Information Brochure

on the Operation of Craigmont Mines Limited

Merritt, B.C.

1 July, 1974

INDEX

GENERAL DESCRIPTION OF MINE AREA

92ISE035 (2W)

The Craigmont Property is situated in south-central British Columbia, about 10 miles northwest of the Town of Merritt and about 120 miles northeast of Vancouver, latitude 50 degrees 12'28" N and longitude 120 degrees 54'48" W. Access to the property is by paved road which connects with No.8 Highway at Lower Nicola and the Canadian Pacific Railway at Coyle.

Moderate local relief ranges from 1,850 elevation at the Nicola River, 2,400 elevation at the plant site, and 4,450 elevation at the west end of the open pit area, to 5,600 elevation at the top of Promontory Hill. The Nicola River and Guichon Creek are the main drainages in the area and occupy broad valleys. Vegetation and climate are typical of the semi-arid "dry belt" of British Columbia, with annual precipitation of approximately 13 inches. Much of the valley bottoms are open and covered with sagebrush. The upper slopes have more rainfall and hence a denser forest cover. Temperature ranges from 20 degrees below zero in winter to 100 degrees above in summer.

The chalcopyrite magnetite-specularite orebodies have a combined strike length of approximately 2,800 feet and extend over a vertical distance of 2,000 feet. Underground ore reserves at 1st November 1973 were 7,900,000 tons grading 1.91% copper. The orebodies lie in steeply south-dipping Triassic Nicola Group rocks which parallel the east-west contact of the south end of the Jurassic Guichon Batholith. In the vicinity of the mine, outcrop exposure is about 10%. None of the orebodies were exposed at surface, but were covered by either a thin layer of glacial till or Cretaceous Kingsvale Volcanic rocks.

HISTORY

The history of the present Craigmont orebody is short, although the Eric showing, situated one mile to the east, was diamond drilled and trenched during or before 1935. The original Company formed to work in the Promontory Hill area was called Pine Crest Mines Limited. This Company was re-organized and incorporated as Craigmont Mines Limited in 1946. In 1954, the fourteen key mineral claims which form the core of the present property were acquired. Prior to 1957 geological and geophysical work was concentrated in the vicinity of Jackson Lake and the Paystin showing. Much of this initial work consisted of the investigation and evaluation of magnetic anomalies of negative magnetic intensity, similar to the anomalies obtained on the Bethlehem Property. It is interesting to note that the anomalies obtained above the Craigmont orebodies were of extremely high magnetic intensity. By the fall of 1957, diamond drilling on the magnetic anomaly in the drift covered area immediately above the No.1 Orebody indicated that an extensive zone of copper mineralization existed. By mid-1958 the results of diamond drill hole No.15 suggested that an orebody of substantial proportions was being delimited. The drill hole passed through the center of the No.1 Orebody, intersecting approximately 640 feet of massive sulphides grading better than 4.0% copper.

Canadian Exploration Limited directed and financed exploration from November 1957 to July 1958, when underground development was undertaken by an operating group, Birkett Creek Mine Operators Limited. This group, formed by Canadian Exploration Limited, Noranda Mines Limited, and Peerless Oil and Gas Company, was dissolved in 1960 and the same interests are presently operating the mine as Craigmont Mines Limited, under the management of Placer Development Limited, the Parent Company of Canadian Exploration Limited.

GENERAL GEOLOGY

The Merritt area is in a copper metallogenic province on a broad belt of copper mineralization extending from Lake Chelan in Washington through Copper Mountain, Princeton, the old Aspen Grove Camp to the Merritt, Highland Valley and Kamloops areas.

In the Merritt area, Jurassic intrusive bodies of granodiorite, quartz diorite and diorite are present. The largest is the Guichon Batholith, which extends from just north of Craigmont to the Highland Valley and from Guichon Creek to the Thompson River.

These rocks intrude the Upper Triassic Nicola Group, a thick volcanic and sedimentary series of agglomerate, breccia, flows, limestones, argillites and greywacke. Many of these poorly sorted sediments immediately adjacent to the batholith have been hornfelsed. The resolution of the sequence and structure in the Nicola series is hindered by lack of outcrop and absence of continuous and distinctive marker horizons. In general, the Nicola rocks are steeply dipping with marked variations in strike from one area to another. This may be interpreted as parallelism of attitude with the strike of the underlying instrusive contacts as a result of forceful intrusion of igneous masses into the Nicola series.

Small areas of Lower Cretaceous Kingsvale and Spences Bridge volcanics and Tertiary Coldwater coal measures also occur in the Merritt area as relatively shallow cappings of the Nicola series.

In the Promontory Hill area, the contact between the Guichon Batholith on the north and the Nicola rocks on the south can be traced roughly from the vicinity of the Mine westward for approximately four miles. The strike of this contact trends slightly north of west for two miles, then swings south of west, while the general trend of the Nicola rocks is approximately south 70 degrees west, with a predominant steep southerly dip.

Impure limestone of the Nicola series outcrops prominently across the top of Promontory Hill and is tentatively correlated with the limestone that forms the host rock for the major orebodies in the Mine.

Lack of continuous outcrop and absence of marker horizons have not permitted resolution of the structure on Promontory Hill. One working hypothesis suggests an anticlinal axis south of the top of Promontory Hill with large drag folds on the limbs. However, other hypothesis may apply and more work is required to resolve the structure.

Agglomerates and flows assigned to the Lower Cretaceous Kingsvale volcanic sequence overlay the Nicola rocks unconformably on the east slope of Promontory Hill and dip gently southward. South of this relatively thin sheet of Kingsvale rocks, the Nicola rocks re-appear, but are intruded by a granitic stock.

On the southwest flank of Promontory Hill, volcanics of Lower Cretaceous Spences Bridge group overlay the Nicola rocks. Minor intrusive quartz prophyry bodies have also been recognized on the western slope.

MINE GEOLOGY

The gross Craigmont structure is that of a large east-west anticline with ore bearing drag folds on the north limb. The Craigmont orebodies lie within a block bounded on

MINE GEOLOGY (cont'd)

the north and east by the Guichon Batholith and on the south and west by separate regional faults. Within this block, the greywackes adjacent to the batholith have been hornfelsed. Immediately south of the hornfelsed greywackes, a copper-iron-magnesium silicate metasomatic (skarn) zone has developed as a dirty limy sediment folded and metamorphosed. Where the limestone contained few impurities, it simply re-crystallized. South of the skarn zone lie more greywackes, some argillites and cherts. Almost all of Craigmonts ore is contained within the skarn zone.

The anticline and subsequent drag folding undoubtedly developed as a consequence of the intrusion of the multistage Guichon Batholith. Areas of disruption - flexures, attenuations and brecciated zones on the drag folds formed areas favourable for formation of actinolite skarn and deposition of copper-iron solutions. The bulk of Craigmont's ore is found in a massive actinolite skarn with varying amounts of magnetite. The Guichon Batholith is the most logical source of the copper rich fluids.

The mineralogy of Craigmont is very simple. Chalcopyrite is virtually the only copper mineral present. Supergene minerals were confined to a narrow oxidized zone immediately above the orebody. Magnetite and hematite account for approximately 20% by weight of the orebody. Non-metallic minerals include actinolite, epidote and garnet. Pyrite is confined to the areas of heavy garnet alteration, but is common as subhedral grains in most of the wall rocks.

CRAIGMONT PRODUCTION

Since commencement of operations in 1961 to 31 April 1974, the Craigmont concentrator has processed 21,629,864 tons or ore grading 1.34% Cu. A total of 1,034,215 tons of copper concentrate, containing over 289,337 tons of copper has been shipped.

OPEN PIT MINING

SUMMARY

The Pit operated from March 1961 to March 1967, and now complete.

The following tonnage was handled:

Overburden by Contractor (6,295,531 cubic yards)	-	11,017,180 tons
Rock by Contractor (1,468,712 cubic yards)	-	3,451,472 tons
Ore and Waste by Craigmont	-	72,989,369 tons
Total	-	87,458,021 tons

A total of 6,922,907 tons grading 1.81% Cu. was hauled directly to the Primary Crusher.

When the pit was completed 7,066,090 tons of ore grading 0.77% Cu. had been stockpiled. This is blended with underground ore as feed to the Concentrator as required.

PIT DESIGN

Mining benches were 33 ft. high, with a 37 ft. berm left every second bench.

This gave a wall slope of 43 degrees for the west and south walls. Due to ground conditions the final overall slope of the north wall was 39 degrees.

The west wall was 1,000 ft. high on completion of mining.

High grade ore (+0.8% Cu.) was sent to the crusher, and low grade ore (0.35 to 0.79% Cu.) was stockpiled.

Top production was at the rate of 50,000 tons per day including ore and waste. A staff of twelve supervised one-hundred and thirty men. There were forty-five men in the repair shop. The engineering staff consisted of five persons.

SLIDE MOVEMENT

In February 1964, wall movement started in the north west corner of the pit. The moving mass was about 1,500 ft., by 500 ft., and a maximum of 250 ft. deep. Total volume was 3.5 to 4.0 million yards. Maximum rate of movement was six inches per day.

The mass consisted of overburden and Kingsvale Volcanic overlaying Nicola rocks, which sloped at 20 degrees. An uncomformity, containing bentonitic clay lubricated the contact.

The slide was stabilized by removing the unbalanced load on the slide and by diverting run-off water away from the slide area. A total of three and one-half million cubic yards of overburden and rock were removed in order to safely assure the continuation of open pit mining.

OPEN PIT MINING (cont'd)

EQUIPMENT

3 - 1400 P & H Electric Shovels with 4½ yard buckets.
1 - 1400 P & H Electric Shovel with 5 yard bucket.
2 - 40R Bucyrus Erie Rotary Drills (9 inch holes).
1 - Boughman Truck (AN-FO mixing).
14- R-27 Euclid Trucks - 27 ton boxes.
5 - R-30 Euclids - 30 ton boxes.
2 - Caterpillar 769 leased - 35 ton boxes.
May 1966, 11 Euclids were sold and 9 more 769's leased.
4 - Tractors
2 - Graders
1 - Michigan Front End Loader (4 yards).
1 - Gardner Denver PR123 Air Trac.
1 - 3000 gallon tank truck with 6½ yard sander.
3 - 3/4 ton trucks.
Capital cost approximately $3,300,000, including buildings, not including crusher, etc.

OPERATING

Rotary Drills: - 9 inch holes up to 43 ft. deep.
Feet drilled per operating shift - 274.
Feet drilled per drilling shift - 373.
Feet drilled per hour - 46.7.
10 feet of sub-grade eliminated big muck on top of blasts on lower benches.
90% of drilling with steel bits - averaging 3,900 ft. per bit.
Pattern was 16 ft. x 18 ft. in ore and 20 ft. x 22 ft. in waste.

Blasting: - Bulk AN was mixed and handled in Boughman Truck. 90% of blasting with AN-FO - remainder with hydromex. Powder factor 0.31 lbs. per ton.

Loading: - Tons loaded per operating shift were 5,294.
Maximum in any one shift - 10,300 tons.

Haulage: - In 1966 when conditions were the same for all trucks, performance was:-
Euclid R.27 - 129 tons/hour.
Euclid R.30 - 141 tons/hour.
Caterpillar 769 - 168 tons/hour.
Average availability of trucks was about 70-75%. Euclid trucks were equipped with diesel engines (336 H.P.) automatic transmissions and torqueomatic brakes. Caterpillar trucks were equipped with 400 H.P. engines, automatic transmissions, disc brakes on the rear wheels and pneumatic suspension.

UNDERGROUND MINING

Extracts from the paper "Some Practical Problems of Sublevel Caving at Craigmont Mines", by E.W. Cokayne.

UNDERGROUND MINING UP TO 1967

While Open Pit Mining was still in operation, exploration, development and production commenced underground. The purpose was to supplement pit tonnages with good grade ore, and to experiment with mining methods.

Two variations of blasthole stoping were tried; benching from sublevels with vertical rings, and horizontal ring drilling from corner raises; in both instances the stopes were backfilled with pit waste. The ground tended to cave.

Transverse cut-and-fill mining was also used in two areas with varying degrees of success. Stopes were mined 25 ft. wide with 25 ft. pillars. Ground support was a major problem.

In 1965, after studying sublevel caving operations in Sweden, it was decided to adopt this method at Craigmont.

REASONS FOR ADOPTING SUBLEVEL CAVING

After our experience with blasthole and cut-and-fill mining, there were only two directions to go - either to more expensive forms of support, such as undercut cut-and-fill or square set stoping, which would be uneconomic; or to a caving method where the weakness of the ground is utilized instead of being resisted. Block caving was not considered suitable because of the relatively small and irregular orebodies surrounded by weak readily cavable waste. Sublevel caving seemed to be the answer. It allows selective mining, requires drift size headings only, leaves no pillars, is well adapted to a high degree of mechanization, and gives low costs as compared to the "support" methods. Its disadvantage is its high dilution factor. However, at Craigmont we feel it is the only practical method for us, and gives the lowest cost per pound possible under our conditions.

CHOICE OF EQUIPMENT

An important decision that had to be made at once was the choice between using large diesel equipment or small air equipment. Large equipment offered high productivity, and the minimum number of orepasses, but high drift maintenance, high mechanical maintenance and high ventilation requirements as compared to air equipment. Due to the bad ground conditions, installation and maintenance of orepasses are high cost items; due to this factor and the advent of shotcreting we decided to go to the larger openings and equipment. We now have two ST4's and seven ST5's. The ST4 units are being converted as engine changes become necessary.

