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SUBLEVEL CAVING AT CRAIGMONT

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INTRODUCTION

In 1957, diamond drilling on a magnetic anomaly indicated an extensive zone of copper mineralization on what is now the Craigmont Mines property. By mid 1958 further drilling established a copper orebody.

Milling commenced in September 1961 at 5,000 tons per day and to the end of February 1970, 14,457,000 tons of ore grading 1.485% copper were processed to yield 730,700 dry tons of concentrate containing 408,482,000 pounds of copper. At present 60% of the mill feed is derived from underground operations and 40% from low grade surface stockpiles.

Craigmont Mines is situated 130 air miles northeast of Vancouver, in the Highland Valley, 10 miles west of the Town of Merritt, a logging, ranching and mining community of about 7,000 people. It is serviced by paved highways, Canadian Pacific Railway, B.C. Hydro and Inland Natural Gas Company. Water is pumped from the Nicola River, a distance of 4 miles and a lift of 800 feet.

In March 1967 the open pit mining operations at Craigmont Mines Limited reached their economic limit and were suspended. Before this it had been decided that a sublevel caving method of underground mining would be used to supply ore to the concentrator after the cessation of open pit production. This paper describes the factors influencing the choice of mining method, some of the problems encountered, current practices and results.

GEOLOGY

The upper Triassic Nicola group is a thick volcanic and sedimentary series of agglomerate, breccia, flows, limestones, argillites, and greywackes. These have been intruded by bodies of granodiorite, quartz diorite and diorite. The largest of these is the Guichon Batholith. Many of the sediments adjacent to the batholith have been hornfelsed.

The fundamental mine structure appears to be a series of drag folds on the north limb of a large anticlinal fold. Where this structure lies within the thermal aureole of the Guichon Batholith calc-silicate skarn alteration is developed and is closely associated with pyrometasomatic copper iron replacement orebodies.

The orebodies are deeply dipping and bordered by greywacke type rocks.

The area has been subjected to considerable faulting and brecciation. This is a major factor in the mining operation.

The chalcopyrite magnetite - specularite orebodies are relatively narrow with a maximum width of 150 feet with a combined strike length of approximately 2,800 feet and a vertical extent of 2,000 feet.

Geological ore reserves at February 1970 were 17,355,000 tons at 1.76% copper of which 14,877,000 tons at 1.95% copper were in situ, and 2,478,000 tons at 0.59% copper were broken reserves in surface stockpiles.

Ground Conditions

The waste rocks, greywacke, andesites and diorite, are relatively incompetent due to the high degree of fracturing and jointing, and all require varying degrees of support.

The ore zones are somewhat less fractured; ground support is still required however, although to a lesser extent than in the country rock. Ground conditions in the main orebody are better than in the smaller, narrower orebodies.

Clayey fault gouge is present in most of the faults; gouge zones may be up to 20 or 30 feet wide. The main ground problems are associated with local weakness rather than pressure.

Shape of Orebodies (See figure 1)

The main No. 1 orebody is approximately 800 feet long and 150 feet wide. It extends vertically from the original top of the open pit at 4200 elevation to just below the 3060 level.

The No. 2 orebody is approximately 1,000 feet long and 150 feet wide and extends from 3060 level to 2400 level.

Both these orebodies have extensions resulting in additional small irregular bodies.

Orebodies are mostly steep dipping, though part of the Wing Orebody, an extension of No. 2 orebody, dips at 50°. This orebody varies in size, but is approximately 400 feet long, 50 feet wide, and about 700 feet high.

MINING PRACTICE UP TO 1967

Open Pit

The pit produced ore from March 1961 to March 1967, 8,198,000 tons at 1.66% copper, and 5,650,000 tons at 0.67% copper. Total material moved including waste and overburden was 87,143,000 tons, which gave a waste to ore ratio of 6.29 to 1. When open pit mining

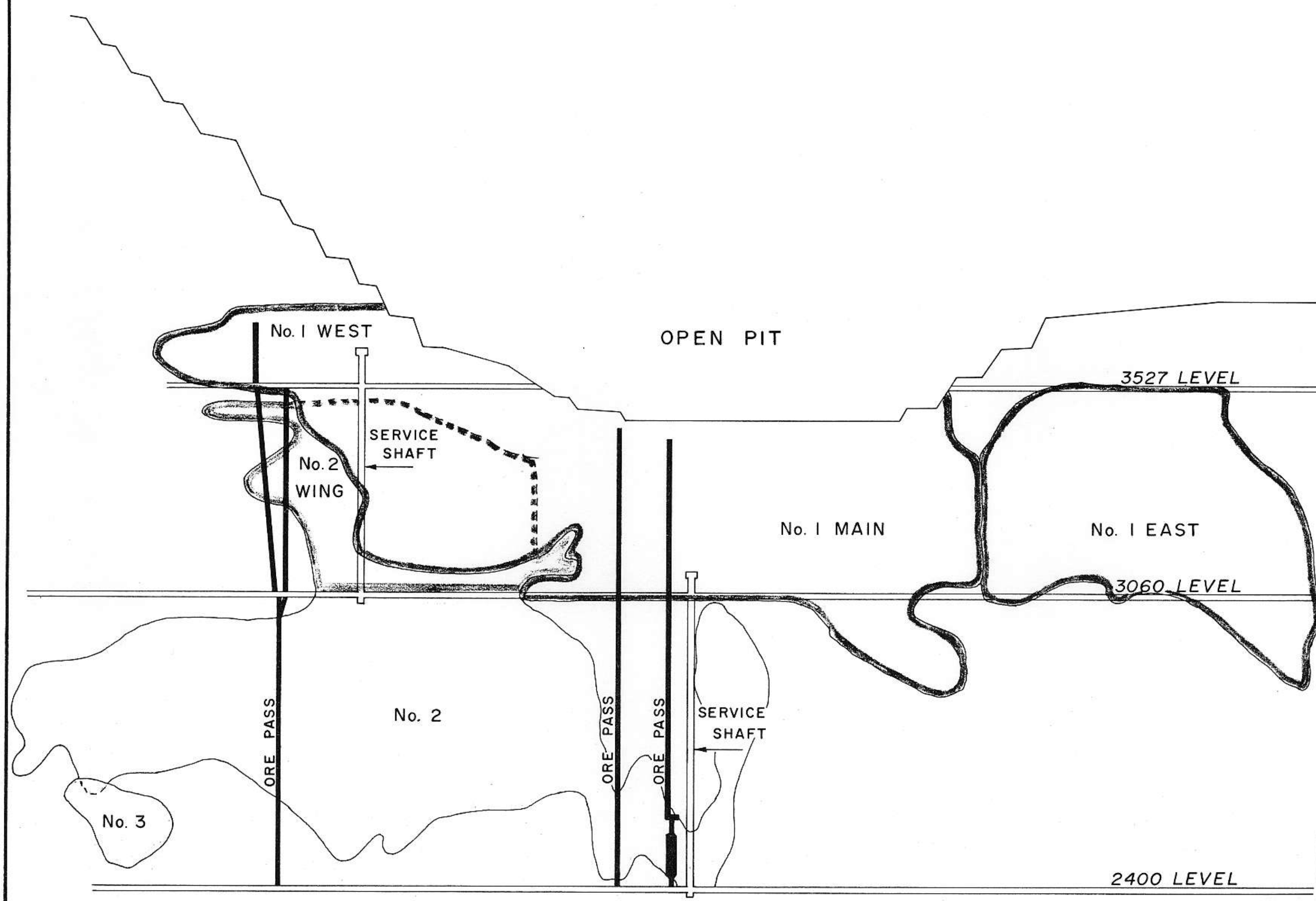


Fig. I

CRAIGMONT MINES LIMITED MERRITT, B.C.	CRAIGMONT OREBODIES

ceased, several million tons of low grade ore remained on surface stockpiles, permitting continuous operation of the concentrator until the underground mine reached design capacity.

