Mine Handout
Apr. '92

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WESTMIN RESOURCES LIMITED
MYRA FALLS CONCENTRATOR

INTRODUCTION

Westmin Resources Limited operates a 3650 mtpd mine and mill complex at the south end of Buttle Lake in Strathcona Provincial Park. Access to the mine site is provided by a 90 km paved highway from Campbell River.

The mine and facilities are nestled in the narrow Myra Valley between Mt. Phillips and Mt. Myra. These steep, rugged mountains rise from the valley floor, at the 300 meter elevation to summits of almost 1800 meters. The steep slopes are heavily wooded with fir, hemlock, and cedar up to 1200 meters where a transition to alpine meadows occurs. The summits are craggy rock faces, some capped with glacial ice. Myra Creek winds its way through the valley dropping several hundred feet into Buttle Lake. Precipitation is high at over 250 centimeters per year, which may include up to 5 meters of snow in winter. The temperature ranges from 32 C in summer to -18 C in winter.

HISTORY

The claims covering the Lynx and Price Mines were originally staked in 1918 by James Cross and Associated of Victoria. The claims covering the Myra Mine were held by the Paramount Mining Co. of Toronto. Some development work was done between 1919 and 1925 with inconclusive results. Interest in the claim group was rekindled in 1959 when the Lynx, Myra and Price claim groups were acquired and consolidated by the Reynolds Syndicate. In 1961 the claims were sold to Western Mines Ltd. Exploration by diamond drilling established underground ore deposits at the present Lynx Mine. An 1100 foot shaft was sunk with horizontal levels at 150 foot intervals to facilitate further exploration. The decision to bring the property into production was made in 1965 when the proven and probable ore reserves reached one million tons. Construction of the concentrator and surface facilities were completed in 1966. Operation commenced at 950 stpd, producing copper and zinc concentrates. Ore was supplied by both the Lynx open pit and underground mines.

Access to the property was by boat, float plane and barge on Buttle Lake until 1968 when a road, built by Western Mines, was completed along the east side of the lake. This 25 mile road linked up with the Gold River road and was paved in 1970. This road not only provided access to the mine but improved public access into Strathcona Park.

Development of the Myra Mine followed Lynx in 1970 when a 2200 foot, 8.5 degree decline was completed. Ore-bearing zones were accessed from levels at 150 foot intervals.

In 1972, to maximize gold and silver recovery from the high-grade Myra ore, production of a lead concentrate began in a separate circuit called the Myra circuit. Later, Myra circuit tails were combined with the Lynx circuit to recover copper and zinc values from the Myra circuit. Production of a Lynx lead concentrate soon followed.

Lynx and Myra ore reserves started a slow but steady decline in the late 1970's. Until then, annual exploration within the Lynx and Myra mines had been successful in maintaining reserves. An aggressive exploration program was launched which soon resulted in the discovery of the Price and HW ore bodies in 1979 and the Lynx West G zone in 1982.

Definition drilling of the HW orebody proved sufficient reserves to justify an increase in production to 3000 stpd. Flowsheet development for a new concentrator was started in 1981. Laboratory bench testing and operation of an on-site pilot plant provided the data for equipment selection and design. Construction of the concentrator was completed in May 1985. Shutdown of the original concentrator coincided with the start-up of the new concentrator on May 7, 1985. Production from the original concentrator facility totaled 6,476,910 tons.

In early 1987 Westmin decided to increase its throughput from 3000 stpd to 4400 stpd. Modifications to the concentrator were primarily in the flotation area. A new flowsheet was developed based on laboratory test results, materials handling capacity was increased and all of the concentrate thickeners were retrofitted with high capacity feed systems. The grinding circuit was fitted with expanded cyclone capacity and the speed and motor size of the rod mills was increased. Two additional Larox PF32 filters were added to increase dewatering capacity and ensure sufficient availability for filter maintenance. In July 1988, the expansion of the concentrator to 4000 tpd was completed. A Foxboro I/A control computer was installed in November 1989 and computer control of flotation circuit air flow was installed April 1990. Computer control of level is in the commissioning phase at the time of writing.