Equipment in use at present is as follows:-

2 - ST4A Scooptrams
7 - ST5 Scooptrams
1 - 3 Boom Gardner-Denver Development Jumbo, with deutz engine (F4L812) and DH123 Rock Drills

UNDERGROUND MINING (cont'd)

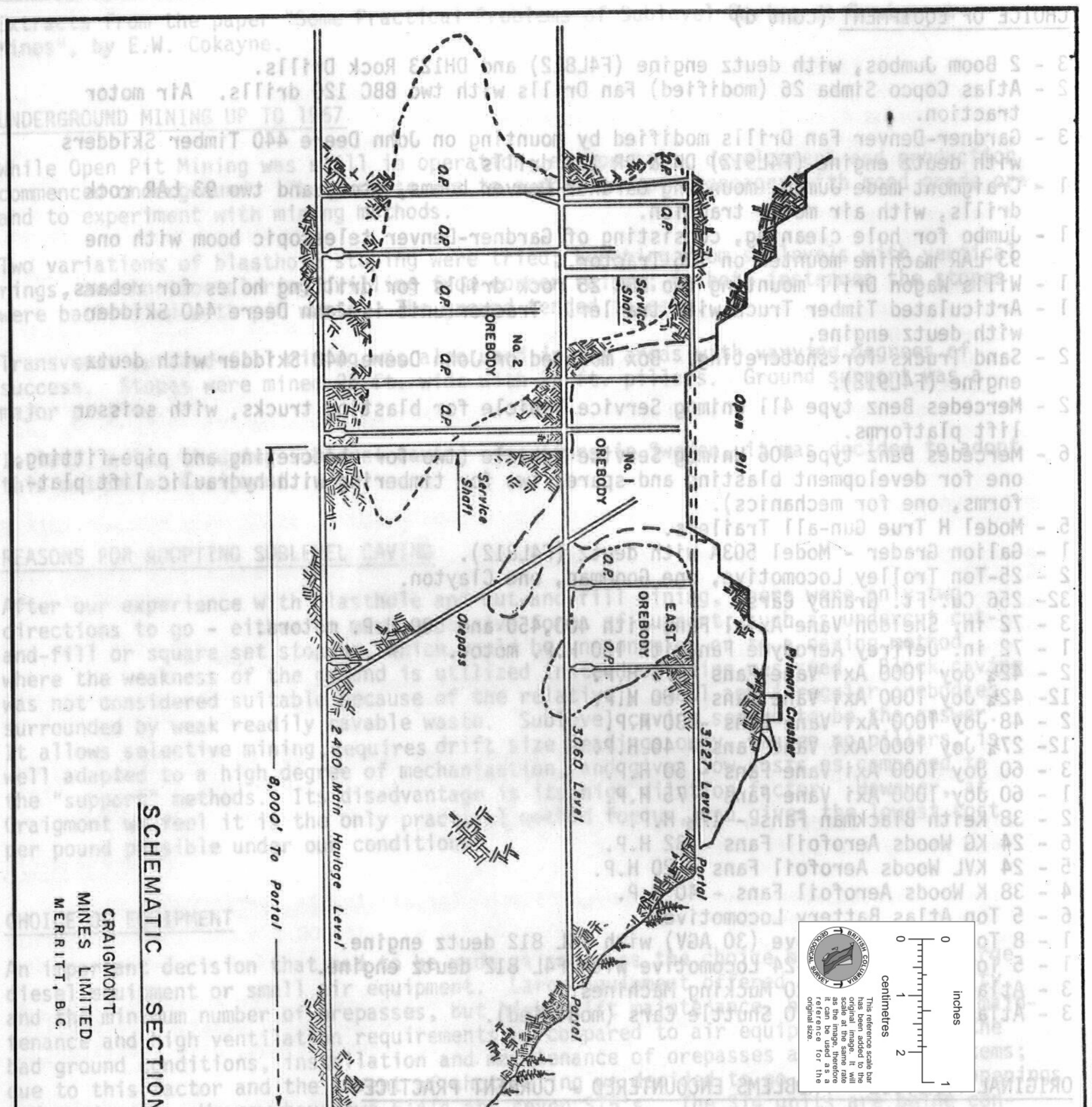
CHOICE OF EQUIPMENT (cont'd)

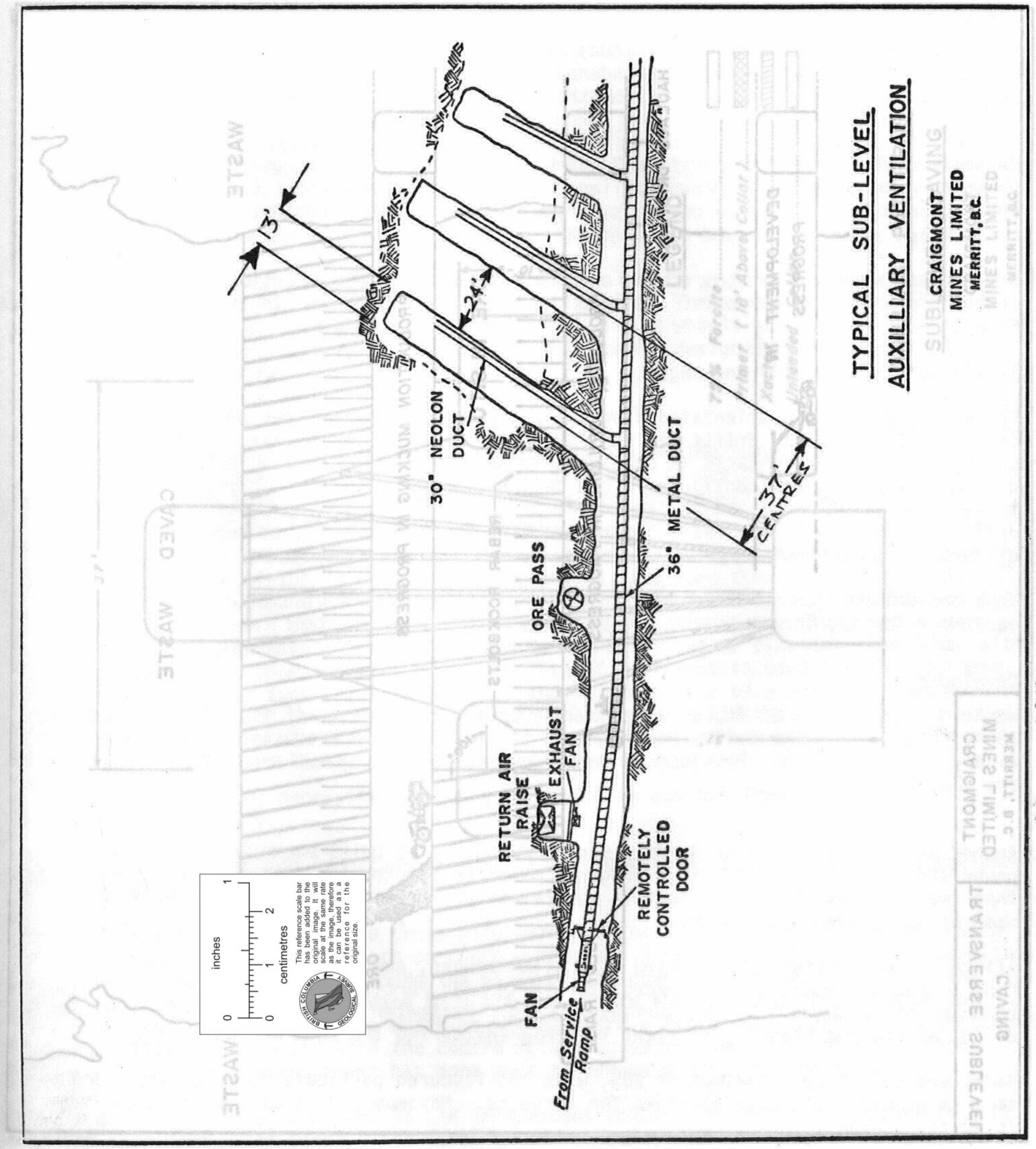
3 - 2 Boom Jumbos, with deutz engine (F4L812) and DH123 Rock Drills.
2 - Atlas Copco Simba 26 (modified) Fan Drills with two BBC 120 drills. Air motor traction.
3 - Gardner-Denver Fan Drills modified by mounting on John Deere 440 Timber Skidders with deutz engine (F4L912) DH & PR 123 drills.
1 - Craigmont made Jumbo, mounting Gardner-Denver booms, etc., and two 93 LAR rock drills, with air motor traction.
1 - Jumbo for hole cleaning, consisting of Gardner-Denver telescopic boom with one 93 LAR machine mounted on TD6 Tractor.
1 - Wills Wagon Drill mounting two BBC 25 rock drills for drilling holes for rebars.
1 - Articulated Timber Truck with trailer. Tractor unit is John Deere 440 Skidder with deutz engine.
2 - Sand Trucks for shotcreting. Box mounted on John Deere 440 Skidder with deutz engine (F4L912).
2 - Mercedes Benz type 411 Unimog Service Vehicle for blasting trucks, with scissor lift platforms.
6 - Mercedes Benz type 406 Unimog Service Vehicle (two for shotcreting and pipe-fitting, one for development blasting and spare, two for timbering with hydraulic lift platforms, one for mechanics).
5 - Model H True Gun-all Trailers.
1 - Galion Grader - Model 503A with deutz (F4L812).
2 - 25-Ton Trolley Locomotive, one Goodman, one Clayton.
32- 256 Cu. Ft. Granby Cars.
3 - 72 in. Sheldon Vane-Axial Fans with 400,450 and 500 H.P. motors.
1 - 72 in. Jeffrey Aerodyne Fan with 150 H.P. motor.
2 - 42¼ Joy 1000 Axi Vane Fans - 75 H.P.
12- 42¼ Joy 1000 Axi Vane Fans - 60 H.P.
2 - 48 Joy 1000 Axi Vane Fans - 30 H.P.
12- 27¼ Joy 1000 Axi Vane Fans - 40 H.P.
3 - 60 Joy 1000 Axi Vane Fans - 50 H.P.
1 - 60 Joy 1000 Axi Vane Fans - 75 H.P.
2 - 38 Keith Blackman Fans - 17½ H.P.
6 - 24 KG Woods Aerofoil Fans - 32 H.P.
5 - 24 KVL Woods Aerofoil Fans - 20 H.P.
4 - 38 K Woods Aerofoil Fans - 40 H.P.
6 - 5 Ton Atlas Battery Locomotives
1 - 8 Ton Sala Locomotive (30 AGV) with F4L 812 deutz engine.
1 - 5 Ton Plymouth FMD-24 Locomotive with F4L 812 deutz engine.
3 - Atlas Copco Cavo 320 Mucking Machines.
3 - Atlas Copco Cavo 510 Shuttle Cars (modified).

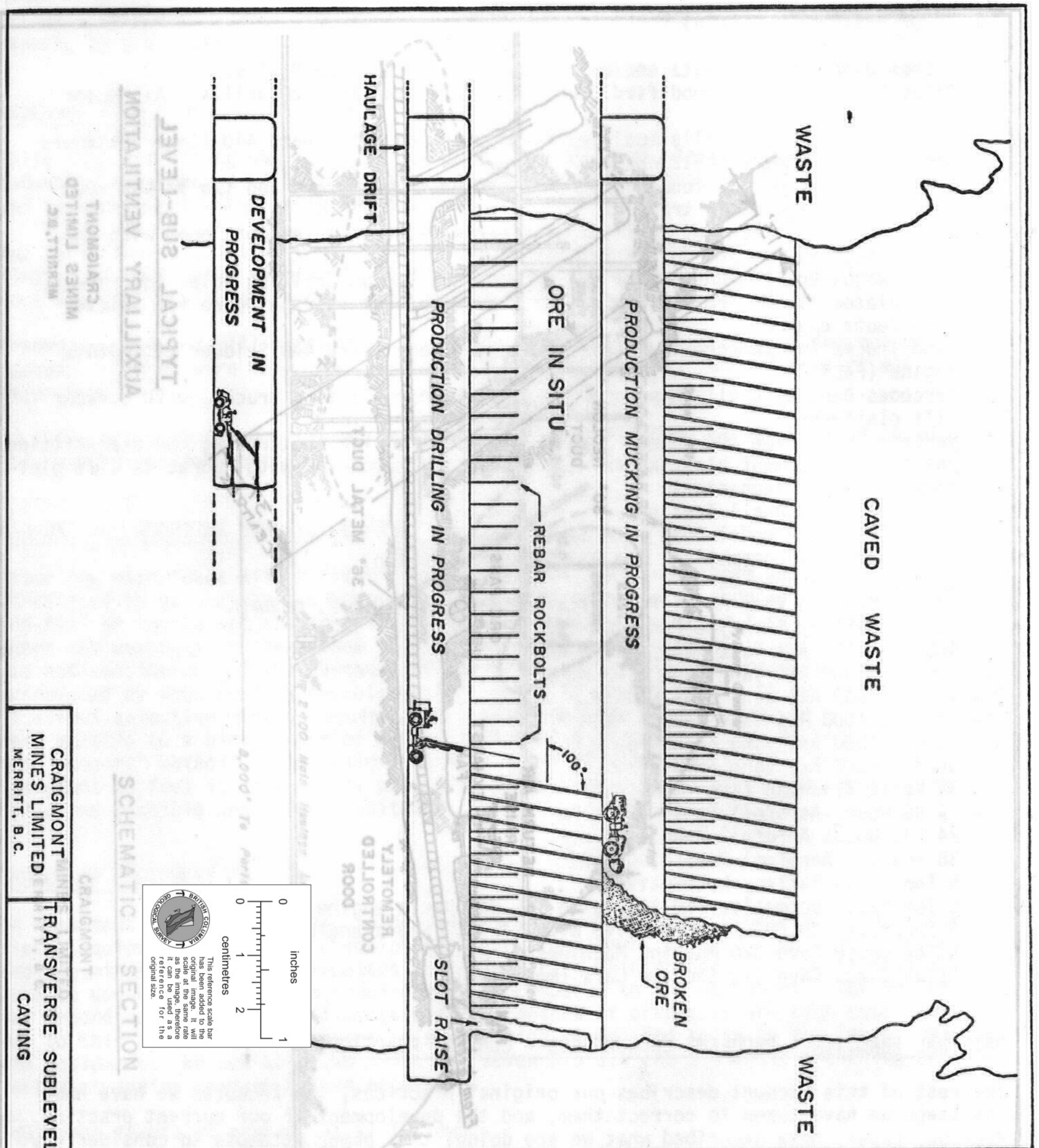
ORIGINAL PRACTICE - PROBLEMS ENCOUNTERED - CURRENT PRACTICE

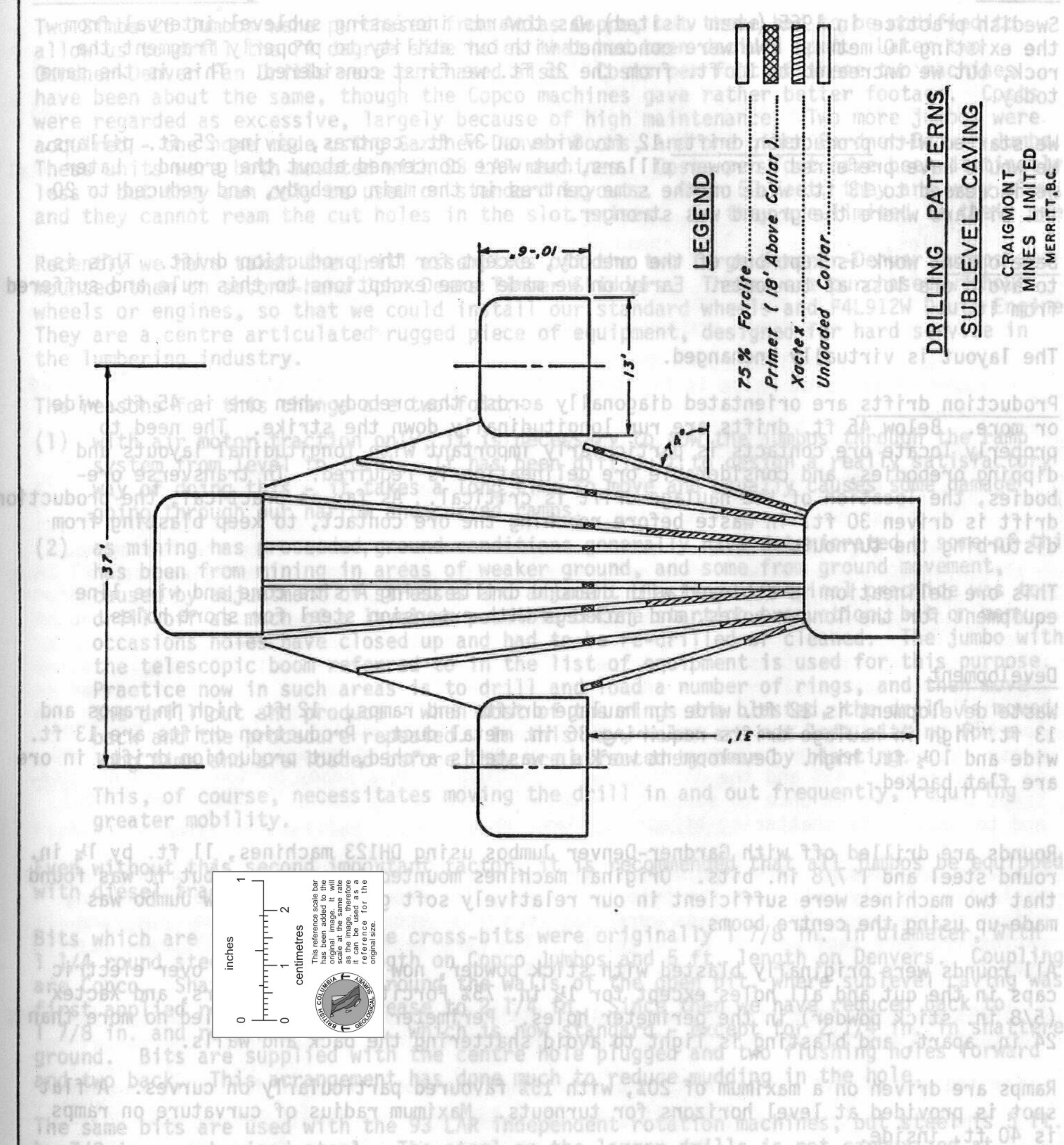
The rest of this account describes our original practices, the troubles we have had, the steps we have taken to correct them, and the development of our current practice. Previous papers have described what we are doing, this paper attempts to consider why. No further attempt is made to describe sublevel caving theory which has already been well covered.