Underground

While the open pit was operating, exploration and development work continued underground.

In anticipation of ground problems, it was decided to commence underground work before completion of the pit, partly to gain information on the orebody, and partly to supplement mill feed with higher grade ore. No mining could be done near the open pit, and no method could be used which permitted subsidence in the pit.

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.

Transverse cut-and-fill mining was used in the No. 2 orebody and in the North Limb orebody, with varying degrees of success. Even though stoping width was limited to only 25 feet, ground support was a major problem.

About 730,000 tons were mined underground by the above methods. In the fall of 1965 the decision was made to adopt the sublevel caving system of mining as soon as possible after the cessation of open pit mining.

REASONS FOR THE CHOICE OF SUBLEVEL CAVING

Experience gained to 1965, from both open pit and underground extraction, gave engineer and operator valuable knowledge of the physical characteristics of the Craigmont ore and the adjacent ground. These characteristics, and the probable extraction effects on adjacent ground were the major factors in determining the underground mining method to be used. As previously mentioned, variations of blasthole and cut-and-fill stoping were tried in initial underground operations with limited success. Mining efficiency was low and costs were high. Some stoping areas gave excessive dilution and sometimes hazardous conditions due to the weak character of both ore and wall rock. Due to the physieal weakness of part of the ore deposit and the lack of competent ground adjacent to the ore, it was logical to turn towards caving methods of ore extraction where the tendency of ground to cave could be utilized instead of resisted.

The Craigmont orebody of contact metamorphic origin is relatively small and contains many irregular, narrow oreshots surrounded by weak, readily cavable waste rock. Because of the small size and irregular shape of most of the Craigmont ore deposit, any type of a block or panel caving system would lack the selectivity required to obtain satisfactory ore recovery without excessive dilution. In addition, high development costs plus poor fragmentation from the harder, tougher parts of the ore deposit ruled out extraction by this type of cave mining.

On examination, a sublevel caving method of mining offered certain advantages for the extraction of the Craigmont ores:-

1. It allowed selective mining of a small, irregular orebody containing ore of both weak and strong physical characteristics, adjacent to extremely weak, fractured, cavable country rock on all sides. It is quite flexible and offers variations that enable it to be adapted to small

irregular ore lenses or large massive deposits.

2. It was a method that took advantage of the caving tendency of the wall rocks. Resulting surface subsidence would be acceptable as open pit operations would be terminated by the time underground mining operations were started.

3. The method did not require large stope openings, but only extraction openings of drift width through the ore and adjacent wall rocks, the size of which would be a compromise between ground stability and productivity of the equipment used.

4. Sublevel caving allows the continuous mining of an ore deposit of high vertical dimension, without tying up a substantial proportion of its ore in pillars. Sublevel caving does not require pillars. Mining starts at the top of the orebody and progresses down, consuming the development openings as it progresses. The method does not require perpetual retention of development openings once the ore on that horizon has been mined.

5. Sublevel caving allows relatively efficient underground production and is well adapted for mechanization with highly productive equipment, the size of which is only limited by the dimensions of the extraction drifts. It allows the use of equipment similar to that used in efficient room and pillar methods of mining.

6. Mining costs and efficiencies, although not as good as Block Caving, large scale Room and Pillar, or Blasthole operations, are superior to other methods of extraction where there is a problem in ground support.

The major disadvantage of sublevel caving is the high dilution which is inherent in the method, and is the price you pay for using it. The percentage of ore recovery depends on the amount of dilution tolerated in the extraction process. The higher the dilution allowed the better the ore recovery, or the lower the amount of dilution tolerated the lower the ore recovery; these factors in turn are dependent on the cut-off grade, i.e. the predetermined grade at which ore draw is to cease. Higher grade ores can stand more dilution in approaching the draw cut-off grade and hence will obtain a better recovery of the mineable ore. The best practice today in Zambia and Sweden approaches an average minimum dilution of from 15% to 20% with an 85% to 90% recovery of the mineable ore. When you consider that no major pillars of ore have to be left in an ore deposit extracted by this method, a total ore recovery of 90% of mineable reserves compares very favourably to the recovery achieved by the best of other mining methods.

In Craigmont's case sublevel caving was the most economic method of ore extraction per pound of copper produced. Due to adverse ground conditions it was the only method that could be used at all, without resorting to a type of square-set or undercut-and-fill method of mining with their high operating costs and low productivity.

PLANNING FOR SUBLEVEL CAVING

In the fall of 1965, it was decided to prepare the mine for sublevel caving. Swedish methods had been studied that summer, and mine planning started.

The first major decision to be made was to choose between operating small low-productivity air-powered equipment in small headings, or large high-productivity diesel-powered equipment in large drifts.

Because of poor ground conditions it was originally planned to keep drifts narrow (about 8 feet by 9 feet high) and use air-powered autoloaders and air shuttle cars delivering to closely spaced orepasses. This was the Swedish practice in mines of similar size with comparable ground conditions.

There was concern over the following points:-

1. Because of short haul range, many orepasses at close intervals would be required. Raw orepasses at Craigmont had given a very short life - after passing about 200,000 tons they would cave out, and have to be abandoned.

2. A good many headings would be required to obtain the initial tonnage target of 3,000 tons per day.

3. Opportunities for a high degree of mechanization were limited.

In March 1965, experiments had begun with the use of shotcrete as a means of ground support in lieu of timber. Preliminary tests indicated that shotcrete would adequately support a 12 foot by 13 foot heading, considered to be the minimum size for use of high productive diesel loaders and 36 inch ventilation duct.

Diesel-powered equipment has a greater haul range, and would require fewer orepasses than short-haul equipment. However, the orepasses would require a sturdy lining to stand up to the greater tonnage. In spite of this, economic studies favoured the larger, more productive diesel equipment, where the orebodies were readily accessible to a ramp system, service shaft, ore and waste passes. At first it was decided to use diesel equipment only in the larger and more competent Main Orebody, and small air equipment in the narrow orebodies; finally in the interests of uniformity, it was felt that diesels should be used throughout.

Some of the relative pros and cons are as follows:-

	Diesel (Large Units)	Air (Small Units)
Productivity	High	Low
Automation	High	Low
Maintenance Costs	High	Low
Number of Orepasses Required	Minimum	Maximum
Ground Control (Drifting, and Production Brows)	Maximum Required	Minimum Required
Ventilation	Maximum Required	Minimum Required

To sum up, although it was known that ground control and ventilation would be a problem with the larger diesel equipment, the reduction in orepasses, the higher potential productivity and the proposed use of shotcrete for ground support favoured the selection of the high productive diesel equipment.

