caused by refractory minerals entrapping the precious metals are often overcome by roasting prior to leaching.

In order to determine the amenability of this ore to cyanidation, a series of preliminary tests were performed on both the raw ore and the flotation concentrates. Because of the great amount of arsenic and antimony present in the concentrate, an oxidizing roast was tried for their elimination.

Roasting the Concentrates

From Flotation Test No. 18, 800 grams of concentrates contained in an eleven-inch fire-clay dish were introduced in a gas fired muffle furnace, the temperature of which was controlled by a Bristol thermoelectric pyrometer. At the introduction of the concentrates into the muffle, the temperature was 1000°F, which was raised at a steady rate throughout the roasting operation. The concentrates were hand rabbled every 10 minutes in order to speed the process of oxidation and to avoid partial fusion or nodulization of the calcine. Very abundant white-bluish fumes of As_2O_3, Sb_2O_3 and SO_2 were evolved from the concentrates especially upon rabbling. At the end of a 2-hour period and at a muffle temperature of 1580° F, the calcine was withdrawn from the furnace, cooled completely, and weighted (540 grams).

From the complete analysis of both concentrate and calcine reported in Table IX, it was noticed that a dead roast was not performed, because of the presence of 2.8 per cent sulfur in the calcine; the elimination of arsenic was 36.5 per cent whereas the elimination of antimony was only 31 per cent. No appreciable amount of base or precious metals was lost by roasting.

Cyanidation Procedure

Because of the small amount of calcine the following procedure was observed in order to obtain the greatest amount of information from a cyanide treatment.

A 200-gram sample of the ore or calcine all ground and stage screened to pass the desired mesh was introduced in a 2-liter Winchester bottle, and the correct amount of solid NaCN introduced to give the desired strength of solution. An initial amount of CaO, as determined by preliminary acidity tests, and 600 cc of distilled water were added; and the bottle was set horizontally on rolls driven by an electric motor. After one-hour run, a 10-cc sample of the clear solution was pipetted out and tested with standard silver nitrate and acid (H_2SO_4) solutions for NaCN and CaO content: any discrepancy from the desired amount was added, and the bottle again agitated on rolls. The solution strength was subsequently checked at the end of 2- and 4-hour runs.

After 12 hours of agitation, the bottle was withdrawn, the solution tested for NaCN and CaO content, and the pulp filtered in a vacuum filter. After thorough washing with distilled water, the cake was carefully transferred back to the bottle, and the combined pregnant solution and washing were sent for gold and silver assay. The desired amount of CaO and NaCN were added to the cake with enough water to give a 3 : 1 pulp dilution, and the bottle again was set on the rolls. The same washing procedure was repeated after 24, 48 and 72 hours of agitation; and at the end of the leaching operations also the dried tailings were sent for assay with the cyanide solutions.

From the analyses, all the necessary information was obtained by calculations, and care was exercized in taking into consideration the amount of solution withdrawn by pipetting for line and cyanide testing.

Table VII. Data from Leaching the Ore and the Roasted Concentrates with Cyanide Solutions.

Sample	Grind	Agit Time hr	NaCN lb/ton sol.	CaO lb/ton ore	% Recovery (Cumulative)		Consumption lb/ton ore (Cumulative)	
					Au	Ag	NaCN	CaO
Ore	100	12	6.0	3	none	11.7	7.8	6.1
Ore	100	24	6.0	3	none	20.1	9.6	7.7
Ore	100	48	6.0	3	none	23.3	11.7	10.0
Ore	100	72	6.0	3	0.2	28.0	13.5	12.2
Roasted Conc.	100	12	25	3	38.5	27.4	45.3	7.3
Roasted Conc.	100	24	25	3	44.7	33.0	50.5	8.3
Roasted Conc.	100	48	25	3	45.6	37.4	61.0	9.2
Roasted Conc.	100	72	25	3	53.2	43.9	71.5	10.9
Roasted Conc.	200	12	25	3	39.6	26.9	44.4	6.1
Roasted Conc.	200	24	25	3	45.6	31.8	48.9	6.9
Roasted Conc.	200	48	25	3	47.0	35.4	58.5	7.7
Roasted Conc.	200	72	25	3	57.2	43.4	69.6	8.8

Cyanidation Results

The results summarized in Table VII emphasize the following points:

1. A preliminary leaching test upon a minus 100-mesh raw ore sample with 6 lb of NaCN per ton of solution gave practically no recovery in gold and a very low recovery in silver, even for a 72-hour agitation time, whereas the cyanide and lime consumption were prohibitively high.

2. Upon leaching the calcine, only an average of about 50 per cent

recovery of the precious metals was achieved with 72 hours of agitation, whereas the cyanide consumption was exceedingly high.

3. An increase in the fineness of grind of the calcine did not appreciably change the recovery, whereas longer times of agitation tend to increase the already high cyanide consumption.

For these reasons, leaching with cyanide solution was not considered a feasible process from either an engineering or an economic standpoint for both the raw ore and the roasted concentrates.

SMELTER SCHEDULES

As already stated, the aim of the mineral engineer in beneficiating ores is to produce a concentrate which will yield the maximum dollar return per ton of ore mined. Sometimes this economic goal, prime mover of industrial operations, contrasts and is in opposition to a metallurgical treatment based purely on theoretical considerations of scientific character. Here, again, it is the hope of the engineer to apply the acquired scientific knowledge to perform a process which will be profitably feasible. A. B. Parson[8] states that "---- the marketing of anything involves a transaction between a seller, and a buyer, who, assuming that the thing in question is valuable to him, wants to get it for just as little as he possibly can. It is probably well to qualify this statement by pointing out that the buyer always should be looking for future tonnage - which is the very essence of his business - and accordingly he does not necessarily negotiate the purchase of a single lot of ore strictly on the basis of the value of that specific lot."

