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OPEN-PIT MINING

at

BETHLEHEM COPPER

By H. G. EWANCHUK

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H. G. Ewanchuk, P.Eng.

Henry G. Ewanchuk was born December 5, 1936, at Vilna, Alberta, where he received his public-school education. He attended high school at Prince George, B.C., then took up geology at the University of British Columbia from which he graduated with honours in 1961. His thesis was "Geology and Ore Deposition of the Silbak Premier Mine Glory-Hole".

In the summer of 1959 he worked on the property of Huestis Molybdenum Corporation at Pacific, B.C., and in the summer of 1960 engaged in field work for Southwest Potash Corporation in northern British Columbia. Following graduation in 1961, he worked for eight months on a salvage operation for Silbak Premier Mines Limited and then spent six months with the Geological Survey of Canada as technical officer in the Vancouver office. In 1962 and the first half of 1963 he was section geologist in charge of ore control, calculation of ore reserves, and development control for Eldorado Mining & Refining Limited.

"Hank" joined Bethlehem in July 1963 as geologist. Two years later he was promoted to mine engineer in charge of ore control, mining planning, design of open pit, and some property exploration. In March 1966, he became manager of mine production of the big Highland Valley copper company.

Synopsis

A brief description is given of the open-pit mining methods employed by Bethlehem Copper Corporation in the Highland Valley, British Columbia. A short discussion of the physical and geological aspects of the Highland Valley area precedes the consideration of the open-pit operation. Detailed information is given with regard to ore reserves, pit design, choice of mine equipment, scheduling of open-pit operations, together with specific reasons for each management decision. The advantages and disadvantages involved in contract versus company mining are outlined, and the Corporation's decision to undertake its own mining is explained.

The Bethlehem property is located in the Highland Valley area of southwestern British Columbia at a latitude of 50° 30′ and a longitude of 121° 00′. The open-pits and milling installations are located at an elevation of approximately 4900-5200 ft. above sea level in an area of gently rolling hills dotted with numerous small lakes. It is characterized by moderate snowfall in winter, moderate rainfall and pleasant temperatures in summer. The mine is serviced by a paved road from the village of Ashcroft, which is located 29 miles to the northwest and has a population of approximately 1200. Ashcroft, which is on the Thompson River, is serviced by both the Canadian National and the Canadian Pacific Railways.

The property consists of 486 mineral claims covering an area in excess of 20,000 acres. The water-table is at a fairly shallow depth below the valley floor. Although water is available from the lakes as well as from some water wells located at an elevation of 3885 ft. above sea level, fresh water has to be supplemented by reclaimed water from the tailings to supply the mill requirement.

A special feature of the open-pit operations at Bethlehem is the relatively small size of the orebodies which are being mined. The East Jersey pit, which was the first to be mined, contained 4,100,000 tons of ore and 6,700,000 tons of waste. The Jersey pit, which is presently being mined, consisted of 42,000,000 tons of ore and 28,000,000 tons of waste. The Huestis zone, an orebody now in the final stages of drilling, will be slightly smaller than the Jersey orebody. Thus the economics of pit design and planning differ from those of the large pits in the southwestern U.S.A. and South America.

History

Copper mineralization in the Highland Valley has been known since the turn of the century. In 1899 a group of mineral claims was staked in the area now known as the Snowstorm zone. In 1915-16 90 tons of ore averaging about 30% copper were shipped. Although exploration was carried on in the area from time to time, it failed to establish an orebody of economic value.

In 1954 the Huestis-Reynolds Syndicate prospected in the Highland Valley and staked about 100 mineral claims covering the Snowstorm, Iona, and Jersey zones. Bethlehem Copper Corporation, Ltd., was incorporated in 1955 and the claims staked by the syndicate were sold to the company. This marked the beginning of a successful programme of exploration. In September, 1955, American Smelting and Refining Co. took an option on the property and during the next 2½ years spent $1,250,000 in exploration — mainly geophysical surveying, trenching and diamond drilling.

Other commitments led American Smelting and Refining Co. to drop its option early in 1958. Bethlehem Copper then proceeded on an underground programme to check the results of Asarco's diamond drilling. A participation and financing agreement was negotiated with Sumitomo Metal Mining Co. of Japan in 1960 to build a 3000 ton/day concentrator and to put the East Jersey zone into production.

A paper presented at the Ninth Commonwealth Mining and Metallurgical Congress, London, May 1969.

Persistent efforts by the company to bring the property into production, backed by the faith of its originators, were rewarded in 1962 with the first shipment of copper concentrates to Japan. Various phases of expansion have occurred, and Bethlehem operates one of the largest open-pit copper mines in British Columbia, and is one of the major copper producers in Canada.

Geology

The property is situated near the centre of the Guichon batholith of lower Jurassic age. The batholith is about 40 miles long in a north-south direction and about 16 miles wide. The rock is a massive, coarse-textured, grey quartz-diorite, called locally the 'Guichon' or older quartz-diorite. Locally, the quartz-diorite has been intruded by granite, quartz-monzonite, several porphyries, and a distinctively younger quartz-diorite. Associated with these varied younger rocks are several bodies of breccia. The origin of this breccia is not definitely known, but there are several possible explanations — (1) explosion, (2) intrusion and (3) intrusive.

Both the breccia and the younger intrusives are host rocks for mineralization. The mineralization consists of chalcopyrite, bornite, molybdenite and minor pyrites, which occur as disseminated grains, fracture filling, lenticular deposits and as irregular coarse blebs. Several ore zones are located close to the contact between the Guichon batholith and the younger intrusives, where breccia may or may not be found. The important point is that the mineralization is controlled by structures and contacts of rock units. Not all structurally disturbed rocks are mineralized, but most of the mineralization is found in dislocated and disturbed rocks and in cross-cutting younger intrusive rock units. In the brecciated areas mineralization is of a higher grade and is more uniformly distributed than in the non-brecciated rock.