Smaller ore blocks requiring considerable vertical access with short hauls to orepasses, were to be mined with the smaller air equipment.

While this planning was proceeding the blasthole and cut-and-fill stopes were phased out.

Parameters

Scale model tests were carried out using crushed ore and multiple drifts to determine optimum development patterns. It was understood that model tests could only give indications and could not be relied upon for statistical results. As a result of the tests and guided by judgement and experience of the operators in regard to ore fragmentation, recovery, dilution and ground support, and consideration of practices in Sweden and other parts of the world, it was decided to use:-

1. Vertical heights of 31 feet between sublevels.
2. Production drifts 13 feet wide x 10½ feet high.
3. Production drifts on 37 foot horizontal centres (to give 25 foot pillars).
4. 74° side holes.

The 31 feet was changed slightly to conform with existing workings. The original choice for pillar width was 20 feet, but was modified to 25 feet because of ground conditions. (See figure 2 and 3). At this time Swedish practice was increasing the sublevel interval to 40 or 45 feet. At Craigmont, difficulty with fragmentation was anticipated at the greater heights. Swedish practice has since returned to about 10 meters. Good fragmentation and an even ore flow when loading in the extraction drifts is of vital importance to optimize recoveries and keep dilution to minimum levels.

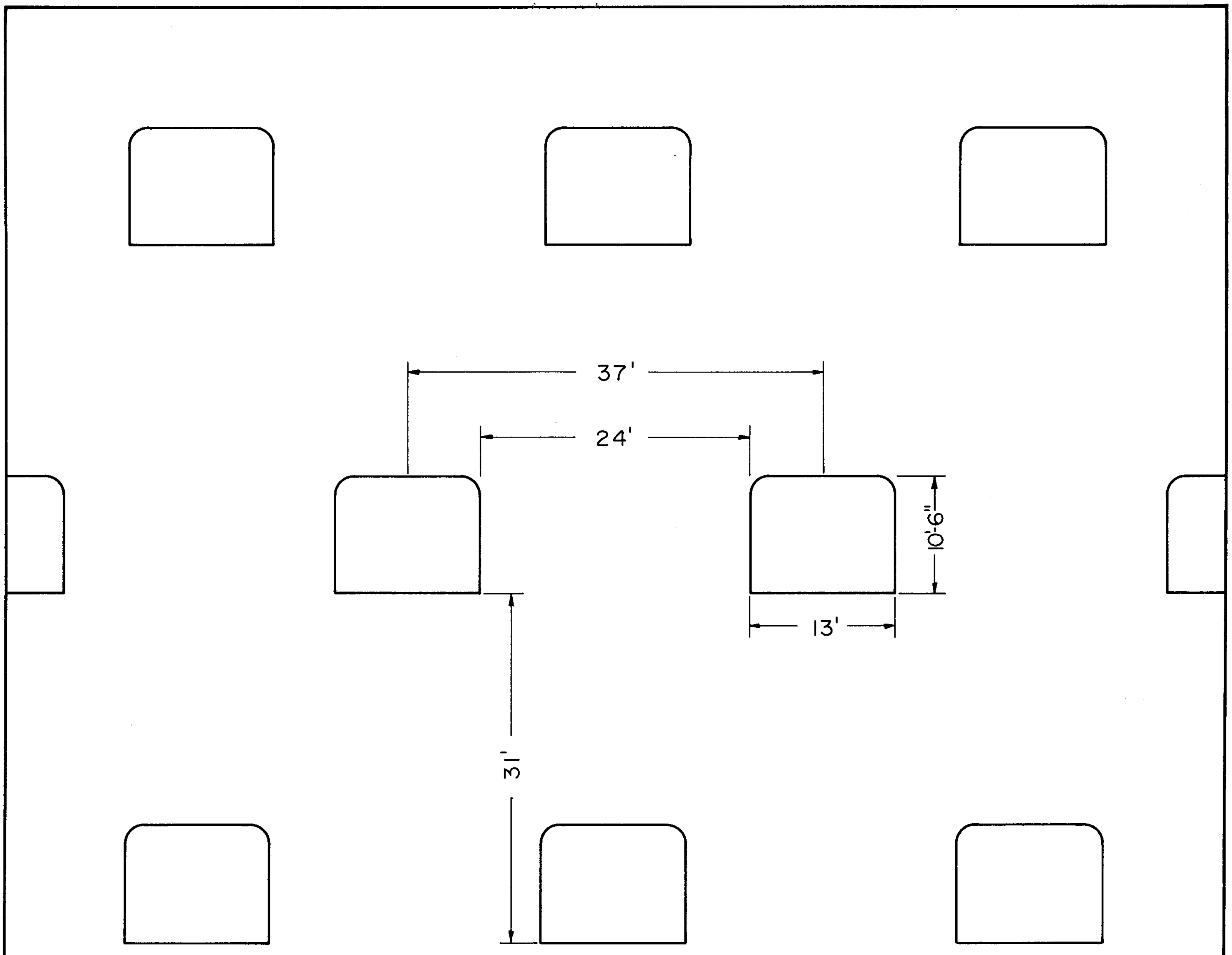


Fig. 2

CRAIGMONT MINES LIMITED MERRITT, B.C.	DRIFT SPACING FOR TRANSVERSE SUBLEVEL CAVING

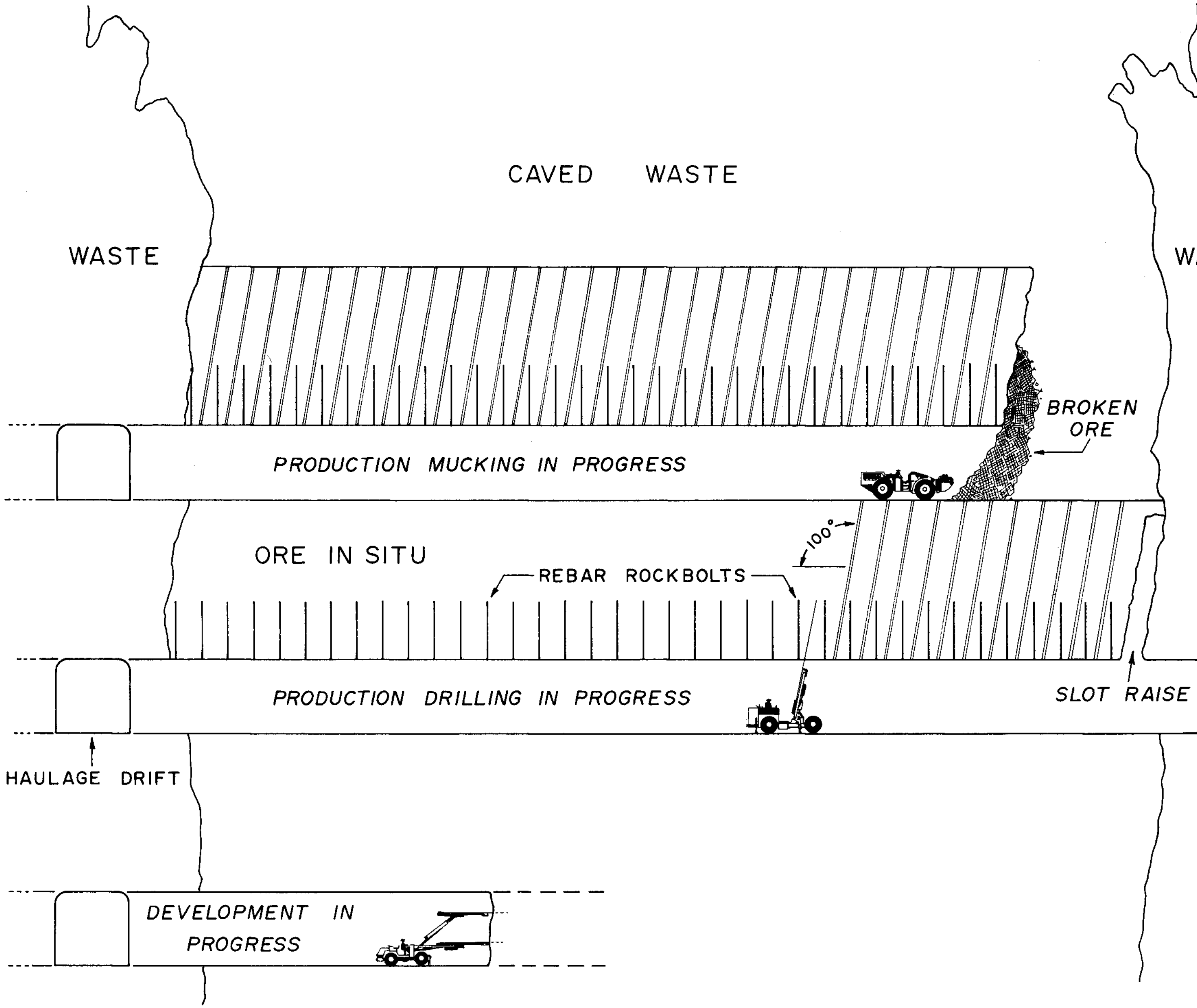


Fig. 3

CRAIGMONT MINES LIMITED MERRITT, B.C.	TRANSVERSE SUBLEVEL CAVING

