671468

MYRA FALLS OPERATIONS

WESTERN MINES LIMITED

(Latitude 49° 35' N, Longitude 125° 36' W, Elevation 1250 feet)

LOCATION, ACCESS AND CLIMATE

The Myra Falls operations of Western Mines Limited are located near the south end of Buttle Lake, approximately thirty-five miles southwest of Campbell River, B.C.

Access to the property is by a paved road twenty-two miles in length which joins highway 28, twenty-three miles west of Campbell River.

Air temperatures reach a maximum of ninety degrees Fahrenheit in summer and a minimum of zero degrees Fahrenheit in winter. Annual precipitation is one hundred and twenty inches which includes fifteen feet of snowfall. Snow cover varies considerably, but may reach a depth of five feet at the plant elevation.

HISTORY AND OWNERSHIP

The claims were originally staked and prospected by James Cross and Associates of Victoria, B.C. The Paramount Mining Company excavated several trenches, drove an eighty foot adit and diamond drilled ten holes totalling 2,169 feet between 1919 and 1925. In 1930 reconnaissance geological mapping of the area was carried out for the Geological Survey, Canada. Several companies examined the property between 1946 and 1960.

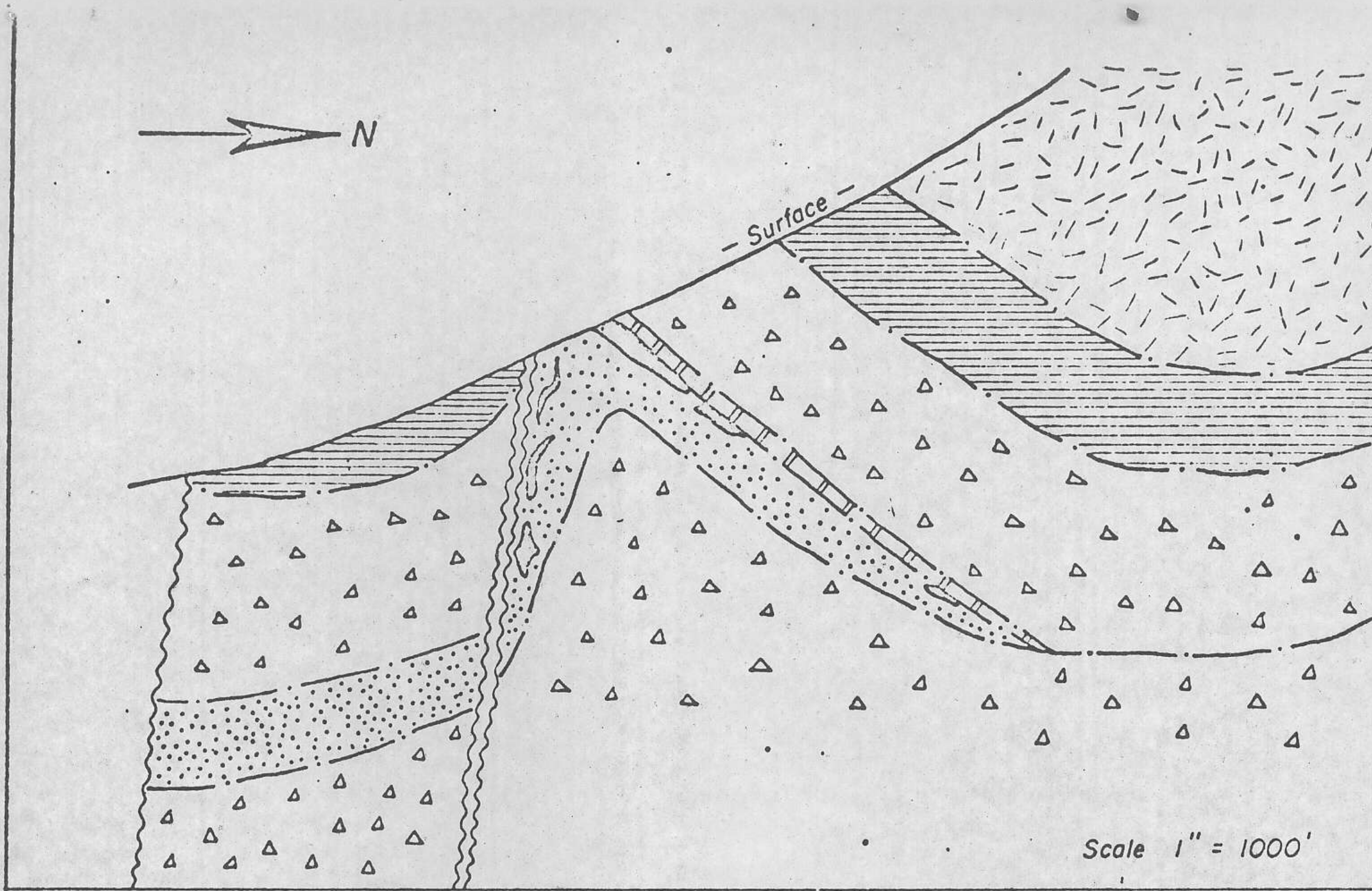
Western Mines Limited optioned the property from P.M. Reynolds and Associates in May, 1961 and commenced production in 1967. The property now consists of the Lynx and Myra underground mines and the Lynx open pit which are integrated to a 930 short tons per day flotation concentrator.

GEOLOGY

The Western Mines ore deposits occur near the north limit of a northwesterly trending belt of Permian volcanics near the geographical center of Vancouver Island.