Fig. 4 shows the pertinent faults in the Jersey and East Jersey mines; particularly noteworthy is the slight elongation of the Jersey orebody parallel to the Jersey fault. In the East Jersey mine the orebody appears to be fault-controlled as it is stretched along the footwall of the East Jersey fault. The main tonnage of the Jersey orebody is on the hangingwall of the Jersey fault; however, the ore is considerably farther from the fault. Fig. 4 does not illustrate the true density of faulting, as there are many faults which were mapped but not shown.

Ore Reserves

The cutoff grade for the ore-reserve calculations is 0.35% copper, any material below this cutoff grade being classified as waste. This waste material is stockpiled in large waste dumps or used in the construction of the tailings dam. The waste removed is not susceptible to leaching owing to the extremely high carbonate and low pyrite content. The Jersey pit is presently being mined. As at 1 January, 1968, the remaining proved ore reserves in the various zones were approximately 70,000,000 tons, grading approximately 0.60% copper.

Other zones of known mineralization

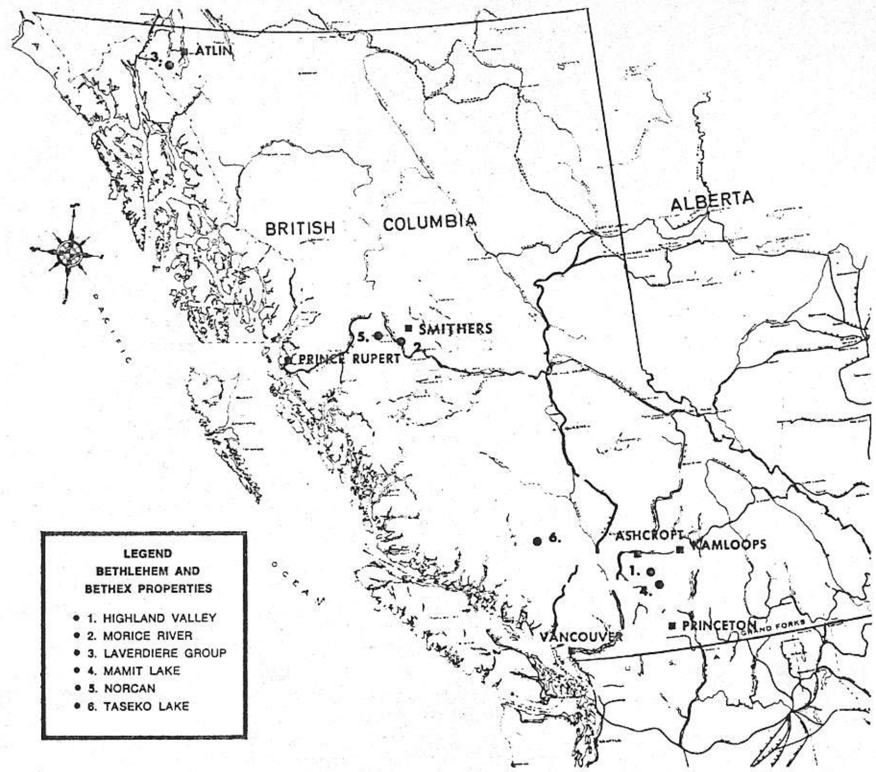


Fig. 1 Map showing locations of Bethlehem Copper Corporation and Bethex Explorations properties

