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A temporary use of the land at Highland Valley Copper

Poul Hansen, President, Highland Valley Copper, Vancouver, British Columbia

ABSTRACT

In mid-1986, a world-class mining enterprise was created in Highland Valley, British Columbia, bringing together the copper mining operations of Lornex and Cominco. In 1988, Highmont joined the new partnership.

Highland Valley Copper is the largest copper mine in North America in terms of both tonnage mined and tonnage milled, and it is one of the largest in the world. Due to its relatively low ore grade its production of copper in concentrate ranks about ninth in the Western World.

The current eighteen year mine plan is based on ore reserves of 761 million tonnes. The daily milling rate is 133 000 tonnes.

At the end of mine life, a total area of approximately 6500 hectares will have been disturbed. At this time, vegetation has been established on about 600 hectares and while work will be done during the operating years, it will not be until the end of mine life, about 2008, that the last half of the land can be reclaimed and revegetated.

Once final reclamation has been accomplished, Highland Valley Copper will have completed its "temporary use of the land". In the meantime, about 1250 people have had employment for about 50 000 man years, at wages ranking among the best in Canada.

Introduction

To provide and sustain our way of life, we need steel, copper, aluminum, zinc and many other minerals.

To sustain mining, we must have adequate ore reserves and replacement of reserves as they are mined. Each year the search becomes more difficult as more of the Earth's surface is investigated and deposits are mined. This calls for aggressive and sustained efforts.

In British Columbia, past and present mining activities have disturbed less than one-tenth of one per cent of the land.

With today's technology, mining operations can be substantially reclaimed and thus complete the cycle of "A Temporary Use of the Land".

Keywords: Highland Valley Copper, Land reclamation, Environmental control.

Paper reviewed and approved for publication by The Metal Mining Division of CIM.

Poul Hansen, president of Highland Valley Copper, was born and educated in Denmark, graduating with the equivalent of a Bachelor of Commerce degree from the Mercantile School of Copenhagen. After spending 26 years with a Danish international trading company, Poul Hansen joined Cominco in 1974 where he successively became chief executive of a utility company, group vice-president for fertilizer operations, chairman of Fording Coal and finally, chief executive of Cominco Europe with mining operations in Europe and South Africa. In 1986, he became president of Highland Valley Copper.

Mr. Hansen is a member of the British Columbia Round Table on the Environment and the Economy, the Vancouver Foundation's Environmental Advisory Committee, Transport Canada's Industry Government Advisory Forum, and is a director of the B.C. Children's Hospital and Junior Achievement of British Columbia.

Highland Valley Copper

Highland Valley Copper was created in mid-1986 by bringing together the Highland Valley mining operations of Lornex Mining Corporation Ltd. and Cominco Ltd. into a new single entity, structured as a partnership.

On the south side of the valley was the Lornex mine which started mining in 1972. In 1981, the Lornex concentrator had been expanded to become one of the largest in the industry.

On the north side was Bethlehem Copper which started mining in 1963. In 1981, this operation was absorbed by Cominco who already owned the Valley orebody located west of the Lornex pit on the south side of the valley. Mining of the original Bethlehem Copper pits ceased in 1982.

The Highmont mill on the south side of the valley was acquired in 1988 when Highmont joined. This mill had been closed down in 1984 when the Highmont copper/molybdenum deposit became uneconomical.

Lornex was wound up at the end of 1988 with the result that Rio Algom Limited, Teck Corporation and Highmont Mining Company obtained direct participation in the cash flow from the partnership. Today's participation in the cash flow is:

50%	Cominco
33.6%	Rio Algom
13.9%	Teck Corporation (including 2.5% from Highmont)
2.5%	Highmont (excluding Teck's 2.5%)

The Partnership

Briefly, the Lornex concentrator had three lines, two using 32-ft diameter SAG mills, and one with a 34-ft SAG mill. Highmont had two 34-ft autogeneous mills.

The **first** plan to take advantage of the merged assets, was to build a crushing and conveying system with two semi-mobile in-pit 60 in. by 89 in. crushers situated in the Valley pit and conveyors from there to the Lornex concentrator.

The crushers feed a two-line 60 in. conveying system capable of a combined 12 000 t/h; the system consists of four main overland conveyors, each 1.25 km long, arranged in two parallel flights.

The project was completed in December 1987.

The **second** plan involved the transporting of the Highmont milling facilities 10 km down the mountain to a new site next to the existing Lornex mill. This project was completed in 1989 and the Bethlehem mill was closed permanently.

The new concentration complex is called the Highland mill. The five grinding lines are operated from a central control room, most of the functions being computer-steered.

Seeding Bethlehem tailings area.

Tailings pond.

From the earlier combined daily milling rate of 120 000 tonnes for the Lornex and Bethlehem concentrators, it is now being operated at 133 000 tonnes and the daily mining rate has increased to about 275 000 tonnes.

Twenty-six Caterpillar 789 190-short ton mechanical drive haulage trucks were purchased and the twenty-two Unit Rig Mark 36 170-short ton trucks were rebuilt. This has given a uniform haulage truck fleet.

The trucks are controlled by a computer-driven dispatch system which together with state-of-the art weighing devices on the shovels gives maximum productivity controls in the mining operations.

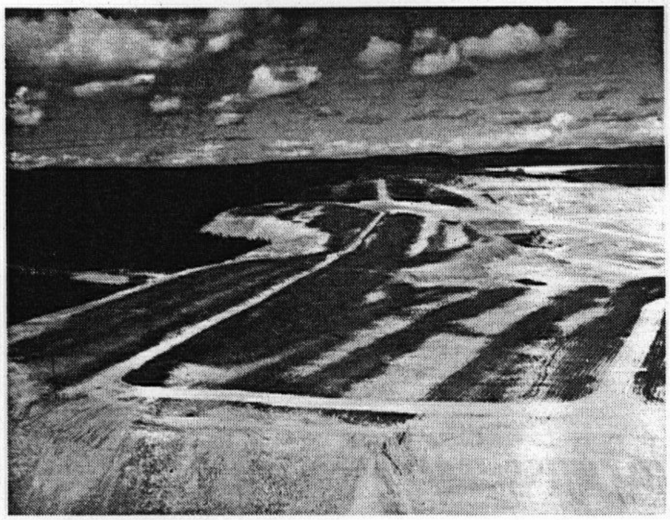
More recently, two new 33 cu yd electric rope shovels were commissioned to replace four small shovels.

Scale of Operations

The current 18-year mining plan is based on ore reserves of about 761 million tonnes at 0.42% copper and 0.006% molybdenum and an average stripping ratio of 0.8.

The annual production will average 160 000 tonnes of copper in concentrate and 4.6 million lbs molybdenum for the first half of the nineties.

The complex is now operating with 1225 total personnel, including management, administration and marketing, compared to more than 1300 required to run the Lornex and Valley operations in early 1986 — and the copper contained in the concentrate has increased by more than 20%.



Reclaimed Highmont waste area.

Highland Valley Copper is the largest copper mine in North America in terms of both tonnage mined and tonnage milled, and one of the largest in the world. However, due to its relatively low ore grade, its production of copper in concentrate ranks about ninth in the Western World.

Markets

On an annual basis, Highland Valley Copper ships over 400 000 tonnes of clean, high grade copper concentrate 40 km by truck to the railhead in Ashcroft and from there, by rail 325 km to North Vancouver. The concentrate is committed to copper smelters in Japan and other Pacific Rim countries as well as to Europe.

Molybdenite concentrate is shipped to conversion plants in Canada and Europe and oxide products are sold to steel mills in Canada and Japan.

Environment

At the time of closure, a total area of approximately 6500 hectares will have been disturbed. Right now, vegetation has been established for one or more years on about 600 hectares and while much work will be done during the operating years, it will not be until the end of mine life that the last half of the land can be reclaimed and revegetated. Starting with about 100 hectares of flat, arable land, this will have expanded to about 3500 hectares at the end of mine life.

Highland Valley Copper has deposited guarantees with the government of over $10 million for the eventual reclamation.

Once final reclamation has been accomplished, Highland Valley Copper will have completed its "Temporary Use of the Land". In the meantime, an average of about 1250 people have had employment for about 50 000 man years, at wages ranking among the best in Canada.

Addendum

This paper was given at CIM's Annual General Meeting on April 30, 1991. The figures quoted have since changed to: 17-year mine plan based on ore reserves of 692 million tonnes at 0.414% copper and 0.0069% molybdenum; 31 Caterpillar 789, 190-short ton mechanical drive haulage trucks and no Unit Rig Mark 36 170-short ton trucks; 1200 total personnel.

Mining at Highland Valley Copper

A. David MacPhail, Manager, Mining, Highland Valley Copper, Vancouver, British Columbia

ABSTRACT

Highland Valley Copper is a large volume producer of copper and molybdenum concentrates. Production is by conventional truck and shovel operation combined with an extensive ore handling in-pit crushing and conveying system.

Grade of the two orebodies is low at 0.4% copper and 0.007% molybdenum and this demands a production rate in excess of 260 000 tpd at a stripping ratio of 1:1. Mining cost has to be kept at minimum levels to keep the mine profitable and thus, it is necessary to keep in the forefront of technology. As well as the in-pit crushing system, the mine has pioneered the use of shovel weighing coupled with a state-of-the-art dispatch system.

Maintenance of the equipment is carried out at the on-site mine maintenance shop with assistance from local shops for engine and wheelmotor rebuilds and other large jobs such as those requiring stress relieving. Shovel and truck rebuilds are undertaken by mine crews. Both long- and short-term planning is designed and scheduled in the mine engineering section. Outside consultants are used to assist in slope design, reclamation and blasting technology.

Introduction

Highland Valley Copper is a partnership between Cominco Ltd., Rio Algom Limited, Teck Corporation and Highmont Mining Company. Initially the partnership was formed to amalgamate the Valley Copper orebody with the Lornex Mill to the mutual advantage of both founding partners. In 1988 Highmont Mining Company joined Highland Valley Copper and brought with it the Highmont holdings within the valley.

The partnership is managed by a committee of six, three of whom are appointed by Cominco and three by Rio Algom.

Highland Valley Copper operates two distinct mines, the Valley mine and the Lornex mine and between the two has measured and indicated ore reserves of 761 million tonnes of 0.408% copper and 0.0072% molybdenum. In addition, the JA Zone orebody, which was included in the last reserves of the Bethlehem Copper Corporation before that company was purchased by Cominco, lies within the boundaries of the partnership. Based on current plans, which ignore the JA Zone, the property has a life of approximately 18 years at conservative metal prices.

Mining is carried out in the two mines simultaneously at a proportion of 80% in the Valley mine and 20% in the Lornex mine, and the ratio is projected to remain much the same over mine life.

Keywords: Highland Valley Copper, Mining strategies, Mine planning, Copper, Molybdenum, Valley mine, Lornex mine.

Paper reviewed and approved for publication by the Metal Mining Division of CIM.

A. David MacPhail received his mine engineering degree from the University of the Witwatersrand, South Africa, in 1958. After working some years in the South African gold mines, he immigrated to Canada in 1964. He joined Cominco Ltd. in 1965 and has held various supervisory positions with that company, both in Canada and Europe.

In 1980 he was appointed development manager of Valley Copper and participated in feasibility studies on the Valley deposit and was responsible for the in-pit crushing and conveying concept developed at that time for the valley orebody. In 1983 he was appointed general manager of the Valley Copper Mine and remained in that capacity until the formation of the Highland Valley Copper partnership in 1986. Since then, he has been manager, mining, for Highland Valley Copper.

Mining Strategy Factors

The economics of mining have been, and are, governed by several parameters;

1. the geology of the deposits;
2. the location of the orebodies relative to the mill;
3. the low grade of both mines;
4. the equipment brought to the partnership at its formation, and the new equipment subsequently purchased;
5. water content of the valley overburden; and
6. reclamation of the disturbed area at the end of mine life.

The Geology of the Deposits

Both the Valley and Lornex deposits originate from the Guichon Creek Batholith but from different phases. The Valley deposit is the younger and was mineralized during the Bethsaida phase while the Lornex deposit was mineralized during an intermediate intrusive phase. Mineralization in both mines is bornite and chalcopyrite with a higher ratio of bornite to chalcopyrite in the Valley mine and the reverse in the Lornex mine. Molybdenum sulphide mineralization is higher in the Lornex mine than in the Valley. Gold and silver are present in small quantities which become significant in the copper concentrate.

The Valley deposit is found on the southern lower slope of the Highland Valley and is covered by glacial till. The thickness of the till increases from 20 metres over the southwestern boundary to 200 vertical metres in the northeast pit slope. The shape of the orebody is roughly circular. Jointing is marked, with two predominant azimuths, one almost parallel to the southwestern highwall and the other north-south. Mineralization is coarse and occurs along a stockworks of quartz veins. Molybdenum mineralization forms a halo around the orebody and copper values increase sharply across the boundary to above 0.3% from < 0.15%. Generally ore within the 0.3% threshold remains between 0.3% and 0.5% Cu.