GEOLOGY

The Myra Falls ore bodies occur in the Myra formation of the Sicker Group. The Sicker group is the oldest stratigraphic unit recognized on Vancouver Island. The Myra formation is composed of variable sequences of differentiated and bedded volcanics, and sediments are volcanic greywacke with interbedded argillite and chert. (Walker)

In the Lynx-Myra-Price system, sedimentary massive sulfide lenses occur within or on the top contact of the host rhyolite beds. The HW system is also a sedimentary massive sulfide lens but is substantially larger than the Lynx-Myra-Price system and is located at the base of a rhyolite bed some 800 to 1000 feet stratigraphically lower than the Lynx-Myra-Price system. The orientation of the different ore zones with each other and the Myra Valley is shown in Figure 1.

The composition of the massive sulfide lenses vary substantially both within and among each ore lens. The most graphic example of the differences between lenses is that the HW ore body averages about 70% by weight pyrite while the Lynx-Myra-Price system averages about 15% by weight pyrite. Higher grade zinc ore is located on the perimeter and higher grade copper ore is near the center of the HW orebody.

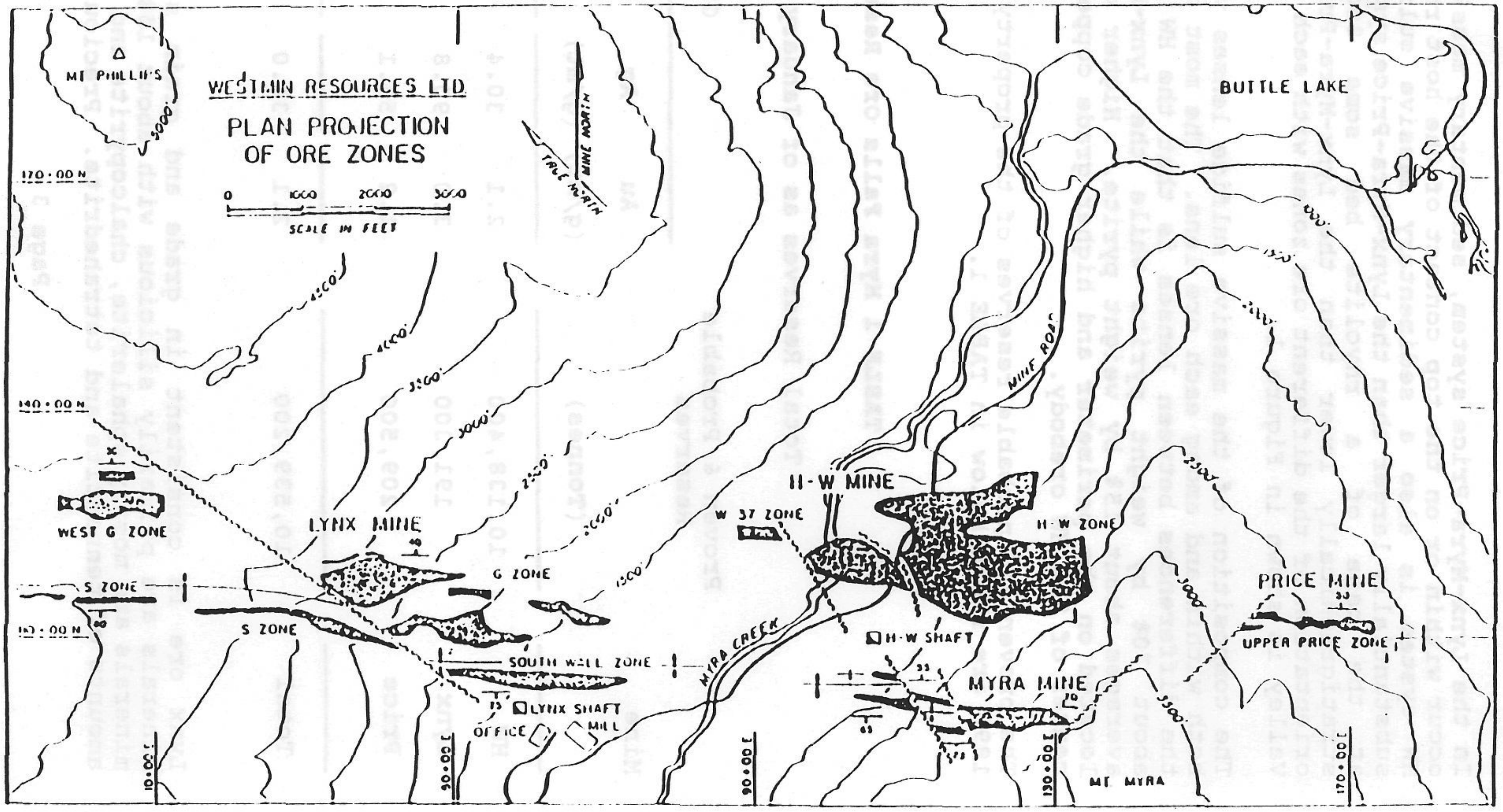
The proven and probable reserves of the property, as of January 1, 1990 are shown below in TABLE 1.

