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MILLING METHODS AND COSTS AT THE LEAD-ZINC CONCENTRATOR OF THE TREADWELL YUKON CO., LTD., AT THE TYBO, NEV.

BY

W. H. BLACKBURN

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MILLING METHODS AND COSTS AT THE CONCENTRATOR OF THE TREADWELL YUKON CO., LTD., AT TYBO, NEV.[1]

By W. H. Blackburn[2]

INTRODUCTION

This paper describing the milling practice at the Tybo concentrator is one of a series of similar papers being prepared by the United States Bureau of Mines on milling methods and costs at the various mills in the United States.

ACKNOWLEDGMENT

A large part of this paper was prepared by H. M. Lewers, mill superintendent of the Tybo plant of the Treadwell Yukon Co., Ltd.

LOCATION

The Tybo mine and concentrator are situated in Nye County, Nev., 70 miles northe of Tonopah, which is the nearest railroad station and general supply center. The distric is accessible by automobile stage either from Tonopah or Ely over an improved gravel highway connecting the two towns. The climate is semiarid, and operations are conducted the year around without difficulty. The elevation at the collar of the shaft is 6,820 feet above sea level.

GENERAL

The concentrator is built on a hillside adjacent to the main hoisting shaft of the mine and was designed and constructed for an all-flotation flow sheet. Two classes of concentrates are produced; lead concentrates which average 90.43 ounces of silver per ton, 62.75 per cent of lead, and 3.50 per cent of zinc; and zinc concentrates which average 13.65 ounces of silver per ton, 2.71 per cent of lead, and 46.57 per cent of zinc. The lead concentrates are shipped to the American Smelting and Refining Co.'s reduction plant at Selby, Calif., and the zinc concentrates to the Sullivan electrolytic zinc plant at Kellogg, Idaho.

All supplies for the mine and mill are hauled from Tonopah in 10-ton capacity Fageol trucks, each equipped with a trailer. The company maintains a rail siding, warehouse, and loading facilities at Tonopah, from which place the concentrates are shipped. The combined capacity of a truck and trailer is about 20 tons of concentrates. A round trip between Tybo and Tonopah requires about 13 hours.

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1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is ad "Reprinted from U. S. Bureau of Mines Information Circular 6430."

2 - One of the consulting engineers, U. S. Bureau of Mines, and manager, Nevada Mines, Treadwell Yukon Co., Ltd., Tybo, Nev.

The mill water is obtained from the mine, and as the supply is more than sufficient to meet the concentrator requirements, mill water is not reclaimed. The domestic water supply is obtained from a small creek above the camp.

Power is purchased from the Nevada-California Power Co. The Treadwell Yukon Co. constructed a branch line from Manhattan to Tybo, a distance of 40 miles, at a cost of about $1,000 per mile. Power is transmitted to Tybo at 55,000 volts, and is stepped down to 440 volts for mine and mill use.

ORE TREATED

The ores treated are complex in character, containing lead, zinc, iron, silver, and gold. The Tybo mine from which all the ores for the concentrator are derived, has been opened to a depth of 976 feet by a vertical three-compartment shaft. The principal vein has an average width of 5 feet, a maximum width of 30 feet, and an average dip of 76°. Mining is done by the shrinkage-stope method without artificial support, except in a few places where stulls are necessary to support a weak hanging wall, and by the stope-and-fill method where walls are soft.

A chemical analysis of a composite sample of the ores between the 710 and 400 foot levels of the mine follows.

Chemical analysis of composite sample of ore between the 710 and 400 foot levels

	Per cent	Ounces per ton
Silica	32.05	-
Iron	15.08	-
Alumina	11.48	-
Calcium oxide	7.73	-
Magnesium oxide	1.62	-
Phosphorous	0.24	-
Sulphur	16.56	-
Lead	7.38	-
Zinc	5.56	-
Copper	0.02	-
Nickel	Trace	-
Cobalt	None	-
Vanadium	None	-
Manganese	0.26	-
Barium sulphate	None	-
Arsenic	0.59	-
Antimony	0.13	-
Cadmium	0.06	-
Gold	-	0.02
Silver	-	10.81

Although most of the silver content in Tybo ore is associated with the galena, the sphalerite and pyrite are both argentiferous, the relative content in the last two minerals being greater in the upper-level ores than in the lower-level ores. Typical assays of pure minerals give the following results:

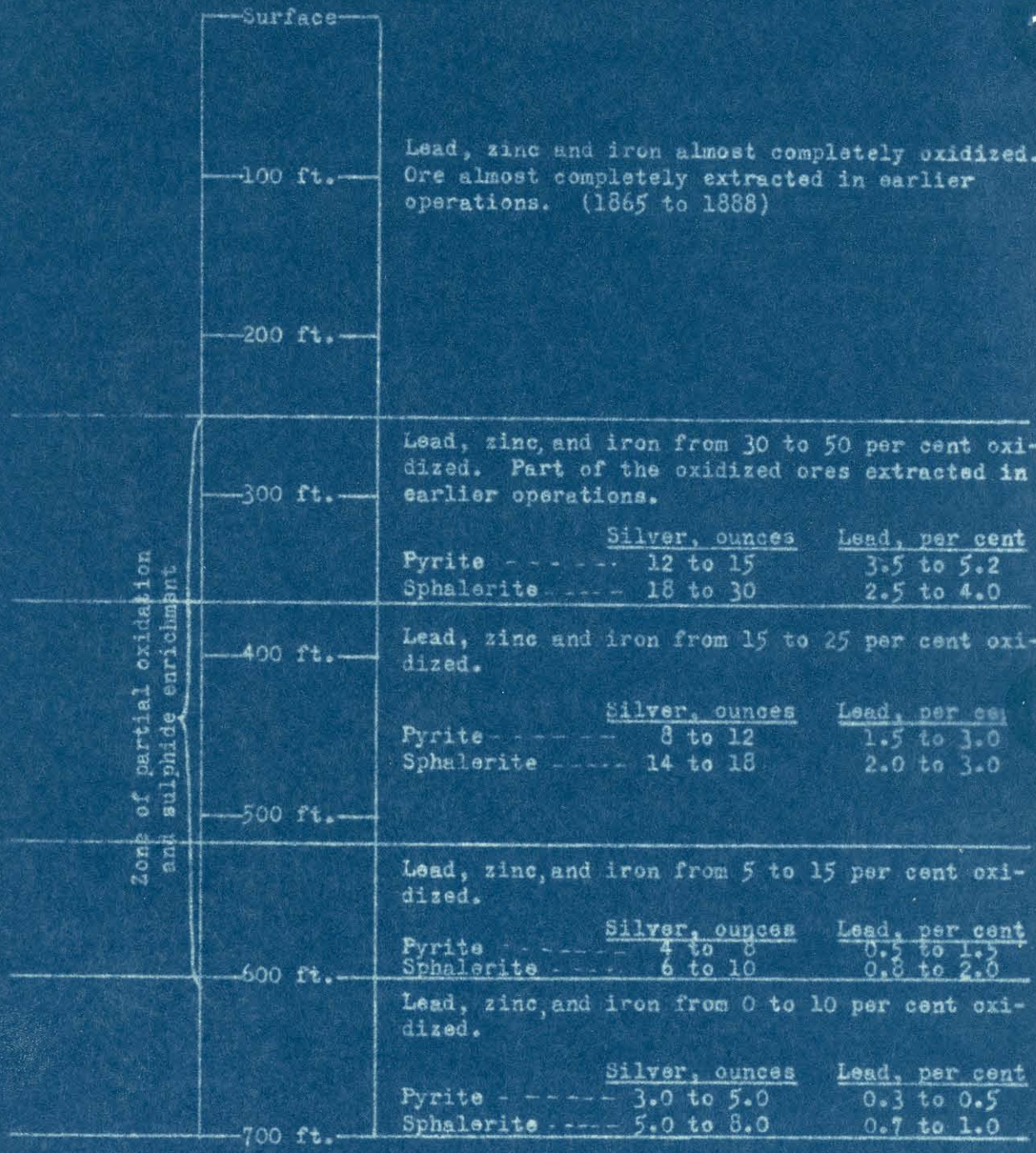

Figure 1.--Vertical section of the Tybo vein, showing approximate degrees of oxidation of the ore body, and varying amounts of lead and silver associated with the pyrite and sphalerite in the different horizons

	Ounces per ton
Galena	110.0 to 130.0
Sphalerite (upper levels)	12.0 to 20.0
Sphalerite (lower levels)	7.0 to 10.0
Pyrite (upper levels)	8.0 to 11.0
Pyrite (lower levels)	3.0 to 6.0

Figure 1 shows a vertical section of the Tybo vein and gives the approximate degrees of oxidation in the ore body and the varying amounts of lead and silver associated with the pyrite and sphalerite in the different horizons.

Some oxidation of the ore is observed on the 710-foot level of the mine and this oxidation gradually increases towards the surface. Above the 400-foot level the ores have undergone considerable oxidation. While practically all of the completely oxidized ores were extracted in former mining operations confined to the area above the 400-foot level, large bodies of partly oxidized ores were left. These partly oxidized ores have comprised a considerable proportion of the concentrator feed up to the present time. The average of seven typical determinations on ores above the 400-foot level is as follows:

	Per cent
Lead in sulphide form	4.4
Lead in oxidized form	3.3
Total lead	7.7

An excessive amount of primary slimes, associated mainly with the partly oxidized ores from the upper levels of the mine, has caused considerable difficulty in concentrator treatment. These slimes are colloidal in character and if present in the mill feed above a critical amount have a tendency to upset completely the equilibrium of flotation operations and to cause lower-grade concentrates and lower recoveries. A large amount of slime in the flotation circuit produces excessive frothing, and when this takes place slime can not be prevented from floating with the concentrates, thus lowering the grade of both lead and zinc concentrates from 5 to 10 per cent in lead and zinc respectively. To overcome this difficulty the ores from the different levels of the mine are mixed as much as is practicable.

The relative distribution of metals in the coarse and fine sizes of mine ore is illustrated by the following analyses of products obtained by screening a sample of ore taken at the shaft bin through a 200-mesh sieve. The minus 200-mesh material amounted to 10.4 per cent of the total weight.

Analyses of ore samples taken at shaft bin and screened through a 200-mesh sieve

	Weight, per cent	Silver, ounces per ton	Lead, per cent	Zinc, per cent	Iron, per cent
Plus 200-mesh	89.6	11.2	7.1	5.0	13.1
Minus 200-mesh	10.4	13.0	15.2	5.9	7.7

The sphalerite so far treated is high in chemically combined iron, pure specimens containing from 50 to 55 per cent of zinc, and 10 to 15 per cent of iron, the iron replacing the zinc isomorphously in accordance with the formula (ZnFe)S. (Zinc sulphide containing 10 per cent or more iron is usually classified as "marmatite".)

