FOR YOUR ENGINEERING NOTEBOOK

FILE NO. BULLETIN NO. M4-B32

SUBJECT MILLS, FLOTATION

Twin J Property
92B/13W
827376

L. SCOTT
RR#2 PAYNE
RD, DUNCAN
B.C.

# TWIN "J" MINES, Ltd.

By R. B. GAYER ★ and J. R. WILLIAMS ★

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**FOREWORD**

Twin "J" Mines Ltd., is located on Mount Sicker, approximately ten miles from Duncan, Vancouver Island, B. C., Canada. The small city of Duncan is between Victoria and Nanaimo on the main Vancouver Island highway and is served by the Esquimalt and Nanaimo Railroad which is operated by the Canadian Pacific.

Design of the mill and experimental work were carried out under the direction of W. J. Asselstine, consulting metallurgist.

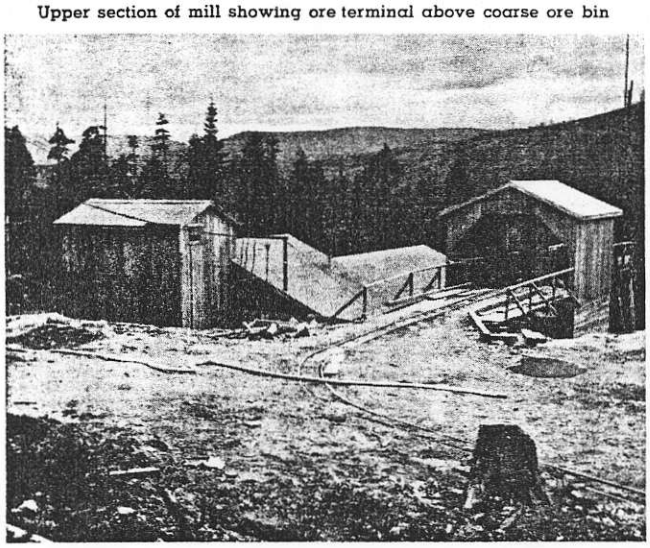
Photographs are by H. M. Wright.

## Introduction

In order to appreciate the problems in treating the Twin "J" ore a brief outline of the geologic and structural occurrence of the ore in the mine will be helpful.

The Lenora-Tyee deposit consists of two westerly striking ore structures referred to as the "North Vein" and the "South Vein." The ore in these structures occurs in narrow, intensely drag-folded shear zones that have been completely replaced and mineralized. The rocks enclosing the shear zones consist of metamorphosed cherty tuffs and black phyllitic slates; the latter known as graphitic schists. These have been intruded and largely replaced by feldspar-porphyry and diorite. Metamorphic action has altered the porphyrys and diorites to a soft, friable schist, locally called a silver schist.

Upper section of mill showing ore terminal above coarse ore bin



From the standpoint of the mill operator there are three distinct types of ore. Two of these occur coextensively within both deposits and consist of a "zinc-barite" ore and a cherty "quartz-copper" ore. The third consists of the intermingled oxidized pro- uct of the first mentioned types. Only the quartz-copper ore is readily amenable to selective flotation. The chief metal constituents of the ore are the sulphides; sphalerite, chalcopyrite, pyrite and minor amounts of galena. The non-metallic constituents are barite and quartz in more or less equal amounts with small inclusions of calcite and alumina. Small erratic amounts of marcasite, bornite, chalcocite, native silver and free gold as well as the oxidation products of the previously mentioned sulphides also occurred throughout both ore bodies.

About 75 per cent of the ore treated was of the zinc-barite type. This was diluted with schistose wall rock and varying amounts of the quartz-copper ore. In addition, oxidized portions of the ore were also mixed with the mill feed.

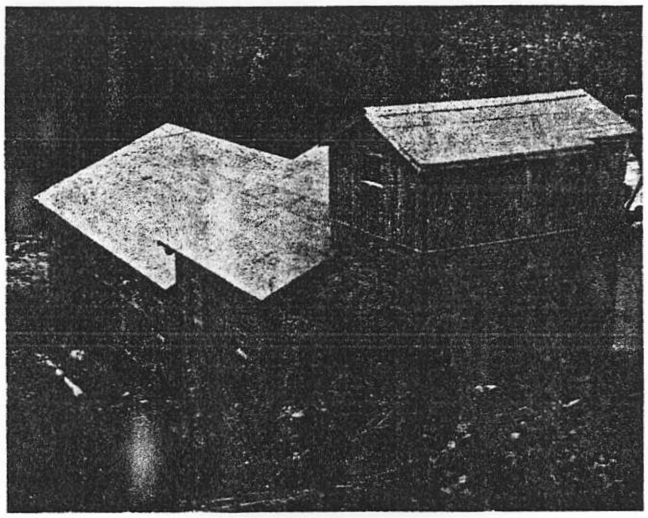
## Milling

### Crushing

Ore was brought from the mine in rocker dump cars, twenty cubic foot capacity, hauled by a Mancha "Little Trammer." The tracks ran over the centre of the coarse ore bin and the cars were hand dumped from either side over a 5 x 18 foot grizzly made up of twenty pound rails set with nine inch clearances. The live capacity of this bin was approximately 140 tons. All of the ore passed from the bin along a short chute controlled by a steel arc-type chute gate to a 10 x 20 Allis Chalmers jaw crusher. The jaws were set at 1¾ inches. Power was supplied through a flat belt drive by a 20 h.p. electric motor. Crusher jaws were lined with manganese steel plates. The stationary plate had a corrugated surface while the moving jaw plate had a flat surface.