TABLE 1 Myra Falls Ore Reserve

Total Reserves as of January 1, 1990

Mine	Proven & Probable Reserves (Tonnes)	Grade Au (g/mt)	Ag (g/mt)	Cu %	Pb %	Zn %
HW	10,138,400	2.1	30.4	2.0	0.3	3.5
Lynx	191,300	3.0	90.8	1.7	0.9	9.1
Price	209,500	1.2	53.1	1.1	1.1	8.3
Total	10,539,200	2.1	32.0	2.0	0.3	3.7

Lynx ore is consistent in grade and grain size. The gangue minerals are primarily silicious with about 15% pyrite. The ore minerals are mostly sphalerite, chalcopyrite and galena, with minor amounts of tennantite and tetrahedrite. Precious metal values are


WESTMIN RESOURCES LTD.
PLAN PROJECTION
OF ORE ZONES
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1000
2000
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SCALE IN FEET
TRUE NORTH
MINE NORTH
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BUTTLE LAKE
WEST G ZONE
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SOUTH WALL ZONE
LYNX SHAFT
OFFICE
MILL
MYRA CREEK
MINE ROAD
W 37 ZONE
H-W MINE
H-W ZONE
H-W SHAFT
MYRA MINE
PRICE MINE
UPPER PRICE ZONE
MT. MYRA
170+00 N
140+00 N
110+00 N

FIGURE 1

CONCENTRATOR

FIGURE 2 shows the mill process.

Crushing

Lynx ore is crushed using the original crushing plant. Ore is feed from a 500 tonne coarse ore bin using a 48" x 12' hydrastroke feeder into a 48" x 36" jaw crusher. Product size from the Lynx crushing plant is minus 6".

HW ore is fed from the 100 tonne surge bin in the headframe onto a 1.4 km long conveyor using a 48" x 12' hydrastroke feeder. This belt discharges into a 3600 live tonne coarse ore bin. A 48" x 16' hydrastroke feeder and a weightometer on the coarse ore bin discharge controls the feed rate to the HW crushing plant. A weightometer totalizes tonnage from the Lynx crushing plant, which is added to the HW crushing plant feed. The HW crushing plant feed is first passed over a 5' x 12' double deck screen. Screen oversize feeds a 5.5' Symons standard cone crusher. Ore passing the 16mm x 28mm lower screen reports to the fine ore bins. Product from the secondary crusher reports to the tertiary screen, which is a 16mm x 28mm single deck. The tertiary crusher, a 5.5' Symons short head cone crusher, is in closed circuit with the tertiary screen. Tertiary screen undersize reports to two 3500 live tonne fine ore bins. In an effort to blend ore, a reversing conveyor alternates the HW crushing plant product between the two fine ore bins at regular intervals. The HW crushing plant has a nominal capacity of 270 tonnes per hour.