BRIEF HISTORY OF CONCENTRATOR OPERATIONS

Ore was first discovered in the Tybo district in 1869. In 1872 the Tybo Consolidated Mines Co. erected a lead smelter, operating two furnaces with a combined capacity of 80 tons per day. Locally burned charcoal was used as fuel. In 1879 the smelter closed down. Soon afterwards a 20-stamp mill was erected, and the ores were treated for their silver content by the "Reece River Process" - that is, pan amalgamation preceded by a chloridizing roast. The records indicate a recovery of 78 to 81 per cent of the silver on ores which averaged from 25 to 30 ounces of silver per ton. This plant operated until 1888, when it was abandoned, due to the exhaustion of the oxidized ores which were amenable to this process.

In 1917, the Louisiana Consolidated Mining Co. acquired the properties and erected a flotation concentrator and a lead smelter. This venture was unsuccessful, due to the high iron and zinc contents of the ores. No method was available at that time for the separation of these minerals. This plant closed down in 1920.

In 1925 the properties were optioned to the Treadwell Yukon Co. After extensive exploration work the option was exercised. Construction of the concentrator was started in November, 1928, and the mill began to produce on May 13, 1929.

PRESENT CONCENTRATOR METHODS

The general flow sheet of the concentrator is shown in Figure 2. The method of flotation employed is a selective separation of the lead and zinc minerals in the original pulp.

Crushing

All ore delivered to the primary ore bin has passed through grizzlies set at 9 inches which are located on the level stations of the mine over the ore pockets. Ore from the mine is delivered through the main vertical three-compartment shaft to the primary ore bin at the shaft collar. This bin has an inclined bottom, is steel lined, and has a capacity of 125 tons.

From the primary bin the ore is fed to a bar grizzly with 4-inch openings, set at an angle of 45°, by a 36-inch Link-Belt apron feeder. The grizzly oversize product goes to an 18 by 30 inch Allis-Chalmers Blake-type crusher set at 4 inches. The crusher is belt-driven at a speed of 250 r.p.m. by a 50-hp. motor. The average power required to operate the crusher under full load amounts to 16-1/2-hp. The crusher is equipped with manganese steel wearing plates having vertical corrugations, and during 12 months of operation the only replacement has been one stationary wearing plate. The normal crushing rate is 65 tons per hour.

The grizzly undersize joins the crusher discharge on a 24-inch, rubber-surfaced belt conveyor and is carried to a 3 by 8 foot Link-Belt vibrating screen, inclined at an angle of 18° and operated at a speed of 1,300 r.p.m. The screen is of manganese steel plate one-fourth inch thick, punched with 1-1/4-inch square holes, and lasts from three to four months. Due to the muddy and sticky character of the ore only a small amount of the undersize is larger than 3/8-inch size.