Crusher product was discharged onto a 6 ply-18 inch conveyor belt which elevated the ore and discharged it onto a 2'x6' Dillon Vibrating Screen with ½ inch openings. Oversize from this screen passed to a 2 foot Symons cone crusher set at ½ inch. The cone crusher was protected from tramp iron by a Dings magnetic pulley at the discharge

*R. B. Gayer, General Superintendent
*J. R. Williams, Mill Superintendent



Twin "J" Concentrator

end of the conveyor belt. Undersize by-passed the cone crusher and discharged onto a short conveyor belt passing beneath the cone crusher. This belt also caught the discharge from the cone crusher and the combined products were carried to a forty foot bucket elevator. Elevated ore was discharged into a 300 ton fine ore bin.

The crushing plant had a capacity of 125 tons per eight hour shift but due to the long underground haul only 90 to 100 tons were crushed per shift. The day shift crusherman spent most of his time in actual crushing while the afternoon crusherman cleaned and repaired the crushing and conveying equipment as soon as the fine ore bin had been filled. No crushing was done on the graveyard shift.

### Grinding

Ore was drawn from the fine ore bin by a 24"x8' Denver Adjustable Stroke Ore Feeder. A weight and head sample taken hourly at this point controlled the rate of feed. Grinding took place in a 7 x 6 cylinder ball mill converted from an old tube mill by the Union Iron Works. The mill was powered by a 150 h.p. electric motor geared to a Falk gear type reducer which turned the mill at a speed of 22 r.p.m. The initial ball mill load was made up of nine tons of manganese steels balls, 5, 4, 3 and 2 inches in diameter. As grinding continued the load was maintained by feeding 5 and 4 inch balls only. This mill was in a closed circuit with a Dorr type 6 x 21 foot rake classifier. Hot water from the mine compressor was added to the ball mill discharge in order to maintain a higher temperature for flotation. This practice was later discontinued as the compressor, operating only sixteen hours out of twenty-four caused an uneven temperature condition that reacted unfavourably in the flotation circuit. The ball mill density was maintained at 70 per cent solids.

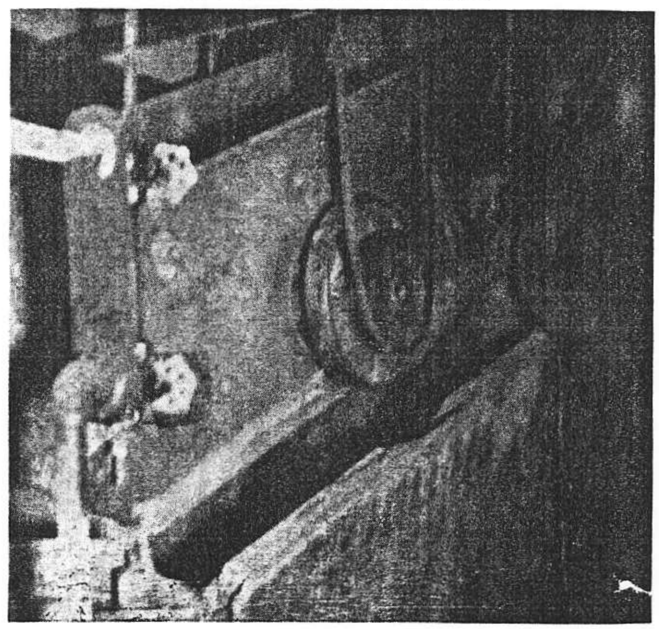
In October, 1943, a series of samples of various mill products were taken and sent to the British Columbia War Metals Research Board for examination and study. Excerpts from this study are shown in Tables 6 to 9. It will be noted that microscopic analysis revealed that 45 per cent of the sphalerite in the copper concentrate was locked with chalcopyrite and that 40 per cent of the chalcopyrite in the zinc concentrate was locked with the sphalerite. It was estimated by Mr. H. M. Wright and Mr. J. M. Cummings that approximately 66 per cent of the locked sphalerite was minus 325 mesh and that all of it was finer than 200 mesh.