Drift Layouts

It was tempting to place as much of the workings in ore as possible as this is in somewhat better ground, reduces development and minimizes costly waste handling. This practice was not uncommon in some mines studied. However, in incompetent ground, ore losses would result at drift turnouts where ground conditions would deteriorate as mining approached, and it would be necessary to blast multiple rings together. In these cases ore recovery would be very poor.

In the wider sections (i.e. Main Orebody) a transverse system of extraction openings was planned, (See figure 4), whereas in the narrow orebodies it was necessary to use longitudinal layouts. In this latter case the location of the drifts is critical and much predevelopment drilling is required to properly define ore boundaries in order to locate extraction drifts correctly. All levels were to be interconnected by ramps at a maximum grade of 20% for easy movement of supplies, equipment and personnel.

Orepasses

Previous experience with raw orepasses had shown that even in the best Craigmont ground they cave out after about 200,000 tons has been passed.

It was decided to provide two armour-plated orepasses for the Main Orebody from 2400 haulage level to 3432 level which would handle about 5,000,000 tons, and another for the westerly orebodies from 2400 to 3060 level to handle about 2,000,000 tons. The top portion of the westerly orepass could be driven raw as it was largely in competent limestone.

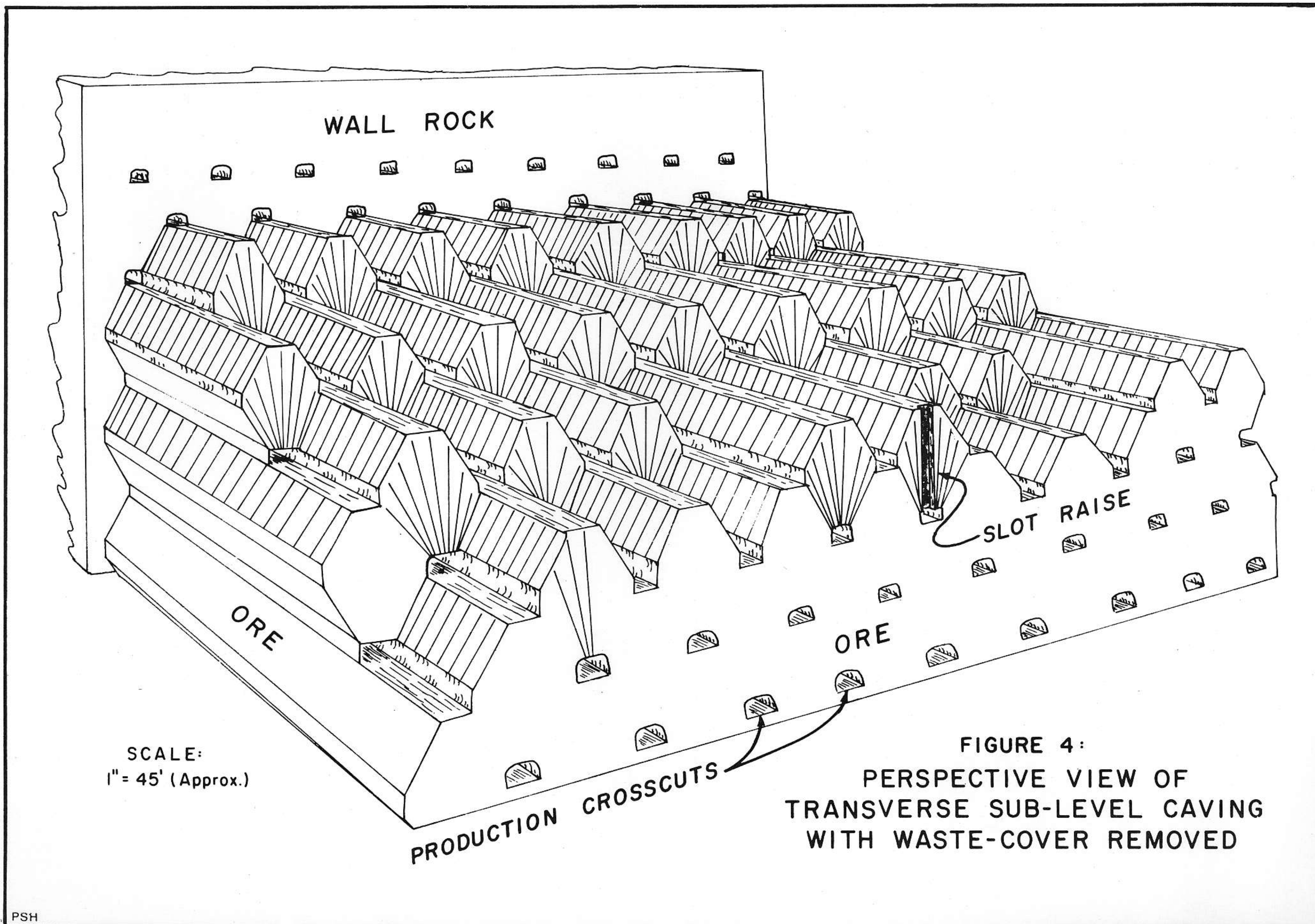


FIGURE 4:
PERSPECTIVE VIEW OF
TRANSVERSE SUB-LEVEL CAVING
WITH WASTE-COVER REMOVED

All tramming would be on 2400 level using two 25-ton trolley locomotives with twelve 256 cubic foot Granby cars per train.