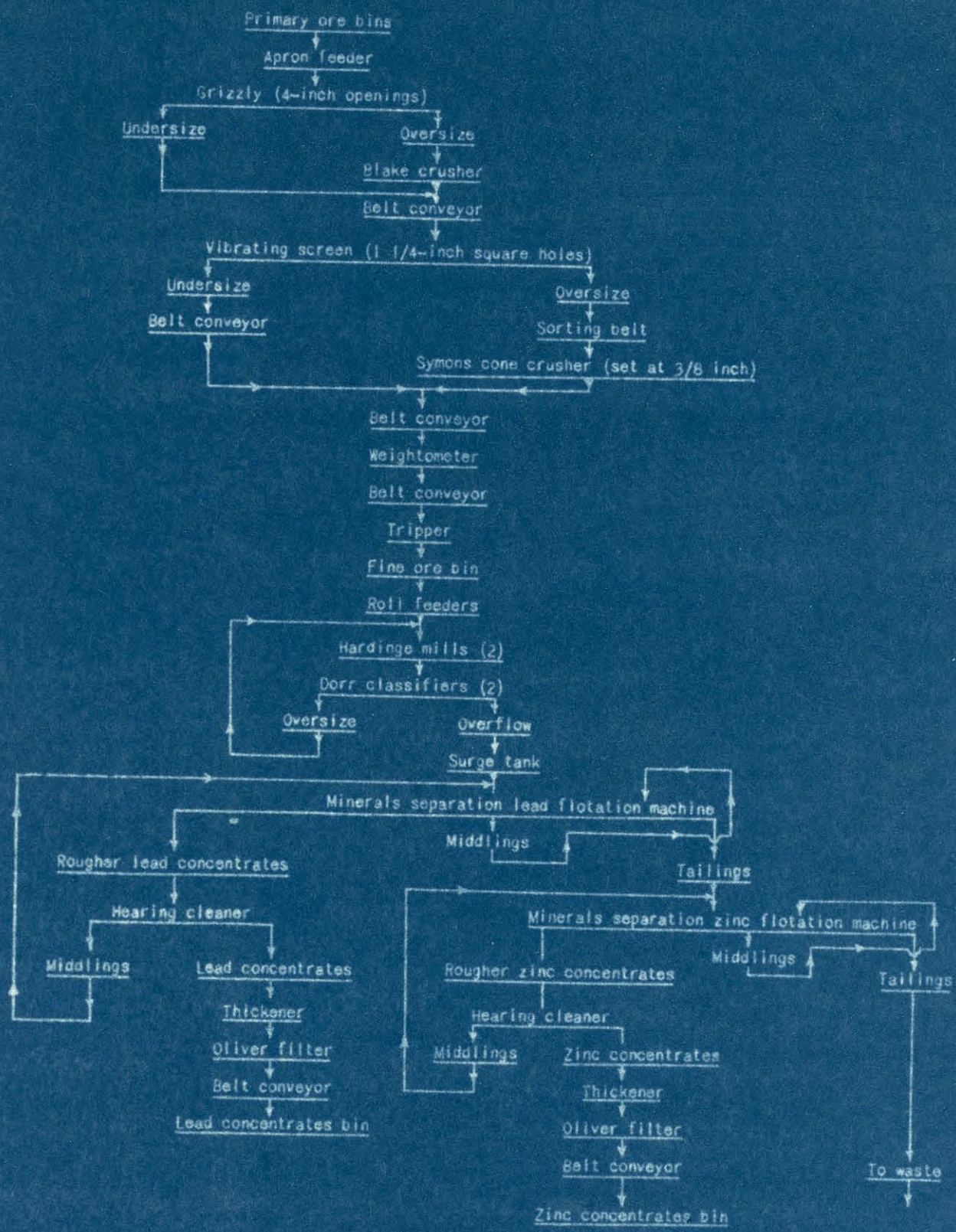

Figure 2.- Flow sheet of Tybo concentrator

The screen oversize goes to a 30-inch, rubber-surfaced belt conveyor where wood, tramp iron, and occasional pieces of waste are sorted out by hand. The ore is usually wet and covered with slime to such extent that it is difficult to distinguish waste from ore. To obtain efficiency from sorting, the ore would have to be thoroughly washed.

The oversize from the vibrating screen is discharged into a 4-foot Symons cone crusher set at 3/8-inch. The Symons crusher is direct-connected to a 100-hp. motor and consumes an average of 36-hp. under full load.

The undersize from the vibrating screen is conveyed on a 20-inch belt to a second 20-inch belt where it joins the discharge from the cone crusher. The combined products pass over a Merrick weightometer to a 20-inch distributing belt conveyor, equipped with a self-propelled tripper, which discharges the products into a 1,400-ton fine-ore bin.

Grinding

The fine-grinding equipment consists of two 8 by 4 foot Hardinge ball mills each operating in closed circuit with a type D, 6 by 25 foot, Dorr classifier. Each ball mill is driven at a speed of 19 r.p.m. by a 200-hp. motor, through a Farrell reduction unit, the reduction ratio being 870 to 159. The average power consumption, under full load, amounts to 139 hp. The ball mills operate with a ball load of 18 tons, a circulating load of approximately 400 per cent and a pulp density between 75 and 78 per cent of solids. Ball consumption, amounting to 2.7 pounds per ton of ore ground, is compensated by the addition of 3-inch "Adamantine" steel balls. Manganese steel liners are used, and their estimated average life is from 14 to 16 months. The feed-cone liners were replaced after 8 months of service, the original cone-discharge liners are still in service, and approximately 80 per cent of the original breast liners are in service after 13 months of operation.

Each Dorr classifier is operated at 14 strokes per minute by a 5-hp. motor which consumes an average of 1-1/2 hp. under full load. The overflow is maintained at a density of 32 to 36 per cent of solids, and these solids average from 68 to 75 per cent minus 200-mesh material. Table 1 gives screen analyses of the ball-mill discharge and classifier overflows. As previously stated, a large amount of slime is detrimental to flotation operations and since an excessive amount of slime is produced in grinding ores from the upper levels of the mine, finer grinding than is shown by the screen analyses of Table 1 is not practiced.

Lead Flotation Circuit

The classifier overflow products, which comprise the feed to the lead flotation circuit, flow by gravity to a surge tank, 24 feet in diameter by 8 feet high, constructed of redwood staves. The pulp flows by gravity from this tank to the head of the lead flotation section.

The equipment in the lead flotation section consists of one 16-cell Minerals Separation Sub-A machine and one three-pan Bunker Hill-Sullivan type Hearing pneumatic cleaner. The cells of the Minerals Separation machine are 18 by 31-1/4 inches and the impellers are 18 inches in diameter.

Depending on the grade and tonnage of the ore treated, the first three to five cells of the Minerals Separation machine produce rough flotation concentrates which are

pumped to the Hearing cleaner unit. The cleaner unit produces finished concentrates and a middlings product which returns by gravity to the head of the Minerals Separation machine. The froths from rougher cells 6 to 10, inclusive, are returned to the head of the rougher machine, froths from cells 11 to 13 are returned to cell 8, and froths from cells 14 to 16 are returned to cell 11. The tailings from the lead section comprises the feed of the zinc flotation circuit.

The spindles of the Minerals Separation machine are driven in pairs at a speed of 330 r.p.m. by eight 7-1/2-hp., alternating-current, 440-volt, vertical motors operating at 1,155 r.p.m. Tex-rope drives are used and after operating over a year are in excellent condition. The power requirement for the impellers of the Minerals Separation machine amounts to 2.6-hp. for each of the 16 impellers when operating under full load. The impellers of the lead section machine have a life of approximately eight months.

Reagents are introduced into the flotation pulp at the ball mills and just ahead of the flotation circuit. The following kinds and amounts of reagents per ton of ore treated are added to the ball mills.

	Pounds
Soda ash	1.30
Sodium cyanide	0.35
Zinc sulphate	0.50

The kinds and amounts of reagents added ahead of the lead flotation circuit per ton of ore treated follow:

	Pounds
Soda ash	0.60
Sodium cyanide	0.30
Zinc sulphate	0.40
Ethyl xanthate	0.10
Cresylic acid	0.50

Soda ash is added to neutralize the acidity of the ore and to maintain the desired alkalinity of the pulp. The alkalinity is determined by pH indicators and is maintained at from 8 to 8.6 pH values. Sodium cyanide and zinc sulphate are used in conjunction to depress the sphalerite and pyrite in the lead circuit. The ethyl xanthate is added as an accelerator in the flotation of lead minerals and the cresylic acid is used as a frothing agent.