Generally speaking, in these days, a custom smelter buys ore and concentrates from independent mines under two types of contracts:

1. Open rate contracts usually called "smelter schedules" which will apply indiscriminately to any lot of ore or concentrate sent to a smelter for treatment. The smelter schedule is usually divided into metal payments and in smelting and refining deductions; any ore will be liquidated as a matter of fact under its condition, and the gross payment, minus shipping, assaying, and other eventual charges, remitted to the shipper.

2. Formal contracts, which are stipulated between a shipper and a smelter, usually have a duration of several years. By formal contracts, a custom smelter insures itself against intermittent and sporadic flow of raw materials, and it can plan in advance with better certainty such items as costs of treatment, labor force, metallurgical treatment, and refining processes. On the other hand, the shipper will be able to obtain higher returns from his ore or concentrate, and, if he complies with the terms of the contract, he will receive a steady income with which to plan more easily his mining and milling processes, and more steadily devote efforts to exploratory and developmental work. In this way both parties gain from a reciprocal understanding, and the mining district can be more efficiently exploited.

Since formal contracts are a personal business between buyer and seller, the following marketing investigation will deal exclusively with smelter schedules.

Considering the geographical location of the mine and the characteristics of the ore constituents, the logical area to market the raw ore or the more advantageous concentrates is the north western part of the United States. In this area the custom smelters that will be willing to accept shipments are those located at East Helena, Montana (lead smelter), Anaconda, Montana (copper smelter), Kellog, Idaho (Bunker Hill, lead smelter), and Silver King, Idaho (Sullivan zinc plant).

Abstracts of the open schedules of these plants follows. Because of the variable and fluctuating characteristics of metal prices, the following quotations have been chosen as average for basis of calculations:*

Gold	\$35.00	per ounce
Silver	\$ .905	per ounce
Copper, electrolytic	\$ .2295	@ lb
Lead	\$ .170	@ lb
Zinc (Western)	\$ .160	@ lb

Anaconda Smelter

Payments

Copper: 96% of Cu content, minimum deduction of 10 pounds per ton, at quotation less 2.5¢ per pound.

Silver: 95% of Ag content, minimum deduction one ounce per ton, at quotation less 2¢ per ounce.

Gold: 100% of Au content, at \$31.81825 per ounce.

Treatment Charge (Maximum \$5.50)

Base charge: \$4.00 per dry ton.

Add: 10% of sum of metal payments in excess of \$15.00 per dry ton.

Add: 12¢ for each 1% Fe.

Deduct: 2.5¢ for each 1% SiO_2 in excess of Al_2O_3.

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* Quotation from Engineering and Mining Journal of March 31, 1949.

Bunker Hill Smelter

Payments

Gold: 100% of Au content @ $31.81825.

Silver: 95% of Ag content at quotation.

Lead: No payment under 2.5%.

2.5-25% pay for 90%)
25-50% pay for 91%) at $ .15434 per pound.
over 50% pay for 92%)

Copper: 100% of Cu content, less 1%, at quotation less 6¢ per pound.

Zinc: No payment under 2.5%.

Over 2.5% for 50% of the content at 25% of quotation.

Treatment Charge (Minimum $11.00)

Base Charge: $8.00 per dry ton.

Iron: Credit (debit) base charge for % Fe over (under) %Zn x 1.77 @ 40¢ a unit.

Silica: Credit (debit) base charge for % of SiO_2 over (under) % Fe x 0.886 @ 8¢ a unit.

Lime: Credit (debit) base charge for % CaO over (under) % Fe x 0.979 @ 10¢ a unit.

Maximum credits for Fe, SiO_2, CaO will be $3.00 per ton.

Silver: Charge all in excess of 50 oz per ton @ 1.5¢ per oz.

Antimony: Charge all in excess of 1% @ 50¢ per unit.

Arsenic: Charge all in excess of 1% @ 50¢ per unit.

Bismuth: Charge all in excess of 0.1% of wet Pb content @ 50¢ per lb.

Moisture: Charge all in excess of 10% @ 20¢ per unit.

Sulfur: Charge all in excess of 16% @ 10¢ per unit.

East Helena Smelter

Payments

Gold: 100% of Au content at #31.81825 per ounce.

Silver: 95% of Ag content at quotation, minimum deduction one ounce.

Lead: If more than 3.0%, deduct 1.5 units and pay 90% at quotation, less 1.8¢ per pound.

Copper: If more than 1.0%, deduct 1 unit and pay 100% at quotation, less 6¢ per pound.

Treatment Charge

Base Charge: If Pb is 20% or less, $8.00 per dry ton. Deduct 10¢ per ton for each unit of Pb over 20%.

Arsenic: Charge all in excess of 1% @ 50¢ per unit.

Antimony: Charge all in excess of 1% @ 50¢ per unit.

Bismuth: Charge all in excess of 0.1% of wet Pb content @ 50¢ per pound.

Zinc: Charge all in excess of 10% at 30¢ per unit.

Sullivan Zinc Plant

Payment

Zinc: 80% of Zn content at quotation.

Lead: 80% of Pb content, in excess of 3%, at quotation less 2¢ per pound.

Gold: 80% of Au content, if in excess of 0.02 oz, at $34.2425 per ounce.

Silver: 80% of Ag content, if in excess of 1 oz, at quotation.

Treatment Charge

Base Charge: $46.00 per dry ton; if Zn is under 45%, additional charge of 50¢ per unit of deficiency.

Iron: 30¢ per unit.

Lead: If Pb under 3%, deduct $1.00 per unit of deficiency.

ECONOMIC CONSIDERATIONS

The erection of a mill for the treatment of an ore must be economically justified by the fact that a profit will be derived from a venture of mineral exploitation where less profit or no profit would be present without beneficiation. On the other hand, different treatment methods must be compared to determine which one will yield a more profitable result. Therefore, the following considerations will be taken into account:

1. Computation of the gross return per ton of ore or concentrate from each single custom smelter to determine which smelter will pay more for the same product.

2. Relative transportation costs by trucks and freight from mill to custom smelter.

3. Comparison of the different treatment methods in relation to ton of ore treated f.o.b. smelter.

The Herbert Gold Mine, situated at about 15 miles from Libby, Montana, is accessible by a good gravel road. It is supposed that a mill would be erected in the near vicinity of the mine.