The Lornex orebody is dominated by the Lornex fault which parallels the long axis of the mine, dipping west, and currently has an exposed trace along the western side of the pit bottom. Thus ore remaining on the lowest benches (12 million tonnes at 0.46% Cu) has to be mined from an ever-decreasing area and will be mined out by 1993.

The remaining Lornex reserves are in a sub-pit along the southern and eastern sides of the mine. The strip ratio of this sub-pit is 0.83:1. Grades improve with depth to reach a high of 0.38% Cu. Molybdenum grades are in the range of 0.017% to 0.008%.

Both mines have pit slopes that are subject to tilting failures through ground water in structures running parallel to pit axes.

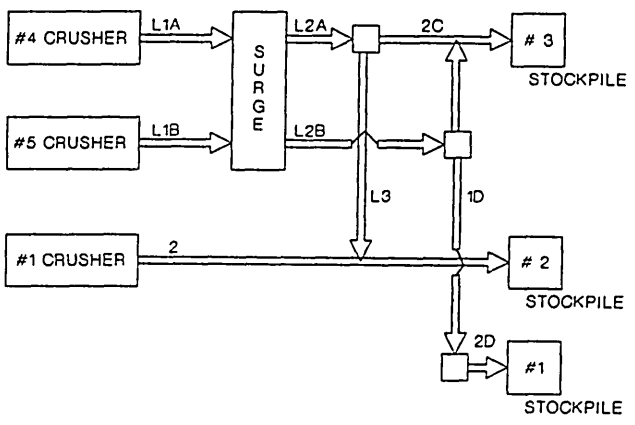


FIGURE 1. Ore flow: crushers to stockpiles.

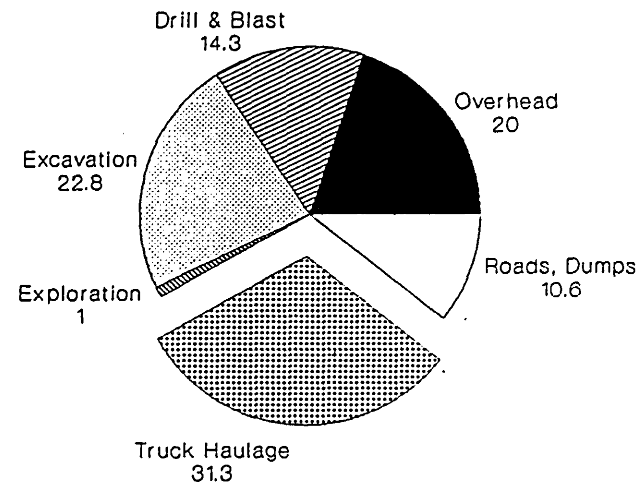


FIGURE 2. Cost centres — percentage of mining cost.

The Location of the Orebodies Relative to the Mill.

The Highland mill ore stockpile discharge is built at an elevation of 1360 metres above sea level, and was sited to process the ore from the Lornex orebody. The crusher to which ore from the Lornex mine is delivered is 650 m from the final pit rim. However, the Lornex mine has been in operation since 1972 and the pit is now deep, requiring an elevation rise of 167 m for the ore trucks.

The Valley mine rim is 160 m below the discharge and 3 kms away. The present bench elevation of the pit bottom is 125 m below the rim and the final bench is designed to be at the 725 m elevation, or 635 m below the mill stockpile discharge. Transportation at present represents 31% of the mining cost and is the major cost centre. Obviously control of transportation expenses is vital to maintaining affordable costs.

The Low Grade of Both Mines

The ore reserves of each mine are:

- Valley mine 627 million tonnes @ 0.418% Cu and 0.0056% Mo;
- Lornex mine 135 million tonnes @ 0.364% Cu and 0.0144% Mo;

Ore reserves are calculated at an equivalent cut-off grade of 0.25% Cu using a molybdenum multiplying factor of 3.5.

Both mines are divided into four grade zones numbered from 1 to 4 with grade ranges of:

Zone 1	> 0.56%
Zone 2	0.36 - 0.56%
Zone 3	0.16 - 0.35%
Zone 4	< 0.16%

It is rare to find grades above 0.6% Cu in any part of the orebodies and the grade mined in the years of operation are:

1986	0.405%
1987	0.435%
1988	0.471%
1989	0.425%
1990	0.429%

With grades of this magnitude it is imperative that the mine use the most efficient mining methods in order to remain competitive.

The Equipment Brought to the Partnership at its Formation, and the New Equipment Subsequently Purchased.

Both Cominco's Valley Copper and Lornex were operating mines at the formation of the partnership. Valley Copper was a relatively small mine with corresponding equipment acquired from Bethlehem Copper; while Lornex was a large mine with equipment to match. Both mines possessed equipment which had seen much use and some which was fairly new. Inevitably the small equipment did not fit the new demands for efficient operation and some of the larger equipment was costly to run. $72 million has been spent on new shovels, trucks and support equipment between 1986 and 1990. Some trucks and shovels have also been extensively overhauled.

Water Content of the Valley Overburden

The Highland Valley is typical of glaciated valleys of British Columbia in that it is gouged out into a V with a large quantity of glacial till to be found in the valley floor while relatively little is found on the flanks. The glacial till has horizons which form aquifers where coarse boulders predominate and aquacludes of silty material. Pre-mining drilling established that there were at least three aquifer horizons within the overburden which would have to be mined during mine life. Subsequent work has defined these horizons in more detail and confirmed that water removal from the overburden is essential for proper slope stability on the northern and eastern slopes of the Valley mine where the mined overburden thickness will be 200 m.

Reclamation of the Disturbed Area at the End of Mine Life

Pits, waste dumps, mill site, tailings and roads will disturb 4286 hectares (9.5 square miles), nearly all of which must be reclaimed to acceptable standards. Reclamation of the area over which mining and milling operations is currently taking place is governed by two reclamation permits. Permit M 11 covers the area disturbed by the former companies Lornex, Valley Copper and Bethlehem; permit M 55 covers the area disturbed by Highmont. While the areas which can now be reclaimed are small, planning and positioning for the ultimate phase is a current activity.

There are other factors which affect the mining strategy, but those listed above are the main ones. These are woven into the plan in the following way.

Mine Planning

The Lornex mine had been planned to the end of its life before the partnership was formed. As the mining was approaching the final sub-pits the effect of the partnership was to slow down development but keep the mining sequence essentially the same as had been planned by Lornex. Mining was taking place roughly

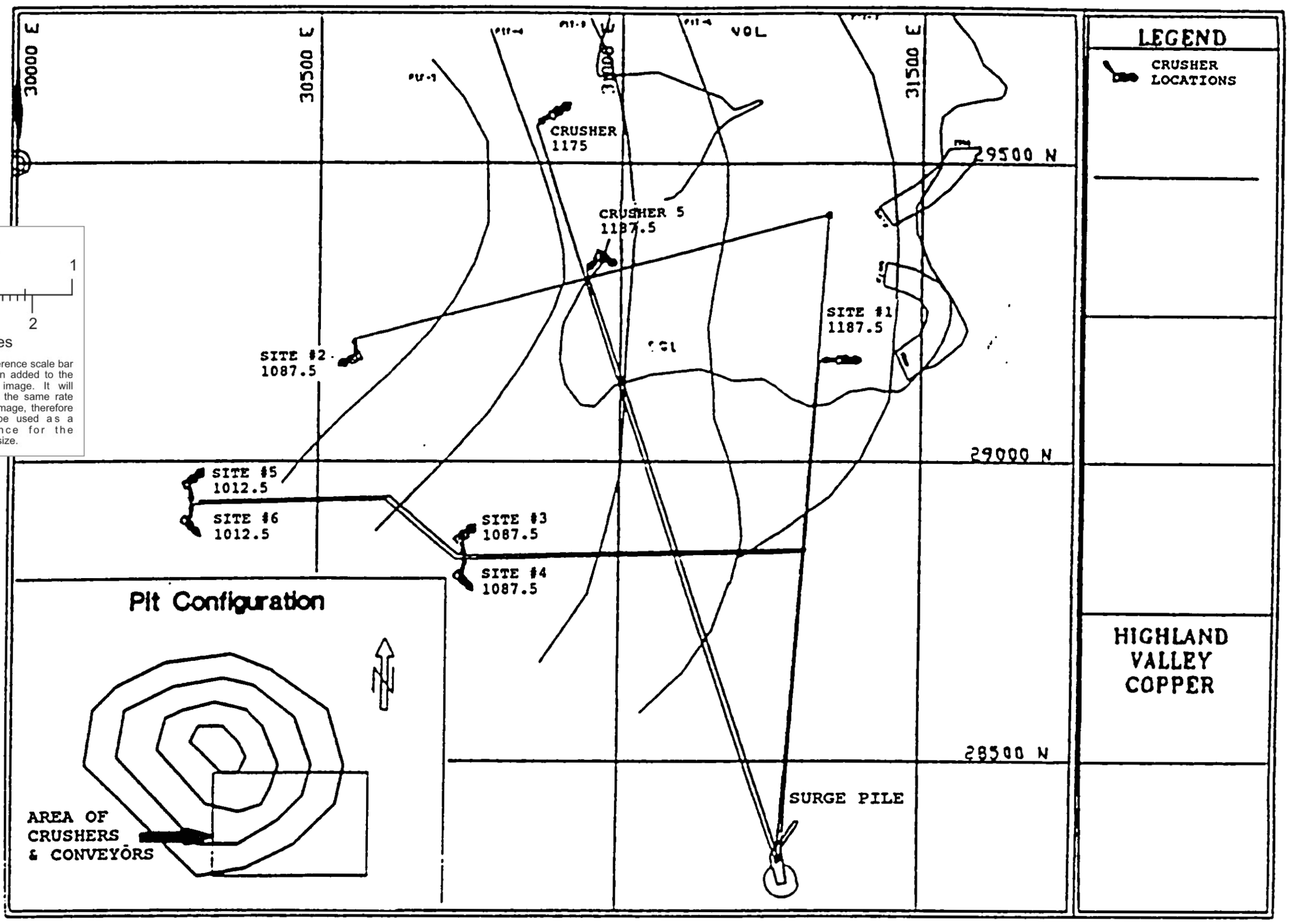


FIGURE 3. Locations and sequencing of crusher moves in Valley pit.

in the centre of the deposit at the lowest elevations of the developed sub-pits. One more sub-pit remained to be developed, pit 10. Currently mining has started in pit 10 but it will be two years before this pit becomes a constant ore producer. The main contribution of the Lornex mine comes from the low strip sub-pits at the lowest elevations of the mine.

The Valley mine was re-planned in 1986 at the time of formation of the partnership because of a different orientation of ore handling required to the Highland mill, because of the big increase in ore production required and because of the very different equipment available. The block model was recalculated and floating cones at 60, 70, 80 and 90 US cents were run. The decision was made to place the ultimate pit close to that delineated by the 80 US cent floating cone and, because the orebody is circular without any markedly higher grade areas, the mine is planned in seven roughly concentric pits each subdivided into several sub-pits. As noted in the earlier section on the geology of the Valley deposit, the grade of ore in the centre is all above 0.3% copper with patches grading up to 0.6%, but this cannot be described as a high grade core.

With the orebody located some distance from the mill, a conveyor belt system was planned to convey the ore from the pit rim to the already existing mill stockpiles.

A schematic of the system, which is designed to convey ore at a maximum rate of 6000 t/h on each belt, is shown in Figure 1. The belts are designed to deliver ore at twice the nominal mill capacity.

Some flexibility is possible in ore handling through the cross-over drawpoints at the surge pile. All three mill stockpiles can be fed from the Valley mine, while Lornex ore can only be conveyed to No. 2 stockpile.

The rationale for constructing the overland conveyor system is the cost savings in operating conveyors as opposed to truck haulage. A post construction audit has confirmed the wisdom of the conveyor choice, proving a payback of 3.3 years.

A breakdown of cost centres for mining shows that transportation of ore and waste forms the major part of the unit cost (Fig. 2).

It therefore follows that if the cost of haulage can be kept constant as the mine deepens then the mine has an opportunity to maintain a constant unit mining cost. Following the logic of using the overland conveyor system and augmented by a study published prior to the formation of the partnership[1], Highland Valley Copper has chosen to use movable in-pit crushers and conveyors for ore transportation in the Valley mine, connected to the surface conveyor belt system. The geology of the orebody is such that a parallel system for waste rock removal from depth is not considered necessary. The advantages of this system are not yet fully apparent as the crushers are close to the original surface elevation. Some saving in haulage cost have been realized in that the crushers are within the ultimate pit and haulage from ore centres are thus reduced by 400 m in distance and about 30 m in elevation. Using theoretical haulage profile calculations this amounts to an approximate saving of 3.7 cents per tonne or \$3.8 million for the 103 million tonnes of ore sent to the Highland mill by the end of 1990.