The density of pulp in the lead circuit is maintained between 30 and 35 per cent of solids.

The character of the ore delivered to the concentrator varies considerably from day to day. Operating results from the treatment of clean sulphide ores derived from the lower levels of the mine are consistent and concentrates ranging from 66 to 70 per cent of lead are readily obtained. Variations from this condition are due to the degree of oxidation and amount of primary slime encountered in the concentrator heads. As the slime content of the flotation feed increases, the grade of concentrates produced decreases, and experience has shown that this condition is not to be overcome by the use and control of reagents.

Zinc Flotation Circuit

The tailings from the lead flotation section flow by gravity through an inverted syphon to the head of the Minerals Separation machine of the zinc circuit. The flotation equipment of the zinc section is a duplicate of that used for the flotation of lead minerals. Depending upon the grade and tonnage of the feed to the zinc section, rough concentrates are produced from the first four to eight cells. These concentrates are pumped to the Hearing cleaner unit, which produces finished concentrates and a middlings product which is returned by gravity to the head of the Minerals Separation rougher machine. Assuming that cells 1 to 4, inclusive, of the Minerals Separation machine are producing rough concentrates, the froths of cells 5 to 8, inclusive, are returned to the head of the rougher machine, and froths from cells 9 to 16, inclusive, are pumped to cell 5. The tailings of the zinc section are conveyed by launder to the tailings pond.

The following kinds and amounts of reagents per ton of original ore treated are added to the pulp ahead of the zinc section.

	Pounds
Copper sulphate	1.30
Ethyl xanthate	0.10
Hydrated lime	1.90
Pine oil	0.01

The copper sulphate is added to reactivate the sphalerite which was depressed in the lead circuit. Lime, which can not be used in the lead circuit on account of the acti of the calcium ion in depressing galena, is used in the zinc circuit as a depressant of pyrite. Pine oil is used as an auxiliary frothing reagent.

The density of the pulp in the zinc circuit is maintained at approximately 25 per cent of solids. Due to the lower pulp density in the zinc circuit as compared to the pulp of the lead circuit, flotation-machine impellers of the zinc circuit have nearly twice the life of impellers operated in the lead circuit.

Power requirements for the impellers of the Minerals Separation machine of the zinc circuit are 2.4 hp. for each of the 16 impellers operating under full load.

Air is furnished to the Minerals Separation machines of both lead and zinc circuits at 1-1/2 pounds pressure by a General Electric centrifugal air compressor direct-connected to a 11-hp. motor consuming 8 hp. under full load. Air is furnished to the Hearing cleaner units of both lead and zinc sections at 4-1/2 pounds pressure by a No. 2 Connersville blower, belt-driven by a 10-hp. motor requiring 11 hp. under full load.

DEWATERING OF FLOTATION CONCENTRATES

The finished concentrates from the lead and zinc flotation circuits are pumped to their respective Hardinge thickeners, each 24 feet in diameter by 8 feet high, by 2-inch Wilfley pumps. The scraping mechanism of the thickeners makes one revolution in six minutes. Thickener feeds contain about 10 per cent of solids and the thickener discharge from 60 to 70 per cent of solids. The thickened concentrates are conveyed through 4-inch pipe lines equipped with Nordstrom valves, to two Oliver filters, each 5 feet 4 inches in diameter by 10 feet wide. Each filter has 167-1/2 square feet of filtering surface, Palma twill style 15 C

being used as the filtering medium. The average life of the filtering cloth is eight months on the zinc filter and three months on the lead filter.

Vacuum at the filters is maintained at about 18 inches of mercury, the elevation of the plant being 6,700 feet above sea level. Two Oliver vacuum pumps, 14 by 8 inches in size, driven by two 15-hp. motors through Lenox short-center drives, are used for this purpose. Compressed air for removing the filter cake is supplied at 10 pounds pressure by one 9-1/2 by 8 inch Oliver compressor driven through a Lenox drive by a 10-hp. motor which requires 9-1/2 hp., on the average, under full load.

The filter cakes drop onto 16-inch belt conveyors and are discharged into their respective bins. The concentrates are transported by truck to Tonopah for rail shipment. No provision has been made for heating the concentrates storage bins in winter, and no trouble has been experienced with frozen concentrates, although the minimum temperature during winter reaches 10° below zero.

Table 2 gives analyses of lead and zinc concentrates.

DISPOSAL OF TAILINGS

The tailings are conveyed in launders by gravity to the disposal ground. Adequate ground is available for tailings storage without impounding, and their disposal does not constitute a serious problem.

SAMPLING AND CONTROL OF OPERATIONS

The flotation feed and the final flotation tailings streams are cut at 15-minute intervals by Galigher automatic samplers. Samples of the tailings of the lead section and of the two final concentrates are cut from the respective launders by operators. A specially designed hand sampler, shown in Figure 3, is used for this purpose.

The daily mill feed and tailings samples are composited for a 15-day period on the basis of tonnage treated, allowing 1 gram for each 10 tons of ore concentrated, and the composite sample is assayed by control methods.

A comparison of the average smelter returns with concentrator assays made on samples taken by hand from May 13 to December 31, 1929, follows:

Average smelter returns compared with concentrator assays of samples taken by hand from May 13 to December 31, 1929

	Lead concentrates			Zinc concentrates		
	Silver, ounces per ton	Lead, per cent	Zinc, per cent	Silver, ounces per ton	Lead, per cent	Zinc, per cent
Smelter	90.43	62.75	3.50	13.66	2.71	46.57
Concentrator	90.65	62.47	3.77	13.39	2.29	46.84

The production of lead concentrates is calculated from the usual formula:

Per cent of mill feed produced as lead concentrates = 100 F-T ÷ C-T where F, C, and T are lead assays of lead section feed, lead section concentrates, and lead section tailings products, respectively.

