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RESEARCH STUDY OF CONTINUOUS HAULAGE FROM OPEN PIT MINES

PREPARED FOR:

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BY:

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FINAL REPORT

ON:

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Research Study of Continuous Haulage
From Open Pit Mines

November 1978

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16. Abstract

It is unlikely, as in the past, improvement in truck efficiency and capacity can fully offset the effects of inflation, fuel costs/shortages and ever increasing haul distances. Therefore, investigations of haulage systems less sensitive to these problems are warranted. Thus, this U.S.B.M. contract research study on alternative continuous haulage systems was prepared. This study identifies the physical, economic and technological parameters in the application of continuous haulage systems with special attention to under pit slope conveyors and skip hoisting systems. The evaluation data was gathered from extensive literature searches, European/North American mine visits, and industry comments. The study concludes that continuous underground haulage is a viable method within the proper geometric configuration.

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FOREWORD

This report was prepared by International Mining Consultnats, Inc., Suite 20, 80 Garden Office Center, Broomfield, Colorado, 80020, under U.S.B.M. contract JO275032. The contract was initiated under the Advance Mining Technology program. It was administered under the technical direction of Spokane Mining Research Center with Mr. Earl Frizzell acting as Technical Project Officer. Mr. Larry L. Rock was the contract administrator for the Bureau of Mines. This report is a summary of the work recently completed as a part of this contract during the period of September 1977 through November 1978. This report was submitted by the authors in November 1978.

SUBJECT INVENTIONS

This is to certify that, to the best of my knowledge, and belief, there are no Subject Inventions made or have resulted from the performance of this contract.

November 1978

William G. Freeman

William G. Freeman
President

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EXECUTIVE SUMMARY

Surface Mining is the predominate mining method in the world today. More than two-thirds of the minerals consumed this year will come from surface mines.

The bulk of iron and copper, for example, comes from large open-pit operations employing traditional truck and shovel mining; but, over the last few years a distressing trend has become apparent in open-pit mines. One of the major direct cost components, haulage, is experiencing a rapid rise. This increase is identified as primarily resulting from:

- Increasing haul length as the major mines work to greater and greater depths
- Rapidly escalating capital and operating costs due to inflation
- Rapid increases in the cost of diesel fuel, lubricants and tires which are tied to the price of crude oil supplies.

These are not new trends but in the past the rapid increases in truck capacity and efficiency has offset increases and in some cases reduced truck haulage costs. However, it is doubtful that truck capacity can increase from the present haul units rapidly enough to offset the present high rates of inflation and escalating fuel costs.

Therefore, the U. S. Bureau of Mines has commissioned this contract research study to evaluate alternatives to traditional truck haulage. This study emphasizes underground continuous haulage techniques and identified several past and present applications. The classifications of continuous haulage systems identified are as follows:

- Systems employing ore/waste passes to feed below pit slope conveyors. Crushing is either in the pit or underground prior to feeding belt conveyors.
- Systems employing ore/waste passes to feed below pit horizontal belt conveyors which in turn feed vertical or inclined skip hoisting shafts. Crushing is either in the pit or underground prior to feeding the horizontal belt conveyors.

- Systems employing ore/waste passes to feed below pit horizontal rail transport, or any transport system other than belt conveyors, which in turn feeds vertical or inclined skip hoisting shafts. Crushing employed depending upon the horizontal transport requirements.

By contract this study was to emphasize the first two classifications of underground haulage systems and continuous surface haulage was to be excluded entirely.

After an extensive literature search, the personnel conducting this research study made mine visits to nine European and North American mines. The mines visited were located in Ireland, Germany, Sweden, Finland, Spain, Mexico, and the United States. The data gathered is summarized and evaluated in the State-of-the-Art survey and Evaluation of the Elements of continuous underground haulage system sections of this report. Detailed descriptions of the mines and their haulage systems are found in the Appendix B.

The State-of-the-Art study established the viable use of continuous underground haulage systems in both past and present applications. These systems have proven their technical and economic feasibility under the site-specific conditions of each property where applied. This study identified the following important geometric relationships for viability of underground haulage systems:

- Laterally constricted orebodies which can achieve a more favorable ore/waste ratio by reducing benching width (no haul roads) or achieve a greater pit depth to extend the surface mining life.
- That the plans for developing the orebody include underground extraction in the foreseeable future.
- Haulage out of the pit would have extremely adverse profiles for trucks, or be too steep for in-pit conveyors or that surface skips or conveyors would hamper pit expansion.
- Errors in orebody deliniation prevent expansion of the pit to accommodate surface haulage.
- The topography between the pit and the plant adversely effect surface haulage making haul roads prohibitively long or slopes excessively steep or lifts too great for belts.

- Climatic conditions favoring the protection of haulage systems by going underground - arctic weather.
- Environmental constraints.

It was determined that the limitations/risks associated with these continuous underground haulage systems actually reflect the limitation/risks of the elements that make up the system. Therefore, the system elements were examined in detail for below pit slope conveyor and below pit conveyor/skip hoisting systems. Lesser detail was employed in examining the more traditional rail haulage components or the more exotic slurry and pneumatic haulage elements.

This study provides extensive material on the application, engineering criteria, limitations, capacity, technologic trends and design risks of the elements which make up continuous underground haulage systems (crushers, conveyors, skip hoists and ore passes). This information and the evaluation procedures employed should be applied in assessing the viability of continuous underground haulage for specific properties.

It was found that the traditional objections regarding capacity, lump size, and reliability for crusher, conveyor belts and skip hoisting have been overcome to a great extent by recent technologic improvements in these haulage elements.

The primary objections to employing these alternative haulage systems as solicited from nine mine managers interviewed during this study reflect both equipment limitation and boundaries in system application. These objections fell into the following areas:

- Belief that continuous haulage and the handling of multiple ore types plus waste are incompatible
- Investment analysis which show that, as long as trucks are required for waste removal, their full value is achieved only through ore and waste haulage.
- Flexibility to adapt to changing pit configuration is seriously impaired.
- In pits with poor ground stability no improvements in ore/waste ratios would be realized by going to underground haulage.

- Belief that conveyor interruptions (availability) would be unpredictable and have serious impact on the overall pit operations.
- Underground systems have a very high capital cost and that this investment represents "all the eggs-in-one-basket."
- Funds are not "available" for such large up-front capital investments.
- Reliable and up to date operating cost and performance data for underground systems is unavailable.
- Reliable design and capital cost estimating is difficult due to the empirical nature of ore pass and rock chamber design.
- Underground systems have limited access and are difficult to maintain and present serious problems to quick repair of malfunctions (i.e. hang-ups in ore passes and major crusher repairs).

In most cases, these objections are from operations which have evaluated continuous underground haulage for application in their pits and rejected it in favor of no change from truck haulage or opted to use other surface systems. However, the majority of these evaluations were made prior to 1973 and recent changes have prompted new interest in underground systems. Some of the reasons for renewed interest were found to be:

- Operating problems resulting from current pit wall instability giving consideration to moving underground.
- Concern that larger sized trucks (350-ton and greater) will reduce operating flexibility and result in serious mismatches to present pit equipment.
- Concern for escalating energy costs and the need to find alternative haulage methods with lower operating costs that are more inflation proof.
- Fear of future fuel shortages.
- Environmental concern for air pollution, dust generation, and noise attributable to truck haulage.
- Environmental considerations such as degredation of scenic views, haul road dust, population centers and noise rule out surface haulage.

Since this study did not include a detailed examination of open-pit utilization of truck haulage (or other non-continuous

haulage systems or continuous surface haulage systems) the extent to which underground systems are economically feasible in future applications will have to be decided on a property by property basis. However, it appears to be generally accepted in the mining industry that:

- Shallow lifts and low to intermediate capacities as in the early years of most open-pit operations favor truck haulage or some other non-continuous surface haulage.
- Intermediate lifts, from pit depths of 500-800 vert. ft., and intermediate to high capacities, say to 40,000 tph, favors conveyor installations.
- Lifts from greater depths, say in excess of 2000 vert. ft. and low to intermediate capacities favors vertical skip hoisting.

Of course, any numerical cut off in capacity of lift is site specific and is continually changing to reflect the impact of technological improvements and escalations in costs for the various haulage systems. The scope of this study precluded any economic comparison of surface to underground haulage systems. Even had such comparisons been developed for a hypothetic mine model the vary process of selecting the design assumptions (such as mine capacity, pit depth, haulage profiles, etc.) would load the comparisons in favor of a specific class of haulage systems.

This study has concluded underground continuous haulage systems to be technically viable where the proper geometric relationships exist. The applications of underground continuous haulage now in operation have demonstrated not only system viability; but, that within the economic and physical constraints of the properties, these underground haulage systems are the optimum (or most economic) haulage method.

The economic evaluation of alternative haulage systems requires going beyond the technical viability of applying the various systems and must also address that:

- The mineral commodity price fluctuations affect mine capacity and life and operating flexibility requirements and have a major impact on selecting haulage systems.

- Overall mine costs and mine planning impact upon haulage system selection.
- Inflation and escalation rates are crucial factors in determining economic viability.
- Technologic trends in the alternative systems are difficult to project but critical to selecting the optimum system.
- The various haulage methods exhibit characteristics which favor specific applications but also have inherent disadvantages.
- Many environmental considerations, such as not despoiling a scenic view or being a quiet dust free neighbor, have to be given a dollar value or in some way considered in assessing haulage system economics.

This study suggests that at least one and preferrably three operating properties be evaluated to establish the relative merits of non-continuous surface haulage to continuous surface haulage to underground continuous haulage. And, that this economic evaluation take the form of a present worth comparison reflecting anticipated inflation, fuel cost escalations, environmental considerations and technologic trends.

From these site specific evaluations the economic viability of underground continuous haulage can be established within a range of application reflecting todays economics and technology. This is desirable in as much as the technical viability of continuous underground haulage from open-pit mines has been established.

1. INTRODUCTION

1.1 Study Background

As previously mentioned in the Executive Summary surface mining is today's predominate mining method, both in the United States and world wide. More than two-thirds of all the minerals consumed are extracted from surface mines. And it is generally accepted that this predominance will continue through the foreseeable future.

It is the superior labor productivity and the cost effectiveness of large scale surface mining techniques and equipment that have allowed the exploitation of the low grade ore deposits that characterize the major open-pit operations of today.

The predominate haulage method employed in large open-pit mines is truck haulage and is reported to represent 45% of total direct pit costs in many open-pits. In most instances this represents trucks operating out of deep pits resulting in relatively long hauls against adverse grades. Every year of operation increases the overall pit dimensions which increases the haulage costs. To compound this problem of increased haul distance, double digit inflation and oil prices are driving up the costs of equipment acquisition and operating costs. Also, labor costs continue to rise in the face of reduced product demands in several mineral industries. While these are not in themselves "new trends," it is felt that the specter of these trends and fuel shortages and environment considerations justifies some modifications to the relative merits of all haulage systems.

From the mid-1950's to today open-pit truck haulage has evolved from small 50-ton tandem axle side dump units to 350-ton single axle end dump units. It has been this phenomenal growth in capacity and improved mechanical efficiency which has not only offset but, in many cases, reduced the total direct

mining cost of surface operations. It seems unlikely that the next 20 years will see such a continued increase in capacity and efficiency of the equipment will be able to offset the observed trends in truck and fuel costs coupled with increasing pit depths.

The demands of industry for handling materials at higher rates over greater vertical and horizontal distances at reduced costs have had very significant effect on other haulage technologies (as well as truck), especially on belt conveyor and skip hoisting. For example, to meet these demands conveyors were designed to handle much coarser materials up steeper inclines and new high strength cable built to relieve some of the problems associated with skip haulage.

The following advantages could be realized by the use of these more continuous haulage systems, belt conveyors and skips, for underground haulage in open-pits:

- Conveyors and skips are capable of high-volume, long distance, low unit costs, and substantial energy savings
- Conveyors are able to handle steep adverse grades
- Fewer men are needed to operate the haulage system
- Conveyor or skip haulage would produce lower noise levels than currently produced by trucks
- Use of an inclined shaft would shelter a conveyor system from climatic conditions

Of course one must realize that not all orebody and pit configurations or mining plans and conditions would permit the economic use of the conveyor and skip haulage systems.

Thus, the evaluation of haulage systems for new open-pit mines and the reevaluation of existing haulage systems must acknowledge and address all design factors, including the final cost sensitivity analysis of the system to:

- Inflation on capital and replacement capital (trucks must be replaced regularly)
- Energy/fuel costs - diesel fuel supply disruptions and projected rate of cost increase

- Labor cost - relative manpower requirements
- Impact of increased haul distance on operating cost
- Ecologic impact
- Design risks and technologic trends

1.2 Study Objective

This study endeavors to determine the state-of-the art of continuous haulage from open-pit mines; and to identify technology gaps, risk areas, mine design considerations and industry acceptance in applying continuous haulage and underground continuous haulage to metal and non-metal open-pit operations.

1.3 Scope of Study

In determining the state-of-the-art of continuous haulage from open-pit mines no geographic, national or mineral industry preferences were observed. Every effort was made to develop as broad a data base as possible, both through comprehensive literature searches and national and international mine visits. The only restriction was that the mine visits or literature reviewed be of open-pit mines (as differentiated from surface coal stripping operations and most sand and gravel pits and quarries). Special effort was directed at identifying and evaluating continuous underground haulage applications employing inclined belt conveyor slopes and horizontal conveyors from below the pit to a vertical shaft using a skip hoist system to the surface.

The applications were identified and then evaluated to establish the:

- Systems advantages and disadvantages
- Limiting condition - such capacity, character and type of material handled, mining and waste disposal methods, climate environmental constraints

- Type of crushing and performance if required
- Types of ore passes or finger raises and problem areas
- Durability and maintenance requirements, ease of excess lubrication, and spillage removal requirements for conveyors and hoists
- Manpower and energy requirements
- When the system was installed - whether at the beginning of the operation or at a later stage of development and the reasoning involved
- Reference and summaries of all site specific economic analysis were available

These continuous haulage applications, once evaluated, were then used in surveying the "Industry Acceptance" of underground haulage. The comments of the nine mine managers contacted were used in preparing Section 4 of this report. The mine managers were specifically asked if continuous haulage concepts were considered at their sites and the reasoning in accepting or rejecting the system.

From all the above mentioned work conclusions pertaining to and recommendations for continuous underground haulage were prepared and presented in Section 6 (Conclusions and Recommendations).

2. STATE-OF-THE-ART IN OPEN-PIT CRUSHING AND UNDERGROUND HAULAGE

Open-pit operation is the predominate mining method used throughout the western world to produce raw material for the metal/non-metal manufacturing industry. Vertical transport of the ore out of these open-pits is invariably an expensive and time-consuming step in the mining cycle. This has influenced mine management over the years to take a more serious look at alternative haulage methods, rather than assuming that the inherent flexibility of trucking ore automatically makes it the least expensive option. One approach, for example, has been to combine conventional truck and shovel operations in the pit with underground crushing and haulage facilities fed by ore passes from the pit bottom.

As should be expected, ore transportation systems for open-pit mines are selected to achieve the lowest cost per ton-mile possible (includes capital investment, operation, and maintenance). For this very reason, several major mining operations already have or are beginning to look at alternatives to truck haulage, especially as pit depth increases and personnel costs escalate. This section presents background information on a number of open-pits in North America and Europe where alternative haulage methods (other than truck) have been investigated and ultimately employed.

2.1 A Review of Underground Crushing and Haulage in Open-Pit Operation

The geometric relationships of the ore deposit - type, shape, size, geology, and surface topography - are the site specific criteria which determine whether underground crushing and haulage in conjunction with open-pit operation make practical economic sense.

This section takes a look at various open-pit operations which have already employed some arrangement of underground crushing and haulage. This is followed by a detailed review of an iron-ore mine project in Labrador which gives a feeling for the design criteria and cost objectives which make this type of mining approach attractive. A brief summary of current open-pit practice in the U.S. comes next with specific reference being made to operations that have adopted continuous ore haulage methods. Finally, a concluding remark forecasts the tentative near term future for future implementation of these ore haulage techniques in other U. S. mines.

2.1.1. Categorization of System Types

The three essential elements to any underground activity in support of open-pit mining include:

- Ore passes from pit bottom to the underground
- Underground or surface crushing facility
- Underground ore transport system to the surface

Ore passes can have the following various configurations:

- Vertical
- Inclined
- With knuckle at bottom
- With inspection raises and drifts
- With bulldozing chamber and storage bins
- A combination of the above

Options for where to crush the ore include:

- In-pit immediately after excavation
- Underground
- No crushing at all (ore fragments suitably reduced during blasting)

Methods of transporting ore from the underground include:

- Conveyor
- Rail
- Shaft
- Truck (not typically a viable option)

Table 2-1 lists several mines which have employed some combination of ore pass, crushing, and ore transport in an underground environment. Sketches of the underground lay-outs at these mines are provided in Figures 2-1 through 2-7. More detailed illustrations and descriptions of the mines visited as part of the program are presented in Section 2.2.

The following section briefly reviews the layout at an iron-ore mine in Labrador and presents an overview of the engineering design and cost analyses which influenced the selection of underground crushing and haulage to support the open-pit operation.

2.1.2 Design Review of an Iron-ore Mine Project--Labrador

The general layout of the area topography at the iron-ore mine site is shown in Figure 2-8. The design layout for the first ore pass and underground transport facilities is shown in Figure 2-9. The principal design criteria influencing the selection of this approach to ore handling included:

- The tunnel would be situated to provide:
  - Future extension to adjacent orebodies. Minimum restriction to pit operations.
  - Use of either skip or truck haulage for the deeper ore in the central portion of the pit.
- The ore-pass collar located at a depth to leave a maximum tonnage of ore available for mining above this elevation. This would take greatest advantage of the truck fleet, which was designed to haul on the level or downhill.

TABLE 2-1

Mines	Ore Pass						Crushing			Haulage			Reference
	Vertical	Inclined	Knuckle	Bulldoze Chamber	Inspection Raise	Combination	In Pit	Underground	None	Conveyor	Rail	Shaft	
Iron-ore, Ontario	X		X					X		X			Figure 2-1
Iron-ore, Ontario		X							X	X		X	Figure 2-2
Copper, Mexico	X							X		X			Section 2.2.5
Iron-ore, Labrador	X	X	X		X	X			X		X		Figure 2-3
Copper, British Columbia		X	X		X				X		X		Figure 2-4
Granite Quarry, Idaho	X							X		X			Section 2.2.6
Iron-ore, Sweden	X						X				X	X	Section 2.2.1
Copper-zinc, Finland	X	X						X				X	Section 2.2.2
Copper, Spain	X							X		X	X		Section 2.2.3
Iron-ore, Minnesota	X						X			X			Figure 2-5
Iron-ore, Minnesota	X						X			X		X	Figure 2-6
Lead-zinc, Ireland		X						X				X	Section 2.2.4
Copper, Arizona		X		X	X				X		X	X	Figure 2-7

Mines which have employed some combination of ore pass, crushing and ore transport in an underground environment.
(Ref. 2.1.1) © *AIME, 1968*

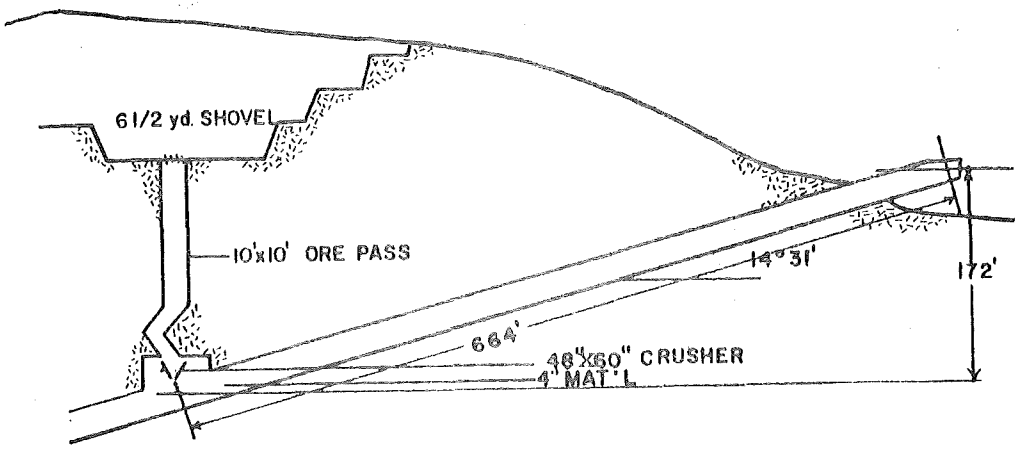


Figure 2-1: Iron-ore Mine, Ontario (Ref. 2.1.1)

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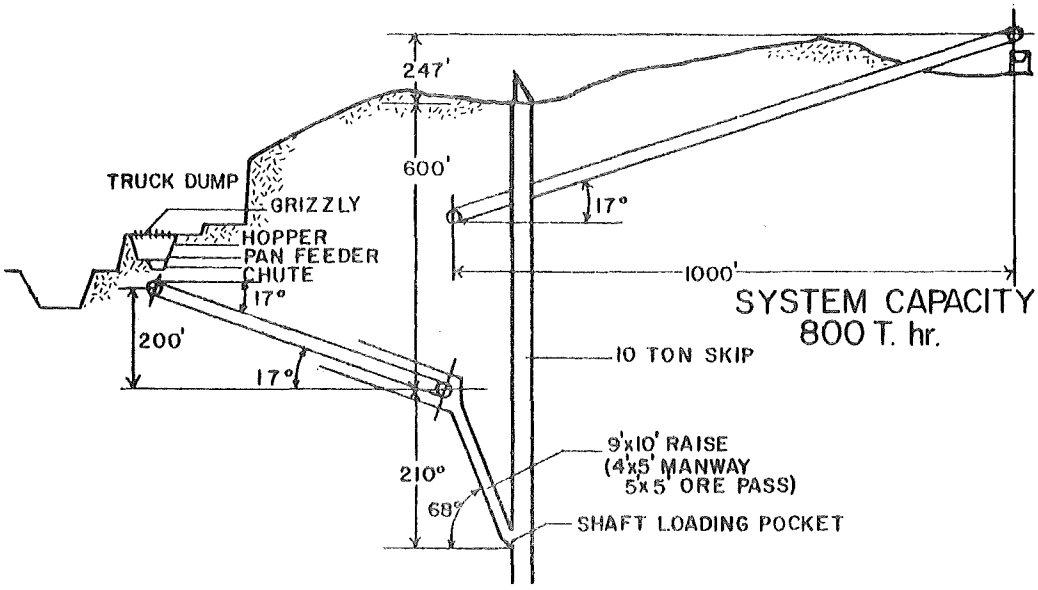


Figure 2-2: Iron-ore Mine, Ontario (Ref. 2.1.1)

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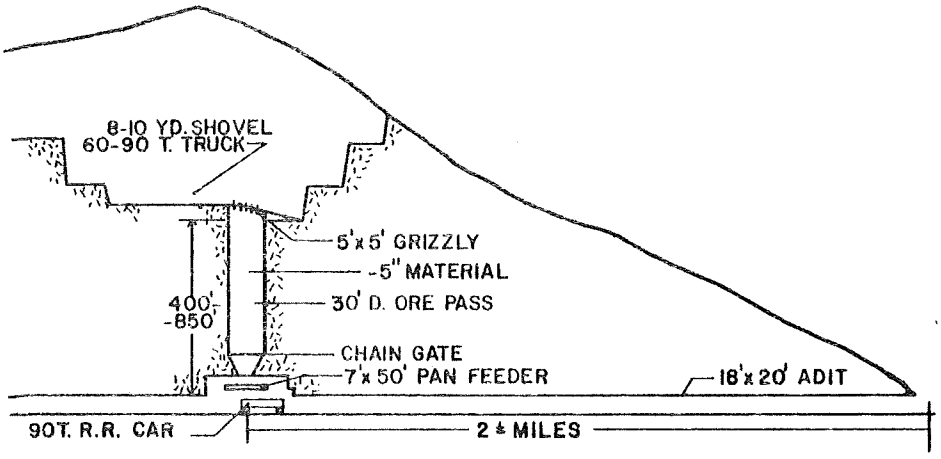


Figure 2-3: Iron-ore Mine, Labrador (Ref. 2.1.1)

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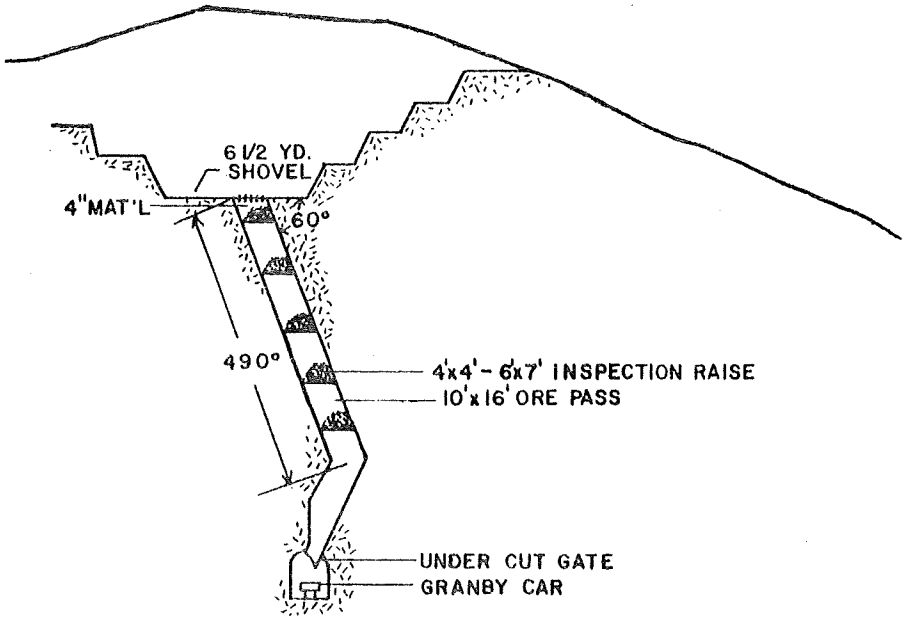


Figure 2-4: Copper Mine, British Columbia (Ref. 2.1.1)

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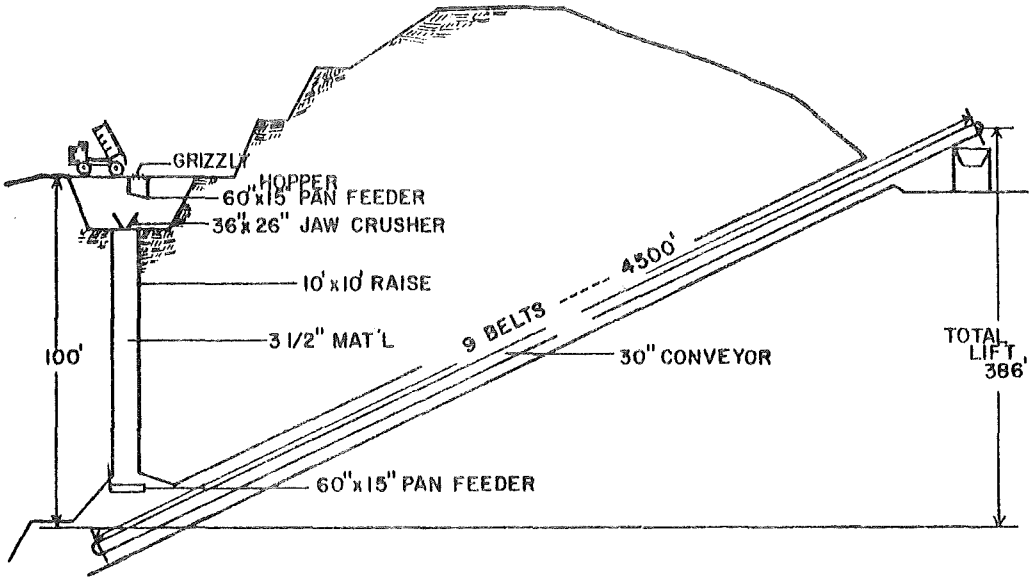


Figure 2-5: Iron-ore Mine, Minnesota (Ref. 2.1.1)

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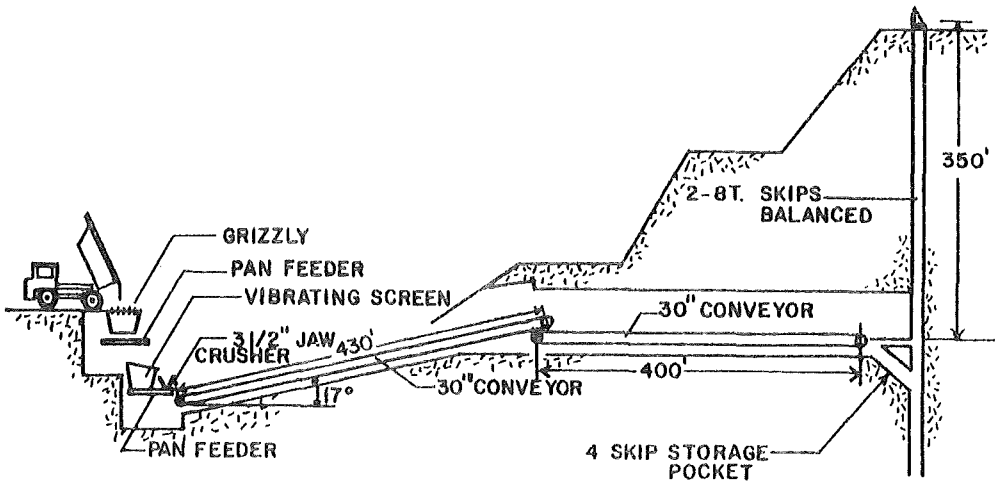


Figure 2-6: Iron-ore Mine, Minnesota (Ref. 2.1.1)

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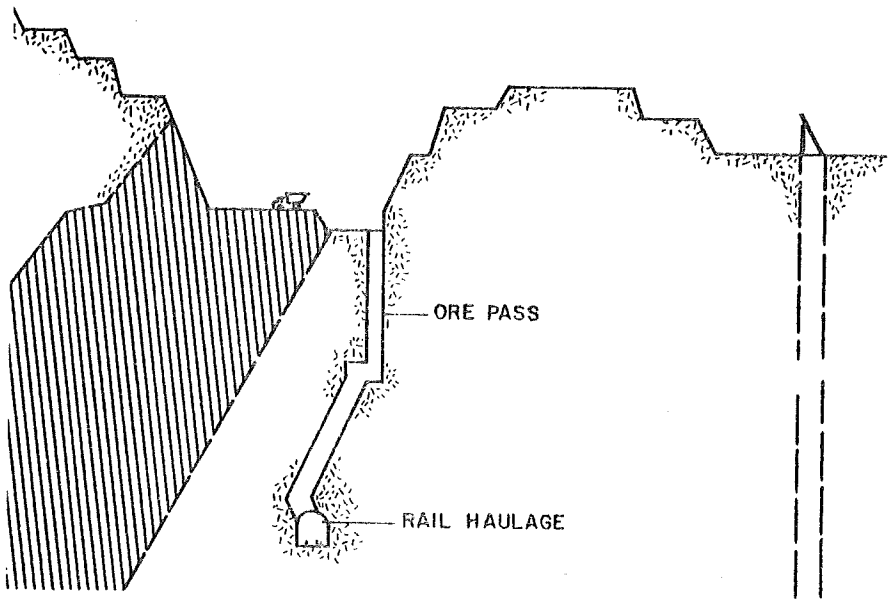


Figure 2-7: Copper Mine, Arizona (Ref. 2.1.1)
© AIME, 1968

- A minimum amount of ore would be tied up on pillars to protect the tunnel during normal pit operation.
- The tunnel grade would be kept at a minimum to favor the movement of the loaded trains.
- All possible mechanical and electrical equipment would be placed underground for protection from blasting.
- Surface structures would be designed as temporary installations since they would be lowered at least twice during the life of the open-pit.
- The ore pass design would incorporate sufficient features to serve as a model prototype for future installations in adjacent orebodies.

A comparison of the estimated costs for the various ore handling approaches which were considered is shown in Table 2-2.