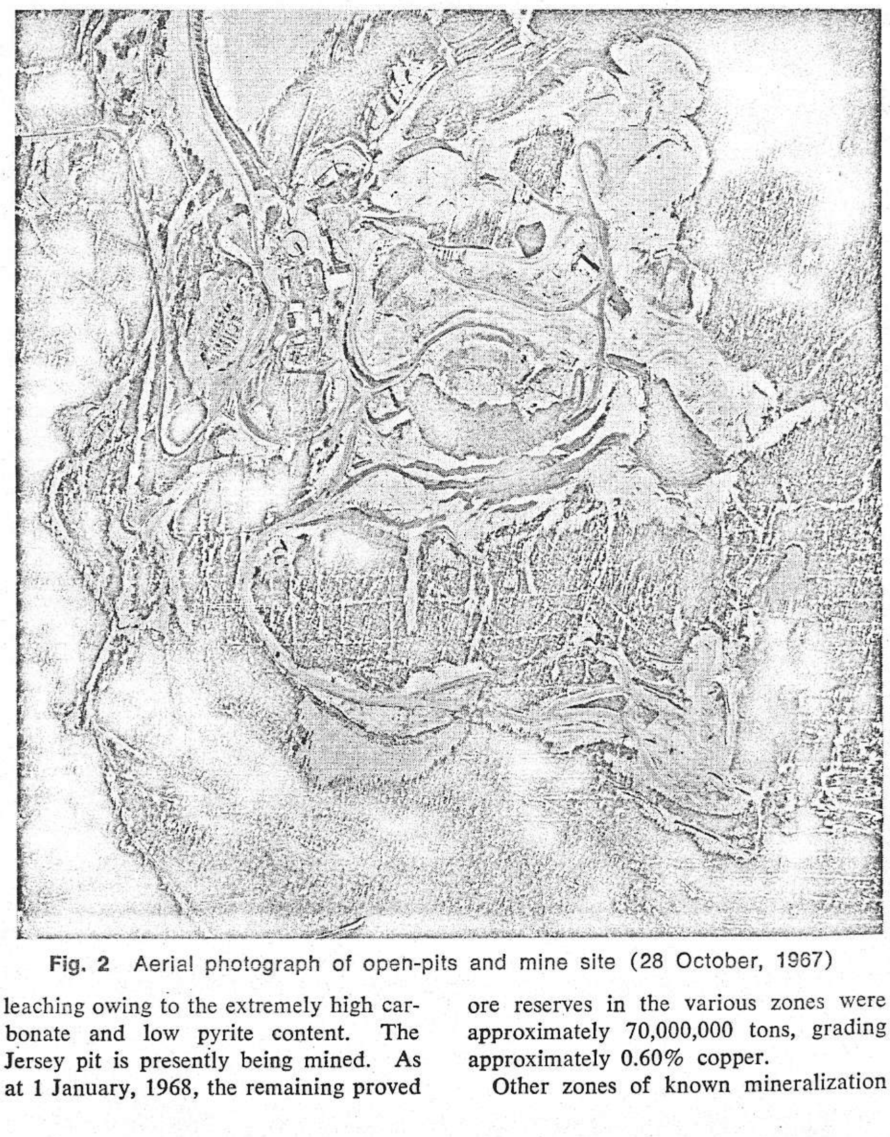


Fig. 2 Aerial photograph of open-pits and mine site (28 October, 1967)

exist — for example, Hank zone, White zone, Spud Lake zones, Simon's zone. The ore potential of these zones will form targets for future exploration. At the current milling rate of 4,000,000 ton/year the present proved reserves will supply the mill for approximately fifteen years.

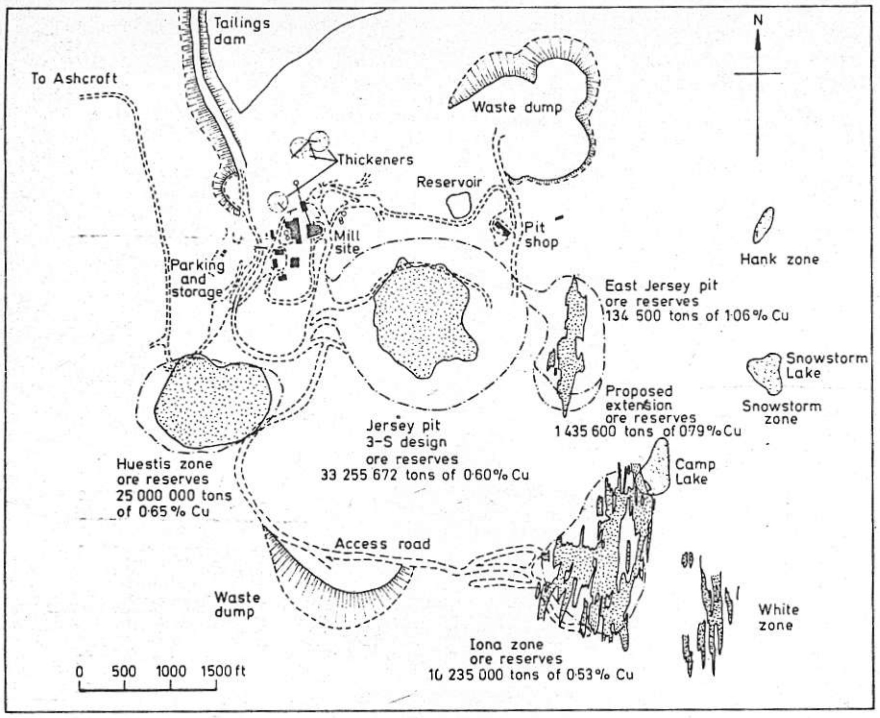


Fig. 3 General site plan showing location of orebodies and ore reserves as at 29 February, 1968

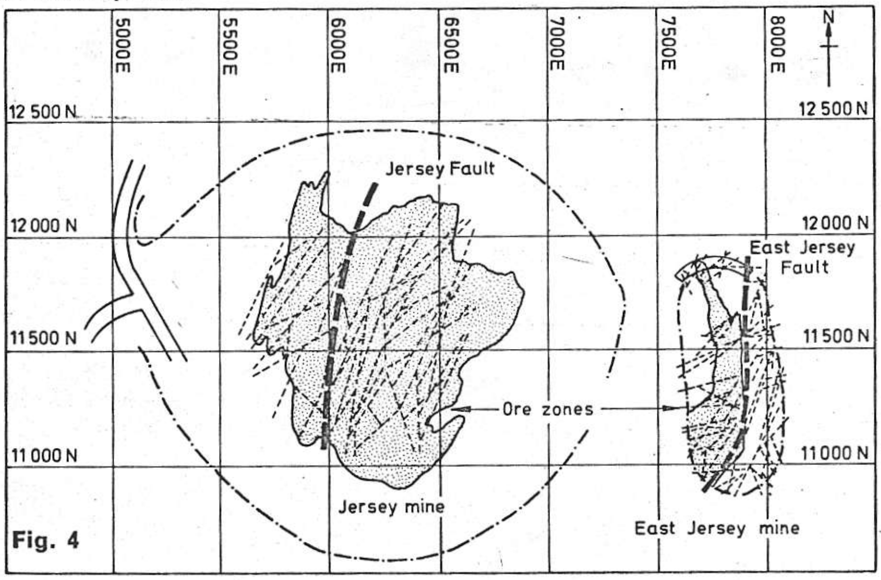


Fig. 4

Pit Design

The Jersey pit is 2400 ft. long and 1900 ft. wide; maximum depth of the pit at completion will be 1050 ft. Mining is carried out on 33-ft. benches. Pit walls are sloped at an angle of 70° and a 25-ft. safety berm is left every second bench to catch any falling rock and to give added stability to the pit wall. Extra-wide berms are provided where necessary because of structural or operational consideration. The pit is designed at an overall slope of 45-48°, although individual wall slopes vary considerably because of the location of haul roads. (A plan of Jersey pit as currently designed is shown in Fig. 1 of Appendix 2. Typical cross-sections through the pit and the location of ore and waste areas are given in Figs. 1-5 of Appendix 1.)