Grinding

The concentrator is designed with two parallel grinding and rougher circuits, each capable of treating 2000 tonnes per day. Both grinding lines are identical and independent. Rod mill feed is drawn from a fine ore bin using two hydraulically driven 60" slot feeders which discharge onto the rod mill feed belt. Mill feed tonnage is measured and feed rate is controlled with the hydraulic slot/belt feeders while rod mill water is ratioed to feed tonnage. Rod mill discharge density is controlled between 78% and 80% by operator checks and ratio adjustments. Rod mill discharge combines with ball mill discharge in a common pumpbox and is pumped with a fixed speed pump to a pair of 20" cyclones. Cyclone feed density is measured and controlled by cascading a pumpbox water flow setpoint from the cyclone feed density controller. Cyclone underflow is ball mill feed, which is 80% to 85% solids. Cyclone overflow is measured and is nominally 40% solids. Product size from the grinding circuit is 70% to 75% -200 mesh and the mean operating work index is 13.4 kwhr/tonne.

associated with the sulfides and are thought to be primarily micro-inclusions in the sulfides. Gold is mainly associated with chalcopyrite and silver is associated with galena, tennantite and tetrahedrite.

HW ore grades are extremely variable. The principal ore minerals are sphalerite, chalcopyrite and galena. Minor amounts of tennantite and bornite are also present. The gangue minerals are primarily pyrite with some barite, quartz and pyrrhotite. The principal gold carrier mineral is electrum with 22 to 30 wt% silver. Grain sizes of up to 50 micron and as small as 2 micron have been identified. Electrum appears to associate with bornite rich ores where it is found in myrmekitic intergrowths of galena with chalcocite. Finer grained electum, which is enclosed in tetrahedrite and tennantite, associates with sphalerite and pyrite. Gold is also in solid solution with other sulphides which are uniformly distributed throughout the ore body. The unprejudiced association of gold and silver with all of the sulfides has resulted in lower gold and silver recoveries from HW ore than was historically experienced from Lynx ore. It is felt lower recoveries are due to pyrite rejection to tails.

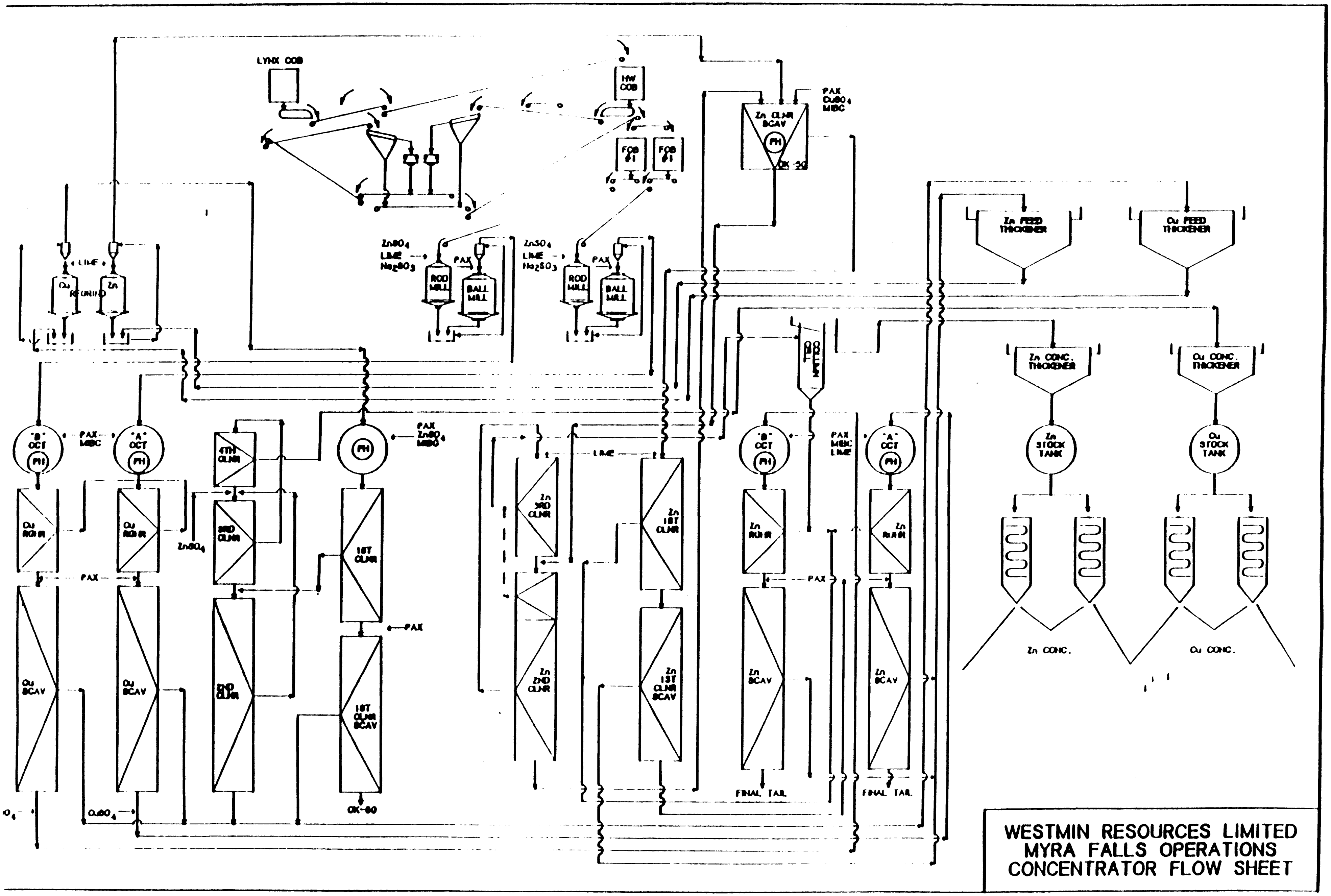
Variation of HW ore grain size is also extremely variable. Pyrite grains have been found from several millimeters to smaller than 10 microns in size. Polished sections show ore mineral grain size variation from a sphalerite matrix with 50 to plus 100 micron pyrite and chalcopyrite grains to a gangue matrix with less than 10 microns chalcopyrite grains and sphalerite grains between 50 and 200 microns.