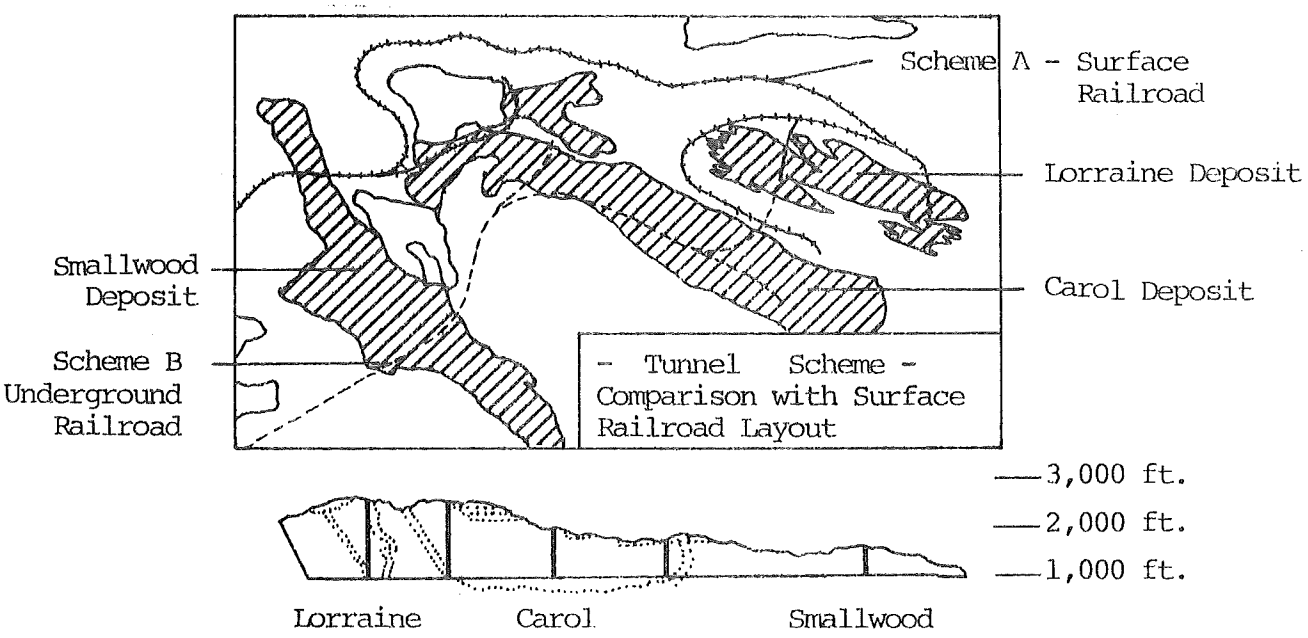


Figure 2-8 Preliminary Design of Rail Haulage Tunnel Scheme-Comparison with Surface Railroad Layout (Ref. 2.1.1) © *AIME, 1968*

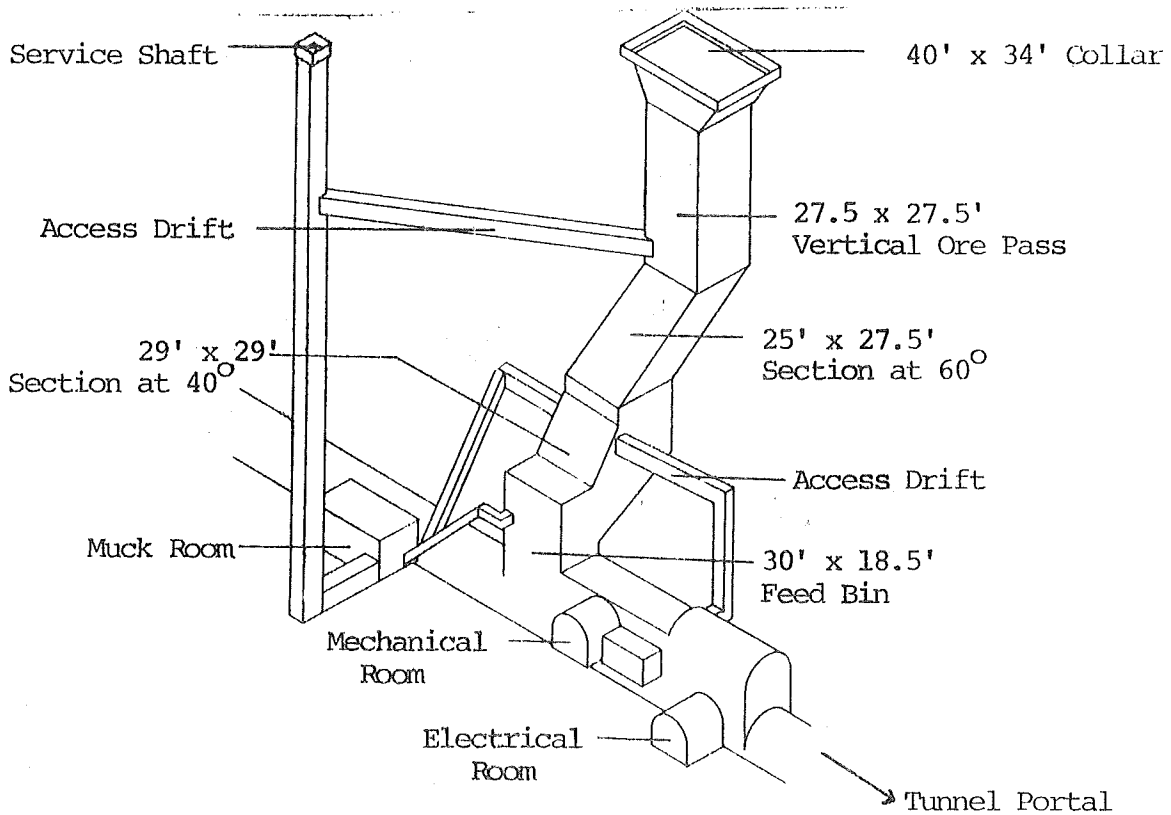


Figure 2-9 Design Layout for Ore Pass and Underground Ore Loading Station for Rail Haulage System at Iron-Ore Project in Labrador (Ref. 2.1.1)

© *AIME, 1968*

TABLE 2-2

COMPARISON OF ORE TRANSPORTATION METHODS AT AN IRON-ORE MINE PROJECT, LABRADOR

(Ref. 2.1.2) © *AIME, 1968*

	Truck-Ore Pass-Tunnel	Truck-Ramp-Surface RR	Direct Into Surface RR
Capital Requirements[1]			
Railroad Trackage:			
Main Line (miles)	6.6	10.1	10.1
Yard and Benches (miles)		11.2	16.6
Tunnel (miles)	3.8		
Rolling Stock:			
Locomotives	4 (60 tons each)	7 (125 tons each)	10 (125 tons each)
Cars (90 short-ton capacity)	95	220	220
Trucks:			
40-ton Rear Dump[2]		10	
60-ton Trailer Type	11		
Ore Transfer Raises	4		
Personnel Required for Ore Transportation Only	94	120	98
Cost Estimates[3]			
Initial Capital Costs (millions of dollars)	15.1	15.7	17.4
Relative Unit Costs:[4]			
Capital Write-off Including Interest (%)	35	37	37
Operating Labor (%)	15	20	16
Operating Supplies and Maintenance Charges (%)	50	64	52
Total *Relative* Unit Costs (%)	100	121	105

[1]Capital requirements are based on estimated production rate of 13,400 tons per shift

[2]40-ton rear dump trucks used only in opening-up benches.

[3]Cost estimates are based on an initial annual capacity of 5.5 million long tons of concentrates

[4]For comparative purposes, unit costs of the three systems are given as percentages adjusted to a base represented by the "Truck-Ore Pass-Tunnel" system.

2.1.3 An Overview of Ore Haulage Approaches in U. S. Open-Pit Mining

The evolution of materials handling equipment in open-pit mining has resulted in larger and more efficient machinery. In non-metal mining operations which typically have softer ore and overburden, excavation is carried on with ever-larger draglines and scrapers. In most large metal ore mines, shovel loading and truck hauling still predominate, but a few move ore by rail and/or conveyors. For loading equipment, massive 25-yard capacity electric shovels are now in use. In trucks, the most widely used are the 170-ton capacity or smaller, but new 350-ton models (Terex 33-19) are now in their third year of test-use and more of these units are expected to be seen. However, with deepening pits the feasibility of trucks must be weighed against a belt conveyor system. In fact, some mine operations have already converted to belt conveying systems resulting in definite economic advantages.

At a mine in Florida, for example, a 2.75-mile belt conveyor system moves deslimed phosphate matrix to the plant on the top belt, and flotation plant sand tailings to reclamation areas on the return side. The 54-inch belt travels 885 fpm, and is driven by steel-belted radial tires in 11 drive modules spaced an average of 2,100 feet apart. This system was designed and built by Continental Conveyor Company and B. F. Goodrich.

In Arizona's surface mines, 170-ton capacity trucks continue to be the prime ore movers. Electric shovels have increased to 20 cu-yd capacity and should remain at that size for the next two decades. Rail haulage has proved efficient and profitable at one open copper pit and, considering the rising cost of diesel fuel, may again be a worthwhile consideration for other large open-pits. Unit trains are typically made up of either six or eleven 80-ton cars for the average 3.4-mile, one-way haul to the crusher and 5.8-mile haul to the waste dumps--distances exceedingly costly for trucks.

At another copper mine in Arizona, a new in-pit ore crushing and conveying system for waste disposal has been erected because of the high tonnage of waste encountered at deeper levels. This system remains a model for other pit operators to study. Previously, all ore from benches was trucked to two 60- by 89-inch gyratory crushers and then conveyed to the mill. All waste was trucked beyond the pit periphery and dumped. A new system required relocation of existing equipment and the purchase of two new crushers, new conveyors and belt, a moveable stacker and ancillary facilities. The changeover reduced truck fleet requirements by 25% and, for this reason, further moves of the crushers are planned as the pit deepens. (See Section 2.2.8 for a more detailed description.)

2.1.4 Conclusion

Despite the above examples, the U. S. is still a long way from embracing the concept of underground crushing and haulage as a viable element in open-pit operation. The closest present mines come to this approach is the use of continuous conveyor haulage out of the pit as mentioned above. The next section presents several examples of foreign operations which have installed and effectively operated underground facilities to realize a lower total mining cost per ton.

2.2 State-of-the-Art Survey

2.2.0 General Discussion

Ten mining sites in Europe and North America were visited to assess the state-of-the-art of continuous haulage from open-pits with emphasis on underground haulage as part of open-pit operation in the hardrock mining industry. Four of the European mines practice, or practiced, combined open-pit and undergound mining (locations are shown in Figure 2-10). These operations are discussed first in this section and include:

- Kiruna Mine - Sweden
- Pyhasalmi Mine - Finland
- Open-pit Mine - Spain
- Lead-Zinc Mine - Ireland

Of the remaining six mines, four are open-pit operations, one was a rock quarry operation for a major dam project, and one is an on-going limestone quarry (locations are shown in Figures 2-10 and 2-11):

- Cananea Mine - Mexico
- Quarry at Dam Site - Idaho
- Nordcement Hannover's Alemannia Works - West Germany
- First Open-Pit Copper Mine - Arizona
- Svappavaara Mine - Sweden
- Second Open-Pit Copper Mine - Arizona

Tables 2-3 and 2-4 summarize the above mines and the reasoning why they were visited in connection to this project. They all were unique in their haulage methods and the rationale why the systems were installed.

In each mine description the presentation is organized to first familiarize the reader with the current operation (history, ore characteristics, mine layout, etc.) and then to discuss the advantages and disadvantages of using

TABLE 2-3

Open Pit Mines Converted to Underground Mines
With Extraction Through Ore Pass Systems

Mine Name	Location	Ores	Present Haulage	Crushing	Comments
Kiruna	Sweden	Iron	Rail and Vert. Skips	Underground	Old open-pit surface crushers to ore pass - rail - vertical skips
Pyhasalmi	Finland	Copper Zinc Iron	LHD's and Vert. Skips	Underground	Surface ore pass to underground crusher to vertical skips
Open-Pit Mine	Spain	Copper Pyrite	Trucks and Conveyors	Underground	Bottom of open-pit to truck adit, underground ore pass, underground crusher, conveyors, rail haulage to surface
Lead-Zinc Mine	Ireland	Copper-Lead Zinc Silver	LHD's and Vert. Skips	Underground	Trucks to surface ore pass, LHD's, to underground crusher, conveyors, and vertical skips

TABLE 2-4

Open Pit Mines - Various Continuous Haulage Methods

Mine Name	Location	Ores	Present Haulage	Crushing	Comments
Cananea	Mexico	Copper	Trucks and Conveyors	Underground	Trucks to surface ore pass, underground crusher, conveyor
Dworshak Dam	Idaho	Aggregates	Trucks and Conveyors	Underground	Trucks to surface ore pass, underground crusher, conveyor
Alemannia Works	Germany	Limestone	Dozers and Shovels and Conveyors	Surface	Dozers to electric shovels feed in portable surface crusher and conveyors
Svappavaara	Sweden	Iron	Shovels and Trucks	Surface	Conventional shovel, truck to surface plank, future underground mining and haulage
Two Mines	Arizona	Copper	Trucks and Conveyors	Surface	Conventional shovel, truck operation to in pit crushers and conveyors

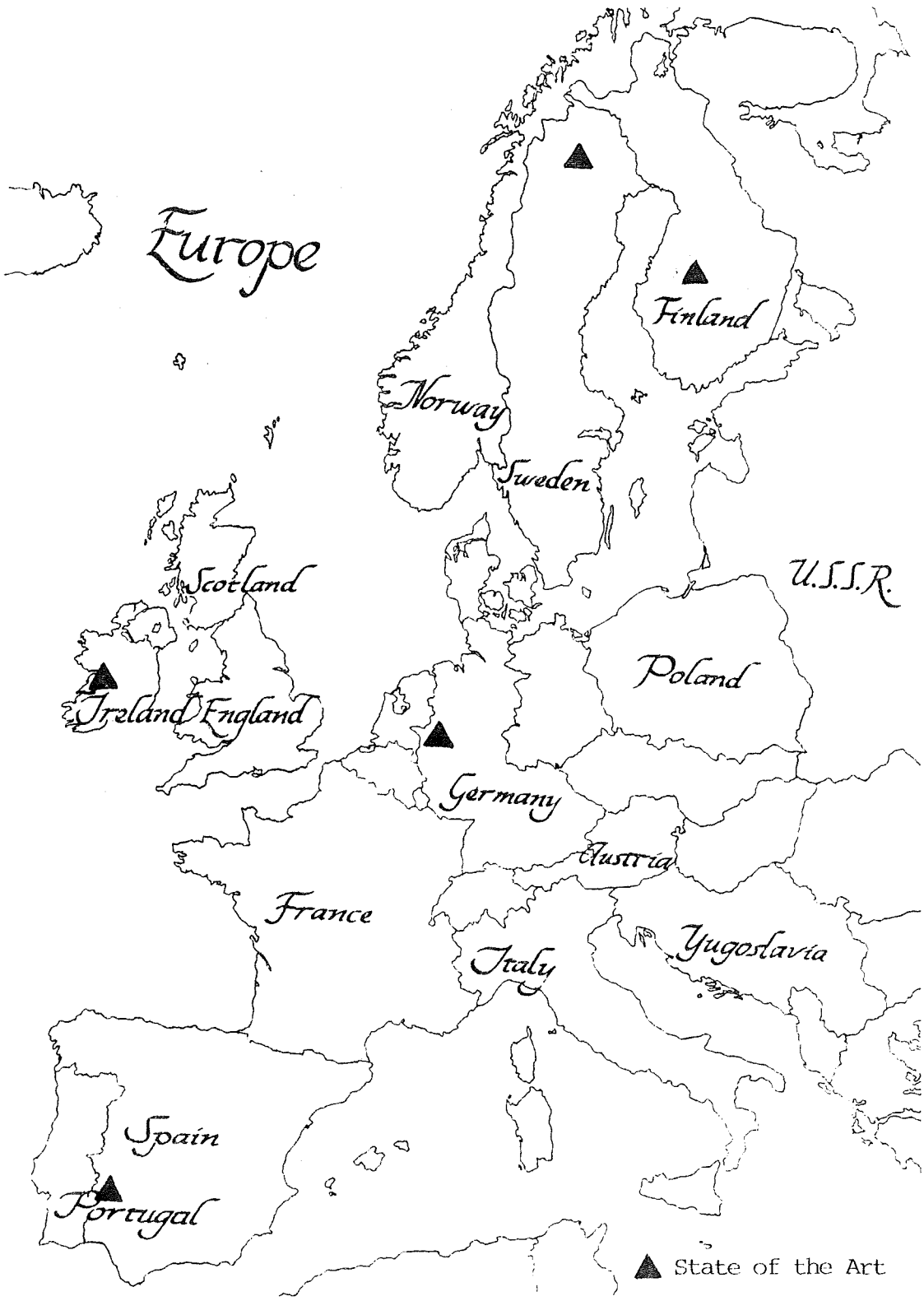
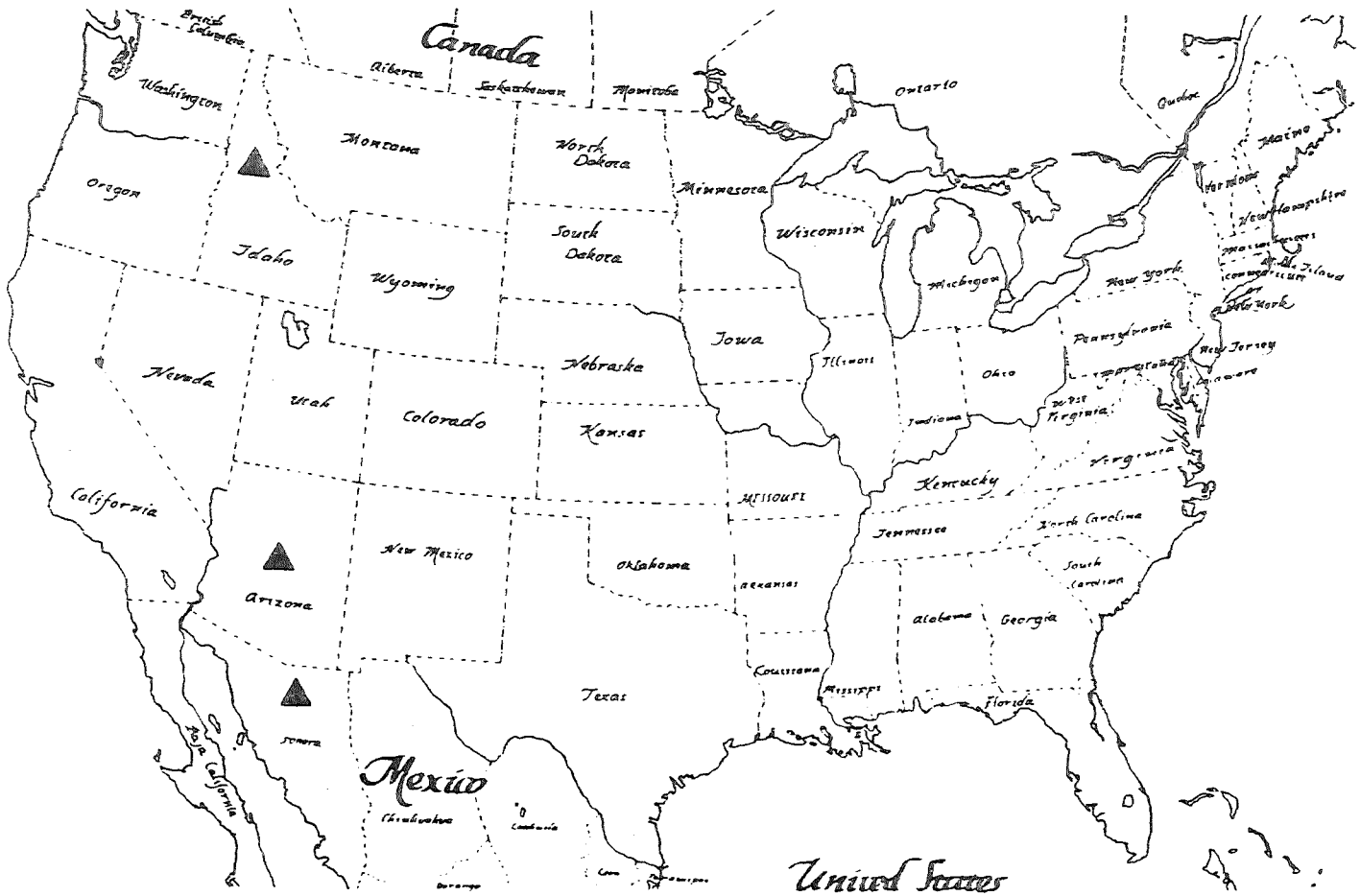


Figure 2-10: Map of the Mine Sites Visited in Europe Where Underground Crushing and Continuous Haulage are Used in Conjunction with Open-Pit Operations



▲ State of the Art

Figure 8-11: Map of Mine Sites Visited in North America to Review and Discuss Use of Underground Crushing and Continuous Haulage in Conjunction with Open-Pit Operation

underground crushing and haulage (manpower requirements and productivity, risk areas, economics, etc.). The specific breakdown of topics goes as follows:

- General Mine History and Orebody Characteristics
- Mine Layout
- Mining Operation
- Mining Equipment
- Durability, Maintenance, Manpower, and Energy
- Risk Areas and Technology Gaps
- Rationale for Mine System Selection -- Constraints and Economics
- Conclusion

The discussion of risk areas and technology gaps in the case of each mine attempts to pinpoint unknown factors which caused considerable concern to management in planning new mine developments or making changes to the existing operation. For example, a forecast of future equipment trends in the 1950's would rarely have envisioned the large-sized haulage trucks used in open-pits today. Nor would the increases in equipment productivity have been fully appreciated to the extent that very low-grade orebodies are now economical candidates for mining. By the same token, there are unforeseen developments yet to come, such as with conveyors (stronger and faster belts), which may make this continuous haulage method more attractive compared to dump trucks.

In summary, this section brings to light the more significant parameters impacting on future mine productivity, and for which there may be no single value, but more likely a range of possible valves.

References and acknowledgements (to include mailing address, phone number, and telex number) have been provided in the appendices for most of the mines visited.

2.2.1 The Kiruna Mine - Sweden

Kiruna stands out not only as a model of modern technological innovation in underground hardrock mining, but perhaps more as the best example of the complexities confronting mine management in timing the switch over from open-pit to underground operations. Following the decision to go underground in 1937, development progressed to the point where by 1954 the majority of ore production could be obtained underground. However, a marked increase in world ore prices at that time influenced management to keep the pit active for an additional 17 years. Moreover, had management foreseen the technological improvements to come in the 1960's with large scale equipment, they might have postponed this decision indefinitely. Nonetheless, the Swedes have persevered to create one of the largest and most productive underground operations in the world, with tonnage output that rivals most open-pit mines. Figure 2-12 is a schematic of the pit operations feeding the underground haulage system. Table 2-5 lists the principal characteristics of the present mining and ore haulage activity.

The Kiruna operation has demonstrated three important points to the mining world.

- Large tonnage can be handled from an open-pit through underground.
- Large tonnage can be automated for underground rail haulage.
- Vertical hoisting of ore was more economical by Kiruna's calculations than the use of inclined conveyors in the deeper mine ($\triangleright$ 500m). (This mine has been in operation for many years and the investment in the facilities for skip hoisting was already depreciated.)

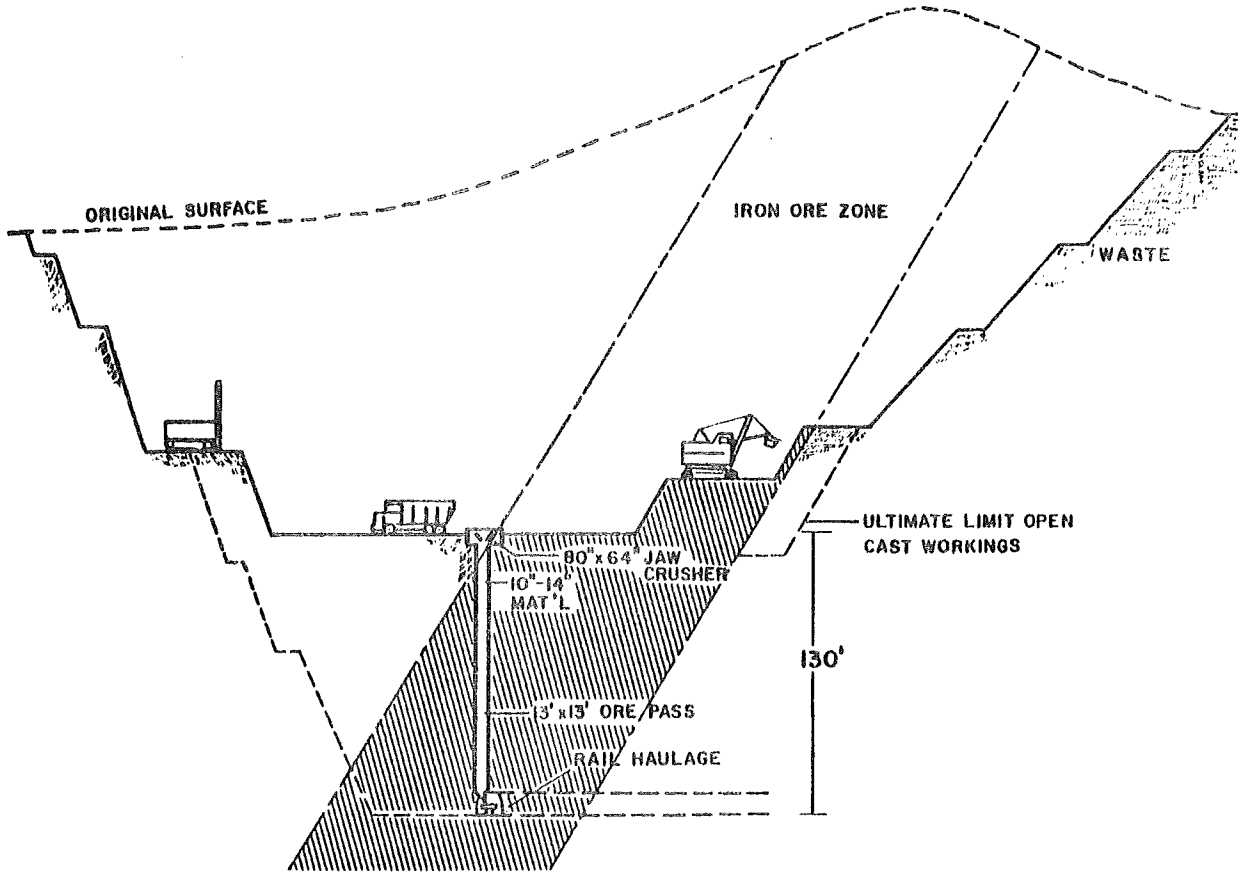


Figure 2-12 Schematic of Pit Operations Feeding the Underground Ore Haulage System (Ref. 2.2.1)

© *Mining Congress Journal, 1972*

TABLE 2-5

FACT SHEET ON THE KIRUNA MINE

- Owner - Luossavaara-Kirunavaara AB (LKAB), a Swedish Government-owned company
- Location - In northern Sweden 90 miles north of the Arctic Circle, 110 miles southeast of Narvik, Norway
- Ore - Iron-Ore
- Pit Operations Started - 1892
- Underground Operations Started - 1938
- Pit Operations Ceased - 1971
- Current Production Rate - 18,000,000 tpy
- Full Production Potential - 30,000,000 tpy or 130,000 tpd
- Potential Mine Life - 50 to 100 year
- Operating Personnel - 1,200 to 1,500 underground
- Equipment:
  - LHDs tend raises/chutes and feed ore into a rail system ahead of the crusher circuit.
  - Crushed ore is hoisted to the surface by vertical skips.
- Operations:
  - Largest underground mine in the world. Highly automated operation. Planning effort under way for hoisting methods.
  - Open-pit mining fed the present underground system until 1971.
- Problems:
  - Despite good productivity, operation is still losing money due to depressed world market for iron ore.

2.2.2 Pyhasalmi Mine - Finland

The key point to note about the relatively new and modern Pyhasalmi Mine is that it was originally designed as a combined surface/underground mining operation (Figure 2-13). Transition to total underground activity (as of 1975) gradually took place during the first 13 years of production. Ore crushing and haulage have from the outset been accomplished underground.

After inspecting the Pyhasalmi operation collective impression of the experienced mining engineers was unanimous on the model-like nature of what had been seen. Not only had the latest mining equipment and techniques been selected, but also top quality managers had been brought in to both plan and run the operation. A number of different mining methods had been practiced to accommodate the varied geological distribution of the deposit. In keeping with a well-planned effort, an underground trial stope was initiated early on after commencement of pit operation to establish the feasibility of longitudinal sublevel stoping without pillars. In addition, a combination of hydraulic and cemented backfilling were employed to aid in ground control.

Another example of the foresight used in carefully planning the mine operation is the dust control procedure for the ore passes, a potential source of high dust levels. High ventilation air counter pressures are maintained at the ore pass drawpoints to force air movement back up into the boreholes, thereby facilitating a more gradual settlement of the dust laden air.

Open-pit activity has virtually ceased, except for the backfill operation with underground mine waste which is first hoisted to the surface via the auxiliary shaft (shown to the left). Ore is hoisted from underground in the main shaft (shown on the right of center in the figure) and processed on site in the milling facility prior to final shipment. Table 2-6 briefly summarizes some of the significant facts on the mine.

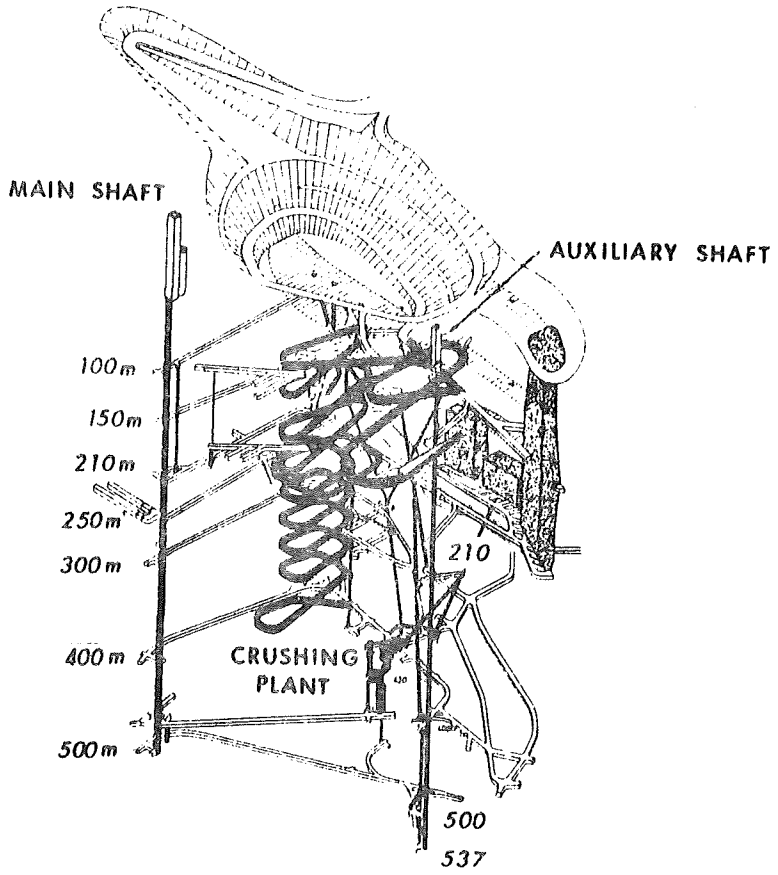


Figure 2-13 Isometric View of the Pyhasalmi Mine (Ref. 2.2.2)

In conclusion, we would compare the Pyhasalmi mine to a well-engineered, finely-tuned machine. Without a doubt, it is the most noteworthy and innovative of the mines visited during this program and is living testimony to the inestimable value of a thorough planning effort.

TABLE 2-6

FACT SHEET ON THE PYHASALMI MINE

- Owner - Outokumpu Oy (Finnish Government)
- Location - 150 km northwest of Kuopio, Finland
- Ore - 0.7% to 0.8% Cu, 2.15% Zn, 33% Fe, 38% S
- Mineralization - chalcopyrite, pyrite, sphalerite
- Pit Operations Started - 1962
- Joint Underground Operations Started - 1965
- Pit Operations Ceased - 1975
- Production - 900,000 tpy
- Potential Mine Life at Present Production Rate - 20 years
- Operating Personnel - 500
- Operations:
  - Open-pit (now defunct) and underground working faces feed ore through raises to drawpoints.
  - Ore hauled via LHD from drawpoints to jaw crusher, then via conveyor to ore skips in #1 shaft.
  - #2 shaft handles hoisting of waste rock and provides ventilation air.
  - Sloped ramp at 8^{o} to the underground is used to transport large equipment and materials.
- Equipment:
  - Two-stage jaw crusher and vibratory feeder system is automated and monitored by television camera.
  - Friction hoist employs a single 10.8-ton skip running at 8.5 m/sec for 530 m. Men and material are hoisted in the same shaft.
  - 5 cu m LHDs haul ore from ore pass drawpoints to the crusher.