Difficulty was encountered in determining the price of transportation by trucks. Gardner[9] gives as average cost, where the trucks are kept busy, loaded from chutes, and dumped into cars or bins without shoveling, 5 cents per ton-mile for 20 to 100 miles in 1935. By taking into consideration the increase in labor, fuel, and maintenance costs, and in efficiency, a final cost of $3.00 per ton for the round trip was considered reasonable.

Table VIII. Freight Charges of Ores and Concentrates from Libby, Montana, in Dollars per Ton.*

Value of shipment per ton		To Butte, Mont.	To Helena, Mont.	To Bradley ** Idaho
Not exceeding	$25.00	5.35	--	--
	$30.00	--	--	4.43
	$40.00	--	4.60	--
Over	$100.00	7.28	7.28	7.18

* A 3% government tax should be added to flat amount of total bill.

** Nearest switching point for both Bunker Hill Smelter and Sullivan Plant.

Table IX. Complete Analyses of Ore, Concentrates, and Calcine from Different Treatment Processes.

Element	Ore	Test No. 6		Test No. 13	Test No. 17	
		Pb Conc.	Zn Conc.	Bulk Conc.	Bulk Conc.	Calcine
Gold	0.25	0.87	0.54	1.10	1.08	1.64
Silver	19.5	115.1	12.9	105.5	95.2	135.8
Lead	2.2	10.5	0.7	10.5	6.9	13.2
Zinc	1.5	3.6	23.7	9.2	9.1	10.7
Copper	0.18	1.02	0.54	1.05	0.94	1.32
Iron	5.2	11.8	11.8	15.0	21.0	22.7
Arsenic	3.8	13.7	--	15.2	11.1	1.5
Antimony	1.3	14.6	--	15.1	10.7	7.4
Sulfur	4.4	19.9	--	25.3	22.7	2.8
Silica	65.8	15.6	11.7	3.6	7.8	12.2
Lime	1.0	0.2	--	0.2	0.2	0.3
Alumina	3.6	1.1	--	0.7	0.9	1.0

Table VIII shows the freight rates of ores and concentrates from Libby, Montana, to switching points nearest to the reduction plants, as quoted by the office of the Great Northern Railroad in Butte, Montana. It is interesting to note that the freight charges of concentrates, the value of which exceeds $100.00 per ton, are about the same without regard to which plant they might be shipped, in spite of the considerable difference in mileage.

Table X. Smelter Returns in Dollars per Dry Ton.

Smelter	Ore	Test No. 6		Test No. 13	Test No. 17	
		Pb Conc	Zn Conc	Bulk Conc	Bulk Conc	Calcine
Anaconda						
Payment	24.33	126.71	--	125.65	116.24	168.12
Deduction	4.02	5.50	--	5.50	5.50	5.50
Net payment	20.31	121.21	--	120.15	110.74	162.62
Bunker Hill						
Payment	24.71	157.34	--	158.68	139.79	211.44
Deduction	11.00	21.07	--	26.55	20.18	14.07
Net payment	13.71	136.27	--	132.03	119.61	197.39
East Helena						
Payment	24.69	149.74	--	150.14	132.59	207.56
Deduction	9.55	21.15	--	23.15	18.90	11.66
Net payment	15.14	128.59	--	126.99	114.69	195.90
Sullivan Plant						
Payment	--	--	84.14	153.37	--	--
Deduction	--	--	62.59	68.40	--	--
Net payment	None	--	21.55	84.79	--	--

For calculation purposes Table IX shows the complete analysis of the ore, of various flotation concentrates, and of the calcine.

In Table X, which indicates payments, deductions, and net returns per dry ton of ore of the same products of Table IX, the following points are clarified:

1. The copper smelter will pay more per ton of dry ore than the lead smelters because of lower base charge and no penalties for arsenic and antimony.

2. The lead smelters will pay more per ton of dry concentrate and calcine thanthe copper smelter because of the lead 'accounted for', in spite of the greater deductions for arsenic and antimony.

3. Between the two lead smelters, Bunker Hill will pay more than East Helena because of the zinc partially 'accounted for' and of a lower deduction on the silver.

4. Except for the high grade zinc concentrates, the Sullivan Plant is not in a competitive position because of the high base charge.

Thus, logical deduction from this analysis is to send the ore to the Anaconda smelter, the lead concentrate, the bulk concentrate, and the calcine to the Bunker Hill smelter, and the zinc concentrate to the Sullivan Plant.

The next step was to determine if the erection of a mill is justified, and what concentration process would be more profitable; namely, if the yield would be greater by shipping either the untreated ore or the two concentrates from a selective flotation or the concentrate from a bulk flotation.

In order to arrive at a consistent and practical solution of this economic problem the following considerations shall be taken into account:

1. The final smelter return must be referred back to ton of ore treated f.o. b. plant.

2. It will be supposed that the shipped concentrates have 10 per cent moisture, whereas the ore and the calcine will be considered dry.

3. The cost of milling is assumed to be uniformly $1.50 per ton of feed. This uniform milling cost is applied whether selective flotation or bulk flotation is performed. Of course, it is recognized that selective flotation is more expensive because of the greater consumption of electric power and reagents, and of the increased handling.

4. An average over-all ratio of concentration of 6.25 : 1 will be considered.

Table XI. Data from Processing, Transporting, and Marketing 100 Tons of Ore.

Items	Ore	Test No. 6		Test No. 13
		Pb Conc.	Zn Conc.	Bulk Conc.
Tons of dry products	100	13.8	2.2	16
Tons of wet products	-	15.33	2.44	17.78
Plant to which shipped	Anaconda	Bun.Hill	Sullican	Bun.Hill
Trucking	$300	45.99	7.32	53.34
Freight	$535	110.00	17.75	127.66
Total transportation	$835	181.06		181.00
Gross revenue from plant	$2031	1880	48	2113
Net revenue	$1196	1747		1932
Milling costs	$ -	150		150
Total revenue per ton of ore treated f.o.b. plant	$ 11.96	15.97		17.82

From Table XI, which indicates the principal steps in arriving at the net total revenue per ton of ore treated and untreated f.o.b. reduction plant, it is easy to see that the greater revenue is obtained from a bulk concentrate, and that the erection of a mill is fully justified, because the net profit of the concentrating process is $3.86 per ton of feed. This net profit per ton of ore is the basis for financial considerations regarding interest withdrawn on capital investment; depreciation, ammortization and interest on bonds, mill alterations, and supervisory expenses should be included in the milling costs.