Choice of Equipment

Load-Haul-Dump - Scooptrams, Scoopmobiles, and transloaders were considered. The bottom dump feature of the transloader would have meant a rather complicated and costly orepass dump design. Headroom and operating characteristics were factors in choosing the scooptram. The ST4 was chosen in preference to the ST5 because of much lower ventilation requirement without serious reduction in productivity.

Drilling - For development, Gardner-Denver Universal Jumbos with DH 123 drills were chosen. For blastholes, Atlas Copco Simba 26's with BBC 120 drills, and Gardner-Denver Fan Drills with DH 123 drills were chosen.

Service Vehicles - Requirements for service vehicles were (a) compact to fit onto an 8 foot by 12 foot cage; (b) diesel-powered; (c) sturdy, with good gradability under load; (d) good load capacity, minimum one ton. Mercedes Benz Unimogs and Getman Scootcretes were chosen.

Shotcrete Equipment - Model H True Gun-all machines were chosen because of compact size, simplicity, portability, wet-mix machine, with good maintenance costs and relative high productivity.

Repair Shop - The success of the mining method would depend to a very great degree on the availability and condition of the equipment. It was therefore decided to build a fully equipped repair shop underground close to the mining area in order to perform all work on equipment, except major engine overhauls.

Ventilation - 2400 haulage level and 3060 incast adit were to be used to bring fresh air into the mine. The 3060 outcast adit and 3500 outcast adit would be used for exhaust. Auxiliary ventilation would be a pressure system using 36 inch steel duct along the main entrance and 30 inch neolon duct in extraction drifts.

PRESENT PRACTICE

Equipment

The following is a list of the major items of equipment in use underground at Craigmont:-

7 - ST4A Wagner Scooptrams with Deutz Engines
1 - 3-Boom Gardner-Denver Development Jumbo, with 44 h.p. engine and DH 123 Rock Drills
3 - 2-Boom Jumbo as above
2 - Gardner-Denver Fan Drills with 2 DH 123 Drills
2 - Atlas Copco Simba 26 (modified) Fan Drills with 2 BBC 120 Drills
4 - Mercedes Benz type 406 Service Trucks
3 - Mercedes Benz type 411 Service Trucks
1 - Flocrete Unit Mounting Model H True Gun-all Machine
1 - Flocrete Unit Mounting 22 cu.ft. True Gun-all Machine
3 - Model H True Gun-all Trailers
2 - Scootcretes - model KD2 and model KD52
1 - Galion Grader - model 503A with Deutz Engine
2 - 25-ton Trolley Locomotives; one Goodman, one Clayton
32 - 256 cu.ft. Granby Cars
3 - 72" Sheldon Vaneaxial Fans with 400,450 and 500 h.p. motors
5 - 2 stage 60 h.p. Joy Auxiliary Fans
5 - 2 stage 40 h.p. Woods Auxiliary Fans
6 - Single stage 32 h.p. Woods Auxiliary Fans
8 - 5-ton Atlas Battery Locis
3 - Atlas Copco Cavo 320 Mucking Machines
3 - Atlas Copco Cavo 510 Shuttle Cars

Trackless Development

Sublevels are 31 feet apart vertically, and are joined by a 20% ramp system; every third sublevel is connected to the service shaft. Extraction drifts are driven at a 3% grade to insure good drainage.

All haulage drifts and ramps driven in waste are 13 feet wide, 12 feet high with circular arched back, except where extra large vent duct is used and the height is increased to 13 feet.

Production drifts are 13 feet wide, 10½ feet high with flat back, to give the best draw characteristics.

Perimeter holes in all headings are drilled not more than two feet apart and blasted with X-actex to reduce overbreak and loose. All headings are fully shotcreted and many have re-bar rockbolts as well, ("Ground Support at Craigmont" by A.J. Petrina in the December, 1968 C.I.M.M. Bulletin).

Using the Gardner-Denver Universal Jumbo, one man can drill two 10 foot development rounds per shift, occasionally three. Including drilling, blasting, ST4 mucking, shotcreting and re-bar bolting, advance per manshift is 1.7 feet.

Drainage - The need to keep headings to a minimum width because of ground control makes it impossible to maintain ditches. A Galion Grader, model 503A is to be put into service shortly; provision has been made to surface roads with crushed aggregate.

Sublevels are connected at frequent intervals by 8 inch diameter drain holes, drilled with a GD 133 mounted on an Airtrac. This keeps upper production levels relatively dry.

Slot-raising - In both transverse and longitudinal sublevel caving systems at Craigmont, it is necessary to provide a slot raise up to 50 feet long at the end of each extraction drift. The most satisfactory means of driving the slot raises has been to drill off the entire raise with the longhole production jumbos and then to blast in stages of up to 10 feet.

Production

Drilling and Blasting - Various fan inclinations, burdens, and drill patterns have been tried ranging from 65° to 90° drilling with 3½ feet to 5 feet burden. Current practice is to incline fans at 80° towards the cave and to blast two fans at a time with a burden of 3½ feet each. Figure 5 shows the drill pattern and the blasting layout.

Because the mining method requires a waste cover over and around the blast to contain the ore, the open pit was filled with coarse waste to a depth of 50 feet prior to the start of mining. This was a mistake and contributed to high dilution in the immediate area. When the broken ore is being drawn, any hang-up permits the fine waste to flood beneath it, cutting off ore and increasing dilution. Good fragmentation is particularly important in sublevel caving and accounts for the relatively high powder factor of 0.9 pounds per ton.

Although the mine is quite dry, there is enough seepage from the open pit to preclude the use of ammonium nitrate, and the major blasting agent is 75% Forcite in 1½ inch by 16 inch cartridges. Hole diameter is 2 inches; anything less makes loading too difficult because of caving holes. Due to occasional brow failure and cramped quarters, more fans are loaded with explosives than are blasted at one time; in other words, after a two-fan blast, there would be four fans already loaded and two more would be loaded before blasting the next two.

Load-Haul-Dump - All sublevel caving ore is hauled by ST4 scooptrams to one of three orepasses which deliver to the 2400 haulage level.

The scooptram engines are de-rated to 105 h.p. so that only 10,500 c.f.m. of ventilating air must be provided; oxycatalyst scrubbers on the equipment controls all gases but the oxides of nitrogen which must be flushed from the drift atmosphere by adequate ventilation.

Haulage distances may be as high as 1,200 feet, although the average is 740 feet (one way). Performance is about 300 tons per man-shift. Of the 7 units, five are kept available.

Brow Support - Due to the high degree of fracturing in the orebody, difficulty was anticipated with brow support as successive sublevel caving fans were blasted. The problem is aggravated by increasing drift width.