2.2.3 Open-Pit Mine - Spain

Problems of significant magnitude currently plague the combined pit/underground operation at this Spanish mine. Pit design, handling problems related to the wet ore, and excessive rail transport distances are some of the major difficulties impacting on mine productivity. Future world interest will focus on the effectiveness of the corrective measures currently being implemented by mine management, especially with respect to the use of inclined conveyors and rail transport for continuous underground haulage.

There are three separate mine workings at the Spanish mine complex. One open-pit mine is abandoned; the second is a very old combination open-pit/underground operation; and the third is a relatively new open-pit. This section will deal principally with ore haulage at the second open-pit mine, and to a lesser extent with the newer open-pit mine. Figure 2-14 provides a cross-section view of the haulage system at the former while Table 2-7 lists some of its principal characteristics.

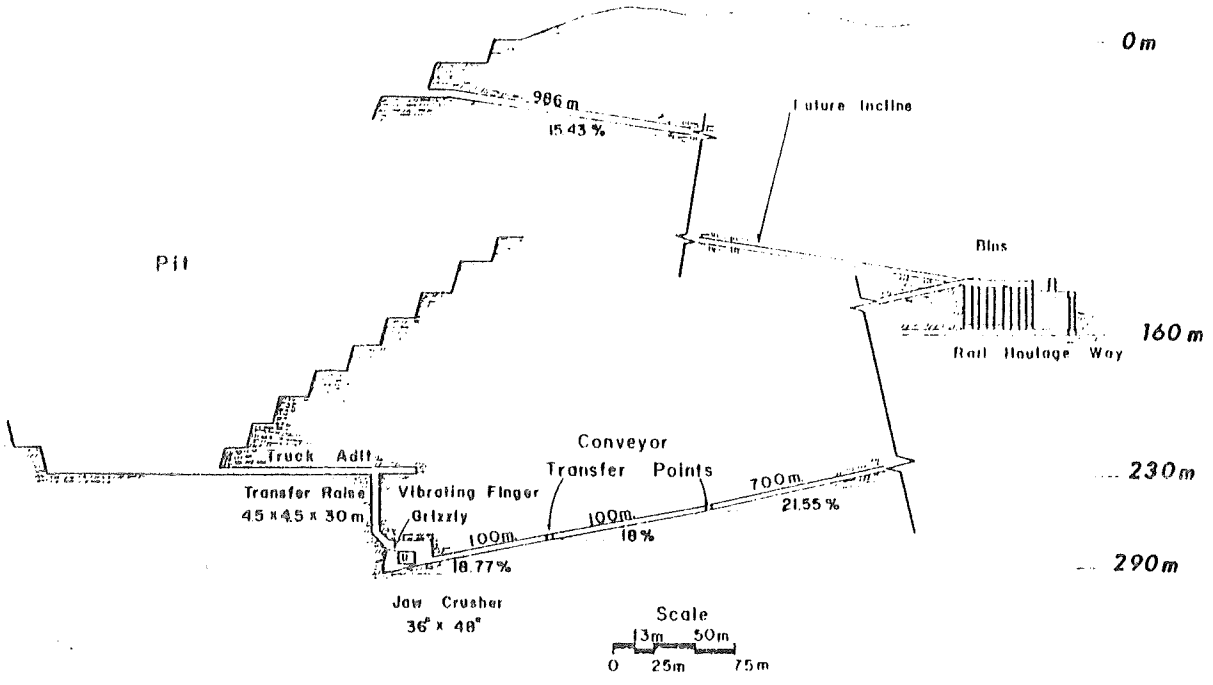


Figure 2-14 Layout of the Spanish Underground Mine Adjacent to the Open-Pit Operation

This open-pit mine is clearly an example of the type of ore deposit which lends itself readily to underground crushing and haulage.

TABLE 2-7

FACT SHEET ON THE SPANISH MINE

- Location - northwest of Seville in southwestern Spain
- Ore - Pyrite and .25 to .7% Cu -- Leaching of mill tailings produces 19% Cu
- Mineralization - Chalcopyrite and calcocite
- Pit/Underground Operations Started - Roman times
- Combination Open-Pit/Underground Operations Started - 1940
- Production - 6,000,000 tpy by 1980
- Operating Personnel - 262
- Potential Mine Life at Present Production Rate - 20 years
- Operations (Open-Pit):
  - Pit ore is trucked into an adit at the pit bottom to a raise that feeds a crusher
  - Underground ore is fed via ore pass to a second crusher
  - A long inclined conveyor system from the crushers feed track haulage units
  - Lengthy horizontal rail network to pyrite stockpile (5.5 km) and mill (7 km)
- Problems
  - Size of the adit is limited to small trucks (35 ton)
  - Equipment maintenance is high because of the acid nature of the ground water
  - Ore production cost is high since track haulage system cannot run at full capacity
  - Track haulage route is excessively lengthy

2.2.4 Lead-Zinc Mine-Ireland

An open-pit mining operation was the only activity envisioned when planning efforts commenced after a limited exploration effort. It was not until after surface facilities had been commissioned and erected that the full extent of the irregular shift in the lie of the orebody was realized through a detailed investigation of the normal fault on which the host rock was situated. Physical limits to further pit development (due particularly to the proximity of the tailings pond) dictated the development of an underground operation for the deeper ore reserves. Under these circumstances, a thorough analysis of the potential mining alternatives was undertaken, resulting in the current economical operation which uses underground crushing and vertical skip haulage.

Open-pit activity ceased as of 1973 except for a limited backfilling operation. The underground mine waste, after separation from the ore in the surface mill facility, is redirected through the pit bottom ore passes to the active stopes. Figure 2-15 provides a cross-section view of the pit crushing and haulage from the open-pit.

Table 2-8 summarizes the key features of the current mining operation and points out the major problems experienced in converting from surface to underground.

The Lead-Zinc Mine is an excellent example of a mining operation that converted from surface to underground and remained profitable, despite adverse circumstances such as lack of qualified manpower skills. Moreover, although the total mine life will have been relatively short when operations are discontinued in 1980, the return of investment has justified an extensive underground development effort, including the acquisition of a raise boring machine to construct the shafts and ore passes. Given the disadvantages of inexperience combined with heavy capital outlay, we found an encouraging case in point for justifying the potential success of open-pit operations combined with underground development, ore crushing and haulage.

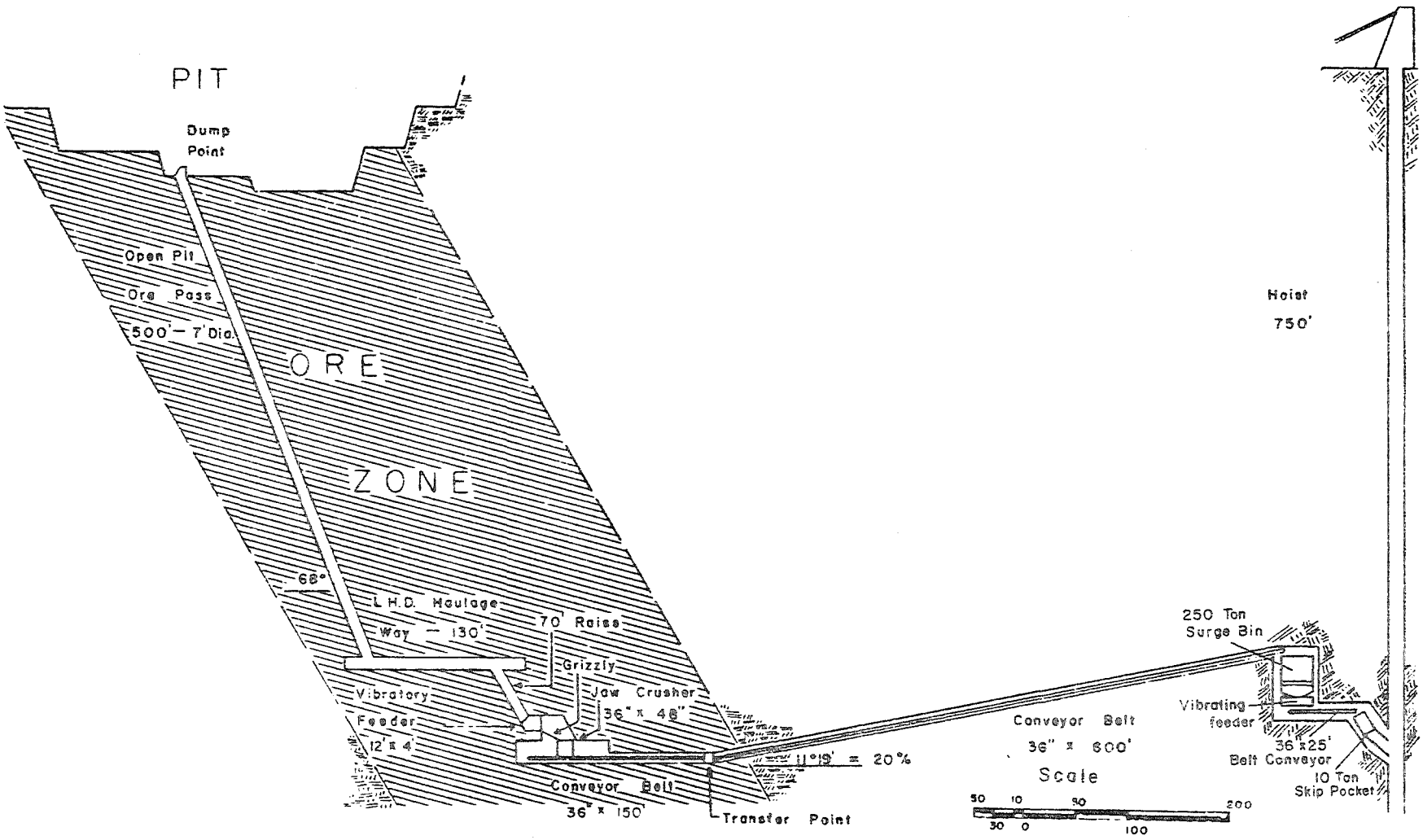


Figure 2-15 Cross-Section of Crushing and Haulage From Open-Pit

TABLE 2-8

FACT SHEET ON THE LEAD-ZINC MINE-IRELAND

- Owner - Irish Base Metals, Ltd.
- Location - 115 miles west of Dublin, Ireland
- Ore - 3.89% Pb, 3.75% Zn, .18% Cu, .9 oz Ag Barite in mill tailings
- Pit Operations Started - 1965
- Underground Operations Started - 1968
- Pit Operations Ceased - 1973
- Production - 600,000 tpy
- Potential Mine Life at Present Production Rate - 2½ yrs. (as of Jan. 1978)
- Operating Personnel - 346 (includes 121 underground miners)
- Operations:
  - -Single ore pass from surface to crusher (7 ft. diameter, 500 ft deep)
  - -Multiple ore passes from underground mining sections to crusher
  - -Skip hoisting of crushed ore to surface
- Equipment:
  - -Ore haulage via LHDs to crusher
  - -Ore and waste are hoisted to surface by 10-ton skip
  - -Unique design for a dump gate at the raise to protect both the trucks from falling in as well as collar caving
- Problems:
  - -Ore would hang up in raise due to its soft, easily compacted nature
  - -Vertical drop of ore damaged chutes at bottom of raise; required conversion to draw points without chutes

2.2.5 Cananea Mine - Mexico

In contrast to the four mines already discussed, the Cananea Mine was a long-standing underground operation which started to convert to open-pit in 1944, but retained the use of an underground crushing and continuous haulage system during and after the transition. Many years of underground experience handling relatively coarse muck through long ore passes had suggested the idea of placing the crusher underground beneath the ultimate bottom of the pit, and using an inclined conveyor to transport the ore to the surface. In addition, investigations into low-cost ore transport and lifting of crushed rock produced in the surface crushing plants had convinced management that conveyors were the most economical haulage medium for their particular needs.

Although most of the open-pit development and underground equipment at Cananea are relatively new, maintenance problems have inhibited the operation from reaching full production potential. But even with recognized design flaws in mine layout added to the maintenance difficulties, tonnage output has still exceeded by 20% the estimated level that would have been achieved with truck haulage. This is strong justification for the decided advantages which continuous haulage can bring to open-pit operation.

Figure 2-16 shows a cross-section of the crushing room. Table 2-9 summarizes the major features of the current operation.

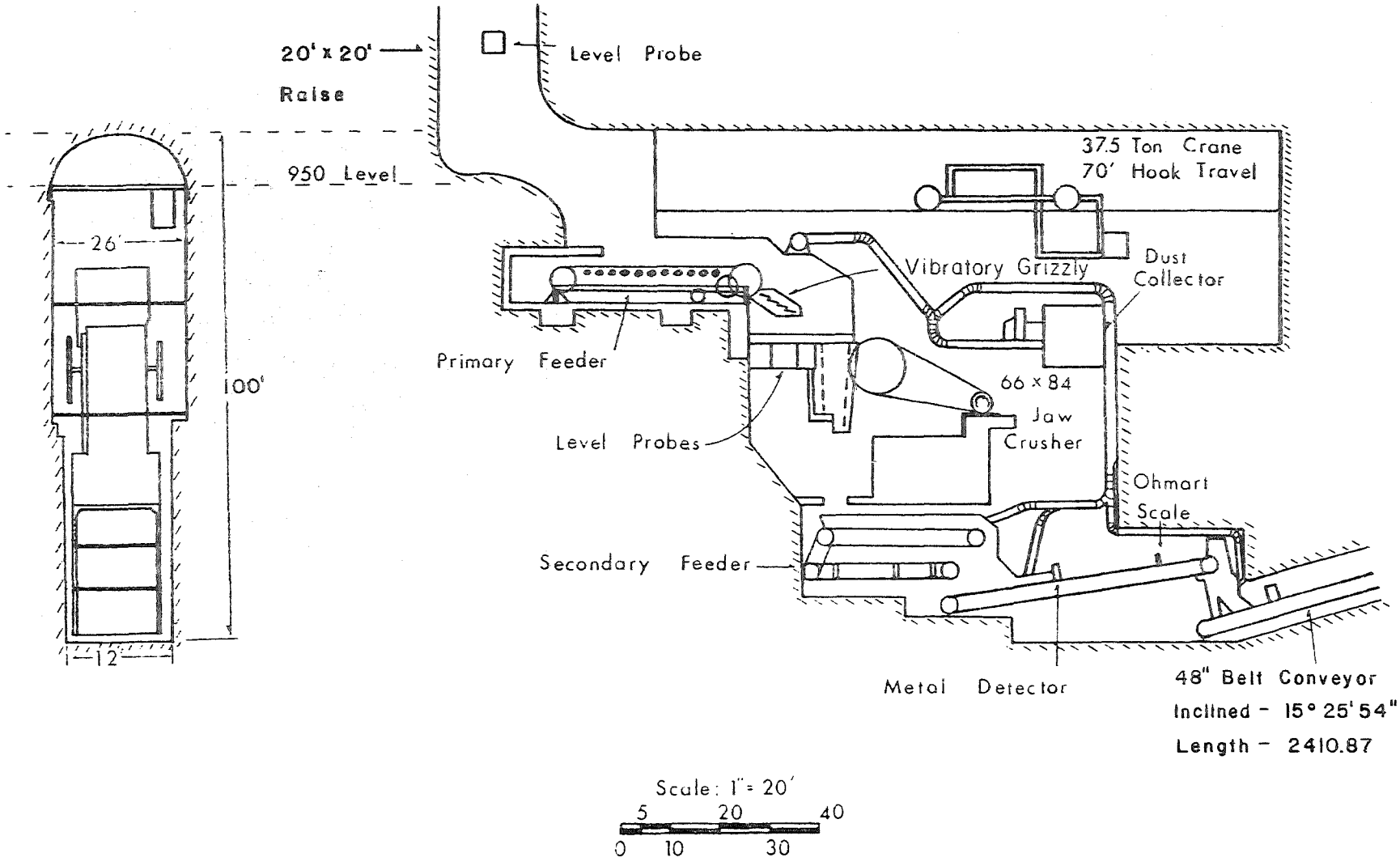


Figure 2-16 Schematic of Underground Crusher Room at the Cananea Mine (Ref. 2.2.3) © *AIME, 1969*

TABLE 2-9

FACT SHEET ON THE CANANEA MINE

- Owner - Compania Minera de Cananea, S.A. (51% - Mexican Government, 49% - Anaconda Corporation)
- Location - Sonora, Mexico, 150 miles south of Tucson
- Ore - .76% Cu
- Mineralization - breccia pipes, granitic intrusive rocks
- Underground Mining Started - 1760
- Opern-Pit Mining Started - 1944
- Production - 20,000 tpd milling; total mining capacity 80,000 tpd
- Potential Mine Life - 75 years
- Operations:
  - Ore from the open-pit goes down through a raise to an underground crusher system, and then via inclined conveyors to the concentration mill.
- Equipment:
  - 120-ton dump trucks in the open-pit
  - Underground jaw crusher
  - Combined underground-overland inclined conveyor network
- Problems:
  - High maintenance costs for the underground facilities
  - Underground facilities located in rich section of ore deposit
  - Underground crusher located too close to pit bottom
  - Erosion in the ore pass due to soft rock
  - Initial high acid water seepage in ore pass and crusher room

2.2.6 Quarry at Dam Site - Idaho

Severe terrain conditions coupled with the close proximity of the proposed quarry site to the dam's left abutment made conventional truck haulage of crushed rock an unattractive approach. Innovative planning based on a thorough study of the area geology led to the design of an underground crushing chamber positioned deep in the heart of the quarry. Continuous belt haulage of the final product through an inclined tunnel to a primary surge pile would feed the required aggregate to the on-site concrete batch mixing plant.

Of course, the above scenario has become history as over 13 million tons of granite gneiss were successfully quarried using the underground crushing operation. This section presents background information on the quarry setup and discusses the thinking that influenced management to accept this novel approach. Figure 2-17 provides a view of the completed dam project and the reclaimed quarry site which is only 600 feet downstream of the dam's left abutment. Table 2-10 summarizes key feature of the rock quarry operation.

To summarize this successful project briefly, it should be noted that a construction wage spiral, which accelerated faster than anticipated when the 6-year project was estimated, hurt the operating costs much less than if a surface crushing facility with truck haulage had been implemented. As stated before underground crushing combined with continuous conveyor haulage can provide excellent results and above average return on investment if used in the appropriate application.

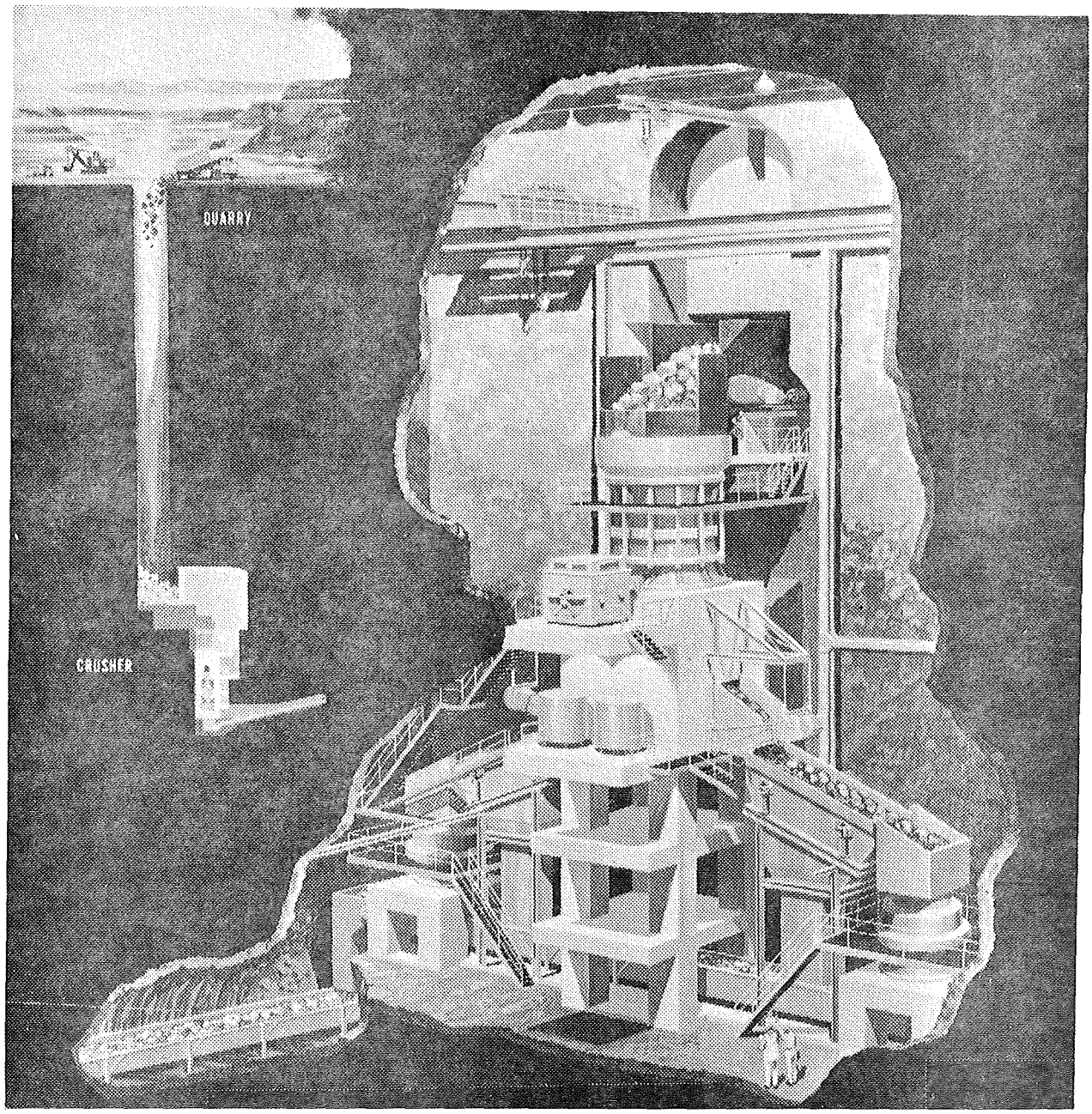


Figure 2-17 *Primary and Secondary Underground Crushing Facilities at The Dam Site in Idaho*

TABLE 2-10

FACT SHEET ON THE OPERATION AT THE QUARRY AT DAM SITE

- Location - North Central Idaho on the North Fork of the Clearwater River
- Contracting Agency - U. S. Army Corps of Engineers
- Ore - Granite gneiss for concrete aggregate
- Start of Quarry Operations - 1966
- Completion of Quarry Operations - 1973
- Quantity Quarried - 13 million tons
- Operations:
  - Truck and shovel feeding vertical shaft on top of hillside quarry.
  - Primary and secondary underground crushing facility.
  - Inclined conveyor in tunnel discharging to outside surge pile near concrete batch mine plant.
- Equipment:
  - 54- x 80-inch primary gyratory crusher.
  - 5½-foot secondary cone crusher.
  - 48-inch wide, 800-foot long conveyor inclined at 10°.

2.2.7 Nordcement Hannover: Alemannia Works - Germany

While there are at present no American users or manufacturers of large primary mobile crushing plants, successful units have already been built and deployed in Europe for the past 20 years. Most of these installations have been in limestone quarries with maximum crusher size to date being a 48-inch gyratory having estimated maximum capacity of 1,000 tph. Only a few of the mobile units have been made with gyratory type crushers, while the majority mount impact-type and jaw crushers. Most of these plants have been used in the cement industry, of which Nordcement has been one of the leaders pioneering this new application.

Experience to date shows that the conservative U. S. mining industry tends to discount the advantages made possible by primary mobile crushing plants with conveyor haulage.

Because of the relatively high production rates required, mine operators feel that any primary crusher installation must be both large and require an extremely high percentage of availability, which of necessity means low maintenance. Consequently, until the confidence in a truly mobile mechanism has been established, the mining industry will lean toward a semi-moveable crusher complete with certain mandatory auxiliary equipment and which need only be moved occasionally in the pit.

European technology is clearly far ahead of the U.S. in this respect. The type of mobile crusher shown in Figure 2-18 has been in operation at Nordcement, Hoever near Hannover, since the early 1960's. Table 2-11 summarizes the principal features of the quarry.

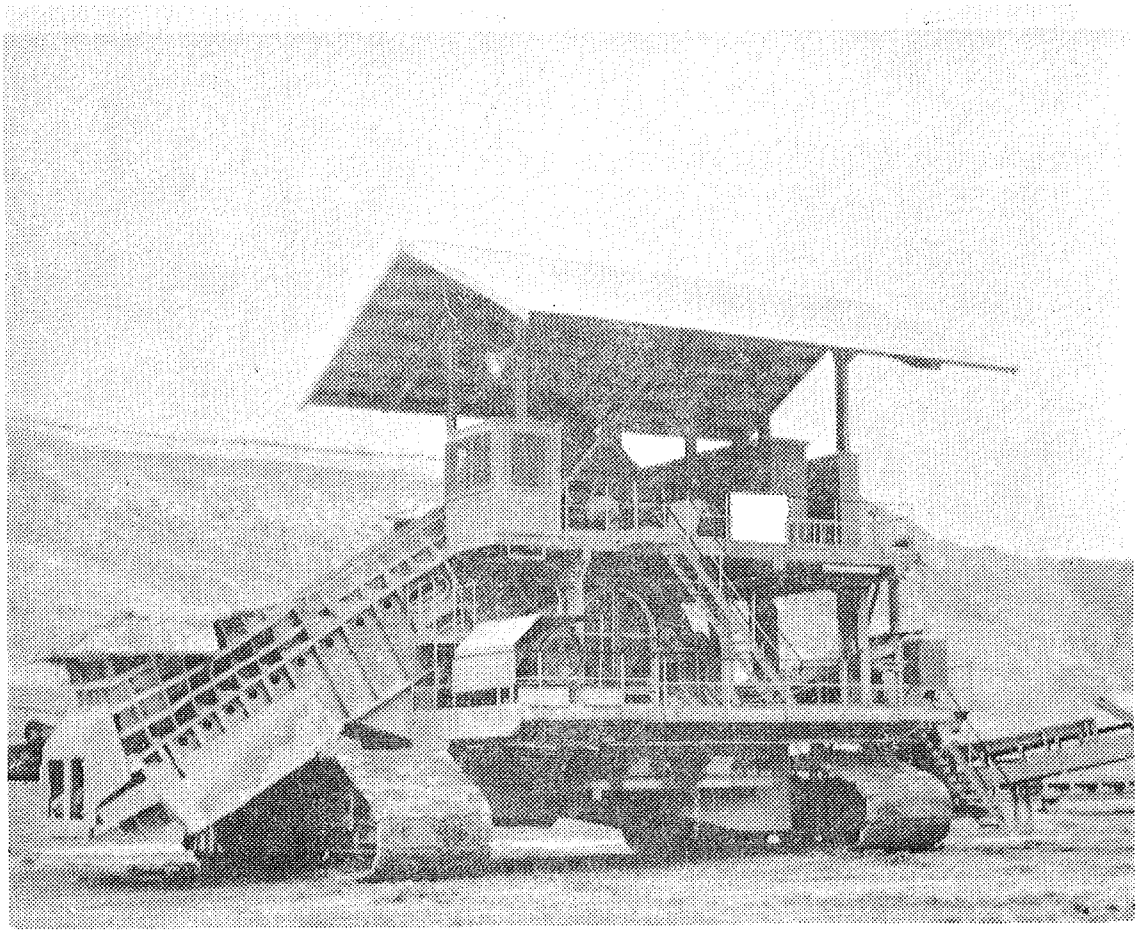


Figure 2-18 *Primary Mobile Crusher (500 tph capacity) Used at the Nordcement Hannover Hoever Works Quarry*

TABLE 2-11

FACT SHEET ON THE ALEMANNIA WORKS

- Owner - Nordcement, Hannover, A.G.
- Location - Hoever near Hannover, Germany
- Ore - Calcareous marl for cement products (75-90% $CaCO_3$)
- Pit Operations Started - Early 1900's
- Production - Maximum pit output of 1,000 tph
- Operating Personnel - 17 equipment operators, 8 per shift
- Operations:
    - Two mobile (impact-type) crushers producing a maximum output of 500 tph each
    - Twin transfer conveyor feeds
    - Two one - meter wide cross pit conveyors
- Problems:
    - Pit floor very soft
    - Ore difficult to mine during wet periods

2.2.8 Open-Pit Copper Mine - Arizona

The increasing interest in the development of low-grade orebodies suggests that, in the case of copper, the average future grade coming from the predominant porphyry type deposits will be on the order of .25%. One particular copper company, which looked at the potential of low-grade mining in the early 1970's, concluded that low-cost continuous haulage of large tonnage production would be essential to cope with fluctuating operating costs and unpredictable world market prices for copper. An engineering-economic analysis showed that the use of a high-capacity conveyor belt running on an incline up the pit wall with in-pit primary crushing offered the best return on investment. This system was subsequently selected and installed versus going to a conventional truck and shovel haulage operation.

This section discusses in some detail the reasons for mine management's decision to favor conveyor haulage over the use of dump trucks. Figure 2-19 gives a view of the current operation, in particular the inclined conveyor rising out of the pit, and Table 2-12 presents a list of principal features about the operation.

The major factor affecting the outcome of the engineering-economic analysis on the different mining systems was the operating cost of the dump trucks--which is very difficult to calculate as a funtion of increasing pit lift. Unit operating costs for crushing and conveying, on the other hand, can be closely estimated by allowing for changes in power consumption for various lifts.

The final "caveat" on this decision-making process involving truck versus conveyor haulage has been well summarized by mine management *(Ref. 2.2.4)*: © *AIME, 1976*

> The long-term consequence of continued truck haulage could be very serious. The possible effects on truck haulage by inflation, fossil-fuel shortage, and high cost of labor have to

be considered. If haulage costs by truck continue up with lift at a geometric rate, they could abruptly shorten the life of the mine. The evaluation of these "irreducible" factors is the vital function of management in the project analysis. It is an essential part of the proper decision.

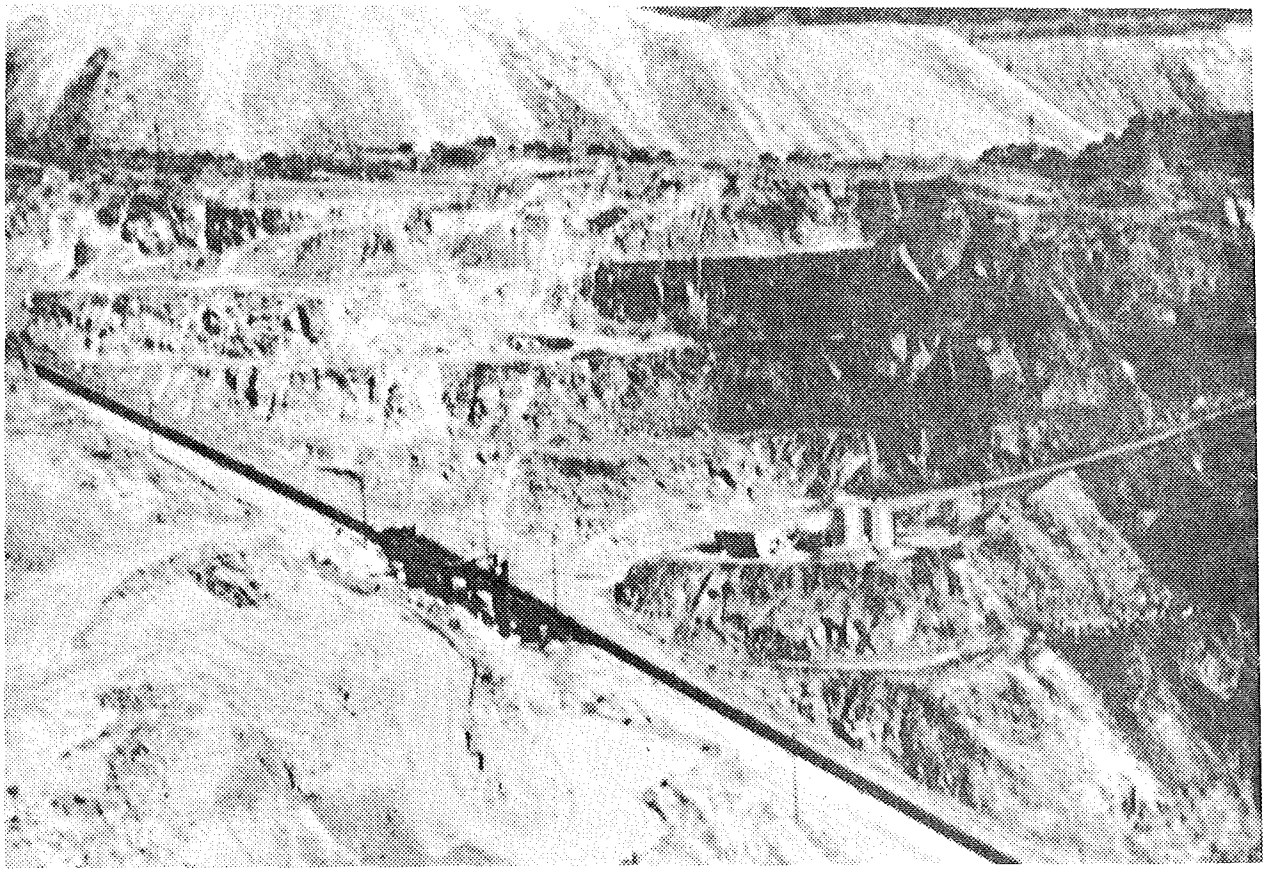


Figure 2-19 Arizona Copper Pit with Conveyor Haulage

TABLE 2-12

FACT SHEET ON THE OPEN-PIT COPPER MINE IN ARIZONA

- Ore - Copper, Molybdenum, Silver
- Mineralization - chalcopyrite
- Pit Operations Started - 1967
- Production - 92,000 tpd of ore, 164,000 tpd of waste rock
- Potential Mine Life at Present Production Rate - 20 years
- Operating Personnel - 2,300
- Operations:
  - Conventional in-pit truck and shovel operation.
  - Inclined conveyor up the pit wall hauls crushed material to the surface.
  - Separate crushing/haulage system for ore and waste rock.
- Equipment:
  - 120 to 170 ton haulage trucks.
  - two double 60-inch gyratory crusher systems.
  - two inclined conveyor systems.
- Problems:
  - Circuitous path of ore conveyor is not an ideal arrangement.