The last problem is to determine if the greater return obtained by shipping the calcine to a smelter will warrant the expense and the capital investment of a roasting process. Again, concentrate and derived calcine should be considered on a common basis remembering that the ratio of concentration of the roasting process is 1.48 : 1, and that the concentrate is shipped with 10 per cent moisture, whereas the calcine is dry.

Table XII shows the returns per ton of ore from the culk concentrate and the calcine, both shipped to the Bunker Hill smelter: by processing the concentrate and shipping the calcine, a gross profit of $3.00 per ton of ore treated is obtained, namely at a ratio of concentration of 6.25 : 1, a gross profit of $18.75 per ton of concentrate.

No information is available about the present cost of roasting a highly arsenical concentrate, but from general considerations upon smelting, it is safe to assume that at $18.75 per ton of concentrate, not only the total cost of roasting is covered, but also a highly remunerative profit will be obtained.

Table XII. Data from Shipping to the Bunker Hill Smelter Concentrate and Calcine from 100 Tons of Ore.

Item	Test No. 17	
	Bulk Conc.	Calcine
Tons of dry products	16	10.8
Tons of wet products	17.78	-
Trucking	$ 53.34	32.40
Freight	$ 127.66	66.40
Total transportation	$ 181.00	98.80
Gross revenue	$ 1915	2133
Net Revenue	$ 1734	2034
Revenue per ton of ore treated f.o.b. plant	$ 17.34	20.34

CONCLUSION

From an engineering point of view the investigation performed tends to demonstrate that the character of the ore is very complex, and that standard methods of treatment are not always justified from an economic basis.

The precious metals do not occur in the free state, but are intimately mixed as chemical compounds with the base metal sulfosalts and the only profitable way of recovery seems to be by flotation.

The interrelated and intergrown minerals require very fine grinding in order to obtain a degree of liberation such that will permit a reasonable separation of wanted and unwanted mineral species. This fact united with the considerable hardness of the ore require a long grinding-time.

Economic considerations point to the fact that one bulk concentrate will be more profitably marketed than a lead and a zinc concentrate from selective flotation. The gain is of approximately two dollars per ton of ore f.o.b. Bunker Hill smelter, which give the highest net returns.

Because of the considerable deductions caused by the high arsenic and antimony assays of the concentrates, their elimination as oxides by roasting will give a calcine whose marketing at the Bunker Hill smelter will yield still higher profits per ton of ore treated.

RECOMMENDATION

In the recommendations for the construction of a mill to process the ore from the Herbert Gold Mine, no mention will be made either to the tonnage to be treated or the specific type of machinery and apparatus to be employed.

From the conclusion drawn it seems that the best method of recovering as much as possible of precious and base metals from the ore is to conform to the following scheme.

The run-of-mine ore, after being crushed in a jaw crusher, would be introduced in a rod mill to which water is added together with 2 lb of $CuSO_4$ and 1.5 lb of CaO per ton of ore. The rod mill should discharge in the pool of a classifier in close circuit with a ball mill for fine grinding. To the overflow of the classifier, 63% minus 200-mesh, 0.2 lb of Minerec and 0.2 lb of amyl xanthate per ton of ore will be added as collectors, and the pulp floated in a series of rougher flotation cells with pine oil as needed. The rougher tailing should be wasted, whereas the rougher concentrate will be introduced in the pool of a classifier in close circuit with a tube mill for further grinding. From the classifier the pulp should be charged to a series of cleaner flotation cells in which the concentrate will be cleaned three times with a consumption of 0.3 lb of CaO, 1.5 lb of Na_2SiO_3, 0.05 lb of minerec per ton of ore, and pine oils as needed. The pH should be kept at above 9.2 in both rougher and cleaner operations. The cleaner tails should be recirculated to the rougher flotation cells, whereas the final concentrate, after being thickened, will be filtered and shipped to the Bunker Hill smelter.

The construction of a roasting plant and bag house is not recommended with the initial erection of the mill. Only after the mill has reached continuous and normal operations, further roasting tests on representative samples of average concentrates will indicate if the erection of a roasting plant is warranted from an economic standpoint.

Further considerations for the erection of a mill and roasting plant must be derived from the rate of mine exploitation and from the amount of ore present, since the capital investment must be completely ammortized during the expected life of the mine.

APPENDIX

This appendix comprehends in tabulated form the results from the gravity concentration and flotation tests together with assays, per cent distribution, reagent consumption, and remarks from each of the tests performed.

TABLE XIII

Tabling Test No. 1

Assay of Heads: 0.25 oz Au; 19.5 oz Ag; 2.2% Pb; 1.5% Zn; 5.2% Fe; 73.1% Ins

Products	% Weight.	Assay				% Distribution			
		Au	Ag	Pb	Zn	Au	Ag	Pb	Zn
Concentrate	3.6	1.62	81.4	6.8	6.9	23.7	15.4	14.2	14.4
Middlings	59.8	0.18	13.7	1.8	1.8	43.6	43.0	62.4	62.3
Tailings	36.6	0.22	21.6	1.1	1.1	32.7	41.6	23.4	23.3
Comp. (calc)		0.24	19.1	1.7	1.7				

Ratio of Concentration: 27.8 : 1.

Reagents in pounds per ton:

Reagent						

Remarks:

Three batches of minus 10-mesh ore were ground for 20 minutes in a laboratory rod mill; the pulps were dewatered, combined, dried and tabled. No satisfactory results were obtained.