The effect of the collapse of brows is serious and is manifested as follows:-

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 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

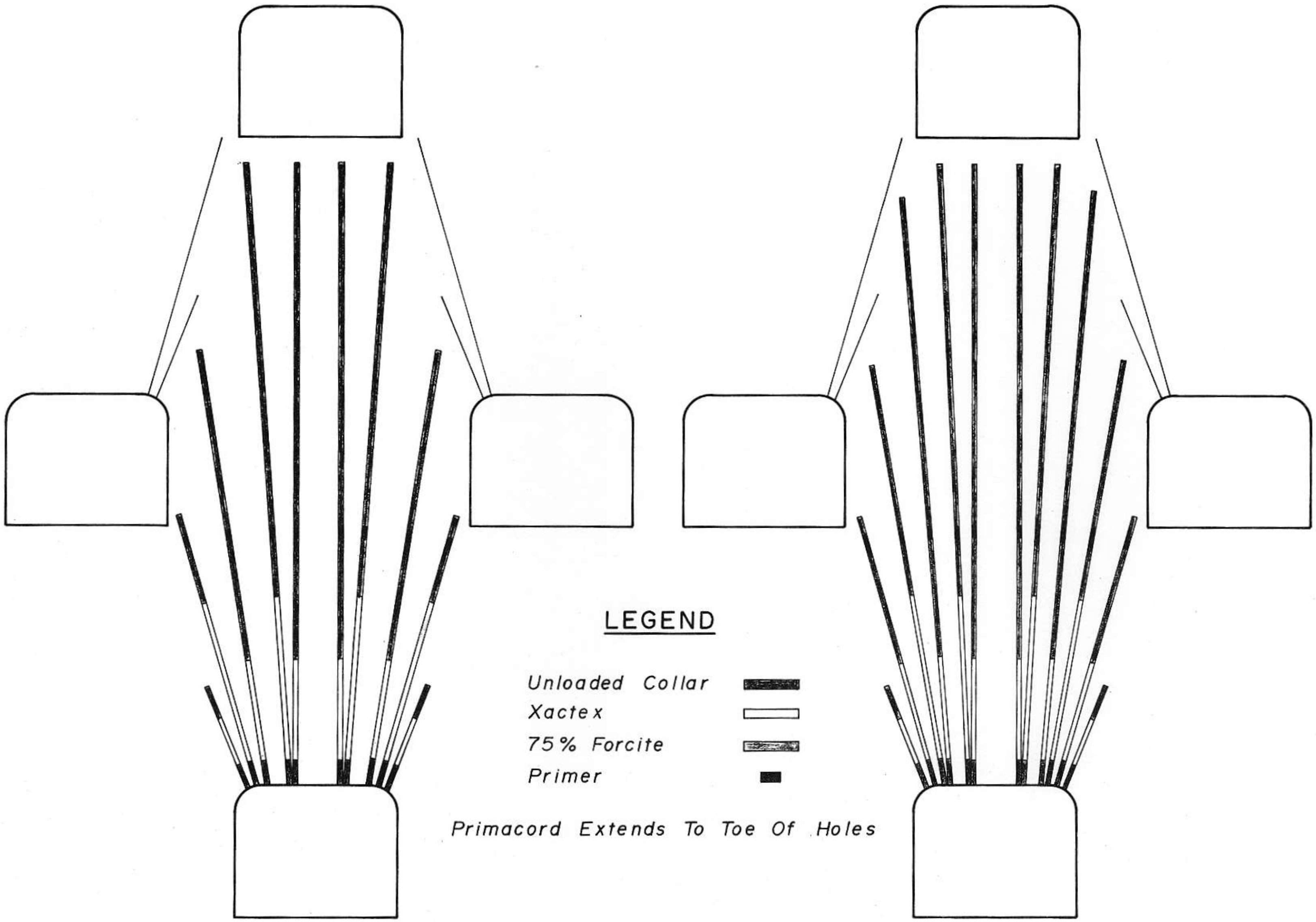


Fig. 5

CRAIGMONT MINES LIMITED MERRITT, B.C.	DRILLING PATTERNS SUBLEVEL CAVING

(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.

The net result of the three conditions described above was to cause low recovery, which in turn made it necessary to consume production drifts at a high rate to maintain tonnage, as well as greatly accelerating the development programme.

Numerous remedies were tried in the effort to control brows.

1. Presplitting of first 10 feet of fan.
2. Use of Xactex near collars of holes.
3. Variations in drill patterns.
4. Expanding shell rock bolts, including bolting timbers to the back.
5. Re-bar bolts installed with Roc-loc.
6. Full length caps supported on bullhorns and blocked to the back.
7. Re-bar bolts installed with mortar grade concrete.

The best results were obtained with re-bar bolts installed with concrete. Vertical rows of up to ten 8-foot re-bars are installed between each fan of blast holes, i.e. 3½ feet apart. Whenever possible, the re-bar holes are drilled with the production jumbo, prior to drilling blastholes; the re-bars are installed in a separate operation and provide support during the blasthole drilling operation.

In addition to the re-bar bolting, judicious use of X-actex (see figure 5) in the lower portion of the blastholes minimize the damage to the brow.

In a few sections of the mine, re-bars are inadequate for brow support, and it becomes necessary to install posts and caps. Obviously this detracts seriously from efficiency, adding another function to the cycle and reducing productivity in all functions.

Orepasses - Ground conditions precluded the use of unlined orepasses except where it was not necessary to pass much more than 250,000 tons. To date, three steel and concrete lined orepasses are in use. These orepasses have so far passed a total of 2,130,000 tons. Maintenance of the lining has been more expensive than anticipated due to the fact that it is not possible to operate with the orepass full because of hang-ups.

In order to prevent any of the three orepasses from inadvertantly becoming too full (and thereby hanging up) a sonic monitor is used in the lower section of each pass. The monitor actuates a depth indicator at each chute on the 2400 main haulage level to indicate which pass should be drawn. Further, if the muck level rises too high a red light goes on automatically at every dumping location for the particular orepass. The depth indicator also prevents the trammers from drawing the chute empty with resultant ventilation and spill problems.

The method of installing these orepasses is described in detail in the paper "Ground Support at Craigmont" in the December, 1968 C.I.M.M. Bulletin and in the paper "Excavating and Equipping a Steel Lined Orepass and Pocket" by E.W. Cokayne.

Haulage - All ore and most waste is hauled from the mine on the 2400 adit level by two 25-ton trolley locomotives pulling trains of twelve 256 cubic foot Granby cars. The run-of-mine ore is dumped into a surface crusher bin by hydraulic dumpers. Waste is dumped along a trestle by a mobile hydraulic dumper, i.e. the train stands still and the dumper moves.

The crusher bin can be closed with air-cylinder-operated doors to keep cold air from freezing wet muck in winter.

The average haul distance is about 8,500 feet with passing tracks at 3,000 and 8,000 feet from the portal.

Chute pulling, tramming and dumping are handled by a single operator. A remote control device permits him to spot cars under the chute from a suitable vantage point beside the chute; similarly, cars are dumped on surface using a remote control. The train itself may be operated either from the locomotive or from a sturdily constructed cabin attached to the last car at the rear of the train. The haulage route is divided into blocks and a system of block lights prevents any more than a single train at a time in any block. Block lights and track switches are actuated by pull switches which can be operated by the trammer from his position on the locomotive (or rear cabin).

The trolley line carries 250 volt A.C. current which is converted to 275 D.C. current in the locomotive. Most of the rail is 60 pounds, but is gradually being replaced with 85 pound rail.