The in-pit crushing and conveying system is planned to be moved six more times. Currently the two crushers dumps are lo-

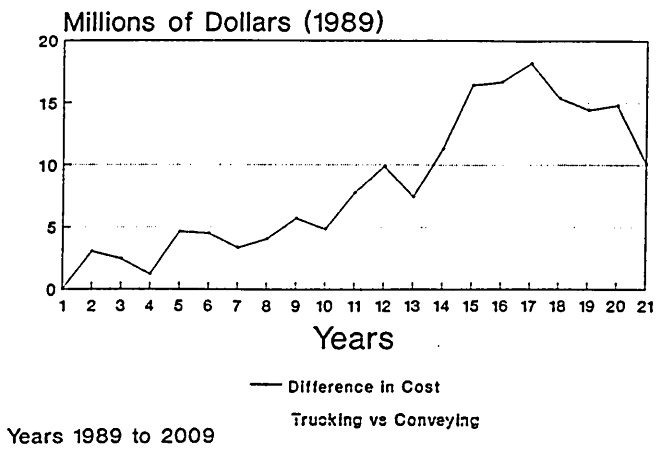


FIGURE 4. Savings in annual operating cost.

cated at elevations of 1200 m and 1187 m and are 500 m apart along the haulage road. The next move is scheduled for the latter part of 1992 and will move No. 4 crusher to the 1100 m elevation. No. 5 crusher will remain on the 1200 m bench but 400 m further east from its present position. Figure 3 shows the planned locations and elevations.

In a follow-up study carried out in 1989 it was estimated that the savings in transportation cost by using the crushers in pit rather than at the pit rim was $181 million (1989 dollars). Figure 4 shows the calculated savings.

Slope Stability.

The east and west walls of the Lornex mine have historically been affected by a tilting mode of failure, which has shown time dependent characteristics. As the mining of the east wall is slow the inter-ramp angle to which the mine has been planned is 35 degrees. This low angle is partly due to the lack of success in dewatering the altered Skeena quartz diorite that comprises most of the wall. The west wall, most of which is Bethsaida granodiorite, is more easily depressurized by horizontal drill holes, and in the Bethsaida areas the wall is designed at an inter-ramp angle of 37 degrees, flattened to 33 degrees when the wall is over 200 m high. Lower down in the wall, the 150 m wide Lornex fault requires a flat angle of 20 to 25 degrees.

Water in the structures is the cause of the tilting failures and to keep the rock walls under control horizontal drill holes are drilled from the pit benches to intersect and drain the water bearing faults and slips. A few new holes are drilled each year, installed with plastic drain pipe and left to drain. Piezometers set at various depths are used to record water levels which show that spring run off has a marked but delayed effect. Horizontal holes are drilled to anticipate the peak readings in May/June each year.

In the Valley mine most of the excavation will be in Bethsaida granodiorite except that on the north and east sides the upper benches excavation will be in overburden to a maximum height of 200 metres. Two almost parallel structures, the Lornex fault and the Osprey shear zone, both striking north-south, provide wall sections where flat angles are required (20 to 30 degrees). Inter-ramp angles in rock vary from 39 degrees on the southwestern highwall to 53 degrees in the southeast corner where the wall is at right angles to the predominant structure. Pre-shearing is being experimented with on the highwall with results inconclusive so far. A tilting failure is evident on the western highwall where water is associated with a highly oxidized zone. Attempts at depressuring the water zones have been partially successful.

In the overburden, dewatering is obligatory to maintaining an inter-ramp slope angle of 32 degrees. The aquifers are being dewatered by 18 vertical wells which together have produced around 4000 l/min. for the last six years. Piezometer readings are obtained from widely spaced drillholes, all of which indicate a steady lowering of the water table.

Dump Development and Reclamation

Mine planning is closely involved in dump development to meet the demands of the reclamation permits. Different solutions are required for specific areas of the property. On the Bethlehem dumps experimentation has shown that a suitable growing material must be hauled to the dump which has been prepared for cultivation, sometimes by resloping or by scarifying the dump surfaces. Mining of overburden in the Valley mine is correlated with reclamation demands and rehandling of suitable material is avoided as it is very expensive. The permit demands that 95% of the Bethlehem slope areas be revegetated excluding the high dumps facing the Highland Valley road which are exempt from vegetation requirements.

The Lornex dumps have proven to be made of material which breaks down sufficiently to allow vegetation to grow on a scarified surface. As mining is still being done in the Lornex mine the waste from pit 10 will be used to build smaller dumps in front of the long slopes and aim for slope heights not exceeding 45 m the maximum slope considered practical for proper revegetation.

Under the terms of the current M 55 permit, the Highmont dumps have to be revegetated to sustain tree growth and will require a metre depth of suitable growth material.

Study of the reclamation requirements has been intense over the past year and every effort is being made to incorporate practical solutions within the current mining practice. This is easier on the Valley mine dumps as they are partly made of overburden material. Scheduling the placement of suitable material has been incorporated into the dump designs. Much work remains to be done to ensure an acceptable surface water drainage system after the mining is complete; tests indicate that the Valley mine rock is unlikely to produce acid drainage but nitrate leaching must be kept within acceptable bounds.

Mining Equipment

At the time of formation of the Partnership mining equipment consisted of:

Trucks: 22 Mark 36 Unit Rigs (154 t capacity)
21 M 100 Unit Rigs (91 t capacity)
11 Wabco 3200Bs (209 t capacity)
17 Wabco 120Bs (108 t capacity)
Shovels: 3 BE 295B1 with a 16.8 m^3 dipper.
1 BE 280B with a 11.5 m^3 dipper.
4 BE 195B with a 10 m^3 dipper.
4 P&H 2100 with a 11.5 m^3 dipper
1 P&H 2300 with a 16.8 m^3 dipper.
Production Loaders: 2 992s
2 Dart 600s
Drills: 8 BE45Rs
Dozers: 8 D9H and D9L; 3 D8L; 8 824 RT
Graders: 10 16G

During 1987, the first full year of the partnership, 81.4 million tonnes were mined which was 86% of the plan. The shortfall was mainly in waste removal in the Valley mine. Truck availabilities were disappointingly low at 68.8% and the cost high.

A study was initiated to examine the cost effectiveness of the truck fleets and concluded that the purchase of new trucks in the 172 t capacity range was warranted. In all, twenty-six 172 tonne Caterpillar 789 trucks were purchased and all M 100 Unit Rigs, Wabco 3200Bs and Wabco 120Bs were retired. With the twenty-

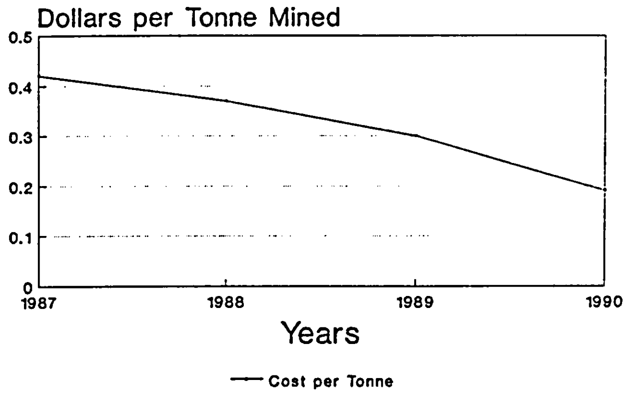


FIGURE 5. Cost per tonne hauled.

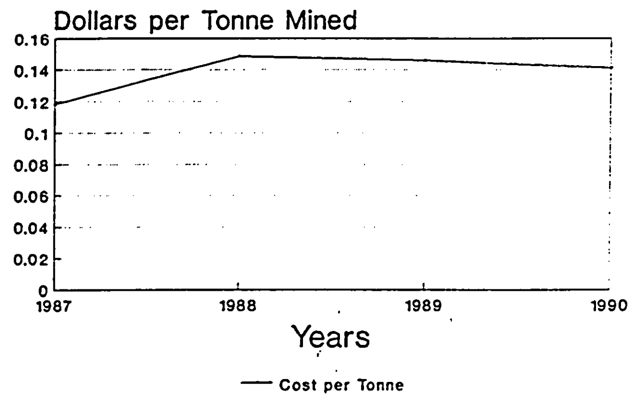


FIGURE 6. Cost per tonne excavated.

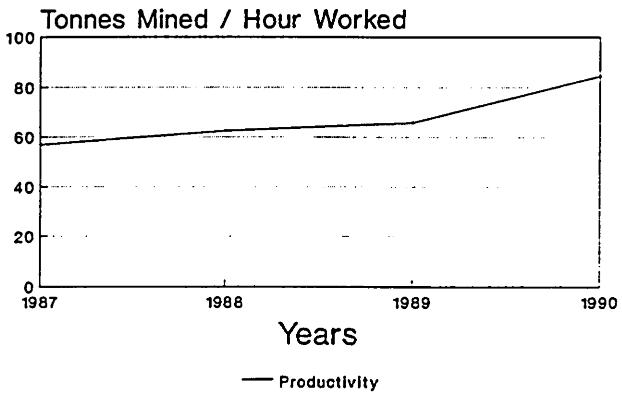


FIGURE 7. Productivity.

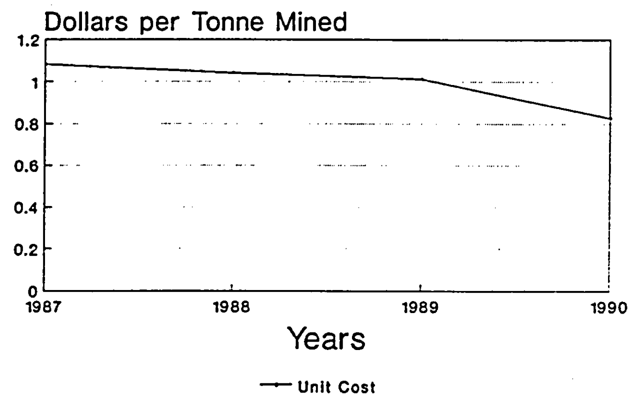


FIGURE 8. Mining cost per tonne mined.

two M 36 Unit Rigs still in service, the mine now operates with 48 trucks, mining an average of 260 000 t per calendar day.

Figure 5 shows the decline in cost of haulage over the past four years. Not all of the decline can be attributed to new equipment but it is a significant factor. Other influences have been the overland conveyor system, commissioned in late 1987; the closure of the Bethlehem mill in mid-1989; the installation of the truck dispatch system and shovel weighing system, both commissioned in 1990.

In August 1988 a shovel study concluded that low productivities, high maintenance costs, advanced age of some shovel fleets and inappropriate size of smaller shovels compared to the size of trucks in the haulage fleet, warranted the purchase of two 33 cu. yd capacity shovels. Three BE 195Bs were retired. Extensive overhaul is now underway on most of the remaining older shovels.

Figure 6 shows the effect of the shovel purchases and current overhaul policy.

In addition to the above-mentioned purchases of new equipment, two other innovations have assisted in raising the productivity.

In March 1990 a truck dispatch system was commissioned. Dispatch is a large scale , computer based system which controls the dispatching of all haul trucks, maximizing mining productivity. Comprehensive reporting features provide detailed mine operating information and time accountability. The most promising result of truck dispatch is to allow mining management by first- and second-line supervision through the provision of up-to-the-minute information to compare against shift goals. At Highland Valley Copper further improvements are expected as first- and second-line supervision become trained in dispatch use.

At the same time as dispatch, a shovel weighing system, which weighs each bucket load and allows the rated tonnage to be loaded onto each truck, was purchased and incorporated into the dispatch system. The shovel weighing system has achieved a high reliability and accuracy, as verified by digital scale readings of loaded trucks. It is installed on five shovels and plans are underway to equip another five over the next two years.

Productivity and mining cost provide a measure of the success of the strategy developed at Highland Valley Copper. History of those parameters is shown in Figures 7 and 8.

Mining in low grade orebodies provides challenges to the mining engineer to keep the mine competitive in the expanding world copper mining industry. At Highland Valley Copper the search for chances to use improved technology is ongoing. Impending advances of interest include steep angle conveyors of high capacity, improvements in in-pit crusher performance and the evolution of bigger and more cost effective machines.

REFERENCE

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HIGHLAND VALLEY COPPER

Blast optimization and stability enhancement through geotechnical mapping at Highland Valley Copper

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ABSTRACT

Poor fragmentation, high toe areas, excessive shovel wear and tear and frequent re-drilling and blasting of certain tough waste rock zones in the Lornex pit in the early 1980s necessitated a better understanding of these zones.