All haul roads are designed on a 10% grade. Originally, the roads were designed to spiral to the ultimate bottom. As the pit developed, the south wall was found to be relatively incompetent due to a number of major faults. The pit was redesigned and a series of switchbacks on the haul roads was put in to relocate the road system away from the south wall of the pit.

Mining Equipment

Exhaustive cost studies were made with various sizes and types of equipment in order to enable the most productive and economical machines to be selected. Diesel shovels and drills were purchased to obtain the maximum mobility and flexibility in scheduling production. Although two larger shovels could have provided the required production, three

Table 1 Jersey Pit Mining Schedule, 1968-73

Bench elevation	*1968 Ore*	*%Cu*	*Waste*	*1969 Ore*	*%Cu*	*Waste*	*1970 Ore*	*%Cu*	*Waste*	*1971 Ore*	*%Cu*	*Waste*	*1972 Ore*	*%Cu*	*Waste*	*1973 Ore*	*%Cu*	*Waste*
4967			169 095															
4933			419 105															
4900			644 200															
4867			1 941 900															
4833	181 400	·407	1 817 100	—	—	—												
4800	313 200	·430	1 988 900	—	—	538 200												
4767	283 600	·445	633 000	114 300	·382	1 591 900												
4733	280 000	·461	246 200	280 700	·441	1 957 100												
4700	—	—	—	913 000	·602	1 257 400												
4667	663 700	·599	100 500	1 081 400	·590	1 167 000												
4633	1 654 600	·686	—	1 398 900	·614	656 000												
4600	1 394 700	·672	—	146 400	·471	125 500	1 470 700	·607	389 200									
4567	253 800	·794	15 000	935 100	·714	18 600	1 565 000	·597	298 800									
4533	—	—	—	605 200	·811	—	1 950 900	·605	365 300									
4500	—	—	—	—	—	—	488 400	·523	54 800	1 927 700	·685	67 700						
4467							—		—	2 341 700	·632	47 700						
4433										1 205 600	·639	136 000	644 200	·554	—			
4400										—	—	—	1 634 200	·556	178 100			
4367													1 486 400	·551	37 800			
4333													1 311 600	·513	69 500			
4300													413 600	·615	41 700	594 900	·585	2 800
4267													—	—	—	914 300	·557	18 500
4233																626 800	·668	109 100
4200																524 300	·657	90 000
4167																275 400	·779	176 000
4133																327 500	·732	28 700
Total	5 025 000	·624	7 975 000	5 475 000	·634	7 311 700	5 475 000	·593	1 067 100	5 475 000	·652	251 400	5 490 000	·552	327 100	3 263 200	·636	425 100

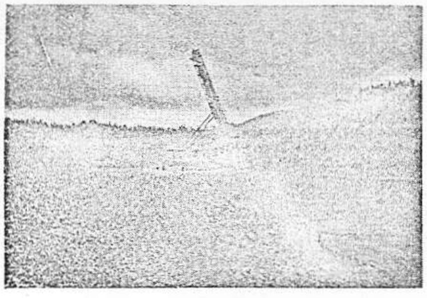


Fig. 5 Bucyrus-Erie 45-R drill on working bench

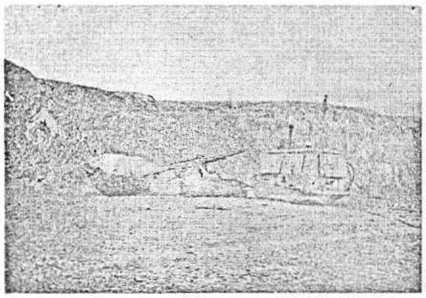


Fig. 6 Gardner Denver 3100 Air Trac drill drilling boulder for secondary blasting

Bucyrus-Erie 88-B shovels with 5½-yd.^{3} buckets were selected. The three smaller shovels were selected so that more uniform blending of ore might be achieved. A well equipped shop on the property enables repairs to be made to any equipment.

Mining Schedules

Long-term mining schedules are prepared in detail and form the basis for all mining operations. Most of the waste rock in the area of the Jersey pit is concentrated on the upper benches. In order to feed the mill at a uniform grade and to maintain the waste/ore ratio at a level where expenditure can be supported by the mining budget or by revenue, long-term mining schedules are necessary. The long-term schedules also serve to indicate the pit life and size of the exploration and development programmes for the new ore zones, and assist in the preparation of budgets. Very little effort is made to average out the production grade in these long-term mining plans. The long-term mining plan proves the feasibility of supplying ore to the mill at any particular rate and indicates the waste/ore ratio which will be required. The results of the overall Jersey pit mining plan at a milling rate of 15,000 ton/day are given in Table 1.

A monthly mining schedule prepared on a composite bench plan is presented in Appendix 4. This schedule is prepared to conform generally to the programme laid out in the overall mining schedule. All ore and waste blocks are shown in detail. Ore grades are arrived at from the detailed assays obtained mainly from the blast-hole samples. The mining schedule is prepared so that a reasonably uniform grade of ore can be supplied to the mill at all times by blending ores from different blocks. This schedule is distributed to various persons in the department concerned with the day-to-day operations in the pit.