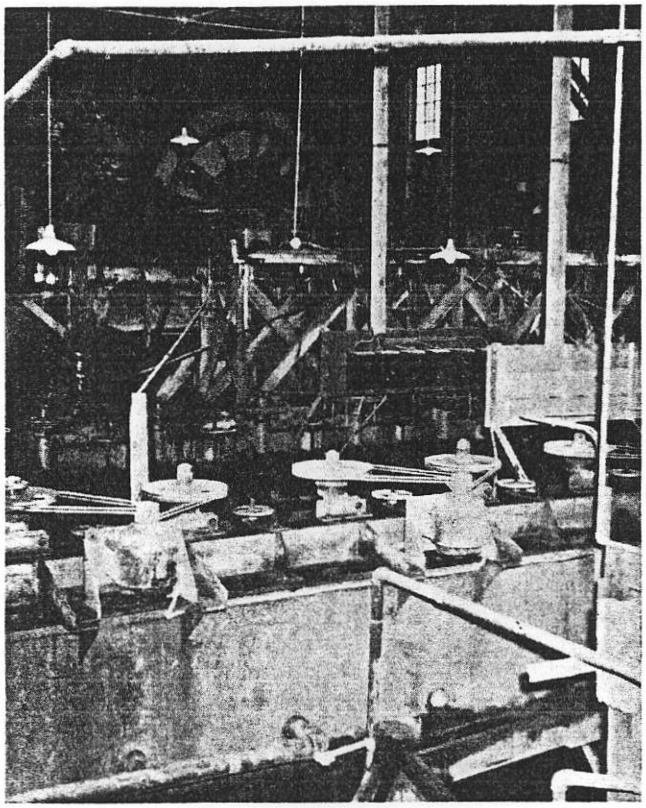
In attempting to release this locked material extremely fine grinding was tried. However, over all mill results suffered due to excessive sliming of the zinc, lead and gangue minerals. By trial and error 82 per cent minus 200 mesh was established as the best possible grind under the given conditions.

### Flotation

The classifier overflow containing 25 per cent solids was laundered to the copper circuit where it was conditioned in a 6'x6' Denver Standard Conditioner. Gravity flow from this conditioner led to an 8-cell No. 18 Special Denver "Sub-A" Flotation Machine. The entire block of cells pulled a rougher copper concentrate which was in turn laundered (see final flow sheet) to a 4-cell No. 12 Denver "Sub-A" Flotation unit for cleaning. Tailing from this cleaner circuit was returned to the ball mill for scrubbing and regrinding.

2'x6' Dillon Screen in crushing plant





General view of mill interior

Under-flow from the rougher copper circuit was discharged into another 6'x6' Denver Standard Conditioner and treated with saturated copper sulphate before entering the zinc circuit. The circuit consisted of a 10-cell No. 18 Special Denver "Sub-A" Flotation Machine. Eight cells made a rougher zinc concentrate which then passed through a two stage cleaning process in the remaining two cells of the circuit.

The final tailings from the zinc circuit were split and a portion passed over a Wilfley table for visual control. Upsets or disturbances in the mill circuit could be noted very quickly by observing the size and contents of the mineral streak on the concentrating end of the table.

Mill tailings were originally discharged into a water course containing a series of small dams in the flat sections of the stream. This stream emptied into the Chemainus River. Due to the turbulence of its course a large portion of the slimes failed to settle out and entered the river. After some controversy a launder was constructed from the bottom of the mill to convey the tailings into an adjacent alder swamp, the lower end of which was damned. The settling area thus provided was sufficient so that the discharge from the swamp was clear and uncontaminated by visible suspended matter.

### De-Watering

Final copper concentrates were pumped to a 10'x5' steel tank thickener by a 1½" Denver Vertical Centrifugal Sand Pump. Zinc concentrates were pumped by a similar pump to a 20 x 8 foot wooden tank thickener.

Two 2 inch diaphragm pumps delivered the thickened pulps to a 4 foot 5 disc Denver Filter. This filter was partitioned, three discs were used for de-watering zinc pulp and two discs for de-watering the copper pulp. The difference in filtering rates between the relatively porous copper concentrate and the more or less finer zinc concentrate caused the moisture content of the zinc cake to be excessive. To compensate for this, one of the copper discs was removed and the performance was improved considerably. A de-watered copper concentrate was formed having 8 per cent moisture but it was difficult to make zinc concentrate with less than 9 per cent moisture.

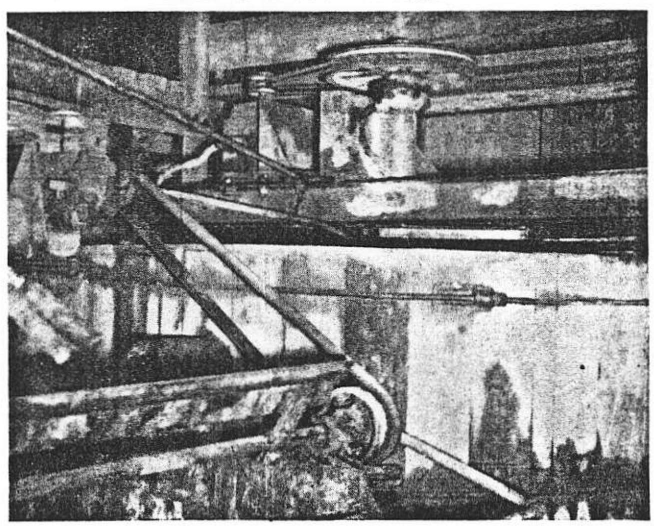
Filter cake was discharged from the filter into two separate concentrate bins of 40 tons each. From the bins dump trucks hauled the concentrate in five ton loads to a loading ramp at Westholme Siding, where the concentrate was dumped into 50 to 70 ton steel gondola cars for shipment to the smelter.

