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GIBRALTAR MINES LIMITED

Technical Information Brochure

July, 1992

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GIBRALTAR MINES LTD.

Location, Access and Climate

The Gibraltar Mine is located approximately 100 miles (161 kilometers) south of Prince George, B.C. on the westerly slope of Granite Mountain and near McLeese Lake. (See Figure 1 - Gibraltar Mines - Location Map)

Access is by way of a 10 mile (16 kilometer) paved highway that joins Highway 97 near the north end of McLeese Lake.

The ambient air temperature ranges from a winter low of minus 38° F. (38° C.) to a summer maximum of 90° F. (32° C.).

The annual precipitation at the minesite is approximately 24 inches (610 mm.) of which 33% falls as snow. A maximum snow cover of 3 feet (1 meter) occurs in late February.

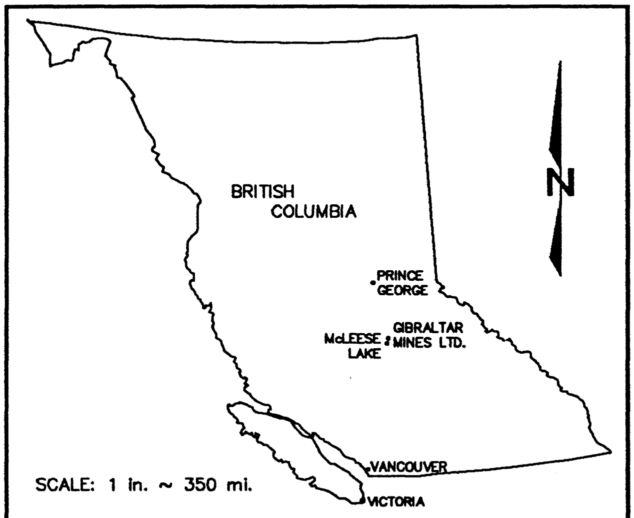

FIGURE 1: GIBRALTAR MINES LTD. LOCATION MAP

History and Ownership

The Gibraltar and Pollyanna properties were, for the most part, explored separately until 1969, at which time they received a combined exploration program.

Gibraltar Property

The Gibraltar property was discovered in 1927 and was known as the Hill property. (B.C.M.M. report 1928). In 1957 Kimaclo Mines Ltd. (N.P.L.) drove a 112 foot (34 meter) adit into a mineralized "shear" called the "Sunset Showings". This was later known as Gibraltar West. During 1958 the property was sold to Major Mines Ltd. (N.P.L.) who then allowed the property to lapse. The property was restaked by J. Hilton on January 1, 1962 who then optioned it to Keevil Mines. Keevil Mines allowed their option to lapse in 1964 whereupon Gibraltar Mines Ltd. (N.P.L.) acquired the property from Mr. Hilton. Cominco then optioned the property from Gibraltar, and in partnership with Mitsubishi outlined the Gibraltar West zone. They terminated their option in 1967. The property was then optioned to Canex-Duval, who were exploring the adjacent Pollyanna property.

Pollyanna Property

Pollyanna was discovered in 1910 and was known as the Rainbow group. After minor work the property was allowed to lapse. It has been known as the Pollyanna since 1925. In 1949 the showings were staked as the Copper King. During this time a shipment of 1000 pounds (450 kg) assaying 10.5 percent copper was made.

Kimaclo Mines restaked the showing in 1954 and allowed it to lapse in 1956. Mr. Robert Glen staked this property in 1963 and optioned it to Keevil Mining Co. who in turn dropped the option in 1964. Duval Corporation Ltd. then optioned the property. In 1967 Canex-Aerial Exploration participated with Duval on an equal basis to explore the claims.

Later, Canex bought Duval's interest and by 1970 held more than 70% of the issued shares of Gibraltar Mines Ltd. (N.P.L.)

Geology

1. Regional Geology

In the vicinity of Granite Mountain (Figure 2 - Generalized Regional Geology), the oldest rocks are a Permian aged regionally metamorphosed sedimentary and volcanic sequence. These rocks are of the Cache Creek group (Tipper, 1959).

Batholithic intrusives of Jurassic-Cretaceous age intrude the Cache Creek group in the Granite Mountain area. An intermittent north-south trending line of batholithic rocks outcrops from Prince George on the east side of the Fraser River fault system. The batholith is composed of granodiorite, quartz diorite, diorite and gniesse. In the immediate vicinity of Granite Mountain, a regionally foliated and metamorphosed quartz diorite occurs which has a chlorite rich diorite margin against rocks of the Cache Creek group. Calcareous members of the latter show some skarn development adjacent to the diorite contact. It is within regionally metamorphosed and foliated quartz diorites that the Gibraltar-Pollyanna copper-molybdenum deposits occur.

2. Geology of the Gibraltar Copper Deposits

The four copper/molybdenum orebodies (Figure 3 - Generalized Geology Gibraltar Area) at Gibraltar Mines are known as Gibraltar East, Gibraltar West, Pollyanna and Granite Lake. The distance between the west end of the Gibraltar East pit and the east end of the Granite Lake pit is approximately 2.5 miles (4.0 kilometers). The distance from the north side of the Pollyanna zone to the south side of the Granite Lake pit is 1.0 mile (1.6 kilometers). These four pits lie entirely within the quartz diorite of the Granite Mountain pluton.

3. Rocks of the Granite Mountain Pluton

Quartz Diorite

The quartz diorite of the Granite Mountain pluton is extremely uniform in its mineral assemblage, but varies in the degree of cataclastic deformation and alteration.

This rock is composed of quartz (25-30%), "plagioclase", which is presently a mixture of albite-epidote-zoisite-muscovite (50-55%), chlorite (20%), which originally was biotite with minor hornblende, and disseminated magnetite (1% or less). The rock is equigranular and generally has a grain size of 0.1 inches (2-4 mm.). The grain size varies and, in the vicinity of the tailings ponds, may be as much as 0.4 inches (10 mm.).

Throughout the Granite Mountain pluton the quartz diorite has been regionally foliated and altered. Mineralogical and textural changes vary according to the degree of cataclastic deformation. When the foliation is weakly developed, the mineral association is quartz-albite-muscovite-chlorite, epidote, zoisite-magnetite. As the intensity of foliation increases the quartz diorite eventually becomes schist like. The mineral assemblage is quartz-oligoclase-chlorite-muscovite plus or minus magnetite. Epidote content decreases as the intensity of the foliation increases.

Locally, "shear zones" within the above rock contain an assemblage of quartz-feldspar-garnet-chlorite.

4. Rocks Intrusive into the Quartz Diorite

Within the saussuritized and foliated quartz diorites of the Granite Mountain pluton, there are three pre-mineral and one post-mineral intrusive rocks. These are:

i) White Quartz Diorite

A rock which exhibits both a sharp and a gradational contact against the saussuritized quartz diorite is a leucocratic porphyritic quartz diorite. This rock is composed of quartz (30%) and saussuritized plagioclase with albitic rims (45%) where the grains range from 0.04 to 0.20 inches (1 to 5 mm.). The remaining 25% of the rock is a fine grained mosaic of quartz and albitic plagioclase with about 1-2% chlorite. This rock is nondirectional to moderately well foliated and is saussuritized.

ii) Quartz-Feldspar Porphyry

A more definite porphyry occurs in the same areas as the leucocratic phase. The quartz-feldspar porphyry crosscuts the saussuritized quartz diorite and apparently crosscuts the leucocratic phase. This latter relationship has not been conclusively demonstrated. The porphyry is composed of 0.12 to 0.20 inch (3 to 5 mm.) phenocrysts of quartz, (30%), and albitic plagioclase (10%) in a white asphanitic matrix. This matrix is composed of quartz, albitic plagioclase and some carbonate, muscovite and zoisite. The rock is not altered.

LEGEND

988	PLEISTOCENE and RECENT Till, gravel, sand, and clay
7	MIOCENE AND PLIOCENE Basalt
685	EOCENE and/or OLIGOCENE Basalt, tuff, brecia, conglomerate, shale, and sandstone
4	PALEOCENE and/or EOCENE Rhyolite, dacite, tuff, and breccia
1	PERMIAN- CACHE CREEK GROUP Chert, argillite, limestone, and greywacke
A	INTRUSIONS JURASSIC-CRETACEOUS Granodiorite, diorite, quartz diorite, and related gneisses

Scale: 1 in. = 7.2 mi. (approx.)

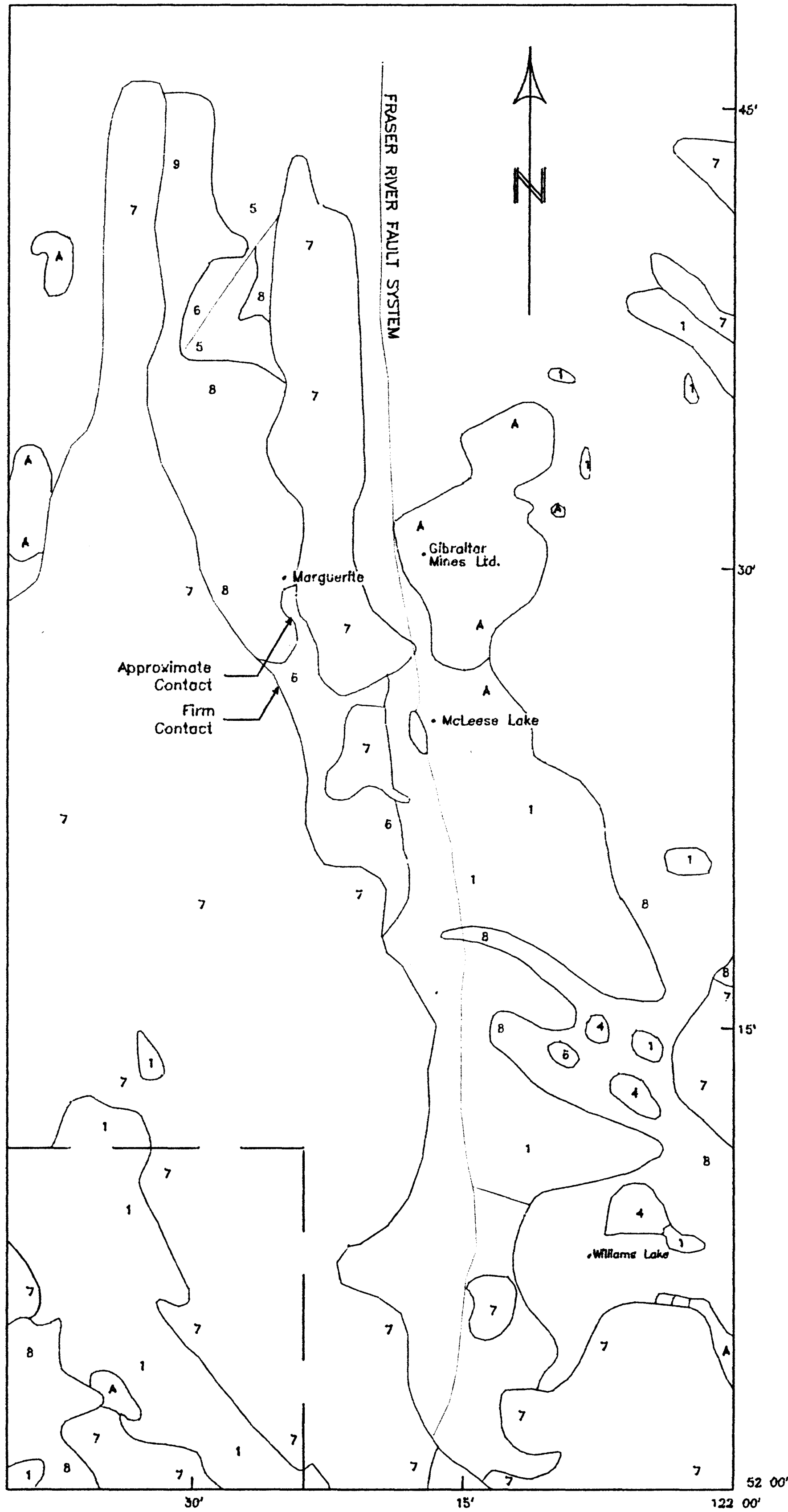

FIGURE 2: GENERALIZED REGIONAL GEOLOGY

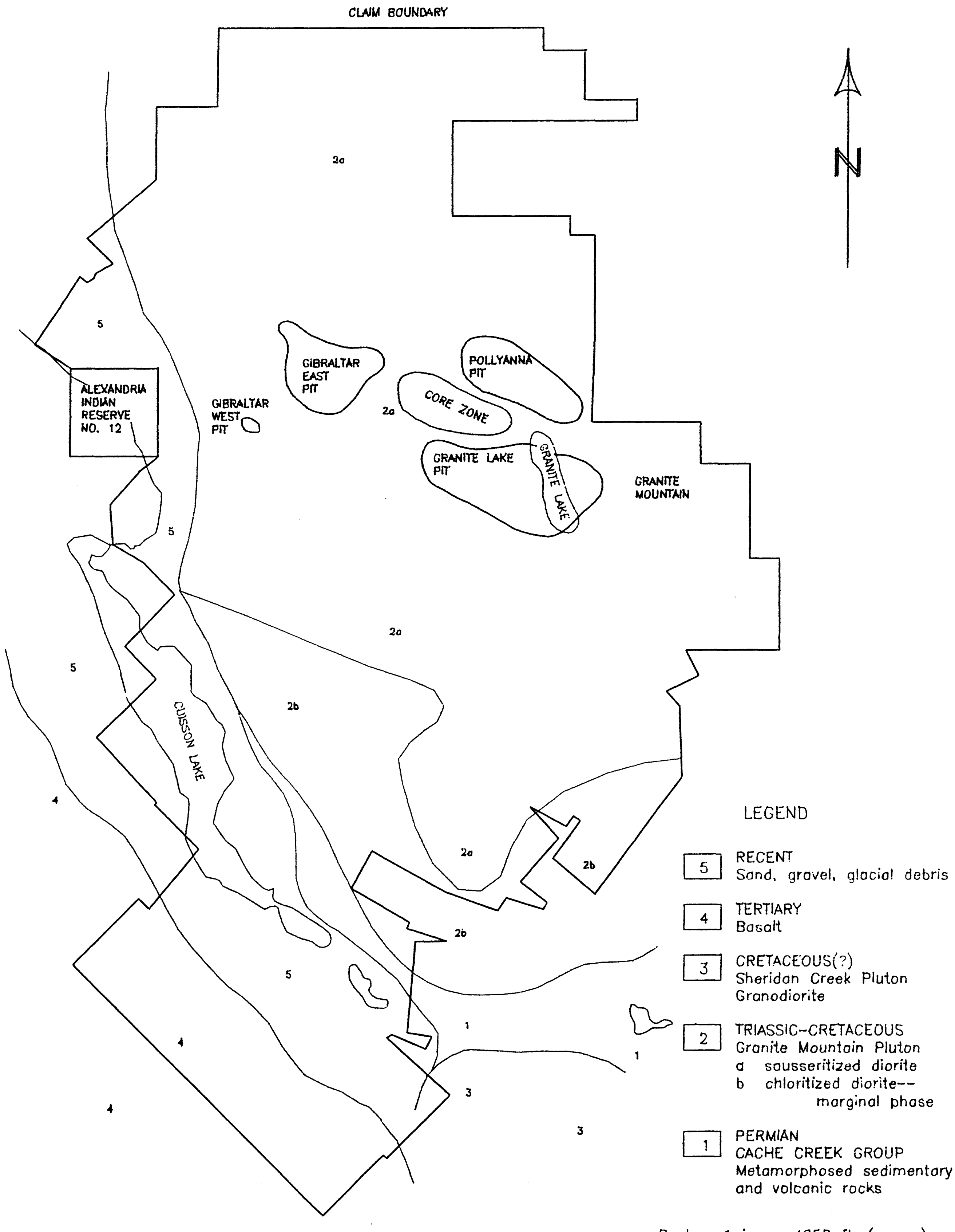

FIGURE 3: GENERALIZED GEOLOGY--GIBRALTAR AREA

iii) Aplite

Another intrusive phase is aplite which is very fine grained and has a sugary texture. This rock is also not altered.

iv) Hornblende Dacite

There has only been one post mineralization rock encountered to date. This rock is a very fine grained chloritized hornblende dacite and is found in the Gibraltar East zone.