A combination of geotechnical mapping, penetration rate data, toe elevation contouring and observations on shovel and drill performance were used to determine the character and extent of these zones. This complementary information was brought together by the geotechnical mapper and the explosives supervisor.

Distinct zones with particular geological characteristics were determined. A classification system with three types of blast and dig-ability was developed for the Lornex pit and later applied to the Valley pit, with considerable success. The three blast zone types are called "poor", "fair", and "good". This system will be described in detail.

The explosives supervisor adjusted the blasting parameters to each zone, in particular, pattern spacing and burden. The resulting powder factors showed considerable variation between the poor, fair and good zones within each pit as well as between the two pits. Pattern geometry and length of collar were also adjusted to improve fragmentation.

Cost savings resulted from less re-drilling and less wear and tear on drills and shovels in the poor zones and more efficient use of explosives in all the zones. An annual estimated cost savings of well in excess of $1,000,000 is realized at Highland Valley Copper. A by-product of the mapping was the determination of unstable structures and water flowing zones. Blasting was then adapted to this information for enhanced slope stability.

Keywords: Highland Valley Copper, Blast optimization, Stability enhancement, Geotechnical mapping.

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Introduction

The Highland Valley Copper mines are located in southern British Columbia, about 300 km east-northeast of Vancouver and 80 km west of Kamloops (Fig. 1).

In 1986, Lornex Mining Corporation Ltd. and Cominco Ltd. joined their Lornex and Valley Copper operations to form the Highland Valley Copper partnership. Production now came from both pits and was milled in both the Bethlehem and Lornex concentrators. By early 1989, the Lornex mill was expanded by the addition of the Highmont mills, to 133 000 t/d and the Bethlehem mill was shut down.

Presently, about 75% of the daily ore production comes from the Valley pit and 25% from the Lornex pit. Total production from the two pits is now about 275 000 t/d including both ore and waste. Total monthly blasting now averages about 6.7 million tonnes of muck, and our target pattern size is 170 blastholes per shot. Bench heights are 12.2 m at the Lornex pit and 12.5 m at the Valley pit.

Geology

Lornex Pit

The Lornex pit is situated on a major contact between two phases of the Late Triassic-Early Jurassic Guichon Creek Batholith (McMillan, 1985) (Fig. 2). The north-south Lornex Fault forms the sharp contact between the youngest Bethsaida phase to the west and the Skeena variety of the Bethlehem phase to the east. The Fault dips steeply westerly from 65 to 85 degrees, and occurs as a zone of intense fracturing and faulting up to 80 m in width.

Skeena Variety of the Bethlehem Phase

The Skeena variety is a medium-grained porphyritic granodiorite and it forms the ore host rock for the Lornex porphyry copper deposit. It is pervasively and intensely fractured and argillicly altered. Hydrothermal alteration occurs as roughly concentric, though elongated, propylitic, argillic, phyllic and K-spar zones. Argillic alteration mainly determines the uniaxial compressive strength of the rock. Fresh to weak argillic or strongly silicified rock exhibits up to 23 000 psi strength whereas intensely argillic rock can have a strength as low as 2000 psi (Wood, 1984).

The Skeena granodiorite exhibits a greater range in rock strengths than the Bethsaida quartz monzonite.

Bethsaida Phase

The Bethsaida phase is totally waste in the Lornex pit and is stripped to gain access to the Skeena granodiorite ore host rock. This phase is a coarse-grained porphyritic quartz monzonite to granodiorite. It consists of quartz eyes in a tightly interlocking matrix

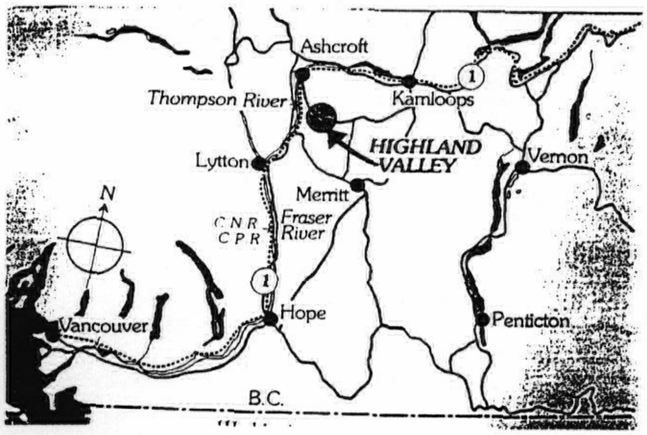


FIGURE 1. Location map of Highland Valley.

of euhedral plagioclase laths, K-spar and biotite 'books'. The strength of this rock normally ranges between 3000 psi in the argillic alteration halo of fault zones up to 18 000 psi or greater in the fresh rock and silicified zones (Wood, 1984; Stacey, 1986). The Bethsaida phase is cut by intermittent strong fault zones dipping moderately east-southeasterly or moderately to steeply west-northwesterly.

These zones consist commonly of thick gouge up to 2 m to 3 m thick and intense sympathetic fracturing with attendant strong argillic alteration (chlorite, sericite and kaolin) haloes. The intervening rock tends to be fresh or weakly argillic with widely spaced vertically and horizontally continuous master joints.

Valley Pit

The Valley pit is situated at present wholly within the Bethsaida quartz monzonite, which forms the ore host rock (Fig. 4). The Bethsaida host rock is bounded on the east by the northerly striking Lornex Fault. In this area the Lornex Fault is near vertical or dipping steeply to the east. The only other rock types in the Valley pit consist of narrow dykes of minor volumetric significance. They are comprised of narrow quartz and quartz feldspar porphyry dykes ranging from 0.3 m to 35 m thick and lamprophyre dykes averaging 4.5 m thick. Most of these dyke rock materials tend to be strong when unaltered.

Similar to the Skeena granodiorite, the Bethsaida phase is also subject to concentric, though circular, hydrothermal alteration zones going from peripheral and vague propylitic, through pervasive argillic, with central phyllic and potassic zones and a strong silicic zone in the southeast end of the orebody. Though statistically concentric, the argillic alteration types tend to occur as elongated haloes along zones of intense fracturing. These zones commonly strike approximately north-south and dip steeply easterly, and display a northerly plunge. Other more local structural trends are north-south fractures and faults dipping steeply westerly, approximately east-west fractures dipping steeply southerly and southeast-striking fractures dipping moderately northeasterly or southwesterly. Once again, the combination of structure and alteration type and intensity largely determines the rock blast-ability in the Valley pit. The eastern side of the pit is overlain by considerable thicknesses (12 m to 214 m) of Pleistocene glacial and Recent overburden.

History of Blasting Problems

The present geotechnical mapping program to determine rock blast and dig-ability zones at Highland Valley Copper was initiated in early 1981 in the Lornex pit (Fig. 2). Certain tough waste rock zones were causing problems high up on the west wall of the Lornex pit, at about the 1497 m bench elevation. Poor fragmentation, high toe areas, extreme shovel wear and tear and frequent re-drilling and

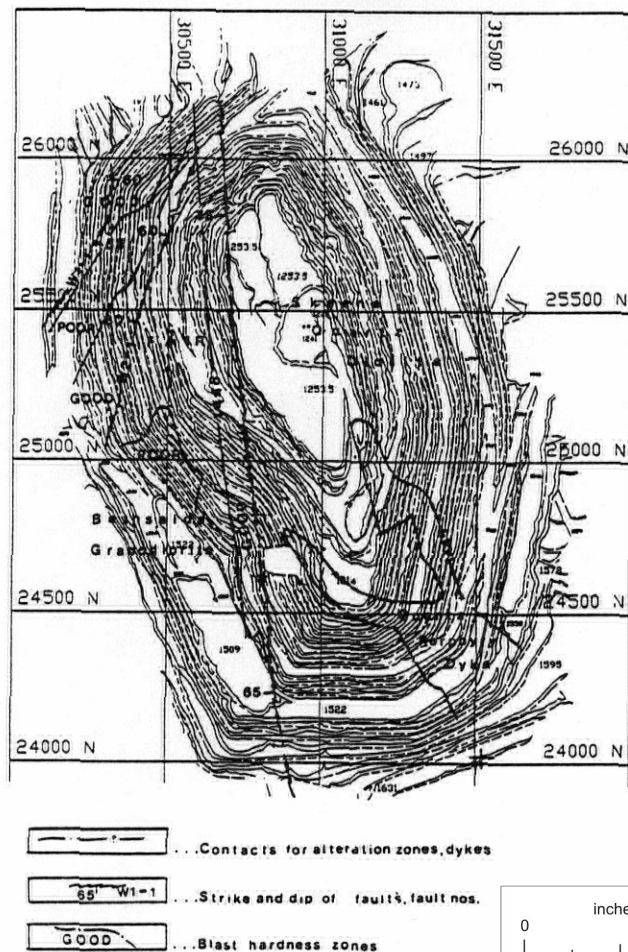
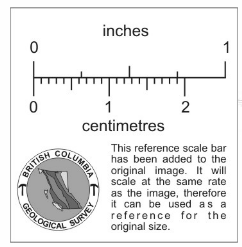


FIGURE 2. Lornex pit — geology.



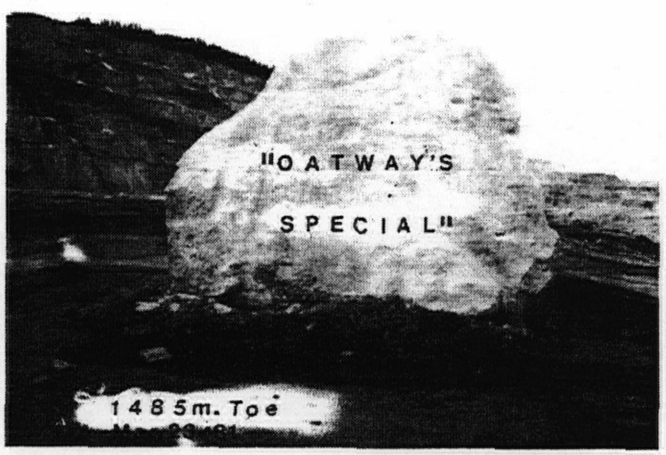


FIGURE 3. The "Oatway's Special" required re-drilling and re-blasting three times.

blasting of these zones led to a request for a geotechnical examination of the recalcitrant rocks. The straw that broke the camel's back was the necessity to re-drill and re-blast one hard knob three times in the southwest corner of the 1497 m to 1485 m bench to break it up (Fig. 3). Adjacent blasted areas displayed very chunky muck in the order of 2 m to 2.5 m diameter or more.

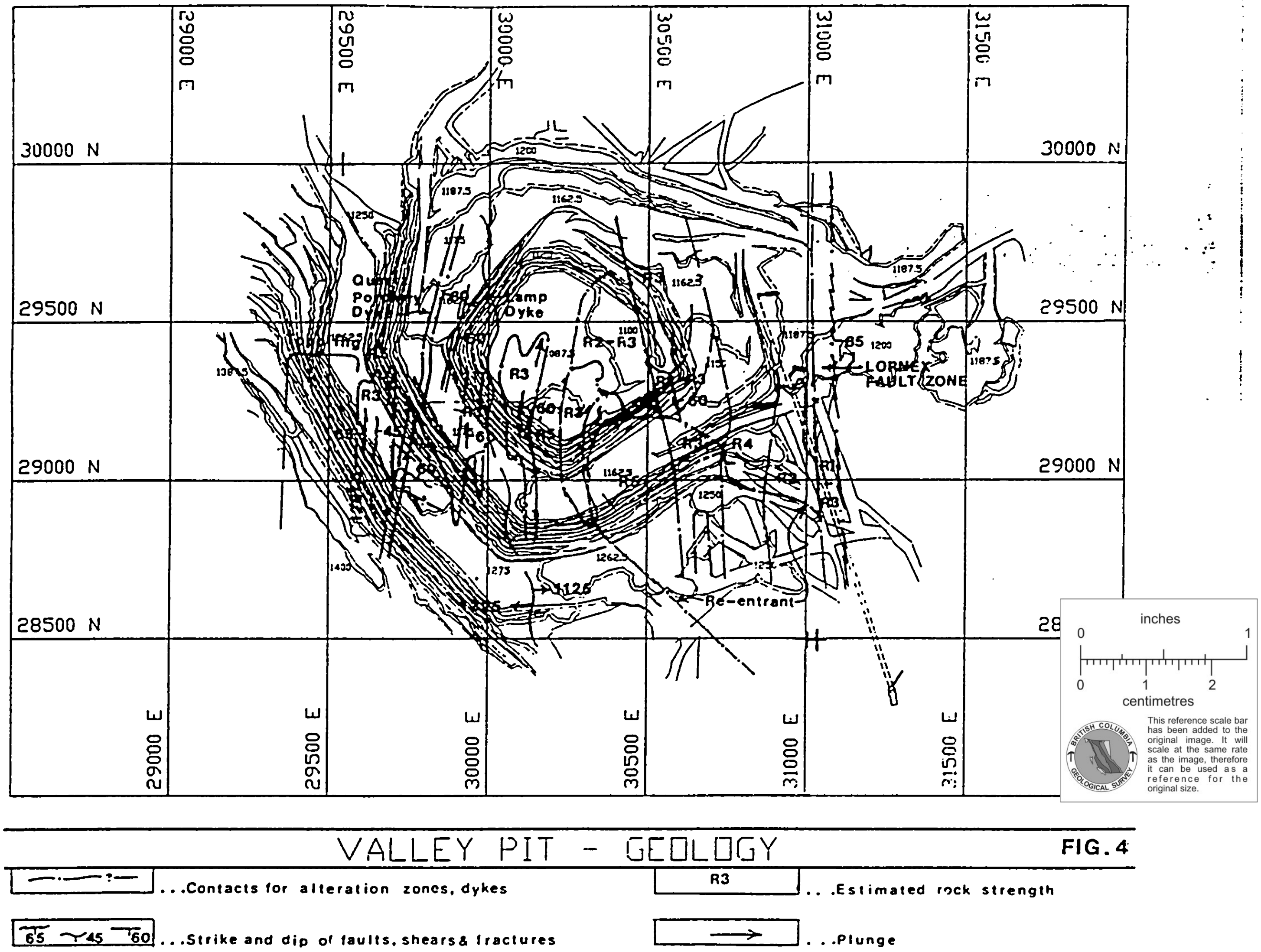


FIGURE 4. Valley pit — geology.