TABLE XIV

Jigging Test No. 1

Assay of Heads: 0.25 oz Au; 19.5 oz Ag; 2.2% Pb; 1.5% Zn; 5.2% Fe; 73.1% Ins

Products	% Weight.	Assay				% Distribution			
		Au	Ag	Pb	Zn	Au	Ag	Pb	Zn
Concentrate	7.8	0.90	52.1	5.8	5.0	29.6	22.3	21.3	24.4
Tailings	92.2	0.18	15.2	1.8	1.3	70.4	77.7	78.7	75.6
Comp. (calc)		0.23	18.1	2.1	1.6				

Ratio of Concentration: 12.8 : 1.

Reagents in pounds per ton:

Reagent					

Remarks:

Three batches of minus 10-mesh ore were ground for 10 minutes in a laboratory rod mill; the pulps were dewatered, combined, and jigged. No satisfactory results were obtained.

TABLE XV

Flotation Test No. 1

Assay of Heads: 0.25 oz Au; 19.5 oz Ag; 2.2% Pb; 1.5% Zn; 5.2% Fe; 73.1% Ins

Products	% Weight.	Assay				% Distribution			
		Au	Ag	Pb	Zn	Au	Ag	Pb	Zn
Cl. Conc.	18.73	0.96	82.1	9.0	7.6	77.1	79.5	81.0	62.7
Cl. Tails	8.70	0.20	13.0	1.2	2.2	7.4	5.9	5.1	8.5
Rough T.	72.57	0.05	3.9	0.4	0.9	15.5	14.6	13.9	28.8
Comp. (calc)		0.23	19.3	2.1	2.2				

Ratio of Concentration: 5.35 : 1.

Reagents in pounds per ton:

Reagent	to Mill	to Rough	to Clean		
Minerec A	0.2				
Areofloat 25	0.05	0.05			
Na_2CO_3	1.5	1.5			
B-24		0.02			
Barret 4		0.02	0.01		
712		0.01			
Pine oil		0.02	0.02		

Remarks:

In rougher circuit the pH was 8.0. Good collection, but brittle froth. Addition of reagent 712 improved froth, but probably floated some quartz.

Too much quartz in cleaner concentrate.

TABLE XVI

Flotation Test No. 2

Assay of Heads: 0.25 oz Au; 19.5 oz Ag; 2.2% Pb; 1.5% Zn; 5.2% Fe; 73.1% Ins

Products	% Weight.	Assay				% Distribution			
		Au	Ag	Pb	Zn	Au	Ag	Pb	Zn
Pb R. Conc.	13.5	0.84	104.2	6.3	3.0	43.8	71.5	69.0	24.2
Zn R. Conc.	12.3	0.76	28.0	1.3	9.7	36.2	17.6	13.0	71.4
Rougher T.	74.2	0.07	2.9	0.3	0.1	20.0	10.9	18.0	4.4
Comp. (calc.)		0.26	19.7	1.3	1.7				

Ratio of Concentration: 7.4 : 1

Reagents in pounds per ton:

Reagent	to Mill	to Pb R.	to Zn Cor.	to Zn R.	
CaO	1.5				
NaCN	0.25				
$ZnSO_4$	1.0				
Pine Oil		0.01	0.01		
Barret 4		0.03			
Z-3		0.2	0.1		
Minerec A		0.2			
$CuSO_4$				1.0	

Remarks:

In rougher circuit the pH was 8.5. Very difficult froth in Pb rougher. Good froth in Zn circuit.

TABLE XVII

Flotation Test No. 3

Assay of Heads: 0.25 oz Au; 19.5 oz Ag; 2.2% Pb; 1.5% Zn; 5.2% Fe; 73.1% Ins

Products	% Weight.	Assay				% Distribution			
		Au	Ag	Pb	Zn	Au	Ag	Pb	Zn
Pb Cl. Con.	10.6	0.82	134.6	6.8	3.8	32.3	68.6	57.3	22.6
Zn Cl. Con.	7.1	0.95	32.1	1.1	13.4	25.2	11.0	6.2	53.6
Pb Cl. Tails	7.1	0.63	19.2	1.9	2.5	16.7	6.6	10.8	10.0
Zn Cl. Tails	9.0	0.48	13.8	1.4	2.0	16.0	5.9	10.0	10.6
Rougher T.	66.2	0.04	2.5	0.3	0.1	9.8	7.9	15.7	3.7
Comp. (calc)		0.27	20.8	1.3	1.8				

Ratio of Concentration: 5.62 : 1.

Reagents in pounds per ton:

Reagent	to Mill	to Pb R.	to Zn Con.	to Zn R.	
CaO	1.4				
NaCN	.25				
$ZnSO_4$	1.0				
Minerec A		0.2			
Z-3		0.05		0.05	
Pine Oil		0.02		0.01	
$CuSO_4$			1.2		

Remarks:

pH maintained at 9.2. Zinc floated quite slowly; it seems that a stronger collector is needed. The finer liberated particles remained in the cleaner tailings.

Concentrates cleaned twice.

TABLE XVIII

Flotation Test No. 4

Assay of Heads: 0.25 oz Au; 19.5 oz Ag; 2.2% Pb; 1.5% Zn; 5.2% Fe; 73.1% Ins

Products	% Weight.	Assay				% Distribution			
		Au	Ag	Pb	Zn	Au	Ag	Pb	Zn
Pb R. Conc.	13.7	0.64	82.6	6.8	2.7	32.4	56.1	53.2	20.8
Zn R. Conc.	22.9	0.66	31.0	2.9	5.4	57.8	36.4	39.2	71.8
Rougher T.	63.8	0.04	2.3	0.2	0.2	9.8	7.5	7.6	7.4
Comp. (calc)		0.26	19.5	1.7	1.7				

Ratio of Concentration: 7.3 : 1

Reagents in pounds per ton:

Reagent	to Mill	to Pb R.	to Zn Con.	to Zn R.	
CaO	1.2				
NaCN	0.25				
$ZnSO_4$	1.0				
Areofloat 25		0.02			
Minerec A		0.1			
Pine Oil		0.01		0.01	
$CuSO_4$			1.2		
Areofloat 208				0.1	

Remarks:

The pH of the rougher tails was 7.8. Faster collection of Zn than in the preceding test.