Mineralization and Hydrothermal Alteration

As outlined in the section on structure, there are at least four stages of veining recognized within the Gibraltar deposits. The four stages of veins are dependent on their relative age relations, which means that the first listed is crosscut by the next listed feature. The four stages are summarized below.

V1 - i - Quartz-pyrite plus or minus chalcopyrite WITH a sericitic envelope. (Sericitic envelope assemblage is quartz, sericite, pyrite plus or minus chalcopyrite with all saussuritized feldspar being made over to a sericite-clay mixture),

ii - Quartz-chlorite-pyrite-chalcopyrite-magnetite plus or minus carbonate WITH a chloritic envelope. (Chlorite envelope assemblage is quartz, chlorite, pyrite, plus or minus chalcopyrite with a pronounced absence of epidote in the saussuritized feldspar).

V2 - i - Quartz-chlorite-pyrite plus or minus magnetite,

ii - Quartz-chlorite-pyrite-chalcopyrite-epidote plus or minus magnetite,

iii - Quartz-chlorite-pyrite-chalcopyrite-epidote plus or minus magnetite,

iv - Quartz-chlorite-bornite plus or minus pyrite (restricted to porphyry area between Pollyanna and Granite Lake zone) all with plus or minus carbonate,

V3 - i - Quartz pyrite chalcopyrite plus or minus magnetite plus or minus carbonate,

ii - Quartz pyrite-chalcopyrite-molybdenite plus or minus magnetite plus or minus carbonate,

V4 - i - Quartz-fine grained chlorite particles - pyrite plus or minus chalcopyrite.

Veins with sericitic and chloritic envelopes are definitely crosscut by V2 veins which do not have envelopes. To date, a vein with a sericitic envelope has not been found to crosscut a vein with a chloritic envelope. Consequently, those veins with either type of envelope are considered to be in Stage 1.

Veining can be parallel to the foliation in the host quartz diorite or it can crosscut the foliation.

Hydrothermal alteration, in the form of sericitic and chloritic envelopes, is both parallel and crosscutting to the foliation in the best quartz diorite.

A second hydrothermal alteration feature was observed in a petrographic study (Simpson, 1970) of the saussuritized plagioclase across the Pollyanna and Gibraltar East zones. It was noted that the amount of sericite relative to epidote in the saussuritized plagioclase could be correlated reasonably well to the copper grade.

Mineral Zoning

An obvious gross zoning of mineralization exists as far as economic copper mineralization is concerned.

There is little mineralization of any kind in the core zone. Immediately outside of, and for some distance, copper minerals (chiefly chalcopyrite) in the primary zone are present in economic amounts. It is within this portion of the mineralized zone that the orebodies occur. Beyond that, mineralization is chiefly pyrite with minor to trace amounts of copper mineralization.

Supergene Zone

The supergene zone consists of the following two divisions:

i) Leached Zone

Over almost all of the mineralized zones there exists a zone that is wholly, or in part, leached of primary sulfide mineralization. It is characterized by abundant limonite and the absence of economic copper mineralization. It is irregular in development and thickness. In the Gibraltar East pit it averages somewhat less than 100 feet (30 meters).

ii) Enrichment Zone

The vast bulk of supergene copper mineralization occurs from the bottom of the leached zone to a depth ranging from 200 feet to 400 feet (60 meters to 120 meters). In local isolated instances, supergene copper minerals occur to depths of 600 feet to 700 feet (180 meters to 210 meters).

This supergene mineralization has created a blanket of secondary enrichment over the copper bearing portion of the mineralized zone. The degree of enrichment has not been closely ascertained. The major effect is that it has added to the tonnage available for milling.

Chalcocite forms at least 85% of the secondary copper mineralization and is mainly present as coatings on the primary sulfide pyrites and chalcopyrites. Cuprite makes up 10% of the secondary mineralization. This is followed by a small amount of malachite, minor amounts of azurite and native copper plus trace amounts of covellite and chrysocolla.

The above two zones are illustrated in Figure 4 (Geology Cross Section--GIBEAST NO. 99).

Interpretative Summary of the Geology of the Gibraltar Copper Deposit

The following is one hypothesis regarding the development of the deposits. The various geological events are:

1. The Granite Mountain pluton intruded the Cache Creek group rocks during Jurassic-Cretaceous time,

2. Deformation of the general area produced simultaneous development of regional foliations and a regional greenschist facies type of metamorphic assemblages, within the quartz diorite of the Granite Mountain pluton,

3. During continued deformation, quartz-feldspar porphyries intruded the pluton which formed a structurally more competent core,

4. After further deformation, a fracture pattern developed around the structurally competent core. This fracture system, which is imposed on and partly controlled by the regional foliation, contains a wide but regionally restricted sulphide zone. Within the sulphide zone, a chalcopyrite-secondary chalcocite molybdenite zone occurs between the low sulphide core and a pyritic halo,

5. At some later time, movements on the Fraser River Fault System uplifted the Granite Mountain Pluton. Relatively down dropped areas were filled by tertiary volcanism. Weathering under arid conditions caused a leached zone and an underlying zone of secondary enrichment,

6. Recent glacial activity then deposited till and gravel over the entire area of the Gibraltar-Pollyanna copper molybdenum deposits.

Ore Reserves

Current ore reserves (January 8, 1992), at a copper price of $1.00 U.S. per pound, are 165,800,000 tons (150,411,200 tonnes) at 0.312% Cu. and 0.0088% Mo. These reserves do not include the "Gibraltar North" ore zone which was indicated in the 1990 drilling program.

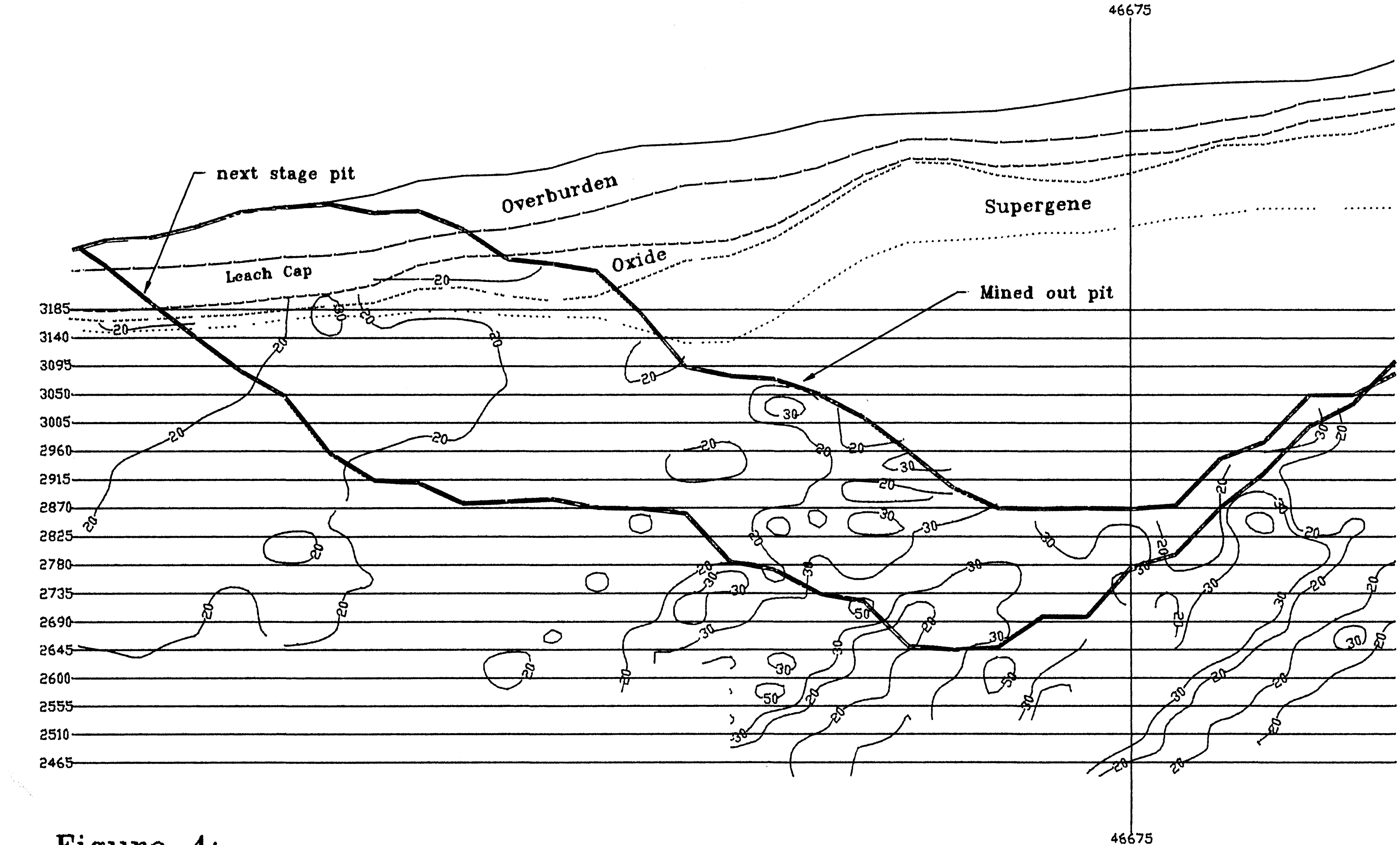

Figure 4:
Portion of Gib–East Section No. 99

50075-N 46675-E AZ-045
%Cu/100 W=100 INV=AUG88
SCALE: 1 in. = 235 ft. (aprox.)

PROPERTY OPERATION

The surface layout of the Gibraltar Mines operation is shown in Figure 5 (General Mine area). Under the direction of the Mine Manager, the operating crew is classified into seven departments as outlined in Table No. 1 .

TABLE 1

DEPARTMENT	STAFF	HOURLY	TOTAL
Administration	1		1
Accounting	10	5	15
Employee Relations	4		4
Engineering & Geology	12		12
Mine	6	67	73
Mill	17	46	63
Plant:			
Crusher Maintenance	1	9	10
Electrical Maintenance	3	12	15
Machine/Weld Shop Maint.	2	14	16
Mill Maintenance	2	8	10
Pit Shop Maintenance	3	18	21
Plant Maintenance	1	8	9
Surface Maintenance	1	9	10
Shovel and Drill Maintenance	1	7	8
Technical Services	3	18	21
Total Plant	17	103	120
TOTAL	67	221	288

MINING OPERATIONS

The total area of the Gibraltar Mines Ltd. property is approximately 4300 acres (1740 hectares). This includes the orebodies, dumps, plantsite, and tailings pond.

As mentioned in the Geology section, the copper deposit is comprised of four separate and distinct orebodies. Each orebody is mined in sequential stages. This sequence is based on computerized pit designs and schedules, thus allowing maximization of the discounted cash flow.

The following design parameters are used:

Overburden bank slope	-	45 deg.
Working face slope	-	67 deg.
Bench interval height	-	45 ft. (13.7 m.)
Safety berms in rock	-	30 ft. - 60 ft. (9.1 m. - 18.3 m.)
Haulage road width	-	100 ft. (30.5 m.)
Haulage road gradient	-	up to a maximum of 10%

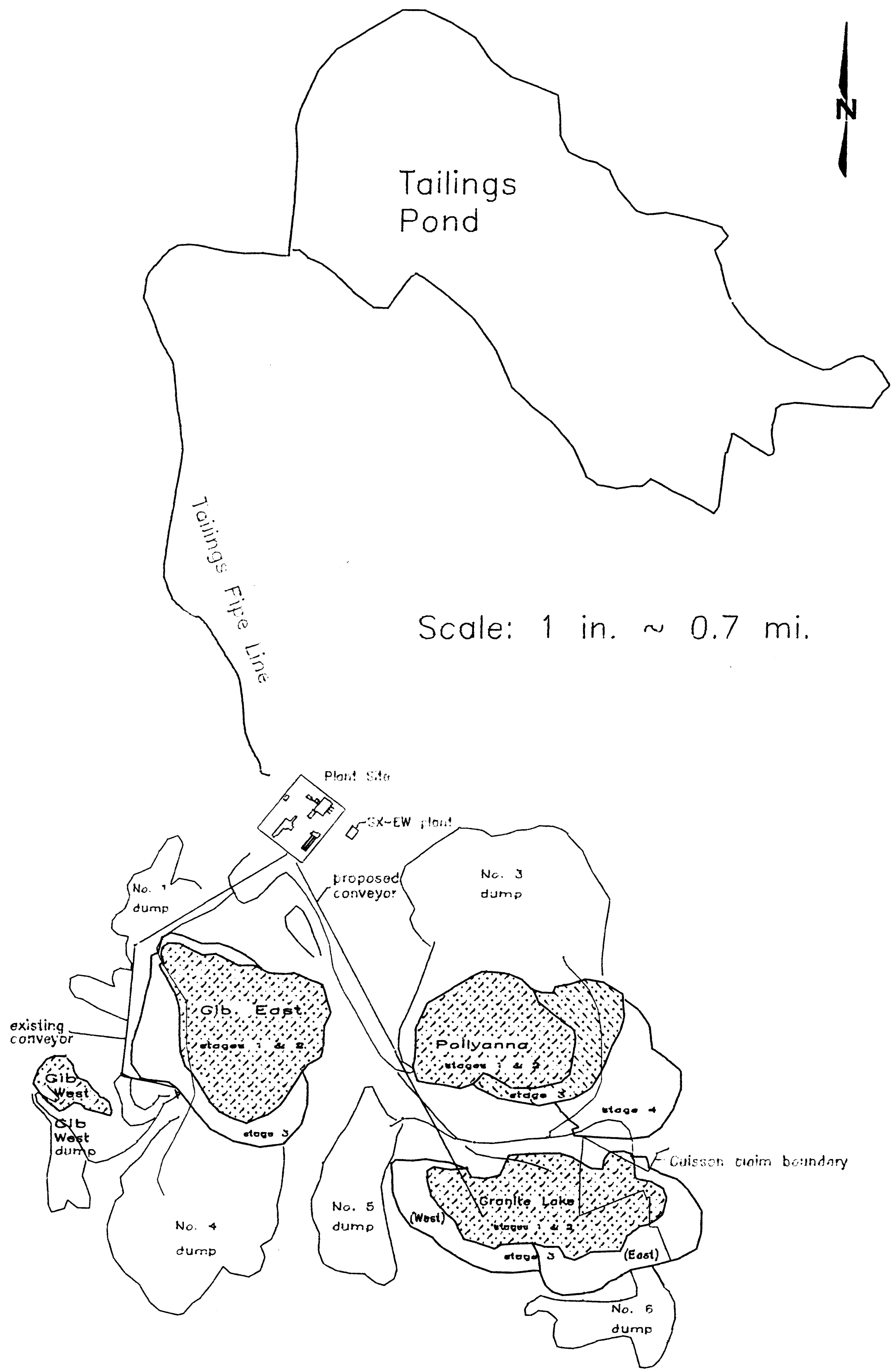

FIGURE 5: GENERAL MINE AREA

Pit Production

As of July, 1991, all Stage 1 and Stage 2 pits have been completed. The Pollyanna Stage III pit is supplying 100% of the ore. Stripping of the Stage III Gibraltar East pit began in the fall of 1990. During the first quarter of 1993, the Pollyanna Stage III pit will be completed, at which time ore will be supplied by Gib East.