Each phase of the operation will be considered separately.


SCHEMATIC SECTION
CRAIGMONT MINES LIMITED
MERRITT, B.C.
Open Pit
Primary Crusher
3527 Level
3060 Level
2400 Main Haulage Level
8,000' to Portal
Portal
Service Shaft
No. 1 OREBODY
No. 2 OREBODY
EAST OREBODY
Orepass
O.P.


TYPICAL SUB-LEVEL
AUXILIARY VENTILATION
CRAIGMONT MINES LIMITED
MERRITT, B.C.
13'
24'
37' CENTRES
30" NEOLON DUCT
36" METAL DUCT
ORE PASS
RETURN AIR RAISE
EXHAUST FAN
REMOTELY CONTROLLED DOOR
FAN
From Service Ramp


WASTE
CAVED WASTE
PRODUCTION MUCKING IN PROGRESS
ORE IN SITU
REBAR ROCKBOLTS
PRODUCTION DRILLING IN PROGRESS
HAULAGE DRIFT
DEVELOPMENT IN PROGRESS
100'
BROKEN ORE
SLOT RAISE
WASTE
CRAIGMONT MINES LIMITED MERRITT, B.C.
TRANSVERSE SUBLEVEL CAVING


DRILLING PATTERNS
SUBLEVEL CAVING
CRAIGMONT MINES LIMITED MERRITT, B.C.
LEGEND
75% Forcite
Primer (18" Above Collar)
Xactex
Unloaded Collar
10'-6"
13'
37½'
7½'
15'

UNDERGROUND MINING (cont'd)

Layouts

Swedish practice in 1965 (when visited) was towards increasing sublevel interval from the existing 10 metres. We were concerned with our ability to properly fragment the rock, but we increased to 31 ft. from the 25 ft. we first considered. This is the same today.

We started with production drifts 12 ft. wide on 37 ft. centres, giving 25 ft. pillars. We would have preferred narrower pillars, but were concerned about the ground. Later we increased to 13 ft. wide on the same centres in the main orebody, and reduced to 20 ft. pillars where the ground was stronger.

Development work is kept out of the orebody, except for the production drift. This is to avoid ore loss at turnouts. Early on we made some exceptions to this rule and suffered from it.

The layout is virtually unchanged.

Production drifts are orientated diagonally across the orebody when ore is 45 ft. wide or more. Below 45 ft. drifts are run longitudinally down the strike. The need to properly locate ore contacts is particularly important with longitudinal layouts and dipping orebodies, and considerable ore delineation is required. In transverse ore-bodies, the location of the haulage drift is critical. As far as practical, the production drift is driven 30 ft. in waste before reaching the ore contact, to keep blasting from disturbing the turnout.

This ore delineation is achieved with diamond drills using A size core and wire line equipment for the longer holes, and jacklegs with extension steel for short holes.

Development

Waste development is 12 ft. wide in haulage drifts and ramps, 12 ft. high in ramps and 13 ft. high in haulage drifts requiring 36 in. metal duct. Production drifts are 13 ft. wide and 10½ ft. high. Development work in waste is arched, but production drifts in ore are flat backed.

Rounds are drilled off with Gardner-Denver Jumbos using DH123 machines, 11 ft. by 1¼ in. round steel and 1 7/8 in. bits. Original machines mounted three booms, but it was found that two machines were sufficient in our relatively soft ground, so a new Jumbo was made-up using the centre booms.

All rounds were originally blasted with stick powder, now Amex is loaded over electric caps in the cut and all holes except for 1¼ in. 75% Forcite in the lifters, and Xactex (5/8 in. stick powder) in the perimeter holes. Perimeter holes are drilled no more than 24 in. apart, and blasting is light to avoid shattering the back and walls.

Ramps are driven on a maximum of 20%, with 15% favoured particularly on curves. A flat spot is provided at level horizons for turnouts. Maximum radius of curvature on ramps is 40 ft. inside.

UNDERGROUND MINING (cont'd)

Production Drilling

Two Simba 26 Jumbos were purchased from Atlas Copco, but these had to be modified to allow us to drill the 74 degree side holes that had been decided upon. Later two Gardner-Denver Fan Drills were purchased also. Costs per foot of these two machines have been about the same, though the Copco machines gave rather better footage. Costs were regarded as excessive, largely because of high maintenance. Two more jumbos were acquired - one home made using Gardner-Denver Booms, and one other Gardner-Denver Jumbo. These units were both mounted with 93 LAR machines. Costs per foot have been distinctly less - but they can only be used on standard layouts. Over 50 feet, they are very slow, and they cannot ream the cut holes in the slot raises. So they are limited in their use.

Recently we have taken the drill assemblies off the two large Gardner-Denver Jumbos and mounted them on second-hand John Deere Timber Skidders. These were purchased without wheels or engines, so that we could install our standard wheels and F4L912W Deutz Engines. They are a centre articulated rugged piece of equipment, designed for hard service in the lumbering industry.

The reasons for this change are two-fold:-

(1) with air motor traction only, it is necessary to tow the jumbos through the ramp system from level to level. It has been difficult to design a really satisfactory way of doing this - it takes a long time to move and usually causes some damage going through our narrow and curved ramps.

(2) as mining has proceeded, ground conditions generally have deteriorated - some of this has been from mining in areas of weaker ground, and some from ground movement, caused by adjustment of stresses and adjacent blasting. Original practice was to drill off as much of a level as possible before starting production, but on many occasions holes have closed up and had to be re-drilled or cleaned. The jumbo with the telescopic boom referred to in the list of equipment is used for this purpose. Practice now in such areas is to drill and load a number of rings, and then move the drill out and produce - when most of the rings are blasted, the drill is moved back and the procedure repeated. In this way, holes are not left sitting for a long time and are loaded before they are affected by near-by blasting.

    This, of course, necessitates moving the drill in and out frequently, requiring greater mobility.

Even without this second important factor, it is recommended that all jumbos be equipped with diesel traction.

Bits which are tungsten carbide cross-bits were originally 1 7/8 in. in diameter, with 1 1/4 round steel of 4 ft. length on Copco Jumbos and 5 ft. length on Denvers. Couplings are Copco. Shattered ground around the walls of the open pit, where sublevel caving was first applied required an increase to 2 1/4 in. Since then we have reduced back to 1 7/8 in. and returned to 2 in. which is now standard - except for 2 1/4 in. in shattered ground. Bits are supplied with the centre hole plugged and two flushing holes forward and two back. This arrangement has done much to reduce mudding in the hole.

The same bits are used with the 93 LAR independent rotation machines, but steel is 5 ft. by 7/8 in. carburized steel. The steel on the larger drills is not carburized at present, though we are seriously considering using it now. We have had a problem with excessive loss of rods in the holes due to mudding, but the increase back to 2 in. bits has helped this.

UNDERGROUND MINING (cont'd)

Slotting

Several different layouts for slotting have been used:-

(1) Originally two short raises were driven conventionally up to each of the sublevels 30 ft. above. Rings were drilled up to the top sublevel kicking into the raises and into the cave above. After some initial success, the top section often failed to break and the method was abandoned.

(2) A fan method was tried in one area, with holes fanning up from the almost horizontal to 70 degrees, slashing down into the drift. Trouble was experienced after about 30 ft. of height was attained; a considerable drift length was required to effect the slot, and ore was left ahead which could not be drawn.

(3) Now a 6 ft. by 6 ft. slot raise is driven up the middle to the top sub. It is drilled off with the production drill (about 50 ft. high), and blasted in 12 ft. sections. Two methods are used to open the slot:-

a) Holes are drilled round the slot to open it up to the fan width. The last round of the slot raise is blasted with these holes. Disadvantages of this method are:-

(i) It is not possible to ensure that the slot raise has broken through to the cave.
(ii) In bad ground it is difficult and sometimes dangerous to get at the slash hole round the raise.

b) Hole are fanned forward into the raise. This is rather like the fan method described above, except that it uses the raise as an opening and holes can be steepened up much faster. In this case the raise round is blasted through and mucked if necessary. This method is standard now.

Blasting

Apart from the importance of good fragmentation, the protection of the brows requires close attention. Brow collapse was the first major problem we encountered, resulting in recoveries of 45% and the difficulty of loading rings already buried in broken rock. Steps were taken at once to support the brows mechanically (this is described later) and to reduce the shattering effect of blasting.

Blasting practice at this time was to use ANFO (except in wet holes) and to bring the explosives fairly well down the hole to fragment the brow. To reduce the shattering effect on the brow, 20 ft. leads were used on milli-second delay blasting caps, and explosives were kept well from the collar. The same Xactex used in development was used to gently blast near some of the collars.

At least two rings were pre-loaded ahead of the blast to minimize the problems of flooding the rings with broken rock when the brow overbroke.

Later the use of ANFO in production blasting was abandoned for the following reasons:-

(1) The need to pre-load, and the fact that holes were not blasted right away.

(2) The problems with shattered ground. Sometimes you could load far more ANFO into the hole than was required. Once I saw ANFO being loaded in one ring and being blown out of the one behind.

UNDERGROUND MINING (cont'd)

Blasting (cont'd)

(3) There were so many ineffective blasts.

(4) Problems with water in the holes.

Because of the shattered ground, and the tendency of the holes to close up after blasting, Primacord is used in all the holes to ensure complete detonation.

The actual loading is achieved with "anti-static hose" using 1¼ in. by 24 in. cartridges of 75% Forcite, and Xactex near the collar. The primer is 18 ft. from the collar. Loading is from a scissor-lift platform on a type 411 Unimog, or from wooden stagings, where multiple rings are being loaded.

Originally 1½ in. powder was used, but we found the smaller size still gave satisfactory fragmentation, and was easier to load in our rough holes.

Load-Haul-Dump

This system needs no description as it is well known. We have had no unusual problems after the maintenance crews became familiar with the scooptrams and established effective procedures. However, productivity has increased from about 300 tons per manshift, over a round trip average distance of 1,800 ft. to 500 tons per manshift over 1,300 ft. Good roads have certainly done something to help.

At first roads were far from good, partly due to the soft ground - the scooptram made holes at turnouts which would fill with water. Our own waste (or even ore) was too soft to use for ballast. Due to the narrow width of the drifts, it was not possible to maintain ditches on the side of the drifts, and drainage was a real problem. Drifts are driven at 3%. Practice now is to bring in crushed and screened diorite from surface to use as ballast. This is oversize from the screening plant that provides sand for shotcrete and aggregate for concrete. It is crushed in the secondary crusher and screened to 1½ in. This is distributed on the road-bed by scooptrams and graded with the Galion Grader. Now a ditch is maintained in the centre with the grader, and as long as the water is run-off this way, the sides are packed hard with the scooptram tires.

Eight inch holes are drilled between sublevels for drainage.

Engine life on scooptrams has been approximately 7,000 hours with 9,500 hours maximum.

Tires used are mostly Firestone 18.00 x 25 - 24 ply, White Pine Compound - slick tread with Michelin Tubes.

Recaps vary from nil to 14 - averaging about 4 per tire. New tires last about 980 hours and recaps about 635 hours. Main problem is with side cuts. Availability of scooptrams is about 75%.

Third and fourth gears have been removed from the transmission.

Ground Control

This has been a major problem in our relatively poor ground.

UNDERGROUND MINING (cont'd)

Ground Control (cont'd)

Originally, we had expected that about 1½ to 2 in. of shotcrete would adequately support our ground, and that this would replace timber. In point of fact we are now using grouted rebar bolts, timber and steel in various sections of the mine as well as shotcreting all the headings. Shotcrete is used in 100% of the headings; Rebars on all production drifts and about half the waste headings; Timber on 15% of the headings; and Steel in special areas, usually in turnouts or where headings have had to be driven through fault zones.

True Gun-all Model H Machines are used to apply wet shotcrete as headings are driven - usually a round at a time, but in stronger ground several rounds may be taken before gunning. Wet shotcrete is used because of the problems of keeping the sand dry in underground storage raises. In one section of the mine, more accessible from surface, dry shotcreting has been used effectively with an Aliva machine and a batch plant feeding self unloading cars. An average of 700 cubic yards are placed each month. Three cubic yards are blown per manshift. Approximately 22 rebars are installed per manshift (including drilling) in development headings where installation is usually on a round by round basis, and 35 rebars are drilled and installed per manshift in production drifts where large numbers are installed at a time.

Shotcrete is found to be very effective in weak ground where no movement or pressure is applied. The thin skin of concrete can seal and bear the weight of loose ground, and hold it together. However, if there is any movement along fault planes or any appreciable pressure, the concrete is subjected to tension or more compression than it can stand. As headings are driven, supervisors judge if they need rebarring as well as shotcreting, or if it is necessary to go straight to timber. After the heading is driven, it is often necessary to do repair work, such as re-shotcreting or adding rebars or timber. Sometimes, due to deterioration of the drift, all three are used, of course, the rebars would have been omitted if the final timber stage could have been predicted.

Rebars are installed in a pattern of fanned 8 ft. deep holes, and consist of 3/4 reinforcing bars grouted into the holes using a two sand to one cement mix, placed with the True Gun-all Machine. Rockbolts with expansion shells have too weak an anchorage in the broken ground. Two major factors in maintaining mine headings are time and mining sequence. As far as practical, the headings should be driven only just before they are required for production preparation. Mining sequence should be strictly adhered to in order to avoid throwing stresses on other areas still in operation. Whenever for some reason, we have strayed from the planned sequence, we have had to pay the price.

Timber sets are constructed using double 10 in. by 10 in. Fir caps standing on 10 in. by 10 in. posts. Posts are battered at 5 degrees to help resist side pressure. Two complete sets are installed per three man shift, using scissor lift platforms on Unimogs. Where space is available, cribs of 10 in. by 10 in. timber are used.

Special steel sets are used where timber is unable to bear the loads. Caps consist of 10 in. by 10 in. x .45 in. wall hollow structural steel tubing stiffened near each end by 1/2 in. steel plates. Posts are steel clad 10 in. x 10 in. fir timbers. Two 10 in. channels are welded together with plate and the post driven through. The purpose here is to provide strength to resist side pressure. These sets though expensive, have been very effective. Many have been salvaged and re-used.

Ground conditions in the sublevel caving in the Main Orebody below the Open Pit have proven rather unpredictable. Depth is not great being about 600 feet below the floor

UNDERGROUND MINING (cont'd)

Ground Control (cont'd)

of the pit. Usually ground conditions have been better here than in most other sections of the mine, but periodically pronounced movement has taken place, throwing considerable weight on ground support and tending to rotate the caps, causing the posts to go off the vertical. At this time some developed areas have caved through two sublevels up to the cave. This condition is presumably caused by a re-adjustment of stresses caused by mining. It eventually settles down. The last such case was over two years ago - we were expecting movement some months ago, but as we are approaching the bottom of this orebody and it is reducing in area, perhaps movement will not be as serious.