## Metallurgy

### Copper

Prior to the operation of this property by the Twin "J" Mines, several mining companies had carried out examinations in the course of which various ore tests were made. These tests were further verified and improved by the examining group for Twin "J" and based on results obtained in laboratories here and in the States. It was concluded satisfactory products and recoveries could be made. Samples

6'x6' Denver Conditioner in copper circuit



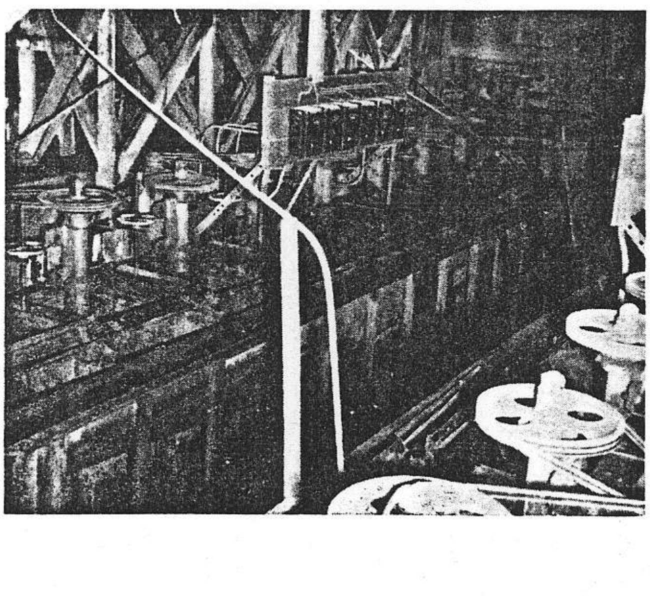
for these tests were obtained by approved methods. It was borne out by operating experience that factors, introduced by actual mining methods and conditions, varied the content of the mill feed from that indicated by the sampling and caused considerable trouble in the mill circuit. Wall rock in the mill feed was not in itself very harmful although it had a tendency to form slimes which had harmful effects later in the circuit.

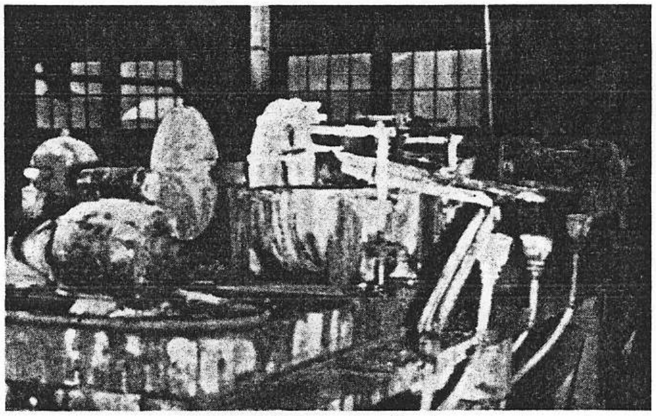
It did, however, when broken and mixed with the ore, act as an absorbent for mine water with which it came in contact in the stope chutes and transfer raises. The water entered the mine through old caved workings in the south vein area and judging from the large encrustations of blue and green copper stains on exposed surfaces in the area the water was saturated with soluble copper salts. These salts were absorbed in the broken schistose wall rock and fines and when released in the mill circuit they had a definite tendency to prematurely activate portions of the sphalerite and cause it to float with the chalcopyrite. In addition to this reaction it was felt that the soluable salts caused further harmful effects in the circuit. This was borne out by the fact that during the rainy seasons the mill feed was wetter and consequently mill performance became more erratic and difficult to control.

In an effort to depress this activated sphalerite various amounts of lime, soda ash, sodium cyanide and zinc sulphate were added to the ball mill. However, the depressing action of sodium cyanide in a basic circuit reacted harmfully and depressed chalcopyrite which was carried through into the zinc circuit.