Drilling patterns vary according to rock type, geological structure and drill hole diameter, but can be summarized as follows:

Hole Diameter	Burden	Spacing
9.875 in. (24 cm.)	21 ft. (6.40 m.)	24 ft. (7.32 m.)
12.250 in. (31 cm.)	26 ft. (7.92 m.)	30 ft. (9.14 m.)

The pit operates seven days per week on a two (12 hour) shift basis.

1. Pre-Production Phase (April 1971 - March 1972)

The following figures summarize pit production statistics from start up (April 1971 - March 1972)

Overburden	- 3,207,000 tons	(2,925,000 tonnes)	(by contractor)
Overburden	- 1,400,000 tons	(1,277,000 tonnes)	
Waste Rock	- 4,134,000 tons	(3,770,000 tonnes)	
Stockpiled High Grade	- 327,000 tons	(298,000 tonnes)	@ 0.41% Cu.
Stockpiled Low Grade	- 1,110,000 tons	(1,012,000 tonnes)	
Total Mined	- 10,178,000 tons	(9,282,000 tonnes)	

2. Production Phase (March 1972 - December 31, 1991)

Overburden	- 22,834,000 tons	(20,825,000 tonnes)	(by contractor)
Overburden	- 33,563,400 tons	(30,448,200 tonnes)	
Waste Rock/Oxide	-192,677,300 tons	(174,793,900 tonnes)	
Stockpiled High Grade	- 5,878,300 tons	(5,332,800 tonnes)	(Dec. 31/90)
Stockpiled Low Grade	- 49,960,400 tons	(45,324,000 tonnes)	(Dec. 31/90)
Mill Feed*	-252,508,600 tons	(229,071,900 tonnes)	
Total Mined	-557,422,000 tons	(505,684,800 tonnes)	

* Includes the mining of High Grade and Low Grade stockpiles

Pit Equipment

Current pit equipment and respective performances appear in Table No. 2 (Pit Production Equipment).

TABLE NO. 2
Pit Production Equipment

Function	Type of Units	No. of Units	Unit Performance
Rotary Hole Drilling	Elect. Marion M-4	1	529 ft./shift (161 m.) 12.25 in. (31.1 cm.) dia. holes
	Elect. Bucyrus-Erie 45R	1	745 ft./shift (227 m.) 9.875 in. (24.8 cm.) dia. holes
	Elect. Gardner-Denver GD100	1	700 ft./shift (213 m.) 12.25 in. (31.1 cm.) dia. holes
Loading	Elect.2100 P&H	2	15,800 tons/shift (14,300 tonnes)
	Elect.2300 P&H	2	26,400 tons/shift (23,950 tonnes)
	Elect.2800 XP/A P&H	1	40,000 tons/shift (36,300 tonnes)
Diesel-Electr. Haulage Trucks	Unit Rig MK36 170 T.	4	4,050 tons/shift (3,674 tonnes)
	Unit Rig MT4000 240 T.	6	8,300 tons/shift (7,530 tonnes)
Dump Maint & Shovel Clean-up	D8N Cat	1	
	D9N Cat	1	
	D10N Cat	1	
	824 Cat	3	
Road Maintenance	16G Cat grader	1	
	14G Cat grader	2	
	Water truck	2	
	Sand truck	2	

Maintenance Facilities:

The maintenance shop is housed in a totally enclosed structure covering 57,800 square feet (5,370 square meters). The shop includes facilities for the complete range of repair and maintenance of pit production equipment, surface equipment and the mine service vehicles. The building includes a machine shop, three welding bays, ten maintenance bays, seven gas service bays, two tire bays, one steam bay, an electrical shop and a tool crib. A re-build shop of 11,200 square feet (1040 square meters) was added in 1984.

Warehouse Facilities:

The mine warehouse is immediately adjacent to the maintenance shop and covers an area of approximately 16,360 square feet (1,520 square meters).

DEWATERING

In the summer of 1973, mining in the Gibraltar East pit progressed below "daylight" on the down-hill side. Groundwater quickly became a costly nuisance in the day-to-day mining operations. Mining, maintenance and blasting costs reflected these problems. Concern was also voiced about adverse effects on pit wall stability.

Regional and Geological Studies

Preliminary investigations included a study of the climate, topography, overburden and bedrock, and variations thereof, that may influence the distribution and flow of ground water.

The pits are located in the Granite Creek drainage system, which has a total catchment area of 1,200 acres. The recharge infiltration area that affected the ground water of the pits was assumed to lie above the 3,400-foot elevation contour. This is the elevation of the bottom of the Granite Lake and Pollyanna Stage 2 pits. The recharge area was found to be 50 square miles. The monthly precipitation charts from Williams Lake Airport and Gibraltar Mines indicated an average annual precipitation of 24 inches. It was assumed that 2% of this water would infiltrate into the recharge area. This amounts to an average annual recharge of 750 (U.S.) g.p.m.

The pits lie within a highly fractured and jointed quartz diorite which is overlain by up to 200 feet of glacial till. The fracture and joint systems, fault zones and adjacent broken material determine, for all practical purposes, the bulk permeability of the bedrock. These features are planar and directional, and vary considerably in frequency of occurrence and in intensity of individual features.

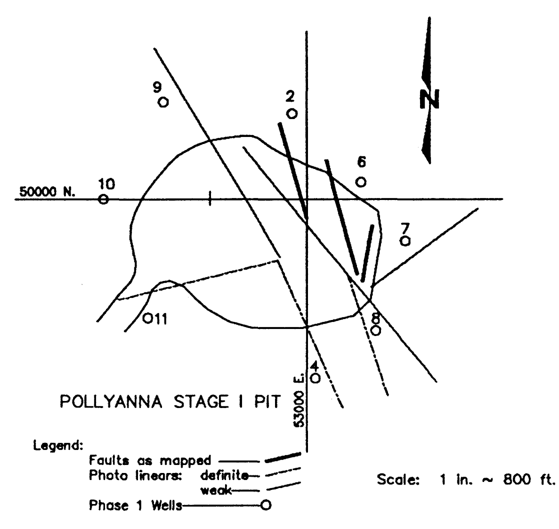

Figure 6: The Pollyanna Stage 1 pit

The most pertinent data used to evaluate groundwater conditions were: the photogeolo gy of the mine area (Figure 6), a study of percentage of faulted and broken rock in diamond drill holes, and geological surface mapping. Four qualitative conclusions resulted from this study:

(1) mass permeability was controlled by two extensive southwest-dipping joint sets,

(2) faults acted as either cut-offs, if gouge filled, or flow conduits, if barren of gouge,

(3) permeability was of a high enough order that effective dewatering was feasible,

(4) the use of deep peripheral wells appeared to be the most effective and economic dewatering method available.

Falling-Head Tests

To substantiate these conclusions, a series of falling-head tests were conducted in the Gibraltar East pit. Testing was done on seven 3-inch-diameter percussion drill holes using packers and stand pipes. The testing procedure was as follows:

The packer stand pipe assembly was installed in the hole at the desired depth (Figure 7). The stand pipe was filled with water, the water supply was cutoff, and the water was then followed down the stand pipe with a water depth
probe. The times corresponding to the various water level readings were recorded with a stop watch. All measurements were taken using the top of the stand pipe as "0" and the time was recorded cummulatively in seconds.

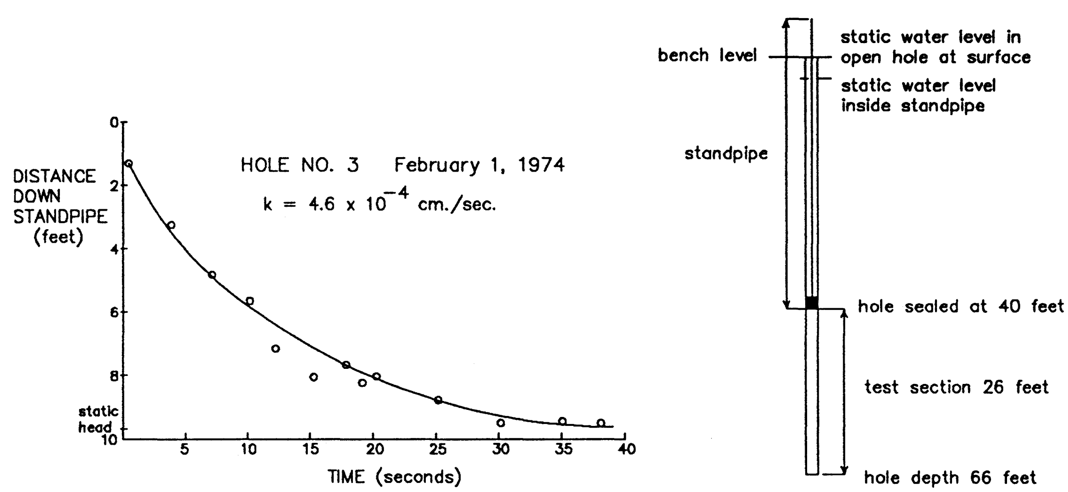

Figure 7: Falling Head Permeability Test

Upon completion of the testing, the permeability was calculated according to the formula:

$$k = \frac{r_c^2 \ \ln\frac{r_e}{r_o} \ \ \ln\frac{h_1}{h_2}}{2L\ (T_2 - T_1)}$$

where:

r_c = internal radius of the casing
r_e = radius at which excess pressure is 0 (estimated at 330 feet (100 meters))
r_o = radius of the test hole
h_1 ,h_2 = excess head above static water level--at time T_1 and T_2.

In holes No.2, 4, and 6, the water dropped too quickly for readings to be taken. The ground represented by these holes was considered to have a high mass permeability. In holes 1 and 3, the results showed mass permeabilities in the order of 10 -4 cm/second, or moderately high mass permeability (Figure 7). Permeabilities in holes No.5 and 7 were of the order of 10-6 cm/second, which is considered a low mass permeability.

These results confirmed that the rock permeability was high enough for effective dewatering to be possible. A consulting firm, Piteau, Gadsby, Macleod Ltd., was then engaged to review all the available data and subsequently recommended a five-well test program for the Pollyanna - Granite Lake pit area.

Deep-Well Pumping Tests

The test program objectives were to evaluate the following parameters and characteristics of the ground-water geology and hydrology:

1. Transmissiblity-- a field measurement of the permeability of the whole saturated thickness of rock to be drained,

2. Storage factor-- the effective porosity of the fractured rock,

3. Specific capacity-- the amount of water an individual well will yield per foot of drawdown,

4. Location of major "conduit" and "dams"-- the optimum location for drainage in relation to the fault zones,

5. Training-- the training of mine personnel in the logging of rotary holes with regard to fractures, water content, and the running of pump tests.

The location of the five pump test wells are shown in Figure 8. The drilling of all wells and installation of piezometers were completed in the summer of 1974. Each of the five production wells were test pumped with a 25-hp 10-stage Webtrol submersible pump. A complete report was submitted by Piteau, Gadsby, Macleod Ltd. in December 1974. The test results led to the following conclusions:

1. The rock in the geological environment containing the Gibraltar pits could be dewatered using peripheral wells,

2. Groundwater flows from northeast to southwest. These flows are controlled by southwest-trending faults and joints,

3. Those southwesterly dipping mineralized joints that have reopened in post-ore time have impressed a secondary level of transmissiblity in a southeasterly direction,

4. The north-to-northwest trending faults provide "cut-offs" or "dams" transverse to regional ground-water flow.

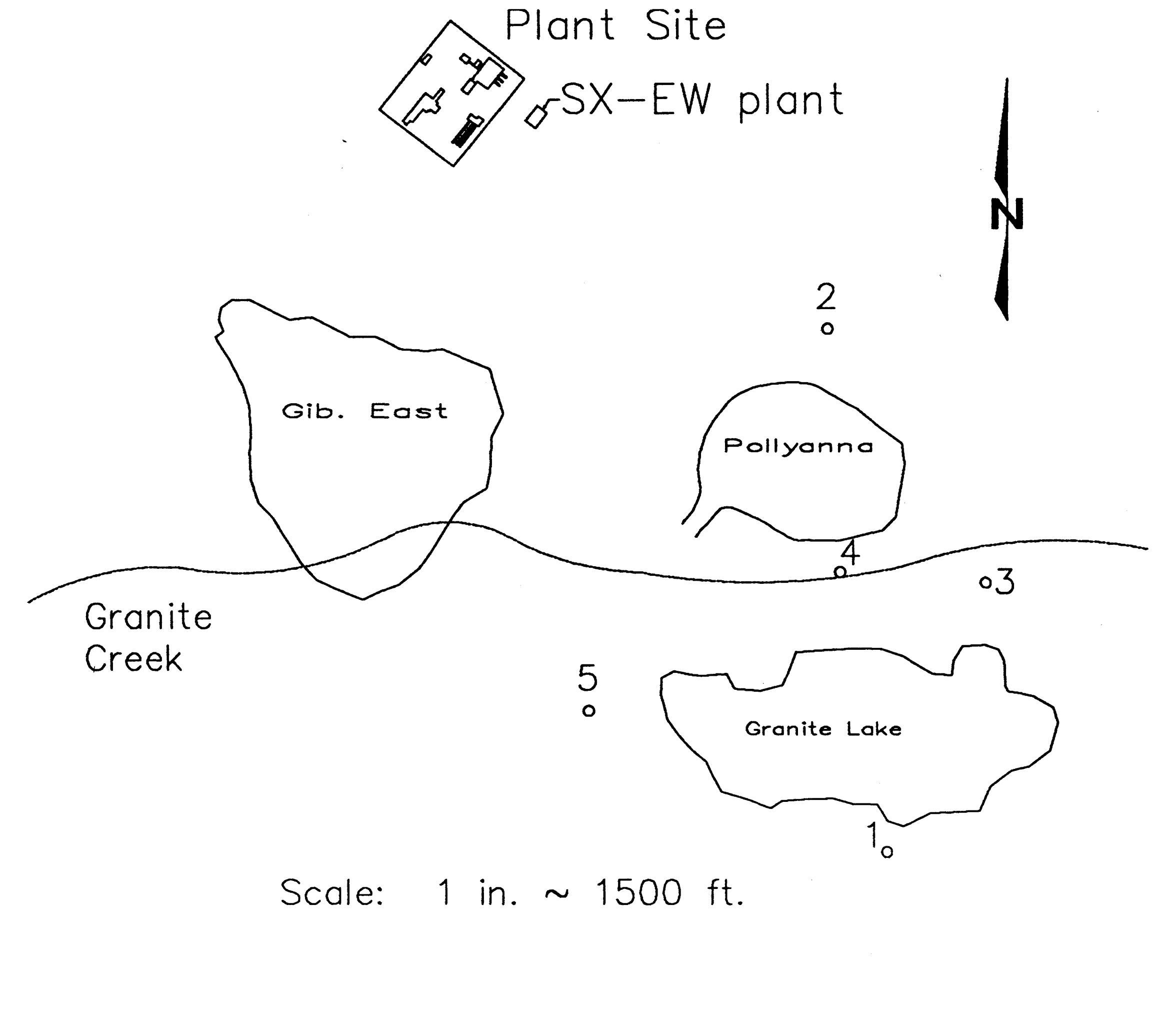

Figure 8: Pump Test Sites

Piteau, Gadspy, Macleod Ltd. did all the determinations of the hydraulic parameters. The results are tabulated below:

Site No.	Transmissibility	Storage Factor
1	5500 - 400	.014 - .001
2	230 - 100	
3	1100 - 325	.0005 - .0001
4	750 - 225	.0006 - .0007
5	420 - 400	.0002

The pump testing also produced some interesting results. For example, pumping at sites 4 and 5 retarded the recovery of site 1, some 3000 feet away. Conversely, pumping at site 2 had no effect on the site 2 observation hole, only 200 feet away. These results indicate both the good interconnections of zones and the strong effect of fault cut-offs.