The production of zinc concentrates is calculated from the formula:

Percentage of original mill feed produced as zinc concentrates = (100-L) (F-T) ÷ (C-T) where F, C, and T are the zinc assays of zinc section feed, zinc section concentrates, and zinc section tailings products, respectively, and L is the per cent of mill feed produced as lead concentrates.

Flotation operations are controlled by the appearance of the froths which are frequently examined by panning with a white enameled vanning plaque.

The pressure filter shown in Figure 4 is used for the dewatering of pulp samples. This filter was designed by H. M. Lewers, mill superintendent, and has been found satisfactory. In operating the filter, a 7-inch diameter filter paper is placed over the canvas and 90-pound pressure mine air is used. It requires less than three minutes to dewater a pulp sample weighing 10 ounces and the filtrate is perfectly clear.

CONCENTRATOR RECOVERIES AND LOSSES

Table 3 gives concentrator results for March, 1930. Table 4 shows the gross values per ton of the heads and of the lead concentrates and zinc concentrates. Table 5 gives extractions of silver, lead and zinc computed as indicated in the table. Table 5 also gives the "economic recoveries" of silver, lead, and zinc. "Economic recovery" of a metal at this plant is computed as the product of the gross value of the metal in the concentrator heads, in dollars, and the recovery of the metal, as concentrates, in per cent. Table 6 gives the metallurgical data for March, 1930.

Most of the silver loss in the tailings is due to silver associated with pyrite, as indicated by the following test. A sample of concentrator tailings, representing one week of operation, was sized by a 200-mesh sieve. The minus 200-mesh material was further separated into sand and slime products, the sand product consisting of approximately minus 200-mesh plus 500-mesh sizes. A separation of the sulphides and gangue contents of the sand product was made with bromoform of 2.82 specific gravity. The sulphides obtained were given a flash or magnetizing roast, and the pyrite was then separated with a magnet. The following tabulation of assay values in the several products obtained indicates the association of silver with the pyrite.

	Silver,	Per cent		
	ounces per ton	Lead	Zinc	Iron
Composite tailings sample	2.76	1.1	1.3	11.6
Slime product, minus 500-mesh sizes	2.30	0.4	0.6	7.7
Bromoform float sand	0.50	Trace	Trace	–
Pyrite	10.25	1.5	0.1	–

The following tabulation gives a comparison of analyses of concentrator heads and final products when treating two different mixtures of lower and upper level mine ores and illustrates the effect of increasing the proportion of upper-level ores on the grade of concentrates.

Comparison of analyses of concentrator heads and final products when treating two different mixtures of lower and upper level mine

	Concentrator feed mixtures, per cent							
	Lower levels = 80 Upper levels = 20				Lower levels = 50 Upper levels = 50			
	Assays				Assays			
	Silver, ounces per ton	Lead, per cent	Zinc, per cent	Iron, per cent	Silver, ounces per ton	Lead, per cent	Zinc, per cent	Iron, per cent
Heads	8.6	5.7	5.0	11.1	9.1	5.8	4.3	12.5
Lead concentrates	96.4	69.5	4.3	6.1	62.8	63.2	4.2	6.7
Zinc concentrates	12.4	1.6	48.6	11.7	13.3	2.4	47.9	11.0
Tailings	2.0	0.87	1.03	12.9	3.0	1.6	0.95	13.2

The lead losses in the tailings produced during the periods covered by the preceding tabulation, were determined to be distributed as follows:

Distribution of lead losses in the tailings

	Concentrator feed mixtures, per cent	
	Lower levels = 80 Upper levels = 20	Lower levels = 50 Upper levels = 50
Lead in tailings, in oxidized form, per cent	0.35	0.80
Lead in tailings, associated with pyrite, per cent	.32	.60
Lead in tailings, association not determined, per cent	.20	.20
Total lead in tailings, per cent	0.87	1.60

The following tabulation gives a screen-assay analysis of the tailings product produced during the period in which the 50 per cent upper-level and 50 per cent lower-level ore mixture was treated.

Screen-assay analysis of concentrator tailings

Screen sizes	Weight, per cent	Assays				Per cent of total			
		Silver, ounces per ton	Lead, per cent	Zinc, per cent	Iron, per cent	Silver	Lead	Zinc	Iron
Plus 100-mesh	19.0	2.2	0.8	0.9	5.1	15.2	9.4	18.1	8.7
Minus 100 plus 150 mesh	13.9	1.6	1.4	1.1	15.5	8.0	11.9	16.2	19.4
Minus 150 plus approximately 350 mesh sands	20.9	1.8	2.0	0.8	19.2	13.7	25.6	17.7	36.1
Minus approximately 350 plus 500 mesh sands	4.3	3.8	0.6	0.7	9.6	5.7	1.9	3.2	3.7
Slimes	41.9	3.8	1.9	1.0	8.5	57.4	51.2	44.8	32.1

CONCENTRATOR OPERATION AND COSTS

Under normal conditions of treating 320 tons of ore per day the mill crew is distributed as follows:

	Number of operators
Crusher	2
Sorting belt	1
Ball mills	3
Flotation	3
Filter and reagents	1
Total	10

The entire concentrator is shut down for a short period twice each month for the examination of ball-mill liners and for general repairs. Repairs and oiling are handled by the mechanical and electrical departments. The total time lost because of enforced shut downs in over a years operation amounted to two hours, caused by defective insulation in a small motor.