TABLE XIX

Flotation Test No. 5

Assay of Heads: 0.25 oz Au; 19.5 oz Ag; 2.2% Pb; 1.5% Zn; 5.2% Fe; 73.1% Ins

Products	% Weight.	Assay				% Distribution			
		Au	Ag	Pb	Zn	Au	Ag	Pb	Zn
Pb Cl. Conc.	13.3	0.86	122.8	9.2	2.8	46.4	85.9	82.5	23.7
Zn Cl. Conc.	2.0	0.84	26.4	1.4	24.8	6.8	2.8	1.9	30.8
Pb Cl. Tail	6.9	0.32	11.4	1.4	2.2	8.9	4.1	6.3	9.6
Zn Cl. Tail	15.2	0.41	7.3	0.5	2.9	25.3	3.3	5.1	28.0
Rougher T.	62.6	0.05	1.2	0.1	0.2	12.6	3.9	4.2	7.9
Comp. (calc)		0.25	19.1	1.5	1.6				

Ratio of Concentration: 6.55 : 1.

Reagents in pounds per ton:

Reagent	to Mill	to Pb R.	to Pb Cl.	to Zn Con.	to Zn Cl.
CaO	1.2	1.3			
NaCN	.25				
$ZnSO_4$	1.0				
Z-4		0.2			
Pine Oil		0.01	0.01		0.01
$CuSO_4$				1.5	
Na_2SiO_3		0.5		0.5	0.5

Remarks:

pH of lead circuit was 8.6; of Zn circuit it was 7.5. Pb concentrate cleaned 2 times; Zn concentrate cleaned 3 times. Probably some $CuSO_4$ added at the first Zn cleaner could have improved the recovery.

TABLE XX

Flotation Test No. 6

Assay of Heads: 0.25 oz Au; 19.5 oz Ag; 2.2% Pb; 1.5% Zn; 5.2% Fe;
73.1% Ins

Products	% Weight.	Assay				% Distribution			
		Au	Ag	Pb	Zn	Au	Ag	Pb	Zn
Pb Cl. Con.	13.8	0.87	115.1	10.5	3.6	50.9	83.5	74.8	29.2
Zn Cl. Con.	2.2	0.54	12.9	0.7	23.7	5.0	1.5	0.8	30.2
Pb Cl. Tail	15.5	0.40	12.0	2.5	2.8	26.2	9.8	20.0	25.4
Zn Cl. Tail	8.7	0.28	3.9	0.3	2.3	10.3	1.8	1.4	11.7
Rougher T.	59.8	0.03	1.1	0.1	0.1	7.6	3.4	3.0	3.5
Comp. (calc)		0.24	19.1	1.9	1.7				

Ratio of Concentration: 6.25 : 1.

Reagents in pounds per ton:

Reagent	to Mill	to Pb R.	to Zn Con	to Zn Cl.	to Pb Cl.
CaO	2.5				
NaCN	0.25				
$ZnSO_4$	1.0				
Minerec A		0.2			
Z-4		0.2			
Pine Oil		0.06		0.02	0.01
$CuSO_4$			1.5	0.5	
Na_2SiO_3			0.5	0.5	1.0
Na Areofloat B				0.2	

Remarks:

Buffered with acetic acid in Zn circuit to a pH of 7.2. Concentrates cleaned 3 times. Relatively great amount of pine oil used: good froth. Table IX gives the complete analysis of the concentrates.

TABLE XXI

Flotation Test No. 7

Assay of Heads: 0.25 oz Au; 19.5 oz Ag; 2.2% Pb; 1.5% Zn; 5.2% Fe; 73.1% Ins

Products	% Weight.	Assay				% Distribution			
		Au	Ag	Pb	Zn	Au	Ag	Pb	Zn
Cl. Conc.	18.6	1.00	83.0	7.1	6.2	72.8	84.3	85.2	80.7
Cl. Tails	11.4	0.30	14.7	1.4	1.8	13.5	9.2	10.3	14.4
Rougher T.	70.0	0.05	1.7	0.1	0.1	13.7	6.5	4.5	4.9
Comp. (calc)		0.26	18.3	1.55	1.4				

Ratio of Concentration: 5.37 : 1.

Reagents in pounds per ton:

Reagent	to Mill	to Cond.	to Rougher	to Clean	
CaO	1.0				
$CuSO_4$		2.0		0.3	
$K_2Cr_2O_7$		1.0		0.2	
Pine Oil			0.02	0.02	
Na_2SiO_3				0.5	

Remarks:

Depression of lead and float of zinc was tried with $K_2Cr_2O_7$. The initial pH was 9.0: because of no good collection H_2SO_4 was added to the rougher to give a pH of 5.3. The pH of the cleaners was 6.5. Both Pb and Zn were floated.

TABLE XXII

Flotation Test No. 8

Assay of Heads: 0.25 oz Au; 19.5 oz Ag; 2.2% Pb; 1.5% Zn; 5.2% Fe; 73.1% Ins

Products	% Weight.	Assay				% Distribution			
		Au	Ag	Pb	Zn	Au	Ag	Pb	Zn
Cl. Conc.	10.8	1.32	78.9	4.7	8.3	52.4	48.4	30.3	64.1
Cl. Tails	17.1	0.42	24.6	2.2	2.1	26.4	23.8	22.4	25.6
Rougher T.	72.1	0.08	6.8	1.1	0.2	21.2	27.8	47.3	10.3
Comp. (calc)		0.27	17.6	1.8	1.4				

Ratio of Concentration: 9.25 : 1.

Reagents in pounds per ton:

Reagent	to Mill	to Cond.	to Rough.	to Clean	
CaO	1.0				
$CuSO_4$		2.0		0.2	
$NaPO_4$		1.0		1.0	
Na Areofloat B			0.2		
Na_2SiO_3				0.5	

Remarks:

The lead was depressed with the phosphate at a pH of 9.2, but also the silver was depressed. Good gold concentrate.