These Permian volcanics, of the Sicker Group, are divided into three units in the mine area; namely, Vent formation, Sharp Banded Tuff formation and Dacitic Tuff formation.

The Vent formation, which contains the orebodies, is comprised of rhyolite and andesite flows and breccias and dacitic tuffs forming submarine volcanic piles. Figure 1, "A Generalized Cross-Section of Lynx Mine Area", shows the typical mine geology.



ORE
VENT. FM.
RHYOLITE
ANDESITE
SHARP BANDED TUFF FM.
DACITIC TUFF FM.
FAULT
"SOUTH WALL" FAULT
CONTACT

FIGURE I.

A GENERALIZED CROSS-SECTION OF LYNX MINE AREA

DRAWN BY ℬ

Faulting is the major structural feature and much of the ore occurs along a major northwesterly-southeasterly fault system. The orebodies are stratabound between an underlying rhyolite breccia and overlying andesite flow along the "north wall", and are dragged down along the "south wall" fault.

The ore lenses have a relatively small cross-sectional area but are remarkably persistent along strike. Ore minerals present are sphalerite, chalcopyrite, galena, tetrahedrite, and bornite in fine grained massive to sparsely disseminated lenses. Gangue minerals are quartz, sericite, chlorite, barite, calcite, and pyrite. Average ore grade is 0.10 ounces of gold per ton, 3.5 ounces of silver per ton, 1.6 percent copper, 1.0 percent lead and 7.5 percent zinc.

Alteration is persistent in the mine area and consists of sericitization, chloritization, bleaching, silicification, pyritization, and foliation with quartz sericite and quartz chlorite schists being formed.

PROPERTY OPERATION

The surface layout of the Myra Falls property is shown in Figure 2, "Surface Plan of Myra Falls Property". A centrally located flotation concentrator is fed from three centres of mining, namely the Lynx Mine, the Myra Mine, and the Lynx open pit. The administration and plant departments are housed in the general service building. The plant department operates hydro and diesel power plants and provides electrical and mechanical maintenance for all operating departments except the open pit which is mined by a contractor.

Personnel distribution as of December 31st, 1973 was as follows:

	Hourly	Staff	Total
Property Management		1	1
Mining	151	11	162
Concentrator and Assay Office	26	12	38
Plant and Surface	44	5	49
Office, Engineering and Geology		28	28
Sub Total	221	57	278
Contractors			11
Total			289