The milling costs vary inversely with the tonnage of ore treated. A summary of costs per ton of original ore, for March, 1930, when treating 320 tons of ore per day, is given in Table 7. Distributions of labor and power and a summary of reagents are shown in Table 8.

Table 1.--Screen analyses of concentrator products

Screen sizes, mesh	Ball-mill discharges, 78 per cent solids		Classifier overflows, 42 per cent solids		Flotation tailings of lead section, 33 per cent solids		Flotation tailings of zinc section, 25 per cent solids	
	Weight, per cent	Cumulative per cent	Weight, per cent	Cumulative per cent	Weight, per cent	Cumulative per cent	Weight, per cent	Cumulative per cent
On 14	9.75	9.75	-	-	-	-	-	-
On 28	4.15	13.90	-	-	-	-	-	-
On 35	3.90	17.80	-	-	-	-	-	-
On 48	5.91	23.71	-	-	-	-	-	-
On 65	10.30	34.01	5.09	5.09	6.12	6.12	6.44	6.44
On 100	11.35	45.36	9.10	14.19	9.36	15.48	8.07	14.51
On 150	10.10	55.46	8.86	23.05	9.15	24.63	10.92	25.43
On 200	6.77	62.23	8.10	31.15	8.41	33.04	7.10	32.53
Through 200	37.77	100.00	68.85	100.00	66.54	99.58	67.47	100.00

Table 2.--Analyses of lead and zinc concentrates

	Ounces per ton		Per cent								
	Gold	Silver	Lead	Zinc	Copper	Iron	Sulphur	Insoluble	Arsenic	Antimony	Cadmium
Lead concentrates	0.18	90.43	62.75	3.70	Trace	6.7	18.3	3.8	1.3	0.7	(1)
Zinc concentrates	0.04	13.65	2.71	46.57	0.10	14.1	28.4	3.3	(1)	(1)	0.5

1 - Not determined.

Table 3.--Concentrator results for March, 1939

	Weight		Assays			Weight, pounds per ton		Total weight			Concentration ratios
	Per cent	Tons	Silver, ounces per ton	Lead, per cent	Zinc, per cent	Lead	Zinc	Silver ounces	Lead, pounds	Zinc, pounds	
Concentrator heads	100.00	9,930.0	3.44	3.95	4.14	113.0	82.8	33,739	1,181,670	822,204	-
Lead concentrates	7.08	703.5	87.10	69.90	4.00	1,193.0	80.0	61,275	842,793	56,280	14.12 to 1
Zinc concentrates	6.29	625.0	14.35	2.55	47.60	51.0	952.0	8,975	31,875	595,000	15.98 to 1
Lead section tailings	62.92	9,225.5	3.60	1.95	4.17	37.0	83.4	33,218	341,381	769,490	-
Zinc section tailings	86.63	8,601.5	3.02	1.35	1.01	27.0	20.2	25,976	328,256	173,760	-

Table 4.--Gross metal values per ton of heads and of lead and zinc concentrates

	Metal prices	Heads	Lead concentrates	Zinc concentrates
Silver	41 cents per ounce	$3.87	$35.71	$5.85
Lead	5.50 cents per pound	6.54	65.89	[1]2.55
Zinc	5.00 cents per pound	4.14	-	47.60
Total		$14.55	$101.60	$56.00

1 - Price of lead in zinc concentrates taken at 5 cents per pound.

Table 5.--Per cent recoveries and "economic recoveries" of metals

	Silver	Lead	Zinc
Extractions:			
Extractions computed from metals in heads and concentrates, per cent	74.9	74.0	72.4
Extractions computed from metals in heads and tailings, per cent	72.3	73.1	72.0
Extractions computed from metals in concentrates and tailings, per cent	73.0	73.3	72.1
"Economic recoveries":			
Silver = 74.9 per cent of $3.87	$2.90	-	-
Lead = 74.0 per cent of $6.54	-	$4.84	-
Zinc = 72.4 per cent of $4.14	-	-	$3.00
Average "economic recovery" = 10.47/14.55 x 100 per cent		73.8	

Table 6.--Metallurgical data for March, 1930

Dry ore treated, tons	9,930
Moisture in ore to mill, per cent	3.8
Hours operated per day	24
Days operated	31
Ore treated per 24 hours, tons	320
Total concentrates produced, dry tons	1,328
Average lead concentrates produced per 24 hours, dry tons	22.7
Average zinc concentrates produced per 24 hours, dry tons	20.2
Recovery of lead, per cent	74.0
Per cent of total lead in lead concentrates	71.3
Per cent of total lead in zinc concentrates	2.7
Recovery of zinc, per cent	79.2
Per cent of total zinc in zinc concentrates	72.4
Per cent of total zinc in lead concentrates	6.8
Recovery of silver, per cent	74.9
Per cent of total silver in lead concentrates	65.4
Per cent of total silver in zinc concentrates	9.5
Ratio of concentration for lead section, tons of original ore into 1	14.12
Ratio of concentration for zinc section, tons of original ore into 1	15.89
Water consumption per ton of ore, tons	5
Ball consumption per ton of ore, pounds	2.7
Average pulp density in lead circuit, per cent of solids	34
Average pulp density in zinc circuit, per cent of solids	25
Average temperature of mill water, °F.	58

Table 7.—Summary of concentrator costs for March, 1930

Ore treated = 9,930.0 tons
Lead concentrates produced = 703.5 tons
Zinc concentrates produced = 625.0 tons