2.2.9 Svappavaara Mine - Sweden

Mine management at Svappavaara, an open-pit operation, working on the Leveaniemi deposit, has already performed an extensive study on the various alternatives concerning the future transition to underground mining. Although results have shown that, based upon currently available equipment and the prevailing world market for iron-ore, pit life should be good for at least 25 years, management believes in maintaining an up-to-date comparative analysis on the underground option. The tell-tale factor has, for the time being, been expressed in terms of a cut-off grade of ore (45% Fe) at which point underground mining should be practiced. With this level of detailed information at their disposal, management feels confident that a timely decision, complete with trial underground mining efforts, will be made.

The long-range planning effort, characteristic of Swedish mining operations, is clearly evident at Leveaniemi (Appendix B.2.1.7). Because of Kiruna's experience in the transition from surface to underground mining during a time of rampant technological development (Appendix B.2.1.1), LKAB will be careful to explore its options before becoming fully committed to the changeover.

The following presents a brief description of current operations at Svappavaara (herein referred to as Leveaniemi) and conclude with a review of management's thinking on how to properly determine the right time for changeover. Table 2-13 lists the principal characteristics of the present open-pit. Figure 2-20 shows an artist's sketch of the fully expanded pit in the year 2000. The existing processing facilities are drawn to scale as shown.

2.2.9.1 Conclusion

What seemed most interesting and important from the standpoint of this program on haulage is the amount of

TABLE 2-13

FACT SHEET ON THE SVAPPAVAARA (LEVEANIEMI) OPEN-PIT MINE

- Owner - Luossavaara-Kirunavaara AB (LKAB), a Swedish Government-owned company
- Location - 47 km southeast of Kiruna, Sweden (north of Arctic Circle)
- Ore - Iron Ore
- Mineralization - Magnetite, Hematite, Ore Breccia
- Pit Operations Started - 1964
- Estimated Start of Joint Underground Operations - 1990's
- Estimated Cessation of Pit Operations - 2,005+
- Production - 14 million tpy - ore and waste
- Potential Mine Life at Present Production Rate - 35 years
- Operation Personnel - 137 (plus an additional 60 belonging to subcontractors)
- Operations - Conventional Open-Pit
- Equipment - Trucks and Shovels
- Problems:
  - Weather: extreme cold in winter; wet conditions during spring thaw
  - Shortage of skilled labor

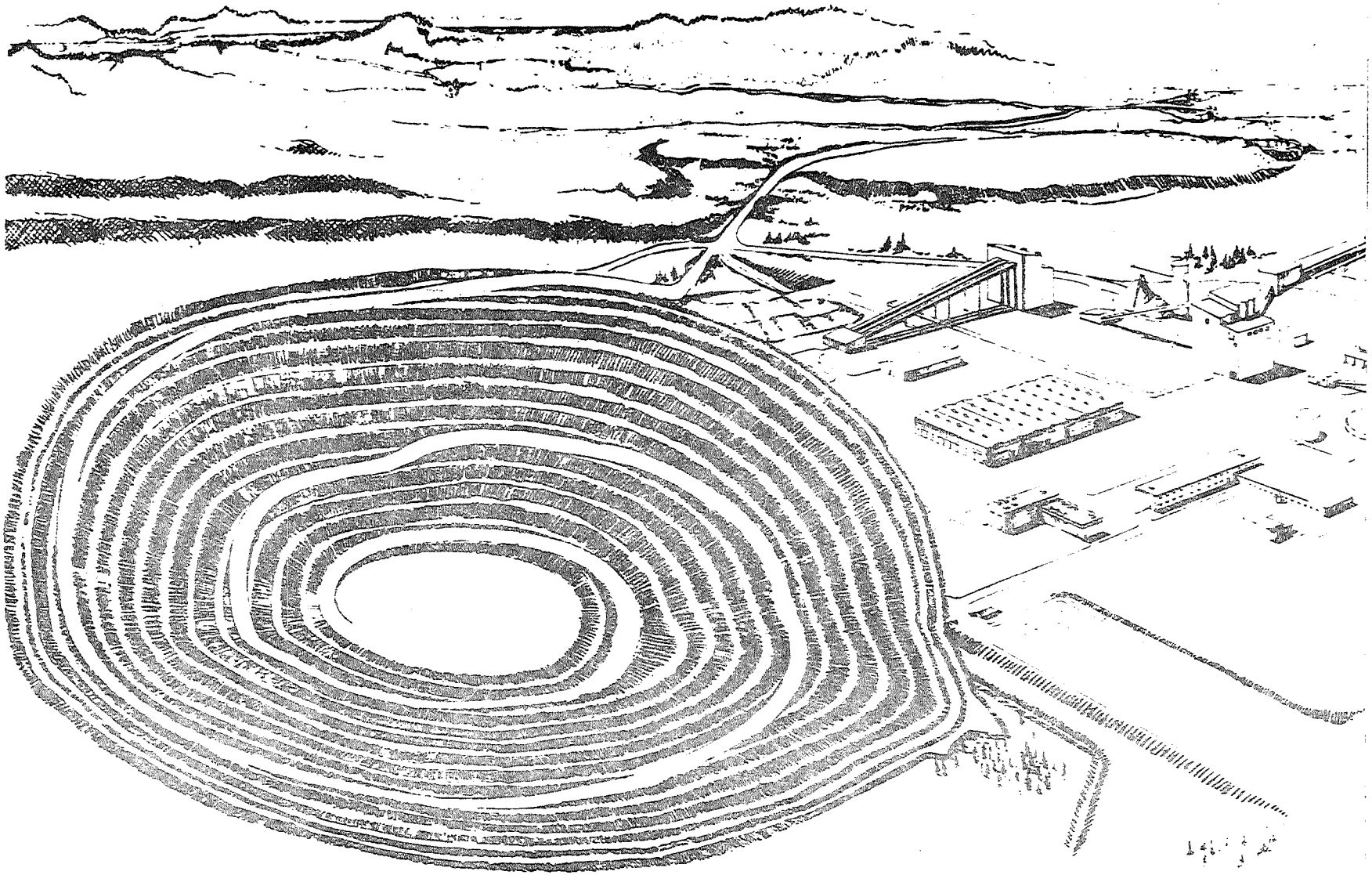


Figure 2-20 An Artist's Sketch Of The Svappavaara (Leveaniemi) Mine In The Year 2000

planning and research done by LKAB to more precisely pinpoint when and how an underground operation should commence. For example, Leveaniemi management expressed one concern about development of the area directly below the present pit and the relationship of this mining plan to the presence of a second orebody nearby. Possible schemes include one to drive a slope between the two orebodies and connect the slope with horizontal haulage drifts. The slope would be equipped with a hoist for ore movement. However, they are very worried about hoisting of ore on an incline because of a lack of experience using this technique. Rates of wear on rails and hoist rope are the principal unknowns.

A second proposal being actively considered for underground development is to drive an incline near the present pit and use it for haulage of open-pit ore now and later for haulage of underground ore. This proposal would only serve the main orebody, however.

In all of their plans it is obvious that LKAB plans a heavy use of diesel equipment for the movement of ore in some manner.

The advantages of the underground road include reduced stripping requirements for the present haul road, a lower maintenance effort on the haul road, and increased slope angles in the open-pit. Men and supplies alike will have access to the underground operations by way of an incline which will be large enough to provide passage for large LHDs.

In summary, Leveaniemi management plans to eventually conclude its open-pit operation. An interim step prior to full underground mining, will be the installation of a below pit continuous haulage system to prolong the pit life and facilitate underground development. This move to continuous underground haulage will gain the following advantages:

- Lower stripping requirements.
- Achieve a greater slope angle on the pit walls.
- Lower maintenance on haul roads.
- Better extraction ratio for the ore.

- The new slope can also be used for future underground mining.
- Lower ore haulage costs are possible with conveyors or skips.

And finally, the haulage methods undergoing comparative cost study include:

- Truck haulage - conventional, open air (Plan PO)
- Truck haulage - road underground, diesel and electrically driven (Plan NA)
- Conveyor Belt - portable open-pit crusher (Plan PO_3)
  - permanent underground crusher
- Skip - vertical
  - inclined

In Table 2-14 there is a comparison of several of the above methods showing the final pit slope angles and the available one that can be recovered. The economics were not included, since the study had not been completed at the time of this report.

TABLE 2-14

PIT SLOPES AND ORE RECOVERY FOR THE LEVEANIEMI PIT FOR THREE HAULAGE PLANS

	PLANS		
	TRUCK HAULAGE (Conventional, Open air)	CONVEYOR BELT (Portable open pit crusher)	TRUCK HAULAGE (Road underground, diesel & electrically driven)
Final Pit Slope	42^{o}	42^{o} (excluding crusher)	42^{o} (underground access haulage roads)
Available Ore (Millions of Metric Tons)			
Magnetite	136	131	128
Hematite	8	8	8
Waste	150	139	59

Recent Stripping Ratio: 1:1.2

With Underground Haulage Road: 1:0.6

2.2.10 Open-Pit Copper Mine - Arizona

One of the fastest growing open-pit copper operations in the U. S. has selected continuous conveyor haulage as the most viable method for hauling both waste rock and ore to the surface. Key factors contributing to this decision included:

- Large volume operation (has averaged almost 100 million tpy of waste and ore over the past 10 years)
- Deep pit (over 700 feet down to top of ore deposit)
- Extensive long-range stripping program (waste-to-ore ratio exceeds 10:1)
- Economics

The evaluation of in-pit crushing and conveying versus truck haulage was the key issue in mine management's planning phase for the open-pit operation. Their assessment concluded the following *(Ref. 2.2.5)*: © *AIME, 1976*

> Conveyors, if used in the proper application, can come out on top of pure truck haulage. Deep pits, with high average lifts can, in many cases, use high capacity conveyors advantageously. Our operation has compiled empirical data on both conveyor and truck haulage that can be extremely useful in the design of material handling systems for open-pits.

Needless to say, conveyor haulage has worked well for this open-pit in its 13 years of operation. This section will provide some insight into the equipment performance analysis made by management during the planning phase and will conclude with some actual data on how well conveyor haulage has fared compared to trucks. Table 2-15 provides basic information on the current open-pit operation.

TABLE 2-15

FACT SHEET ON THE OPEN-PIT COPPER MINE

- Location - 30 miles southwest of Tucson, Arizona
- Ore - Cu, MoS_2, Ag
- Mineralization - Chalcopyrite, calcocite, covellite, bormite
- Open-Pit Mining Operation Started - 1965
- Annual Production Rate (1976) - 7.5 million tons ore
  76 million tons waste
- Potential Mine Life at Present Production Rate - 25 years
- Operations:
  - Truck and shovel feed in-pit crushers
  - Inclined conveyors haul material to surface
- Equipment:
  - Electric shovels - 15-cu-yd buckets
  - 75- and 100-ton dump trucks
  - 54 x 80-inch gyratory crushers
  - 60-inch steel belts for main conveyor haulage
- Problems:
  - Conveyor capacity overdesigned
  - 100-ton dump trucks too large for crusher transfer chute design.

3. EVALUATION OF THE ELEMENTS OF CONTINUOUS UNDERGROUND HAULAGE SYSTEMS

Section 2 of this report presented the state-of-the-art on continuous haulage for the ten open-pit mines identified as using, have used, or planning to use continuous haulage techniques. All of the mines were visited and detailed reports are presented in the appendix. These reports demonstrate that no existing or proposed operation can restrict its haulage investigations to any single system. Each mine visited had identified several haulage methods, both surface and/or underground, applicable to their orebody configuration. And, detailed "economic evaluations" were undertaken to select the optimum haulage system.

This section elaborates on the elements that were considered in the selection of the various underground haulage systems evaluated by the mine operators and planning engineers. Thus, emphasis is placed on continuous underground haulage methods for open-pit mines and leaves the discussion of the well-established truck haulage and surface conveyor systems to others.

It is important to note that none of the mine operators, nor the literature reviewed contributed sufficient capital or operating cost data to allow an "economic evaluation" of comparative systems either between operations or as utilized by any of the operators in selecting the haulage system adopted. The mine operators generally regarded economic studies which utilize capital and operating cost data reflective of the companies performance and competitive posture as proprietary. Therefore, only descriptions of the underground haulage systems and discussions of the elements involved in preparing an "economic evaluation" are addressed in this section.

The elements identified as being necessary for mine management to compare the various underground haulage systems (as to their advantages and disadvantages) rationale when they should be installed, versatility, economics, etc., are listed below and

detailed in this report section:

- Geometric relationships
- Crushing
- Conveying
- Ore passes
- Hoists
- Others

3.1 Geometric Relationships

An ore deposit's geometric relationships inevitably determine the kind of ore haulage system that can be practically and economically implemented. This is especially true when the decision has already been made to transport ore underground from the open-pit and a variety of haulage methods--rail, conveyor, skip--are possible. The key parameters which make up the so-called "geometric variables" are:

- Nature of the orebody
- Local topography
- Regional climate
- Proposed mine plant layout

The following sections review each variable in detail and conclude with examples of open-pit mine layouts which, because of the geometric relationships, are ideal candidates for underground and continuous haulage transport systems.

3.1.1 Nature of the Orebody

Orebody analysis is a complex process that must consider many items related to the geology and the configuration of the deposit. The first step is to establish the general geology through an exploration phase which will determine the following:

- Geology--local and regional geology.
- Mineralogy--type of minerals, their relationship to the orebody and the host rock.
- Reserves--the quantity of minerals within the orebody.
- Grade--the quality of the minerals.

With the above information the ore deposit configuration can be mapped out. Items of particular interest include:

- Deposit Location--near the surface, deep underground, etc.
- Deposit Shape--massive, vein, tabular, disseminated, etc.
- Variations in ore grade and mineralization throughout the deposit.

These details serve to determine the mining methods that are most applicable which in turn dictate the ore haulage techniques that warrant evaluation.

3.1.2 Topography

Haulage system design must take into account the relationship of the local topography to the ore deposit. For example, if the deposit is located in a mountainous region, continuous haulage from underground may be a practical approach. The economic gain from minimizing elevation changes in the transport path and shortening the haul distance can be quite significant. All things considered, it is probably much cheaper to bring the ore out through the mountain than to truck it up and over.

In areas where the topography is flatter, the gains from underground haulage beneath the pit are harder to define. In many cases underground transport of open-pit ore is not a viable method at all.

3.1.3 Climate

The climate of the region can affect the type of haulage system selected, but to a lesser extent than the topography and the orebody configuration. For example, underground haulage makes sense in locales where cold temperatures persist throughout much of the year. In regions where moisture and precipitation are significant problems, underground

transport again offers a viable alternative to the unavoidable maintenance problems with surface haul roads. This latter point is perhaps the most important to mine management when trying to determine what possible adverse impacts the weather may have on the mining.

3.1.4 Mine Plant Layout

The last major consideration in the choice of a haulage system is its relationship to the ore processing plant and shipping terminal. If the mine is a new property, then the plant location can be modified to fit the haulage requirements. But if the haulage system has to be modified after the establishment of the mine plant, very few changes become possible for economic reasons. In short, once the processing plant and tailings ponds have been built, it is generally too expensive to move them in order to establish a new haulage system or change the pit configuration.

3.1.5 Open-Pit Mine Candidates for Underground Haulage

Underground haulage from open-pit mines is practiced world wide. Operations which utilize underground transport generally do so for reasons of orebody configuration. The following classification covers most cases:

- Narrow laterally constricted orebody.
- Orebody with a large depth dimension.
- Orebody with elevation disadvantage relative to the processing plant.
- Orebody separated from the processing plant by a significant horizontal distance.

Some mines make use of underground haulage for other reasons such as:

- Environmental considerations.
- No requirement for primary crushing.

The following sections give specific examples of the above cases.

3.1.5.1 Deep Laterally Constricted Orebody--(Figure 3-21)

This type of orebody is generally small and deep. The stripping ratio can be quite high. If the pit is taken to great depths, the haul road must have several spirals for the trucks to exit the pit. Open-pits in this classification include:

- Tynagh, Ireland
- Pyhasalmi, Finland
- Rio Tinto, Spain
- Questa, New Mexico

3.1.5.2 Orebody with Extensive Depth--(Figure 3-21)

Underground haulage can be a viable method with orebodies that are so deep that they must be mined both from open-pit and underground. Mines in this classification include:

- Tynagh, Ireland
- Pyhasalmi, Finland
- Rio Tinto, Spain
- Climax, Colorado
- Consolidated Sullivan, B.C., Canada
- Kiruna, Sweden
- Svappavaara, Sweden
- Butte, Montana

- Questa, New Mexico
- United Verde, Arizona

3.1.5.3 Orebody with Elevation Disadvantage Relative to The Processing Plant--(Figures 3-22 and 3-23)

This kind of orebody generally has the most complex geometric relationship which makes the selection of a mining and haulage system a difficult process. The ideal configuration is where the orebody is at an elevated level on a mountain and the processing plant or shipping point is at a much lower location. (See Figure 3-22.) The following mining operations are good examples:

- Carol Project, Labrador, Canada
- Dworshak Dam site, Idaho
- Eagle Mountain, California
- Questa, New Mexico
- Sierrita, Arizona
- Kendall Quarry, Washington

Another possible configuration is where the surface haulage must go up out of the pit and over another elevation rise to the processing point. (See Figure 3-23.) The following mines are characteristic of this situation.

- Cananea, Mexico
- Carlin, Nevada
- Spruce, Minnesota
- Susquehanna, Minnesota
- Algoma, Ontario, Canada
- Caland, Ontario, Canada

3.1.5.4 Orebody a Long Distance from Processing Plant--(Figure 3-24)

One advantage of any continuous haulage system is that it generally becomes more economical the longer the haul distance to the processing plant. Mines that fall in this category include the following:

- Rio Tinto, Spain
- Carlin, Nevada
- Eagle Mountain, California
- Carol, Labrador, Canada
- Nordcement Hannover, Germany
- Twin Buttes, Arizona
- Kendall Quarry, Washington

3.1.5.5 Environmental Considerations--(Figure 3-22)

In some cases where the open-pit is located near an urban area, it is desirable to locate the processing plant underground in order to minimize the ecological and environmental impact. An example is the proposed quarry site near Golden, Colorado where concern for the possible noise pollution and general unsightliness has influenced planners to consider underground haulage.

3.1.5.6 Orebody with Long Distance Rail Haulage and No Crushing---(Figure 3-25)

The use of primary crushers in any continuous haulage system is essential where belt conveyors or ore skips are used to bring the rock to the surface. In a few cases, however, crushing has not been necessary and a single ore pass to an underground rail haulage system has proven to be the most

economical transport means. Mines that fall in this category include the following:

- Carol Project, Labrador, Canada
- Bingham Canyon, Utah

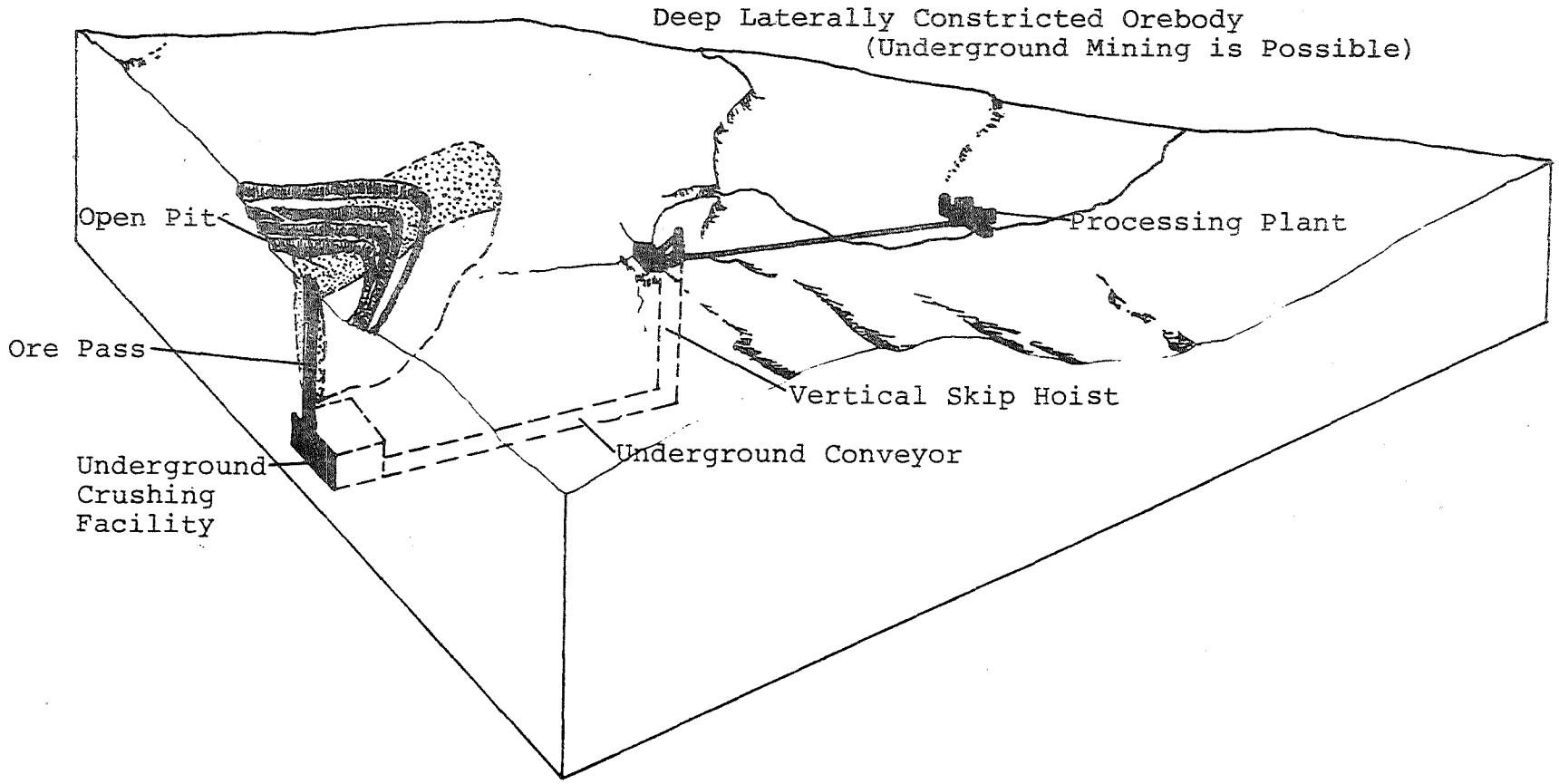


Figure 3-21 Geometric Relationship Example: Open-Pit Mine For Deep Laterally Constricted Orebody

Haulage System Network: Ore Pass, Underground Crusher, Conveyor, Vertical Skip Hoist, Overland Conveyor

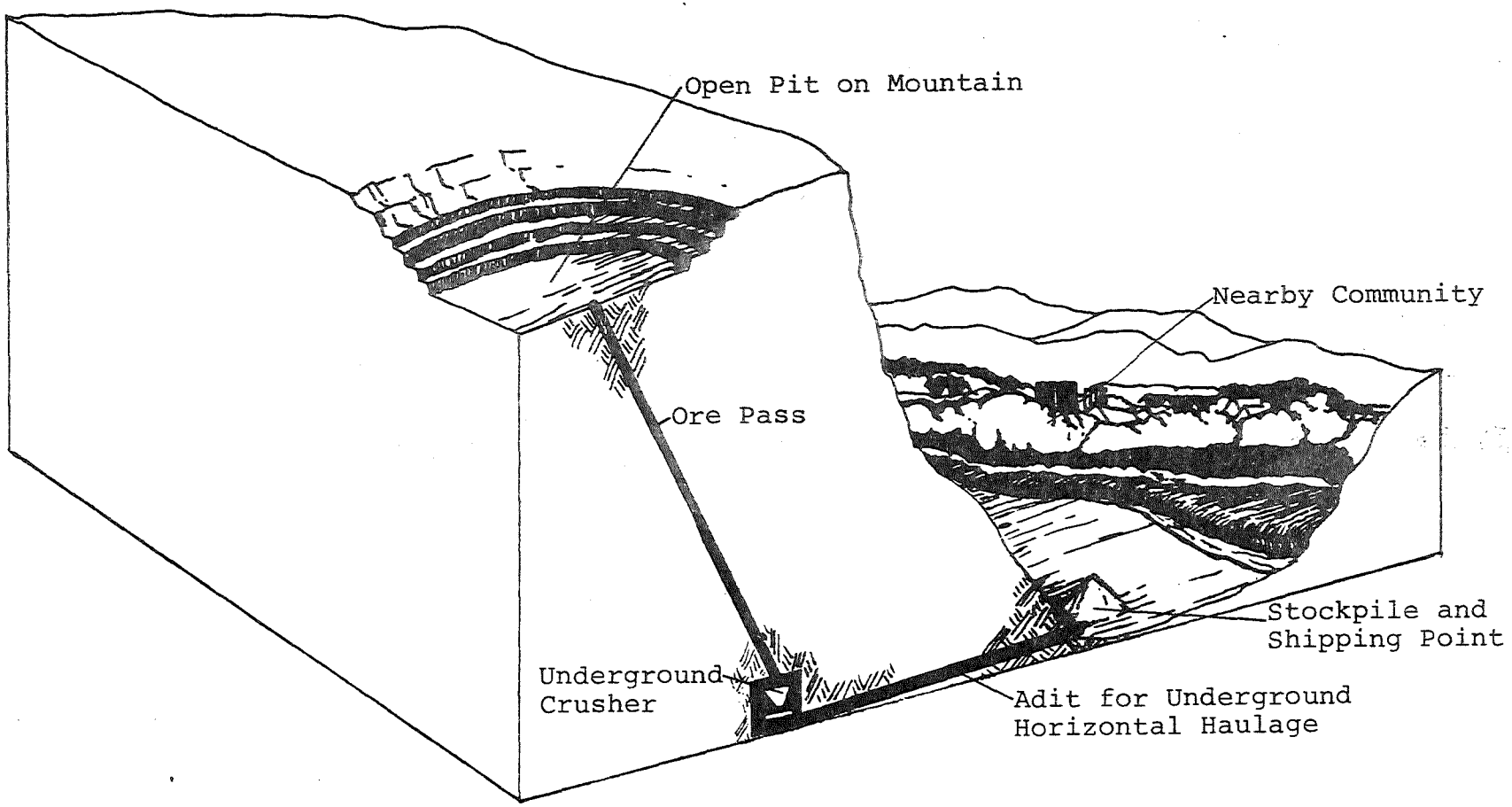


Figure 3-22 Geometric Relationship Example: Open-Pit Mine At An Elevated Level On A Mountain With Ore Shipping Point At Lower Elevation. Additional Ecological Factor Proximity To Nearby Community.

Haulage System Network: Ore Pass, Underground Crusher, Horizontal Haulage, Stockpile and Shipping Point

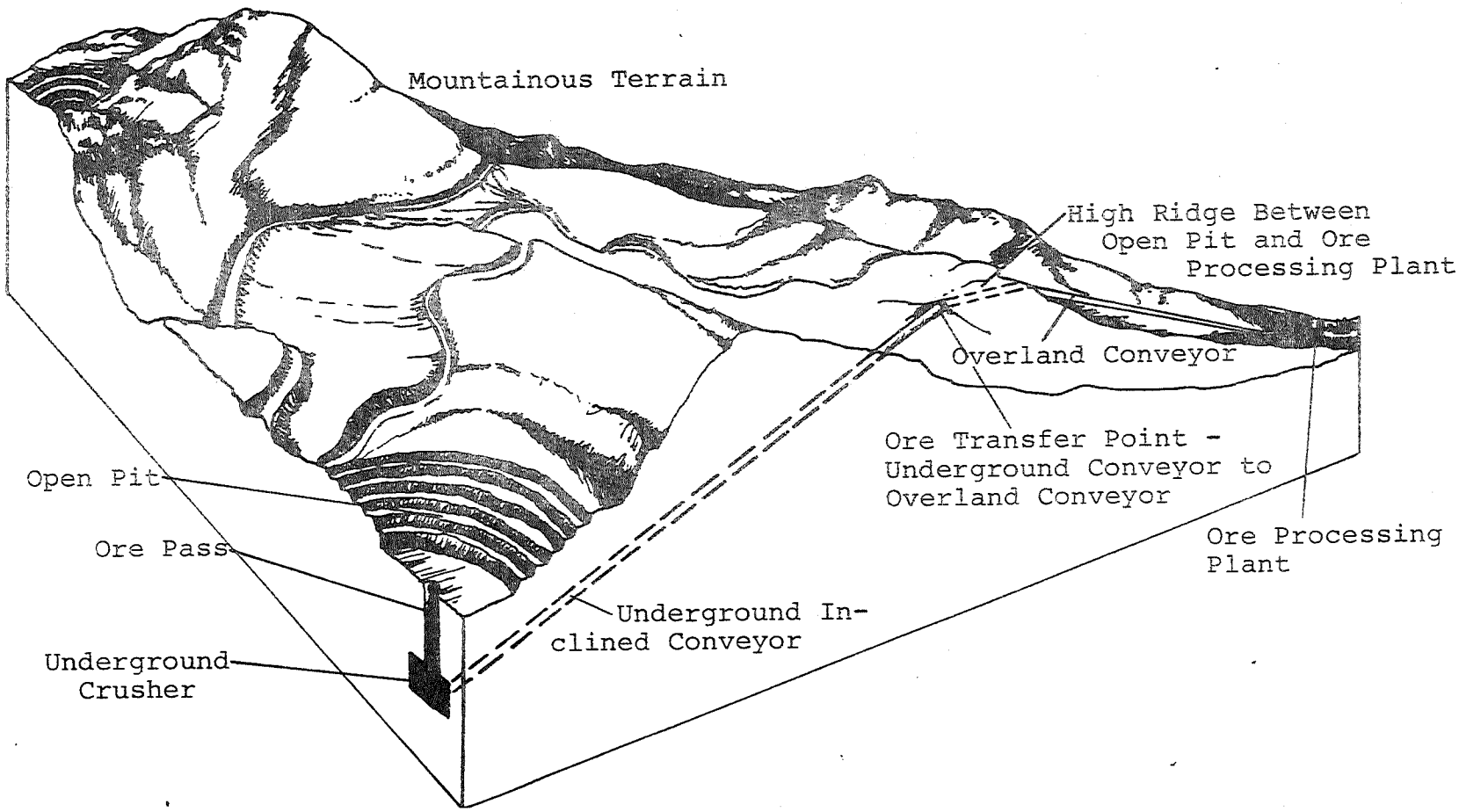


Figure 3-23 Geometric Relationship Example: Open-Pit With Elevation Disadvantage For Direct Transport of Ore To The Processing Plant.