Design Procedure

The design of the dewatering system of the Pollyanna pit was projected to be a three-phase project, each subsequent phase conditional on the success of the previous one. Phase 1 consisted of drilling six new wells and equipping eight in total, including two old pump test wells. If phase 1 failed to attain the desired results, six additional wells were to be installed as phase 2. The final phase, if required, added two in-pit wells.

The wells were strategically located so as to intercept major groundwater conduits and dewater isolated fault blocks. It should be noted that the wells are more numerous on the northeast side of the pit (Figure 8); such a pattern was dictated by the southwest flow of the groundwater beneath the area. The wells were drilled to a minimum depth of 250 feet below the final pit floor.

The wells were constructed in several stages (Figure 9). Initially, a 9 7/8-inch-diameter hole was drilled through the overburden and 10 feet into the bedrock. An 8-inch-inside-diameter casing was then set into the hole. Following this, a 7 7/8-inch-diameter hole was drilled to the desired depth. Two hours of surging removed cuttings and cleansed the well wall. The well was then cased with a 6 1/4-in.-inside-diameter, light-wall, perforated steel casing. The perforations were cut vertically with an acetylene torch. They were approximately 1 foot in length and 1/8 to 1/4 inch wide. The number of perforations were governed by the amount of water pumped during the drilling of the well.

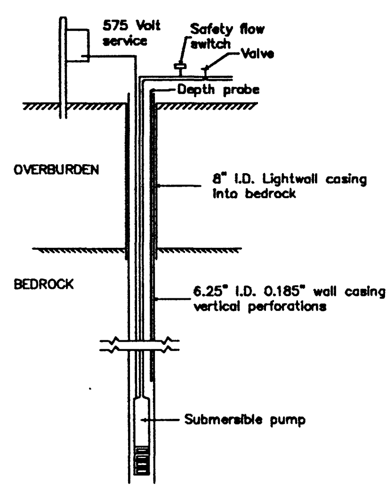

Figure 9: Typical Well Design

Operation and Maintenance

The major operating problem has been the control of the discharge rate of the well. Initially, high-low cut-off probes were installed in the well to operate the pump within a certain depth range of water. The number of starts and stops of the pump was minimized by throttling the flow rate with a valve at the discharge. This method was found to be unreliable due to iron content in the water and the deposition of solids (carbonates) on the ends of the probes. The current method involves a time-delay relay. A single probe in contact with the water turns the pump on. If the water level drops below the probe, the pump is automatically turned off after a pre-set time. This method of water level control is very effective and requires minimum maintenance.

A further problem developed with regard to the original selection of pump sizes. The actual production of a well is unpredictable and in fact can increase or decrease long after its installation. As an example of this, Well No. 9, when first started, produced 130 (U.S.) g.p.m. while Well No. 10 produced only 75 (U.S.) g.p.m. After one year of pumping, the wells reversed in volume production; No. 9 decreased to 40 (U.S.) g.p.m. and No. 10 increased to 85 (U.S.) g.p.m.

Maintenance on the wells has been minimal. The major problem has been damage to the thrust bearing caused by the hammer effect when the pump stops. To correct this a second check valve was installed half way up the well. One well had a cave-in above the pump resulting in damage to the pump impellers. A defective pump can be pulled and replaced in one 8-hour shift.

Results

1. Phase one of the Pollyanna Stage I pit was successfully dewatered to a depth of 585 feet (180 metres) below the original surface. At this point some short in-pit wells were required to supplement the deep wells. This trend has continued with the larger stage II pits, where inpit wells are required at the lower benches.

2. It is difficult to compare blasting costs of the Granite Lake and Pollyanna pits due to inflation and changes in types of blasting agents. In 1978, not allowing for the above factors, the blasting cost had dropped by 1.6 cents per ton. In the Granite Lake pit, 68% of the total blasting agent used was slurry whereas the Pollyanna pit required 31%.

3. Tire life and costs have also significantly improved. Ignoring inflation, a 48-ply tire in 1975 had a life of 2,140 hours and a cost of $1.91/hour; in 1978, tire life was 2,951 hours at a cost of $1.43/hour. This represents a reduction in tire costs of approximately 1.4 cents per ton.

4. Fewer problems associated with freezing and plug-ups in the crushing and screening circuits have been experienced due to the drier ore coming from the pit.

5. Since 1978 the dewatering program at Gibraltar Mines Ltd. has continued and is still proving to be successful.

GIBRALTAR MINES INPIT CRUSHING SYSTEM

An initial study was completed in 1977 on the feasibility of crushing and conveying mill feed ore in the Gibraltar East Stage II pit. The results of this study indicated:

1. A reduction of ore haulage cost of 16 cents per ton (1977 dollars),

2. The probability of saving money with a conveyor-crusher system was high while the probability of losing money was low,

3. The same conveyor could be used for mining of the Granite Lake and Pollyanna Pits,

4. The conveyor system would delay the purchase of additional haulage units by 2 years,

5. An expansion of the maintenance facilities could be delayed,

6. A used conveyor system at Hudson Hope became available for purchase. The conveyor system itself was in excellent condition (ie. the belt, rollers, frame, etc.) while the drive systems and electrics required repairs and modifications. This system was eventually purchased and resulted in substantial savings in the capital cost of the project.

Wright Engineers Ltd. were assigned the responsiblity of designing a new primary crusher and modifying the purchased conveyor to Gibraltar's specifications. Construction of the crusher-conveyor system was awarded to Commonwealth Construction Ltd. Site preparation commenced in May of 1979 and the system was commissioned in April of 1980. The system was in use from April of 1980 to August of 1986 when ore was depleted in the Gibraltar East Stage II and Gibraltar West pits.

In 1992, with the pending expansion of the Gibraltar East Stage III pit interfering with the right of way on both of the long conveyors, Kilborn Engineering was commissioned to re-design the conveyor system. Gibraltar chose to act as the principle contractor, sub-contracting most of the civil, structural and mechanical construction to TNL Construction and Western Belting. All of the electrical requirements were sub-contracted to J.B. Electric. Commissioning is expected to occur in August of 1992.

General Arrangement

The inpit crusher is an Allis Chalmer 54"x74" Superior Gyratory crusher (see Figure 10) which is driven by a 600 h.p. variable speed DC motor. The crusher building itself is a conventional concrete structure with no surge bin capacity. The throughput onto the first conveyor (no. 30 conveyor) will be controlled automatically by adjusting the operating speed of the crusher. The hydroset position will also be automatically adjusted to maximize crushing effort.

The mill feed then drops directly onto No. 30 conveyor which is 84 inches (2.1 meters) in width, 257 feet (67.6 meters) in length and has a designed capacity of 3300 tons per hr. (3000 tonnes per hr). It is driven by a 150 h.p. motor at a speed of 5.2 feet per sec. (1.6 meters per sec.).

Making an 80 degree directional change to get around the outer boundaries of the pit, conveyor 30 discharges onto conveyor 31. Conveyor 31 is a 66 inch wide belt conveyor travelling at 600 feet per minute with a nominal capacity of 3300 tons per hour. It has a vertical lift of 18 feet, overland length of 891 feet and is powered by a 500 h.p. head pulley drive. This conveyor passes underneath the haulage roads in the pit through a concrete tunnel with a length of 295 feet.

No. 31 conveyor then discharges the ore onto No. 32 belt with an 83 degree directional change. Of similar construction and capacity to conveyor 31, conveyor 32 is 3000 feet (940 meters) in length. It is powered by two 850 h.p. motors coupled to either end of the head pulley and one 500 h.p. motor on a secondary drive. The maximum grade is 20% which results in a lift of 305 feet (100 meters).

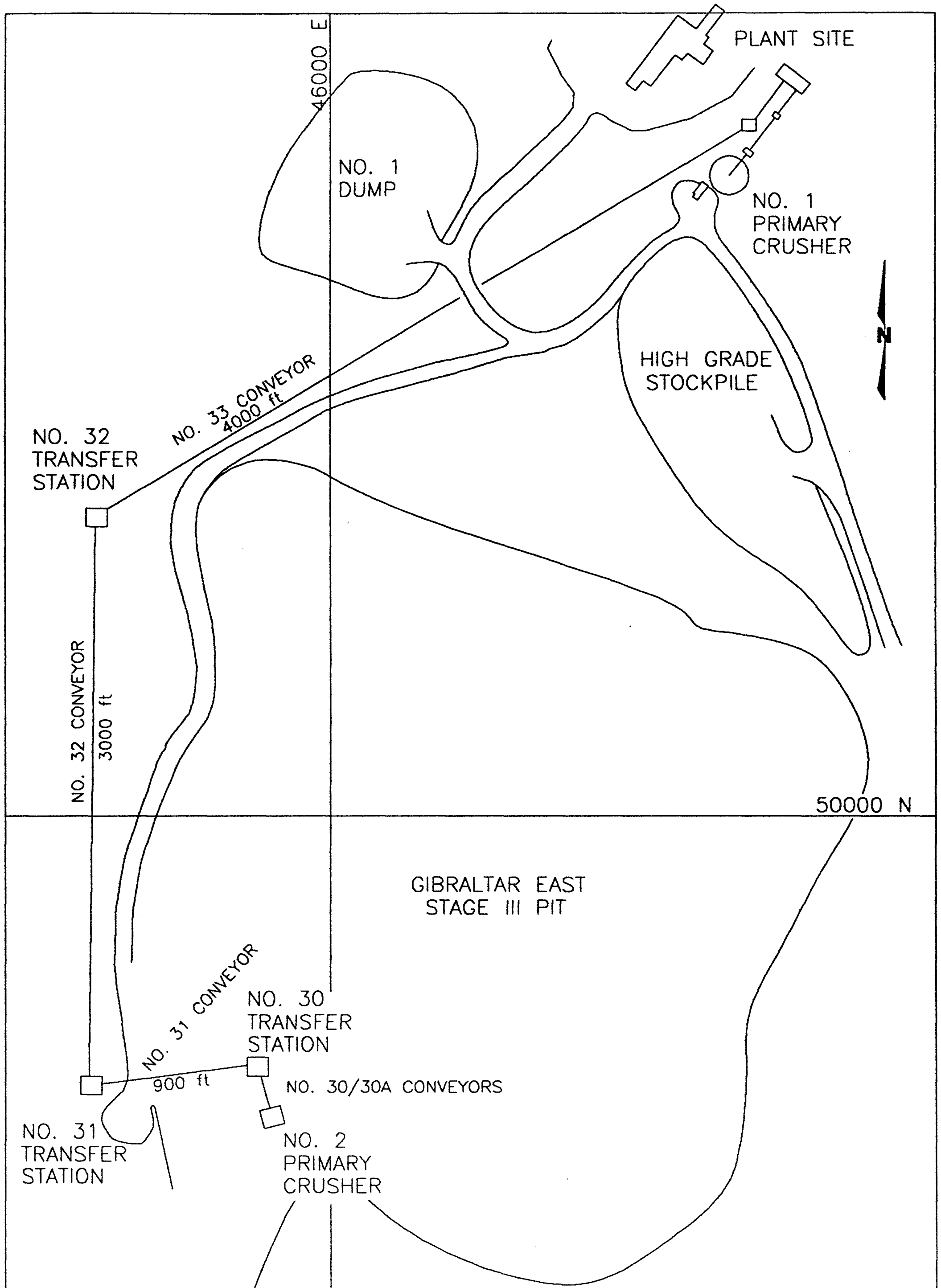

FIGURE 10: GIB EAST CONVEYOR
Scale: 1 in. = 800 ft. (approx)

No. 32 belt extends to a transfer point where there is a directional change of 58 degrees. The transfer point then discharges the high grade ore onto No. 33 belt which is powered by a mid-station drive with one 850 h.p. motor. This is assisted by a high tension snub drive pulley that is powered by a second 850 h.p. motor. No. 33 conveyor is 4020 feet (1225 meters) from center to center and has a vertical lift of 160 feet (48 meters) (maximum grade is 4.5%).

No. 33 belt then deposits the mill feed onto No. 2 conveyor at mid-station which transfers the ore to the crushing and screening plant.

Most of the equipment in this installation, including the belts, drives and motors is of mid-60's vintage. In spite of the dated equipment, the installation has been up-graded to state-of-the-art technology with PLC controllers and the longest non-utility use of fibre-optics communication cable in the Cariboo.

The PLC's will be programmed to maintain crusher throughput to a set point by varying crusher speed, and to maximize crushing effort by varying the hydroset position (which changes the crusher opening). The PLC's will also monitor several other key operating parameters on the system and will alarm and possibly shut-down should any unusual situations develop.

Dust control and clean-up within the crusher are accomplished through conventional means (wet scrubbers and dribble belts). Dust control at the conveyor transfer stations is not a problem because none of the stations are housed.

THE GIBRALTAR MINE PLANNING SYSTEM

The Gibraltar Mine Planning System (GMPS) (Figure 11) is made up of several computer programs that can be separated into four main processes or routines. The system uses diamond drill data to produce an ore block model, smoothed pit designs, and a schedule for mine production. A recent addition to the system has been a computer aided drafting package that allows "smoothed" pit designs to be constructed with far greater efficiency and accuracy.

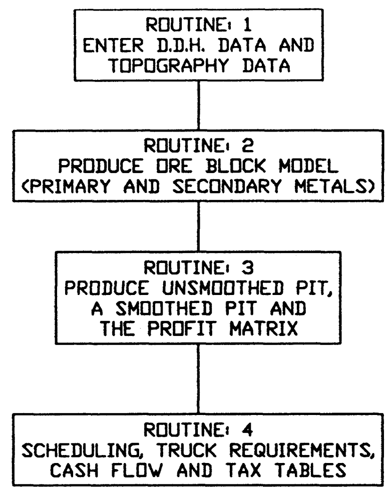

FIGURE 11: THE GIBRALTAR MINE PLANNING SYSTEM

Upon completion of diamond drilling, the core is logged and data such as metal grades, overburden, leachcap, and bedrock contacts are entered into the first routine (Figure 12). Also, the Digital Elevation Model (DEM) of the topographic contours are entered. The first routine then interpolates topographic data as well as producing a bedrock and leachcap surface, which can then be checked to ensure continuity before proceeding to the second routine.