Control of ground is the "name of the game" at Craigmont. There are many mines with worse ground, where much more support is required; our problem is associated with providing the minimum support to satisfy safety and draw control requirements. Most of our workings cave shortly after we abandon them.

Draw Control

As we all know this is the crux of the matter with sublevel caving. As mentioned above, when we first started, recovery was 45%, and the draw was out of control - we were grateful for a good stockpile of ore from the open pit to feed the mill while we got squared away.

The primary problem was the collapse of the brows after blasting. Numerous remedies were tried to control this failure:-

(1) pre-splitting of first 10 feet of fan.
(2) use of Xactex near collars of fans.
(3) variations in drill patterns.
(4) expanding shell rock bolts between fans (including bolting timbers to the back).
(5) rebar bolts installed with epoxy.
(6) full length caps supported on bullhorns and blocked to the back.
(7) rebar bolts installed with mortar grade concrete.

The full-length caps on bullhorns were effective only about 50% of the time. The best and most economical results were obtained by grouting a row of six 8 ft. x 3/4 in. rebar bolts between each fan of blastholes.

Pre-splitting proved ineffective. Xactex near the collars is still used. Various fan inclination burdens and drill patterns were tried, ranging from 65 to 90 degrees and from 3½ to 5 ft. burden. Current practice is to incline fans at 80 degrees towards the cave and to use a burden of 3½ ft. (4 ft. in one section of the mine). Usually two rings are blasted at a time, but sometimes if fine waste invades the ore pile very early in the draw, better results are obtained blasting rings one at a time.

The use of brow blocks (10 in. x 10 in. timber caps supported by bullhorns) improved recoveries considerably and provided the opportunity to experiment with other methods. Rebar bolts installed with epoxy were successful, but the use of mortar was cheaper, easier to apply and equally effective. This is our present practice and generally gives good results. This is not to say that brows are always perfectly square after blasting as we would wish them to be, but we do seem to be getting acceptable recoveries.

The effect of brow collapse is as follows:

UNDERGROUND MINING (cont'd)

Draw Control (cont'd)

(1) Ore floods the drift and covers the next row or rows of holes to be blasted. When this happens, the holes are dug out, if possible, (a somewhat hazardous operation) or they are lost.

(2) If the brow is uneven, i.e. not flat and horizontal, the ore funnels down through the high spot. This reduces the width of ore flow with consequent ore losses.

(3) When the brow overbreaks, the ore floods further back into the drift than was intended. This prevents the scooptram from digging in its proper position relative to the broken ore. As a result, a thin draw of ore occurs with waste being pulled down the brow without loading the full depth of ore blasted. In this case, single ring blasts give best results.

(4) When the brow caves back a fair distance, or there are high spots from previous sloughing during the development drifting, and it is obvious that the brow will end up in a high spot if only one or two rings are blasted (thus causing ore loss), it may be necessary to blast more than two rings at a time to pass the high area. Recovery will be poor from such a blast, but records are kept to plan for overdraw on lower levels.

As well as trying to keep square brows, the following steps are taken to control draw:-

(1) Primary blasting is supervised to ensure good fragmentation for an even and continuous flow of ore, i.e. ensuring that holes are properly cleaned and loaded to the toes and that proper blasting procedures are followed. "Bridging" also results from improper loading and blasting.

(2) During the draw, hang-ups or partial hang-ups are blasted immediately to ensure continuous flow.

(3) Information is provided for supervisors and operators so that they may recognize the desired cut-off points. This is closely supervised.

One of the most important duties of the production shiftboss is to make the decision as to when ore recovery from any given blast has reached the optimum in terms of dilution and recovery.

(1) Grade of ore in situ is determined by sampling the sludge produced from blasthole drilling; the samples are taken from the drift floor after drilling.

(2) Tonnage blasted can be computed from drill hole layouts.

(3) With the above information, the ore-waste ratio is calculated for the appropriate cut-off. For example, assuming the waste dilution has no copper values, the ore in place is 2% copper, and the desired cut-off is at 0.5% copper, then mucking should stop when the muckpile appears to consist of 75% waste and 25% ore. The theoretical number of scooptram buckets to achieve this cut-off is provided to the supervisor in advance. Distribution of copper values in Craigmont orebodies is very erratic and wide fluctuation in grade are common, consequently the theoretical number of buckets to achieve a given cut-off grade can vary widely from fan to fan

UNDERGROUND MINING (cont'd)

Draw Control (cont'd)

(4) If the supervisor finds after drawing the theoretically correct number of buckets that the pile still appears to be above cut-off grade, he will naturally continue to draw. If the opposite conditions occurs, i.e. cut-off grade reached substantially too early, he will draw an extra ten buckets as waste in the hope of getting the ore to run again.

(5) It is considered impractical to sample and assay to determine cut-off because:-

a) It is difficult to get a reasonable sized representative sample of a muckpile usually consisting of fine waste and relatively coarse ore.

b) Three or four buckets can change the grade of the pile considerably so that sampling would have to be done at very frequent intervals to be meaningful.

c) A great number of working places would be necessary to maintain production while awaiting assay results.

Flooding

The Open Pit, unfortunately, catches much of the run-off from melting snow in the spring. A ditch is maintained just above the pit wall to catch most of this, but the snow between the ditch and the pit inevitably runs in, and water also drains into the ground and comes out in the pit wall. This water carries mud with it from alluvial deposits. This mud lays in the bottom of the pit, and is a serious menace to the draw points 200 to 250 feet below. Draws tend to pipe up to surface very rapidly and tap the mud. The difficulty here is to tickle it sufficiently to stop it building up in the pit, but at the same time to be able to control it in the production drifts. There have been several runs of mud which have pushed the scooptram right out of the production drift into the haulage drift. Fortunately no injuries to men or equipment have resulted.

Draw had to be restricted to try and prevent piping up to the pit floor, resulting in some ore losses. In some cases mud would reach the muckpile after only 15% of the draw had been removed. Mud entered the gaps in the caved rock and set up almost solidly after water drained off. This caused trouble with blasting, allowing no room for expansion of the rock, and resulted in "bridges".

Orepasses

This has always been a major problem at Craigmont, due to the fact that raw raises will cave out rapidly when used to pass rock. Raises are steel lined and concreted, with bins above the chute. Ore packs and hangs up if the raises are filled, so they are operated empty with high maintenance cost. This however has been the subject of another paper.

Ventilation

When sublevel caving was first started, approximately 150,000 c.f.m. of air was passed through the mine using one 400 h.p. 73 in. axial flow in the exhaust drift. Auxiliary ventilation in the sublevels was provided by 32 h.p. fans with 30 in. flexible duct. Due to air losses through the cave, increasing use of diesel equipment and the raising of the

UNDERGROUND MINING (cont'd)

Ventilation (cont'd)

standards of acceptable ventilation conditions, it was necessary to make major changes involving driving additional drifts and substantially increasing mechanical ventilation.

Primary ventilation now passes 380,000 c.f.m. of fresh air using two adits as intakes. Each adit is equipped with a two stage 73 in. vane axial direct driven variable pitch 500 h.p. 1,200 RPM fan capable of delivering 190,000 c.f.m. at 13.5 in. SWG. Fans are installed in airlocks. Air is exhausted through two routes, one equipped with the original 400 h.p. fan, and the other with a 150 h.p. axial fan. Secondary ventilation is provided by twin 60 h.p. fans through 36 in. metal duct in the haulage drift, and 30 in. flexible duct in the production drifts. On levels below the cave, fans are also installed in the exhaust raise to balance pressure on the level and minimize leakage through the cave.

The use of diesel equipment underground adds greatly to the cost of ventilation.

At present, installed ventilation capacities are as follows:-

Primary Ventilation - 1450 H.P.
Secondary Ventilation - 1750 H.P.

Maintenance

This vital operation is carried out almost completely in an underground shop, 280 ft. long 20 ft. wide and 16 ft. high. It is serviced by two 5-ton overhead cranes. Improvements in maintenance procedures during the past few years has done much to improve the mine efficiency.

Cavo Units

There is one independent section of the mine where a modification of Cavo units powered by Compressed Air have been used instead of using diesel equipment. The same principles as far as sublevel caving are applied here and no description is included.

RESULTS

Production

The operation was designed for 3,000 tons per day on three shifts per day seven days per week. We had difficulty reaching this figure. However in 1972, we averaged 4,700 with a peak of 5,400 tons per day in June. During 1974 production has averaged in excess of 5,000 tons per day.

This improvement is attributed largely to the following:-

(1) Improved ventilation.

(2) More working places available - this is partly a function of improved ventilation, i.e. two scoops can operate where only one could before.

(3) Improved maintenance of equipment.

UNDERGROUND MINING (cont'd)

Production (cont'd)

(4) Improved brows.

(5) Improved roads.

(6) General improved supervision and operator skills through experience.

(7) Improved organization.

Efficiency

Total number of men working underground are as follows:-

	Supervision	Work Force	Total
Mining Department	25	170	195
Mechanical Maintenance	6	44	50
Electrical	1	6	7
			252

Tons per manshift from underground has increased from 15 to 27.

Recovery & Dilution

Recovery has improved from the 45% mentioned above, to a cumulative average of 89%. Dilution however, has increased somewhat and is now 34%.

MILLING OPERATIONS

The preliminary metallurgical design at Craigmont was based on the results of samples obtained from the original 3500 level drift during 1959. Pilot plant test runs and other research testing for design purposes were conducted at the Canadian Exploration Ltd. Laboratory at Salmo, B.C. The design flow sheet was altered in progressive stages from 2,000 to 3,000 and finally 4,000 tons per day as the ore reserve situation became better defined. Construction work on the 2400 level mill site commenced in late 1960. The official opening of Craigmont Mines Limited took place on 15 September 1961, a little over two years from the time that the metallurgical assessment of the original ore samples commenced.

Mill throughput to date has varied widely both on a daily and a monthly basis, and has depended on the varying hardness of the ore. The grindability of the ore appears to be related to the oxide Fe content which varied from lows of 6 to 9% at the top of the Open Pit Orebody, to 25 to 30% in the lower section of the Open Pit. Daily tonnage has thus ranged from 4,000 to over 7,000 tons per day. The average tonnage throughput obtained since the inception of operations in September of 1961 has been 5,100 tons per calendar day, and performance has been above that anticipated.

The paper following describes in detail the various milling functions from the gyratory crusher at the 3700 level of the Open Pit operations to the tailings disposal area at the 2200 level.

PRIMARY CRUSHING

At present 100% of the mine output comes from underground; some low grade stockpile ore remains at the Open Pit.

Underground ore is hauled on the 2400 level in 16 ton side-dump cars and dumped into a 600 ton surge bin ahead of the 48 x 60 in. jaw crusher. Crusher discharge at minus 5 in. is conveyed to No.2 stockpile.

There is a metal detector located at the belt onto which the jaw crusher discharges. Tramp iron from the mine is removed at this point.

TABLE I - Operating Data - 2400 Level Jaw Crusher

Operating Personnel - One man per shift - 3 shifts per day - 7 days per week.

Capacity - 300 to 600 TPH.

Jaw Life - Upper plates - 3.6 million tons. Lower plates - 1.5 million tons.

Pit ore is dumped from 50 ton trucks directly into a 42 x 65 inch gyratory crusher. Crusher discharge at minus 5 in. goes to No.1 Stockpile at the 3650 level, prior to being transported by cable belt conveyor 5,800 feet downhill to No.2 Stockpile at the 2450 level.

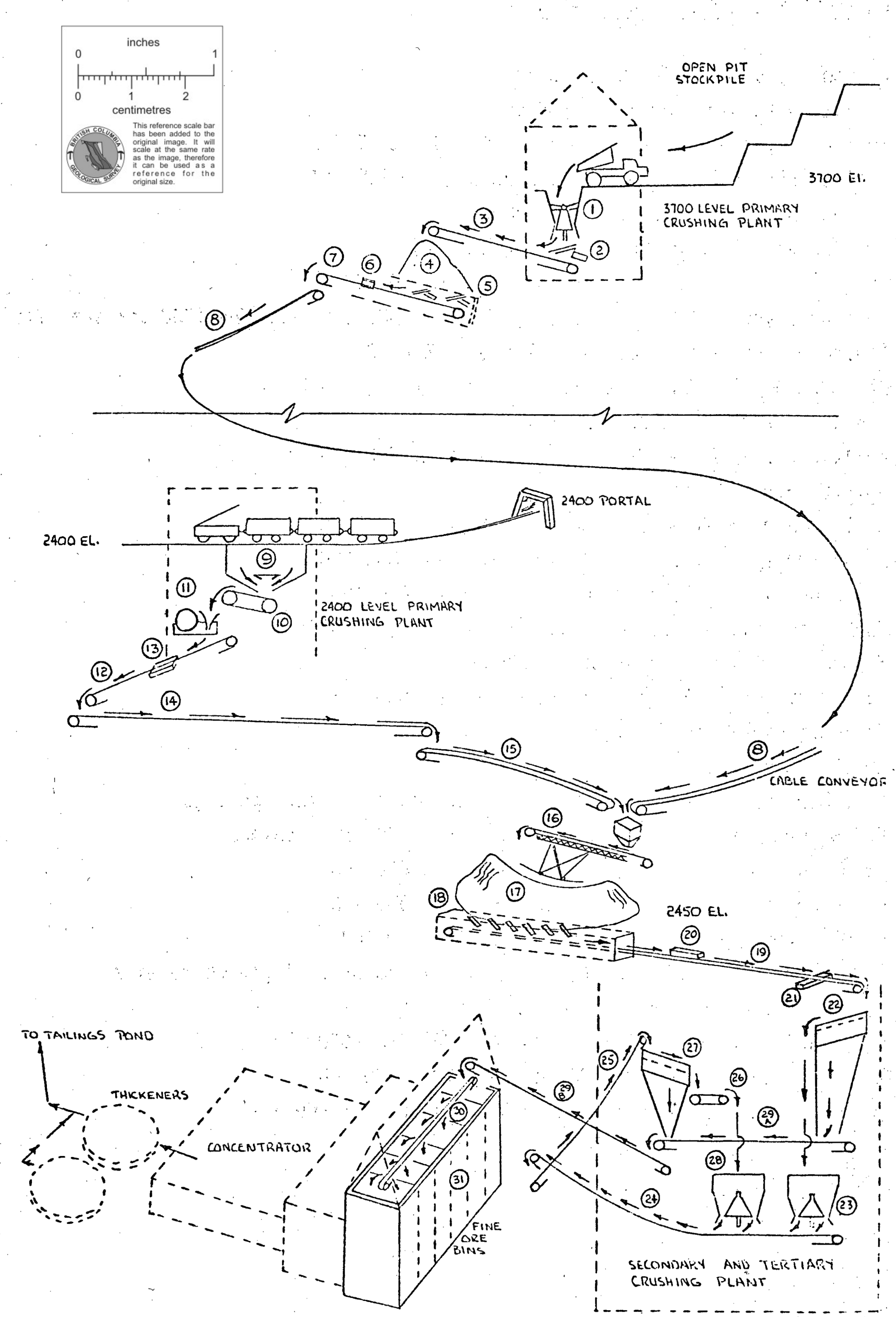
TABLE II - Operating Data - 3700 Level Primary Crusher

Operating Personnel - One man per shift - 1 shift per day - when operated.

Capacity - 800 to 1,000 TPH.

Mantle Life - 2¼ million tons.