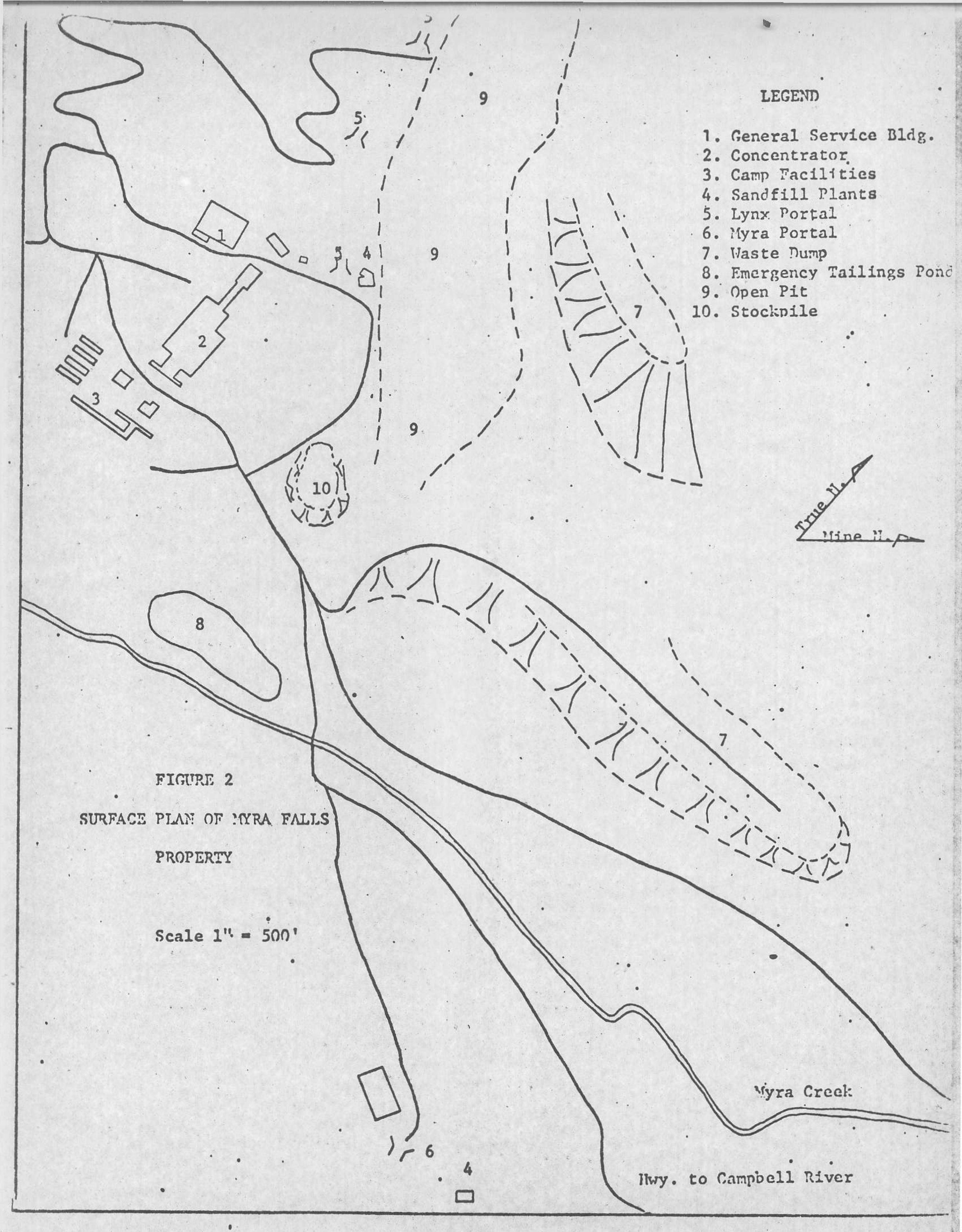


FIGURE 2

SURFACE PLAN OF MYRA FALLS PROPERTY

In 1973, the cost of processing 354,240 tons of ore was as follows:

	1973 Cost Per Short Ton Milled
Mining	$ 13.62
Milling	2.74
Plant Expense	2.28
Haulage and Terminal	.69
Administration, Housing, Head Office Expense	3.46
Total Operating Cost	$ 22.79

MINING OPERATION

Daily production as of December 31st, 1973 was 930 short tons per day. Source of ore in 1973 was 59 percent from the Lynx mine, 21 percent from the Myra mine and 20 percent from the Lynx open pit. Seventy-four percent of underground production was mined from 27 cut and fill stopes and the remainder from one longhole stope.

Cut and Fill mining is done in accord with the following parameters:

Maximum panel width	12 feet
Breast height	8 feet
Distance between millholes	100 feet average
Type of millholes	Prefabricated steel
Fill type	Classified mill tailings
Filling method	Hydraulic emplacement, tight filled to within three feet of the back.
Mucking floor	8 inches of 1:5 cement/ tailings mixture.

Longhole mining is done in accord with the following parameters:

Spacing of drawpoints	10 feet, centre to centre, on alternate sides of scram drifts
Spacing of slots	Approximately 85 feet centre to centre
Drilling method	Vertical rings (upholes only)
Ring burden	4 feet
Toe spacing	8 feet
Size of blastholes	2 inch diameter

Underground haulage is accomplished by battery powered locomotives pulling 60 cubic foot Granby-type cars. Haulage drifts are spaced at 150 foot vertical intervals.

Ore from the Lynx mine is hoisted in a $26\frac{1}{2}$ foot by $7\frac{1}{2}$ foot vertical shaft and hauled by train from the shaft collar to the concentrator. Myra ore is trucked up an $8\frac{1}{2}$ degree incline to surface and then to the concentrator.

Underground production equipment is tabulated in Table No. 1, "Mine Production Equipment".

Open pit production is performed on a contract basis in accord with the following parameters:

Average pit slope	$45^{\circ} - 50^{\circ}$
Bench height	18 feet
Safety berms	35 feet wide every 72 feet
Roadway width	35 feet
Blasthole size	5 inch diameter
Blasthole spacing	11 feet by 11 feet
Subgrade drilling	2 feet

Pit production equipment required for the removal of 65,000 cubic yards of material per month is as follows:

Two Northwest 80 D shovels

Two Joy TPH rotary drills

Three Gardner-Denver air tracs

Two caterpillar 769 B trucks

Seven haulpak 966 C trucks

One caterpillar D8 bulldozer

One caterpillar D7 bulldozer

Two caterpillar 988 front end loaders

One caterpillar 12E grader

One watering truck

Since start of operations 1,553,805 tons of ore and 14,694,544 tons of waste have been mined from the open pit.

Tabulation of Mine Operating Statistics

Production and development for 1973 was as follows:

	Myra Mine	Lynx Mine	Combined Since Start-up
Ore to concentrator (tons)	72,615	281,625	2,534,995
Broken ore reserves (tons)		43,849	
Horizontal Advance (feet)	5,338	4,852	68,871
Vertical Advance (feet)	2,277	2,712	29,496
Surface diamond drilling (feet)	9,288	13,129	76,044
Underground diamond drilling (feet)	39,499	41,948	566,206

Deferred development (1973)

Horizontal Advance (feet) 2,312

TABLE NO. 1

MINE PRODUCTION EQUIPMENT

FUNCTION	TYPE OF UNIT	NO. OF UNITS	OPERATING SHIFTS/DAY
Blasthole drilling	Gardner Denver DH-99	2	1
Development and stope drilling	Joy SAL 60M stopers	20	2
	Denver RB 83 stopers	17	2
	Joy AL 60M jacklegs	51	2
	Denver FL83 jacklegs	25	2
	Atlas Copco BBC 25 jacklegs	2	2
Underground level haulage	Assorted battery locomotives	18	2
	60 cubic foot Granby type cars	26	2
	30 cubic foot rocker type cars	8	1
	40 cubic foot Granby type cars	11	2
	52 cubic foot Granby type cars	6	2
	Getman Scootcretes	4	1
Mine haulage to concentrator	Battery locomotive	1	2
	90 cubic foot Granby type cars	8	2
Service hoisting	Ingersol Rand 1 drum 72" by 48"	1	2
Production hoisting	Nordberg 2 drum 84" by 54"	1	2
Decline haulage	Jarvis Clark JDT-410 trucks	3	2
LHD units	Wagner ST-2B scooptrams	2	3
Mucking machines	Eimco 12 B	1	1
	Eimco 21 B	11	2
	Eimco 630	2	1
	Atlas Copco LM56	3	2