TABLE XXIII

Flotation Test No. 9

Assay of Heads: 0.25 oz Au; 19.5 oz Ag; 2.2% Pb; 1.5% Zn; 5.2% Fe;
73.1% Ins

Products	% Weight.	Assay				% Distribution			
		Au	Ag	Pb	Zn	Au	Ag	Pb	Zn
Cl. Conc.	14.4	1.12	111.5	9.5	9.2	64.1	79.5	68.4	79.4
Cl. Tails	14.4	0.48	21.3	3.4	1.4	27.4	15.2	24.5	12.1
Rougher T.	71.2	0.03	1.5	0.2	0.2	8.5	5.3	7.1	8.5
Comp. (calc)		0.25	20.2	2.0	1.7				

Ratio of Concentration: 6.95 : 1.

Reagents in pounds per ton:

Reagent	to Mill	to Cond.	to Rough.	to Clean	
CaO	8.0				
$CuSO_4$		2.0			
Na Aerofloat B			0.2		
Minerec A			0.15		
Pine Oil			0.02	0.03	
Na_2SiO_3				1.5	

Remarks:

Additional Assays: Cl. Conc.: 13.3% Fe, 10.0% Ins
Rougher T: 1.5% Fe.

Futile attempt to depress iron at a pH of 12.2. Concentrate cleaned 3 times.

TABLE XXIV

Flotation Test No. 10

Assay of Heads: 0.25 oz Au; 19.5 oz Ag; 2.2% Pb; 1.5% Zn; 5.2% Fe; 73.1% Ins

Products	% Weight.	Assay				% Distribution			
		Au	Ag	Pb	Zn	Au	Ag	Pb	Zn
Cl. Conc.	11.8	1.20	104.4	9.6	10.2	53.8	65.0	57.9	74.1
Cl. Tails	13.4	0.65	39.1	5.0	2.0	32.8	27.8	34.4	16.6
Rougher T.	74.8	0.05	1.8	0.2	0.2	14.4	7.2	7.7	9.3
Comp. (calc)		0.26	18.9	2.0	1.6				

Ratio of Concentration: 8.48 : 1.

Reagents in pounds per ton:

Reagent	to Mill	to Rough	To Clean		
CaO	10.0		1.0		
$CuSO_4$	2.0				
Minerec A		0.2			
Z-4		0.2			
Pine Oil		0.02	0.05		
Na_2SiO_4			1.0		

Remarks:

Additional assays: Cl. Conc.: 13.6% Fe: 8.4% Ins
Rougher T.: 1.5% Fe.

Futile attempt to depress the iron at a pH of 12.4. Concentrate cleaned three times. Probably some minerec in cleaner will improve the silver grade of the concentrate.

TABLE XXV

Flotation Test No. 11

Assay of Heads: 0.25 oz Au; 19.5 oz Ag; 2.2% Pb; 1.5% Zn; 5.2% Fe; 73.1% Ins

Products	% Weight	Assay				% Distribution			
		Au	Ag	Pb	Zn	Au	Ag	Pb	Zn
Cl. Conc.	16.3	1.04	93.4	9.0	8.6	56.2	83.5	76.6	85.5
Cl. Tails	14.1	0.82	15.1	2.7	1.2	38.1	11.6	19.8	10.3
Rougher T.	69.6	0.02	1.3	0.1	0.1	5.7	4.9	3.6	4.2
Comp. (calc)		0.30	18.3	1.9	1.6				

Ratio of Concentration: 6.14 : 1.

Reagents in pounds per ton:

Reagent	to Mill	to Rough	to Clean		
CaO	12.0		1.0		
$CuSO_4$	2.0				
Minerec A		0.2	0.2		
Pine Oil		0.02	0.02		
Z-5		0.2			
Na_2SiO_3			1.5		

Remarks:

Additional assays: Cl. Conc.: 14.1% Fe; 14.5% Ins.
Rougher T.: 1.1% Fe.

Futile attempt to depress the iron at a pH of 13.4. Concentrate cleaned three times.

TABLE XXVI

Flotation Test No. 12

Assay of Heads: 0.25 oz Au; 19.5 oz Ag; 2.2% Pb; 1.5% Zn; 5.2% Fe; 73.1% Ins

Products	% Weight.	Assay				% Distribution			
		Au	Ag	Pb	Zn	Au	Ag	Pb	Zn
Cl. Conc.	10.5	0.78	122.8	8.5	10.0	34.3	70.2	58.7	76.3
Cl. Tails	21.5	0.62	20.4	2.6	1.2	55.8	23.9	36.8	18.8
Rougher T.	68.0	0.03	1.6	0.1	0.1	9.9	5.9	4.5	4.9
Comp. (calc)		0.24	18.3	1.5	1.3				

Ratio of Concentration: 9.53 : 1.

Reagents in pounds per ton:

Reagent	to Mill	to Rough	to Clean		
CaO	10.0		1.0		
$CuSO_4$	2.0				
Minerec		0.2	0.2		
Areofloat 25		0.02			
Areofloat 31		0.02			
NaCN			0.1		
Na_2SiO_3			4.0		
Pine Oil			0.01		

Remarks:

Additional assays: Cl. Conc.: 10.0% Fe; 12.4% Ins.
Rougher T.: 1.3% Fe.

Futile attempt to depress the iron at a pH of 12.0 and with different collectors, and to depress the gangue with great amount of sodium silicate. Probably the rougher concentrates should be reground to improve the liberation.

TABLE XXVII

Flotation Test No. 13

Assay of Heads: 0.25 oz Au; 19.5 oz Ag; 2.2% Pb; 1.5% Zn; 5.2% Fe; 73.1% Ins

Products	% Weight.	Assay				% Distribution			
		Au	Ag	Pb	Zn	Au	Ag	Pb	Zn
Cl.Conc.	12.5	1.10	105.5	10.5	9.2	54.9	66.3	70.7	68.1
2nd Cl. Tail	7.2	0.62	29.2	3.5	1.7	18.1	10.7	13.8	7.4
1st Cl. Tail	11.7	0.24	10.7	1.1	1.5	11.3	6.3	7.0	10.5
Rougher T.	78.6	0.05	4.2	0.2	0.3	15.7	16.7	8.5	14.0
Comp. (calc)		0.24	18.9	2.0	1.6				

Ratio of Concentration: 8.04 : 1.