Solutions

Geotechnical Mapping

Mapping of the whole bench began immediately. The initial mapping simply involved recording any observations that might be pertinent to blast-ability, i.e. alteration zones, structural frequencies and orientations and muck sizes and shapes. Patterns began to emerge and it was discovered the rocks could be sorted into three distinct zones of varying blast-ability. On the west wall of the Lornex pit alone, at least 12 km of mapping of active faces and final walls went into the development and refinement of this blast zone system. Another important point is that with the low powder factors at Lornex and Cominco's Valley Copper, it is possible to map the blasted muck accurately as dug because the structures are well preserved and subject to little displacement. This facilitates projecting structures and alteration zones to the next bench with confidence.

Rock Mass — Blasting Zone Classification System

The geotechnical mapping on the west wall of the Lornex pit resulted in classification of the Bethsaida quartz monzonite into the categories *poor, fair,* and *good* (Fig. 5).

"Poor" means tough Bethsaida quartz monzonite, with an R5 (Geological Society of London and Golder Associates R1 to R5 rock strength estimates) field estimated rock strength. R1 is 182 psi to 730 psi while R5 is greater than 29 200 psi. In practice, the R2 range of 1825 psi to 7300 psi is where most of our weaker rocks lie and it conforms to the *good* zone. R3 is intermediate, from 7300 psi to 14 600 psi and conforms to the *fair* zone. Most of our stronger rocks, while called R5, are probably closer to R4 in field estimated strengths.

The Bethsaida quartz monzonite in the *poor* zone can either be fresh to weakly argillicly altered or intensely argillicly altered but intensely silicified (approximately 35% to 40% quartz content at Lornex and 39% to 59% at Valley Copper). Joint spacing is commonly 1.5 m to 6 m and joints tend to be master joints with 12 m or more (bench height) vertical continuity and continuity along strike of commonly 50 m or more. These master joints divide the rock up into large blocks which, when blasted, result in very chunky muck.

"Good" means R2 or R3 rock strength material, usually moderate to intense to intensely argillicly altered Bethsaida quartz monzonite cut by frequent fractures, shears and gouge-filled faults. The intense fracturing and the strong alteration renders the rock soft and friable and blastable into fine muck. It takes much less explosive energy to break this rock than the *poor* zone rock. On the Lornex west wall, these zones are caused mainly by the laterally

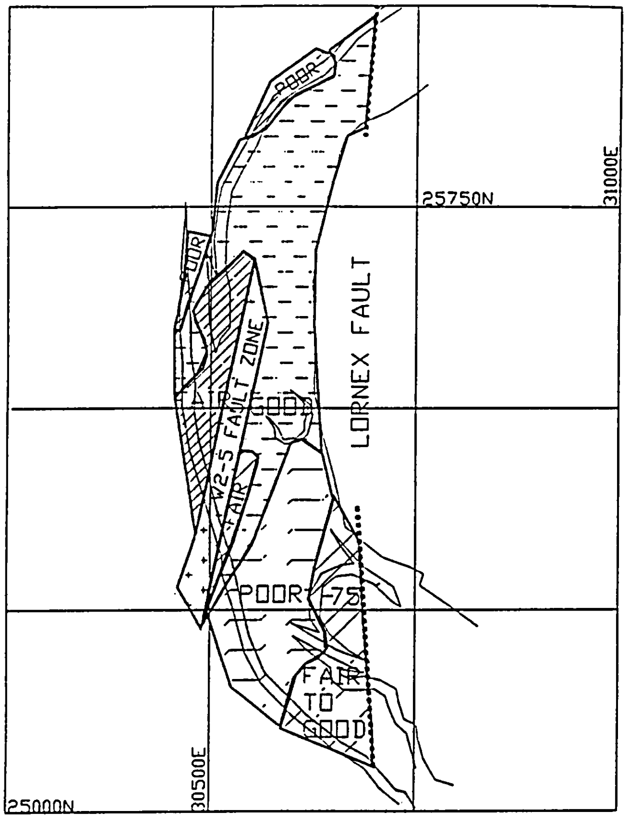


FIGURE 5. Lornex pit 1351 (4432) bench — dig-ability plan.

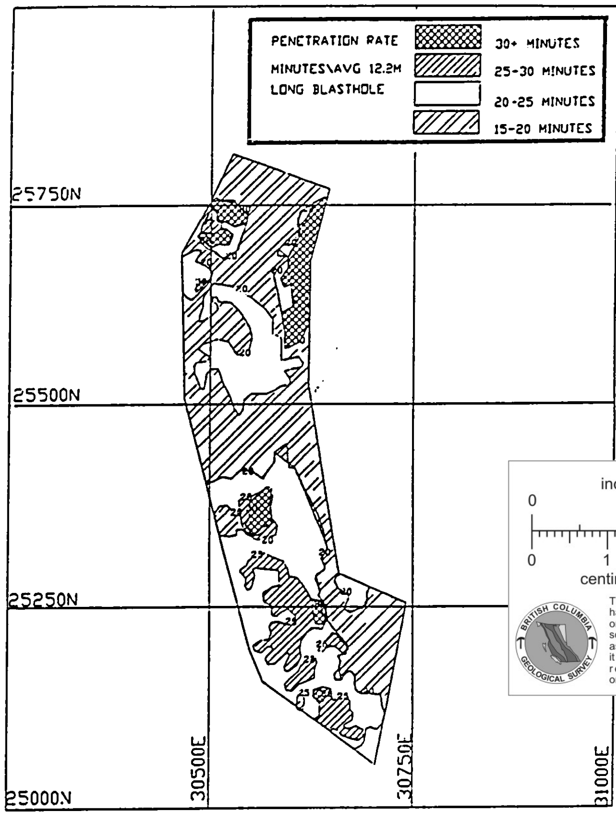


FIGURE 6. Lornex pit 1351 bench — blasthole drill penetration plan.

continuous major gouge-filled faults, striking 010 to 025 degrees and dipping either 55 degrees easterly or 60 to 85 degrees westerly (the W1-1 and W2-1 to W2-5 fault systems). The strong alteration occurs mainly as hydrothermal haloes enclosing the faults.

The third, or intermediate zone, is called "fair" and consists of moderate to moderate-intense argillicly altered Bethsaida quartz monzonite with an average R3 uniaxial compressive rock strength. It is commonly well fractured and yields blocky muck approximately 0.3 m in diameter. This material is also fairly easy to break with explosives.

Paradoxically, the blasting terms *poor, fair,* and *good* are the opposite of their meaning to slope stability. When these terms were first shown to a slope stability consultant, he jokingly said, "You've got it in reverse, because your *poor* rock means *good*, i.e. competent rock to me".

Application of Classification System

The blast zone classification system was later applied with considerable success at Lornex to hard zones in the Skeena granodiorite on the east wall and to the quartz porphyry dyke on the south wall. After the amalgamation of Lornex and Cominco's Valley Copper, this system was applied to various shells of the Valley pit and is part of an ongoing program there now. Originally, some Valley ore went to the Bethlehem crusher. The requirement for less than 15 cm diameter crusher feed meant a powder factor up to 0.5 kg/t was needed to break it up. Needless to say, little mapping was necessary in these areas. When the Bethlehem crusher was shut down and the in-pit crushers came on line in 1987, the powder factors were substantially reduced and geotechnical mapping became of paramount importance again.

Even though the Lornex and Valley pits are adjacent to one another, the blast-ability of the rock in the two pits is quite different. The ore in the Valley pit requires substantially more energy than the ore in the Lornex pit to break it to an acceptable size to maintain shovel productivity and crusher throughput. This difference is attributed to the greater intensity of fracturing and argillic alteration in the Lornex pit compared to the Valley pit. The greater susceptibility of the Skeena granodiorite to hydrothermal alteration may, in part, be due to its finer grain size thus greater surface area available for alterations. It may also be due to having some primary minerals in its' composition of a higher temperature of formation than the Bethsaida quartz monzonite minerals (McMillan, 1985).

Blast Zone Outlining Procedure

A combination of the geotechnical mapping, penetration rate data, toe elevation contouring and observations on shovel and drill performance were used to determine the character and extent of the blast zones. This complementary information was brought together by the geotechnical mapper and the explosives supervisor. The procedure used to develop the blast zones was as follows:

1. map the actively mined faces and the final walls of the bench recording all pertinent geotechnical information;
2. using a hydrothermal alteration model, the alteration zones were projected along the predominant structural trends across the bench;
3. blasthole drilling penetration rate maps for the holes on the bench above were developed by the blasting supervisor;
4. toe elevations for the mined-out bench were contoured by the field geologist in suitably small contours of 0.3 m to 0.6 m, to show the high relief areas;
5. shovel and drill performance were monitored as they drilled and mucked the bench. Areas which required re-drilling due to poor fragmentation and tough-digging zones for the shovels indicated hard zones;
6. the penetration rate data and toe elevation data were then used

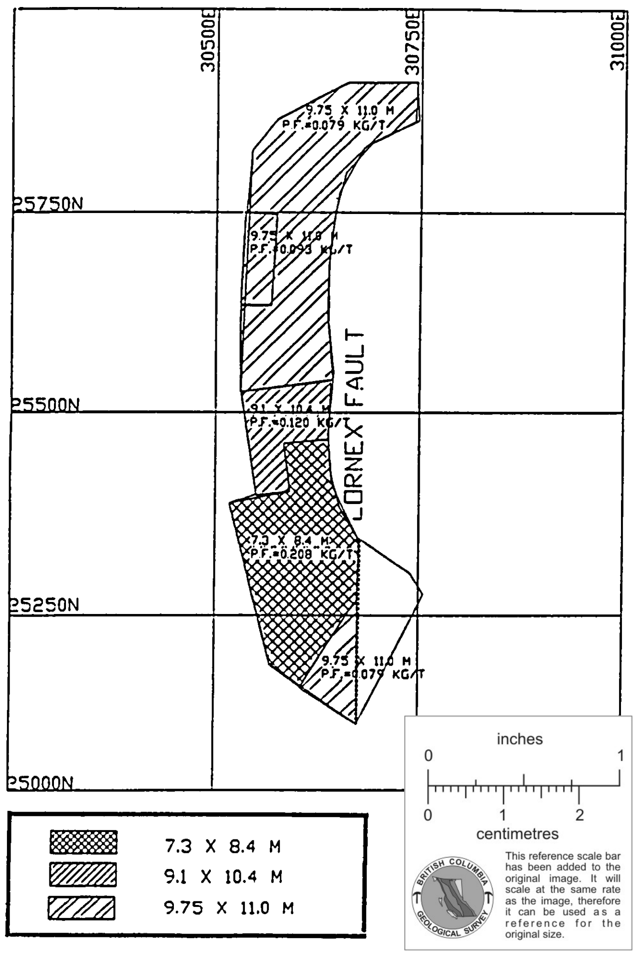


FIGURE 7. Lornex pit 1351 (4432) bench — pattern sizes.

to verify or modify the blast zone contacts as projected across the bench from the geotechnical mapping.