TABLE NO. 1 con't

FUNCTION	TYPE OF UNIT	NO. OF UNITS	OPERATING SHIFTS/DAY
Service Vehicles	Unimog 411	1	2
	Allis Chalmers Grader	1	1
Slushers	Joy air $7\frac{1}{2}$ H.P.	1	2
	Joy air 15 H.P.	2	2
	Joy electric 10 H.P.	2	2
	Joy electric 20 H.P.	9	2
	Joy electric 25 H.P.	16	2
	C.I.R. air $7\frac{1}{2}$	4	2
	C.I.R. air 15 H.P.	2	2
	Sala electric 25 H.P.	1	2
	Sala electric 30 H.P.	1	2
	Gardner Denver 5H.P.	1	1

Mine personnel distribution as of December 31st, 1973 was as follows:

Miners	75
Timberman	10
Diamond Drillers	8
Trackmen	2
Mucking machine operators	9
Motormen	10
Hoistmen	3
Cagetenders	3
Truck Drivers	4
Fill men	8
Underground labourers	14
Shift bosses	12
Mine Captains	2
Mine superintendents	2
Open pit contractor	11
Total	173

The underground mines operate ten shifts per week and the open pit operates five shifts per week.

Undergound Mine costs for 1973 were as follows:

	Cost/short ton milled
Stoping	$ 3.54
Development	2.59
Haulage	.89
Hoisting	.22

Equipment maintenance	.44
Fixed and programmed	4.04
Geology, engineering & assaying	.71
Total underground	$ 12.43

MINERAL PROCESSING

The 930 short tons per day concentrator operates on a continuous basis. During 1973 the plant operated at rated capacity for 96.46% of the total available time.

The 200 ton per hour crushing and screening circuit is shown schematically in Figure 3, "Crushing and Screening Flowsheet for 930 TPD". This section of the plant operates 8 hours per day only.

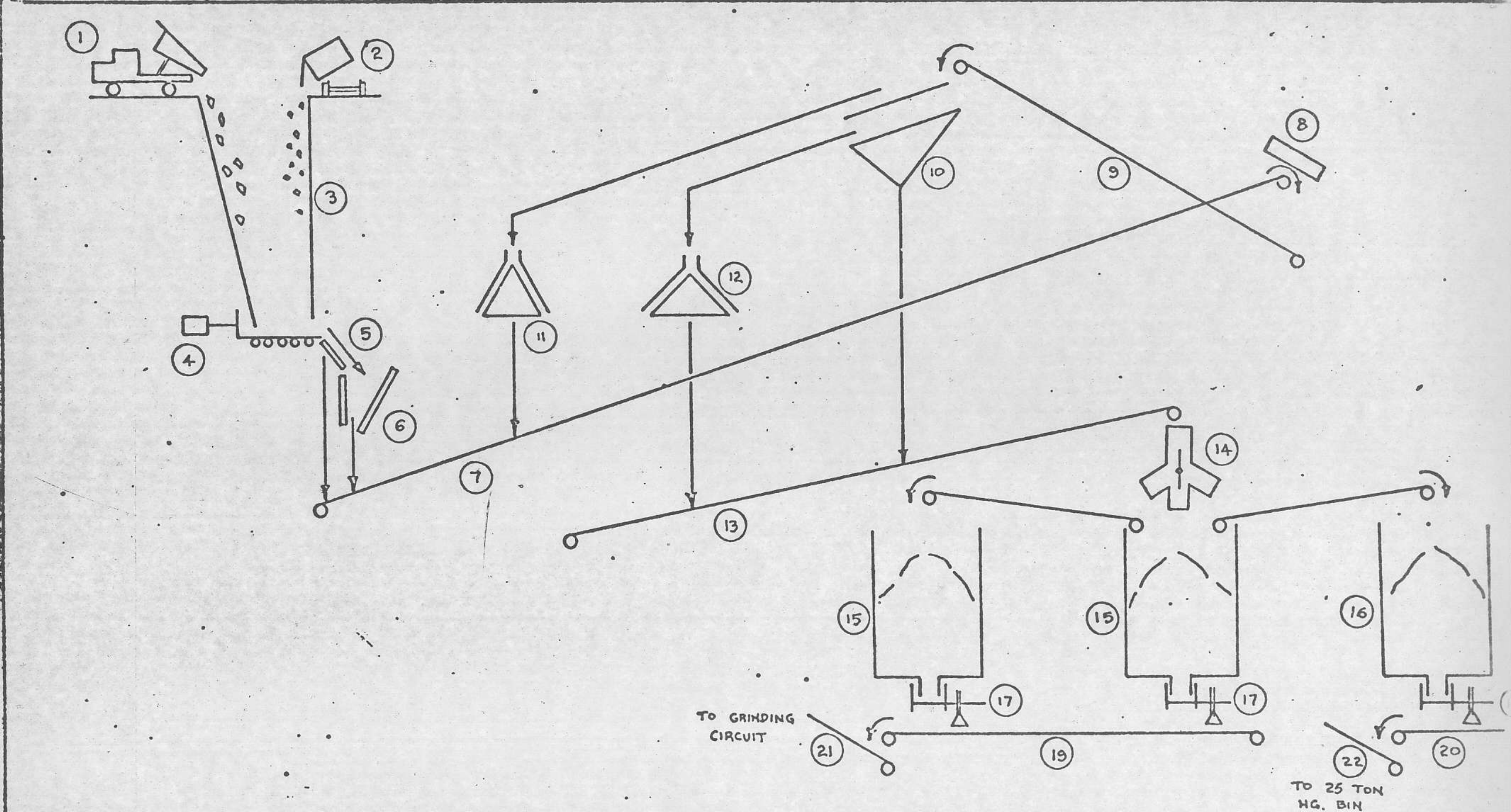
High grade ore is crushed once per week and stored in a separate fine ore bin.