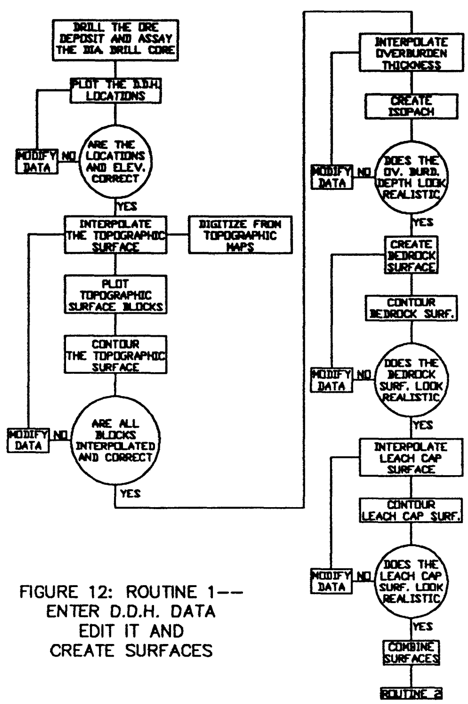

FIGURE 12: ROUTINE 1—
ENTER D.D.H. DATA
EDIT IT AND
CREATE SURFACES

The second routine (Figure 13) incorporates various geostatistical methods (such as Kriging, Inverse Distance Squared, etc.) to interpolate a preliminary ore block model, from the diamond drill hole (DDH) and Stage I blasthole assay (BAS) data. It then interpolates ore blocks for each bench (Figure 14) and estimates that block's work index and secondary metal grades (such as molybdenum).

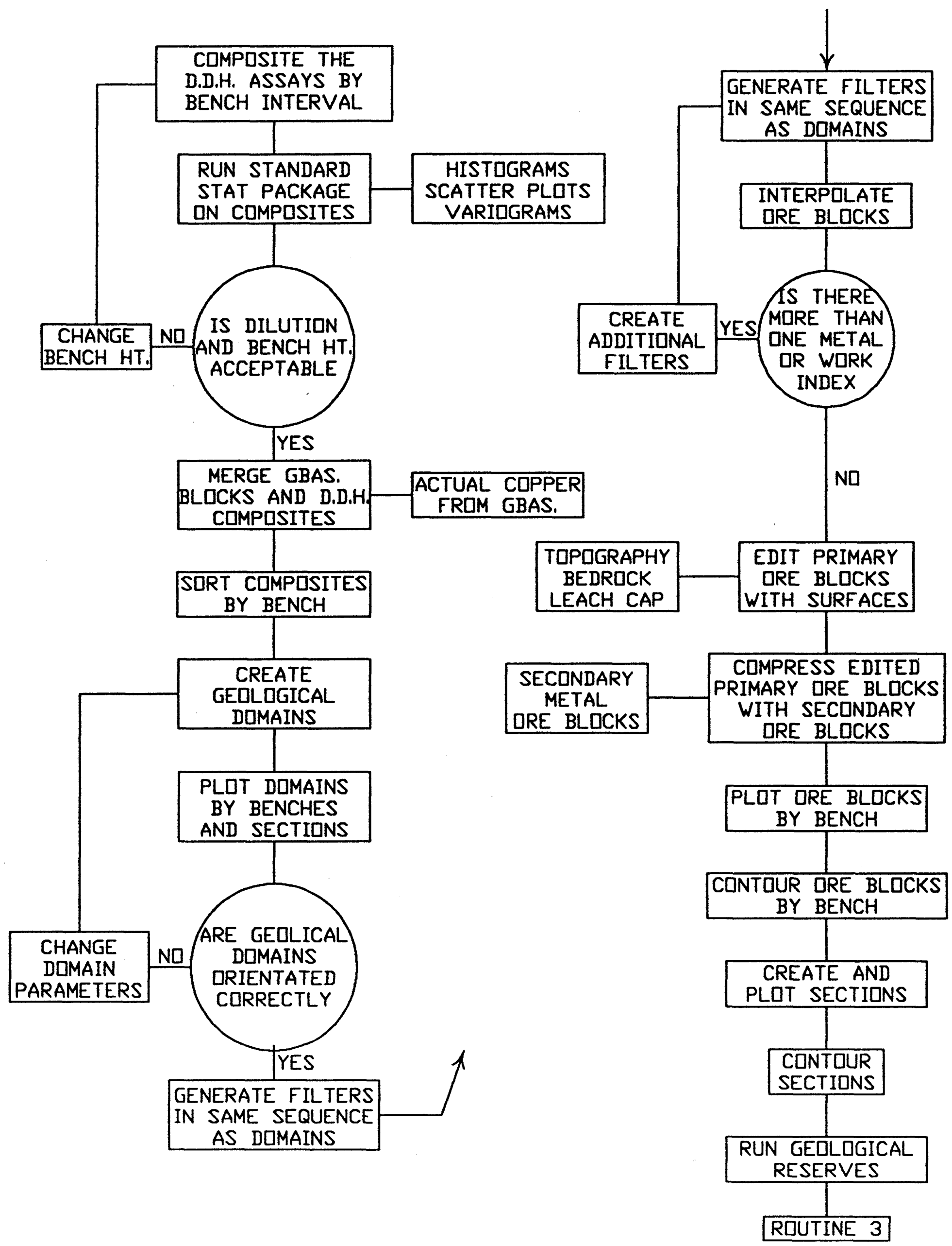

FIGURE 13: ROUTINE 2--

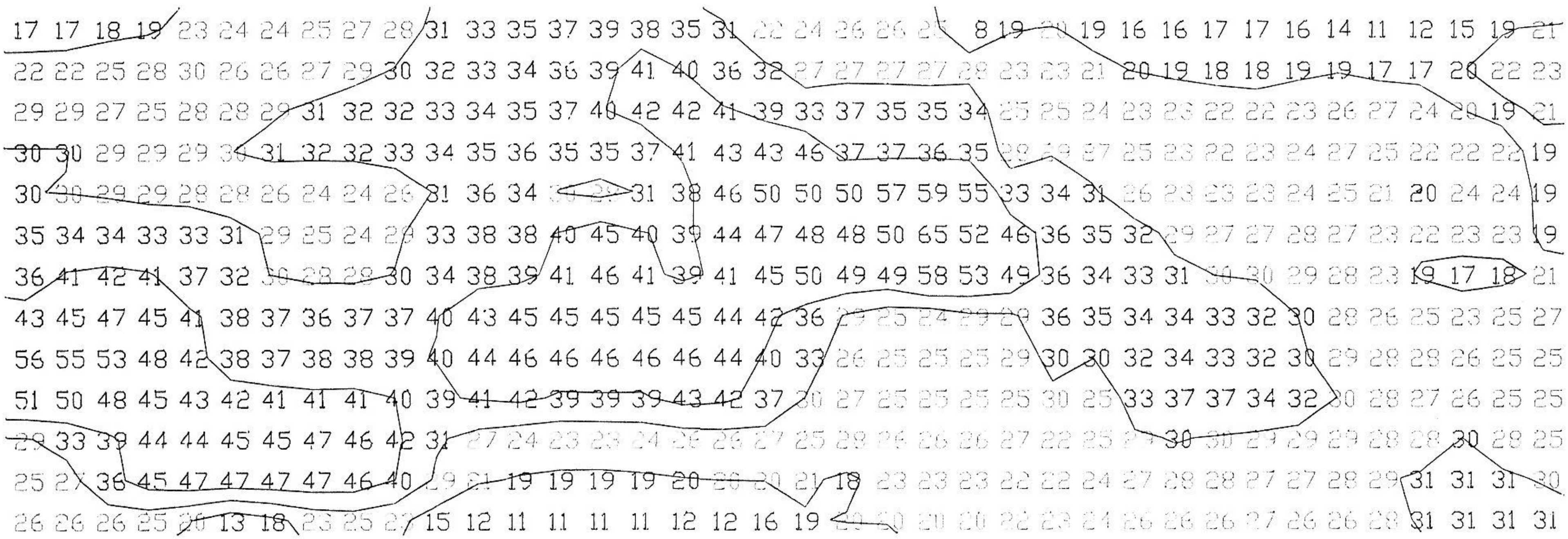

FIGURE 14: TYPICAL ORE BLOCK MODEL

The third routine (Figure 15) optimizes an "unsmoothed" pit design from the ore block model. The input parameters include estimates of copper price (ie. net smelter return), recovery, secondary metal prices, and overhead costs. They also include estimates of operating costs such as the mining of ore and waste, overburden, the incremental increase in mining cost for each bench below the entrance, and the cost of milling. The "unsmoothed" design does not make allowances for ramps, minimum mining widths, or other physical restrictions associated with the design.

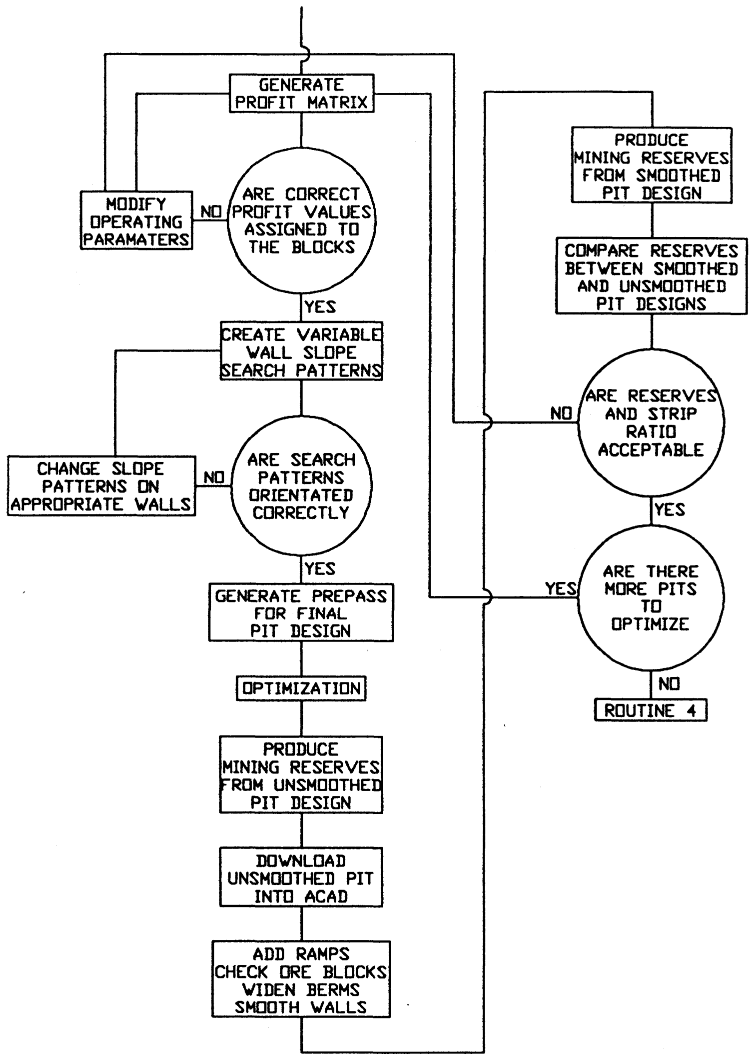

FIGURE 15: ROUTINE 3-- UNSMOOTHED PIT DESIGN

Graphic workstations and AutoCAD software is used to provide a system to assist the engineer in the process of developing a practical ("smoothed") pit design, based on the optimization. A contoured version of the unsmoothed design, together with topographic and current asbuilt surfaces, are loaded into AutoCAD, (in color coded layers based on benches)(Fig. 16). Grade values from the interpolated mineral inventory are also loaded into the drawing, with grades outside the design in a "subdued" color. The grades contained within the unsmoothed pit are coloured red, blue, green, brown, or yellow.

"Prototype" drawings provide a standardized format for all drawings, eliminating the need to recreate grid lines, limits, and other drawing parameters that are unique to each pit area. This standardization also allows information to be easily transferred between drawings developed from the same prototype.

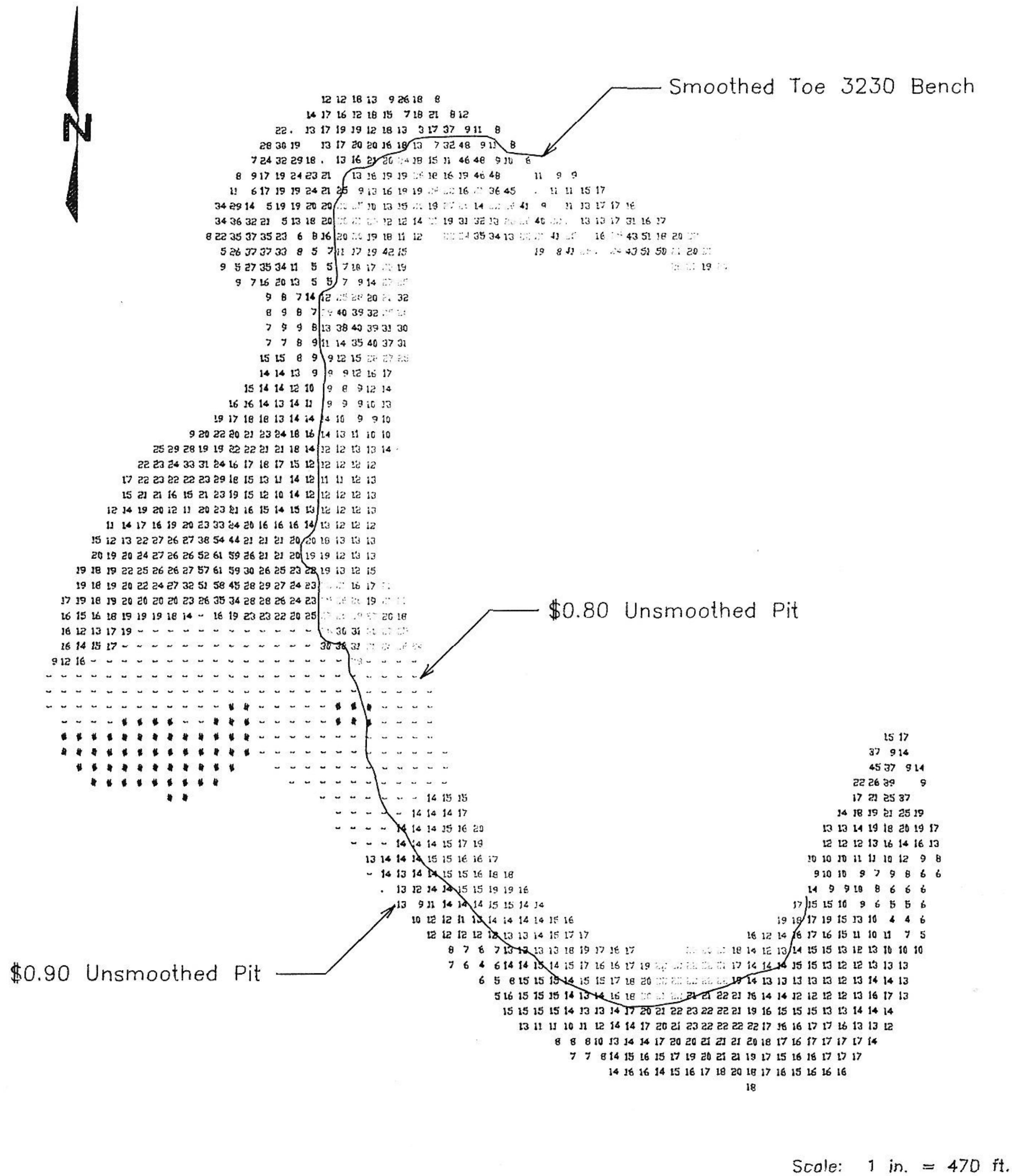

FIGURE 16: UNSMOOTHED PIT DESIGN-- GIB EAST STAGE III, 3230 BENCH

"Autolisp" routines have also been written to facilitate smoothing:

1) A ramp routine may be used to "rough in" a haulage ramp, fitted from the bottom bench of the design to the surface. This routine can also be used to construct the final design of ramps between benches. It automatically assigns the correct elevations, linetypes, layer, etc.