MINING

Mining methods at Westmin vary with the size and dip of the ore zones. Cut and Fill and Avoca are used extensively at Lynx with some Room and Pillar mining where ground conditions and lens sizes permit. Mining at HW is primarily Room and Pillar, Retreat Sublevel Stoping and Blasthole and Longhole Stoping. Some Cut and Fill mining is also carried out in some of the steeply dipping areas.

Ore from Lynx is slushed, trammed and hoisted to an upper level where it is trammed by track to a surface coarse ore bin prior to crushing.

HW ore is transferred from stopes to ore passes using load-haul-dumps. The ore is trammed from the ore passes to a 1000 tonne surge bin. Due to the wide variation of stope ore grades the ore is blended underground by controlling tramming patterns. Ore is crushed underground to minus 6" in a 48" x 42" jaw crusher and is hoisted to a 100 tonne surge bin in the headframe using 10 tonne skips.



WESTMIN RESOURCES LIMITED
MYRA FALLS OPERATIONS
CONCENTRATOR FLOW SHEET

FIGURE 2

The copper and zinc scavenger concentrates are reground in essentially identical circuits. Both regrind mills are 7' x 12' Dominion rubber lined mills charged with 1" balls and driven by a 250 HP motor. The mills are in closed circuit with six 9" Linatex cyclones. Scavenger concentrates are thickened prior to regrinding and thickener underflow densities are controlled using underflow pump speed. Thickener underflow joins regrind mill discharge which feeds the regrind cyclones. Cyclone underflow ranges beteen 65% and 75% solids and cyclone overflow is nominally 35% solids. The copper and zinc regrinds feed their respective first cleaners. Grinding statistics are shown below in Table 2.

TABLE 2 Grinding Statistics

	Rod Mill	Ball Mill	Cu Regrind	Zn Regrinc
Manufacturer	Dominion	Dominion	Dominion	Dominions
Size, feet	8 x 12	11.5 x 15	7 x 12	7 x 12
Horsepower	400 Induction	1100 Synchronous	250 Induction	250 Induction
Critical speed, %	76	75	73	73
Grinding media	3.5" x 11.5'	Mix 20% 1.5" 50% 2.0" 30% 2.5"	1"	1"
	1060 steel	1060 steel	1060 steel	1060 steel
Consumption, kg/tonne	0.31	0.77	.15	.15
Liners	Noranda Wave 4" lift Ni-Hard	Rubber 5" lift	Rubber 4" lift	Rubber 4" lift
Product size (P80)	425 microns	190 microns	38 microns	38 microns