Although the Jersey pit is designed for an overall waste:ore ratio of 0.655:1, a major portion of this waste is concentrated on the upper benches. In order to obtain the desired ore production at the average pit grade, production has been scheduled at, or in excess of, a waste to ore ratio of 2:1 for the first few years. A temporary road system was designed so that ore production could be obtained from lower benches while waste from the top benches was removed. Mining of ore and waste has to be closely planned and tight schedules have to be maintained.

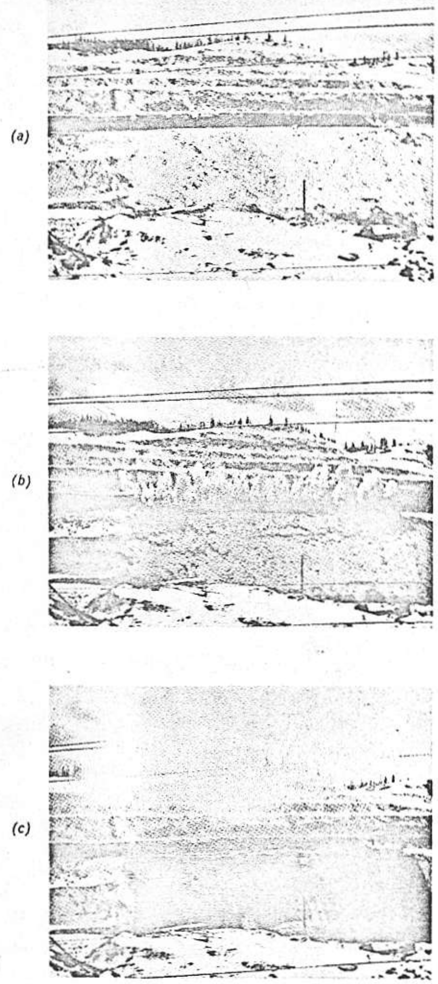


Fig. 8 Blasting sequence in Jersey pit: (a) pre-blast; (b) blasting; (c) after blast. Blast consisted of 118 40-ft holes of 9⅞-in diameter, 48 200-lb of AN-FO, 800 lb of Hydromex T-3 blasting slurry, 118 sticks of 2 in x 16 in Forcite and 118 blasting caps being used. Note slight amount of displacement in (c)

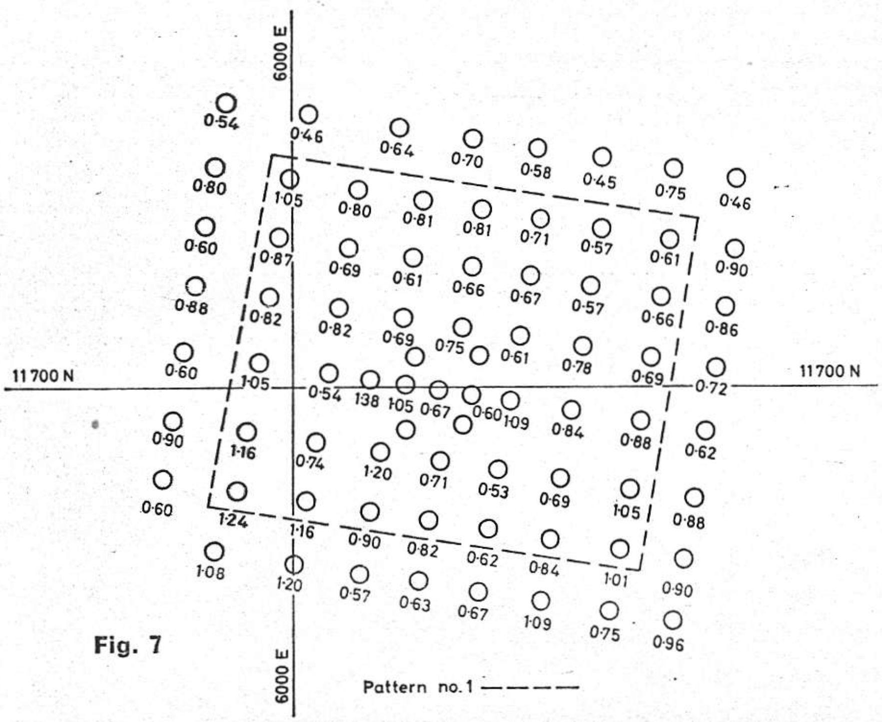


Fig. 7

It is expected that by mid-1969 the major tonnage of waste will have been removed. After this period the waste: ore ratio will be 0.25:1 or less. At that time it is planned to utilize the extra capacity to strip other proved ore zones, such as the Huestis and Iona. Figs. 1-5 of Appendix 2 show the Jersey pit in plan as it is mined and developed. The first-year plan corresponds to the pit as at 1 May, 1968. Subsequent plans show the development of the pit over the next four years — until the pit is mined out. If production plans, grade of ore and the structural nature of the rock deviate from those expected, the mining plan is revised accordingly.