Haulage System Network: Ore Pass, Underground Crusher, Underground Inclined Conveyor, Overland Conveyor, Ore Processing Plant

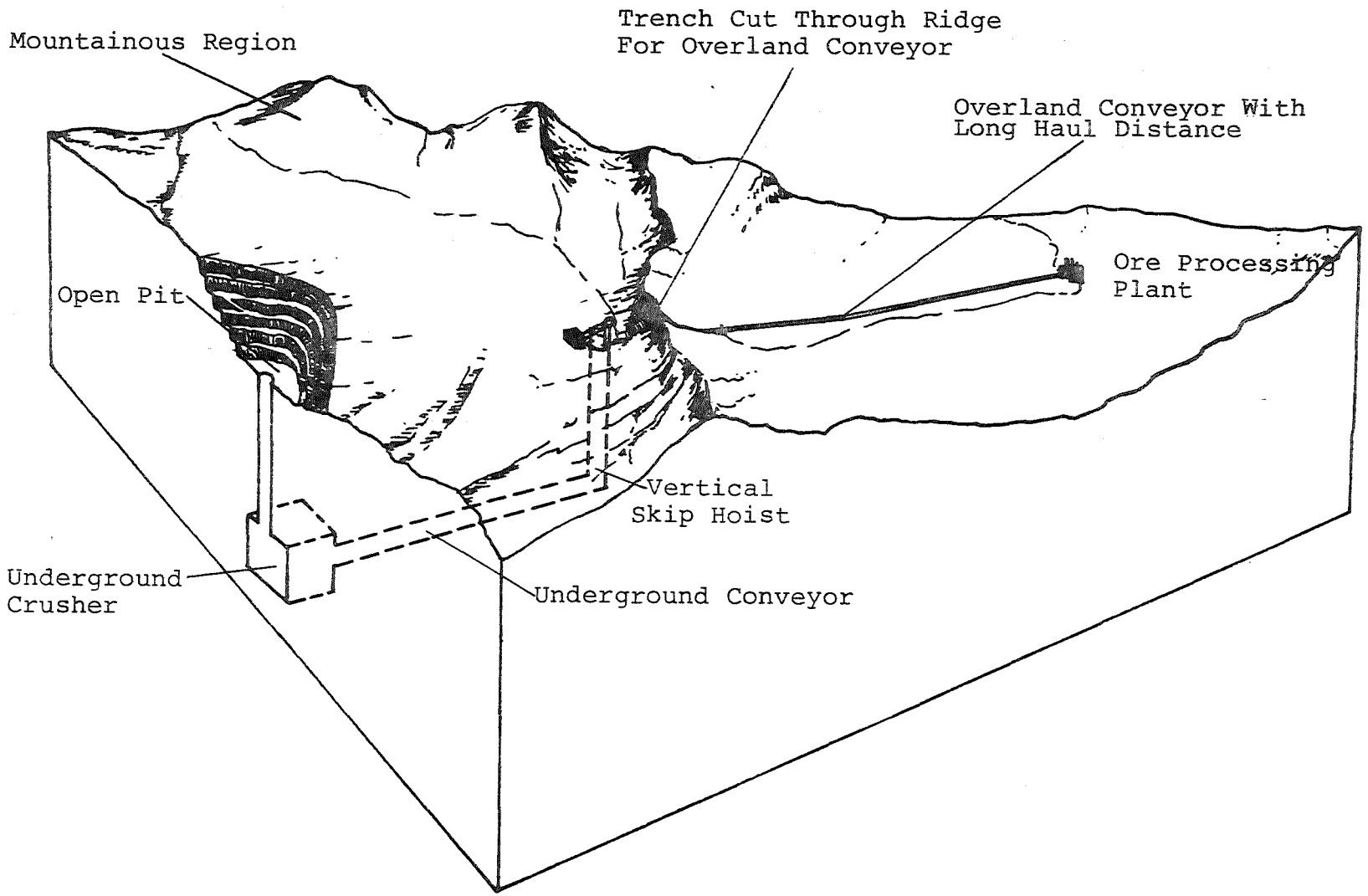


Figure 3-24 Geometric Relationship Example: Open-Pit with Elevation Disadvantage For Direct Ore Transport and Long Haul Distance to Processing Plant

Haulage System Network: Ore Pass, Underground Crusher, Underground Conveyor, Vertical Skip Hoist, Overland Conveyor

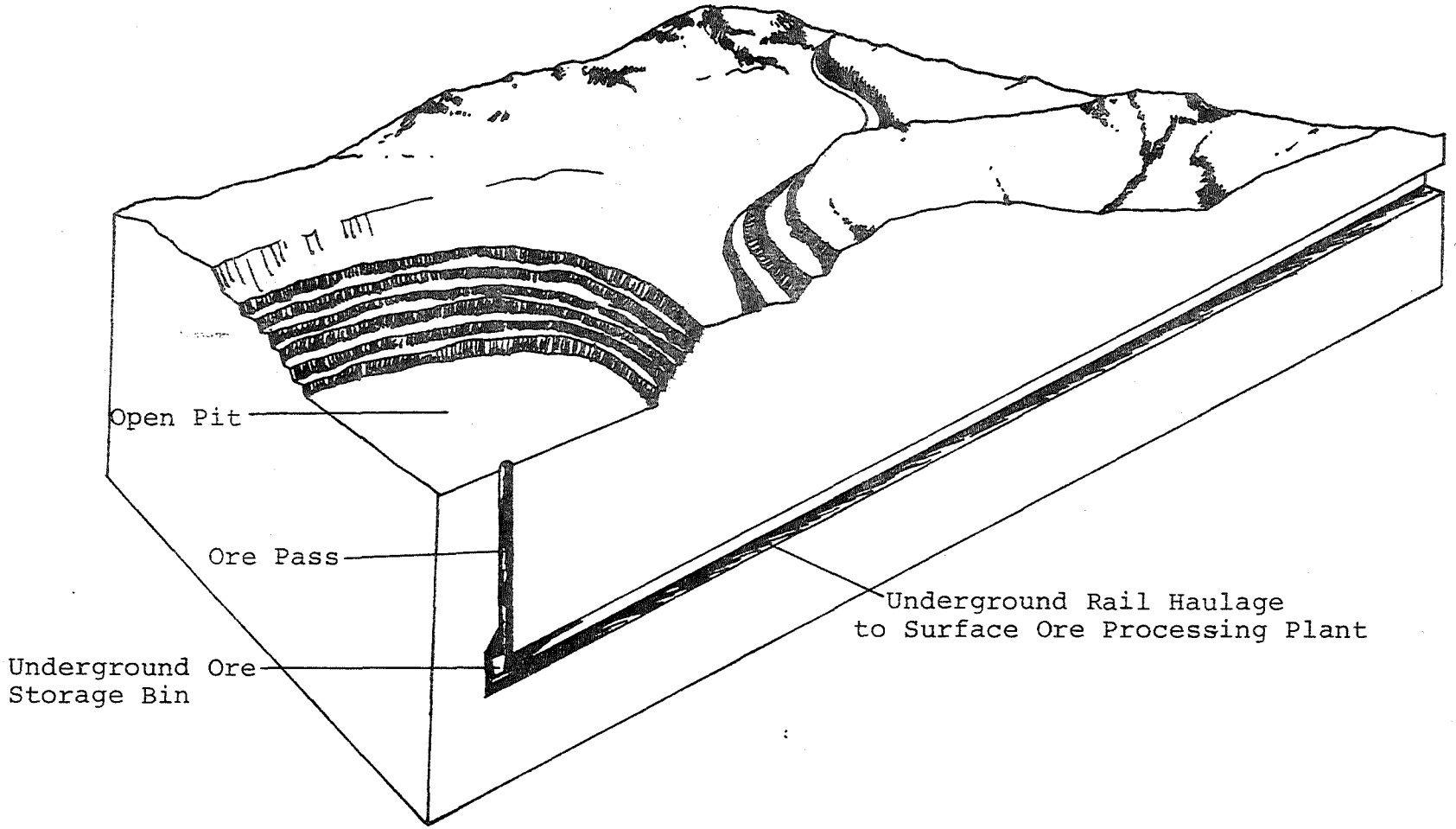


Figure 3-25 Geometric Relationship Example: Open-Pit Without Requirement For Primary Ore Crushing and With Long Haul Distance To Processing Plant.

Haulage System Network: Ore Pass, Underground Storage Bin, Underground/ Surface Rail Haulage

3.2 Crushing

This section briefly outlines the principal factors considered by mine planning engineers, operations supervisors, and management personnel in determining the type of plant to be installed, the location of the facility, and the specific equipment to be used.

3.2.1 Types of Plants

For sake of definition, there are four basic plant types:

- Permanent - A plant with any combination of primary, secondary, tertiary crushing and/or grinding capabilities which are integral part of the main ore processing facility.
- Semi-Permanent - A plant capable of providing either primary or both primary and secondary crushing on site at an open-pit. The plant can be either an underground or surface installation. (see Figures 3-26 and 3-27, respectively.)
- Mobile - A modular unit plant which can move about in the pit at a relatively slow rate of advance either on rubber tires, a crawler track assembly or a hydro-mechanical walking mechanism. (See Figure 3-28.)
- Portable - A modular unit plant of relatively low capacity, mounted on trailers for over-the-road transport, and normally associated with aggregate production. (See Figure 3-29.)

For the purposes of this discussion, further detail on semi-permanent plants and mobile crusher units only is presented in Sections 3.4 and 3.5, respectively.

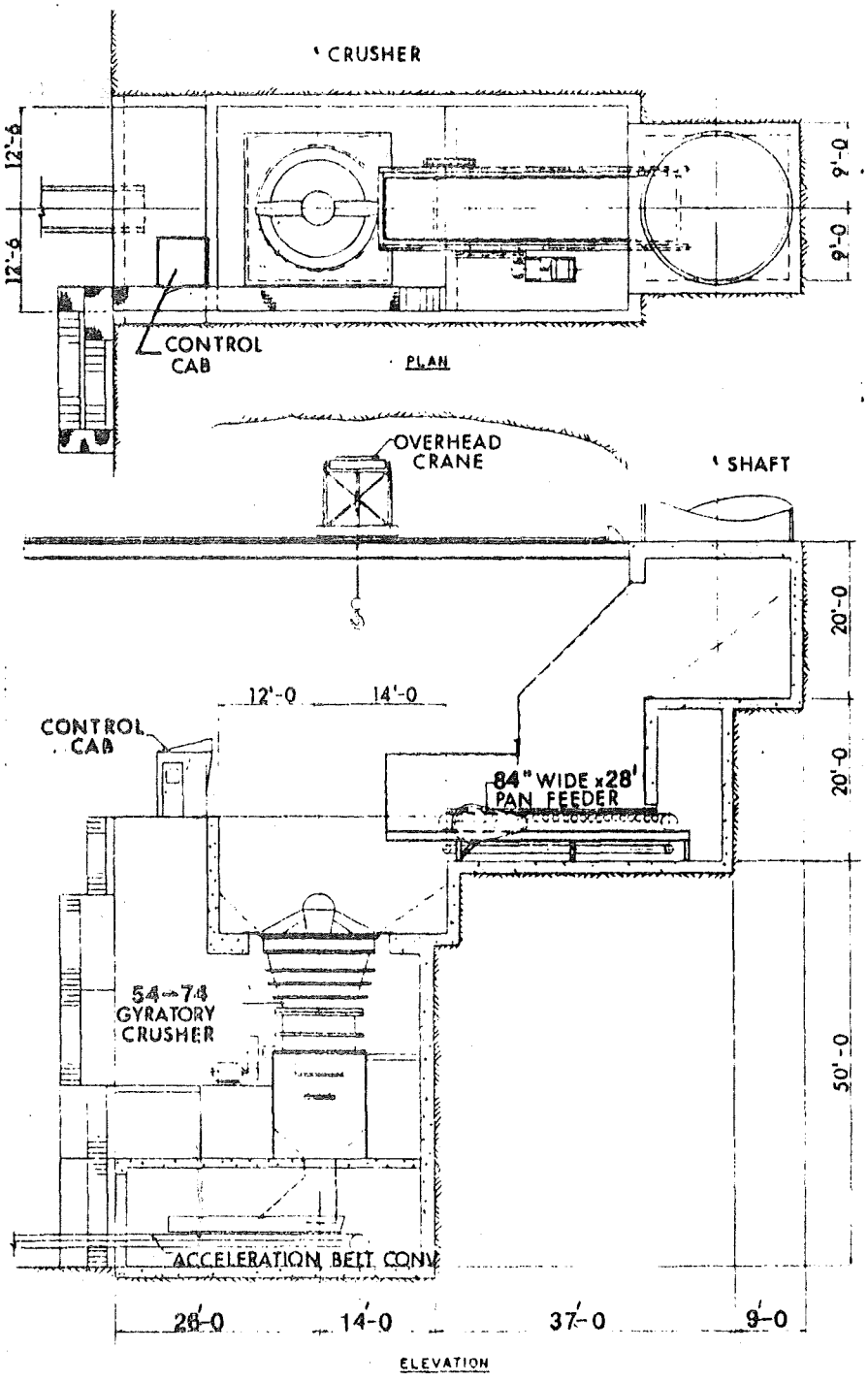


Figure 3-26 Illustration of Semi-Permanent Underground Crusher Facilities (Ref. 3.2.1)

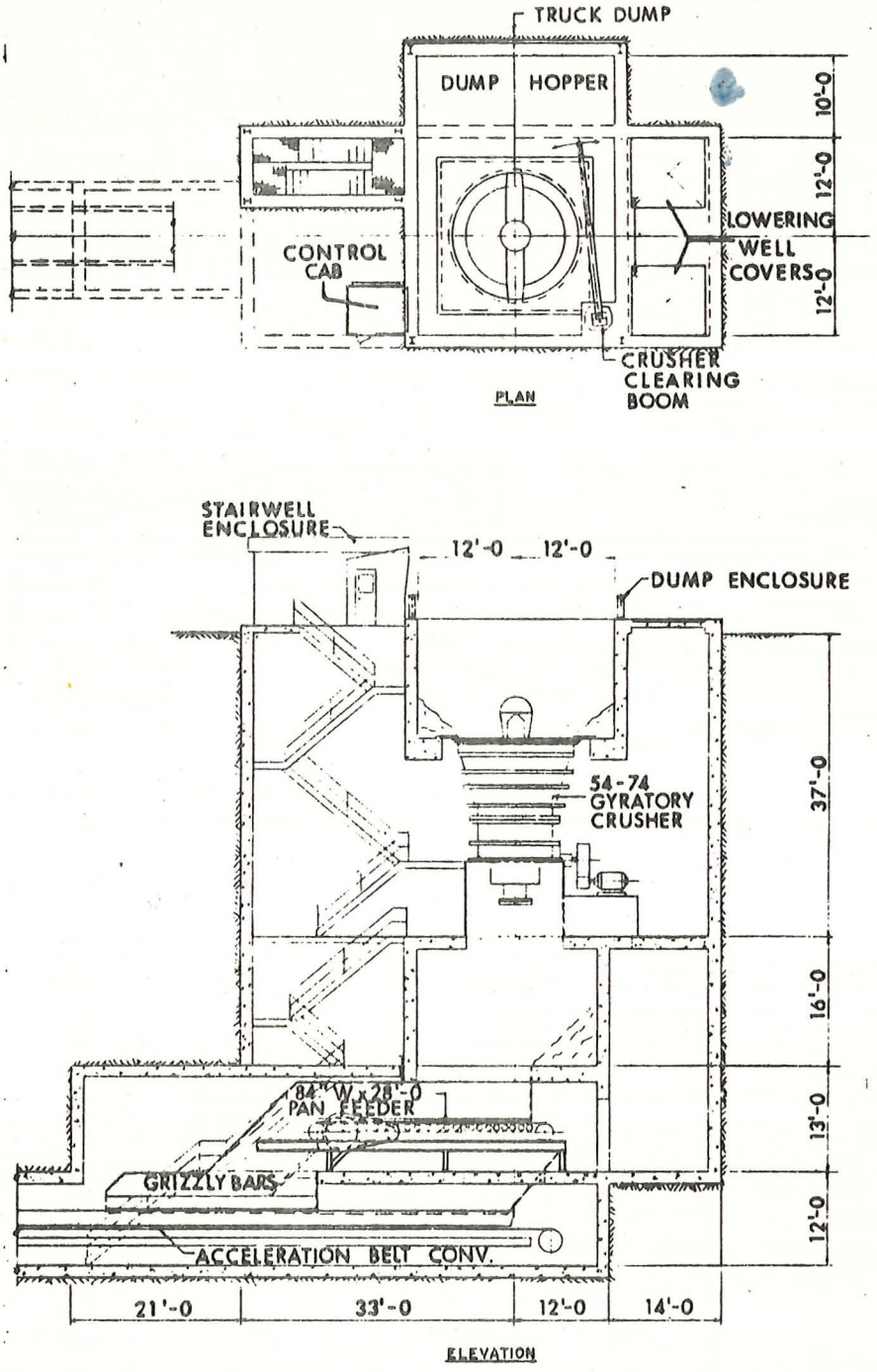


Figure 3-27 Illustration of Semi-Permanent Surface Crusher Facilities (Ref. 3.2.1)



Figure 3-28 A Mobile Crusher Unit of 1,000 tph Capacity

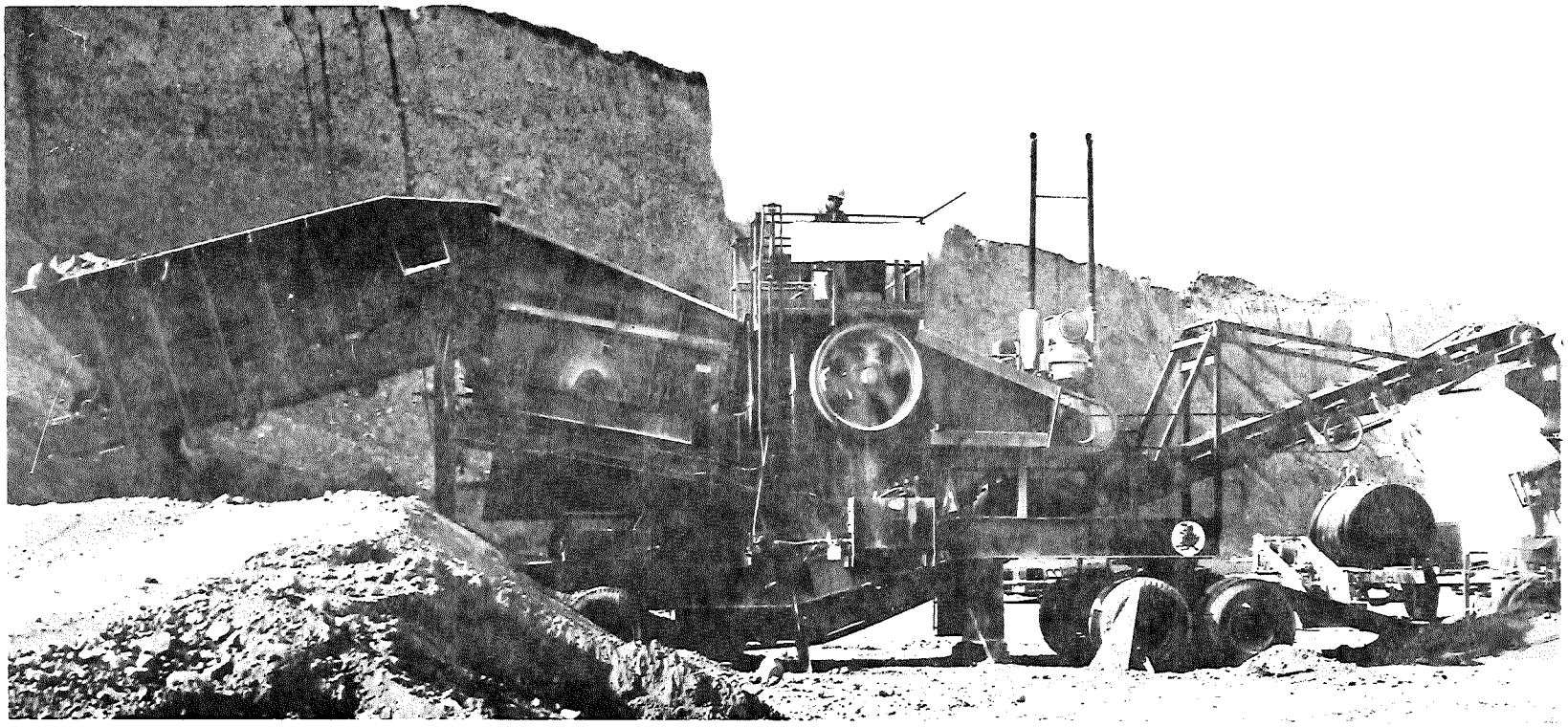


Figure 3-29 A Portable Crusher Unit of Low Capacity (10 to 25 tph) Typically Found In A Quarry Operation

- Waste
- Overburden

- Area Hydrology

With the above information in hand, the planning team is faced with several options concerning the crusher facility:

- Install a unit, either on the surface or underground, during stripping operations. Use it for ore and/or waste processing once the pit becomes operational. *(Ref. 3.2.4) ©EM/J 1967*
- Install adequate crushing capacity at a central location for the pit's total estimated production requirements. Use one crusher to process ore and a separate unit for waste. They can be either surface or underground facilities. *(Ref. 3.2.5) © Mining Congress Journal, 1978*
- Install multiple modular crushing facilities, either surface or underground, that will have adequate total capacity to process ore and/or waste material as the pit develops. These units can be located at varying depths and close to ore and waste centroids.
- Install several mobile crushing units with adequate capacity to meet total pit production. Each unit would be coupled with a shovel and be capable of processing the full production capacity of the excavation equipment. Separate systems may be required for ore and waste preparation. The option exists to process one or the other at a semi-permanent installation. *(Ref. 3.2.6) © AIME, 1960 (Ref. 3.2.7) © MacLean Hunter Publishing Co., 1968 (Ref. 3.2.8) © AIME, 1968 (Ref. 3.2.9) USBM, 1976*

In addition, possible combinations of the above and any other candidate crushing systems identified by the planning team will be considered.

Other important factors which affect crusher selection include the choice of haulage system(s) between the open-pit and processing plant. The key decision pertains to surface transport versus underground haulage, the latter case providing the option to locate the primary crushing facility underground. Section 5 treats this issue in depth.

3.2.4 Semi-Permanent Crushing Plants

The principal difference between permanent and semi-permanent plants is that the latter is restricted to utilization at a single location by one factor--economics. This plant can be moved. A permanent facility, by comparison, is restricted forever to a single location due to its integral relationship with the main ore processing facility. The next two sections discuss the pros and cons of locating a semi-permanent crusher either on the surface, in the pit, or underground.

3.2.4.1 Surface Location

Appendix C presents typical arrangements for semi-permanent surface plants which utilize either jaw or gyratory crushers ranging in capacity from 1,000 to 10,000 tph. The primary difference between the units of large and smaller capacity, other than the equipment size, is the volume of the containment facility.

Among the advantages of a surface facility are:

- Easy accessibility for repair and maintenance.
- Flexibility as regards relocation in the pit.
- Less expensive installation cost.

The major disadvantages include:

- The requirement for larger equipment to accommodate surges in feed.
- The inherent loss in efficiency due to surge feeding.
- Possible air and noise pollution.

3.2.4.2 Underground Location

Appendix C also shows typical underground installations with capacity identical to the surface installations previously presented.

Among the advantages are:

- Increased crusher efficiency through the elimination of surge feeding.
- Utilization of raises and/or shafts for material storage.
- A totally enclosed and environmentally protected installation.

The disadvantages of this type of facility include:

- Possible bridging or plugging of ore passes and shafts.
- Increased capital cost for the development of service tunnels, crushing chamber and raises.
- Reduced accessibility.
- Extra dust control and ventilation.

3.2.5 Mobile Crushing Plants

The introduction of mobile crushing equipment to the hardrock mining industry could have significant positive impact on open-pit operations in the U. S. Well accepted in both European mining activities and the U. S. aggregate industry, there has been little interest shown in this type of equipment by large open-pit operators. This reflects the current mismatch in size between the 500 to 1,500 tph mobile units that are available and the 4,000 tph capacities desired by U. S. operators.

Mobile units are available utilizing jaw crushers, jaw gyratory units, impact crushers, single and double rolls, single and double hammer installations. (see Figure 3-31) These plants can be mounted either on rubber tires, crawler track assemblies, or a hydromechanical walking mechanism, depending on the equipment weight and pit floor characteristics. The

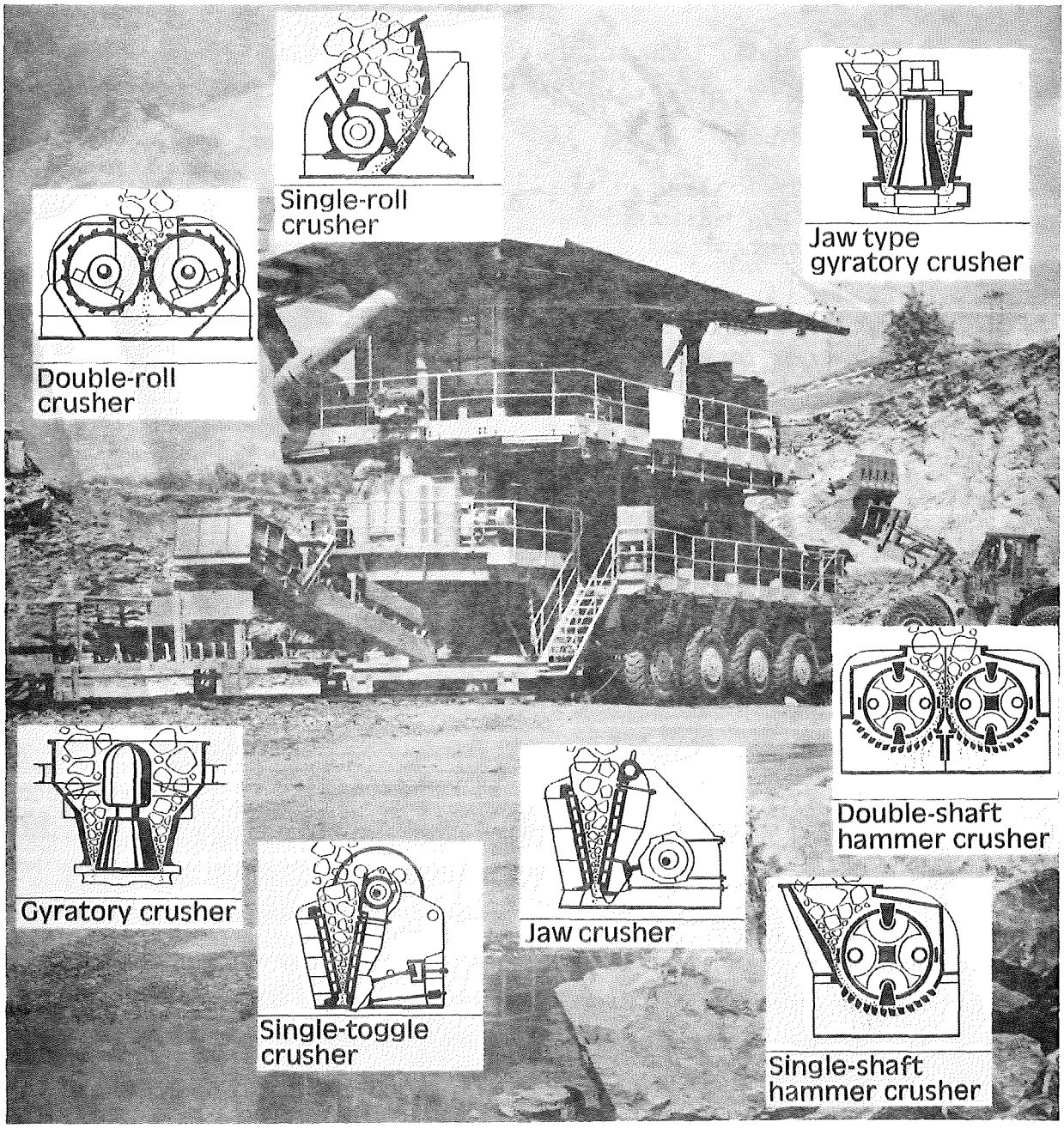


Figure 3-31 *Mobile Crusher Unit With Illustration Of The Different Types Of Crushers Available For Implementation*

maximum in-pit speeds are approximately 8 and 15 fpm for the "walking" and track-mounted units, respectively.

The advantages of mobile crushing units compared to semi-permanent facilities include:

- High tonnage output per man-shift due to the increased output of the loading shovel.
- Reduced labor, maintenance, and power requirements for ore transport by using conveyor haulage.
- Greater flexibility in planning overall pit development.
- Blending of different ore types and grades is more readily accomplished.

The basic disadvantage is that additional front-end costs are incurred due to the investment required to purchase or construct a mobile arrangement for a crusher. In addition, operating costs for mobile crushers will be somewhat higher than for permanently installed conventional units. Even so, the combination of mobile crushers and conveyor haulage can be shown to offer appreciable cost savings over truck and shovel operation.

3.2.6 Procedure for Selection of Crusher Type

The selection of a crusher type is dependent upon mine planning variables peculiar to each site and upon the specific material being mined. These variables include:

- Material characteristics
  - Type of ore
  - Bulk density
  - Crushability (Work Index)
  - Moisture content
  - Screen analysis (% of fines)
  - Maximum lump size of feed
- Production required

- Other quarry and haulage equipment specifications (sizes)
- Method of feeding
- Product size desired

Each of the crusher types normally associated with surface mining applications (see Section 3.2.2) are best suited for duty on certain type materials. Jaw and gyratory crushers, for example, work best in the reduction of hard, tough and abrasive ores. Impactors are usually involved in the reduction of medium to soft ores. Roll-type toothed crushers are normally utilized for breaking sticky medium-to-soft ores.

The maximum lump size and throughput capacity for the mined material become the decisive factors in the final crusher selection. One of the analytical tools used in this process is shown in Figure 3-32, which compares crusher capacity to critical input dimension (CID)* of the feed. Another trade-off to review, particularly in the case of an underground operation, is the estimated crusher height versus CID of feed (see Figure 3-33).

3.2.7 Crusher Economics for Semi-Permanent Installations

The important cost factors related to crusher selection, and for that matter any piece of in-pit capital equipment, include:

- Equipment purchase price
- Installation cost
- Direct operating and maintenance costs
  - Labor
  - Energy
  - Water

*"CID" refers to the maximum value of the minimum side dimension of an ore/waste particle to be fed through the crusher.