Summary

Blast Zones and Blasting Optimization

Initially it was decided to map each of the benches with the intention of improving the rock fragmentation. Once each bench is mapped and a blast zone outlined, a blast pattern size is determined for each zone (Figs. 5, 7, 8 and 10). When a bench has been mined out and mapped, a new blast plan is outlined for the next bench. Some blasting practices were changed unilaterally to improve fragmentation, while others are adjusted each time a blast plan is made up. In the former category are the practices of loading to a collar and changing the pattern geometry. On a regular basis, pattern spacing and burden are adjusted for each blast in each pit. The standard subgrades are not changed. In the Valley pit the standard subgrade is 1.5 m and in the Lornex pit it is 1 m.

Blast Parameter Changes — Irregular

Loading to a collar was introduced in the latter part of 1986. Up to this point, poundage for each hole was assigned and loaded, regardless of the hole depth. It was assumed that all blastholes were drilled to the same depth. Loading to a collar actually lowered the over-all powder factor and gave a much more consistent heave to the muck pile. This also allowed for a better visual check of a blast and helped determine the amount of energy released by the explosive.

In early 1986, a staggered blast pattern was introduced to replace the square pattern. In the staggered pattern the 'spacing' equals 1.15 times the 'burden'. With this pattern there is more overlap in the area of interference of explosive energy between adjacent rows of holes, which results in improved fragmentation.

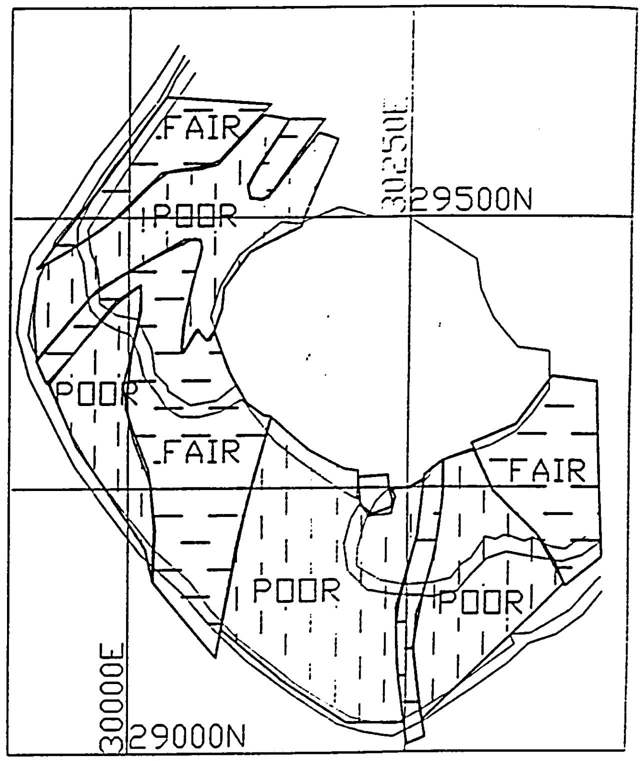


FIGURE 8. Valley pit 1125 bench — dig-ability plan.

Blast Parameter Changes — Regular

When the blast plan for each new bench is made up, a particular powder factor is assigned to each blast zone, according to the blasting results, digging performance and geotechnical characteristics of that zone on the last bench. Tough digging and very chunky muck mean the powder factor must be increased, whereas fine muck and easy digging indicate the powder factor can be reduced. The extent of the blast zones with a particular powder factor may also change from bench to bench, depending on the dip and plunge of the structural/alteration make-up of the zones. If the powder factor is adjusted, a computer program automatically changes the burden and spacing. Originally, the actual powder factor adjustments were only approximations to the changed conditions. Time and experience helped to refine them.

Pattern Size Evaluation

After a pattern has been blasted, the broken rock is monitored to see if the digging is adequate and the fragmentation acceptable for both the shovel digging the muck and the crusher, if it is ore. As a general statement, the ore in the Valley pit must be blasted more for the crushers than the shovels. The opposite is true in the Lornex pit in all zones except the *poor* zones. The ore is broken for shovel productivity, not crusher throughput. Once the shovel digs the ore, the muck is quite small, and easily crushed.

Poor digging is manifested by chunky and poorly broken muck, a lot of scraping of the face by the shovel to dislodge chunks, shaking of the boom cables and rocking on the tracks as the bucket breaks loose a hard chunk. This is very hard on the shovel.

The topography of the toe can be a good indication of the success of the blast. High toes of 1.5 m to 3 m were common in the

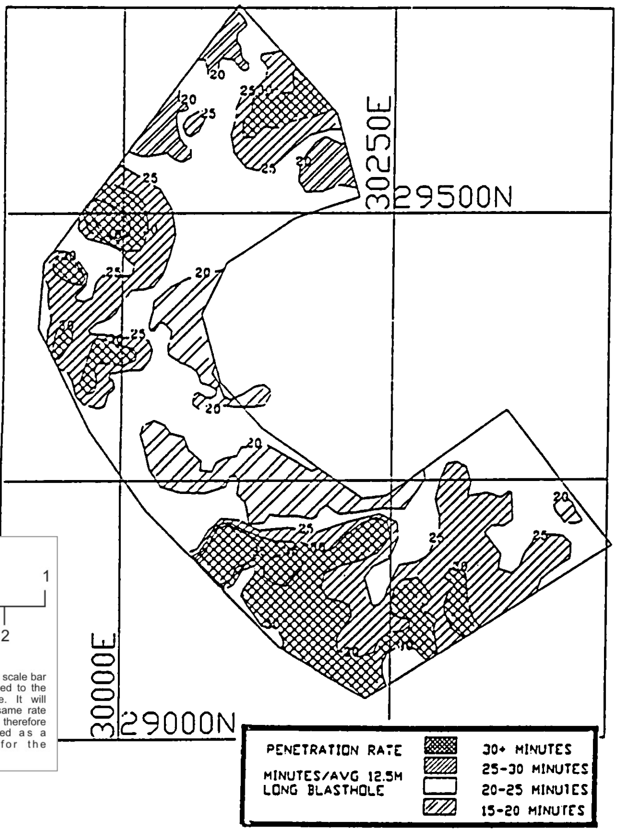


FIGURE 9. Valley pit 1125 bench — blasthole drill penetration plan.

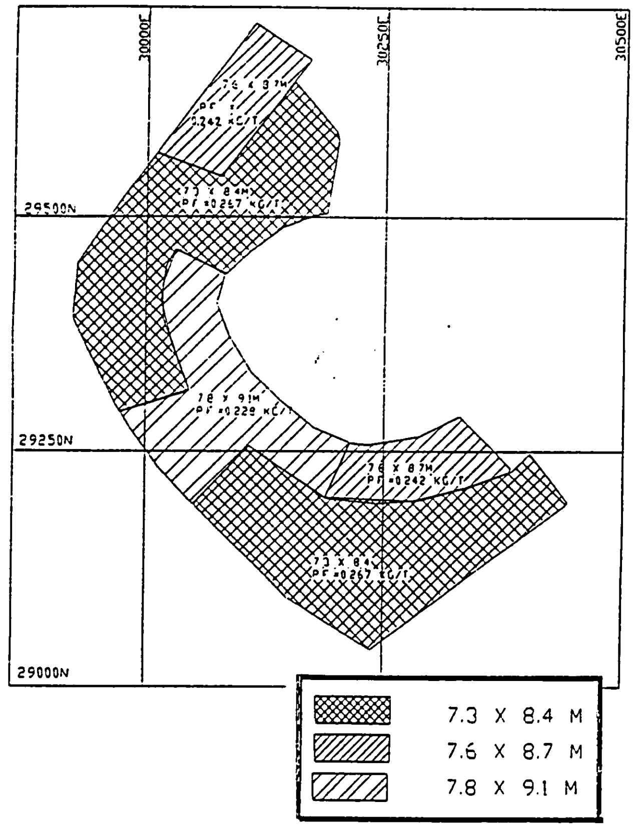


FIGURE 10. Valley pit 1125 bench — pattern sizes.

poor zones in the early days until the blasting was refined. They could be used to correlate with the geotechnical mapping and usually there was a good correlation with *poor* zones. On the other hand, low toes and fine, floury muck indicate over-blasting and can be adjusted accordingly.

With each bench the blast zones continue to be refined from the previous bench. The fragmentation from each zone has become more consistent and the shovel productivity more predictable (the target shovel productivity for the P&H 2800ZPA shovels is 2800 t/h and the smaller BE 295B shovels is 1965 t/h). With experience it has also become more evident that the geotechnical mapping and drill penetration times have the greatest impact on determining "blast-ability" zones.

Results

Lornex 1351 Bench

As can be seen in Figure 5, the Lornex 1351 bench on the west wall is subdivided into three blast zones: *poor, fair* and *good.* The northern minor *poor* zones represent fresh to weakly argillicly altered Bethsaida quartz monzonite. The central *fair* and *good* zones represent the gouge-filled W2-5 fault zone and its moderate-intense argillic alteration halo merging with the strongly altered and fractured structural domain along the Lornex Fault. The southern extensive *poor* zone represents intensely silified and intensely kaolinized Bethsaida quartz monzonite. A *fair* to *good* zone occurs adjacent to the Lornex Fault and reflects fracturing parallel to the fault.

The tough southern *poor* zone is the same one which caused so much trouble on the 1497 bench in 1981 when the blast mapping started. It was blasted with a 7.3 m by 8.4 m pattern and a 0.208 kg/t powder factor. By contrast, the soft, weak W2-5 fault zone was blasted with a 9.75 m by 11 m pattern and a 0.079 kg/t powder factor.

Both the southern *poor* zone and the northern *good* zone correlate well with the penetration rate contours. The *poor* zone coincides with the 25 minute contour and the *good* zone along the W2-5 fault correlates with the 15 to 20 minute contours. The only complication is the 30 to 55 minute zone close to the junction of the W2-5 fault and the Lornex Faults. This badly broken area was probably also water bearing which resulted in caving holes and extra time reaming the holes out.

Valley 1125 Bench

Reference to Figure 8 shows two main *poor* zones on the 1125 Valley Pit bench; a northern northeasterly-striking narrow zone and a southern broad north-south zone. The northern zone consists of very chunky tough Bethsaida quartz monzonite bound together by a closely spaced network of quartz-bornite stringers and hardened by quartz-grain silicification. The southern zone is partly due to fresh to weakly altered Bethsaida on the west side and the re-entrant late stage quartz vein network on the east side resulting in tough digging. Both these geological zones correlated closely with 35 to 40 minute/hole penetration rate contours (Fig. 9). Both zones were blasted with 7.3 m by 8.4 m tight patterns and a powder factor of 0.267 kg/t.

The central *fair* zone is a vertically persistent zone (88 m vertical continuity so far) that exhibits closely spaced fracturing (commonly 0.3 m spacing or less) and moderate and moderate-intense argillic alteration. This zone strikes northerly and dips consistently easterly at about 60 degrees and plunges to the north. It correlates quite well with the 20 minute penetration rate contours. It has been blasted on a 7.8 m by 9.1 m pattern on this bench, with a powder factor of 0.228 kg/t. This means pattern spacing was widened due to this more friable zone, thus saving explosives and still achieving the good fragmentation.

A comparison of the 1125 m southwest Valley pit bench and the 1351 m (4432 ft) Lornex pit bench blast plans shows considerably higher powder factors for the Valley pit compared to the Lornex pit, for the same rock type. The southern main *poor* zone at Lornex had a 0.208 kg/t powder factor, whereas both the *poor* zones at Valley had a 0.267 kg/t powder factor, for an increase of 28%. This mainly reflects the less intense fracturing in the Valley pit. Even the *fair* zone in the Valley 1125 bench has a higher powder factor than the *poor* zone in the Lornex 1351 bench (Figs. 7, 10).

Typical pattern sizes from the two pits are as follows:

	Pattern Size	Powder Factor
Lornex pit — 1351 bench west wall		
Poor zone	7.3 m × 8.4 m	0.208 kg/t
Fair zone	9.10 m × 10.4 m	0.120 kg/t
Good zone	9.75 m × 11.0 m	0.079 kg/t
Valley pit — 1125 bench west wall		
Poor zone	7.3 m × 8.4 m	0.267 kg/t
Fair zone	7.8 m × 9.1 m	0.228 kg/t
Good zone	9.1 m × 10.4 m	0.174 kg/t

Presently the combined powder factor of the two pits is 0.200 kg/t. Individually the Lornex pit averages 0.100 kg/t and the Valley pit averages 0.218 kg/t.

Estimated Cost Savings

By accurately determining the "blast-ability zones", explosives, blasting accessories, blasthole drilling and shovel and truck operations are optimized and a substantial cost savings can be realized. On an annual basis an estimated cost savings of well in excess of a $1,000,000 is realized at Highland Valley Copper. At present, most of the areas with strong contrasts in blast-ability i.e. between *poor* and *fair* or *poor* and *good*, are in three different shells in the Valley Pit. The $1,000,000 estimate comes from these three shells, as well as other areas of the Valley and Lornex Pits with a *fair* to *good* contrast. The zones which show sharp changes in blast-ability have the biggest impact on cost savings. The 1351 bench in the Lornex pit can be used as an example. Approximately 2 200 000 t were blasted in total. If accurate interpretation of the geological mapping and drill penetration rates had not been completed to determine the "blast-ability" zones, either of two things would have happened. The area would have been over-blasted to ensure the shovels would not have any *poor* digging or the hard zone would have been under blasted resulting in *poor* digging or even redrilling.