Craigmont Mines Ltd.
CRUSHING AND CONVEYING FLOWSHEET



CRUSHING & CONVEYING FLOWSHEET LEGEND

(1) Gyratory Crusher - 42 in. x 65 in.
(2) Vibrating Feeder - 60 in. x 108 in.
(3) No.1 Conveyor - 190 ft., 42 in., 300 FPM, 900 TPH
(4) No.1 Stockpile - L.L. 5000 SDT plus D.L. 10,500 SDT
(5) 2 - 36 in. x 72 in. Vibrating Feeders
(6) No.1 Conveyor Scale (Skinner Weightometer)
(7) No.2 Conveyor - 370 ft., 30 in., 350 FPM, 400 TPH
(8) No.3 Conveyor - 5800 ft., 30 in., 300 FTM, 400 TPH, decline 1200 vertical ft.
(9) Coarse Ore Bin - L.L. 600 SDT plus D.L. 600 SDT
(10) Apron Feeder - 5 ft. x 16 ft., and Grizzly
(11) Jaw Crusher 48 in. x 60 in.
(12) No.14 Conveyor - 260 ft., 36 in., 350 FPM, 600 TPH
(13) No.5 Conveyor Scale
(14) No.15 Conveyor - 660 ft., 36 in., 350 FPM, 600 TPH
(15) No.16 Conveyor - 450 ft., 36 in., 350 FPM, 600 TPH
(16) No.4 Conveyor - 120 ft., 36 in., 450 FPM, 800 TPH
(17) No.2 Stockpile - L.L. 5000 SDT plus D.L. 31,700 SDT
(18) 6 - 48 in. x 72 in. Vibrating Feeders
(19) No.5 Conveyor - 360 ft., 36 in., 350 FPM, 600 TPH
(20) No.2 Conveyor - Transweigh Scale
(21) Metal Detector
(22) S.D. Screen - 5 ft. x 12 ft. Perforated Plate - 1 in. openings
(23) Hydrocone Crusher - 14 in. x 84 in. - set 7/8 in. openings
(24) No.6 Conveyor - 170 ft., 36 in., 350 FPM, 800 TPH
(25) No.7 Conveyor - 125 ft., 36 in., 350 FPM, 800 TPH
(26) No.8 Conveyor - 16 ft., 30 in., 250 FPM, 350 TPH
(27) S.D. Screen - 8 ft. x 16 ft., ½ in. x 3 in. slots, single chute wire
(28) Hydrocone Crusher - 5 in. x 84 in., set 3/8 in.
(29) No.9 (A&B) Conveyors 170 ft., 30 in., 350 FPM, 600 TPH
(30) No.10 Conveyor - 165 ft., 30 in., 450 FPM, 600 TPH
(31) Fine Ore Bin - 8,400 SDT (L.L. 8,500 SDT, D.L. 1,500 SDT)

MILLING OPERATIONS (cont'd)

TABLE II (cont'd)

Concave Life - Bottom three rows turned from dump side to opposite at 2¼ million tons - life 3½ million tons.

CABLE BELT CONVEYOR

The 5,800 foot cable belt conveyor transports ore from No.1 stockpile at the 3650 level to No.2 stockpile, 1,200 vertical feet below when required to augment the underground supply. The prime reason for having chosen the cable belt over a conventional conveyor system was the lower capital cost. Cable belt operating costs to date are similar to, or better than, those expected for a comparable standard conveying system.

Contrary to standard conveyor installations, the tension in the cable belt is taken by two 1½ in. diameter wire ropes, which run on grooved pulleys, set at 25 foot intervals on steel and concrete stands. Raised grooves molded along the edge of the belt on both sides hold the belt on the cables during the conveying and return cycles and there is only sufficient tension maintained in the belt to prevent its creeping downhill along the cables. Lateral strength to support the load between the cables is provided by 3/16 in. square spring steel bars molded transversely between the 2-ply fabric body of the belt on 3 in. centres. At the discharge end of the conveyor, the belt is separated from the cables, which pass through the tensioning mechanism at this point, and is fed back onto the cables for the return cycle. A similar separation occurs at the feed end of the conveyor where the belt passes through its tensioning mechanism and the cables pass over the drive sheaves. The 200 foot belt sections are joined by wire-hook hinged fasteners.

The belt is driven by a 4,200 V., 720 r.p.m., 370 h.p. wound rotor induction motor which is geared to the two, seven foot driving sheaves. The sheaves have a single groove with a twelve degree slope to centre, and the cables are wound 2½ times symmetrically around their respective drive sheaves to eliminate lateral thrust on the driving mechanism.

The motor drives the belt until the load on the belt is such that the belt motion is self sustaining. At this point the belt drives the motor at super synchronous speeds and power is generated and fed back into the circuit. The generation of power, brakes the belt to a constant speed, some three to four percent, about the synchronous speed of the motor, and when fully loaded the system yields approximately 325 h.p.

An important consideration in the design and operation of the conveyor is to prevent even a short duration over-load. The characteristics of an induction motor is such that the torque increases only for a short range above synchronous speed, then decreases with further increase of speed. Exceeding the critical speed for torque may result in a run-away of the conveyor. To prevent over-loading the conveyor, while at the same time loading it to the maximum allowable, an automatic tonnage control regulates the feeding of the conveyor from No.2 stockpile by varying the amperage on the vibrating feeders.

Shoe type brakes are applied by counterweights to two 30 in. diameter drums on the driving mechanism and are hydraulically released. On shutdown, the braking is initiated immediately, four seconds later the power cuts off, and at the end of seven seconds the brakes are fully applied. Emergency stops occur with power failure, over-speed, belt off and belt slippage. Over-speed protection is provided above the normal 300 f.p.m. running speed.

MILLING OPERATIONS (cont'd)

CABLE BELT CONVEYOR (cont'd)

The belt is uncovered for its entire length and this gives rise to a number of operating problems. During rainy periods the ore becomes damp during its twenty minute travel down the hill. Wet fines drip off on the return journey and can build up to the point where they lift the belt off the cable. Tripper switches have been installed to meet this situation and stop the conveyor when the belt leaves the cables. Normal practice is to stop the belt during the periods of abnormally heavy rainfall. During the winter months the belt is run twenty-four hours per day to prevent frost forming in the rope grooves. Frost or snow allows the belt to slip on the cable and folds form on the return side. Tripper switches stop the belt when a fold occurs and the fold must be manually worked out of the belt.

The belt is cleaned by two rotating nylon brushes mounted at, and just back of the discharge point. During the winter when the belt is run empty, compressed air is used to blow snow from the belt at a point 300 feet back from the discharge. A similar arrangement is used to blow fines from the upper side of the returning belt just prior to its being fed back onto the upper pulleys.

TABLE III - Operating Data - Cable Belt Conveyor

Operating Personnel - Feed End - 1 man per shift - 2 shifts per day, 5 days per week.

Operating Personnel - Discharge End - 1 man per shift - 3 shifts per day, 7 days per week.

Length - 5,800 feet.

Vertical Distance - 1,200 feet.

Maximum Grade - 16 Degrees.

Average Grade - 12 Degrees.

Cable Life - 2¼ million tons on north cable - 2 3/4 million tons on south cable.

Power Generated - 0.6 KWH/Ton when running at 350 t.p.h.

Belt Replaced - Belt life 10 million tons.

SECONDARY & TERTIARY CRUSHING

To remove tramp metal, the conveyor belt from the lower stockpiles passes through a tramp metal detector prior to entering the secondary crushers. The high magnetite content of the Craigmont ore prevents the use of a conventional electro-magnet or electronic metal detector. It was, therefore, necessary to develop a detector that did not employ magnetic forces directly.

Experimental work conducted by Rens Company showed that metals passing through the field of seach coil produced and effect on the "Q" factor of the coil that was several times larger than the effect produced by the ore. The "Q" factor is the ratio of the inductance to the resistance of the coils. The detector consists of coils mounted below the conveyor so arranged in a monitoring circuit that tramp metal will stop the conveyor at a pre-set surge in the "Q" factor. The operator then removes the offending piece of metal from the ore streams. Variations in temperature alter the resistance of the coil and, during periods of sudden climatic change, frequent adjustment of the contact gap is required. The detector has given satisfactory performance since it was installed during September of 1962.

MILLING OPERATIONS (cont'd)

SECONDARY & TERTIARY CRUSHING (cont'd)

The feed to the 14 x 84 in. Hydrocone secondary crusher is screened on a 5 x 12 foot vibrating screen using perforated plate with 1 in. diameter round openings on staggered centres. The secondary crusher discharge is screened on a 8 x 16 foot single deck vibrating screen with ½ x 3 in. slots, 0.225 in. wire, prior to entering the 5 x 84 in. Hydrocone tertiary crusher which is in closed circuit with the 8 x 16 foot screen.

Both the 5 x 12 foot and 8 x 16 foot screens are inclined at 20 degrees. The undersize from both the screens at 15 to 20% plus ½ in. is conveyed to the fine ore bin where a tripper conveyor distributes the ore over the six compartments of the bin.

TABLE IV - Operating Data - Secondary & Tertiary Crushing

Operating Personnel - 3 men per shift - 3 shifts per day - 7 days per week.

Secondary Crusher Setting - 7/8 in.

Secondary Crusher Mantle Life - 2½ million tons.

Secondary Crusher Concave Life - 2½ million tons.

Tertiary Crusher Setting - 3/8 in.

Tertiary Crusher Mantle Life - 900,000 tons.

Tertiary Crusher Concave Life - 900,000 tons.

8 x 16 ft. Screen Cloth Life - 6 weeks.

5 x 12 ft. Perforated Plate Life - 4 - 5 months.

Operating data for the various phases of crushing and conveying are given in Tables I, II, III and IV.

GRINDING

The fine ore storage consists of a six-compartment, reinforced concrete bin of 8,400 tons live capacity. A transverse slot feeder delivers ore from the full width of each compartment and the steep angle of the compartments walls eliminates hang-ups of ore.

By means of transverse conveyors, three bins feed each of the duplicate grinding sections. The ore is continuously weighed on the rod mill feed conveyor and discharge from the rod mill at 80% solids. The rod mill discharge at 25-30% plus 10 mesh enters the ball mill - cyclone closed circuit and joins the ball mill discharge, which is maintained at 80% solids. The joint stream gravitates to one of two 10 x 8 SRL-C pumps, each of which feeds two Krebs D20 cyclones. One of the cyclone circuits is always spare. Each grinding circuit has an Autometrics PSM 100 installation to automatically regulate the product size at 56-60% minus 200 mesh. The cyclone overflow density controls the final product size, but this density must be varied over the range 46-56% solids, with variations in the hardness of the ore, and hence the tonnage treated.

TABLE V - Grinding Cyclone Data

Cyclone Size - Krebs D20B

Vortex Finder - 6 3/4 in.

MILLING OPERATIONS (cont'd)

TABLE V (cont'd)

Apex Size - 3½ in. ceramic.

O/F Density - 46-56% solids.

U/F Density - 85-90% solids.

Inlet Pressure - 7 - 9 psi.

Pump Speed - 560 RPM.

TABLE VI - Grinding Mill Data

	Rod Mills	Ball Mills	Regrind Mill
Size	9'6" x 12'	11' x 14'	6' x 8'
Horsepower - installed	600	900	150
- drawn	400	930	95
Speed - R.P.M.	19.0	18.4	24.7
- % critical	74.5	77.8	79.0
Grinding medium	3½" x 11'6" rods	2" balls	1½" balls
Type	0.85% C Steel	Forged Steel	Forged Steel
Consumption lb/ton	0.50	0.49	0.028
Liners	Ni-hard integral shell liners	Ni-hard double wave shell liners	Trelleborg rubber liner & lifter bar

Slow speed (240 R.P.M.) synchronous motors drive the mills, directly connected to the pinion through a Fawick air-clutch. The air clutch has permitted the use of a synchronous motor of standard design, rather than a specially designed motor with high starting torque, which would be more costly and would have lower efficiency. The clutch also provides ease of operation in starting or inching the mills. The use of synchronous motors has resulted in a reduction of over-all power consumption since operation of the motors on a lead power factor balances lag factors in the remainder of the property load centres, and a reduction of overload peaks during starting.

A system of interlocks operates from the Fawick air clutches on the mills back through the conveyors in series to the slot feeders, so that no spillage of ore can occur in the event of conveyor failure, etc. The air clutches also cannot function unless the motors are synchronized, so that the mills cannot be started unless full power is available. In the event of power failure, the clutches release, and the ore-stream stops.

Changes in the grinding circuit since the commencement of operations have included the increase in the speed of the mills, the change from one-piece to two-piece wedge bar lifters, and finally to integral liners in the rod mills. Rubber liners were installed in one ball mill, but were subsequently rejected due to poor wear. The mill speeds were increased in April, 1962 from the original speeds of 16 R.P.M. for the rod mills and 16.5 R.P.M. for the ball mills. The integral liners have resulted in reduced downtime for lifter replacement. The mill discharge pumps, 10" x 8" SRL, each feed two cyclones and have been slowed to 560 R.P.M. to reduce the circulating load. An automatic on-stream particle size analyser has been installed on each of the two grinding sections.