The grinding and flotation sections consist of a high grade circuit shown in Figure 4, "High Grade Circuit Flowsheet for 80 TPD" and a main circuit shown in Figure 5, "Main Concentrator Flowsheet for 850 TPD". The former treats the Myra Falls ore containing high gold and silver values. Feed to the main circuit consists of Lynx mine, open pit and Myra Falls standard ores. The flow through the concentrator is summarized below.

High Grade Circuit

1. Crushed, minus $\frac{3}{4}$ inch discharge from the high grade fine ore bin is fed to a 25 ton surge bin. Feed from the surge bin is directed to a 50 H.P. powered open circuit rod mill, which in turn feeds a closed circuit 40 H.P. powered ball mill. The classified product, at approximately 70% minus 200 mesh, is fed to the flotation circuit at 45% solids.

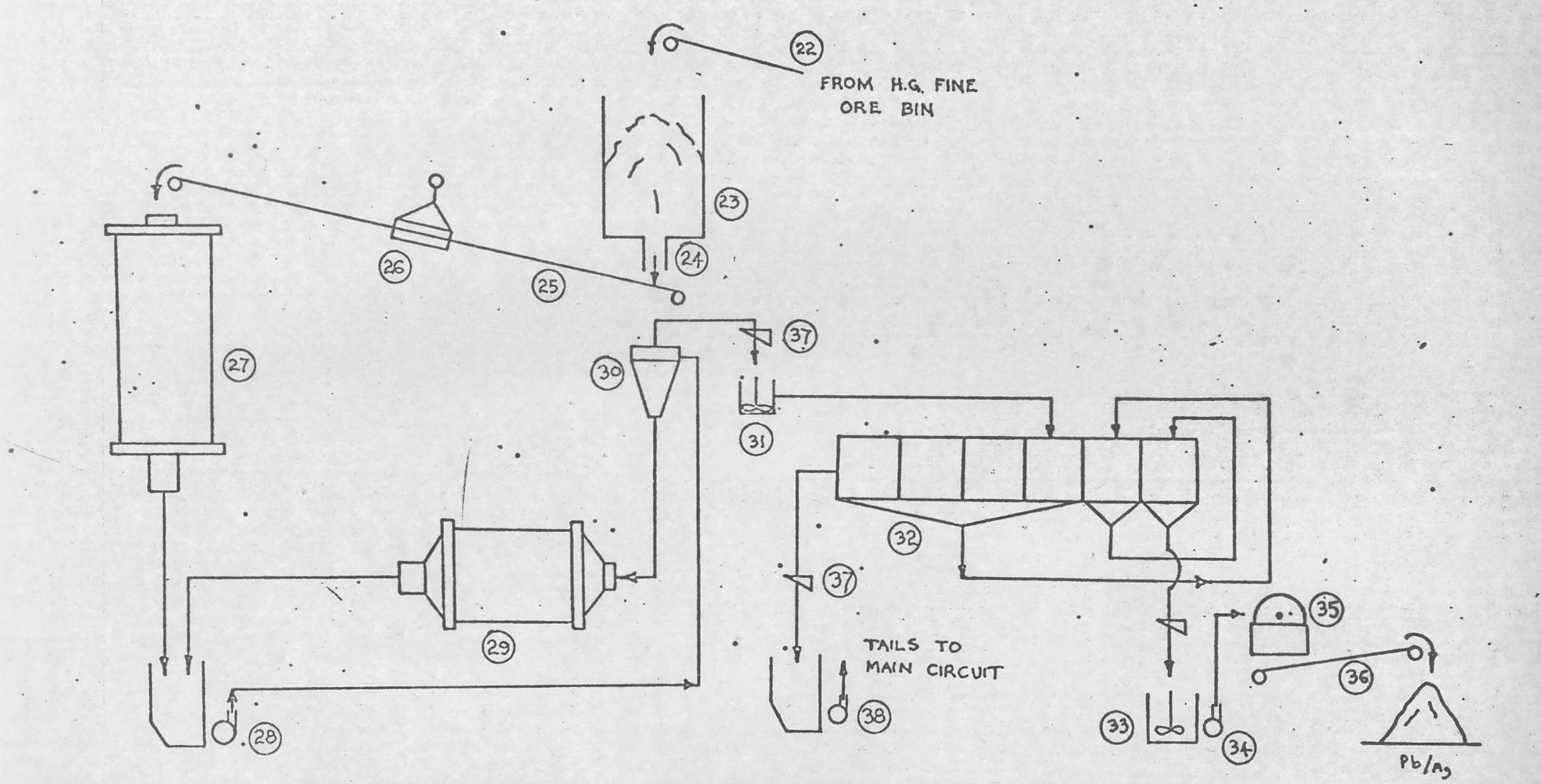
2. The high grade flotation circuit consists of one bank of Denver # 16 Sub A Cells. Four cells are used as roughers, followed by two stages of cleaning at one cell per stage. The concentrate is filtered in a two leaf disc filter, yielding a product approximately 40 - 50% lead at 7% moisture. Tailings from the high grade circuit are directed to the main circuit for recovery of copper and zinc.