FLOTATION

Flotation feed grades are variable. Copper ranges from 1.2 to 4.5%, zinc ranges from 2.5% to 10.5%, lead ranges from 0.2% to 1%, and iron ranges from 10% to 38%. A great deal of effort goes into blending ores underground, however, rapid grade fluctuations do occur regularly. A Courier 300 on stream X-ray analyzer assays 16 streams for Cu, Pb, Zn, Fe, and %solids. Assays are reported every 5 minutes with each stream being reassayed every 15 minutes. Sixty- three Outokumpu OK8 (300 cu ft) machines and one OK50 (1765 cu ft) flotation machine are used in the flotation circuit. Air and level control for each bank of cells is managed by the Foxboro computer. The primary sensor for air control is an annubar and the final control element is an air actuated butterfly valve. The primary sensor for level control is an Outokumpu float and angle transmitter and the final control element is an air actuated dart valve. The originally supplied Outokumpu field controllers are maintained for backup, startup and shutdown. Nearly all flotation reagents flows are measured by magnetic flow meters and metered into the circuit using small ball valves, which are regulated by the Foxboro computer. In some cases, one flow loop serves two addition points by splitting the flow with small in-house design pinch valves. MIBC and lime are the only reagent flows not measured. MIBC is metered with pulsa feeder pumps and lime is pulsed into the circuit using air actuated red jacket valves. Lime is added to a pH setpoint. All reagent setpoint changes are made by operators. However, feedforward control of collector to the copper roughers is currently being evaluated.

Copper

There are two independent identical copper rougher circuits, one for each grinding line. Each cyclone overflow is gravity fed to its own 400 cu ft conditioner at the head of each copper rougher circuit's ten OK8 cells. Each copper rougher is split into three independent banks. The first two banks are the first and second roughers which have two cells per bank. The last bank is the rougher scavenger which has six cells. The pH in the roughers is controlled between 10.5 and 11.2, depending on lead and iron heads. The primary sensor is a pH probe in the rougher conditioner; a red jacket valve pulses lime into the rod mill feed chute. An equal mixture, by weight, of Potassium Amyl Xanthate (PAX) and Aerofloat 208 are stage added. The collector blend, called MIX, is added at the ball mill feed chute when Cu heads are high. MIX is also added at the copper conditioner and the copper rougher scavenger transition box. Zinc sulfate and sodium sulfite are added to the rod mill for sphalerite and galena depression. Zinc sulfate is used for zinc depression when the iron heads are low and sodium is used when lead is high but is limited as zinc rougher recovery is affected. MIBC is also added to each copper rougher conditioner.

Copper rougher concentrates from both circuits are combined and pumped to the copper second or third cleaner feed depending on grade. Rougher scavenger concentrates from both circuits are combined and pumped to the 40' copper regrind feed thickener. Thickener underflow at 35% solids is pumped to the copper regrind pumpbox where zinc sulfite is added to depress zinc, sodium sulfite is added to depress lead, MIX is added to reactivate copper, and lime is added should the pH drop below 10.6 which helps aid lead depression. The copper cleaners' four stages of countercurrent cleaning have a total capacity of ten OK8 cells. The fourth cleaner is a single OK8, the third cleaner bank has two OK8 cells, the second cleaner bank has four OK8 cells, and the first cleaner bank has three OK8 cells. Copper regrind cyclone overflow reports to the copper cleaner conditioner which feeds the copper first cleaner. Copper first cleaner tail feeds the copper cleaner scavenger whose tail reports to the zinc rougher conditioners and whose concentrate reports to the copper regrind feed thickener. The cleaner scavenger bank has three OK8 cells. The copper final concentrate reports to the 32' copper concentrate thickener.

Zinc

Copper rougher scavenger tail and copper cleaner scavenger tail are pumped to the 400 cu ft zinc rougher conditioners. There are two independent identical zinc rougher circuits which follow from the copper roughers. Note, this requires the copper cleaner scavenger tail, which reports to the zinc rougher conditioners and is a single stream, to be split. Each zinc rougher has a total of ten OK8 cells. The first four cells make up the rougher bank, and the last six make up the rougher scavenger bank. Copper sulfate is split between each copper rougher scavenger tail box and each zinc rougher scavenger drop box. Lime is pulsed into each zinc rougher conditioner, where the pH is controlled between 11.5 and 11.8, using red a valve. The alkaline pH is required to depress iron. MIX is also split between each zinc rougher conditioner and each zinc rougher scavenger drop box. MIBC is also added to each zinc rougher conditioner.