2) A set bench routine will display only the pertinent information (ie. layers) required for smoothing and developing the design of a particular bench, -- all other information is temporarily "frozen", or hidden from view.

3) Various other routines have also been written to be used in conjunction with the normal AutoCAD commands. An example of this would be the offset command. Once an existing line has been drawn (such as a toe) then the "offset" command will produce the intermediate and crest of the next 2 benches in a matter of seconds (Figure 17). By using the Autolisp routine "CI", the toe line is changed to an intermediate line with the appropriate color, elevation, thickness, linetype and layer by simply entering the correct bench number (eg. 3905 bench).

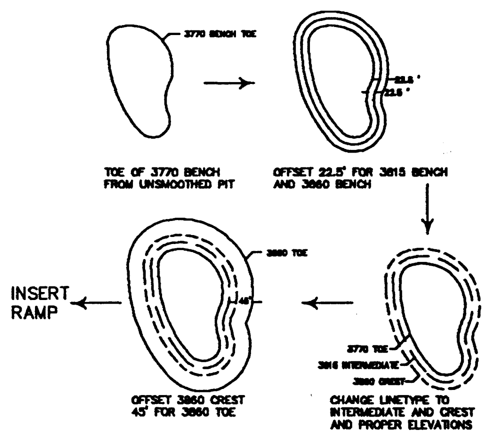

FIGURE 17: USE OF THE OFFSET COMMAND TO PRODUCE A SMOOTHED PIT DESIGN

Once the final design has been completed, the data is fed back into the third routine for the calculation of ore reserves (Table 3) and mine scheduling.

TABLE 3

R E S E R V E S --GRANITE LAKE Stage 2 North
Mineral inventory as of dec88 pit code.....(080c22mar88n)

	-------hg--------			-lg-&-highoxide-			waste	ob	total	strip	-----oxide-----			----supergene----			------primary----		
bench	tons	cu	mos2	tons	cu	mos2	tons	tons	tons	ratio	tons	cu	mos2	tons	cu	mos2	tons	cu	mos2
3905	0	.000	.000				9375	0	9375	9.99	0	.000	.000	0	.000	.000	0	.000	.000
3860	93750	.258	.011	0	.000	.000	184375	0	278125	1.97	37500	.296	.009	18750	.226	.011	37500	.236	.012
3815	408750	.272	.011	0	.000	.000	215000	0	623750	0.53	0	.000	.000	268125	.285	.008	140625	.247	.018
3770	1817500	.313	.013	0	.000	.000	394375	0	2211875	0.22	0	.000	.000	434375	.286	.010	1383125	.321	.014
3725	2545000	.331	.014	0	.000	.000	706250	0	3251250	0.28	0	.000	.000	309375	.342	.015	2235625	.330	.014
3680	2511250	.312	.015	0	.000	.000	430000	0	2941250	0.17	0	.000	.000	135000	.303	.012	2376250	.312	.016
3635	2767500	.312	.019	0	.000	.000	305625	0	3073125	0.11	0	.000	.000	18750	.316	.009	2748750	.312	.019
3590	3440625	.324	.021	0	.000	.000	328125	0	3768750	0.10	0	.000	.000	0	.000	.000	3440625	.324	.021
3545	1931250	.280	.021	0	.000	.000	300000	0	2231250	0.16	0	.000	.000	0	.000	.000	1931250	.280	.021
Tot	15515625	.312	.017	0	.000	.000	2873125	0	18388750	0.19	37500	.000	.000	1184375	.302	.011	14293750	.313	.018

Total copper cutoff hg=0.20

```
copper price...........................U.S.$ 0.800
exchange rate (cnd/us)................ 1.176
OPC................................... 0.000
copper price (nsr).....................CND$ 0.000
recovery.............................. 0.000
content............................... 1.000
moly  price (nsr)......................CND$ 0.000
recovery.............................. 0.000
content............................... 0.600
reduction in ox ore tons....................     0%
milling                              cost/ton.. 0.000
overhead (to mining)                 cost/ton.. 0.000
ore mining                           cost/ton.. 0.000
lg mining                            cost/ton.. 0.000
waste mining                         cost/ton.. 0.000
overburden stripping                 cost/ton.. 0.000
mining cost increase per bench       cost/ton.. 0.000
entrance bench................................ 3950

average profit per ton milled               $0.000
est. B.E. (CND$ 0.546 nsr)             U.S.$0.000
```

The fourth routine (Figure 18) is used in the development of mine production schedules. A program (MINSIM) simulates mine production and provides estimates of mining rates, truck requirements and operating costs which are then used in the long range schedule. Alternative economic evaluations for the long range production schedule are produced through a CASH FLOW analysis (electronic spreadsheet), which forecasts and summarizes cost, revenues, taxation and the time value of money (DCF) for various pit designs, sequences and cut-off grades. Major capital expenditures can easily be evaluated through these programs.

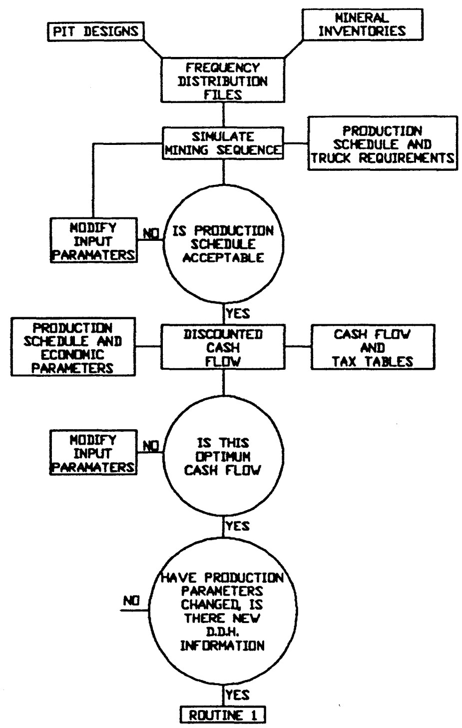

FIGURE 18: ROUTINE 4-- SCHEDULING

Production Engineering

The objective of production engineering is to ensure a continuous supply of mill feed while meeting budgeted production within the framework of the long range mine plan. Short term scheduling scenarios must be quickly assessed to determine their effect on the medium and long range plans. AutoCAD is used to integrate daily production control with short and medium range planning. The results of this integration are accurate projections, volumetric reconciliations and analyses of various "what if" scenarios enabling timely and informed planning decisions to be made.

Production survey control is maintained with the aid of a Geodimeter 422 Total Station instrument which interacts, through a separate data collector, with an on-property network of PC's and Sun workstations. Mine planning functions are serviced by a series of in-house programs developed to integrate production control with mine planning via an AutoCAD based system. The system is designed to allow easier flow of information between planned mining schedules and actual production, permitting faster response to changes in important mining parameters such as strip ratio, mill throughput, pit design and equipment availability.

Production Engineering System - Overview

The foundation of the production engineering control system is the Total Station instrument - it is the connection between design and production. Shovel advances -ie. toes and crests- are picked up along with drilled blasthole locations on a daily basis Monday to Friday. This information, in the form of identifying code, northings, eastings and elevations, is stored in the instrument in the field, transferred to the data collector and downloaded into files via a PC-XT computer for later manipulation within the network of PC's and Sun workstations.

The files of shovel advances and drilled blastholes are used as the basis for the creation of data exchange files, which are used for updating the AutoCAD survey drawings - one drawing per pit. The survey drawings form the basis for several other drawings used in the planning and scheduling process. These drawings are:

1. Daily dig plans.
2. Weekly priority drawings.
3. Asbuilt/current drawings.
4. OP39 generated schedule drawings.
5. Reconciliation drawings.

How these drawings are used in the planning and scheduling process will be outlined in the following section.

Integration of Production Engineering With Mine Planning

1. Daily dig plans -

Once the information from the daily survey has been transferred to the AutoCAD survey drawing and the drawing has been updated, a daily "dig plan" is created. This plot consists of:

- toes and crests of the active benches
- toes and crests of all the benches in the pit
- blastholes drilled since the last survey
- high grade/waste boundries
- nonblasted pattern layouts with the actual location of the borehole, ie. the accuracy of drilling
- drilled, blasted but unmined boreholes with associated assay values

The updated "dig plan" is plotted at a scale of 1"=50' by 12:00 noon where it remains in the engineering office until 4:00 p.m. The drawing is then taken to the pit office where the general foreman adds any additional information that he feels is required for the night shift foreman, such as cat work or cable jobs, equipment moves, etc.

Assays from holes that are mined out are used to calculate the average grade of ore hauled to the primary crusher and waste hauled to the dumps. The holes are erased from the autocad drawing and from a hold file. Drilled holes are added to the hold file directly from files downloaded from the survey instrument data collector. Each hole in the hold file contains various "fields". These fields are assay information, blast data, mined out date, etc. No hole can be removed from the hold file until all the fields have been completed. For example, if a hole was mined out but the blast had not been processed by the mining engineer, then that hole would remain in the hold file. However, since it has been mined out, it would not appear on the daily dig plan plot.

A simple algorithm may be run which takes all of the copper assay values in the hold file, arithmetically averages them into fifty foot square blocks (bas blocks) containing one assay value and puts the information into a data exchange file (dxf file) suitable for using in an AutoCAD drawing. These bas blocks are then used for short term scheduling.

2. Weekly priority drawings -

These are AutoCAD drawings that are generated weekly from the survey drawings and from bas blocks brought in from the hold file. They are used to represent available drilled and blasted reserves by bench along with expected shovel and drill positions necessary to achieve the weekly production target. The weekly priority drawings are plotted on scale 1"=200' and may be compared directly with similar drawings generated for three and six month schedules using OP39 - the interactive scheduling package. Any variations in actual advance versus scheduled advance are quickly seen and action can be taken to revise the short or medium range plans as necessary to fit into long range planning objectives.

Figure 19 Weekly priority drawing

3. Asbuilt/current drawings -

The asbuilt/current drawing is an AutoCAD drawing of a pit which represents the as-mined surface of the whole pit as of a certain date. Should it be necessary to revise short and medium range plans, using the OP39 interactive scheduling package, the asbuilt/current drawing must be modified in order to create surfaces suitable to be read by OP39. Also, an updated mineral inventory must be created using information from the assay values in the hold file. This mineral inventory is known as the 'active' inventory.

It is at this point in the scheduling process that the advantage of maintaining an AutoCAD survey drawing for each pit on a daily basis is fully realized. The process of modifying the asbuilt/current drawing is simply that of transferring the data from the appropriate layers (benches) in the survey drawing to the asbuilt/current drawing. Before the Total Station was purchased there were no Autocad survey drawings. Any modifications to the asbuilt drawing had to be made by a time consuming process of digitizing toes and crests from the mylar dig plans, transferring this information to a drawing and correcting any errors that occurred in the digitizing process before modifying the actual asbuilt drawing.

4. OP39 generated schedule drawings -

OP39 is a mine scheduling program developed to simulate mine development and the depletion of ore reserves. The program uses the GKS graphics utility (similar to AutoCAD). Input requirements are files of the compressed mineral inventory (in this case the 'active' inventory, based on up-to-date bas blocks) and OP27 type surfaces for topography and pit designs.

For short range scheduling (ie. 3 months) or where production levels are constant for a known period, OP39 alone is adequate. However, for longer range schedules (ie. 6 to 12 months), where production totals must balance with ore reserves, the general development is first established with a spreadsheet known as 'PC scheduler'. PC scheduler is used to summarize the results and evaluate truck requirements when the schedule is finalized.

After a mining schedule is finalized in OP39, an AutoCAD drawing is generated containing a plan view of the pit with the bench advances and mineral inventory shown on the active benches. Some modification of this drawing is required before it is suitable for presentation. A book of plans on scale 1"=200' showing advances and potential broken mineral inventory by month for the schedule period is then plotted and put together for presentation purposes. Should any problems show up in the proposed schedule as a result of subsequent discussion, changes can be made by simply rerunning OP39, taking into account the suggested changes.

5. Reconciliation drawings -

Two types of reconciliation drawings are produced:

i) Total volumetric reconciliation (tonnage).
ii) Projected vs actual ore boundary occurrences.

i) Total volumetric reconciliation -

At the end of each month, surfaces are created from the elevations in the AutoCAD survey drawing representing the volume of material mined over the period. The calculated tonnage from this volume is then compared against the production as reported from the cumulative daily truck count used in the pit statistics. Through this process the truck tonnage factor is calculated.

ii) Projected vs actual ore boundary occurrences -

As each bench in a pit is finished, a drawing is prepared showing the occurrence of projected ore boundaries vs actual ore boundaries on that bench. This drawing serves as a visual check on ore placement and provides an early warning system for expected changes in short and medium range schedules.

As a result of the computerization and integration of daily production engineering with mine planning, it is possible to quickly combine up-to-date asbuilt information with new drilling information. This allows scheduling that is based on current conditions. With the OP39 interactive scheduling package, production can be scheduled rapidly and accurately for 3, 6, and 12 months, or even longer periods, as required. These drawings provide the mine planner with a medium in which to evaluate operation's concerns: ie., the most effective utilization of equipment, the coordination of stripping with ore production, and the final adjustment of actual ore reserves versus that which was originally forecasted. The completed drawing is a comprehensive illustration of the production forecast which is then used in presentations to management and operating personnel.

MILLING OPERATIONS AT GIBRALTAR MINES LIMITED

The Gibraltar concentrator commenced operation in March of 1972. The mill was designed to handle 30,000 tons/day. Current capacity ranges from 37,000 to 48,000 tons/day depending on ore hardness. Mill head grades are approximately 0.30% copper and 0.014% molybdenite. Recovery for copper and molybdenum has averaged 78.5 - 80.0% and approximately 40%, respectively. The crushing, milling and tailings disposal are on a 7 day a week, 24 hour/day timetable. There are four operating crews which work on a 12 hour shift schedule. Some features of the Gibraltar concentrator are:

a) Only 2 stages of crushing are utilized, whereas three stages of crushing are more common for large tonnage operations,

b) A coarse mill feed of 40% +3/4 inch ore is possible,

c) A coarse cyclone overflow size which feeds the rougher flotation circuit. Normal grinds are between 35% and 45% +65 mesh. Recovery is acceptable at these grinds with clean ore, averaging 78% to 80% for copper.

d) The centre-line method of tailing dam construction is used to avoid dam failure due to seismic loading,

e) Column flotation technology is used extensively throughout the concentrator flowsheet,

f) Most of the concentrator unit processes are heavily automated using a distributed digital control system to maximize tonnage throughput, copper and molybdenum concentrate grade, and recovery.

Figure 20 (page 36) is a simplified flowsheet of the Gibraltar concentrator and crushing circuit.

PRIMARY CRUSHING

Minus 3 foot ore is crushed in two 54" x 74" gyratory crushers, each powered by a 600 HP motor with a closed side setting of 5 1/2" and an eccentric throw of 1 5/8". The mantle life is approximately 6.0 million tons and concave life is 6.0 to 8.0 million tons.