It was the opinion of one metallurgist, who spent considerable time and study on the problem, that

8 Cell No. 18 Special Denver "Sub-A" in copper circuit





Denver Wet Reagent Feeders

the main trouble in making the copper-zinc separation was due to the galena present in the ore being slimed in the grinding circuit and acting as a promoter for the sphalerite. It has been proven by recent investigation in Australia that slime lead will definitely cause zinc to float. This slime coating on the zinc particles made it impossible for depressors to work satisfactorily. In order to counteract this condition an attempt was made to flocculate the lead particles before they were deposited on the zinc. To accomplish this an excessive amount of lime was added to the grinding circuit. Immediately after the high pH was introduced into the circuit the lead appeared on the flotation cells as large floccules. However, the high pH had a depressing effect on the copper.

At the suggestion of Mr. Williams, soda ash as well as lime was added to the grinding circuit and this brought about a favourable condition for the copper-zinc separation. Why the addition of soda ash should have this effect is not clearly understood. This mixture was used throughout the balance of the operation and gave satisfactory results. (See final flow sheet.)

In the original flow sheet the copper circuit consisted of an eight cell flotation machine, five cells making a rougher concentrate followed by three stage cleaning in the remaining three cells. (See preliminary flow sheet.) Sodium cyanide and zinc sulphate were added in the last cleaner cell to depress zinc. The middlings from the operation discharged into cells two and three and thence on into the rougher circuit.

The small amounts of zinc depressants from this cleaning operation were thus carried into the rougher circuit and had a harmful effect there. In addition, this cell arrangement caused a large circulating load of middlings to build up. After various unsuccessful attempts to alter this condition a small unit of four flotation cells was purchased and installed in the copper circuit. (See final flow sheet.)

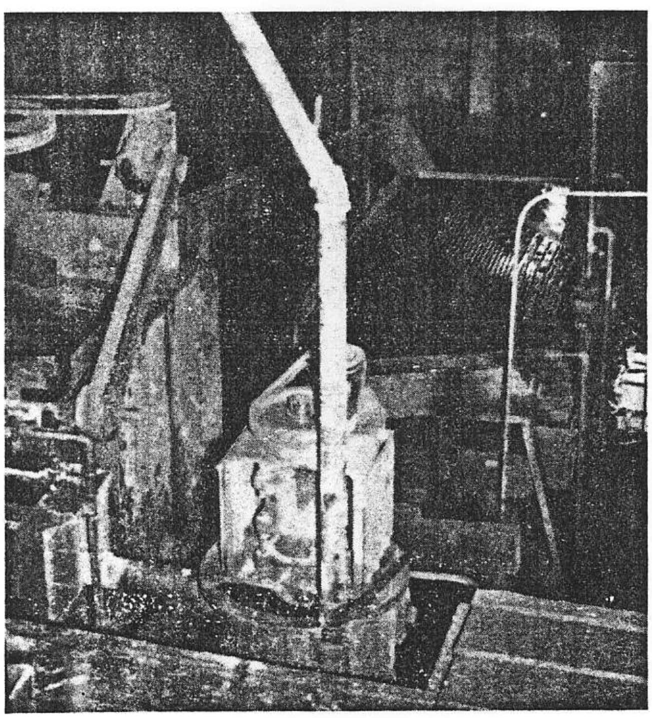
The new cells were used as cleaners and the tailing from this operation was pumped back to the grinding circuit. These middlings were returned to the grinding circuit at various points in order to avoid as much sliming as possible. It was found, however, that the primary consideration was the effect of the middlings on the ball mill density. To keep the density as high as possible the middlings were introduced as wash water in the classifier sands return launder.

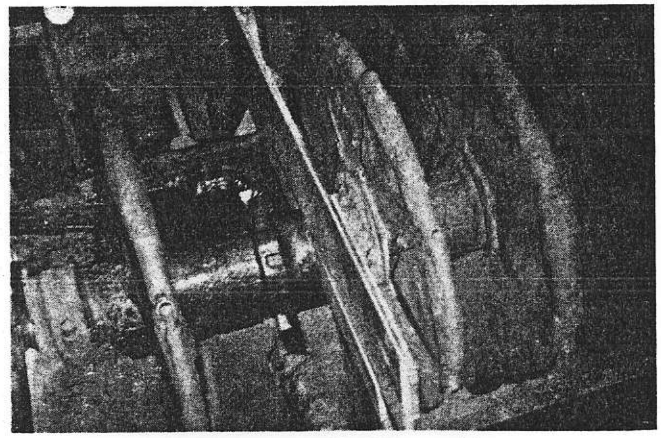
An attempt to overcome the condition by adding the middlings to the ball mill discharge seemed satisfactory at first but over a period of time a circulating load of middlings was built up in the copper circuit which tended to lower the grade of copper produced.

This change in the copper circuit showed improvement both in grade of concentrate and in recoveries. Under this arrangement the last four cells in the copper rougher circuit showed considerable amounts of slimed zinc, probably due, in part, to the regrinding of the copper middlings. This could have been overcome by the installation of a small regrind unit and hydroclassifier, which was indicated by early test work and included in the original mill specifications.