Servicing

All supplies, material and equipment are brought into the mine on rail-mounted flat cars or mine cars, through either the 2400 or 3060 adit levels. 2400 and 3060 levels are connected in the vicinity of the orebodies by a vertical winze equipped with a friction hoist and 8 feet 3 inch by 12 feet 10 inch cage. A similar hoisting installation joins 3060 and 3500 levels; it is this installation which provides access to currently producing orebodies.

All current ore production is obtained from trackless sublevel caving above the 3060 adit level. All supplies and materials are transferred from flat cars to storage areas at shaft stations, or to trackless vehicles for delivery to the required areas.

The following vehicles are used for this servicing:-

1. 3 model 406 and 3 model 411 Unimogs are used for transporting explosives, pipe, timber, shotcrete sand and cement, etc.

2. 1 model 406 Unimog equipped with hydraulically elevating deck is used for rockbolting and certain timbering applications.

3. 2 Getman scootcretes, with local modifications, are used by maintenance crews for servicing other trackless equipment.

4. 500 gallon tank cars, rail-mounted, are used for diesel fuel and are parked near shaft stations.

5. Frequent use is made of ST4 scooptrams, when available, for transfer of supplies and material.

Power is taken underground at 4160 volts, transformed to 550 volts for hoist and fans, 110 volts for lighting, etc. Compressed air capacity is 8,000 c.f.m. at 105 p.s.i.

Ventilation

The primary mine ventilation passes 320,000 c.f.m. of fresh air using the 2400 level haulage adit and 3060 level service adit as intake. Each of these levels is equipped with a two stage 73" vane-axial direct-driven variable pitch 500 h.p., 1200 RPM fan, capable of delivering 160,000 c.f.m. at 13.5 inches S.W.G. Fans are installed in an air lock off the main adit, relatively close to the portal. Air is exhausted from its distribution throughout the working areas by an isolated dual system using the 3060 and 3500 level ventilation exhaust adits.

Figure 6 illustrates a typical sublevel with its secondary ventilation distribution.

The original secondary ventilation system for air distribution to the working face used 30-inch neolon duct and 40 horsepower fans. This was eventually modified to 36-inch spiral-lock duct in the haulage drift, 30-inch neolon in the production cross-cuts and 60 horsepower fans.

The extensive use of diesel equipment, along with stringent legislative controls makes ventilation a major cost item. Ventilation systems are under constant review and are continually being modified to suit the ever-changing geometry of the mine workings.

Maintenance

A well equipped repair shop is maintained underground near the centre of mining activity. It is 230 feet long, 20 feet wide and 16 feet high. A 5-ton electric overhead crane runs the full length. Illumination is by mercury vapour lights.

At the April, 1969 C.I.M.M. Meeting, underground maintenance at Craigmont was described in detail in a paper by A.K. Croteau and will not be discussed further here except to say that all work on trackless equipment with the exception of major overhauls is done in the underground shop.

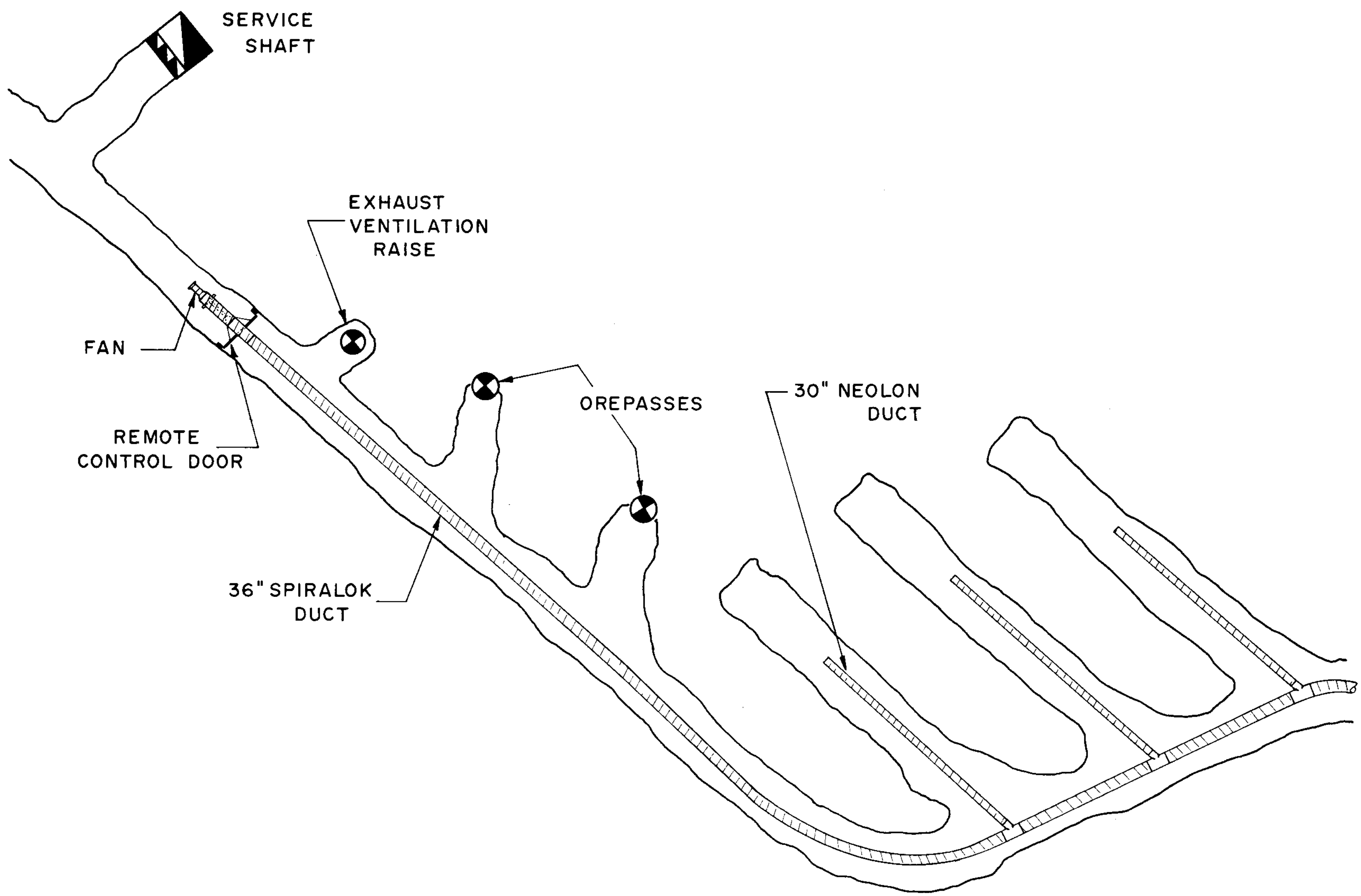


Fig. 6

CRAIGMONT MINES LIMITED MERRITT, B.C.	AUXILIARY VENTILATION ON TYPICAL SUB LEVEL

As sublevel caving progresses downward through the orebodies, the shop will be relocated at a lower elevation.