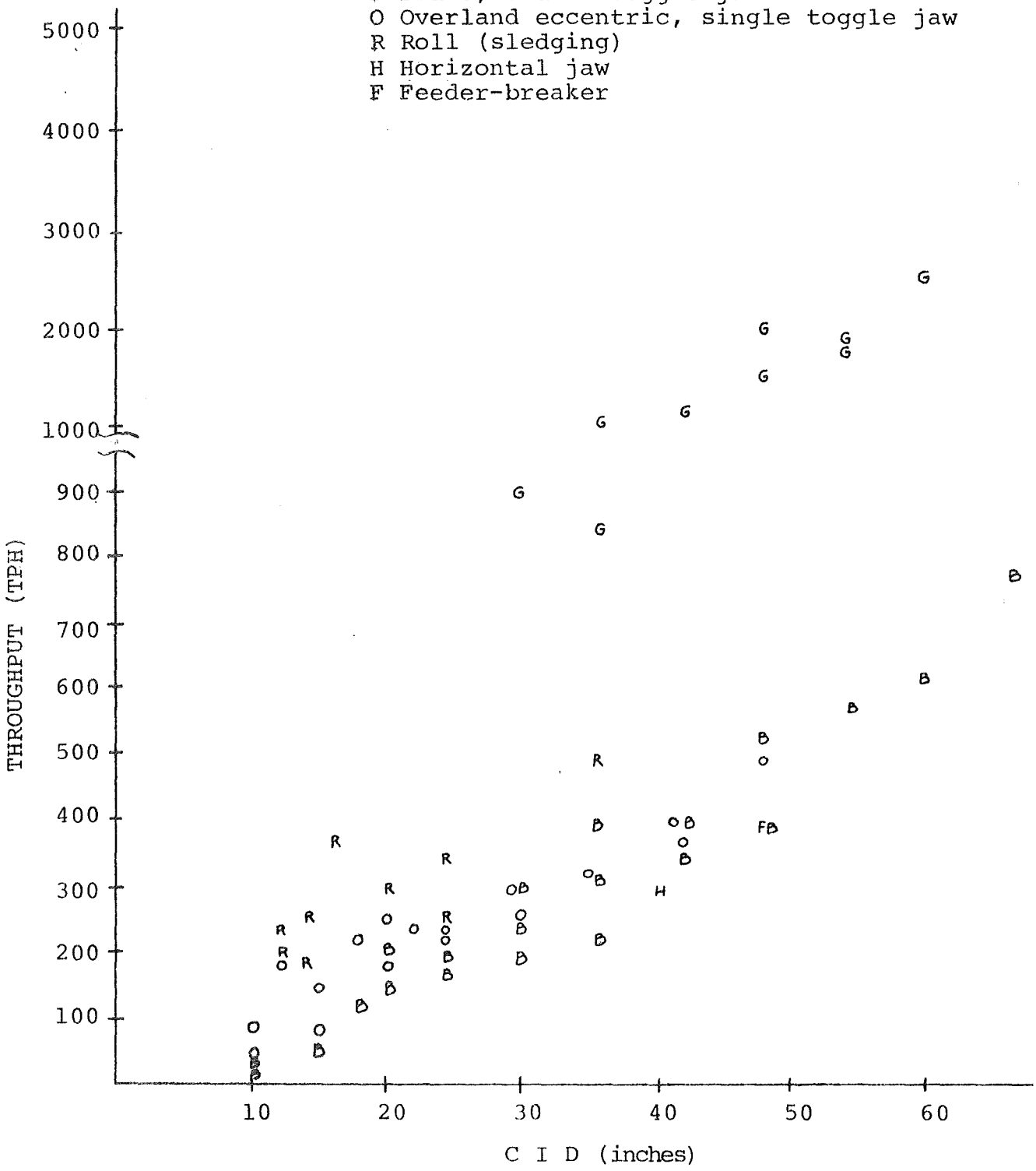
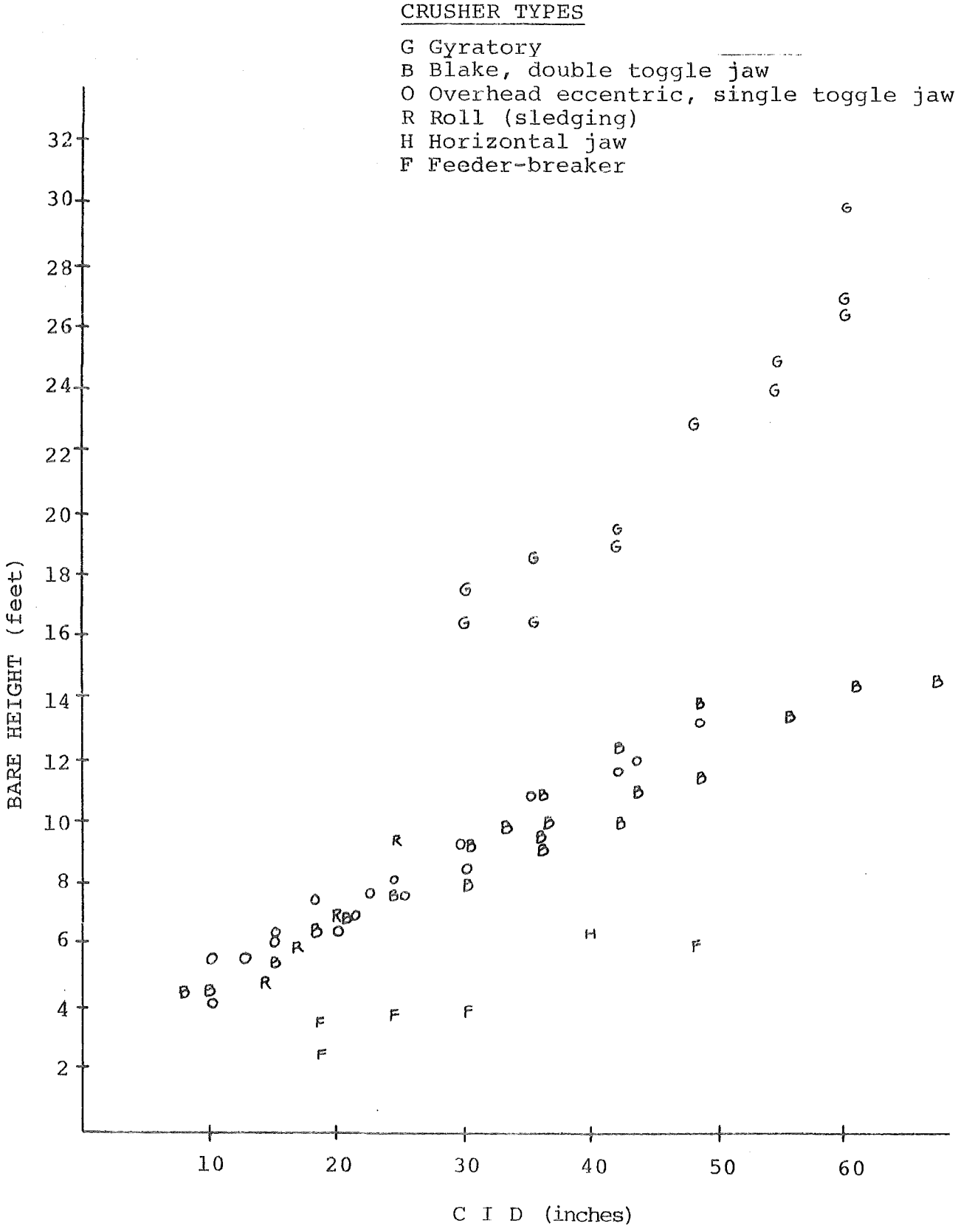


Figure 3-32 Crusher Throughput Versus Critical Input Dimension (Ref. 3.2.10) USBM, 1976



Figures 3-33 Crusher Height Versus Critical Input Dimension (Ref. 3.2.10) USBM, 1976

- Repair parts
- Supplies

- Indirect operating costs
  - Administration
  - Lab analyses
  - Ore stockpiling and storage
  - Taxes, licenses, and insurance
- Non-operating costs
  - Legal
  - Financing
  - Marketing

In trying to optimize the equipment purchase price, a couple of important tradeoffs must be reviewed. The first pertains to crusher cost and the associated throughput capacity as a function of feed CID. Figure 3-34 shows crusher cost versus CID; Figure 3-32 has already illustrated crusher capacity versus CID. (Inflationary effects should also be taken into account by applying an appropriate price adjustment factor, such as the Marshall & Swift cost index.)

The second tradeoff looks at horsepower requirements per ton of ore crushed as a function of ore throughput capacity. The information shown in Figure 3-35 has been derived in Appendix C. The difference in horsepower requirements between surface and underground facilities can be attributed to the additional equipment, necessary lighting and environmental considerations required for the latter.

Once the above tradeoffs have been established, total capital cost as a function of crusher throughput capacity is generated as shown in Figure 3-36. This information was developed through preliminary design estimates on equipment, material and labor costs for a variety of facilities with capacities ranging from 1,000 through 10,000 tph (see Appendix C).

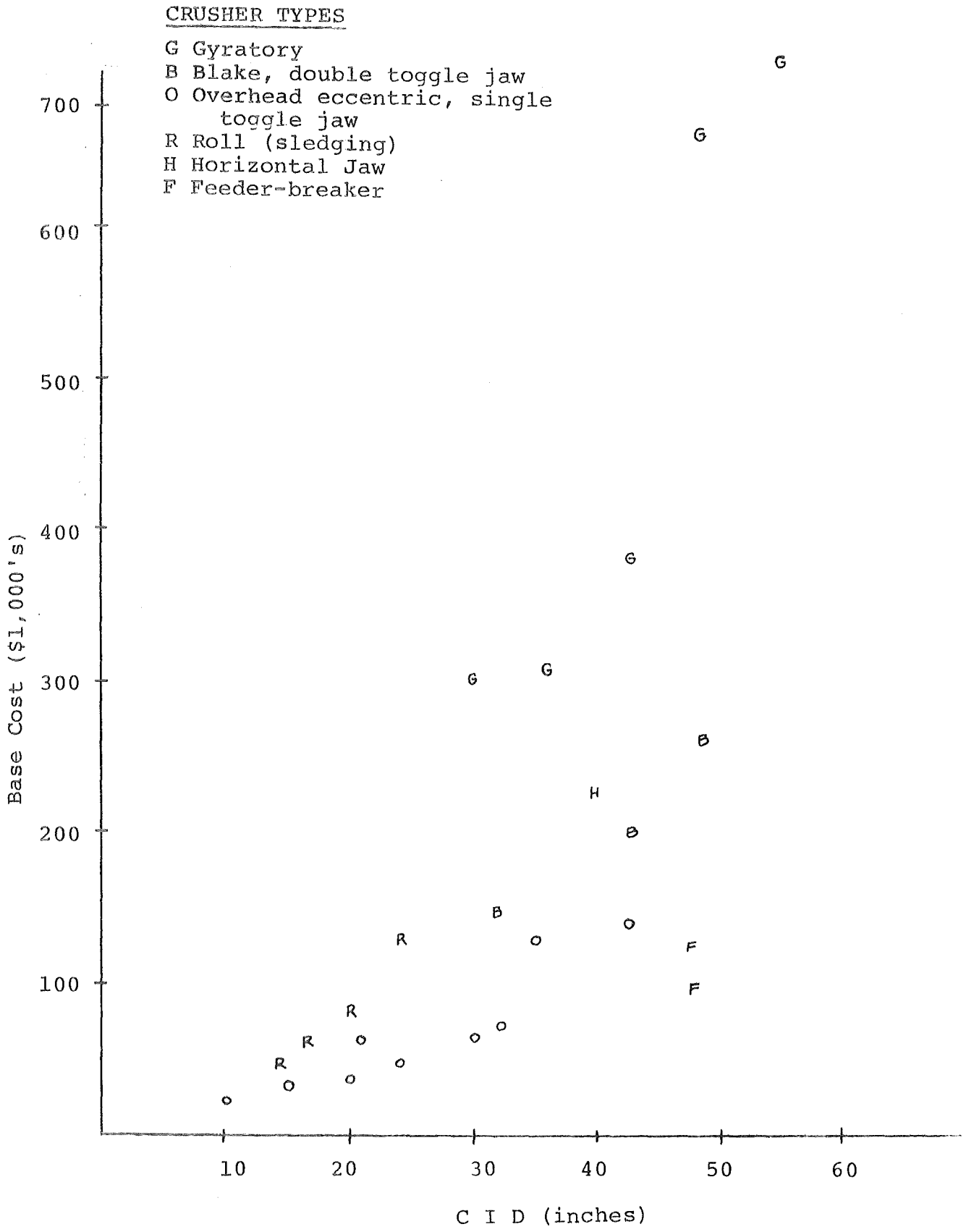


Figure 3-34 Crusher Cost Versus Critical Input Dimension (Ref. 3.2.10) USBM, 1976

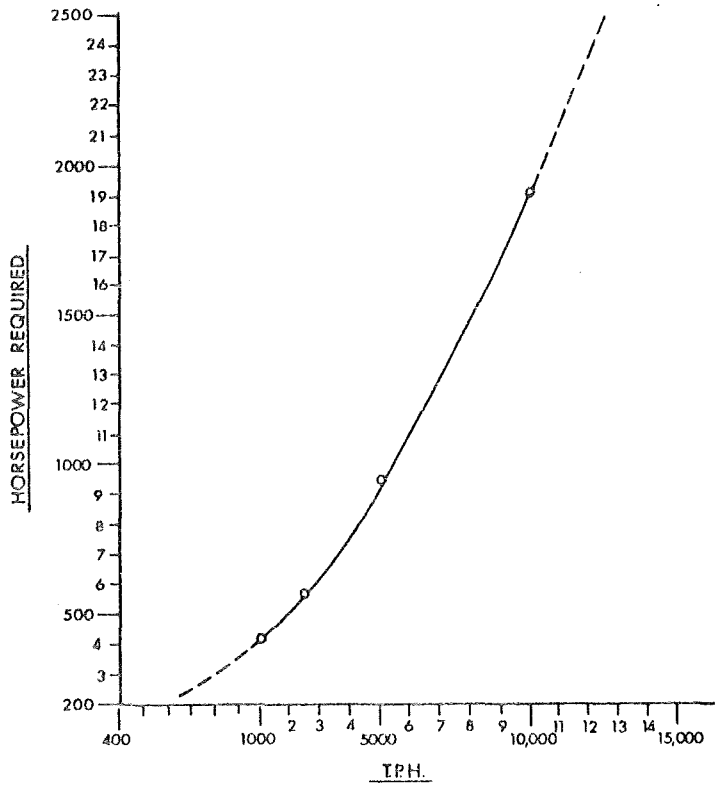


Figure 3-35 Crusher Horsepower Requirement Vs. Total Facility Capacity (Ref. 3.2.1)

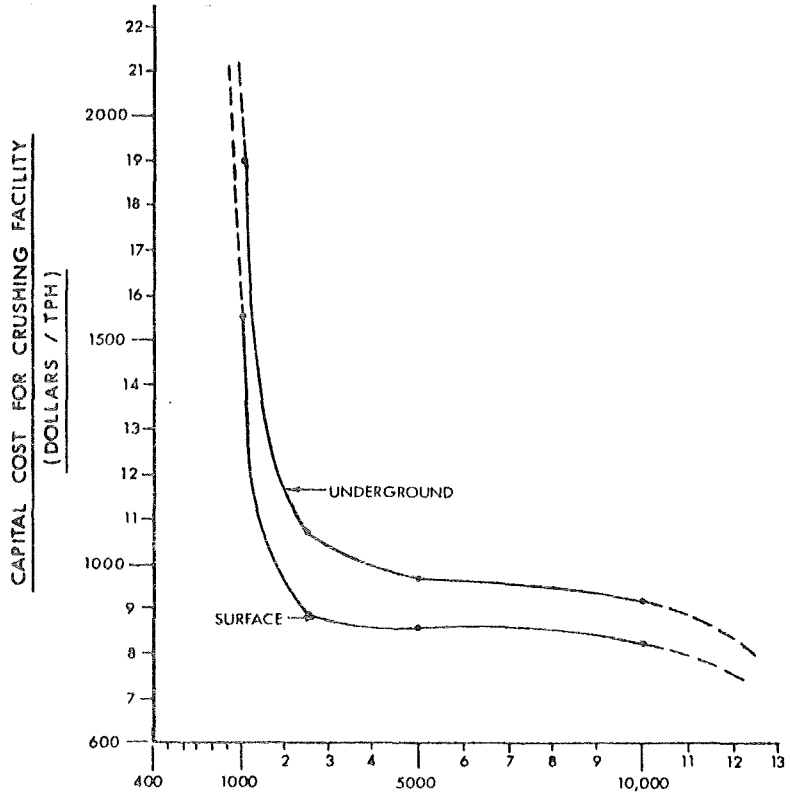


Figure 3-36 Crusher Facility Total Capital Cost Vs. Throughput Capacity (Ref. 3.2.1)

3.2.8 Crusher Economics for Mobile Facilities

The economics of mobile crushing units was not reviewed in sufficient depth during this program to make a valid comparative assessment against semi-permanent installations. Section B.2.7.7 has presented some information which illustrates the higher capital costs of mobile crushers and the lower operating costs (when haulage is included as part of the total analysis). The clear-cut advantage of using a mobile crusher is in applications where:

- Conveyor haulage out of the pit is the most economical ore transport technique.
- Multiple face operations on the order of 1,000 tph output constitute the mining method.
- Adverse truck haulage conditions exist in the pit due to weather and/or area hydrology.

3.2.9 Concluding Remarks

Selection and choice of location for a crusher is more than just a task of comparing equipment types and associated costs. The entire mining method must be treated as a system with each element offering a range of possible alternatives. This point will become increasingly apparent in the next three sections on haulage which point out how feed size and throughput capacity affect transport equipment selection as well.

3.3 Conveyor Systems

Open-pit mining continues to show a need for higher speed, higher capacity systems and components based on increased world-wide demand for raw materials. Discussions with equipment suppliers and systems engineers relating to customers' inquiries indicate that this trend is accelerating. Continental Conveyor, for example, has experienced an increased number of requests for information on 84- and 96-inch wide belts while belting manufacturers, such as Uniroyal and American Duck, have stepped up efforts to develop newer high strength belts of 1,500 to 2,000 piw. This section will briefly review the current trends in conveyor technology, summarize the design process, and highlight the potential applications of conveyor systems (both surface and underground) to future open-pit operations.

3.3.1 Background

The trend toward belt conveying of ore and waste in open-pit mines has been slow in the U. S., but now appears to be gaining momentum. Reasons for this change of events have been well summarized in a recent AIME-SME paper *(Ref. 3.3.1)* *© 1978*

> The depletion of high grade ore reserves has mandated a tremendous increase in the size and complexity of the ore haulage systems necessary for open-pit mines. These increased demands were served by rail haulage systems in the early large open-pit copper mines while later developed properties opted for the flexibility and lower initial costs of truck haulage. Now, we find the trucks being displaced as large overland belt conveyors tackle the problem of handling these ever larger quantities of low-grade ores with accompanying overburden ratios that can run as high as 10:1 or even 15:1.

Most of the recent developments in conveyor technology have taken place outside the U. S. while our country spearheaded the evolution of ever larger haulage trucks.

The following excerpt emphasizes this point: *(Ref. 3.3.1)* *© AIME, 1978*

> The basic development of large conveyors with shiftable sections and modular belts has generally taken place overseas during the last few decades. The coal fields of central Europe proved an ideal testing ground for conveyor innovation because the low density and natural friability of the material created an ideal medium for transporting by belt conveyor. Now these proven ideas are being considered for and implemented into both the soft and hard rock mining industries of the United States.

Increasing U. S. interest in conveyor usage relates to its capability of moving larger tonnages and its improved flexibility for relocation. The following elaborates on these points: *(Ref. 3.3.1)* *© AIME, 1978*

> Tremendous quantities of overburden have been moved with some of the systems subsequently evolving into waste rock removal systems. The problem of limiting the maximum size of transported materials has been overcome by locating the primary crusher in the pit and using the flexibility of mine trucks for the short haul from the pit face to the crusher.
>
> The advent of shiftable belts with rotary drive platforms and the ability to lengthen or shorten a belt by inserting or deleting standard lengths of unanchored conveyor tables is making the use of such conveyors increasingly attractive. Disassembly and reuse in a new location or a new use at an original location are two options open for the long-term planning of pit conveyors.

3.3.2 Mining Parameters Affecting Conveyor Design

In the design process, certain assumptions regarding the application of belt conveyors to open-pit mining operations must be made. For example, specific information on the nature of the conveyed material should be known in order to size and select the proper unit. In order to allow for the greatest range of material bulk densities, 100 lb/ft^3 is a reasonable planning figure for the hardrock industry. The

material lump size range, with nominal values from 6 to 12 inches, covers a representative cross-section of primary crusher product output sizes.

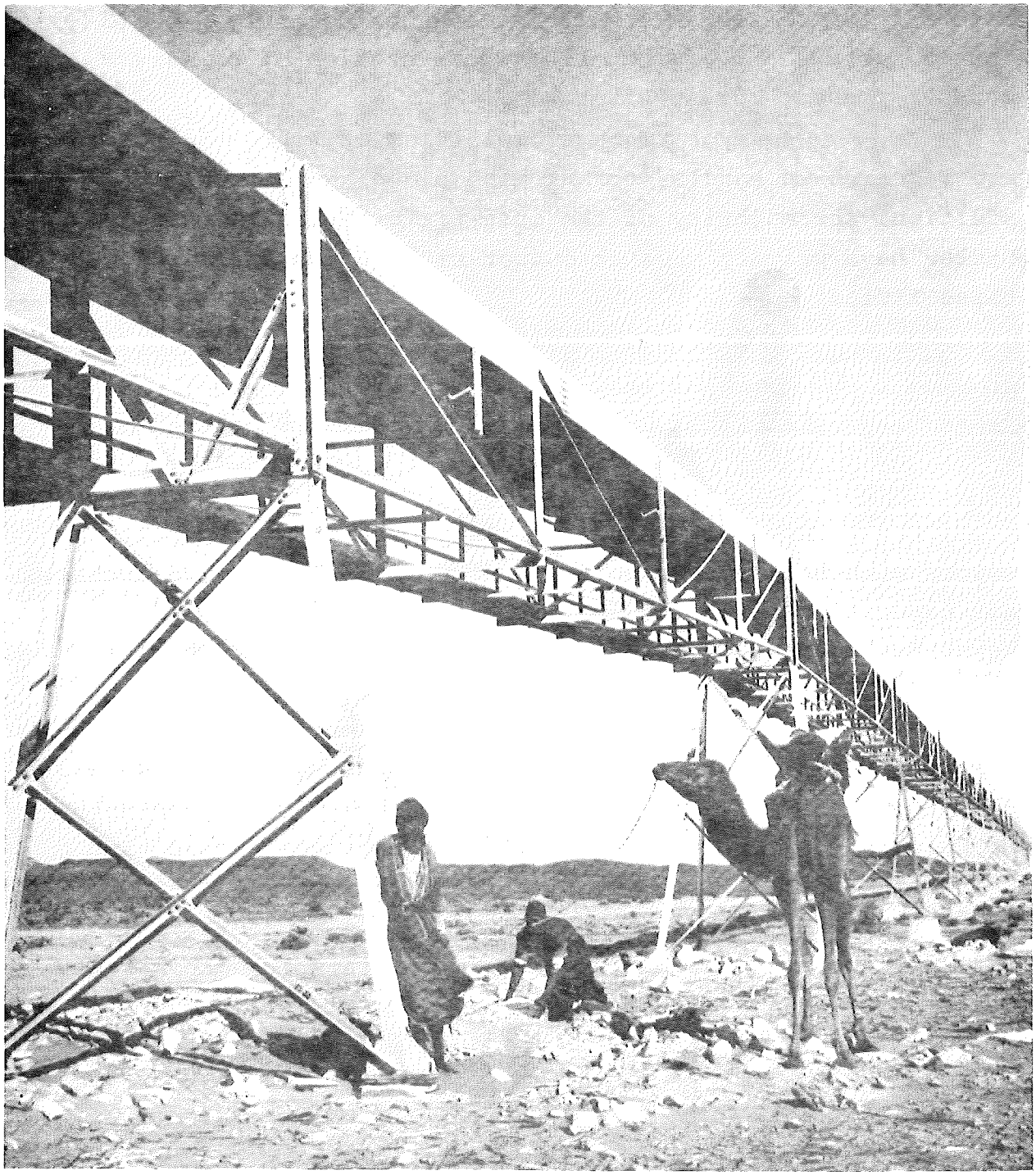
Conveyor tonnage capacities of 1,000 to 10,000 tph are required to handle current mine planning methods. In addition, the majority of the open-pit mines in the United States have primary crusher output rates which fall within these limits.

3.3.3 Principal Conveyor Design Parameters - Speed and Width

Present technology has produced lengthy belt conveyors with belt speeds up to 1,200 fpm, widths of 120 inches, and capacities of 40,000 tph. For example, the longest conveyor haulage network in use today is the phosphate handling system in the Spanish Sahara, Africa (62 miles long made up of 10 sections with a maximum individual length of 12 miles). The belt speed in this case is set at 885 fpm. (see Figure 3-37.) Another example is West Germany's Rheinische Braunkohlenwerke which has installed a 40,000 tph conveyor system with belt width of 118 inches running at 1,180 fpm. To set parameters for this discussion, the hardrock mining industry is interested in both high capacity (conveyors which can move at least 5,000 tph) and high speed (a minimum of 750 fpm).

3.3.3.1 Speed

Conveyors have increased in speed through the years to keep pace with rising mining capacities while holding capital costs to a minimum. Speeds were not increased dramatically though because of transfer point maintenance problems and potentially excessive belt cover wear. In fact, some belts



Fiaure 3-37 *Section Of The World's Longest Conveyor Erected In The Spanish Sahara (62 Miles In Length Running At 885 fpm)*

running at speeds of 600 to 700 fpm and handling abrasive or lumpy materials were fed by accelerator belts in order to limit the major cover wear to a relatively short and much less expensive belt.

With the emergence of the long overland conveyor, speeds took a substantial increase due to the reduction in capital cost realized by using smaller belts for a given capacity and the reduction in belt cover wear due to belt cycle time. The experience gained on these conveyors has led to the use of high speeds for much shorter conveyors. An example is an existing unit on 900-foot centers handling 10,000 tph of iron ore pellets and operating at a speed of 1,200 fpm.

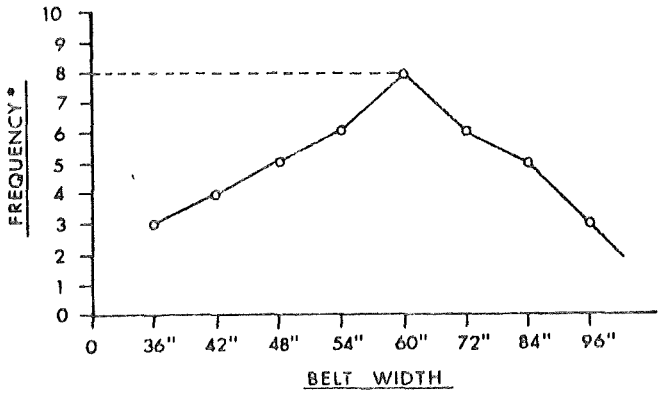
There are many unknown factors in belt conveyor design that still preclude very high speed application. With the present state of technology it appears that 1,200 fpm is the maximum belt speed that can be used with confidence. Friction factors, the dynamics of idlers and pulleys, optimal design of cross-sectional areas, and selection of low-wear materials are just a few of the performance factors to be determined on prototypes at speeds above 1,200 fpm.

3.3.3.2 Belt Width

At present, the largest belts made are 120 inches wide which reflects the limitation of current manufacturing procedures. Belt widths are also controlled by economic considerations such as transfer point costs, shafting sizes, idler designs, etc. It appears that conveyor widths above 120 inches will not find general use in the near future, although wider belts will probably be used for special application.

3.3.3.3 Tradeoff Between Capacity, Speed, and Width

Figure 3-39 shows the tradeoff between conveyor capacity and speed as a function of belt width. Experience to date indicates that for a mining operation with output ranging between 3,000 and 10,000 tph, the 60-inch wide belt is most often preferred (see Figure 3-38). This stands to reason as illustrated in Figure 3-39 where it is apparent that the 60-inch wide belt can deliver a greater total tonnage over the range of expected operating conditions. A similar argument can be made for application of the 36-inch wide belt to mining operations ranging between 1,000 and 3,000 tph.



* Number of TPH rates in 1000's achievable by various belt width systems in the belt speed range 300-1100 FPM and 1000-10000 TPH capacity range.

60" BELT ENCOMPASSES THE GREATEST TONNAGE RANGE (see Belt Speed vs. Tons/Hr. Graph)

FREQUENCY VS. BELT WIDTH GRAPH

Figure 3-38 Frequency vs. Belt Width - 60" Belt Most Preferred Between 3,000 and 10,000 tph. (Ref. 3.3.2)

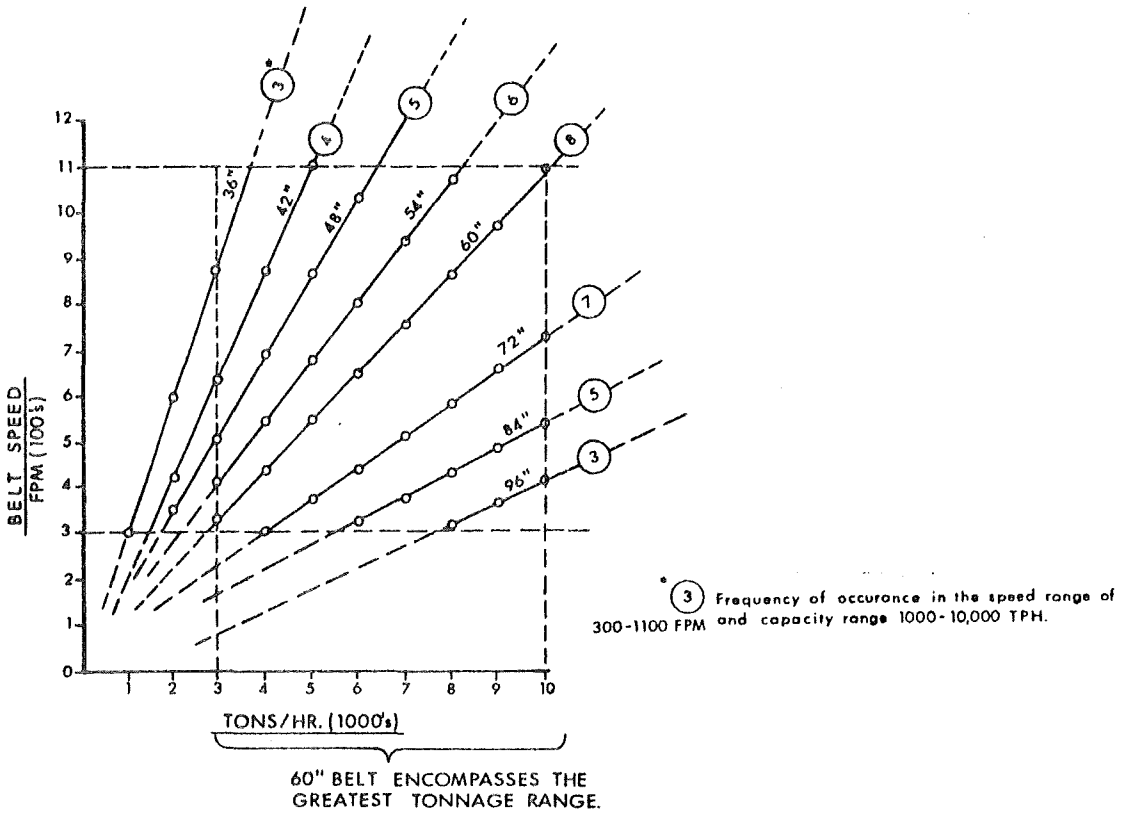


Figure 3-39 Tradeoff Between Belt Capacity, Width And Speed

3.3.4 Other Conveyor Design Parameters

With increased capacity and speed, larger belts and longer centers, the loadings and operating conditions on the individual components of a conveyor have increased substantially. This section looks at the new designs resulting from operating conditions and high loads on:

- Transfer points
- Idlers
- Pulleys
- Drives

3.3.4.1 Transfer Points

Accepted critieria in the design of transfer points (reduce belt cover wear, minimize horsepower requirements,

turbulence and dusting), require that the material be loaded on the belt as close to the direction of belt travel and speed as possible. With the many materials being handled today, fairly steep chute slopes are required. In some instances, a direct belt-to-belt transfer is required because of the severe problems associated with chute plugging. One development in conveyor design that has evolved to reduce the number of transfer points is the use of horizontal curves.

One of the most critical aspects of transfer point design is the manner in which the conveyor is loaded. Improper loading causes a number of problems, one of the more serious being difficulty with proper belt tracking due to unbalanced forces at the load point. One method for safely loading the main conveyor belt with the primary crusher discharge is by the use of feeders or accelerating belts. Feeders provide a relative degree of protection for the main belt by reducing the effects of impact forces at the load point. They also allow for a consistent and properly distributed cross-section of material flow, thereby enabling the belt to operate more efficiently with fewer shutdowns and less maintenance and repair time. The accelerating belt provides essentially similar protection.

In terms of feeder/accelerating belt of application, main conveyor belt speeds and primary crusher product discharge rates are the determining factors. Apron feeders are utilized more often in the case of large capacity and lower belt speed conveyors. The use of an accelerating belt for the case of faster main belts permits the material to approach the main belt speed within a short distance of the loading section, thereby minimizing the resistance of the material flow in the direction of belt travel. This provides for proper material flow to the main belt, enabling it to stabilize the load more quickly (an especially valuable feature when using an inclined conveyor).

The major unknowns requiring further design testing for high belt speed (above 1,200 fpm) include:

- Overflow problems as a function of belt speed and height from which the material is dropped.
- Additional horsepower requirements for the acceleration zone.
- Added dust control problems.
- Increased rates of wear on transfer point components.

3.3.4.2 Idlers

The standard rigid frame idler continues to find wide use in high capacity conveyors. This type of idler, with slight modification, has found use on wire rope supported conveyors . In Europe, the garland idler has found extensive use in a variety of applications. This is a ball bearing non-greaseable, idler that is made in three- and five-roll profiles. (see Figure 3-40.) It is beginning to find use in North America and its general acceptance should continue to escalate.

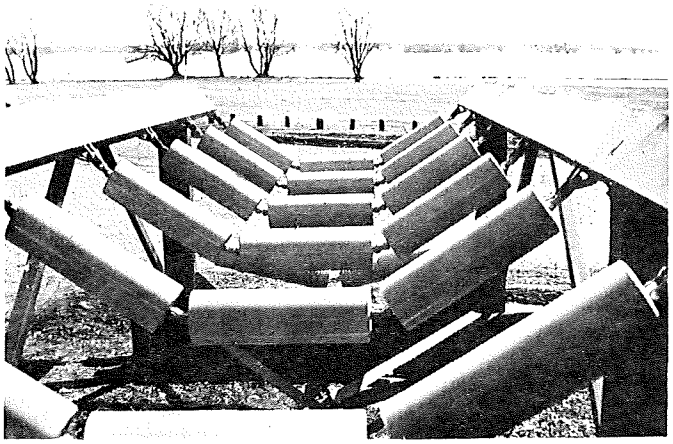


Figure 3-40 Conveyor Frame Section Showing The Garland Idler (Ref. 3.3.3) © Mining Congress Journal, 1977

The garland idler has some features that make it ideal for long overland conveyors and high capacity units:

- Ability to change profile due to pivoted support points which allows better training.
- Quick release arrangements for replacement of defective idler while conveyor is operating.