Test work was carried on at this point and further refinements to the copper circuit were made. It was determined by a series of tests and mill runs that

1½" Denver Vertical Sand Pump handling final zinc concentrates





5 disc 4' Denver Disc Filter used for dewatering both copper and zinc concentrates

a pH of 11.2 in the copper circuit plus careful control of collectors and frothers would produce a 23 per cent copper concentrate with good recoveries for this type of ore. This pH was maintained by adding one part soda ash to three parts lime at the rate of approximately 5.2 pounds per ton to the ball mill. Z-6 and R-208 in the ratio of Z-6 to two parts of R-208 was fed at the rate of 0.03 pound per ton to the copper conditioner. Additional Z-6 was added in the Number 5 cell of the rougher circuit.

## Zinc

The zinc circuit was a source of trouble, due chiefly to the necessity of maintaining a high grade concentrate. During the first four months of operations the zinc concentrate was stock piled. During this period an average grade of 45 to 46 per cent zinc was maintained in order to realize the best recoveries possible. Upon conclusion of final arrangements with the smelter it was determined that it would be necessary to maintain an average shipping grade of 50 per cent zinc. This meant that future production must be maintained well above 50 per cent in order to allow mixture with the previously stockpiled zinc.

In the preliminary circuit (see flow sheet) conditioned pulp flowed by gravity to cell Number 1. Cells four to ten inclusive were used to produce a rougher concentrate which was returned to cells one to three for cleaning. This system was discontinued as the average grade produced was around 45 per cent zinc.

The zinc circuit was altered to that shown on the final flow sheet. Conditioned pulp was fed into number three cell and cells three to ten inclusive pulled a rougher concentrate which passed through a two stage cleaning operation in cells one and two.

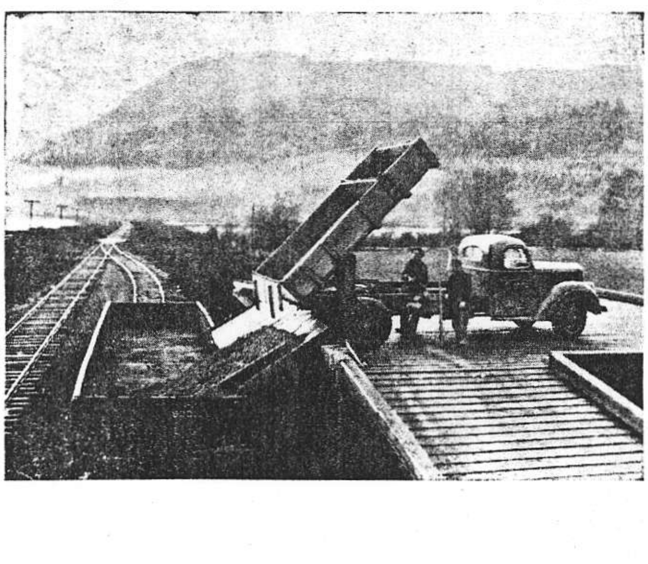
This arrangement gave a final zinc concentrate averaging between 50 and 53 per cent with tails between 0.5 and 1.5 per cent zinc. It is felt that an average grade of 49 per cent zinc with a final tails of around 0.4 could have been maintained by the installation of separate cleaning cells. This is borne out by the results shown in Table 5. Dilution from water in the zinc cleaner cells kept the density of the rougher circuit at around 16 per cent solids. A pH of 10.8 was maintained in the zinc circuit. A long series of mill tests, over a pH range varying between eight and twelve, were made. It was found that, all conditions remaining the same, a low pH gave a low grade concentrate and low tails while a high pH gave a high grade concentrate with high tails.

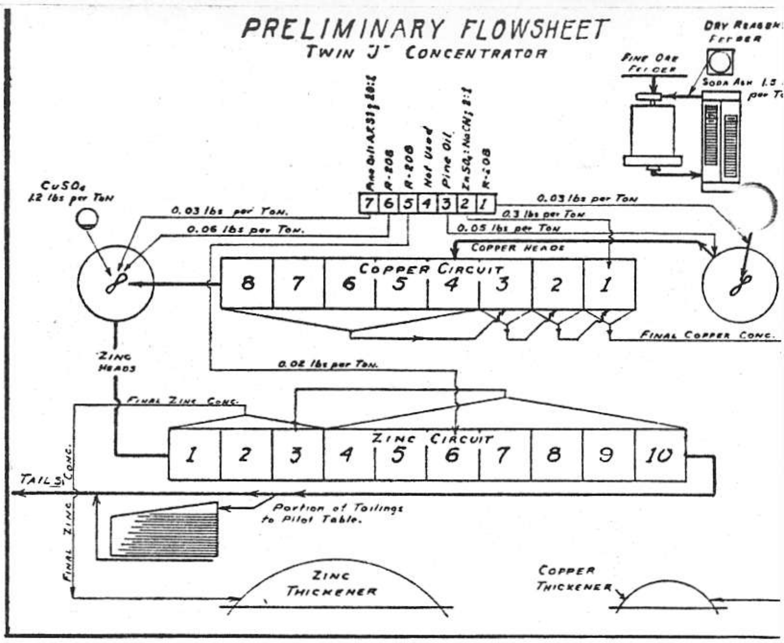
Barite and carbonaceous matter from the "graphitic" schists seemed to be activated by the copper sulphate in a high pH circuit and caused a lowering of the grade of zinc concentrate. Attempts were made to depress this gangue material with various starch mixtures and compounds with slight success. Careful reagent control, particularly in the case of copper sulphate and pine oil, seemed to give the best results.