Drilling

Primary drilling is done with the 45R Rotary drill, drilling 9⅞-in. diameter holes. The 33-ft. benches are drilled with 40-ft. holes, giving a sub-grade of 7 ft. The Jersey pit rock is relatively easy to

Table 2 Assay and Screen Analysis of Hole Cuttings (Hole 4,4930/26)

Sample no.	*999*			*1000*			*5929*		
Mesh, Tyler	*Weight, g*	*Box 1, %*	*% Cu*	*Weight, g*	*Box 2, %*	*% Cu*	*Weight, g*	*Box 3, %*	*% Cu*
$+\frac{3}{8}$ in				1·3	0·1	1·08			
$-\frac{3}{8}$ in	1798·3	56·7	1·83	974·5	50·5	1·81	647·7	44·2	1·70
−10 mesh	460·3	14·5	1·72	284·1	14·7	1·89	199·1	13·7	1·87
−20 mesh	325·6	10·3	1·66	218·4	11·3	1·96	158·6	10·8	1·85
−35 mesh	215·8	6·8	1·98	156·4	8·1	2·45	133·9	9·2	1·92
−60 mesh	77·5	2·4	3·06	57·6	3·0	2·74	63·1	4·3	2·97
−100 mesh	102·1	3·2	4·14	78·1	4·1	4·74	88·8	6·1	4·22
−200 mesh	192·6	6·1	4·43	158·2	8·2	4·68	171·4	11·7	5·41
Weighted assay	3172·2	100·0	2·07	1928·6		2·30	1463·4		2·40
Composite			2·21			2·10			2·12
Pipe sample			2·70			2·72			3·02

drill. A major part of the drilling is carried out with steel tooth tricone bits equipped with tungsten carbide buttons on the outside of the rollers for gauge protection. Carbide button bits are used in the harder ground. Holes are drilled on a 28 ft. x 28 ft. pattern — except in the vicinity of the pit walls, where the burden on the hole is cut down almost to half.

The lighter burden permits less explosive charge in the hole, and thus minimizes damage to the pit walls. The pit walls are trimmed with 3-in. percussion holes, where needed. The holes are staggered on successive rows to avoid toes between holes. Experiments with wider pattern drilling are now in progress. The plans are to maintain the present burden but to increase the spacing of the holes.

Efficient drill performance depends on a combination of the following considerations. (1) Sufficient volume of air must be available at the bit to move the cuttings into the annulus as soon as they are chipped loose — otherwise they will stay under the bit and be reground, resulting in slower penetration and excessive bit wear. (2) Sufficient down pressure must be applied to the bit to overcome the rock strength. The drill must have the power to turn the drill stem continuously with the full down pressure on the bit, at a rate of 45-50 rev./min. in hard rock and 60-70 rev./min. in softer rock. A faster speed will only regrind the cuttings and again increase bit wear. (3) The compressor has to provide the air at sufficient pressure to force it through the piping system, to clean the bit bearings of dust and grease, and to clean the hole of dust and cuttings, as well as muddy water if drilling is being done in wet ground. About 10 lb./in.^{2} is required to overcome friction losses, 25-35 lb./in.^{2} is needed at the bottom of the hole, and an extra 10 lb./in.^{2} is required as a safety factor.

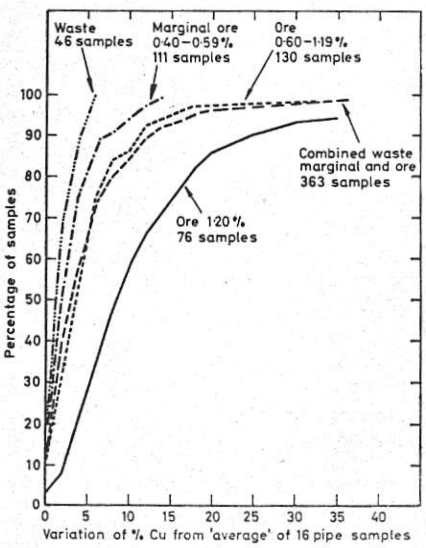


Fig. 9 Cumulative percentage of sample variation for 4 pipe samples compared with average of 16 pipe samples

All these considerations must be in balance if the most efficient drill performance is to be obtained. The best indication of such balance is an even wear pattern on the bit. Excessive wear around the outside of the rollers would mean that rock chips and pieces of rock are coming down the hole and being reground — indicating an insufficient volume of air. Too fast a gauge loss could result from excessive drilling speed. A slow penetration rate could indicate that the bit is insufficiently loaded. A close check is kept on all these factors and each bit is examined for its wear pattern as it is removed from the drill.

The drill reamer — stabilizer at the bottom of the drill stem is equipped with a water separator for wet drilling. It has not been necessary to use water for drilling during a major part of the time. Water is used for drilling in summer when the dry weather creates an extreme dust problem.

The footage drilled per 8-h. shift averages about 600 ft., although a drilling rate of 90-100 ft./h. is not uncommon in softer ground.

Blasting

Ammonium nitrate-fuel oil mixture is used for blasting in all holes, except in very wet ground. The AN-FO mix is supplied by Canadian Industries, Ltd., at the hole. The ammonium nitrate and fuel oil are carried on the truck in separate tanks and are mixed as they are fed into the hole. The hole is primed near the bottom and a detonating cord is used through the complete length of the hole. A stock of 2 in. x 16 in. 70% forcite is generally used for priming. Pentamex primers are used in very hard or wet ground. Short period electric blasting caps are used in blasting. In starting a new bench, extra holes are drilled (Fig. 7).

The holes are delayed away from an open face or broken rock. The usual practice is to blast behind a previously broken muckpile. This method has several advantages: (1) there is better fragmentation and less fly rock; (2) there is little lateral movement of the broken material — the rock remains essentially in its original position and mixing of ore and waste is thus avoided; and (3) when shovelling, the face of the broken muckpile remains almost vertical — thus the ore sorter knows at all times where the face is in relation to ore or waste markers.

In the vicinity of the pit walls and berms it is desirable to keep seismic vibrations to a minimum. Any blasting in these areas is done to an open face. The last few feet are trimmed with an air-trac drill and 3-in. diameter holes.