The *poor* zone was outlined and contained 800 000 t. It was blasted using a powder factor of 0.208 kg/t. The remaining 1 400 000 t were blasted with an average powder factor of 0.093 kg/t. If the west arm had been blasted as if it was all a *poor* zone at the high powder factor, approximately $100,000 would have been spent needlessly. On the other hand, if the hard zone had been under-blasted and it resulted in hard digging for the shovel and or reblasting a similar cost would have been incurred.

The truck and shovel operation has also benefited from better fragmentation in terms of increased productivity (personal observations) as well as decreased maintenance costs. Maintenance personnel cite various positive improvements to this effect. They include the following: less shovel breakdowns in the last few years, less replacement of crowd transmissions, better dipper life, and increased teeth and adaptor life. Trucks also benefit from less impact on the truck box during loading of smaller sized chunks. Truck operators also appreciate the gentler loading resulting from the finer muck.

Two other factors which have optimized blasting are the conversion to emulsion from regular slurry and ANFO in January 1988 and an increase in blasthole diameter from 25 cm to 26.9 cm. The former conversion resulted in an expansion in the blast pattern size by 30% with no loss in fragmentation. This resulted in a 30% reduction in drilling requirements. The 15% increase in blasthole cross sectional area resulted in a further reduction in drilling meterage of 15%, by increasing the volume of explosive in each hole.

Slope Stability Enhancement

In the Lornex pit, a useful by-product of the regular blast hardness mapping was structural and hydrological information pertinent to slope stability. The pervasive fault fracture pattern of N20 degrees E striking major faults dipping 60 degrees westerly and southeast striking joints and minor faults with a steep dip explained the structural basis of the toppling failure mechanism on the west wall. Artesian water intersected and mapped along the toe of each successive bench on the westwall indicated the importance of water pressure as a driving force in promoting toppling failure on this wall. It also helped in targeting the annual horizontal drainhole programs on this wall.

In the Valley final southwest highwall, blasting is modified not only according to the hardness zones but also relative to the orientation of structural discontinuities. The two main structural trends were north-south and southeast-striking shears and faults. Before mapping revealed these trends, the break angles for blasting just happened to coincide with them. Because there was considerable crack propagation along them into the final wall, one of the remedial measures was to change the break angle. With the break angle now crossing the structure, crack propagation was reduced. Other factors also helped to reduce this problem, but a knowledge of the structural trends was a key factor in their solution. Blast hardness mapping also revealed a concentration of north-south-striking, steeply west-dipping faults on the upper final southwest highwall. This explains the toppling failure occurring on this wall now and helps alleviate the fear of a catastrophic failure.

Another slope stability by-product of the blasting according to blast-ability zones is that where *fair* or *good* zones are close to the pit wall, blast vibrations will be reduced by the lower powder factor. This is beneficial because these zones are usually highly fractured or faulted and susceptible to blast damage. On the other hand, where a *poor* zone is adjacent to a wall, the higher powder factor can be alleviated to some extent by early-blasted line holes and blasting to a free face, or pre-shear holes.

Conclusions

Coordination and cooperation between the geotechnical mapper and the blasting supervisors at Lornex and Highland Valley Copper has very significantly optimized blasting and reduced costs. Over nine years of experience have shown the undeniable existence of distinct geological zones in the local granitic rocks that strongly affect blasting and slope stability.

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Optimal tailings management at Highland Valley Copper

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ABSTRACT

This paper reviews tailings management at Highland Valley Copper, and outlines main features of its tailings storage facility enclosed between the H-H and L-L dams. Key management focusses are: energy efficiency, cost effectiveness, safety and integrity of the tailings impoundment, environmental protection, and reclamation after decommissioning. Main features of the tailings facility include: high international standards for seismic and flood design criteria, water management and water quality monitoring, operations and construction optimization, reclamation and decommissioning plan, and geotechnical challenges associated with the complex geology of the L-L tailings dam foundation.

Introduction

Highland Valley Copper stores its tailings mainly in the Highland Valley tailings facility (ultimate capacity: 1.8 billion tonnes). Smaller tailings storage facilities were operated by predecessor companies: the Number One Pond, the Trojan Pond at Bethlehem and the Highmont Pond at the Highmont operation as well as backfilling the Heustis and Jersey Iona Pits at Bethlehem. These areas are currently not in use, and are in the reclamation phase prior to decommissioning. This paper focusses on tailings management in the main facility. As indicated in Figure 1, the facility includes three tailings dams: these are, going from east to west, the H-H, J-J and L-L Dam, named after the alignment alternatives in the initial feasibility studies. From 1970 to 1977 Canadian Bechtel Ltd. advised Highland Valley Copper's predecessor company Lornex, on tailings storage at the H-H and J-J Dams and on the L-L Dam. Since the completion of the L-L Starter Dam in 1977, Klohn Leonoff Ltd. has provided ongoing consultation and construction monitoring of the raising of the L-L and the H-H Dams. The J-J Dams is currently buried by deposited tailings. This paper reviews key tailings management focusses and main features of the tailings facility.

Tailings Management Focusses

The main objectives of tailings management at Highland Valley Copper are to operate an efficient and cost effective tailings facility, and to protect the environment from possible undesirable influences from the tailings impoundment. Key management focusses are: energy efficiency, cost effectiveness, safety and integrity of the impoundment, environmental protection, and reclamation after decommissioning. During the long operating mine life (about four decades under the current mine plan), the geometry of the tailings storage and distribution facilities and tailings operation considerations change with time.

Keywords: Highland Valley Copper, Tailings management.

Paper reviewed and approved for publication by the Metal Mining Division of CIM.

However, by focussing on the key issues, Highland Valley Copper is able to plan, modify, construct and operate in an orderly manner its tailings facility. The mine has been monitoring the quality of surface water and groundwater around the facility before and during the mining operation. With ongoing research into various reclamation strategies and techniques, the mine is preparing for the ultimate return of the tailings proper to appropriate land use after the decommissioning of the mine.

Highland Valley Tailings Storage Facility

The tailings facility is located in the broad U-shaped valley at the north end of the Highland Valley on Pukaist Creek, which offers excellent storage potential because of favourable storage volume vs impoundment elevation characteristics, water-tight valley walls and competent dam foundations.

The original Bechtel scheme envisaged the storage of 680 million tonnes of tailings initially in the Upper Pond bounded by the H-H and J-J Dams at a crest elevation of 1303 m. Thereafter, the remaining storage would be provided by the Lower Pond between the J-J and L-L Dams. The reason for adopting this scheme involving an ultimately redundant Middle (J-J) Dam was the lower initial capital cost of the scheme compared with the two-dam scheme involving the H-H and L-L Dams only. With the rapid rise of energy costs since 1973, the relative advantage of the three-dam scheme was eroded quickly. In 1975, a decision was made to convert the tailings facility to the two-dam scheme in order to reduce exposure to rapidly increasing energy costs.

As shown schematically in Figure 2, the existing tailings storage facility is approximately 9.6 km long, and a pond is formed upstream of the L-L Dam. A portion of the tailings flows from the Highland Mill to the L-L Dam along a 914 mm diameter pipeline located on the north valley slope. Flow is by gravity to the H-H Dam with assistance by two booster pumps located at the H-H Dam and two pumps in series between the H-H and L-L Dams. Tailings delivered to the L-L Dam are double cycloned to produce sand for dam construction. The remaining 78% of the tailings is discharged via three 914 mm diameter pipelines into the tailings pond near the east abutment of the H-H Dam. Both the H-H and L-L Dams are to be raised annually from their respective present heights of 36 m and 103 m to ultimate heights of 107 m and 166 m to meet the ongoing tailings storage requirements of the mining operation.

Design Criteria

To safeguard the environment, the tailings pond was designed to be closed to the environment. A vertical impervious glacial till core was constructed along the centreline of each tailings dam which extended down to tie into a core trench in the dense layer of impervious till underlying the valley between the dams. High international standards were adopted for seismic and flood design criteria (Scott *et al.* 1988) in the design of the dams and the tailings impoundment. Based on detailed review of the site seismotectonic setting, historical earthquakes, regional and local geology, the design earthquake selected for the facility is a maximum credible event corresponding to a magnitude 6.5 earthquake as-

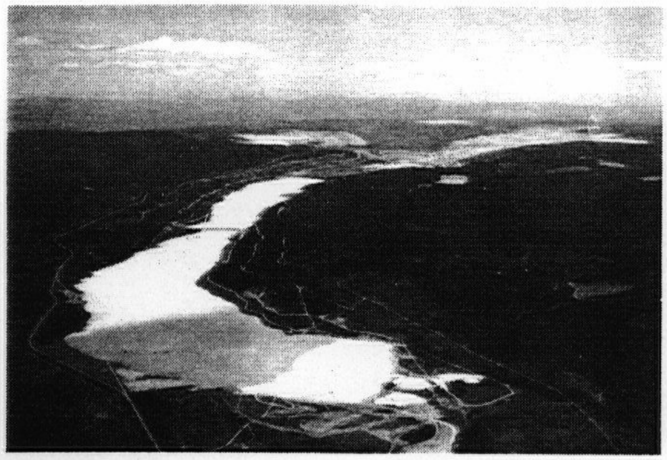

FIGURE 1. Highland Valley tailings impoundment (L-L Dam in the foreground).

sumed to occur at the nearest fault, the Guichon Creek Fault, about 15 km from the H-H Dam and 23 km from the L-L Dam.

The tailings pond is operated as a closed system, and is designed to store a design flood inflow volume of 40 600 cubic decametres without release. This design flood is arrived at from the more conservative of the following two different criteria: (1) the sum of the average annual runoff, the 100-year flood and the 24-hour probable maximum flood (PMF); and (2) the sum of the average annual snowmelt runoff and the runoff from a 120-hour probable maximum precipitation (PMP) assumed to occur during the snowmelt period. An additional freeboard of 2 m is added for preventing overtopping of the L-L Dam by waves, although the tailings beach formed in front of the dam tends to mitigate the wave action.

Water Management and Quality Monitoring

Highland Valley is located in the rain shadow of the Coast Mountain Range in an extension of the upper Sonora Desert of the United States. Most of the precipitation comes down as snow in the winter. Approximately 60% of the runoff in the creeks flowing in the vicinity of the mine occurs in May and June. Over the years systems have been developed to provide mill makeup water from catchment areas around the mine including tailings areas feeding water into the mill process by gravity or other low head sources such as groundwater dewatering wells around the Valley Pit and other well fields near the mine. The drainage basin flowing into the tailings pond between the H-H and L-L Dams provides half the surface area for the collection system. After 20 years of operation, the total volume of freewater in the tailings pond has not changed significantly, although it does undergo yearly fluctuation with runoff. A 12 000 HP pumpstation located on the Thompson River originally constructed to provide makeup water to the mine has been relegated by these systems to emergency backup or insurance against extremely dry years.

Nominal seepage from the tailings impoundment occurs mainly through the lower sand and gravel/clean sand aquifer underlying the relatively impervious glacial till foundation and to a lesser extent through the shallow dam foundations and impervious glacial till cores of the H-H and L-L Dams.

Water quality of both surface water and groundwater around the impoundment has been continuously monitored by Beak Consultants starting shortly prior to the development of the facility in 1972 and by Eco-Tech since 1987. The data to date indicate little effect of tailings deposition within the impoundment on water monitored at strategic sampling points. Similar results are expected in the future, because no significant increases of impoundment seepage are anticipated.

Operations and Construction Optimization

Tailings and water operations at Highland Valley Copper involve pumping tailings, cycloned-sand slurries and water over long distances against high heads. Continuous efforts have been made to improve the tailings and water systems and their components to reduce pumping-related energy consumption. Similarly, annual raising of tailings dams for accommodating ongoing tailings storage requirement also dictates heavy energy consumption in terms of diesel fuel usage by earth-moving construction equipment. The mine has developed design and construction measures that have realized considerable energy savings over the last two decades. Briefly outlined in the following are highlights of these energy conservation approaches relative to the tailings facility (Scott, 1991).

Tailings Transportation System

The original tailings transportation system relied heavily on pumping. Since 1974, the mine has made continuous modifications to the system with the emphasis on utilizing gravity transportation through large-diameter polyethylene pipelines. Highland Valley Copper is a North American bench-mark case in the hydraulic design of these pipelines on an extremely flat slope (Scott 1977). When pumping becomes necessary due to ongoing increase of the tailings level in the impoundment, the tailings transportation system is raised in stages to minimize pumping against unnecessary heads in early years of operation.