Zinc rougher concentrate from both circuits are combined and pumped to the zinc second cleaner feed. Zinc rougher scavenger concentrate from both circuits are combined and pumped to the 40' zinc regrind feed thickener. Thickener underflow at 30% solids is pumped to the zinc regrind pumpbox where copper sulfate and MIX are added to reactivate zinc, and lime is added to raise the pH to 12.0 which helps aid iron depression. The zinc cleaners' three stages of countercurrent cleaning have a total capacity of twelve OK8 cells and one OK50 cell. The third cleaner has two OK8 cells, the second cleaner bank has four cells, the first cleaner has one OK50 and three OK8 cells. Lime is added to the third cleaner feed to bring the pH up to 12.3 to depress iron. Zinc regrind cyclone overflow reports to the zinc first cleaner. Zinc first cleaner tail feeds the zinc cleaner scavenger whose tail is split and reports to the zinc rougher scavenger drop box and whose concentrate reports to the zinc regrind feed thickener. The zinc cleaner scavenger bank is three OK8 cells. The zinc final concentrate reports to the 32' zinc concentrate thickener.

Table 3 shows typical metallurgical performance of the Concentrator. Table 4 shows typical reagent consumptions.

TABLE 3 Typical Metallurgical Performance

	ASSAYS					RECOVERY				
	Au	Ag	Cu	Pb	Zn	Au	Ag	Cu	Pb	Zn
Head	2.19	30.3	2.02	0.23	3.65					
Cu Conc	9.83	201.4	23.86	1.14	4.25	33.63	49.78	88.40	37.83	8.73
Zn Conc	3.44	74.9	1.51	1.68	49.23	9.40	14.77	4.47	44.42	80.65
Tail	1.44	12.4	0.17	0.05	0.45	56.97	35.45	7.13	17.76	10.62

TABLE 4 Typical Reagent Consumption

REAGENT	CONSUMPTION Kg/tonne
Aeroflot 208	0.054
Potassium Amyl Xanthate	0.079
Copper Sulfate	0.48
Sodium Sulfite	0.008
Zinc Sulfate	0.087
Lime	3.26 Includes Surface Water Treatment
Superfloc 1201	0.017 Includes Tailing Thickener

to the mill water head tanks at this point. An annubar flow meter measures the flow. Nearly 80% of the water used by the concentrator is reclaimed from the surface water treatment system. Excess water, which represents anywhere from 1 to 5 times the water used by the concentrator, is discharged into Buttle Lake. Discharge pH is measured and a weir is used to measure the total flow discharged to the lake. Figure 3 shows the surface water treatment system layout.

Power Generation

The Myra Falls Operations use between 15MW of power during the winter and 12 MW of power during the summer. One unique feature is the power generation facilities as all power is generated on site.

There are two lakes above the mine site, called the Jim Mitchell and Tennent Lakes. The Jim Mitchell feeds the Thelwood power station. The 8.2 MW Thelwood generator is driven by a Gilkes impulse pelton wheel. At maximum capacity the pelton wheel uses two jets of water, supplied at 580 psi. Maximum water consumption is 39,200 USGPM. A Woodward UG8 govenor controls the pelton wheel's output and by using only one jet the generator can be operated down to 1 MW. Tennent Lake feeds the Tennent power station. The 3.0 MW Tennent generator is driven by a Gilkes impulse pelton wheel, similar to the Thelwood unit; however, the Tennent pelton wheel is smaller and only has a single jet. It is also controlled by a Woodward UG8 governor. The Tennent, whose water is supplied at 2000 psi, can operate as low as 0.5 MW.