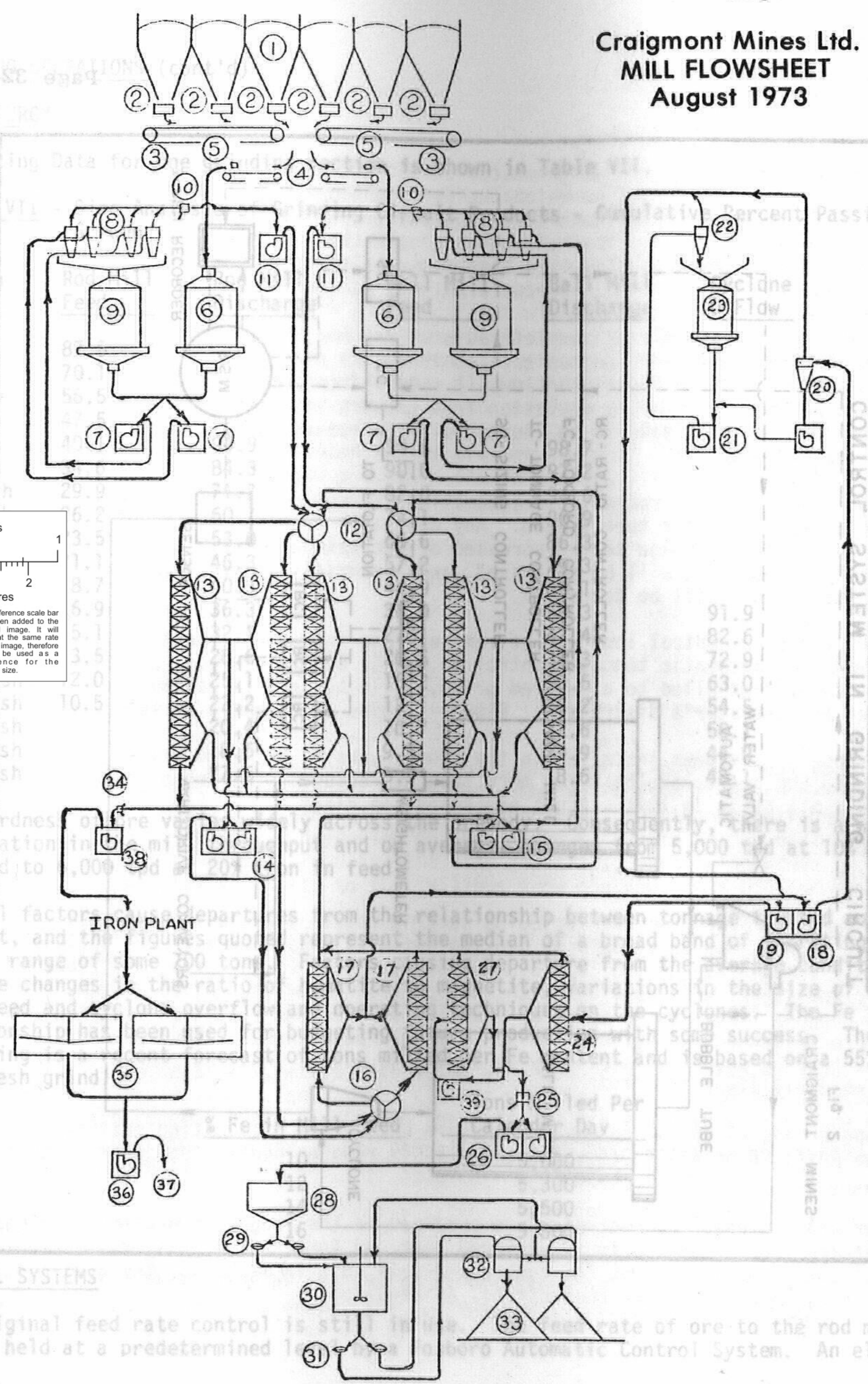
CONCENTRATE (COPPER) LEGEND

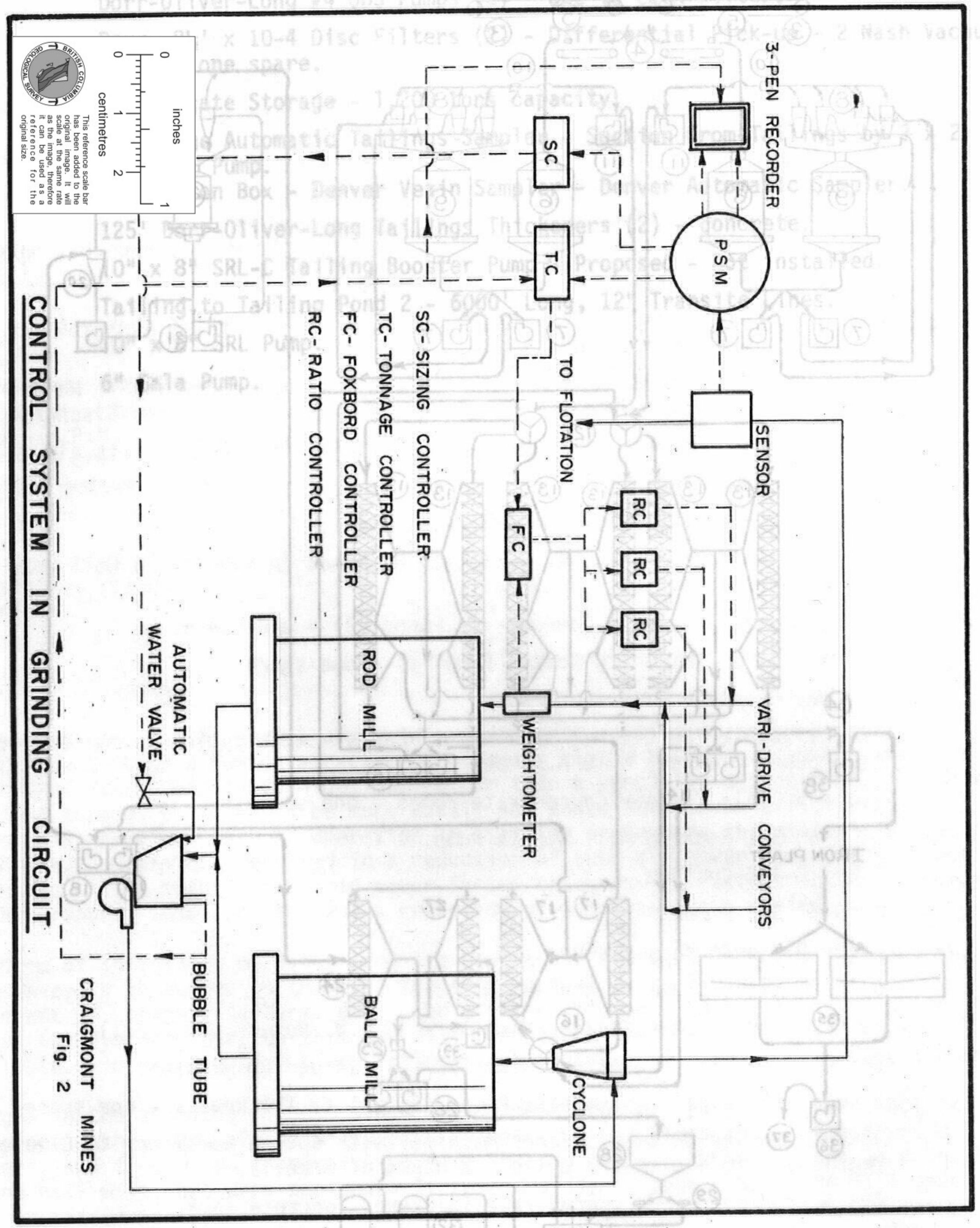
Ref.	Equipment
1.	Concrete Fine Ore Bin - 6 Comp. - 8,400 tons cap.
2.	Slot Feeders (6) - 30" Heavy Duty Belts - U.S. Vari-Drive.
3.	24" Transverse Conveyors (2).
4.	30" Rod Mill Feed Conveyors (2).
5.	Skinner Scales Weightometers - Converted.
6.	9½' x 12' Dominion Rod Mills - 600 hp Tamper Synchronous Motors - Fawick Air Clutches on Pinion - Chute Feeders - Trommel Screen Discharges.
7.	10" x 8" SRL-C Pumps (4) - One spare on each section.
8.	Krebs D20B Cyclones (8) - two per each pump - 3½" ceramic apex - 6 3/4 Vortex Finders - Inlet area 21 sq. in.
9.	11' x 14' Dominion Ball Mills (2) - 900 h.p. Tamper Synchronous Motors - Fawick Air Clutch on Pinion - Chute Feeders - Trommel Screen Discharge.
10.	Single Stage Automatic Samplers - sample taken from Autometrics air eliminator
11.	10" x 8" SRL-C Pumps (2) - one spare.
12.	6 - way Pressure Distributor (2) - one spare.
13.	18 Cell #48 Agitair Rougher Flotation Machines (6 Banks) - 6 Cell Open Flow Units - 15 h.p. motors per 2 cells.
14.	10" x 8" SRL-C Rougher Concentrate Pumps (2) - one spare.
15.	10" x 8" SRL-C Middling Return Pump (2) - one spare.
16.	3 - way Pressure Distributors.
17.	10 Cell Denver DR Cleaner Machines (3 Banks - with option to switch one bank to a Super Cleaner) - 10 h.p. per 2 cells.
18.	Two - 6" Sala Cleaner Concentrate Pumps - one spare.
19.	6" x 6" SRL Regrind Mill Bypass Pump.
20.	Krebs D-15B Cyclone.
21.	6" x 6" SRL Pump.
22.	Krebs D10 Secondary Cyclone.
23.	6' x 8' Allis Chalmers Regrind Mill.
24.	10 Cell Denver DR Recleaner Machine - 10 h.p. Motors per 2 Cells.
25.	Denver Automatic Concentrate Sampler - 1 cut per 20 minutes.
26.	5" x 5" SRL Final Concentrate Pumps (2) - 1 to Thickeners - one spare.
27.	10 Cell Denver DR Supercleaner Machine (with option to switch to Cleaner Machine) - 10 h.p. per 2 cells - 2 banks of 5 cells.
28.	50' Dorr-Oliver-Long Concentrate Thickener Concrete Tank.
29.	Dorr-Oliver-Long #4 ODS Pumps (2) - one spare.
30.	12' x 12' Concentrate Storage Tank - Denver Agitator.

CONCENTRATE (COPPER) LEGEND (cont'd)

31. Dorr-Oliver-Long #4 ODS Pumps (2) - one to each Filter.
32. Dorr- 8½' x 10-4 Disc Filters (2) - Differential Pick-up - 2 Nash Vacuum Pumps - one spare.
33. Concentrate Storage - 1,200 tons capacity.
34. 3 - Stage Automatic Tailings Sampler - Suction from Tailings by 2 x 2 Vac Seal Pump.
Collection Box - Denver Vezin Sampler - Denver Automatic Sampler.
35. 125' Dorr-Oliver-Long Tailings Thickeners (2) - concrete.
36. 10" x 8" SRL-C Tailing Booster Pump - Proposed - not installed.
37. Tailing to Tailing Pond 2 - 6000' Long, 12" Transite Lines.
38. 10" x 8" SRL Pump.
39. 6" Sala Pump.

Craigmont Mines Ltd.
MILL FLOWSHEET
August 1973





CONTROL SYSTEM IN GRINDING CIRCUIT

CRAIGMONT MINES
Fig. 2

MILLING OPERATIONS (cont'd)

METALLURGY

Operating Data for one grinding section is shown in Table VII.

TABLE VII - Size Analysis of Grinding Circuit Products - Cumulative Percent Passing Screen

Screen Size	Rod Mill Feed	Rod Mill Discharge	Ball Mill Feed	Ball Mill Discharge	Cyclone O'Flow
.525"	87.5				
.371"	70.1				
3 mesh	56.5				
4 mesh	47.5				
6 mesh	40.1	94.9	95.6	98.7	
8 mesh	34.6	84.3	90.0	97.2	
10 mesh	29.9	71.7	82.0	94.6	
14 mesh	26.2	60.7	74.1	90.9	
20 mesh	23.5	53.0	66.6	86.3	
28 mesh	21.1	46.3	57.2	79.3	
35 mesh	18.7	40.3	44.9	68.1	
48 mesh	16.9	36.3	36.0	58.3	91.9
65 mesh	15.1	32.1	27.1	47.4	82.6
100 mesh	13.5	28.6	20.5	38.3	72.9
150 mesh	12.0	25.1	15.2	30.5	63.0
200 mesh	10.5	22.2	12.1	25.2	54.5
270 mesh		20.4	10.7	22.6	50.1
325 mesh		18.6	9.5	19.9	44.9
400 mesh		17.6	8.9	18.6	42.1

The hardness of ore varies widely across the orebody. Consequently, there is a wide fluctuation in the mill throughput and on average, changes from 5,000 tpd at 10% iron in feed to 6,000 tpd at 20% iron in feed.

Several factors cause departures from the relationship between tonnage treated and iron content, and the figures quoted represent the median of a broad band of operating results, with a range of some 700 tons. Factors causing departure from the average condition include changes in the ratio of hematite to magnetite, variations in the size of rod mill feed and cyclone overflow and operating techniques on the cyclones. The Fe tonnage relationship has been used for budgeting future production with some success. The following is a recent forecast of tons milled per Fe content and is based on a 55% -200 mesh grind.

% Fe in Mill Feed	Tons Milled Per Calendar Day
10	5,000
12	5,300
14	5,500
16	5,800

CONTROL SYSTEMS

The original feed rate control is still in use. The feed rate of ore to the rod mills can be held at a predetermined level by a Foxboro Automatic Control System. An electri-

MILLING OPERATIONS (cont'd)

CONTROL SYSTEMS (cont'd)

cal signal, generated at the weightometer and directly proportional to the feed rate, enters a recorder controller on which the desired tonnage has been pre-set. Should the input signal not balance the controlling circuit, a correcting signal, which is converted to air pressure, simultaneously adjusts the variable speed drives on each of the three slot feeder belts via a diaphragm valve motor. Independent manual control of the proportion of the tonnage drawn from each of the feeders is also available.

Both grinding sections are on complete automatic tonnage and particle size control. An Autometrics PSM (particle size monitor) continuously monitors the particle size and percent solids of the cyclone overflow. A signal from the PSM system is compared with a pre-set particle size. Any deviation from the pre-set size results in water to the grinding sump (cylone feed) to be increased or decreased such that the desired particle size is maintained.

The feed tonnage to the rod mill is automatically regulated by monitoring the grinding sump level with a bubble tube system. Any deviation from a pre-set sump level with result in the feed tonnage to be increased or decreased. Thus, if the level drops below set point, the tonnage will be increased, and conversely if the level rises above set point, the tonnage will be decreased.

The particle size control loop can be run separate from the tonnage control loop by switching from automatic tonnage control to manual control. Thus, particle size can be automatically controlled to a desired grind, while tonnage is raised or lowered manually at the operator's discretion. Figure I shows a schematic of the control system.

Sonar sensors have been installed in the feed chutes of both rod mills. Should the chute plug and break the beam for more than two seconds, the sensor automatically shuts off all belts in that circuit and sounds on alarm.

FLOTATION

The cyclone overflow from each grinding section passes through a three stage automatic sampling system, and these samples provide the mill head samples. The cyclone overflows then pass through one of two 10 x 8 SRL-C pumps to the rougher feed distributor, where the cyclone overflow is combined with the scavenger middling return stream. The feed to the rougher flotation is split between the six rougher flotation machines, which consist of eighteen #48 Agitair cells. A single pass through these machines produces a final tailing which is sampled by an automatic cutter and then pumped to the iron media recovery plant.

The rougher distributors are sealed pressure units thus eliminating spillage. There are two units in parallel, each feeding all six rougher banks. One unit is kept as a spare.

Rougher and scavenger concentrates are produced on the rougher flotation machines. The concentrate launder system is so arranged that, by insertion of plates, the whole or part of the concentrate from the centre six-cell unit in each bank may be directed to either rougher or scavenger concentrate, as required.

MILLING OPERATIONS (cont'd)

FLOTATION (cont'd)

The combined rougher concentrates are pumped to the cleaner circuit which consists of four 5-cell units. The cleaner flotation mechanisms are Denver D-R. The cleaner tailing is combined with the scavenger concentrates and pumped to the head of the rougher circuit. The cleaner concentrate is pumped by a 6" Sala Pump to a D15B Krebs cyclone. The underflow from the cyclone is pumped by a 5 x 4 SRL pump to the regrind circuit, where it is combined with the regrind discharge. The combined pulp is pumped by a 6 x 6 SRL to a D10B cyclone in closed circuit with the mill. The overflow from the D10B and D15B cyclones is pumped by a 6 x 6 SRL to the recleaner circuit which consists of two 5-cell banks. The recleaner tailing joins the cleaner tailing and is returned to the head of the rougher circuit. The recleaner concentrate is pumped by a 6" Sala Pump to the supercleaner circuit which consists of two 5-cell banks. The supercleaner tailing is also returned to the head of the rougher circuit. The flotation mechanisms on the recleaner and supercleaner are Denver D-R. The supercleaner concentrate is sampled automatically and pumped to the dewatering circuit. The S'Cleaner bank is by-passed if grade permits.

REAGENTS

Reagent consumption is as follows:-

Lime	0.725 lb/ton
Aerofloat 242	0.003 lb/ton
Dowfroth 250	0.025 lb/ton
Z-200	0.010 lb/ton

The lime consumption is not governed by the flotation circuit, but by requirements for flocculation in the tailing thickeners. The reclaimed water from these thickeners returns to the mill water tank, and the lime addition used, maintains the rougher circuit pH at above 10.8 - 11.0 depending on the amount of pyrite in the mill feed. Automatic pH control is used to regulate the lime additions, part being added to the cyclone overflow pumpbox and parts to the tailings from the rougher flotation machines.

Quicklime is slaked and the slurry is stored in an agitator from which it is continuously circulated through a distribution loop. The lime slurry is delivered from the loop at the addition points by solenoid valves that activate red-jacket valves and is controlled either by manually ~~set~~ percent-o - cycle timers in the case of the thickener feed or by the pH meter for the flotation feed.