Of course, the inevitable tradeoff for this well-designed component is a higher capital cost.

There has been a trend throughout the world to increase idler spacing for sake of cost reduction. Idlers spaced at 8 to 10 feet or more are in successful use in the U. S., but basically to transport light material such as coal. One factor that limits idler spacing is the load capacity of the idler itself. With heavy materials and wide belts, the idler bearing capacity is limited. This could result in the use of 5 or more roll idlers in the future but the added expense of more rollers and heavier frames might not offset the reduction in cost due to increased spacing.

The effect of increased idler spacing on horsepower is not completely defined since the type of material handled, type of belting used, and alignment all have an effect on the overall friction factor of a conveyor.

3.3.4.3 Pulleys

The standard pulley (flexible type) has been in use for many years and with some modification has been adapted for use on steel cable belts. In this design, the bending moment at the pulley hubs is distributed between the pulley and its shaft in direct proportion to their respective stiffnesses. As loads increase, there was a trend toward the use of a rigid type welded pulley which, because of its extremely high stiffness, carries most of the loading. More recently, a cast end disc pulley has found extensive use in Europe. It is constructed

with an expansion hub that eliminates any welds between the shaft and end disc where control of the welds and fatiguing of welds has been a characteristic problem.

May factors contribute to the eventual fatigue and failure of pulleys:

- Rigidity of bearing supports which induce stress concentrations in the pulley.
- Pulleys machined without attention to a close tolerance specification for the diameter.
- Pulleys not perpendicular to the centerline of the belt.
- Improper belt splices.
- Belt overloading.
- Material buildup on the pulleys.

Failure of a pulley during mine operation for any reason is expensive and dangerous. The cost of lost production, as well as the cost of the materials, machinery and labor to put the system back in operation, could be substantially more than reliable pulleys and good maintenance programs.

3.3.4.4 Drive Systems

The requirements for conveyor drives with horsepower ratings ranging up to 12,000 hp per flight have become more complex due to higher speeds, increased lengths and the use of multiple drives. These requirements have resulted in the design of starters that give proper torque control for one or more drives during acceleration.

Intermediate drives have gained in popularity due to their ability to reduce belt tension which in turn allows longer belts, additions to existing systems, or increased capacities. Figure 3-41 shows a cross-section of an intermediate drive in use at the present time using radial pressure tires to provide driving traction. However, use of intermediate drives, especially a series of units, can result in a substantial

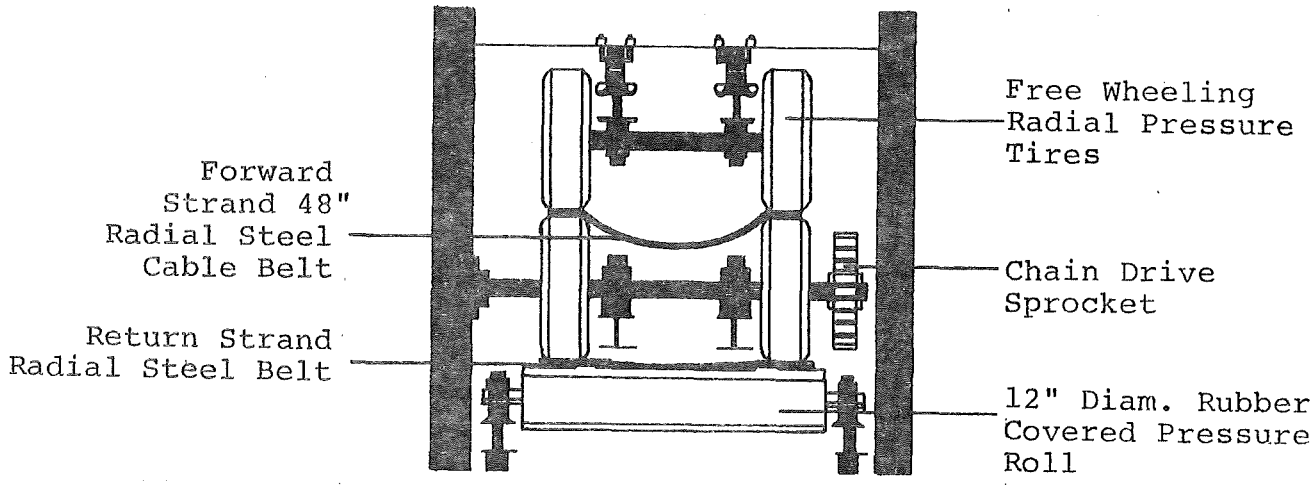


Figure 3-41 Cross-Section Intermediate Drive System. Radial Tires Provide Driving Traction for Conveyor (Ref. 3.3.3) © *Mining Congress Journal, 1977*

increase in electrical cost due to the necessity of synchronizing the drives. There are various other features which do not lend themselves to the handling of large lump sizes. In addition, backstop requirements, especially in the case of inclined conveyors, need to be investigated.

Another type of intermediate drive that has been under test for almost three years is shown in Figure 3-42. It has been installed on a belt handling 16,000 tph of material and has given excellent operating results. Another belt conveyor with 3,600-foot centers has recently been built incorporating this type of drive.

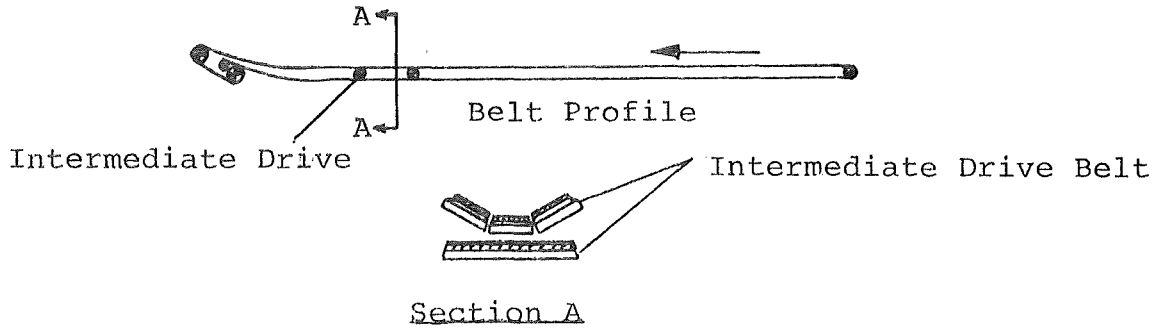


Figure 3-42 Profile Of Intermediate Drive Installed On A Belt Handling 16,000 tph (Ref. 3.3.3) © *Mining Congress Journal, 1977*

3.3.5 Conveyor Cost Computations Using a Case Study

The following illustrates the cost computation procedure for conveyor design by using a case study. Since this is a general description not specific to a given open-pit, site data such as location, nature of the mining operation, depth of overburden, pit slope angles and specific material characteristics other than bulk density (material surcharge angle, angle of repose, etc.) have not been considered. The study assumes the set of conditions described below.

The conveyor(s) are located in an open-pit mining operation with a conveyor profile (shown in Figure 3-43; horizontal to incline) which may undulate with concave and convex curves as required to fit the site-specific requirements. The horizontal portion will vary from 200 to 2,000 feet, depending upon the size and number of in-pit primary crushers being utilized to feed the conveyor and other mining method factors. The inclined section will vary from 400 to 2,500 feet with overall lifts of 100 to 800 feet for any single flight. For lifts greater than 800 feet, a transfer point is required to an entirely new conveyor (in the case of larger belts only). The additional conveyor(s) will have a maximum lift of 800 feet also. The number of conveyors and transfer points depend upon the amount of lift required. The pit depth will vary from 500 to 2,400 feet.

The conveyor horsepower will vary from 100 hp to 12,000 hp per conveyor, depending upon capacity and required lift as shown in Figure 3-44. The conveyor profile may require open cut work in hardrock in order to maintain a maximum incline angle of 18^{o}. Conveyor capacities will range from 1,000 through 10,000 tph utilizing material with a bulk density of 100 lbs/ft^3.

The conveyor belt will be a 60-inch wide steel cable belt, except for tonnage rates below 3,000 tph where a 36-inch wide belt would be required. The belt will have a

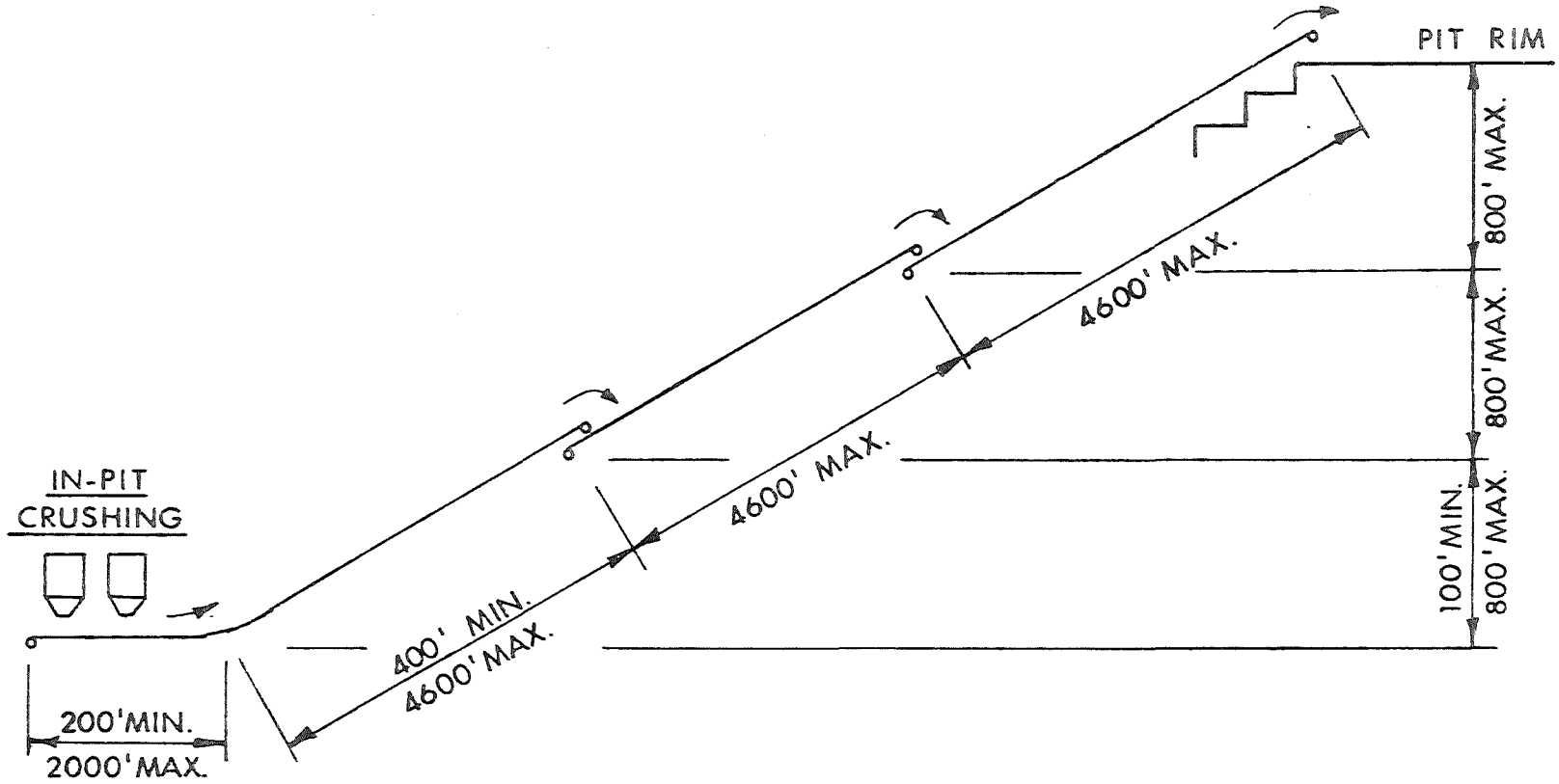


Figure 3-43 Case Study: Conveyor System Profile For Various Pit Depth Conditions
(Ref. 3.3.2)

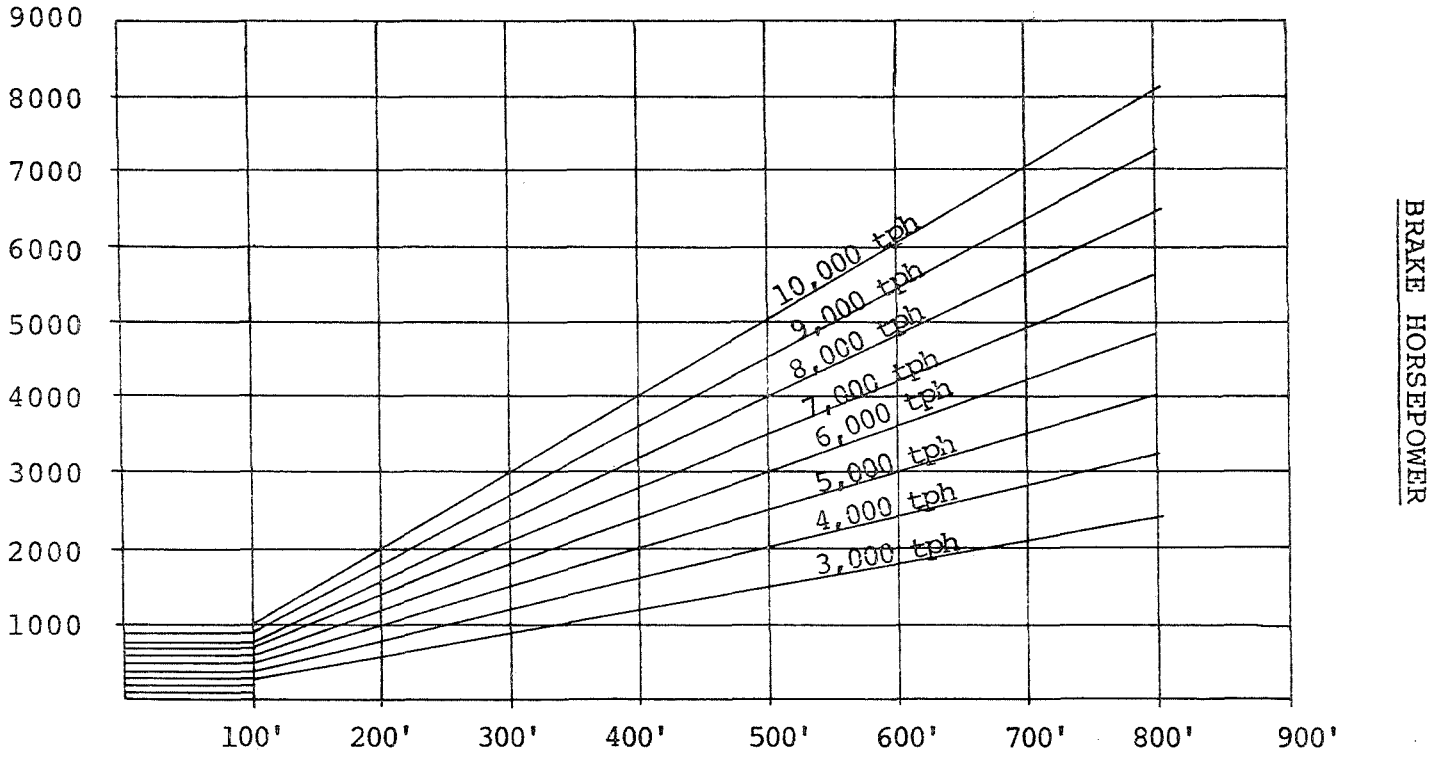


Figure 3-44 Brake Horsepower Required For Inclined Conveyor System As A Function Of Lift And Capacity Requirements (Ref. 3.3.2)

maximum tension rating of 4,400 piw and will utilize an 80% load capacity on 35^{o} troughing idlers with an estimated surcharge angle for the load profile of 20^{o} and an angle of repose for the material of 30^{o} to 35^{o}. The belt speed will range from a nominal 300 to 1,100 feet per minute for various conditions of lift and capacity.

The results of a series of calculations, which are discussed in depth in Appendix c, indicate that the 60-inch wide belt is the favored candidate as it best encompasses the majority of the variable loading conditions above 3,000 tph. The 36-inch belt provides an economical solution in the lower capacity range.

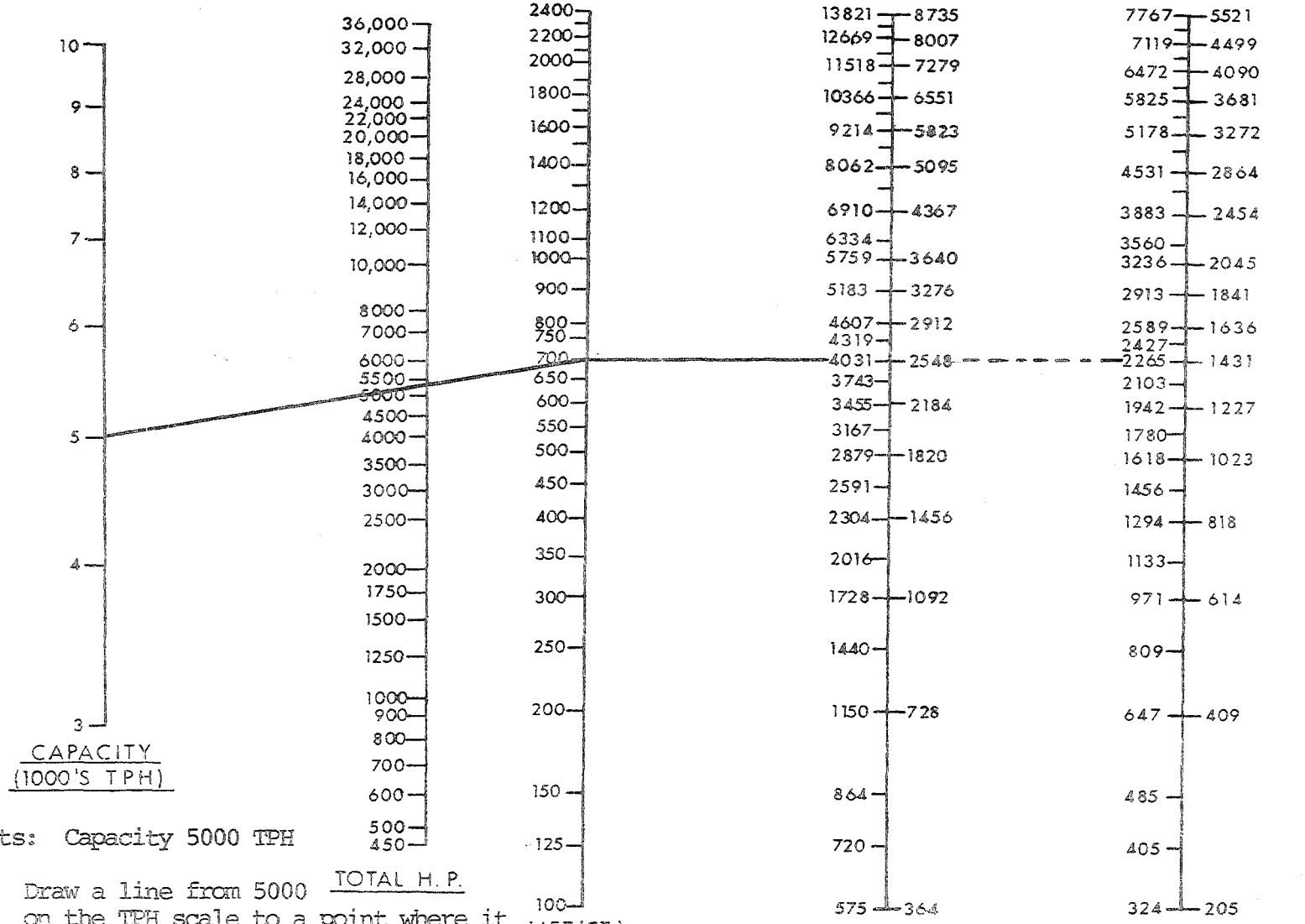
The method and type of feed to the conveyor is from a primary crusher with a nominal discharge of 12-inch maximum lump size. The 36-inch wide belt limits the maximum lump to a nominal 8 inch. The conveyor may be fed by an accelerating belt or belts, depending upon the number of stages of acceleration required.

The Cost Nomographs shown in Figures 3-45 and 3-46 provide the mine planning engineer with an easy method for computing the respective installed costs of the 60-inch and 36-inch wide in-pit conveyor haulage systems defined above.

3.3.6 Pros and Cons of Conveyor Haulage

Briefly stated, conveyor haulage provides the following advantages over other forms of ore transport:

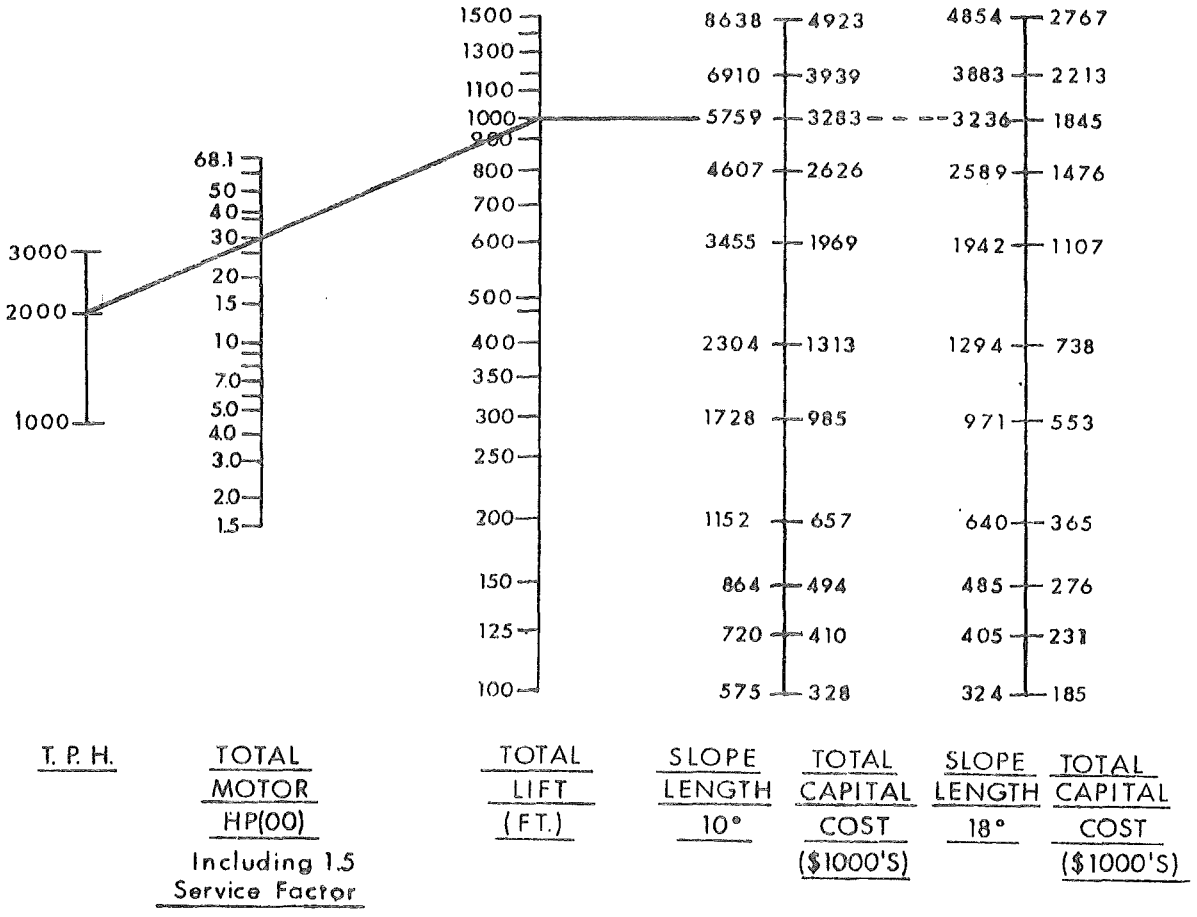
- Reduced energy consumption
- Lower manpower requirements
- Ability to negotiate steeper angles from the pit to the surface
- Less hydrocarbon pollution
- Lower maintenance requirements
- Lowest operating cost for certain ranges of horizontal and vertical distance



Requirements: Capacity 5000 TPH

Procedure: Draw a line from 5000 on the TPH scale to a point where it intersects 700 on the total lift scale. Total motor HP column reads approximately 5300. Total installed conveyor cost ($2,548,000) and slope length (4031 ft.) are found by reading horizontally from the 10° slope length column.

Figure 3-45 Procedure For Using Cost Nomograph To Estimate Price Of Single 60" Belt Conveyor Flight For Case Study (System Capacity Ranging from 3,000 to 10,000 TPH). (Ref. 3.3.2)



Requirements: Capacity - 2,000 TPH

Procedure: Draw a line from 2,000 on the TPH scale to a point where it intersects 1,000 on the total lift scale. Total motor HP column reads approximately 3,000. Total installed conveyor cost ($3,283,000) and slope length (5759 ft.) are found by reading horizontally from the 10° Slope Length Column.

Figure 3-46 Procedure For Using Cost Nomograph To Estimate Price Of Single 36" Belt Conveyor Flight For Case Study (System Capacity Ranging From 1,000 to 3,000 TPH). (Ref. 3.3.2)

The principal drawbacks of conveyor haulage include:

- Loss of flexibility in altering the mining plan.
- Danger of having all the "eggs in one basket" should the conveyor break down.
- Primary crushing in the pit is required.

3.3.7 Concluding Remarks

The use of high capacity belt conveyors will continue to expand due to increasing demand for raw materials and the necessity of minimal capital expenditures. Overland conveyors have found a place in industry and mining where they are competitive with truck and rail haulage. Short center conveyors are finding application in high speed and/or high capacity use. The use of conveyors with horizontal curves shows an interesting new approach for both short and long conveyors. The requirements for longer and/or higher capacity conveyors will continue with the continued development of intermediate drives. The future development, testing and use of high speed conveyors (above 1,200 fpm) will continue to provide higher capacities and lower capital costs.

Present-day economics require that belt conveying systems be evaluated as an alternate to truck haulage as pits deepen and hauls lengthen in hard rock mining. The future availability of electric power and the almost certain increasing cost of truck fuels may one day dictate the use of continuous conveying systems at many large open-pit mines. Such mines, with significant ultimate depths projected, should be planning now to use conveying systems in conjunction with local mine truck haulage. Although documentation is limited, the indicators are that large and lengthy belt conveying systems will, in the future, be supplanting mass truck haulage in selected areas of the hard rock mining industry.

3.4 Ore Passes

The one common element in all continuous underground haulage systems from open-pits is the need for passes to transfer the excavated material from the surface to underground handling points. The prime factors affecting ore pass design are the geometric relationships of the open-pit. The main design variables pertaining to an individual ore pass include:

- Location in the pit
- Depth (may change as a function of time)
- Size and shape (cross-sectional dimensions which can vary as a function of depth)
- Angle of inclination
- Estimated throughput (expected life)
- Desired storage capacity
- Loading and discharging area design considerations
- Maintenance considerations
- Ore pass development (possible geotechnical problems)
- Economics (installation and operating costs)

The following sections review some of the above variables in greater detail, pointing out the kinds of decisions mine planners are faced with.

3.4.1 Choice of Location

One of the first tasks in ore pass design is to locate the ore pass system based upon the evolution of pit layout over its total life expectancy. Whether one or more ore passes are used, a centroidal location(s) must be determined since the main objective is to cut down on the length of haul by trucks. A computer simulation of the mining plan can simplify this determination, especially since the time dependent nature of this calculation can become somewhat complex. In

addition to locating the centroid(s), a rock mechanics study must also be performed to determine whether the rock is competent for an ore pass system.

3.4.2 Size and Shape Considerations

The size of an ore pass is generally determined by its intended use and the type of material to be passed through. If the primary crusher is underground, then the opening must be large enough to handle the pit-run rock without causing "problems" in the systems. The rule-of-thumb for planning purposes has been to design the minimum cross-sectional dimensions of the ore pass 3 to 6 times the maximum rock dimension handled. Several pits, by careful blasting or by the use of a grizzly, have had much smaller ore passes than the above rule would dictate.

One main advantage of a larger ore pass is that it can be used for surge capacity. However, the development and maintenance costs will also be much higher. Another little-known benefit is that the actual cross-sectional area through which material movement takes place will decrease as you approach the discharge point. The theory is that the material creates a natural funnel effect within the ore pass. (see Figure 3-47.)

The smaller ore passes of 7- to 10-foot diameter are generally more economical in development cost and maintenance but are limited in size and quantity of material that can be handled. If the rocks are relatively small due to use of a surface crusher or being of a type that undergo attrition easily in the ore pass (due to low shear or compressive strength), then smaller ore passes are preferable. The problems with the interlocking of and cohesion between the rock particles are greater with the smaller ore passes and this fact must be weighed against the lower development costs.

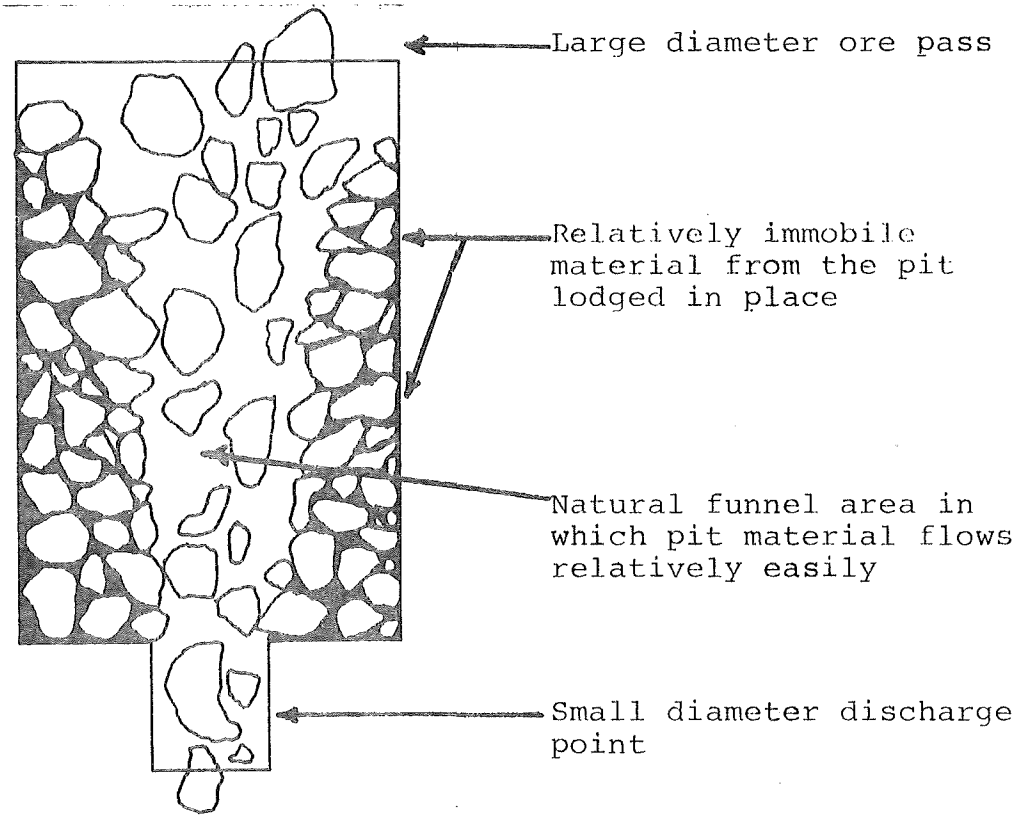


Figure 3-47 Illustration Of Funnel Effect Resulting In Larger Ore Passes With Cross-Sectional Dimensions Significantly Greater Than Those Of The Discharge Point.

The following factors illustrate situations where larger ore passes are preferred.

- Need for surge capacity (underground crushing, for example, is the intended mode of operation)
- Large tonnages being moved hourly
- Large individual rock sizes
- Natural size distribution in the excavated rock conducive to packing
- Possible segregation of the material in the ore pass
- Tendency of the ore to crush and pack
- High friction factor on the ore pass walls
- High incidence of moisture and freezing, causing hang ups

Most ore passes are circular in shape to give maximum strength, though if they are driven by conventional methods, a square or rectangular shape is generally used. Timber raises

are not in use as major ore passes from the surface because of their high maintenance factors. Long rectangular passes have slightly less tendency of hanging up than the circular ones, especially where there is a slot opening at the bottom or a large drawpoint.