Water Transportation System

To ensure a reliable supply of make-up water for its mill usage, the mine installed at the outset a raw water pumping system with a capacity of delivering 8000 gpm of water from the Thompson River against a total dynamic head of 1400 m over a distance of 22 km. Since then, the mine has developed an elaborate surface and groundwater collection system which collects surface water from seven nearby watersheds with a total area of 260 km^2, and mines groundwater from the Highland Valley. The collection system includes a series of water retention reservoirs, diversion ditches and several well fields. The Thompson River pumping system has become a back-up component with considerable savings on energy consumption.

Originally an independent reclaim water system was installed to reclaim supernatant water from the tailings pond for the mill usage. Ongoing upgrading in terms of combining the raw water and reclaim water system and looping and re-looping pipelines has reduced pipeline friction loss significantly thereby reducing electricity consumption,

Tailings Storage Facility

Economic considerations and energy conservation have influenced the evolution of the design and construction of the main tailings storage facility (Scott and Lo, 1984). Major decisions in this regard include: the adoption of the modern centreline scheme for both the H-H and L-L Dams, the utilization of the wedge-shaped delta between the H-H and L-L Dams above the L-L Dam crest for tailings storage, the early conversion to an energy efficient two-dam storage scheme, the reduction of the size of the L-L Starter Dam, and the adoption of cost-saving hydraulic placement methods to construct the L-L Dam mainly of low-cost, dense cycloned-sand damfill with excellent engineering properties.

Reclamation and Decommissioning Plan

Reclamation plans and techniques are being developed on an ongoing basis. Growth performance of grasses and legumes on

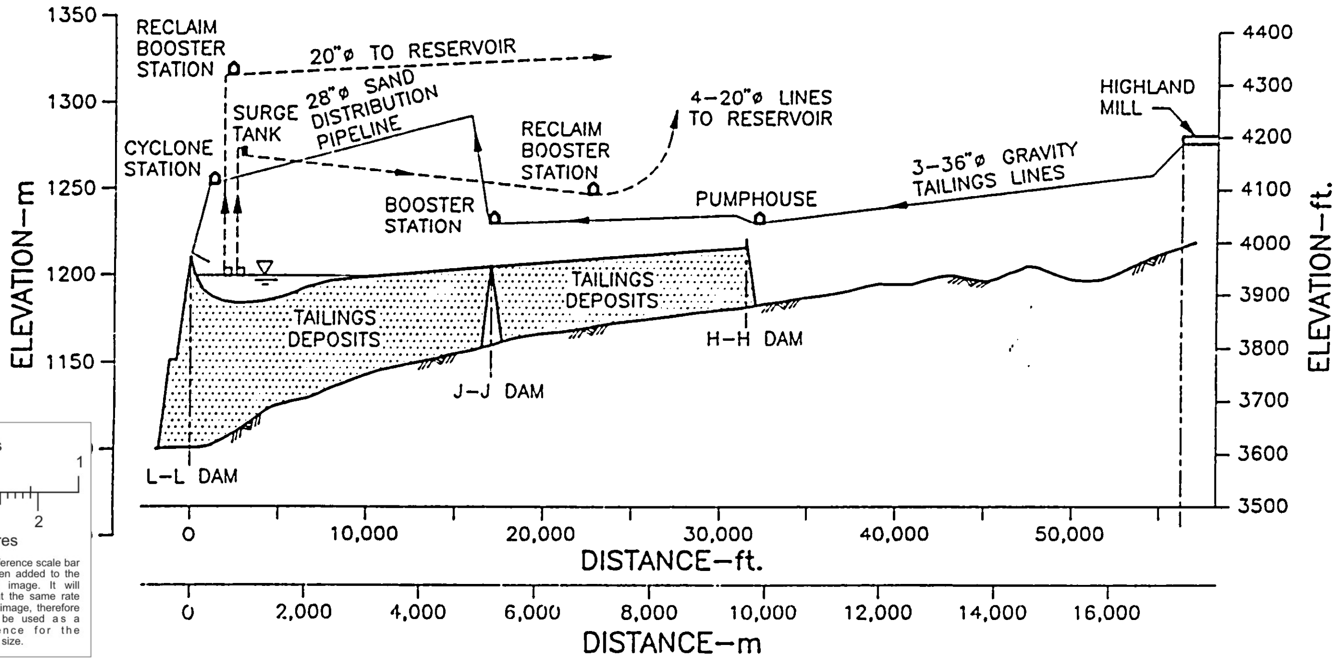

FIGURE 2. Schematic sketch of present Highland Valley tailings facility.

the tailings in areas reclaimed on inactive tailings impoundments is monitored and evaluated for future reference in reclamation planning. Ultimate land uses for the long and gently sloped (0.3%) tailings pond may include: seed production, enhanced grazing for cattle and wildlife, hay production, tree farming and public recreation.

It is envisioned that creeks flowing into the tailings pond, which is sloping from the H-H Dam toward the L-L Dam, will naturally irrigate the reclaimed pond area and form a shallow lake located about 200 m upstream of the L-L Dam. Conceptual design of a permanent spillway system located on the south abutment of the L-L Dam involves alternatives with or without diversion channels around the tailings impoundment. The spillway will consist of an approach channel, a control structure located in a rock cut and an outlet channel. Its location will be determined prior to mine closure upon detailed investigation.

The spillway control structure may consist of concrete culverts embedded in a concrete free overflow crest structure. The culverts would pass normal flows, and most flood flows due to the attenuating function of surcharge storage provided by the L-L Dam above the invert elevation of the culverts. The overflow structure would generally not pass water unless the culverts became blocked or inflow rates approached those of a design PMP event. For channels located within glacial till, riprap armour and concrete flume are required to prevent erosion. A rock-lined plunge pool will be used to dissipate energy of the outflow prior to its return to the original Pukaist Creek course downstream.

Geotechnical Challenges

The unusually complex geological setting of the L-L Dam has provided continuous geotechnical challenges for the dam designers. The 43 m high L-L Starter Dam was constructed in 1976-1977 over compressible lacustrine deposits up to 12 m thick in the valley bottom (Burke and Smucha, 1979). Subsequent annual raises involved excavation of lacustrine deposits downstream of the Starter Dam and constuction of a downstream buttress berm (Klohn, *et al.*, 1982). The dimensions of the buttress berm for the ultimate L-L Dam has been determined using a state-of-the-art nonlinear effective stress finite-element dynamic response analysis embodied in the TARA-3 computer program. The analysis provides a coherent picture of the dam's dynamic behaviour including acceleration and deformation fields as well as the distribution of seismically induced porewater pressures when subject to the design earthquake motion (Scott *et al.*, 1989).

While the south abutment of the L-L Dam is founded on dense basal till overlying sound granodiorite bedrock, its north abutment is underlain by a glacial till sheet of varying thickness overlying volcanic and sedimentary sequences of bedrock. The complex bedrock stratigraphy at the north abutment presents interesting foundation conditions in the form of high foundation pore pressures and seepages as well as low foundation shearing strength.

The area has been instrumented with piezometers and inclinometers. Seepage and stability control measures include: supplementing the natural upstream glacial till blanket with additional impervious till materials to achieve a reliable impervious upstream blanket with minimum design dimensions, extending the upstream tailings beach in the area to reduce the seepage gradients through the foundation bedrock, removing glacial till cover to expose the volcanic bedrock for pressure relief in the downstream area, providing appropriate filter and drainage zones in the downstream dam shell, and adding downsream buttress berms over sedimentary bedrock areas.

To cope with the design challenges, the mine has adopted from the outset the observational approach, using instrumentation to monitor key foundation performance parameters such as pore pressures and movements. For a given year, construction is limited to what is required. However, plans are prepared to deal with situations as detected by the ongoing instrumentation monitoring program.

Conclusions

Tailings management at Highland Valley Copper has many facets, and undergoes continuous evolution as dictated by the changing geometry and operating conditions. The main principle guiding the design, construction, operation and eventual decommissioning of the tailings facility, however, remains the same. This principle is to operate an energy efficient and cost effective facility to meet the ongoing mining requirement of tailings storage as well as to provide the long-term protection of the surrounding environment.

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Industrial Minerals - Today and Tomorrow

Industrial Minerals — Today and Tomorrow, The Raw Materials to Build the Upper Midwest, a workshop, will be held in Minneapolis, Minnesota, **September 10-11, 1992** and a field trip to the Minnesota River Valley Area, September 12, 1992.

The conference themes are: industrial mineral resource demands; resource availability; environmental issues and solutions; land use conflict and resolution; research needs; and zoning case studies-promises and pitfalls.

This Workshop on Industrial Minerals will address the resource-environment issues that confront us, and it will propose possible solutions to launch us into the next century with less-than-fragile foundations. The conference is designed for those who will identify and provide the materials, those who make decisions about zoning and regulation in areas containing usable resources, and those who will communicate to and educate the nation about the required balance between improving the infrastructure and preserving the environment.

In conjunction with this workshop, the Minnesota Geological Survey will conduct a one-day field trip on September 12, 1992, to the Minnesota River Valley area. Stops will include dimension stone quarries in fossil-rich Ordovician limestone and the 3.6 billion-year-old Rainbow Granite as well as a clay pit near Redwood Falls.

Co-sponsors, the U.S. Geological Survey, the U.S. Bureau of Mines, and the Minnesota Geological Survey, along with the state geological surveys of Illinois, Indiana, Michigan, North Dakota, Ohio, South Dakota, and Wisconsin, invite you to attend. For additional information and to receive a Registration Circular contact: Gary B. Sidder, U.S. Geological Survey, P.O. Box 25046, MS 905, Denver, CO 80225, U.S.A.; Tel.: (303) 236-5607; Fax: (303) 236-5603.

CIM

Environmental planning of mines

Environmental planning for mining projects, with pointed emphasis on the application of risk assessment in the decision-making process, is the focus of the Risk Assessment/Management Issues in the Environmental Planning of Mines Conference, **October 5 to 7, 1992,** to be held at the Mariott Pavilion Hotel in St. Louis, Missouri.

The design of new mines, and the planning of remedial actions at existing mines are explored. The use of health risk assessment, environmental risk assessment, probabilistic risk assessment and economic risk assessment in decision making are covered in addition to risk communication.

The considerable amount of risk assessment technology developed by professionals outside the mining industry contributes to the conference.

Program areas include: Plenary session; Risk Assessment in Mine Development; Risk Assessment/Management at Superfund Sites; Pre-Acquisition and Operational Audits; Economic Risk Assessment Related to Environmental Decision Making; and Miscellaneous Aspects of Risk Assessment/Management.

A short course, entitled "Risk Assessment Principles and Practices for Evaluating Environmental Aspects of Mining Projects" will be held prior to the conference. The course presents the principles of probabilistic risk assessment, toxicological aspects of risk assessment and health and environmental risk assessment.

For additional information, contact: The Meetings Department, SME, P.O. Box 625002, Littleton, CO 80162; U.S.A., Tel.: (303) 973-9550; Fax: (303) 979-3461.

CIM

Zinc to cover the Canada pavilion

Canadian zinc-producing companies have joined together to showcase their metal to the world by sponsoring the zinc facade of the Canada Pavilion at Expo '92 in Seville, Spain.

Two facades of the Canada Pavilion, totalling 3200 m^2 are covered with 1 m by 1 m sheets of pre-weathered zinc. Approximately 50 tonnes of zinc was used.

The Canada Pavilion is one of the most distinctive zinc-clad structures in the world. Zinc was chosen for its versatility and recyclability. The exterior of the Pavilion demonstrates the beauty of zinc, a product that naturally and permanently retains its initial colour and patina. The material will take on a "life" and an "inner glow" that will give the Pavilion a striking and everchanging image throughout the day and night.

The use of zinc on the Canada Pavilion ties in with the main theme of the exposition: The Age of Discovery. "Discovery will be manifested in part through the architectural achievements of pavilions at Expo '92. Visitors to the Canada Pavilion will discover how zinc can be used in architectural design. Zinc has been used widely in architectural applications in Europe for many decades. It is hoped that by sponsoring projects such as the Canada Pavilion, the market for zinc in architecture will be developed in North America.

The Canada Pavilion has a dominating presence just inside the main gate of the Expo '92 site. Located on a principal avenue, the Pavilion has the advantage of a double facade, attracting the attention of monorail passengers and of pedestrains using the main gate. In addition, plans are being made for other zinc promotional programs, including a Zinc Day. About 40 million people from around the world are expected to attend the exposition.

Expo '92 is scheduled to run up to October 12, 1992.

For more information, contact: Stephen Wilkinson, Tel.: (416) 869-1850; Fax: (416) 869-3698; or Ralph Eastman, Tel.: (604) 682-0611; Fax: (604) 685-3041.

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