### Lead Separation

Laboratory tests were run on the copper concentrate and a satisfactory lead separation was obtained. In order to check this a test run was made by dividing off the four small cleaner cells. Lack of conditioning and low cell capacity gave indifferent results and the project was discontinued. Further efforts to obtain a lead separation were tried by passing the copper concentrate over the pilot table. A good grade of lead concentrate, carrying extremely high values in gold and silver, was obtained. The over-all recovery was low and when it was taken into consideration that the gold and silver values were already recovered in the copper and concentrate, the cost of necessary equipment to obtain a lead concentrate was not justified.

Loading concentrates at Westholme, Vancouver Island, B. C.






## Ore Testing Equipment

The need for a well equipped ore testing laboratory in conjunction with the mill was recognized and a small room was partitioned off for this purpose during the construction of the mill.

Some difficulty was met in obtaining the equipment so that it was not until after the first of the year that the entire laboratory was assembled. Even at that late date much use was made of the equipment; it is felt that results obtained justified the expenditure.

Laboratory equipment consisted of 500 and 2000 gram batch type Denver "Sub-A" Flotation Machines, lab ball mill, vacuum filter, La Motte comparator and standard colorimetric comparison vials ranging upwards to a pH of 13.6, microscope, scales and miscellaneous laboratory equipment including a full set of ore testing reagents.

On each shift an operator was trained in the use of the comparator and thus accurate alkalinity control was maintained.

## Mill Data

### Table 1

### Reagent Consumption

| Reagent | Average April & May, 1944 Lbs. per ton | Average of 11 months Lbs. per ton |
|---|---|---|
| A. F. No. 31 | .005 | .009 |
| R. No. 208 | .09 | .09 |
| Pine Oil | .08 | .106 |
| Copper Sulphate | .53 | .704 |
| Zinc Sulphate | .406 | .36 |
| Soda Ash | 1.3 | 1.08 |
| Lime | 3.4 | 3.47 |
| Sod. Cyanide | .13 | .114 |
| Sod. Silicate | .20 | None |

### Table 2

### Consumption of Steel Wearing Parts

| Part | Lbs. per Ton |
|---|---|
| Jaw Crusher Plates | 0.0081 |
| Symons Cone Liners | 0.0065 |
| Ball Mill Liners | 0.384 |
| Grinding Balls | 2.06 |
| Filter Cloth | 120 days |

## Table 3

### Milling Costs

| Item | Avge. cost for 10-month period per ton milled |
|---|---|
| Crushing and Conveying | 0.2068 |
| Crushing and Conveying—repairs | 0.0403 |
| Grinding and Classifying | 0.6344 |
| Grinding and Classifying—repairs | 0.0338 |
| Flotation | 0.30 |
| Flotation repairs | 0.0225 |
| Reagents | 0.2005 |
| Filtration | 0.0842 |
| Filtration—repairs | 0.0084 |
| Loading Concentrates—mill | 0.0025 |
| Tailings Disposal | 0.0214 |
| Building Maintenance | 0.0013 |
| Water Supply | 0.0664 |
| Assaying | 0.1279 |
| Experimenting | 0.0365 |
| Supervision | 0.0707 |
| Mill Office Expense | 0.0047 |
| Mill General | 0.0456 |
| Total Ore Treatment | $1.9077 |

## Table 4

### Handling Concentrates

| Item | Average cost per ton milled over 10-month period |
|---|---|
| Trucking—Mill to Westholme | $ 0.2199 |
| Road Maintenance | 0.0216 |
| Loading—from Stockpile and at Westholme | 0.0149 |
| Miscellaneous | 0.0199 |
| Total | $ 0.2763 |

## Table 5

### Average Analysis 192 Hour Mill Run on Uniform Ore—See Final Flow Sheet

| Products | Av. per 24 hrs. | Wt. % | Analysis Au. Oz./T | Ag. Oz./T | % Cu. | % Zn. |
|---|---|---|---|---|---|---|
| Heads | 128 | 100 | 0.049 | 2.07 | 1.22 | 5.16 |
| Cu. Con. | 5.13 | 4.05 | 0.71 | 19.53 | 23.04 | 10.5 |
| Zn. Con. | 10.93 | 8.50 | 0.11 | 9.23 | 1.42 | 49.21 |
| Tails | 111.94 | 87.45 | 0.012 | 0.57 | 0.20 | 0.62 |

Cu. recovery: 75.49% in Copper Concentrates.
Zn. recovery: 81.22% in Zinc Concentrates.