	Operating labor	Power	Supplies	Miscellaneous	Total
Sorting	$0.017	–	–	$0.001	$0.018
Crushing	.035	$0.023	$0.019	.002	.079
Grinding and classification	.054	.210	.176	.005	.445
Flotation	.073	.064	.359	.006	.502
Filtering	.015	.005	–	.001	.021
General mill expense	.050	.016	–	.151	.217
Total	$0.244	$0.318	$0.554	$0.166	$1.282

Table 8.—Distributions of labor and power and summary of reagents for March, 1930

Ore treated = 9,930.0 tons
Lead concentrates produced = 703.5 tons
Zinc concentrates produced = 625.0 tons

Labor (man-hours per ton concentrated):	
Sorting	0.027
Crushing	.047
Grinding and classification	.078
Flotation	.097
Filtering and mixing reagents	.022
Miscellaneous	.067
Total man-hours per ton of ore concentrated	0.338
Power, kw.h. per ton:	
Crushing	1.77
Grinding and classification	15.84
Flotation	4.85
Filtering	0.39
Miscellaneous	1.19
Total kw.h. per ton of ore concentrated	24.04
Reagents (pounds per ton of ore treated):	
Soda ash	1.90
Sodium cyanide	0.65
Zinc sulphate	0.90
Copper sulphate	1.30
Ethyl xanthate	0.20
Cresylic acid	0.50
Lime (hydrated)	1.90
Pine oil	0.01
Miscellaneous:	
Steel balls, pounds per ton of ore concentrated	2.70

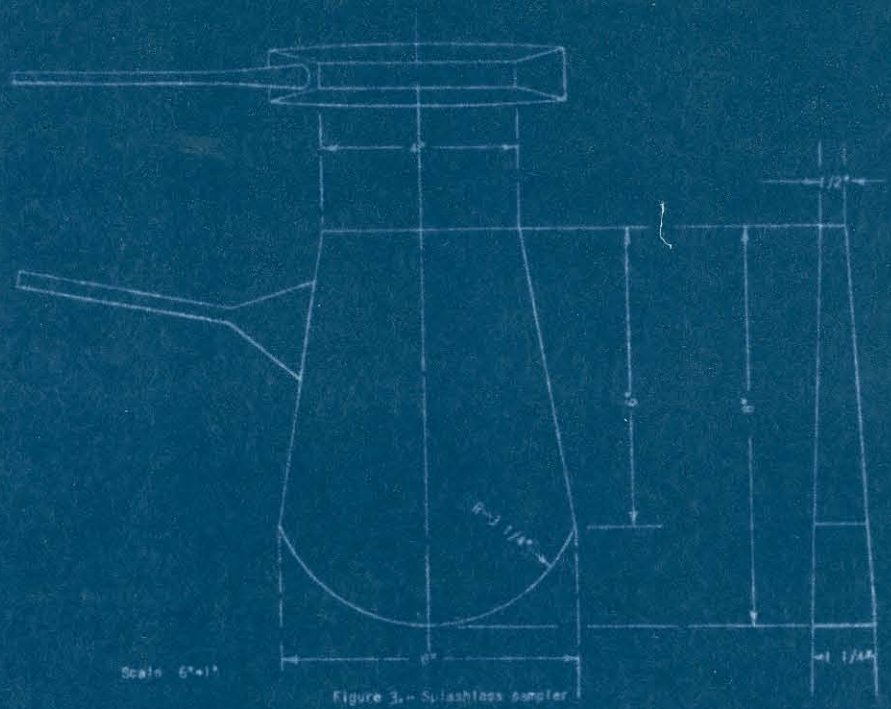

Figure 3.- Splashless sampler

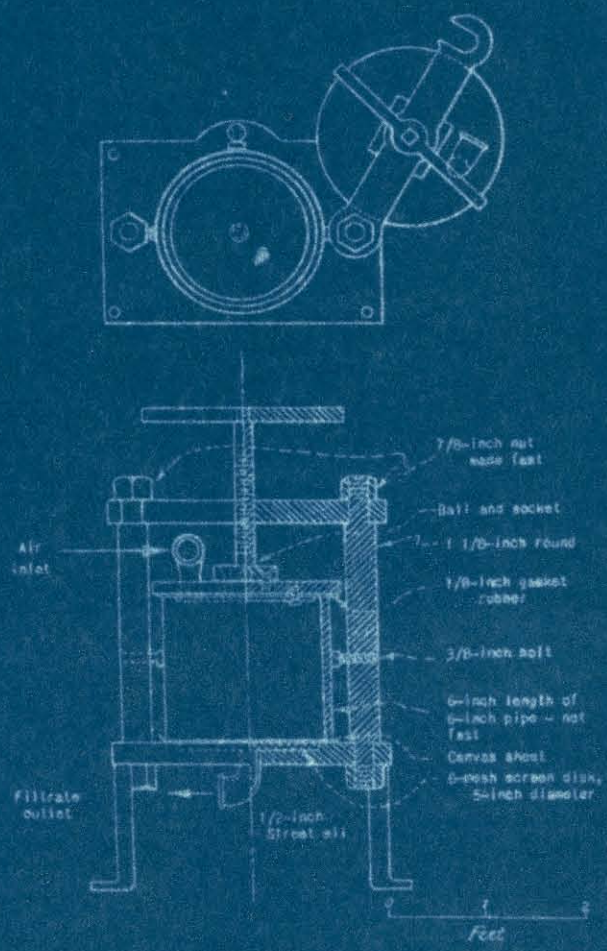

Figure 4.- Laboratory filter