In addition to the hydro plants, a 9 MW diesel generating plant is maintained. During the wet months both hydro plants operate at maximum capacity and diesel generators provide the remaining 4 MW of power. During the dry season the diesel plant is operated to capacity and several diesel air compressors are rented to replace existing electric compressors. In recent years up to 4 MW of additional diesel capacity has been rented to maintain operations. Depending on lake levels as little as 0.5 MW of hydro electric power is available during the dry season.

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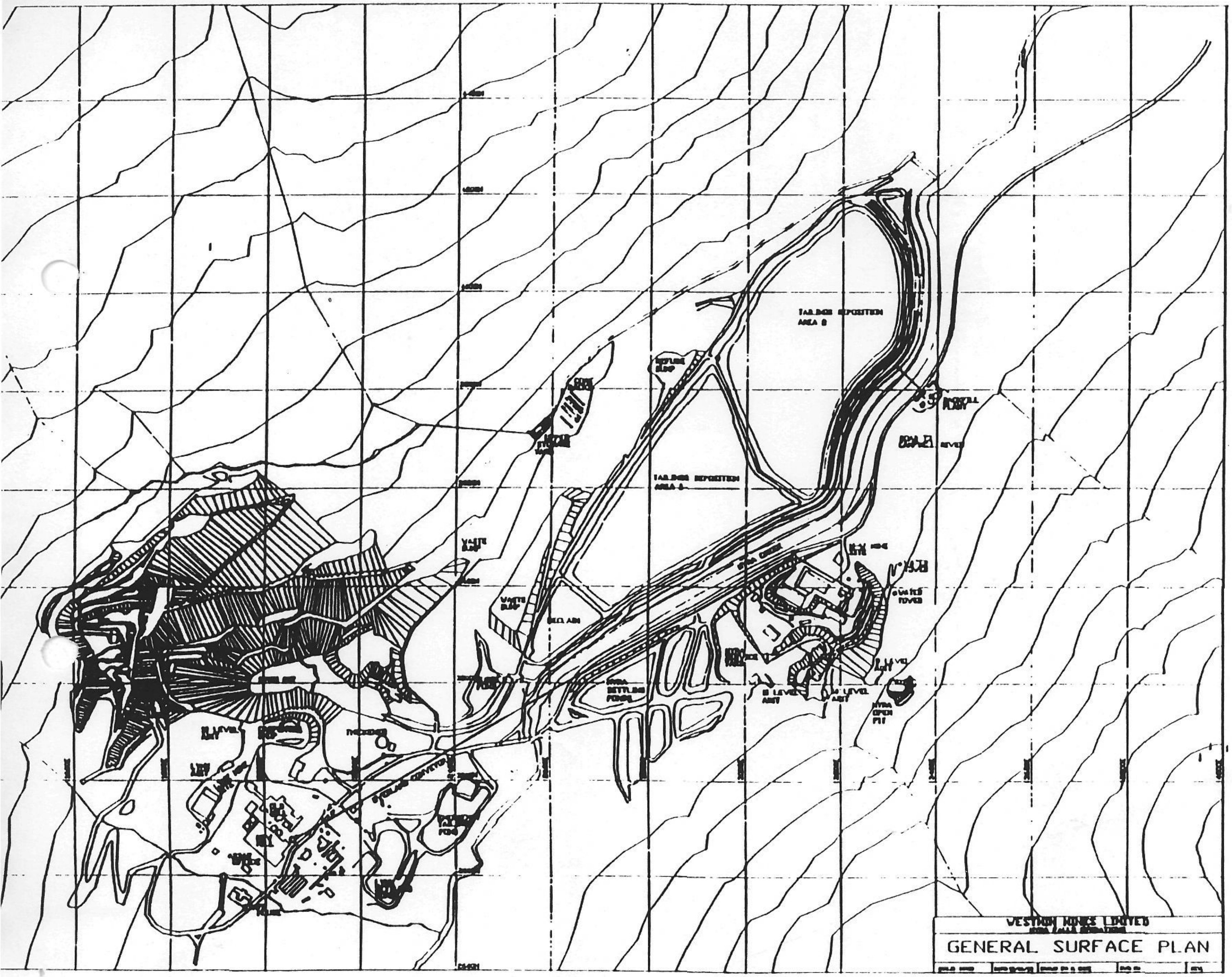

FIGURE 3