Every effort is made to pump the water from set holes, to insert a plastic liner and to use AN-FO for the explosive charge. Two submersible electric pumps with an overall diameter of 5⅜ in. (Prosser Series 1300) are used in tandem to pump out the holes. A portable gen-

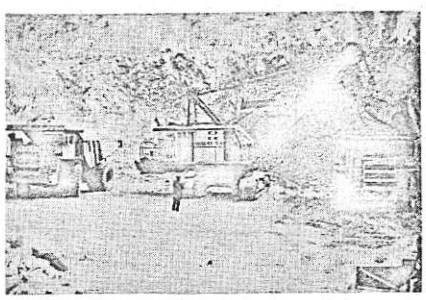


Fig. 10 Bucyrus-Erie 88-B shovel in a typical loading position with 50-ton Haulpaks on either side

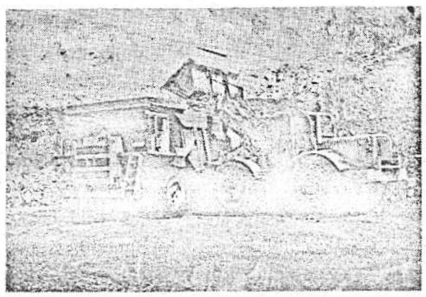


Fig. 11 Caterpillar 988-B front-end loader filling 50-ton truck

erator, the pumps and the discharge hose are carried on a pick-up truck, which moves to the hole for pumping. A plastic liner is lowered into a hole as soon as it is pumped dry and the AN-FO charge is loaded. If the hole is excessively wet and cannot be pumped, it is loaded with a metallized slurry (Hydromex in 8-in. diameter cartridges) primed with a Pentamex primer. Since Hydromex is almost five times as costly as AN-FO, every effort is made to keep the Hydromex consumption to a minimum. Generally, very little of the metallized slurry is used during summer. In extreme cold weather in winter, pumping of holes is not very practical and slurries have to be used. A non-metallized slurry, Hydromex T-3, is now under trial. This new slurry provides the advantages of a waterproof blasting agent with comparable speed of detonation at a lower cost.

Grade Control

The average ore grade in Jersey pit is 0.60% copper and the cutoff grade between ore and waste is 0.35% copper. With this close margin between the ore and the waste grade, it was very important to establish a reliable method of grade control.

Cuttings from each blast-hole in Jersey pit are assayed and form the basis of grade control. Initial attempts at sampling included the taking of grade samples at random from a representative sample of the entire pile of cuttings. These grade samples showed a wide variation in copper content. Attempts to eliminate human error included sampling with a mechanical fork-shaped sampler, a mechanical pipe sampler, a pie wedge-shaped sample cutter placed next to the drill collar, and wooden boxes placed in the various quadrants around the hole. It was found that the higher copper content occurred in the smaller particle size, although distribution of particle size varied with each blast-hole (Table 2).

The plotted results of the tests indicated that an average of the pipe samples gave sufficiently accurate results when compared with the control sample, although individual pipe samples varied appreciably (Fig. 9). At present, grade control at Bethlehem is based on the following factors. (1) Geology is utilized in selective sorting by predicting the location of contacts, dykes and waste-ore boundaries. (2) Representative samples are taken from the blasthole cuttings. These assay grades from samples are marked on stakes located on the muckpile with a plane table. Ore and waste areas are indicated by coloured ribbons. (3) The cuttings are sampled with a 4-in. diameter piece of iron pipe. Cuts are taken from the pile of drill cuttings after the cuttings from the sub-grade drilling have been removed. The cuts are combined to make one sample and then split using a Jones riffle to a convenient size for assaying. (4) In order to minimize any lateral movement of rock, blasting is done behind broken muck, so far as this is possible. (5) Personnel are trained to estimate visually the value of low-grade ore blocks.

The effectiveness of this method of grade control is reflected in a comparison of the reported mine grade and the millheads — generally between 1 and 2% at Bethlehem.

Excavating and Hauling

The broken muck is loaded by Bucyrus-Erie 88-B shovels (Fig. 10) equipped with 5½-yd.³ buckets. Three shovels work in each shift and are supplemented by a Caterpillar 988-B front-end loader (Fig. 11). The loader is used for berm clean-up, loading in tight working areas, and developing good shovel faces after blasts, and it replaces a shovel in case of breakdown or removal from service for scheduled maintenance. An extra loader is available in case of breakdowns of the scheduled shovels, as well as during maintenance and service work. The three shovels are so placed about the pit that moves are generally confined to short distances.

The haulage fleet consists of 15 50-ton Haulpak rear-dump trucks. A shovel generally works with 4 or 5 trucks and the front-end loader with 2 or 3, depending on the haul distance as well as the speed of loading. Idle time on equipment is cut to a minimum and the supervisors' pick-ups are all equipped with radios; maximum mobility of the equipment and operations can thus be achieved.

Eleven or twelve trucks work on any one shift and, normally, two spare units are available. Two or three trucks are usually in the shop for servicing and maintenance. As the fleet works a 24-h. day, seven days per week, it is essential to service every unit once in each 24-h. period.

Pit, Dump and Road Maintenance

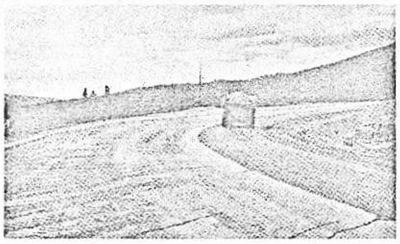
In order to obtain equipment efficiency and low tyre costs, all effort is made to maintain the pit floors and haul roads in top condition. A grader, packer, water truck and other subsidiary equipment are utilized for road and waste dump maintenance.