ITEM	NO	DESCRIPTION	ITEM	NO	DESCRIPTION
1		ORE TRUCKS	14	1	3 WAY SPLITTER
2		UNDERGROUND ORE CARS	15	2	1200 T. FINE ORE BINS
3	1	400 T. COARSE ORE BIN	16	1	1200 T. HIGH GR. ORE BIN
4	1	PAN FEEDER	17	12	MEXICAN TUBE FEEDERS
5	1	4" GRIZZLY	18	3	MEXICAN TUBE FEEDERS
6	1	3'x4' JAW CRUSHER	19	2	30" CONVEYOR
7	1	30" CONVEYOR	20	1	30" CONVEYOR
8	1	ELECTROMAGNET	21	1	18" CONVEYOR
9	1	30" CONVEYOR	22	1	24" CONVEYOR
[illegible]	1	5'x12' DOUBLE DECK SCREEN			
[illegible]	1	SECONDARY CRUSHER - 4½' STD			
[illegible]	1	TERTIARY CRUSHER - 4' S.H.			
[illegible]	1	24" CONVEYOR			

FIG. No. 3

CRUSHING & SCREENING FLOWSHEET FOR [illegible]

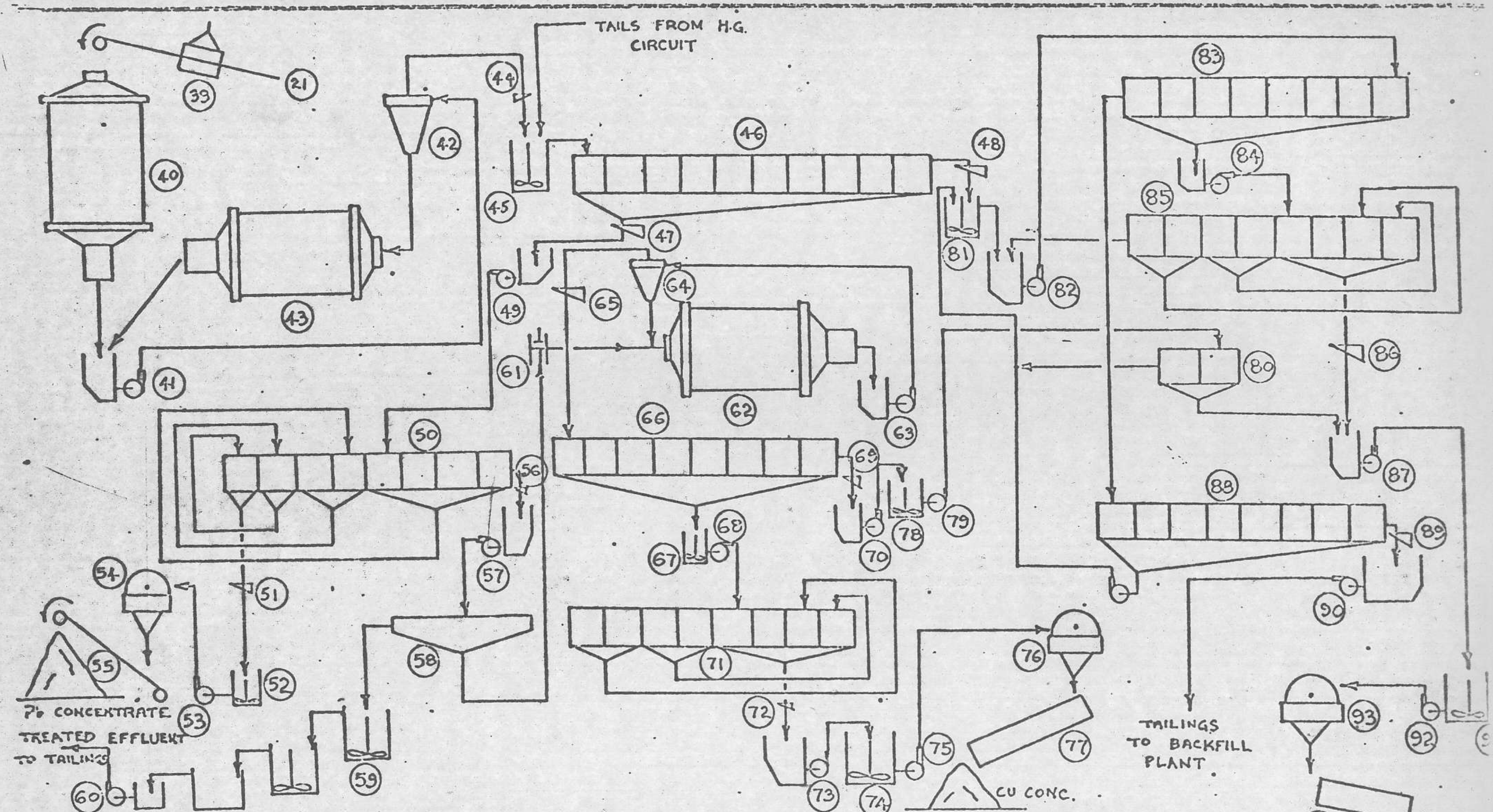


ITEM	NO	DESCRIPTION	ITEM	NO	DESCRIPTION
[illegible]	1	24" W CONVEYOR	35	1	4' DISC FILTER
[illegible]	1	25 T BIN	36	1	24" W CONVEYOR
[illegible]	1	SLOT FEEDER	37	3	SAMPLERS
[illegible]	1	18" W CONVEYOR	38	1	2'x2' SRL PUMP
[illegible]	1	WEIGHTOMETER			
[illegible]	1	3'x 8' ROD MILL			
[illegible]	1	3'x3' SRL PUMP			
[illegible]	1	4'x4' BALL MILL			
[illegible]	1	10" CYCLONE			
[illegible]	1	3'x3' CONDITIONER			
[illegible]	1	BANK - 6 No. 18 SUB A CELLS			
[illegible]	1	CONCENTRATE STOCK TANK			
[illegible]	1	2'x2' SRL PUMP			