Present operation utilizes the primary crusher located at the plantsite and a new "pit" crusher to the West of the Gib East pit. Discharge from the primary crusher is conveyed to two parallel double deck 8' x 20' Ripl-Flo screens. The undersize, accounting for 35% of the material is conveyed to a covered fine ore storage bin. Oversize from the primary screens is conveyed to an open air coarse ore stockpile. Details of the primary and secondary screens are given in Table 4.

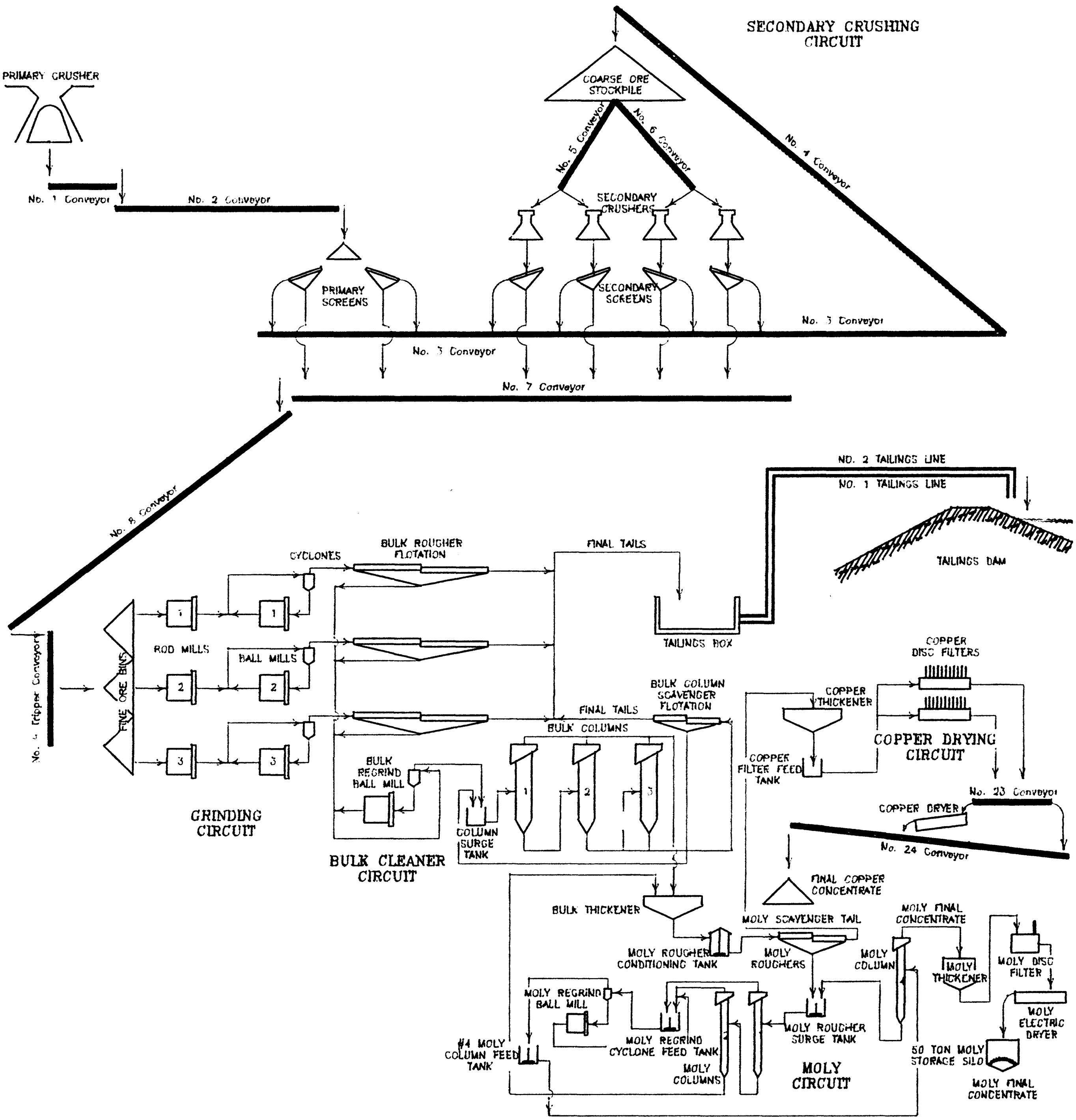

Figure 20: Crusher and Mill Process Flowsheet

TABLE 4

SCREENING

	Primary	Secondary
Make and Model	Allis-Chalmers double deck Ripl-Flo XXH	Allis-Chalmers double deck Ripl-Flo SH
Size	8′ x 20′	6′ x 14′
Number of Units	Two	Four
Screens - Top Deck	2" x 2" cast panels	2" x 2" x 5/8" wire
- Consumption	11 - 2′ x 4′ panels per month	16 - 28" x 72" screen cloths per month
-Bottom Deck	1"x3"x1/2" wire cloth	7/8"x3"x1/2" wire cloth
Consumption	6 - 4′x8′ cloths/month	14 - 28"x72" cloths/month
Amplitude	3/8"	5/16"
Slope	25 degrees	20 degrees
Speed	800 R.P.M.	890 R.P.M.

SECONDARY CRUSHING

Secondary crushing is performed in closed circuit with the coarse ore stockpile. Six 54" x 10' Nico Hydrastroke feeders discharge to two parallel 48" conveyors. Each conveyor feeds two 13" x 84" Hydracone crushers operating in parallel, and discharging onto a double deck 6' x 14' Ripl-Flo screen equipped with 2" x 2" wire screen cloths on the upper deck and 7/8" x 3" screen cloths on the lower deck. The Hydracone crushers are driven by 500 HP motors and have a closed side setting of 3/4" with an eccentric throw of 1 1/2". Mantle life is 0.5 million tons while the concaves last 1.0 million tons.

FINE ORE STORAGE

A 60,000 ton covered stockpile is fed by an automatically controlled tripper conveyor. The stockpile has an estimated live capacity of 30,000 tons. There are 27 evenly spaced draw points with Mexican type tube feeders discharging to nine variable speed cross conveyors. Three of these conveyors combine to feed one rod mill conveyor for each of the three grinding circuits.

GRINDING

There are three identical parallel grinding circuits consisting of a 13.5 foot diameter by 20.0 foot long open circuit rod mill and identical ball mill in closed circuit with five 30 inch diameter Krebs cyclones. Both mills are powered by 2,500 HP synchronous motors. The mills of each grinding circuit discharge to a common launder and to one of two pumpboxes. A 20" x 18" SRL centrifugal pump moves slurry to the 30" cyclones. Cyclone underflow flows by gravity back into the ball mill while cyclone overflow flows by gravity to a rougher flotation bank. See Table 5 for data details on the grinding mills.

The grinding circuit has been automated to maximize circuit throughput, while maintaining a grind of 40% +65 mesh for optimum flotation recovery. The grinding control strategy uses cyclone overflow density to control the fineness of grind to the bulk rougher flotation via the pumpbox water addition. The pupmbox level is then used to control rod mill feedrate. Total recirculating load is also monitored and intervenes in the control strategy if it becomes too high. Tables 5 and 6 list more details on the grinding circuits.

TABLE 5
PRIMARY GRINDING DATA

	ROD MILLS	BALL MILLS
No. of Units	Three	Three
Size	13.5 ft. diam. x 20.0 ft.	13.5 ft. diam. x 20.0 ft
Motor	2,500 HP Synchronous 240 RPM, 4160 volts	2,500 HP Synchronous 240 RPM, 4160 volts
Power Drawn	1900 - 2200 HP	2200 - 2450 HP
Mill Speeds	#1 - 16.5 RPM 78% Critical Speed #2 - 16.5 RPM 74% Critical Speed #3 - 16.5 RPM 78% Critical Speed	All Mills 17.3 RPM 81% Critical Speed
Drive	Air Clutch with Single Helical 320 tooth gear	Air Clutch with Single Helical 320 Tooth gear
Media	180 tons Carbon 1090 4"x19.6′long steel rods	181 tons 2.5" diam. forged steel balls
Shell Liners	Noranda wave, Ni-Hard	Rubber
Trommel Screens	A.R. punchplate with 1" x 1.5" slots	A.R. punchplate with 5/8" x 3" slots

TABLE 6
ADDITIONAL GRINDING DATA

Cyclones	Type	Krebs D30	30" diameter
		Apex	5" diameter
		Vortex Finder	11.5" x 28"
Cyclone Feed Pumps		Primary	20" x 18" SRL centrifugal pump
		Standby	16" x 14" SRL centrifugal pump
Grinding Media		Rods	0.6 lb/ton
Consumption		Balls	0.47 lb/ton (2.5" dia. balls)
Power Consumption			6.0 - 7.0 kWhr/ton
Ore Work Index			10.0 - 15.0
Feed Rate/Section			440 - 750 ton/hour
Rod Mill Feed			40% +3/4"
Cyclone Overflow			35% - 45% +65 mesh
			28% - 40% -200 mesh
Densities		Rod Mill	81% solids
		Ball Mill	74% - 76%
		Cyclone Overflow	45% - 50%

COPPER AND MOLYBDENUM FLOTATION

The three grinding circuits each feed an individual flotation bank of 16 Denver 600H flotation cells. Rougher flotation is carried out at 42% - 44% solids and is operated at a pH of 9.8 to 10.4 with lime addition to the rod mill. Sodium isopropyl xanthate and R200M (a thiocarbomate) are added to the cyclone overflow as flotation collectors, and Oroform F2 is added to the ball mill underflow launder as a frother. At times when depression of pyrite is difficult even with additional lime, R200M alone is used in place of sodium isopropyl xanthate. Emulsion (emulsified fuel oil and water) is added to the flotation feedbox as a collector for molybdenite. See table 4 for more details on reagents used at Gibraltar.

As with the grinding circuit, the rougher flotation banks are fully automated with the computer adjusting reagent feed rates, air volume, flotation density and flotation cell levels in order meet a flotation concentrate grade established by the operator. This automation is used in conjunction with a Outokumpu Courier 300 on-stream x-ray analyzer. Analysis of the flotation feed, flotation tails and flotation concentrate assays are used by the process control computer to control the rougher flotation cells as described. See table 7 for more information on flotation metallurgy.

The tailings of the rougher flotation cells are final tailings. Final tailings are dumped into a concrete sump in the concentrator basement where they flow by gravity down one of two 36" diameter steel pipelines. At approximately 1200 feet from the tailings box the tailings are booster pumped by a 20" x 18" SRL pump to the tailings impoundment site, which is approximately 14,000 feet away.

Concentrate from all three rougher flotation cell banks are combined and pumped to a bulk regrind circuit. The bulk regrind mill is a 9.5 foot diameter by 14 foot long ball mill powered by a 670 HP motor and in closed circuit with two to three 15" diameter Krebs cyclones. Depending on the final copper concentrate grade, thiocarbomate or sodium isopropyl xanthate are used as a cleaner circuit flotation collectors and are added to the bulk regrind discharge box.

The bulk regrind cyclone overflow moves by gravity to a column surge tank. A 10" x 8" SRL pump on the column surge tank pumps to one of two 40' x 7' diameter column flotation cells. Normally two column flotation cells operate in series with the tail of the first column feeding the second column. The concentrate from both column flotation cells is combined and is pumped to a bulk thickener as a bulk concentrate grading 27% to 30% copper and 1.5% MoS_2. The tails of the second column is pumped to a bank of sixteen 300V Denver flotation cells. A third 7' diameter column is used as a spare. The concentrate from the cleaner scavenger flotation cells flows by gravity to the column feed surge tank where it combines with bulk regrind cyclone overflow. The tails of the cleaner scavenger flotation cells flows by gravity to the final tailings box.

Bulk concentrate is thickened in an 80 foot diameter Dorr-Oliver thickener to 50% - 60% solids in preparation for the molybdenum separation circuit. This thickener underflow is pumped to a tank where it is conditioned for 1 hour with CO_2 gas to a pH=8.5. Sodium hydrosulphide, the depressant used in the molybdenum separation circuit, is added to the moly rougher feed. Sixteen Denver DR30 flotation cells produce a molybdenite rougher concentrate which is pumped to two 36" diameter by 40 foot high moly flotation columns for the first stage of cleaning. The tails of the moly rougher flotation cells is sent to the copper thickener for final dewatering of the copper concentrate. The concentrate of the two 36" diameter moly columns flows by gravity to a moly regrind circuit which is a 6 foot diameter by 12 foot long ball mill (200 HP motor) in closed circuit with one 10" diameter Krebs cyclone. Cyclone overflow is pumped to a 30" diameter moly column which produces a final concentrate product grading 85% to 90% MoS_2 and less than 1.0% Cu. See table 7 for more details on the molybdenum flotation metallurgy.

TABLE 7

REAGENT CONSUMPTION

Bulk Circuit		
Sodium Isopropyl Xanthate	0.002 - 0.004 lbs/ton	rod mill feed
R200M	0.005 - 0.008 lbs/ton	rod mill feed
HX-23 (collector)	0.0063 lbs/ton	rod mill feed
P-2 (frother)	0.0147 lbs/ton	rod mill feed
Fuel Oil/Emulsion (collector)	0.0123 lbs/ton	rod mill feed
Lime (depressant)	0.9006 lbs/ton	rod mill feed
Molybdenum Separation Circuit		
Sodium Hydrosulphide (depressant)	11.630 lbs/ton	conc. feed
Dewatering Circuit and Water Treatment Circuit		
CFA-40 (flocculant)	0.0005 lbs/ton	rod mill feed
Millsperse 801 (descalant/antiscalant)	0.0031 lbs/ton	rod mill feed
Millsperse 805 (descalant/antiscalant)	0.0008 lbs/ton	rod mill feed

TABLE 8

FLOTATION METALLURGY

Bulk Circuit	%Copper	%MoS2
Mill Feed	0.28 - 0.30	0.005 - 0.020
Rougher Concentrate	9.0 - 11.0	
Scavenger Tail	0.04 - 0.11	
Cleaner Scavenger Tail	0.30 - 1.50	
Bulk Concentrate	27.0 - 30.0	0.8 - 1.3
Molybdenum Separation Circuit		
Moly Rougher Feed	27.0 - 31.0	0.8 - 1.3
Moly Scavenger Tail (Cu. Conc)	29.0 - 31.0	
Moly Final Concentrate	0.80 - 1.50	
Recoveries		
Bulk Roughers	75.0% - 82.0%	
Bulk Column and Scavengers	93.0% - 98.0%	
Moly Separation Circuit		
Total Recovery	72.0% - 80.0%	20.0% - 40.0%

CONCENTRATE FILTERING AND DRYING

Copper concentrate is thickened in an 80 foot diameter Dorr-Oliver thickener with the overflow combining with the bulk thickener and flowing to an internal reclaim water system. The underflow of the copper thickener is pumped at 65% solids to a 16 foot by 16 foot filter feed holding tank.

One of two 8'6" diam. by 10 disc Dorr-Oliver disc filters produces an average cake moisture of 12%. This is either conveyed directly to the storage shed or sent to a gas fired rotary kiln dryer. Concentrate shipments are 9.0% moisture in the summer and 7.5% in the winter.

Molybdenum concentrate is thickened, filtered on 1 disc 4 foot diameter Dorr-Oliver disc filter and dried in an 60 kW electric dryer to a moisture of less than 10%. The concentrate which grades 85% - 90% MoS2 (molybdenite) is loaded in 4000 pound bags in preparation for shipment.