Aerofloat 242 is automatically mixed with water at a 10% solution. Feed points, controlled by solenoid valves, are to the cyclone overflow pumpbox and to the 7th and 13th cell of each rougher bank. Z-200 and Dowfroth 250 are added to the cyclone overflow pumpbox by Clarkson feeder. A Clarkson 12 point reagent distributor is used in feeding Dowfroth 250 to the 7th and 13th cells of the agitair banks.

METALLURGY

Since start-up the metallurgy of the Craigmont ore has progressively become more complicated. It appears that the deeper the orebody goes, the more finely disseminated the copper becomes and at the same time the pyrite content varies widely. The underground

METALLURGY

ore occurs as chalcopyrite in a non-sulphide matrix with small quantities of pyrite.

Hematite and magnetite are a major constituent mineral ranging from a combined assay of 5 - 40%. This oxide iron has a direct relationship on the hardness of the ore and thus the fineness of the grind. Current practice is to produce a grind of 58% -200 mesh.

Concentrate grade varies between 28 and 30% copper depending on the pyrite content of the feed and operating efficiency in the cleaning - regrind - recleaning circuit. A concentrate grind of 85 - 90% -200 mesh is desired along with a maxium moisture of 9.0% water.

Tables VIII and IX show metallurgical results for ore treated to July 1974.

TABLE VIII - Metallurgical Balance

Product	Tonnage	% Copper	% Recovery
Concentrate	52,145	28.64	96.8
Tailing	982,862	0.050	2.2
Feed	1,035,007	1.49	100.0

TABLE IX - Analysis of Products

		Concentrate			Tailing		
Tyler Mesh		% Wt.	% Cu.	% Dist	% Wt.	% Cu.	% Dist
	+35				0.5	0.113	0.96
-35	+48				2.9	0.132	6.90
-48	+65				6.6	0.158	18.65
-65	+100				11.8	0.119	24.97
-100	+150	4.2	18.17	2.66	10.4	0.055	10.21
-150	+200	9.4	24.38	8.00	9.8	0.029	5.05
-200	+270	10.6	27.18	10.05	5.7	0.023	2.34
-270	+325	7.6	27.43	7.31	7.1	0.021	2,64
-325		68.2	30.18	71.99	45.2	0.035	28.28
		100.0	28.61	100.00	100.0	0.056	100.00

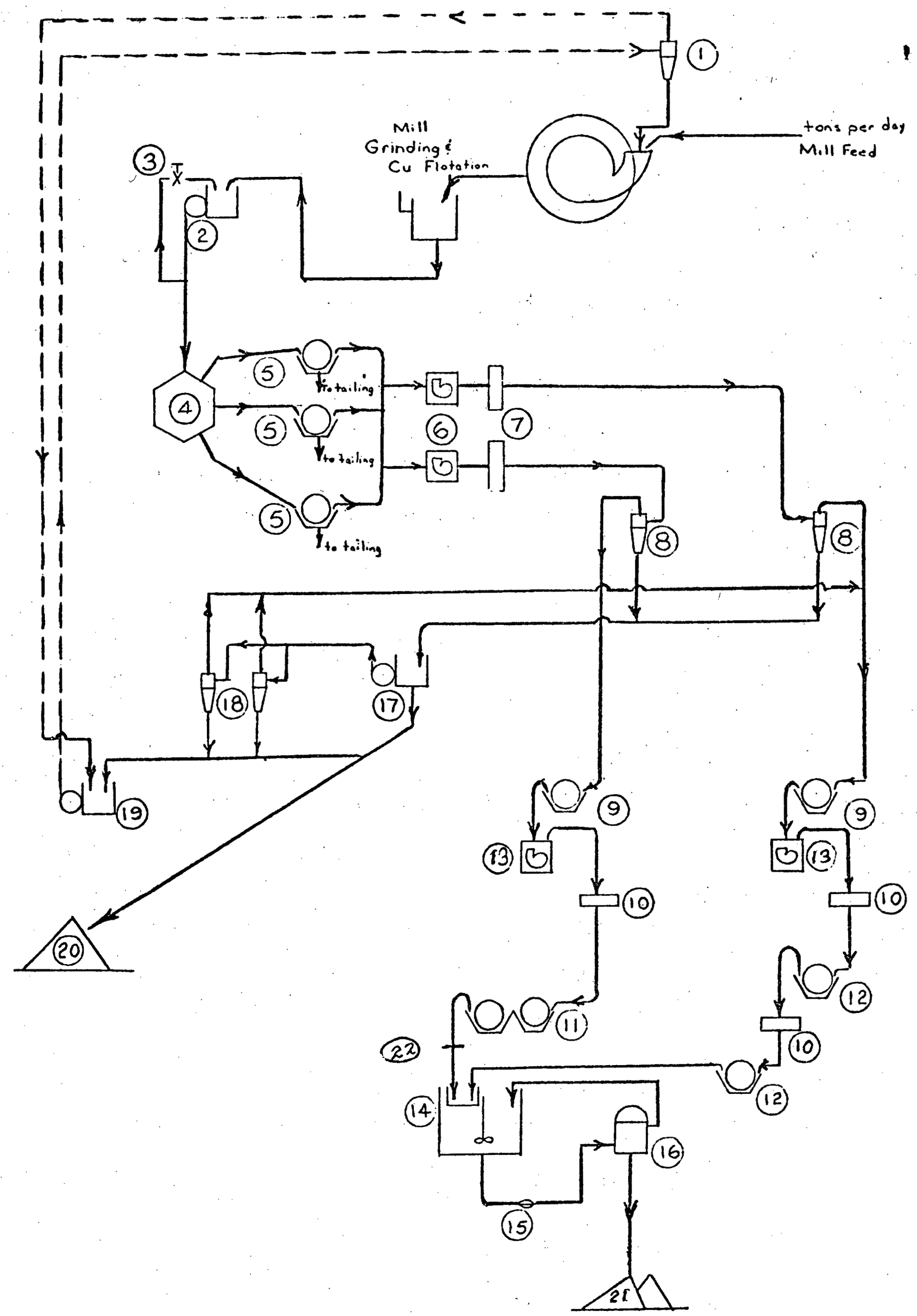
DEWATERING

Thickening to 60-65% solids must precede filtration to prevent settlement of sands in the filter boot and lines. The final concentrate is pumped by a 5 x 5 SRL pump to the 50' concentrate thickener. The thickener underflow is withdrawn continuously by one of two O.D.S. pumps. The pump can be speeded up or slowed down by regulating a timer switch and thus controlling the thickener underflow density. The thickener underflow feeds a 10' x 10' stock tank with agitator. The filters are fed from the stock tank by O.D.S. pumps, one to each filter. The filters are 8½' x 4 disc, American disc filters. Each filter is equipped with an agitator in the bath and a variable, low vacuum pick-up for control of the cake thickness, in addition to the variable speed drive.

IRON PLANT FLOWSHEET LEGEND

Ref.	Equipment
1.	Proposed Dewatering Cyclone.
2.	10" x 8" SRL Pump.
3.	Recirculating Valve.
4.	6-Way Pressure Distributor.
5.	36" x 72" Rougher Magnetic Separator (3)
6.	5" x 4" SRL Pump (2)
7.	Demagnetizing Coils (2)
8.	Krebs D10B Cyclones (2)
9.	Sala 14" x 67" Cleaner Magnetic Separators (2)
10.	Demagnetizing Coils (3)
11.	32" x 22" Recleaner - Dual Drum Magnetic Separator.
12.	36" x 24" Recleaner Magnetic Separators (2)
13.	3 x 3 SRL Pumps & Pump Sumps - with Probe Type Level Control (2)
14.	Tonnage Measuring Tank & 10' x 10' Denver Stock Tank.
15.	ODS Pump.
16.	8½' x 10 - 4 Disc Filter.
17.	3" x 3" SRL Pump.
18.	Krebs D6B Cyclones (2)
19.	3 x 3 SRL Pump.
20.	Coarse Magnetics Impound Area.
21.	300 Ton Cap. Iron Concentrate Loading Shed.
22.	Single Stage Denver Automatic Sampler.

Craigmont Mines Limited
IRON PLANT FLOWSHEET
August 1972



MILLING OPERATIONS (cont'd)

DEWATERING (cont'd)

The cake moistures obtained from the system average 8.5 to 9.5%. Aerodri 100 is used as a filter aid at a rate of 0.4 lb/ton of concentrate. The Aerodri 100 results in a moisture reduction of approximately 1% as well as a quicker releasing filter cake. Moisture is also regulated by adjusting the filter cake thickness. The optimum, about ½ in., is maintained by adjusting the density of the filter feed; i.e. if the cake is too thin, density is raised and if too thick, density is lowered.

Both lime and Alfloc 8863 are required for flocculation in the concentrate thickener. Laboratory testing has shown that both reagents are necessary. While lime alone produces a clear overflow, the flocs are fine and settlement is relatively slow; on the other hand, Alfloc 8863 alone produces large, fast settling flocs, but cannot produce a clear overflow. The same considerations apply to the flocculation of the mill tailings.

IRON MEDIA RECOVERY PLANT

The iron media recovery plant was started up in November 1969, to produce minus 325 mesh magnetite. The fine magnetite is being shipped to the coal companies in B.C. and Alberta for use in their heavy media separation plants.

The copper flotation tailings contain 10 to 20% magnetics which is equivalent to 3 to 8% available magnetics for recovery. Available magnetics is defined as the magnetic material in the minus 325 mesh fraction. Recovery of available magnetics is approximately 45%. Thus, iron media production ranges from 70 tpd at 10% magnetics to 180 tpd at 20% magnetics.

The copper flotation tailings is pumped by a 10 x 8 SRL-C pump to three 36" x 78" Sala drum rougher magnetic separators. The concentrate from the rougher separators is pumped by one of two 5 x 4 SRL pumps to one of two D10B Krebs cyclones. A demagnetizing coil is installed on the pump discharge line prior to the cyclones. The purpose of the coil is to demagnetize the magnetic particles so that more efficient cycloning can be achieved. The D10B cyclones have different size apexes. At low magnetics, and hence low tonnage, the cyclone with the smaller apex is used and at high tonnage the cyclone with the larger apex is used. A cyclone overflow density of 17-20% solids is desired to maintain a sizing of 90% -325 mesh. At very low tonnages the D10B cyclone underflow is recycloned in a 6" Krebs cyclone. The result is an additional one ton per hour of iron media produced. The cyclone overflow from the 10" and 6" cyclones is combined and cleaned on two 24" x 67" Sala drum cleaner magnetic separators. The cleaner concentrate is pumped by one of two 3 x 3 SRL pumps to two double drum recleaner separators. A demagnetizing coil is installed on the feed line to the recleaner separators to "defloc" the particles for easier recleaning. The recleaner concentrate flows to a 10' x 10' agitated stock tank at about 55% solids and containing 90% -325 mesh magnetics. The concentrate is pumped by an O.D.S. pump to an 8½ x 3 disc, American disc filter. The moisture content of the filter cake is 10%.

TABLE X - Metallurgical Data - Fe Plant

Tons Concentrate Per Month	3,800
Concentrate (-325 Mesh)	90.9
Concentrate (% Magnetics)	96.5
Recovery of avail magnetics	44.6

MILLING OPERATIONS (cont'd)

TAILING DISPOSAL

Mill tailing leaves the iron media recovery circuit at about 30% solids and is thickened to an average 50% solids in the two 125 foot thickeners. Thickener underflow is removed by spigotting through Gamma Gauge controlled pinch valves and flows by gravity to the tailing disposal area through about 1 mile of 12 in. Transite and Wood Stave line, having a slope of ½% between a series of pressurized drop boxes.

Tailing is retained behind a dam 4,000 ft. in length across the valley below the plant site. A total of some 230 acres is available above the dam for eventual tailing disposal. The dam is built by spigotting behind two-foot high wooden retention fences which are set back 2½ ft, with each lift.

Six decant towers were provided for water reclamation from the pond above the dam. Reclaimed water from the tailing pond is pumped to the mill head tank averaging about 500,000 U.S. gallons per day.

An automatic control system was installed to regulate the density to the underflow. Variations in the underflow density from a pre-set level are detected by a gamma gauge, from which a signal enters the control circuit and actuates pneumatically-controlled pinch valves in the underflow lines. Under varying load conditions the underflows are maintained at an optimum level for flow in the distribution line to the tailing dam, and maximum recovery of water is obtained in the thickener overflow.

Some 55-60% of the water used in the mill is recovered by the tailing thickeners and the need to maximize this recovery is assuming greater importance as the total useage on the property increases with the development of underground operations.

WATER TREATMENT

The river water supplied to the property has a total hardness of 100-150 ppm largely, as calcium and magnesium bicarbonates. The reaction of these bicarbonates with the lime content of the reclaimed water produces an excessive scale in the mill water system unless chemical stabilization of the precipitating calcium is achieved. The condition also appears to be aggravated by the presence of fine ore floccules in the reclaimed water.

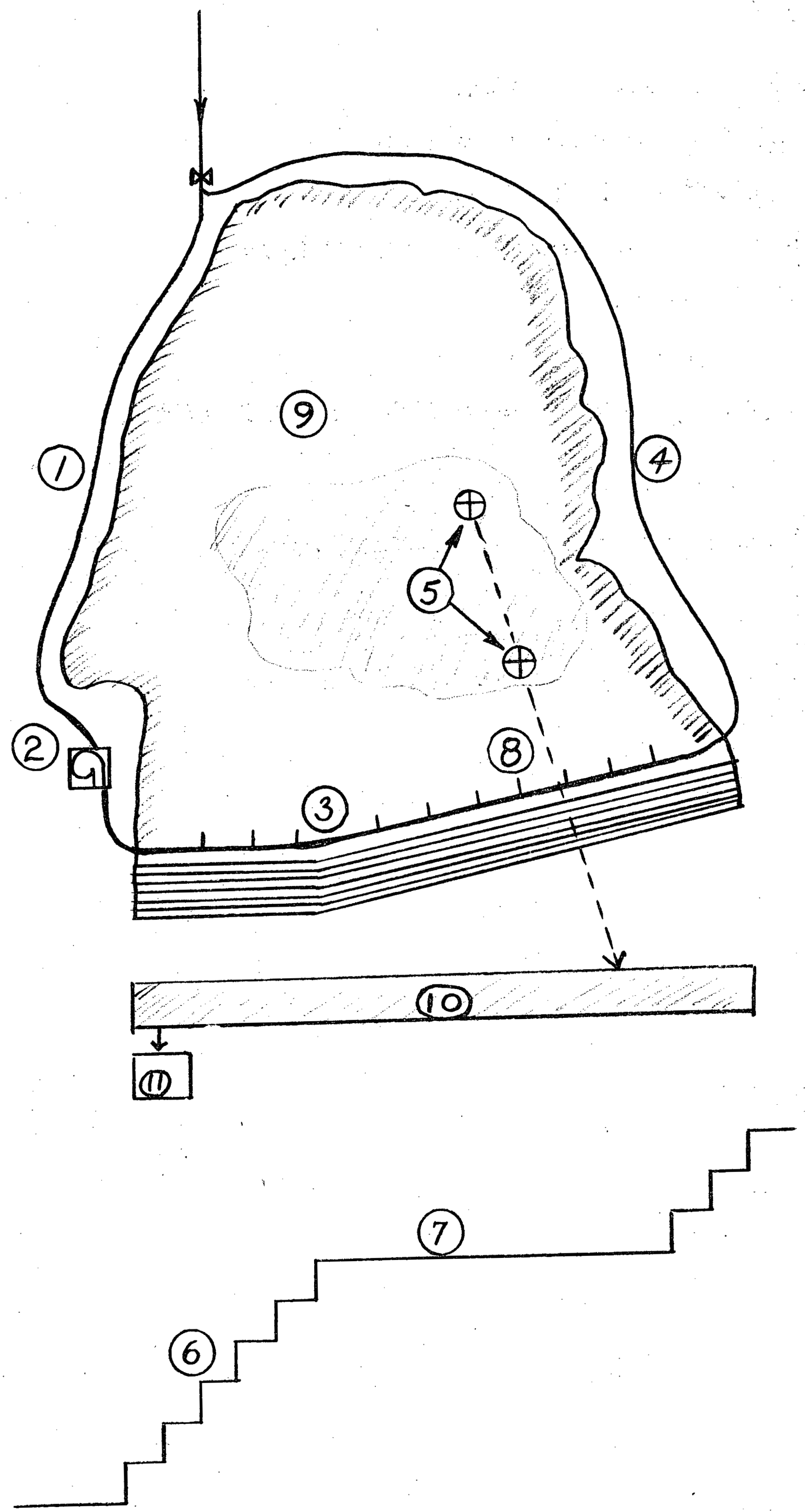
A sodium polyphosphate additive has been used successfully since August 1962, and since that date no delays have occurred in production due to scaling of the mill water pipes. Additions of polyphosphate are made to the river water at the Nicola River pump and the overflow water from both tailing thickeners. Approximately 30 lb. is used per day, or .008 lb/ton of ore year to date, to maintain a level of 1-2 ppm polyphosphate in the mill water circuit.