FIG No. 4

HIGHGRADE CIRCUIT

FLOWSHEET FOR 30T.P.[illegible]



ITEM	NO	DESCRIPTION	ITEM	NO	DESCRIPTION	ITEM	NO	DESCRIPTION	ITEM	NO	DESCRIPTION
[illegible]	1	WEIGHTOMETER	53	1	2"x2" SRL PUMP	67	1	4'x6' CONDITIONER	81	1	8'x8' CONDITIONER
[illegible]	1	8'x12' ROD MILL	54	1	4' DISC FILTER	68	1	3"x3" SRL PUMP	82	1	6"x6" SRL PUMP
[illegible]	2	5"x6" SRL PUMPS	55	1	CONVEYOR	69	1	SAMPLER	83	1	BANK - Zn ROUGHER - 8 DR.
[illegible]	2	20" CYCLONES	56	1	SAMPLER	70	1	2"x2" SRL PUMP	84	1	3"x3" SRL PUMP
[illegible]	1	9'x9' BALL MILL	57	1	3"x3" SRL PUMP	71	1	BANK - Cu RECLEANERS - 8 SUB A	85	1	BANK - Zn CLEANER - 8 SUB A
[illegible]	1	SAMPLER	58	1	40' DOUBLE TRAY THICKENER	72	1	SAMPLER	86	1	SAMPLER
[illegible]	1	5'x5' CONDITIONER	59	3	CHLORINATION TANKS	73	1	2"x2" SRL PUMP	87	1	2"x2" SRL PUMP
[illegible]	1	BANK Cu-Pb ROUGH - 10 DR. CELLS	60	1	SRL PUMP	74	1	Cu. CON. STOCK TANK	88	1	BANK - Zn SCAVENGER - 8 DR.
[illegible]	1	SAMPLER	61	2	4" DIAPHRAGM PUMP	75	1	2"x2" SRL PUMP	89	1	SAMPLER
[illegible]	1	SAMPLER	62	1	7'x7' BALL MILL	76	1	6' DISC FILTER	90	1	8"x6" SRL TAILINGS PUMP
[illegible]	1	3"x3" SRL PUMP	63	1	3"x3" SRL PUMPS	77	1	3'x24' ROTARY DRYER	91	1	ZINC STOCK TANK
[illegible]	1	BANK - Cu/Pb SEPARATION - 10 SUB A	64	2	10" CYCLONES	78	1	5'x6' CONDITIONER	92	1	2"x2" SRL PUMP
[illegible]	1	SAMPLER	65	1	SAMPLER	79	1	3"x3" SRL PUMP	93	1	6' DISC FILTER
[illegible]	1	[illegible] CONCENTRATE STOCK TK	66	1	BANK Cu CLEANERS - 8 SUB A	80	1	BANK - SECONDARY ZINC - 2 DR.	94	1	4'x24' ROTARY DRYER

FIG. No. 5

MAIN CONCENTRATOR FLOWSHEET FOR 550 T[illegible]

Main Circuit

1. Crushed, minus $\frac{3}{4}$ inch material from the fine ore bins is fed to an open circuit 350 H.P. powered 8 by 12 ft. rod mill. The rod mill discharge feeds a closed circuit 400 H.P. powered ball mill. Cyclone overflow, at 60% minus 200 mesh, is fed to the Cu-Pb rougher cells at 45 to 47% solids.

2. The Cu-Pb rougher circuit consists of a single bank of 10 Denver # 24 DR cells. The cells are operated for maximum Cu-Pb recovery, with zinc and gangue being depressed. The tailings from the cells average 0.09% Cu, 0.15% Pb and 7% Zn. Approximately 35% of the total Zn reports to the Cu-Pb concentrate. This is subsequently removed at a later stage.

3. Concentrate from the Cu-Pb rougher cells is pumped to the Cu-Pb separation circuit. Copper is depressed with cyanide and a lead concentrate is recovered. The circuit consists of a bank of eight Denver # 24 Sub A Cells. Four cells are used as rougher-scavengers, the remaining four make up three cleaning stages. Final lead concentrate is pumped to the filter section wherein a three leaf disc filter is utilized to produce the end product. The concentrate averages 35 to 40% lead at 7% moisture.

4. The tailings from the Cu-Pb separation circuit are pumped to a 40 ft. thickener. This stage serves not only to increase density, but also to remove effluent high in cyanide. The latter is treated prior to disposal.

5. The 40 ft. thickener underflow at 35% solids is pumped to a regrind circuit. This consists of a closed circuit 150 H.P. powered ball mill. The cyclone overflow at 95% minus 200 mesh is fed to the copper circuit at 20% solids.

6. The copper circuit consists of one bank of cleaner cells (8 Denver # 24 Sub A) followed by a bank of recleaner cells. The latter consists of 8 Denver # 24 Sub A Cells. Recleaner tails are returned to the cleaners via the thickener. Tails from the copper circuit (cleaner cells) are pumped to the zinc circuit.