PROCESS CONTROL IN THE CONCENTRATOR

A Fisher ProVox distributed digital control system is the equipment used for automation in the concentrator. The grinding circuits, rougher flotation cells, bulk column flotation cells, reagent addition pumps, bulk column scavenger flotation cells, moly rougher flotation cells, and moly column flotation cells have all been automated on the ProVox system.

ACID WATER NEUTRALIZATION SYSTEM

Mine acid water is collected in a series of collection ponds surrounding the lower drainage area of the entire minesite. All of these ponds pump to #4 collection pond which is the central collection point for the Gibraltar property. Normally the pH of this water would be 4.0 to 5.0 and carry copper and iron in solution. The acid water is pumped periodically from #4 pond to the mill neutralization tank where it is mixed with slaked lime to neutralize the water to a pH=8.5 and precipitate metals. This neutralized water and precipitate is directed down the spare tailings line for disposal at the tailings impoundment area.

TAILING DISPOSAL

Tailings, at the rate of 10,000 USGPM is pumped from the mill to the tailings dam construction site through one of two 36" diameter steel pipes. The impoundment area is a natural valley with two small saddle dams located to the east and northwest. The main tailings dam is constructed with tailings. Cycloning of the tailing produces sand used in the centerline method of dam construction. A 36" diameter header pipe, running along the entire length of the dam is pressurized by a 20" x 18" Warman booster pump. A series of 30" diameter cyclones mounted on portable 15 foot towers are moved as required by a bulldozer. Cycloning at the tailings dam is only done during the summer.

Sand production is controlled to about 12% passing 200 mesh with a minimum percolation rate of 3.0 inches/hour. The phreatic surface is very low as found by piezometers installed along the dam. The ultimate tailings dam height will be 350 feet.

An extensive rock and gravel finger drain system has been constructed under the sand portion of the dam to provide good drainage at the dam base. A seepage collection dam and pumphouse located in the valley below the main dam collects and returns all water to the tailings impoundment area. A floating barge on the main pond reclaims 80% of all water for use in the mill circuit. Fresh makeup water is used only for pump glands, cooling systems, drinking water, reagent mixing water, moly circuit concentrate launders, onstream x-ray analyzer, and wash water systems.

SOLVENT EXTRACTION AND ELECTROWINNING PLANT (SX-EW)

The Leaching Circuit

The SX-EW plant was constructed in 1986 to recover soluble copper from the low grade waste dumps. To achieve this the dumps are sprinkled with a mildly acidic solution. While percolating through the dumps, the solution contacts the surfaces of liberated copper minerals and dissolves the soluble copper into the form of copper sulphate. The leaching reaction is a combination of bacterial action that oxidizes the sulphides and the solubilization of copper into the acid solution. The plant was designed to produce 14.3 tons per day of cathode copper. Declining available copper values leachable from the dumps have reduced actual production from a peak of 16 tons/day in 1987 to 10 tons/day at present. New oxide material from the operating pits is offsetting the decline. For the bacterial action to occur, the dumps must be near the ideal pH of 2.1 - 2.3 and the necessary factors (oxygen, solution distribution, and dump temperature) must be present.

A total of 3500 to 4500 US g.p.m. is pumped continuously from the plant to the top of the three operating dumps. The solution percolates through the dumps and discharges into the perimeter ditches at the toe. The recovered pregnant solution flows by gravity into collection ponds. Solution from other dumps is pumped to the No. 3 dump pond, which acts as a feed reservoir for the SX plant.

The solution is distributed across the dump via a grid system of 2" pipes drilled at approximately 10' intervals (1/8" diameter holes). A separate grid system is used in winter which discharges solution down cased 7' deep (frost line) holes on a 60' x 40' grid. Surface distribution is not possible in winter due to ice build-up.

The dumps are divided into approximately 100,000 square foot grids and each area is operated on a 14 day leach, 56 day rest cycle to allow the bacterial reactions to take place. Leaching is, however, continuous when a new grid is brought on line while the copper oxide minerals are leached out, as these are not dependant on bacterial leaching.

Solvent Extraction / Electrotwinning

The leach solution or PLS flows by gravity from the No. 3 dump pond to the solvent extraction plant at a rate of 3500 - 4300 US gpm. Plant throughput is temperature dependant and at the low end of the scale in winter. Solution is distributed to three mixer settlers where it is contacted with organic comprising kerosene and an ion exchange agent (Acorga 5038). The copper is extracted into the organic phase which settles out as a layer on the aqueous PLS and is separated by weirs. The now copper barren PLS (raffinate) is returned to the dumps for further leaching. The PLS flows in parallel through the three mixer settlers while the organic flows in series from one to the other.

The copper loaded organic from the extractors flows to a strip mixer settler where it is contacted with strong acid (165 gr/litre). The copper is stripped off the organic ion exchange reagent in this environment. The organic and strong aqueous solution (rich electrolyte) are separated in the settler. The organic is then recycled to the extraction circuit.

The rich electrolyte, containing 36 - 40 gr/liter of copper is filtered to remove entrained organic and then heated to 110 F by lean electrolyte heat exchange and a hot water heat exchanger. It is then pumped to an electrotwinning circuit where the copper is electroplated onto stainless steel cathode blanks using the ISA process.

The solution discharged from the cell house is a lean electrolyte, which is pumped back to the stripper mix box in the solvent extraction plant. There are 32 electrowinning cells with 30 cathode and 31 anode plates in each cell. The cycle for removing cathode plates from the electrowinning cells for copper cathode plate removal lasts about 6 days with about 5 cells being stripped of copper sheets every day. The sheets of copper are steam washed, bundled into 2.5 ton bundles, and sold as a final product. Metallurgical information about the solvent extraction and electrowinning plant is contained in Table 9. Figure 21 is a flowsheet of the entire solvent extraction electrowinning circuit.

TABLE 9
SX-EW PLANT DATA

SX Section - Extraction

Flowsheet: 3 extraction stages, parallel aqueous, series organic

PLS Compostion

Grade	0.45 - 1.0 g.p.l. Cu
Fe+++	2-3 g.p.l.
Suspended Solids	10 - 30 ppm
pH	2.1 - 2.2
Raffinate grade	0.08 - 0.2 g.p.l. Cu

Organic Composition

Reagent	Acorga 5050 (5 - 6% v/v)
Diluent	Shellsol 160
Wetter	RA 30
Loaded organic	2.0 - 2.1 g.p.l. Cu
Barren organic	1.0 g.p.l. Cu
PLS Flowrate	3,500 USgpm (winter, 0 degrees C)
	4,300 us gpm (summer, 12 degrees C)
O:A ratio	1.1:1
Extraction	75 - 80%

Design Criteria

Mixer residence time:	2.7 (summer) - 3.4 (winter) minutes/stage
Settler area:	1.1 (summer) - 1.3 (winter) US gpm/ft^2
Settler Depth (utilized):	30" (10" org, 20" Aq)

SX Section - Stripping

Flowsheet: single stage

Electrolyte advance flowrate	350 US gpm
Electrolyte flowrate to mixer	900 US gpm
O:A ratio:	1.7:1
H_2SO_4 in strip electrolyte:	165 g.p.l.

Design Criteria

Mixer residence time:	3 minutes
Settler area:	1.5 US gpm/ft^2
Settler depth utilized:	30" (10" org., 20" Aq.)

<u>**Electrotwinning**</u>

Nominal max. production rate:	30,000 lb/day
Cathode production:	ISA process - stainless steel blanks
Number of cells:	32
Cathodes/cell:	30 of 990 mm x 990 mm
Anodes:	Rolled Pb - Ca - Sb
Current density:	27 amps/ft^2 nominal maximum
Current efficiency:	88%
Harvesting frequency:	Based on cell weights. Nominal 7 days growth period.
Rich electrolyte:	38 g.p.l. Cu
Lean electrolyte:	30 g.p.l. Cu
Solution temperature:	43 degrees C

<u>Dumps</u>

Set up as grids:	+/- 100,000 ft^2
Operating cycle:	14 days on leach 56 days on rest
Solution rate:	4 - 5 US gpm/1,000 ft^2
Total tonnage:	123 million tons at 0.15% Cu
Anticipated O/A recovery:	39%

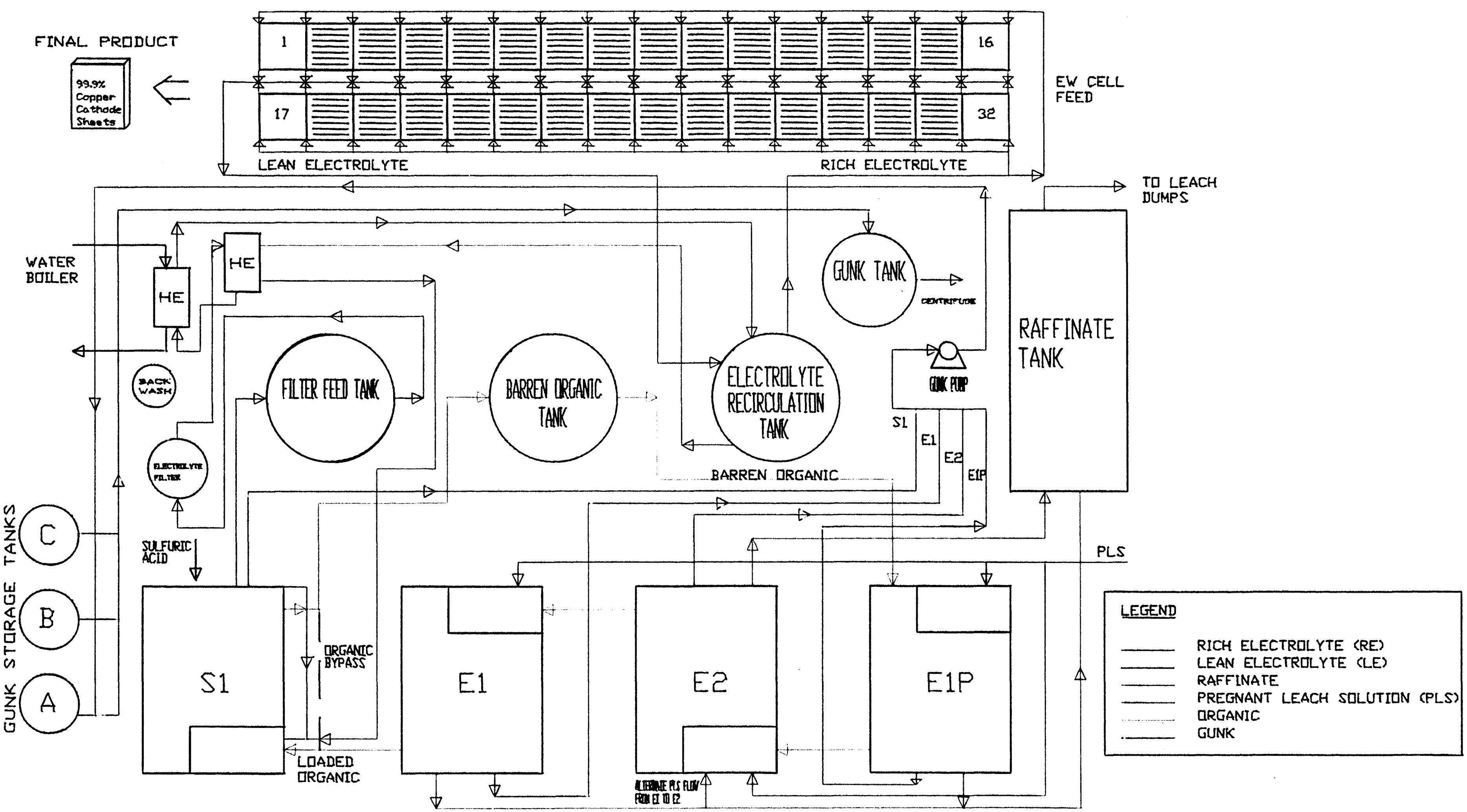

Figure 21: Solvent – Extraction Process Flowsheet

ENVIRONMENTAL ENGINEERING

Environmental control at Gibraltar Mines Ltd. is a top priority and is achieved by following corporate goals. These goals are:

Maintain compliance with all environmental regulatory permits.

Maintain an effective management system of personnel and resources to address all current and future environmental requirements.

Develop and implement preventative strategies and facilities which will ensure environmental security.

Efficiently assess environmental information from monitoring and control programs on an ongoing basis.

Effectively measure and communicate the status of environmental control practices.

Identify and investigate areas of potential liability and develop strategies and programs that will ensure that long term liabilities are minimized.

Gibraltar Mines Ltd. operates under a zero discharge concept, that is to say no water is discharged from the operation into the receiving environment. Environmental staff closely monitor operating performance to ensure high standards are maintained. This requires monthly, quarterly, and annual assessment of sampling programs for both surface and subsurface water quality as well as water balances to account for water usage on the property. These assessments are conveyed to Government agencies at a regular frequency. Parallel sampling programs are also carried out by Government as a quality control check on sampling procedures and analytical methods.

Gibraltar Mines must deal with a complex environmental problem termed Acid Rock Drainage (A.R.D.). A.R.D. is a term used to define drainage that occurs as a result of natural oxidation of sulphide materials contained in rock that is exposed to air and water. A.R.D. will not occur if either the sulphide minerals are non-reactive or if the rock contains sufficient natural potential to neutralize the acid. For practical purposes, the principal ingredients required to promote A.R.D. processes are reactive sulphide minerals, oxygen and water. Oxidation reactions are often accelerated by biological activity given suitable conditions.

This chemical and biological reaction yields acidic water that has the potential to dissolve and mobilize heavy metals such as copper contained within the waste rock. Drainage from waste rock piles that are undergoing this process therefore contain heavy metals which may be detrimental to ecosystems. Gibraltar Mines Ltd. has installed a complex system to collect and pump A.R.D. to the tailing impoundment where it is neutralized.

Realizing the potential for A.R.D. to remove heavy metals from the otherwise non reusable waste rock piles, Gibraltar Mines Ltd. tested, engineered and carried out feasibility studies to evaluate using this natural process to extract copper. On the merit of this work, a Solvent Extraction and Electrowinning Plant was constructed to collect economic values of copper from A.R.D. sources on the minesite. Additional acid is added to waste piles to further promote this copper extraction. Although copper extraction is the primary function of the process it may have a very significant and beneficial side benefit. By way of acid addition, many of the sulphide minerals, especially the fine fraction, will be consumed in the process thus decreasing the intensity of A.R.D. that will evolve from the waste rock dumps with time. Any reduction in this sulphide content is viewed as a significant achievement environmentally.

Once leached and of no further use, waste rock piles will be reclaimed to provide an aesthetically pleasing land mass. To date 1773 hectares of land has been disturbed due to mining activities. Of this disturbance, 19 % has been fully reclaimed using native and non-native grasses and legumes species.

Some of the research programs that are geared towards providing environmental security to the operation, now and long into the future are:

Acid - base accounting of all waste materials to evaluate acid generating properties.

Reclamation steps required to mitigate acid generation processes.

Reclamation test plots for best selection of agronomic species.

Environmental impact assessment and risk analysis.

On going hydrological and hydrogeological assessment.

Fishery potential of the area including trout establishment in the tailing and seepage ponds as well as in the mined out pits.

Closure concepts geared for decommissioning the minesite once ore reserves are depleted.

Gibraltar mines is dedicated to the long term environmental security of its operation. By working together, employees demonstrate their shared commitment to a better environment.

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