Organization (See Figure 7)

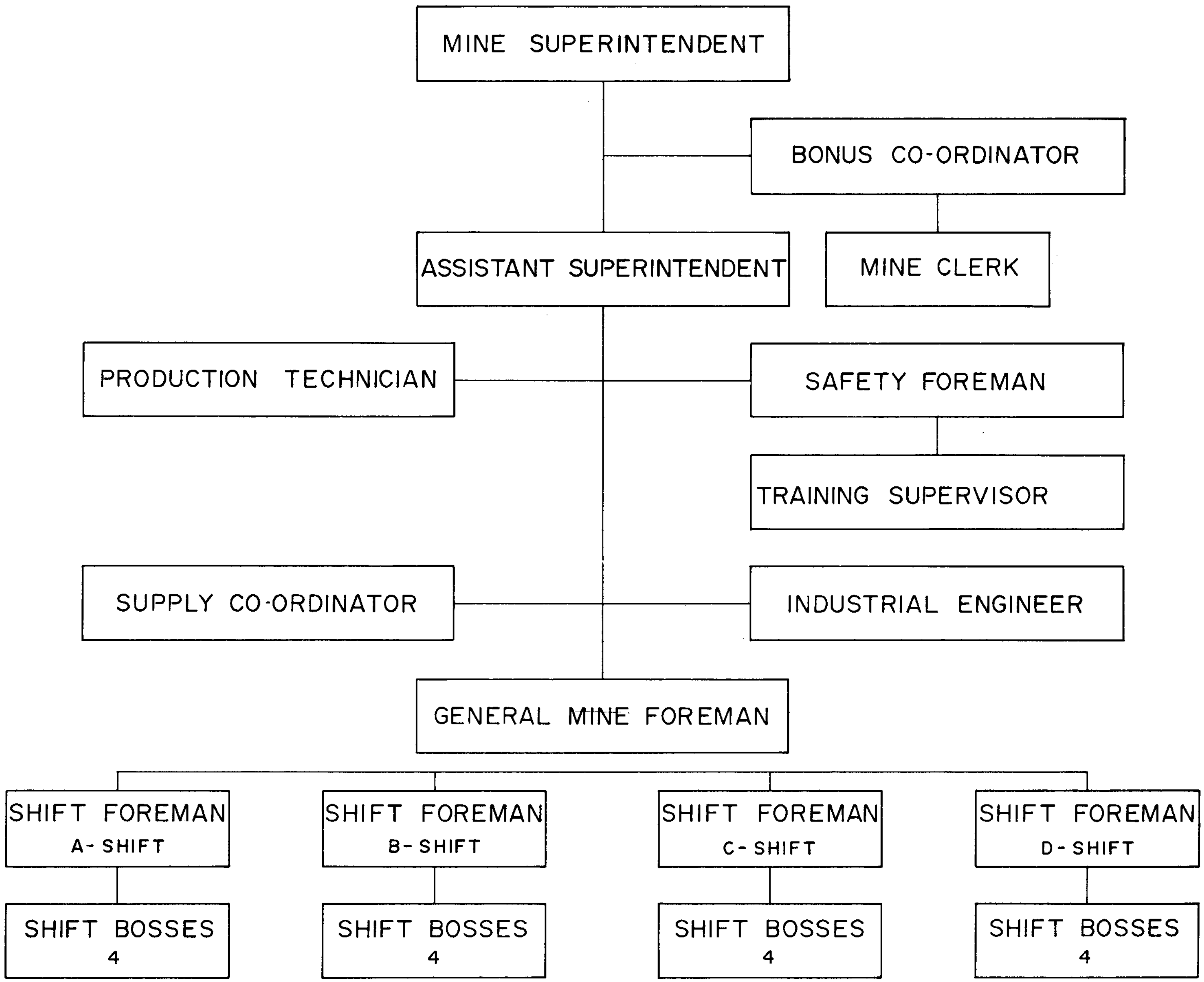
The underground mine is operated 24 hours per day, 7 days per week. Approximately 200 workmen are divided into four shifts (6 days on, 2 off) and are supervised on each shift by four shift bosses and a shift foreman. Overall co-ordination is provided by the mine superintendent, his assistant, and a general mine foreman; in addition, several technical personnel perform staff functions, reporting to the superintendent or assistant. A full-time training supervisor is employed to train and instruct workmen on the various pieces of trackless equipment.

A continuous operations always poses scheduling and communication problems. These are partly overcome by having the shift foreman leave the mine two hours before the end of his shift, and prepare a detailed work schedule for all men and equipment on the on-coming shift. The schedule is verified by last minute phone calls from the supervisors still remaining underground. It is then duplicated on a copier, distributed to supervisors coming on, and discussed in detail. The shift-by-shift scheduling falls within the framework of longer range schedules prepared by the production technician. Yearly schedules are prepared by the engineering department and are up-dated as required.

The duties of the four shift bosses on each shift are divided roughly as follows (there is some overlap):-

(a) trackless production
(b) trackless development
(c) all track development work, plus No. 1 east orebody
(d) transportation: supplies, material, ore, waste

ORGANISATION OF CRAIGMONT MINING DEPARTMENT



CRAIGMONT MINES LIMITED MERRITT, B.C.	*Fig. 7*

Draw Point Control

This very important aspect of the sublevel caving operation is the responsibility of the shift boss in charge of trackless production. He controls the draw by:-

1. Trying to ensure a square brow at the original drift back.
2. Supervising blasting to ensure good fragmentation for an even and continuous flow.
3. During draw, immediate blasting of any hang-ups or partial hang-ups to ensure continuous flow.
4. Knowledge of desired cut-off point and ability to recognize it.

Cut-off Point

(1) The mill is fed mainly by the underground mine and topped up to capacity by drawing from surface low grade stockpiles. Theoretically, to maximize profits, no underground ore which is lower grade than that available from stockpiles should be sent to the mill. However, a profit can still be made from some areas handling underground ore of lower grade than surface stockpiles. Here lower cut-off grades are used.

(2) 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.

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

(4) 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 fluctuations in grade are common, consequently the theoretical number of buckets to achieve a given cut-off grade can vary widely from ring to ring.

(5) 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 condition 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.

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

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

2. 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.

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

A recently purchased P.I.F. Analyser may prove to be of use in determining cut-off grade at the extraction face in reasonable time.

Ore Recovery and Dilution

Recovery is expressed as the ratio of copper units in the mill feed to the copper units that were available in the ore drilled and blasted.

Dilution is expressed by the relationship:-

$$\text{Percentage Dilution} = \frac{\text{tons barren waste in mill feed}}{\text{tons ore in mill feed plus tons barren waste}} \times 100$$

The overall figures at Craigmont since sublevel caving started and to February, 1970 are:-

Transverse Mining -	recovery	92%
	dilution	34%
Longitudinal Mining -	recovery	82%
	dilution	22%

CONCLUSION

For the particular ground conditions, shape of orebodies and grade of ore at Craigmont, sublevel caving has been proven to generate greater profitability than other alternative mining methods.

Performance in the underground mine, including maintenance personnel and all supervisors, is approximately 15 tons per manshift. This figure, which can be improved, is substantially higher than that which could be attained by other mining methods at Craigmont when extraction of all the ore is considered.

ACKNOWLEDGEMENTS

The writers thank those who assisted in the preparation of this paper, and particularly those whose efforts enabled sublevel caving to be successfully implemented at Craigmont.

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