## Table 6

### Screen Analysis—Classifier Overflow

| Mesh | Wt. Grams | % Distribution | % Cu. | % Zn. |
|---|---|---|---|---|
| Plus 65 | 2.5 | 0.5 | 2.4 | 9.0 |
| Minus 65 Plus 100 | 7.5 | 1.5 | 0.8 | 2.60 |
| Minus 100 Plus 150 | 55.0 | 11.0 | 1.09 | 4.50 |
| Minus 150 Plus 200 | 25.0 | 5.0 | 2.4 | 5.3 |
| Minus 200 | 410.0 | 82.0 | 1.45 | 5.9 |

500 gram sample used.

Microscopic Examination: (H. M. Wright and J. M. Cummings—Project No. O.D. 13, Progress Report No. 1, War Metals Research Board.)

No detailed study of this product was made, but the microscope showed considerable interlocking of chalcopyrite and sphalerite.

## Table 7

### Screen Analysis—Copper Concentrate

| Mesh | Wt. Grams | % Distribution | % Cu. | % Zn. |
|---|---|---|---|---|
| Plus 65 | 0.1 | 0.05 | ........ | ........ |
| Minus 65 Plus 100 | 0.2 | 0.10 | ........ | ........ |
| Minus 100 Plus 150 | 1.6 | 0.80 | 17.20 | 12.40 |
| Minus 150 Plus 200 | 11.3 | 5.65 | 20.30 | 13.70 |
| Minus 200 | 186.8 | 93.4 | 20.60 | 11.60 |

200 gram sample used.

Microscopic Examination: (H. M. Wright and J. M. Cummings; Project No. O.D. 13, Progress Report No. 1, War Metals Research Board.)

A statistical grain count indicated that at least 45 per cent of the sphalerite present was locked with chalcopyrite. The locked sphalerite is about 66 per cent minus 325 mesh and it is all finer than 200 mesh. The free sphalerite is in approximately the same mesh range.

## Table 8

### Screen Analysis—Zinc Concentrate

| Mesh | Wt. Grams | % Distribution | % Cu. | % Zn. |
|---|---|---|---|---|
| Plus 65 | 0.1 | 0.05 | ........ | ........ |
| Minus 65 Plus 100 | 0.2 | 0.1 | ........ | ........ |
| Minus 100 Plus 150 | 2.2 | 1.1 | 1.3 | 35.5 |
| Minus 150 Plus 200 | 9.6 | 4.8 | 1.2 | 46.2 |
| Minus 200 | 187.9 | 93.95 | 1.1 | 49.9 |

200 gram sample used.

Microscopic Examination: (H. M. Wright and J. M. Cummings—Project No. O.D. 13, Progress Report No. 1. War Metals Research Board.)

Of the chalcopyrite present in the zinc concentrate 40 per cent of it is locked with sphalerite.

## Table 9

### Screen Analysis—Final Tails

| Mesh | Wt. Grams | % Distribution | % Cu. | % Zn. |
|---|---|---|---|---|
| Plus 65 | 2.5 | 0.5 | 0.24 | 1.08 |
| Minus 65 Plus 100 | 7.5 | 1.5 | 0.12 | 3.6 |
| Minue 100 Plus 150 | 62.0 | 12.4 | 0.096 | 0.43 |
| Minus 150 Plus 200 | 28.0 | 5.6 | 0.22 | 0.96 |
| Minus 200 | 400.0 | 80.0 | 0.18 | 0.67 |

500 gram sample used.

Microscopic Examination: (H. M. Wright and J. M. Cummings—Project No. O.D. 13, Progress Report No. 1, War Metals Research Board.)

A study of the zinc tailings shows that about 85 per cent of the sphalerite is free, leaving only 15 per cent locked with chalcopyrite.

## Table 10

### Analysis of Cu Concentrate Shipped April 1944

(Average of 5 carload lots)

| Au. | Ag. | Cu. | Zn. |
|---|---|---|---|
| 0.834 Oz/T | 20.14 Oz/T | 20.58% | 13.84% |

Change of flow sheet (see Final Flow Sheet) for latter part of April gave following result:

| Au. | Ag. | Cu. | Zn. |
|---|---|---|---|
| 0.71 | 19.53 | 23.14 | 10.5 |

## Table 11

### Analysis of Zinc Concentrate Shipped March, 1944

(Average of 5 carload lots*)

| Au. | Ag. | Pb. | Cu. | Zn. | Fe. | Cd. | As. | Sb. |
|---|---|---|---|---|---|---|---|---|
| 0.091 | 7.9 | 0.2 | 1.2 | 49.6 | 2.85 | 0.205 | 0.24 | 0.06 |

*The above carload lots include some low grade zinc concentrates from stockpile.

# FLOW SHEET OF TWIN "J" CONCENTRATOR