Reagents in pounds per ton:

Reagent	to Mill	to Rough	to Clean		
CaO	1.5		0.3		
$CuSO_4$	2.0				
Minerec A		0.2	0.05		
Z-4		0.2			
Na_2SiO_3			1.5		
Pine Oil		0.02	0.05		

Remarks:

Additional assays: Cl. Conc.: 15.0% Fe; 6.6% Ins.
Rougher T.: 2.2% Fe.

The rougher concentrates from two tests were combined and reground in a laboratory rod mill for 12 minutes. This process gave a better silica depression. The pH of both rougher and cleaner operations was 9.3. The concentrates were cleaned three times. Table IX gives the complete analysis of the concentrate.

TABLE XXVIII

Flotation Test No. 14

Assay of Heads: 0.25 oz Au; 19.5 oz Ag; 2.2% Pb; 1.5% Zn; 5.2% Fe; 73.1% Ins

Products	% Weight	Assay				% Distribution			
		Au	Ag	Pb	Zn	Au	Ag	Pb	Zn
Cl. Conc.	12.2	1.04	117.8	9.0	6.8	50.0	73.6	69.1	53.2
Cl. Tails	9.5	0.59	29.4	3.5	1.9	22.1	14.3	21.0	11.6
Rougher T.	78.3	0.09	3.0	0.2	0.7	27.9	12.1	9.0	35.2

Ratio of Concentration: 8.2 : 1.

Reagents in pounds per ton:

Reagent	to Mill	to rough	to Clean
CaO	2.0	1.0	1.0
LAmHCL		0.08	
Pine Oil		0.01	0.02

Remarks:

Additional assays: Cl. Conc.: 14.8% Fe; 9.0% Ins
Rougher T.: 2.5% Fe.

The pH was kept at 10.4 in order not to float the silicates. Too much froth in rougher; good collection; concentrate cleaned three times.

TABLE XXIX

Flotation Test No. 15

Assay of Heads: 0.25 oz Au; 19.5 oz Ag; 2.2% Pb; 1.5% Zn; 5.2% Fe; 73.1% Ins

Products	% Weight.	Assay				% Distribution			
		Au	Ag	Pb	Zn	Au	Ag	Pb	Zn
Cl. Conc.	15.1	1.04	101.4	7.0	8.8	62.0	80.0	77.9	83.0
Cl. Tails	9.3	0.55	17.6	1.6	1.3	20.2	8.6	10.9	7.5
Rougher T.	75.6	0.06	2.8	0.2	0.2	17.8	11.4	11.2	9.5

Ratio of Concentration: 6.64 : 1.

Reagents in pounds per ton:

Reagent	to Mill	to Cond.	to Rough.	to Clean.	
CaO	2.0		1.0	1.0	
LAmHCl		0.08			
$CuSO_4$		1.0		0.3	
Pine Oil			0.03		
Na_2SiO_3				1.0	

Remarks:

Additional assays: Cl. Conc.: 13.0% Fe; 12.0% Ins
Rougher T.: 2.1% Fe.

No iron depression at a pH of 11.0. Remarkable good results for amine flotation.

TABLE XXX

Flotation Test No. 16

Assay of Heads: 0.25 oz Au; 19.5 oz Ag; 2.2% Pb; 1.5% Zn; 5.2% Fe;
73.1% Ins

Products	% Weight.	Assay				% Distribution			
		Au	Ag	Pb	Zn	Au	Ag	Pb	Zn
Cl. Conc.	8.0	0.81	150.6	10.6	5.0	25.7	62.9	48.8	25.2
Cl. Tails	18.0	0.67	25.4	3.3	2.9	47.9	23.9	34.2	32.9
Rougher T.	74.0	0.09	3.4	0.4	0.9	26.4	13.2	17.0	41.9

Ratio of Concentration: 12.5 : 1.

Reagents in pounds per ton:

Reagent	to Mill	to Rough	To Clean		
CaO	2.2	1.0	0.8		
LAmHCL		0.05			
Pine Oil		0.01	0.02		
Na_2SiO_3			1.0		

Remarks:

This test demonstrates the necessity of adding copper sulfate for a good collection of the zinc and of minerec for the complete recovery of the silver.

TABLE XXXI

Flotation Test No. 17

Assay of Heads: 0.25 oz Au; 19.5 oz Ag; 2.2% Pb; 1.5% Zn; 5.2% Fe; 73.1% Ins

Products	% Weight.	Assay				% Distribution			
		Au	Ag	Pb	Zn	Au	Ag	Pb	Zn
Cl. Conc.	16.8	1.08	95.2	6.9	9.1	79.8	80.6	71.5	76.2
2nd Cl. Tail	6.0	0.32	17.3	2.1	1.7	8.5	5.2	7.8	5.1
1st Cl. Tail	13.1	0.08	7.3	1.1	1.4	4.6	4.8	8.9	9.1
Rougher T.	64.1	0.02	2.9	0.3	0.3	7.1	9.4	11.8	9.6
Comp.(calc)		0.22	19.8	1.6	2.0				

Ratio of Concentration: 5.95 : 1.

Reagents in pounds per ton:

Reagent	to Mill	to Rough	To Clean		
CaO	1.5		0.3		
$CuSO_4$	2.0				
Minerec A		0.2	0.05		
Z-5		0.2			
Na_2SiO_3			1.5		
Pine Oil		0.02	0.05		

Remarks:

Twelve single batches of the general condition of test No. 13 with regrinding of the rougher concentrates and triple cleaning, for subsequent roasting and leaching tests. The complete assay of the concentrate is reported in Table IX.

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