Repairs and Maintenance

A heavy equipment maintenance shop, equipped with the latest type of tools and equipment capable of effecting all repairs and complete overhaul of any unit on the job, is available at the mine site. Service and repair crews work around the clock, although major repairs and overhaul jobs are concentrated in the day-shift. All equipment is regularly serviced and is thoroughly checked by mechanical crews, according to a preventive maintenance schedule. To date, an uninterrupted mining operation has been achieved.

Cost

The ultimate test of a commercial operation is in the cost of production. Records of production cost at Bethlehem are kept in complete detail. Estimates of cost are made each week and any variations in costs are analysed to find the areas where results may be improved. Direct costs of mining per ton moved, including the charge based on the useful life of all mine equipment as well as reserve accounts to cover any major maintenance and tyre replacement costs, range between 25 and 29¢/ton. A detailed analysis is shown in Table 3.


(a)


(b)

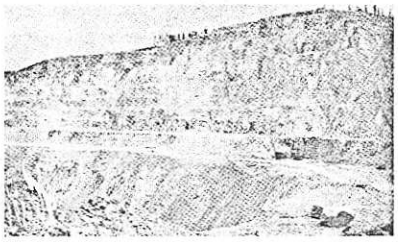

(c)

Fig. 12 (a) Typical haul road; (b) hauling on the tailings dam; (c) typical haul road in the pit, 4833 bench (note excellent condition of berms to 5167 bench at top)

A major portion of the waste is used to build the tailings dam. The extra haul involved has been estimated to cost 3.2¢/ton and is included in the mining cost and reflected in the haulage section.

Apart from the overall efficiency of all operations, two items require special mention because of their effect on costs — (1) the condition of haul roads and (2) the number of temporary ramps built and relocated. Poor haul roads result in slowing down the hauling units and cause a sharp rise in tyre costs. The use of a large number of temporary haulage ramps caused by frequent ramp relocation is a result of poor planning and always results in a slowing down of operations and in higher costs.

Cost control is a continuous operation.

A relaxation of vigilance results in higher unit costs.

Contractor Versus Company Mining

In 1961 Bethlehem Copper was one of the first mining companies in western Canada to introduce a contractor into its operation to do all the mining. The mining operations were taken over from the contractor by the company on 1 November, 1967. Why did Bethlehem bring in a contractor? Why were the mining operations taken over later? The answers to these questions are of great value to a large number of companies attempting to develop new orebodies.

In 1961 Bethlehem was a small company which was trying to develop a marginal-grade orebody on a very limited capital investment. By having the mining done by a contractor, a saving of about $2,500,000 in capital costs was realized. Although the contract price included a profit for the contractor, it also offered the company a fixed cost for the mining operation as well as a reduction in initial capital. The result of feasibility reports indicated that the contractor's price for mining could be carried with no undue dilution of the profitability of the initial operation.

As was previously noted, production at Bethlehem was rapidly accelerated from a starting rate of 1,000,000 tons of ore per year in 1963 to in excess of 4,000,00 ton/year in 1967. During this period the contractor's fee for mining services had gradually increased to a point which necessitated a re-evaluation of the situation. The results of the study indicated that (1) sufficient capital was available for purchase of the necessary equipment; (2) greater flexibility and efficiency could be realized by control of the actual physical aspect of the mining being in Bethlehem's hands; and (3) a substantial saving in operating costs could be realized. Bethlehem assumed control of the mining operations with its own equipment and personnel on 1 November, 1967. The forecast savings in mining costs have been achieved and the overall improved efficiency of the operation is a matter of record.

Table 3

Drilling	1·5 ¢
Blasting	2·0 ¢
Loading	5·0 ¢
Hauling	11·5 ¢
Pit, dump and road maintenance	4·0¢
Supervision, engineering and miscellaneous	3·5 ¢
Total	27·5 ¢

Personnel and General Considerations

The Mine Production Department at Bethlehem consists of 23 salaried and 130 hourly rated employees. Of these, 15 are involved in engineering, 112 in operating, and 26 in mechanical functions. An organization chart for the Department is given in Appendix 3.

Mine rescue and industrial first aid courses are available and employees are encouraged to participate in them. Key employees attend courses of training made available by equipment manufacturers. Opportunities are provided to all interested employees to learn to operate new equipment and vacancies for operators and foremen are, so far as possible, filled by promotions from within the organization. Safe working habits are encouraged and safety tours of the pit are made every month by a Committee of both labour and management representatives.

Production per man-shift (operating personnel only) ranges from 510 to 550 tons.

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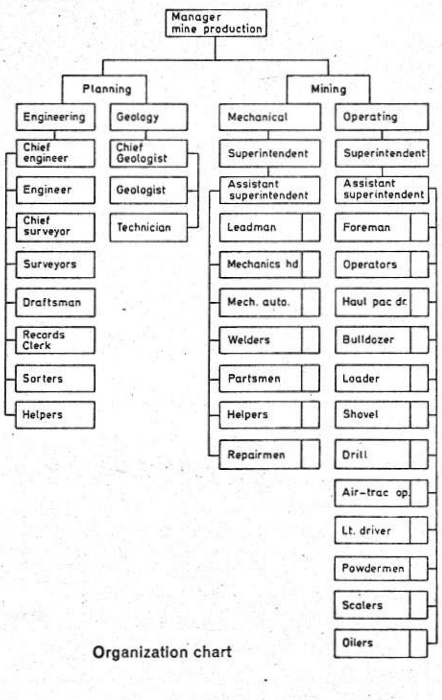

Organization chart