Copper concentrate is filtered in a four leaf disc filter followed by drying to approximately 6% moisture in a 3 ft. rotary dryer. Final concentrate averages 28% Cu by dry weight.

7. Feed to the zinc circuit consists of Cu-Pb rougher tails (refer 2 above) and copper circuit tails (refer 6 above). The latter is treated separately in a bank of two Denver # 24 DR Cells. Concentrate from these cells flows directly to the filter feed tank, the tailings are mixed with the Cu-Pb rougher tails feeding the main zinc circuit.

The main zinc circuit consists of one bank of eight # 24 Denver DR roughers followed by a similar bank of scavengers. Scavenger concentrate is returned to the rougher feed. Scavenger tails, at approximately 0.7% Zn, represents final plant tails, and are pumped to the tailings disposal system. Rougher concentrate is cleaned in a bank of eight Denver # 24 Sub A Cells. Cleaner tails return to the roughers, final concentrate is pumped to the filter feed tank. Zinc concentrate is filtered on a 6 leaf disc filter followed by drying in a 4 ft. rotary dryer. Concentrate averaged 52 to 54% zinc at 6 to 8% moisture.

8. Following filtration, all concentrates are directed to small holding bins of 200 to 400 ton capacity. From these bins the concentrate is trucked daily to the storage and ship loading facilities in Campbell River. Copper concentrates are loaded directly onto ships for delivery to Japanese smelters. Zinc concentrates are shipped either to Japan, or barged to Seattle for shipment to U.S. smelters. The lead concentrate is trucked to a rail siding in Courtenay B.C. for shipment via rail barge to Vancouver and then to Trail, B.C.

9. The use of instruments for control purposes is limited. The chlorination plant, completed in early 1974, includes a flowmeter, pH and ORP recorder. The balance of the operation is controlled by manual densities and pH. Clarkson feeders are employed for reagent addition. Daily assay samples are collected by automatic samplers. These are supplemented by manually taken control samples every four hours.

Tabulation of Mineral Processing Operating Statistics

The Milling Statistics for the twelve month period ending December 31st, 1973 are as follows:

Total Mill Feed (High Grade)	29,530 tons at .262 oz/T Au; 22.6 oz/T Ag 4.11% Pb
Total Mill Feed (Main Circuit)	351,709 tons at 1.37% Cu., .97% Pb., 8.22% Zn
High Grade Concentrate Produced	2,483 short dry tons at 2.52 oz/T Au; 2.22 oz/T Ag 44.2% Pb
High Grade Recoveries - Au	82.4%
Ag	84.2%
Pb	92.3%
Copper Concentrate Produced	13,590 short dry tons at 27.0% Cu.
Copper Recovery	76.3%
Lead Concentrate Produced	6,450 short dry tons at 37.7% Pb
Lead Recovery	71.0%
Zinc Concentrate Produced	44,736 short dry tons at 53.3% Zn
Zinc Recovery	82.5%
Grinding Media	1.9 lbs steel per ton milled

Reagent Usages:

Lime		3.7 lbs.per ton milled
Copper Sulfate		0.8 lbs.per ton milled
Depressants	($ZnSO_4$)	2.7 lbs. per ton milled
	(Na_2SO_3)	0.5 lbs. per ton milled
	(NaCN)	0.6 lbs. per ton milled
	(SO_2)	0.3 lbs. per ton milled
Collectors	(AF 208)	0.08 lbs. per ton milled
	(Amyl Xanthate)	0.1 lbs. per ton milled
	(Z 200)	0.03 lbs. per ton milled
Frother	(MIBC)	0.1 lbs. per ton milled
Soda Ash		0.06 lbs. per ton milled
Flocculant		0.01 lbs. per ton milled
Chlorine		1.2 lbs. per ton milled

<u>Mineral Processing Operating Crew Strength</u> = 37

	Hourly Rated	Staff
Operators	19	6
Maintenance	3	1
Metallurgy and Assaying	3	4
Superintendent		1
	25	12

Mineral Processing Operating Costs (1974)

	$/ton mill feed
Operating Labour	0.73
Maintenance Labour	0.15
Operating Supplies (incl. reagents)	1.53
Maintenance Supplies	0.20
Assaying	0.21
Pollution Control Supplies	0.24
Supervision and Overhead	0.70
Total Mineral Processing Operating Costs	$3.76 per ton milled

Tailings Disposal System

The waste material from the mill is divided into two flows. The main flow consists of the zinc tailings and includes all the solid material. Dissolved heavy metals, are very low in this portion of the tailings. The zinc tailings are cycloned prior to disposal. The cyclone underflow, containing approximately 50% of the total solids, is used as mine backfill. The overflow is pumped to a tailings raft on Buttle Lake. The secondary tailings flow from the mill comes from the 40 ft. thickener (refer 5 above). The overflow from this thickener contains practically all of the dissolved copper and cyanide which must be removed prior to disposal. This effluent is treated with chlorine in order to destroy the cyanide, breaking it down into N_2 and CO_2. The copper is precipitated out of solution by pH adjustment. The treated effluent flows into the main tailings line.

The tailings are directed, through a submerged outfall, to the bottom of Buttle Lake. A flocculant is added at the drop box located on a raft anchored several hundred feet off shore. The tailings are discharged from a 32 inch pipe approximately 80 ft. below the surface. The solids settle to the bottom of the lake, which is 100 ft. below the surface. No adverse effects on the lake have been recorded in approximately eight years of operation.