Scale in the water pipes has thus been kept to a minumum, and regular checks are made of the polyphosphate level in the water circuit. Should the polyphosphate level drop, due to improper addition or a seasonal change in the hardness of the water, scale again starts to build up. Higher than normal additions have a tendency to dissolve or loosen existing scale.

TAILING IMPOUNDMENT LEGEND

1. West Line - 6,000 ft. of 12 in. transite.
2. Proposed 10 x 8 SRL-C Booster Pump (if required).
3. Dam wall line - 4,000 ft. of 10 in. Transite.
4. East Line - 6,000 ft. of 12 in. Wood Stave pipe and Transite.
5. Decantation Towers - concrete.
6. Wooden Fences - 2 ft. in and 2 ft. up.
7. 18 - 20 ft. road allowance every 12 vertical ft. Present height of dam is 72 ft.
8. 20 - 30 1¼" Rubber Spigot Hoses.
9. Tailings - total area presently 230 acres containing about 20,000,000 tons of tailing solids.
10. Collection Ditch - 4,000 ft. long.
11. Pump House - 500 U.S. Gallons per minute to Mill Head Tank.

TAILING IMPOUNDMENT



MILL SUPERVISION AND OPERATING CREW

Mill Superintendent	1
Mill	
- Foreman	1
- Shifters	4
- Grinding Operators	4
- Flotation Operators	4
- Filtration and Thickening Operators	4
- Mill Helper and Buckers	9
- Iron Plant Operators	1
Crushing & Conveying	
- Foreman	1
- Primary Crusher Operators	5
- Cable Belt Operators	6
- Secondary Crusher Operators	8
- Helpers	2
Tailing Dam	
- Operators (seasonal)	4
Research & Assay	
- Mill Metallurgist	1
- Chief Assayer, Senior Assayer, Assayers (2)	4
- Research Metallurgist	1
- Laboratory Technicians and Assayers	4
Routine Maintenance	
- Mechanics	5
- Oiler and Spare Mechanic	1

Water Supply

Fresh Water Supply - Nicola River, approximately four miles from property. Water is pumped in two stages of pumping, with total static head of 800 feet. All water is chlorinated, and PO_4 added to prevent lime scaling.

No.1 Pump House - Located at Nicola River at elevation 1840 feet. Storage tank capacity 38,000 gallons. Pumps: 3 x 75 H.P. Pamona Fairbanks-Morse delivering 540 gallons per minute at 450 foot head each.

No.2 Pump House - Located 2.5 miles from Nicola River at elevation 2230 feet. Storage tank capacity 20,000 gallons. Pumps: 3 x 75 H.P. Pamona Fairbanks-Morse delivering 540 gallons per minute at 450 foot head each.

Main Storage Tank - On property at elevation 2620 feet. Construction - Wood Stave. Capacity: 250,000 gallons, 200,000 gallons for fire hydrant line, 50,000 gallons for fresh water supply.

Mill Storage Tank - At elevation 2460 feet. Fed from Main Storage Tank, Tailings Pond Reclaim water, Tailings Thickener Reclaim water, Compressor Cooling water. Construction - Wood Stave. Capacity: 65,000 gallons.

Main Water Lines are 12" diameter cast iron pipe.

Fresh water is sampled weekly throughout the property to ensure it meets Department of Health standards for drinking water.

Plant Electrical Statistics

Hydro Primary Supply Voltage - 60,000 volts.

Main Substation Capacity - 12.8 MVA

Primary Distribution Voltage - 4,160 volts.

Secondary Distribution Voltage - 550 volts.

Total Connected Horsepower - 17,454 H.P.

Maximum Peak Power Demand - 9,800 H.P.

Average Monthly Consumption - 5,500,000 Kilowatt Hours

Average Power Factor - 99%.

Average Load Factor - 86%.

Average Monthly Power Cost - $37,000.00.

Emergency Generator - 1250 KVA (Cooper-Bessemer)

Most motors over 100 H.P. operated on 4160 volts, 3 phase.

Smaller motors operate on 550 volts, 3 phase.

Domestic power operate on 110 volts, single phase.

PLANT SERVICES (cont'd)

Natural Gas

Natural Gas is supplied by the Inland Natural Gas Company from its main trunk line which runs 2.5 miles easterly from the property.

Supply Lines - Length	-	2.5 miles
- Size	-	3" inside diameter
- Pressure	-	760 pounds per sq. in.

In the Natural Gas Receiving Station, the gas pressure is reduced to 60 p.s.i. for distribution around the property, i.e. directly to the 3700 level Open Pit Shop, the Central Heating Plant, and Mine Air Heaters.

Compressed Air

In Surface Powerhouse - 2 CIR XVHE - 2 @ 3,300 c.f.m.
- 2 CIR XVHB - 2 @ 1,100 c.f.m.

On 2400 Level Underground - 1 Belliss-Morcom WH200 @ 2,000 c.f.m.

In Concentrator (stand-by) - 1 CIR @ 375 c.f.m.

At present there are four XVII Ingersoll-Rand Compressors at the 2400 Level Powerhouse; two 1,100 c.f.m. and two 3,300 c.f.m.; supplying compressed air for the underground mining operation, and one 375 c.f.m. Ingersoll-Rand electric powered stand-by compressor at the mill. In addition the 2400 level underground compressor station houses a 2,000 c.f.m. WH200 Belliss & Morcom Air Compressor, supplying air to underground services.

Heating

Plant Buildings are mainly hot water heated from a 62 H.P. Cleaver Brooks Boiler at a centrally located building, adjacent the warehouse. Auxiliary heat is provided by radiant and hot air exchange units. Except for a minor amount of propane, all fuel is Natural Gas.

2400 Level Service Buildings

All buildings are Butler prefabs, concrete foundations, steel frame, metal clad with automatic sprinkler systems installed, insulated with limpet asbestos.

Main Office (Air Conditioned)	7,600 square feet floor area
Dry Change House & First Aid Station	12,200 " " " "
Warehouse	6,000 " " " "
Carpenter Shop	2,000 " " " "
Service Shops (Machine & Electric)	14,400 " " " "
Open Pit Shop	11,880 " " " "
Powerhouse	5,300 " " " "

PLANT SERVICES (cont'd)

2400 Level Service Buildings (cont'd)

Assay & Metallurgical Research Building	4,000 Square feet of floor area
Concentrate Storage Slab	12,500 (Capacity 4,000 tons of concentrate)

2400 Explosive Magazines

Cap House	300 square feet of floor area
Explosives	950 " " " " "

Communications

Property Telephones	50 line automatic dial system
Outside Telephones	2 lines
Teletype	From property to Vancouver
T.W.X.	From property to Computer in Calgary
Portable Radio	13 sets in Surface Mobile Equipment 11 sets in Underground

SAFETY & FIRST AID

Full time Safety Supervisor, responsible for all safety, first aid and fire prevention.

First Aid Attendants - At least one fully certified first aid man on property at all times.

First Aid Stations - Standard mine first aid room.

Ambulance - Thames panel - on property at all times.

Fire Prevention

All buildings on the 2400 level are automatic sprinkler installed. Fire hydrants are in close proximity to all buildings.

LABOUR

All labour is hired on the property where a personnel officer is on duty during normal working hours.

The services of the Canada Manpower are available as may be required at times to fill vacancies for tradesmen.

CREW STRENGTH (at 31 May, 1974)

	Crew	Staff	Total Employees
Underground	182	24	206
Mill	48	14	62
Engineering		16	16
Plant Department	117	17	134
Accounting	10	14	24
Safety	5	5	10
Administration & Management		5	5
	362	95	457

The operation is on a continuous 6:2 schedule.

LAND RECLAMATION

Topography, Climate & Vegetation

Acreages of land occupied and disturbed by the mining operation are as follows:-

Open Pit	68
Waste Dumps	546
Plant Areas	131
Tailings Pond	282
Total	1,027

The topography of the local area is generally hilly, with elevation varying between 2000 and 6000 ft. above sea level. The hills are well glaciated and interspersed with wide valleys and numerous lakes. Ground cover is glacial till comprised mainly of sand, gravel, and boulders, covered in places with top-soil.

The climate is semi-arid. The winters are fairly cold with the average minimum temp. being 10 degrees F, while snowfall is approximately 50"/year. Summers are generally hot and dry as indicated by an average maximum temp. of 80 degrees F. Rainfall is approximately 8"/year giving a total precipitation of 13"/year.

The low amount of precipitation dictates the type and degree of vegetation in the area. Uncultivated areas have a sparse covering of various grasses, which turn brown and become dormant during the summer months. Grasses to be found in the area include the following: wild and foxtail barley, various wheatgrasses, pine grass, alfalfa, clovers, red top, couch grass, timothy, orchard grass, brome, downy brome, Kentucky blue and fescue. Drought resistant shrubs and flowers in the area include sainfoin, yarrow, dandelion, fireweed, lupines, mustard weed, and several varieties of thistle.

Forest growth is not considered dense with the major species being Douglas Fir, Spruce, Ponderosa Pine, Jack Pine, Yellow Pine, and Aspen.

Reclamation Programme

A two year programme was started in October 1969 to study the growth of vegetation on the tailings dam and on the waste dumps in the pit area. Local climate and soil conditions indicated a very drought resistant and alkaline tolerant seed mixture was necessary.

The initial plan called for seeding and fertilizing of the face of the tailings dam (14 acres) and 207 acres of pit waste dumps.

Programme Highlights

October 1969:

Soil sampling of the tailings dam indicated an alkaline soil of pH 8.4 which was extremely deficient in plant nutrients of N, P and K.

14 acres of the tailings dam and 207 acres of waste dumps were seeded by aerial spraying at the rate of 76.6 lbs/acre. Fertilizer spread over the same area was a 10-30-10 variety at 289 lbs/acre. The seed mixture consisted of the following:-

LAND RECLAMATION (cont'd)

Programme Highlights (cont'd)

October 1969:

Annual Ryegrass	15%
Boreal Fescue	20%
Crested Wheatgrass	20%
Streambank Wheatgrass	8%
Slender Wheatgrass	10%
Pubescent Wheatgrass	7%
White Clover - double - inoculated	8%
Rhizoma Alfalfa - double - inoculated	12%

There was no ground preparation prior to seeding and fertilizing. After seeding, the 14 acres of the tailings dam was raked. There was no raking of the waste dumps.

Due to the lateness of the season and lack of rainfall, the seed did not germinate until the following spring.

April 1970:

In April 1970 the entire area was re-fertilized by aerial spraying, this time using the 20-20-10 variety of fertilizer at 200 lbs/acre. The upper benches of the tailings dam had only 5% growth while the lower drier benches had no growth at all. Watering of the tailings dam by truck was attempted and abandoned as unsuccessful.

Germination on the waste dumps covered only 4-5% by area and grew to a height of several inches. Rainfall for May, June, July and August was only 1.10 inches as compared with 5.32 inches for the same period in 1969. It was assumed that the hot temperatures and lack of moisture killed a good portion of the grass.

Summer 1970 - Spring 1971:

In August 1970 the 14 acres of the tailings dam and 24 acres of waste dumps were re-fertilized with 10-30-10 at 211 lbs/acre. The tailings dam was also re-seeded at 64 lbs/acre. Seed mixture was identical to the first. A cool, wet 1971 spring produced a favourable growth on both the dam and waste dumps. It became evident that in order to grow vegetation on the tailings dam, irrigation would be required.

Summer 1971

In July 1971 an irrigation system using reclaim water was installed on the tailings dam covering approx. 1/3 of the 14 acres. The system was used regularly during the summer months. Growth responded immediately and became fairly heavy. After approx. 2 years of fertilizing and seeding the following species of grasses and plants were identified on the dam: - Rye grass, streambank wheatgrass, crested wheatgrass, pubescent wheatgrass, creeping red fescue, downy brome, alfalfa, mulleins, sweet clover, and Russian thistle. It was noted that the white clover legume in the first two mixtures failed to germinate.

In September 1971 the entire area was re-fertilized at 200 lbs/acre with a 13-16-10 fertilizer. The tailings dam was re-seeded at 36 lbs/acre with the following mixture:-

LAND RECLAMATION (cont'd)

Programme Highlights (cont'd)

Summer 1971:

Creeping Fescue	19%
Annual Rye	16%
Crested Wheatgrass	33%
White Clover, double inoculated	16%
Rhizoma Alfalfa, double inoculated	16%

Costs

Costs of the two year programme:-

Seed	$ 8,634	)
Fertilizer	6,423	)
Application by Plane	4,512	) - Cost/Acre $136
Freight	1,789	)
Labour	1,980	)
Irrigation System	6,818	)
Total	$ 30,156	

Fall 1972

In September 1972 an additional 100 acres of the pit waste dumps were fertilized and seeded. A 19-19-19 fertilizer was spread at a rate of 200 lbs/acre and a seed mixture spread at a rate of 70 lbs/acre. The seed mixture consisted of the following:

Tetraploid Perenial Rye	5%
Annual Rye	5%
Perenial Rye	3%
Creeping Red Fescue	15%
Crested Wheatgrass	10%
Pubescent Wheatgrass	15%
Inter. Wheatgrass	5%
Tall Wheatgrass	15%
Sainfoin	5%
Trefoil	7%
Rhizoma Alfalfa	15%

No other cultivation was carried out. The tailings dam irrigation system was used only intermittently during the summer because of an above average rainfall.

By this time it was apparent that the natural re-generative process was well established on both the tailings dam and the pit waste dumps.

The cost of the 1972 programme was $58/acre.

1973:

There was no seeding or fertilizing in 1973. An additional 3,000 ft. of irrigation pipe was installed on the tailings dam to increase the sprinkling coverage. The 14 acres of seeded area were sprinkled on a regular basis during the summer.

LAND RECLAMATION (cont'd)

Programme Highlights (cont'd)

1973:

Cost of this additional equipment including labour for installation and operation was $4,367.00 which is $312.00 per acre for the tailings dam.

Future Programme:

During 1974 our efforts will continue to be directed at maintaining the existing growth on the waste dumps and tailings dam. Some re-seeding or re-fertilizing may be carried out if required. The irrigation of the tailings dam will be continued as necessary.

In the more distant future as we retreat from parts of the property, it is likely that our reclamation programme will be expanded to include these areas.