3.4.3 Angle of Inclination

The following summarizes the advantages and disadvantages of vertical versus inclined ore passes:

- Advantages of vertical ore passes
  - Cheaper development costs
  - Less erosion of the walls
  - Less segregation of the rocks
- Advantages of inclined ore passes
  - Possible less hang up problems with arching of the rock
  - Force vector analysis shows that an unbalanced component on the upper side of the ore pass tends to keep the material moving better (Ref. 3.4.1)
  - Can be designed with orientation so as to pass through competent rock
  - Several can be driven to a common discharge point underground
- Disadvantages of angle ore passes
  - Maximum angle must be greater than 60^{o} in almost all cases
  - Higher erosion problems, possible slabbing of the footwall rock

3.4.4 Loading Methods

There are several methods of dumping into the ore passes, each of which generally depends on the material haulage system being used in the pit and other local conditions. A few are listed on the following page:

- If conveyors are used, the crushed rock is loaded directly into the pass without any special equipment except safety guards around the opening.
- When the primary crusher is located above the ore pass, the opening has to be well reinforced for safety and to prevent erosion.
- Trucks can use temporary or permanent berms to back up against while dumping into the ore pass. However, care must be taken, especially with the temporary berms, to make them large enough to prevent the trucks from backing into the hole.
- One operator has devised a steel door structure over the ore pass that opens up as the truck bed raises. It was constructed for safety and ventilation reasons (Ref. Fig. B-27).
- An older unloading system for trucks with side dumping is a grappling hook that dumps the truck bed sideways into the ore pass. *(Ref. 3.4.1)* *© AIME, 1968* This is an elaborate system and is generally used in cases where the ore pass is near the bottom of the pit.

One consideration in designing a temporary dump point is the proposed method of lowering it as the pit develops. The most common approach is to fill the ore pass to the top, blast the new bench around the pass to the desired depth, and then bulldoze the rock into the opening until sufficient room is obtained for the normal shovel and truck operation. Another method that was used at an Idaho dam site was to ramp down into a slot from the new bench area and truck the rock up around to the ore pass, until the shovel breaks through to the ore pass. This gives a solid bench and a clean cut for the trucks to operate in (see App. Fig. B-36). Depending upon the configuration of the pit, the lower bench could be started at a distance from the ore pass and could then progress towards it, with the final blast being pushed into the ore pass by dozers, front-end loaders or shovels.

A grizzly at the dump point is one way to avoid large rock from entering the ore pass. A grizzly is also

necessary to keep timber and steel out of the system. Where large volumes of ore are to be moved, it may be necessary to provide a drill rig or impact hammer at the grizzly for clearing hang ups.

3.4.5 Discharge Points

For purposes of breaking the fall of the rocks in the ore pass, improving safety, and preventing damage, mine planners may change the angle of the ore pass near the bottom, install a knuckle, or put in rock boxes at the point of discharge, The problem with a knuckle is the possibility of creating a point of hang up, in which case an access drift must be driven to the knuckle. Either a knuckle or a rock box is necessary ahead of a feeder in a primary underground crusher system. Chutes might also be used to prevent possible damage by falling rock. A final alternative is the installation of a draw-point discharge where the bottom of the draw in effect breaks the drop of the rock. However, this requires a horizontal material transport system, possibly the use of LHDS, in order to get the material moving again. The following reviews each alternative in some detail.

The discharge point of an ore pass system can pose the most critical design requirements. There are several methods of discharging from ore passes and these will vary for each mining plan. The simplest is the use of draw-points from which the ore then can be transferred to other passes and bins ahead of crushers or loading systems. LHD equipment is typically used although trucks may be required if the haul distance is fairly extensive. The need for maintenance is low, and in the case of a large opening, the problems with hang ups are decreased. Draw-points also allow an LHD to blend ore from several areas. This generally necessitates a concrete floor if a considerable amount of ore is to be moved from the draw-point.

When several ore passes are used to feed to a central ore pass, the same rules apply. At the intersection of the ore passes it may be necessary to use reinforced concrete to protect

the brow. Care must also be taken to install a relatively large angle at the intersection.

The second method is to construct chutes at the bottom of the ore pass. When properly designed, these chutes are generally very expensive to build, but the ore can then be loaded onto the underground haulage system at a high uniform rate. This system, however, generally has the highest rate of hang up, especially if the ratio of chute ore pass size to maximum rock dimension is not large enough.

Automatic and manual gates can also be installed to provide a great deal of control over material movement. For example, ore can be loaded directly into ore cars, trucks, or onto conveyors. Automatic feeders, which can be used in combination with the gates, work extremely well when dumping onto conveyor belts.

If ore passes feed directly into the crushing system, then the design becomes more elaborate. The crusher needs a feeder system; generally a heavy duty apron or vibratory feeder is preferred. It is best to feed the rock from the back section of the feeder which will reduce the amount of reinforcement (concrete, steel, rail or steel plate) required in this area. Off-sets, such as knuckles and rock boxes, will also reduce the maintenance level.

The ore pass should always have some ore in it to protect the feeder systems. If the ore pass must be emptied for maintenance reasons, then before the large rocks are dropped into it, several loads of sand should be used to break the damaging fall of the coarser material.

3.4.6 Ore Pass Development

Most methods now used for development of ore passes (except down-drilling) necessitate the extraction of the blasted material through the underground workings. These methods include:

- Conventional--open raises

- Conventional--two compartment raises
- Alimack raise system
- Down-drilling--raise borers
- Up-drilling--raise borers

The following sections review the advantages and disadvantages of each.

3.4.6.1 Conventional Open Raise

This is the simplest of all raising methods, but it rarely exceeds 100 feet in length and is not considered economical for distances greater than 50 feet. The pros and cons of this technique are listed below:

- Advantages
  - Easy and fast set-up
  - Set-up time fast for short distance raises
- Disadvantages
  - Damage to ladders and pipes from the blasted rock
  - Poor ventilation after a blast
  - Loss of efficiency as the raise goes higher
  - Danger in barring-down after each blast
  - The maximum economical size is approximately 100 sq. ft.
  - Slow development time

3.4.6.2 Conventional Two Compartment Raise

This method is used for greater lengths than the conventional open raise. The maximum economical length is between 100 and 200 feet, though it can be driven farther. The pros and cons of this technique are listed below:

- Advantages
  - The timber manways protect the miners and equipment
  - A larger cross-section can safely be blasted than with open raises

- Ventilation is better than with open raises
- Blasted rock can be loaded through chutes

- Disadvantages
  - High cost for timber
  - High maintenance cost for ore pass if timber manway is left in
  - Maximum economical length is only 200 feet
  - Slow development time

3.4.6.3 Alimack Raise System

For the past 25 years, the Alimack raise system has been used throughout the world. It has unlimited capability as regards to raise size and length, though in recent years raise boring equipment has become more economical in the softer and medium hard rock formations. The raise climber has a self-propelled working platform with a personnel cage suspended beneath the platform (see Figure 3-48). The drive pinions are powered by air, electricity, or a diesel-hydraulic engine. The raise climber travels on a single guide rail which is supplied in curved and straight sections of 3 and 6 foot lengths. All scaling, drilling, and loading of the powder is done from the working platform. The raise climber then descends to a safe section of the shaft during blasting. The air and water supply lines located in the guide rail create a fine mist after the blast to dissipate the blasting fumes and dust after the detonation of the explosives.

For driving blind raises, the Alimack system can choose from one of three drive systems, depending on shaft length:

- Air driven--Maximum 500 foot length with the limiting factor being air pressure availability
- Electric--600- to 1,500-foot length with the limiting factor being the electric cable weight
- Diesel-hydraulic--unlimited length

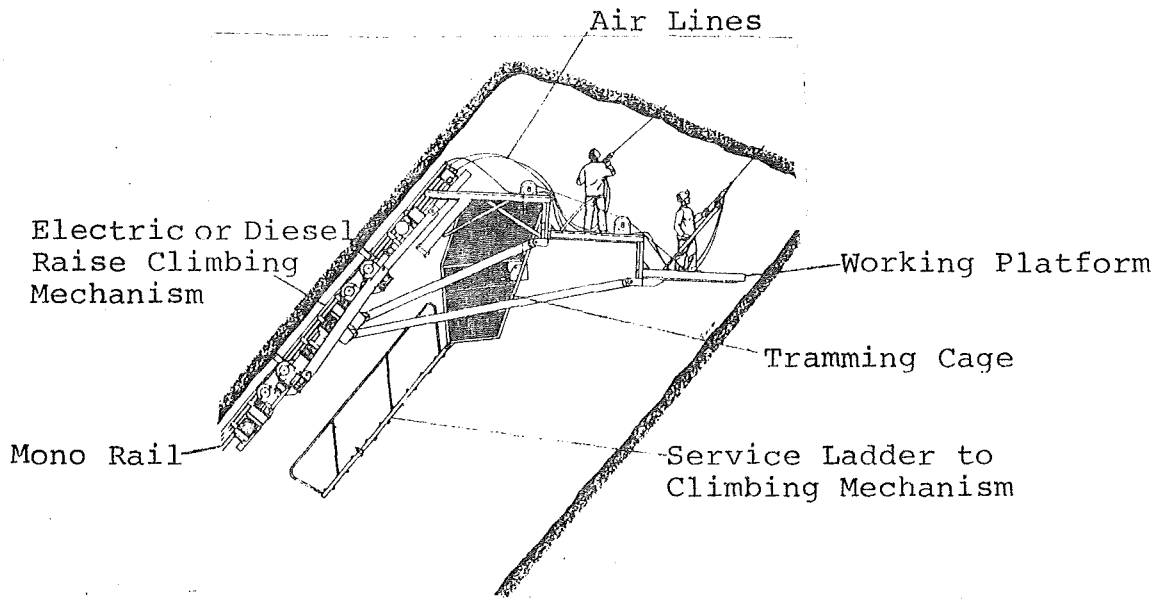


Figure 3-48 Alimack Riase System

The basic Alimack raise system can be used with a ring-drill set-up which can enlarge the raise from 20 to 30 feet in diameter.

The advantages of the Alimack System include:

- Fast set-up time
- Ability to drill and blast several times per shift
- Improved ventilation over conventional methods
- Fast access to the face for men and materials
- Stable platform for drilling and loading holes
- Unlimited length can be driven
- Can be driven either vertically or on an incline
- Can create smooth wall through precision blasting
- With the use of ring-drills a large cross-section can be blasted (up to 30 feet in diameter)

The disadvantages include:

- Preparation time for original set-up is slow and expensive

- Skilled miners are required for equipment operation.
- Initial capital investment is high.
- The walls must be strong enough to hold anchors.
- Average development cost per foot is higher than for raise borers in softer rocks.

3.4.6.4 Raise Boring

The most recent advances in raise development have been in the area of raise borers. The newer raise borers are drilling deeper and generally in harder rocks. The two possible methods of raise boring include up-reaming and blind hole drilling. Up-reaming is generally the cheaper of the two methods, but there must be access to underground hoisting to remove the cuttings.

The advantages of raise boring are:

- In soft and medium-hard rock the development costs are the cheapest of all methods.
- There is no cycle time and therefore total drilling time is reduced.
- The smooth round surface of the raise is a superior shape for coping with extreme ground pressures and reducing the wall friction factor.
- Since the rock is drilled and not blasted, the raise walls are not fractured, thus adding strength to the raise.
- The safety is improved by not having miners working in the high-risk areas of the raise.
- The noise level is much lower than with the use of pneumatic drills in "bald-headed" raises.

Among the disadvantages are:

- Economic advantage is much less in very hard rocks.
- Since only a limited number of ore passes would be drilled in the continuous haulage system, the mobilization and demobilization costs could be excessive.
- Blind drilling is limited in depth.

- For up-reaming the cuttings must be extracted by underground haulage.

3.4.7 Maintenance

The biggest single problem in any ore pass system is the potential for hang ups which impede the downward movement of the material. They can result from the following causes:

- Rock size is too large in relation to the size of the ore pass.
- The graduation in size of the rock is such that it contributes to the arching phenomenon characteristic of a hang up.
- The slope of the ore pass is too flat.
- Knuckles can be the point of arching or hang ups.
- A large percentage of fines in the material and/or moisture can cause packing of the rock. Freezing can cause the same problem, but generally is not a significant factor.

When hang ups occur, placing concussion bombs in the ore pass will generally start the rock moving. The explosive can either be placed at the bottom of the pass or raised up on the end of a series of wooden poles to the arched area. In extreme cases, a hole may be drilled from the surface, or new access drifts driven to the point of hang up. In a small ore pass, an access raise is sometimes driven or placed beside the ore pass. This is generally more expensive and of itself can create additional maintenance problems.

The second major maintenance problem in ore passes is erosion of the walls. This is especially acute where the rock is soft. Where the ore pass is on a slope, the footwall will show a great deal of wear (as much as 60 feet in one mine). If the rock has a tendency to slab, then either the angle of slope or the direction of the ore pass must be changed. Another method for reducing the erosion, if arching is not a problem, is

to keep the ore pass full. The friction factor between the rock and the wall remains the same, but the impact forces of the falling rock are eliminated. Concrete is a possible solution for the erosion problem because it has greater wearability than many rock types. However, considering the added costs to install the concrete lining (not to mention the possibility of its breaking off large slabs after a certain amount of wear), it is not generally used except at the bottom to protect the discharge areas.

3.5 Hoisting

There are several large underground ore hoisting systems in use in the United States. One of the largest from a tonnage standpoint is located at a copper mine in Arizona. Capacity for this four-shaft system is in excess of one hundred thousand tons per day, and recently they hoisted their 300-millionth ton. From a depth standpoint, a silver mine in Idaho has the greatest single lift hoist designed to operate from 7,700 feet but presently operating at approximately 5,700 feet.

This section will review the kinds of hoists typically used in underground ore transport and will discuss the design tradeoffs considered by the mining engineer in evaluating hoist systems. It is important to note that hoist selection is always a site specific process, although these are general guidelines and geometric relationships which favor their application.

3.5.1 Types of Hoists

There are essentially two types of hoists:

- Drum
- Friction

Several design variations of each are available, but the basic difference is that the drum hoist wraps and stores the hoist rope on the drum, while a friction hoist has the hoist rope pass over the drum connected to some type of counterweight. Friction hoists are best suited for single-level hoisting, while drum clutched hoists work well for multi-level hoisting.

3.5.1.1 Drum Hoists

Drum hoists are usually classified as single or double drum. The single drum hoist can be either a service or

production hoist with a cage or skip in balance with counterweight. It can effectively service one or more levels except when there are two skips in balance which would dictate single level hoisting.

The other hoist type is a double drum arrangement with one or both drums "clutched." Service cages and production skips can serve several levels efficiently. The advantage of the clutch drum is that it permits quick adjustment of the ropes during initial stretching. With the double clutch one compartment can be used for men and supplies while the other is not operating.

Another type of drum hoist that was developed for the deep mines in South Africa is the multiple drum hoist. It has two ropes per drum that permit hoisting from a great depth using smaller ropes and smaller drums. It has the greatest advantage below 6,000 feet in depth.

The friction hoist can use either single or multiple ropes for hoisting. The single rope has a lower safety factor as well as lower power consumption. The following are some of the advantages of the friction hoists over the drum types:

- The hoist diameter is less with smaller multiropes
- The weights and sizes of the equipment are less.
- The direct drive system is mechanically simple.
- Multiple ropes give greater safety.
- Capital costs are less.
- Can be operated at a greater speed.

Some of the friction hoist disadvantages are relative to the drum type:

- Energy consumption per ton of ore hoisted is greater.
- With two skips in balance, hoisting must be from one level.
- It is difficult to keep the ropes in balance.

- The friction hoist cannot be easily adapted to shaft extensions.
- A deeper shaft sump is required to accommodate the tail rope loop.
- The rope-changing equipment costs more.
- The balance rope requires a heavier suspension gear.
- Maximum economical depth is in the range of 4,000 feet.

3.5.2 Design Parameters for Selecting Hoist Type

Considerations involved in determining the type of hoisting system include:

- Mechanical requirements
- Electrical requirements
- Shaft layout
- Head frame and hoist house arrangement
- Rope/cable arrangement

Besides the above, there are certain external design parameters such as state and federal laws, type of power available, type and location of material being hoisted, present and future production requirements.

3.5.2.1 Drive System

Availability of power will determine what type of drive system is used. Most manufacturers prefer DC power which requires a motor generator set to convert outside AC line power to DC. At present, these units are used only on hoist systems with horsepower requirements of 3,000 hp or more. A new system currently available is the cyclo-converter drive which is a direct drive slow-speed AC motor that is controlled by varying the cycles of the AC voltage. This

system is only economical at horsepower levels of 7,000 to 10,000 hp. For power systems where 200 hp or less is required, the new static AC drive is proving popular. It uses silicon-controlled rectifiers and AC wound rotor motors to control motor voltage.

Hydraulic drives are popular in some cases where the equivalent horsepower requirement is 300 hp or less. Types available include hydrostatic drives, high torque motors that are low speed and speed reducers that utilize high speed motors. In all cases the initial capital cost is quite reasonable, but overall life is shorter than conventional drives and maintenance is higher.

3.5.2.2 Brake System

In all cases, the type of braking system is important. Two types are currently being used:

- Parallel motion
- Jaw

The parallel motion brake applies pressure at two points on the break ring 180^{o} apart. In early hoist systems, this method was preferred over a jaw brake because it gave uniform wear on brake shoes. Newer modifications of the jaw brake have overcome the wear problem and are now used on more hoists.

Due to federal and state laws, a brake system must have controls that provide a dead-man switch and a gravity backup. The dead-man switch requires the operator to keep his foot on the switch at all times while the hoist is operating. If his foot is removed, the hoist will automatically go into emergy braking. The gravity backup is provided in case of power failure. This system generally uses heavy weights on a brake lever. The weights are kept in the off position by applying hydraulic or pneumatic power during

normal operation. In the case of power loss, the brakes are automatically applied by gravity.

3.5.2.3 Shaft Layout

Depending on depth and production requirements, a certain type of shaft will be sunk. Presently most shafts in the United States are circular and concrete-lines. Shaft diameters will range from boreholes of 6 feet to conventionally sunk shafts up to 30 feet. The shaft design must take the following points into consideration:

- Tonnage requirements and type of ore being hoisted
- Depth
- Ground conditions
- Ground water to be pumped
- Ventilation requirements
- Size of mining equipment to be lowered underground
- Location of shaft in relationship to ore body
- Surface or underground shaft
- Vertical or inclined configuration

Other requirements could be dependent on state regulations. For example, it may be possible to use rope or wood for guides. Most states require wood guides if men are to be hoisted, although at least two western states, namely Utah and Wyoming, will allow rope guides on man trips if the hoist is multi-rope. Some states will also allow steel guides, although most still require wood.

In the case of hoisting on slopes, most of the above items will apply. However, there are a few unique problems. The first is that of rope wear. For slopes of any length it is necessary for the rope to lay along the bottom of the slope or incline. By rubbing against the floor, increased rope wear results. To overcome this problem, there are several types of

rollers to keep the rope off the floor and reduce friction. These rollers are high maintenance items. Another method is to apply rubbing timber and replace it on a regular schedule. The second problem is that of rail maintenance. In some cases, the weight of ore and skip may reach 60 tons. For such conditions, it is extremely essential to provide good track maintenance.

3.5.2.4 Maintenance Requirements

Daily maintenance of friction hoists is greater than for the drum types and this differential increases with shaft depth. Generally it takes about one hour more each day for routine maintenance. The unscheduled downtime is also greater for friction hoists, as is the repair time. The ropes on the friction hoists cannot be re-shackled as is the case during periodic drum-type inspections, so the safety factor during initial design is higher.

Most of the new hoists have automatic lubrication systems, which include flood lubrication combined with ring oiling on the hoist sleeve bearings.

In the recent years, automatic skip loading systems have been used. The load measurement is by weight with volumetric backup included to guard against overflow of low density material.

Deflectors are located below the skips to direct spillage into pockets which are similar to loading pockets. The spillage can then be manually loaded into the skips. The deflectors are interlocked to ensure that they are out of the shaft prior to hoisting from lower levels. If friction hoists are used, then an auxiliary hoist might be used to hoist the spillage to the loading pocket. Another method for handling spillage is to drive a slope to the spillage pocket and then haul the material back out to the ore pocket by LHDs.

3.5.2.5 Power Requirements

Normally, the power consumption per ton of ore hoisted is greater with the friction hoists. On the other hand, drum hoists can cause significant voltage fluctuation in the power system, causing power peaks up to 250 percent over normal.

3.5.2.6 Personnel Requirements

Ore hoists can be operated on manual, semi-automatic, or fully automatic control. With the use of T.V. monitors, automatic skip loading can also be accomplished, but generally one man still must be used to operate the automatic ore hoisting system. The maintenance crews will vary from one to three men for normal hoist maintenance. The number required for shaft repairs will depend on the type and condition of shaft, type of guides, and the amount of water.

3.5.2.7 Deepening of Shafts, Slope, and Ore Passes

One of the more serious problems with continuous haulage from underground is that once the ore pass, slope or shaft system is established, it is difficult and expensive (but not impossible) to modify. When a vertical shaft is deepened, all production from the hoist facility must stop. This could mean a loss of production for several years if the shaft is the only point of hoisting. An expensive alternative is to sink another shaft to the lower levels. Many times, it is preferable to raise the second shaft from the bottom of the original shaft. This type of sinking is expensive, since the waste rock is oftentimes hoisted up the ore haulage shaft.

One possible method is to raise the shaft using the Alimack System (see Section 3.4.6.3).

If proper safety precautions are taken, a conveyor slope can be lengthened downward without loss of production, except when the waste material has to be conveyed to the surface. A bulkhead can be installed in the slope to protect the shaft sinkers. When the added slope is completed, the conveyor can be lengthened downward or a transfer point put in at the extension interchange.

Deepening the ore pass system causes the least difficulty, provided there is no underground crusher. The deeper ore pass system is installed and a transfer raise is driven between the old and new ore passes.

The problems with deepening usually result from insufficient planning prior to mine development. Other factors relating to changes in economics, product demand, technology advances, etc., can still affect the initial mining concept in an unforeseen way. Nonetheless a detailed study during the preliminary planning phase must try to anticipate the full range of consequences pertaining to deepening the shaft.

3.5.2.8 Economics

Recent comparative studies *(Ref. 3.5.1)* have shown the following relationships for shaft versus slope installations:

- The capital cost per vertical foot, excluding equipment, favors shaft installations at depths greater than 580 vertical feet. This assumes an equivalent usable cross-sectional area and calculations based on a 12 ft. x 20 ft. slope on a -15° grade versus a 20 ft. inside diameter vertical shaft.

- The haulage capacity of comparative cross-section area production shafts and conveyor slopes favors slopes in all cases due to the inherent higher limits of conveyor capacity.

- The minimum capital cost for production systems (multiple shafts vs. single slope, or large diameter hoisting shafts vs. smaller cross-section conveyor slopes, or other combinations) generally favors shafts in the combined range of depth greater than 1750 feet to 2750 feet, with production requirements 5000 tpd to 30,000 tpd. While below those parameters conveyor slopes have the most favorable capital cost.

3.6 Other Haulage Methods

This section reviews other less popular alternatives for future ore/waste haulage systems, namely:

- Rail
- Slurry
- Pneumatic hoisting

3.6.1 Rail Haulage

Although rail is currently the most common method of underground transport, trackless haulage and conveyors have made considerable progress as prime haulage methods in recent years. This study on continuous haulage did not include a detailed study of rail haulage, but since this method is being used for large tonnage movement it will be discussed briefly.

Rail haulage from underground ore pass discharge points requires that the system must be well designed for fast, maintenance-free loading and dumping of the rail cars. Several methods can be used, such as:

- Automatic gates in chutes
- Aprons
- Vibrating feeders

The use of slushers to load trains applies to small tonnages and would not be applicable to continuous haulage systems. With the use of TV monitors, the load points can be automated as is the case at an iron ore mine in Sweden, where a number of trains load ore from 12 galleries and 50 chutes for transport to the crushers. The whole operation is remotely controlled from a central control room. Another outstanding automated rail haulage system is the recently installed one at a mine near Denver, Colorado, that will haul 30,000 tpd over a 14.4 mile distance to the concentrating plant. *(Ref. 3.6.1)* © *AIME, 1975* One Canadian mine uses 100-ton cars and 15-car trains to haul iron ore 8 miles to the processing plant. An apron feeder loads

the train at a rate of 2,000 to 5,000 tph. *(Ref. 3.6.2) © AIME, 1968*

There are several advantages in rail haulage:

- More flexible than conveyors
- Can also be used to transport men and material
- Fewer skilled maintenance people are required to maintain track haulage equipment
- More than one product can be hauled
- Capacity of system can be varied by adding or subtracting equipment
- Can handle a coarser rock than conveyors
- Can have frequent direction changes

The disadvantages include:

- More difficult and expensive to install
- Can haul only on low grades--3 to 3½%
- Operating procedures are dependent on human decision and performance
- Training of operating and installation personnel is more difficult

In coal mine design, rail haulage is recommended over belt conveyors when the haul distance is over 4 to 5 miles *(Ref. 3.6.3) © American Mining Congress, 1975.*

3.6.2 Slurry Haulage

Surface slurry haulage has gained ground in the last few years, with a large number of systems being installed for many different types of materials such as coal, limestone, copper, iron ores, etc. The present haul distance varies from a few miles up to 273 miles. The maximum tonnage being moved is 12,000,000 tons per year through a 20-inch line (Table 3-16).

Slurry pumping from underground mines has very limited application and several conditions must exist for this system to be viable.

TABLE 3-16

SUMMARY OF SELECTED COMMERCIAL SLURRY PIPE LINES

Slurry Material	System or location	Length (miles)	Diameter (inches)	Annual thruput millions tons/year	Initial Operation
Constructed					
Coal	Consolidation	108	10	1.3	1957
	Black Mesa	273	18	4.8	1970
Limestone	Calaveras	17	7	1.5	1971
	Rugby	57	10	1.7	1964
	Trinidad	6	8	0.6	1959
	Colombia	17	7	0.4	1944
Copper concentrate	Bougainville	17	6	1.0	1972
	West Irian	69	4	0.3	1972
	KBI Turkey	38	5	1.0	
	Pinto Valley	11	4	0.4	1974
Magnetite concentrate	Tasmania	53	9	2.3	1967
	Waipipi (land)	4	8	1.0	1971
	Waipipi (offshore)	1.8	12	1.0	1971
Magnetite and Hematite	Pena Colorado	30	8	1.8	1974
Gilsonite	American Gilsonite	72	6	0.4	1957
Tails	Japan	44	12	0.6	1968
Nickel refinery tailings	Western Mining	4.3	4	0.1	1970
Under construction					
Magnetite and Hematite	Sierra Grande	20	8	2.1	
	Brazil	250	20	12.0	
	Mexico	17	10	1.5	
Planned					
Coal	U.S.A.	1036	38	25.0	
	U.S.A.	750	22	9.0	
	U.S.A.	800	30	16.0	
	U.S.A.	180	24	10.0	
	Canada	500	24	12.0	
Phosphate	Australia	200	16-22	4.0-6.0	
Magnetite and Hematite	Africa	350	18	6.6	
	India	36	2-22	10.0	
Limestone	Australia	44	8	0.9	

(Ref. 3.6.4)

© Coal Age, 1975

- Abundant and reliable supply of water
- The material must be crushed to a fine size to maintain proper pumping velocities
- The haul distances and tonnages being pumped, because of the costs of the fixed plant, will have the greatest influence on the economics

Other critical design factors for a slurry include:

- System should be operated continuously
- Reliable power
- Designed for unscheduled shutdowns
- Anticipate abrasion and corrosion
- Possible degradation of the material during transport

Most of the underground slurry pumping systems are only supplements to existing conventional haulage, except for the pumping gilsonite at a mine in Utah, and coal mines in Germany, Japan, Poland, and Russia *(Ref. 3.6.4)* © *Coal Age, 1975*

One of the prime economic factors affecting the use of slurry pumping from underground has been commented on by Donald Ashe, Vice-President, Allen-Sherman-Huff Pump Co. *(Ref. 3.6.5)* © *Mining Congress Journal, 1965:*

> The economical factors of hydraulic transportation and hoisting are not clearly understood. Each installation would have to be made on its own merits.

3.6.3 Pneumatic Hoisting

Recently pneumatic systems have been designed and manufactured to move 3-to 4-inch material 2,000 feet horizontally and hundreds of feet vertically using low pressure compressed air. The maximum capabilities of pneumatic hoisting at this time for single stage systems are 250 tph and 650 feet of vertical lift, and 100 tph for 1,500 feet vertical lift *(Ref. 3.6.6)* © *CIM Bulletin, 1975*

The ease of installation and versatility of pneumatic hoists are excellent; also they are cheaper to install in and around existing equipment. In addition, they do not have the watering and dewatering problem of slurry lines.

The energy consumption of pneumatic hoists is much higher than with conventional hoists or conveyors. The maintenance requirements are also high, especially if the material is abrasive.

4. U. S. MINE MANAGERS' CRITIQUE ON CRUSHING AND UNDERGROUND CONTINUOUS HAULAGE

Feedback from U. S. mining industry personnel was solicited on the foreign and domestic practices described in Section 2.2 in order to gain insight into how crushing and underground continuous haulage might be more popularly received here at home. Presentations were arranged with various mining companies to discuss the findings of this program, and to receive input on their experiences in evaluating the possible implementation of similar systems. This section presents the results of these meetings and provides a comprehensive assessment of open-pit mining industry opinion on:

- Ore passes
- Underground crushers
- Moveable vs. fixed crushers
- Truck haulage
- Continuous haulage (underground and above ground)

4.1 Selection Process

General critieria which governed the selection of mining companies to provide feedback included the following:

- The company should have potential orebodies with geometric relationships conducive to potential application of underground crushing and continuous haulage systems.
- Mine management should be progressive and open-minded regarding new ideas and have expressed interest in evaluating this type of technology for possible application to current mining operations.
- Mine engineering and operating personnel should be familiar with the concept and theory of underground continuous haulage and know about other sites where such systems have been implemented.
- The mine should be either a metal or non-metal open-pit operation.

- There should be a mix of geographic locations to provide a range of regional prejudices and experiences concerning such systems.

The above criteria led to the selection of nine different mining companies which expressed willingness to receive the presentation. Table 4-17 provides general information on the companies that were ultimately involved. Figure 4-49 illustrates the geographic location.

4.2 Reasons for Mining Company Interest

There were several factors common to most of the mining companies which had influenced their initial interest in underground crushing and continuous haulage. These included:

- Current operating problems with local pit wall stability which gives rise to the consideration of moving underground in those same areas.
- Concern that larger sized trucks (350-ton and greater) will impose limitations to flexibility of operation and be a mismatch to present pit equipment.
- Concern for escalating energy costs and the need to find alternative haulage methods with lower operating cost.
- Fear of future fuel shortages.
- Environmental concern for air pollution attributable to truck haualge.
- Environmental concern for dust generation and noise emission near urban areas.

Some of these same companies had already decided against the implementation of underground crushing and/or continuous haulage for a variety of reasons:

- Belief that continuous haulage and the handling of multiple ore types plus waste are incompatible.
- Investment analysis which shows that, as long as trucks are required to accomplish stripping, the full value of their estimated production life is achieved only through their use in ore and waste haulage.

TABLE 4-17

LIST OF MINING COMPANIES RECEIVING PRESENTATIONS AND PROVIDING FEEDBACK ON UNDERGROUND CRUSHING AND CONTINUOUS HAULAGE SYSTEMS

Type of Mine	Mineral	State(s)	Reason for Selection
Open-Pit (2)	Copper	Arizona	Orebodies have appropriate geometric constraints and mine personnel are familiar with these systems.
Open-Pit	Iron-ore	Michigan	Orebody has appropriate geometric constraints.
Quarry	Limestone	Eastern U.S.	Mine personnel have evaluated similar systems for potential application.
Open-Pit	Copper	Montana	Mine personnel have evaluated similar continuous haulage systems for potential application.
Open-Pit	Gold	Nevada	The operation is spread throughout a large open-pit and the orebody has appropriate geometric constraints.
Open-Pit	Iron-Ore	California	Mine personnel have evaluated similar systems for potential application.
Open-Pit and Underground	Molybdenum	Colorado	Closely parallels several of the European mines that were visited.
Open-Pit	Molybdenum	New Mexico	Mine personnel are currently evaluating similar continuous haulage systems.

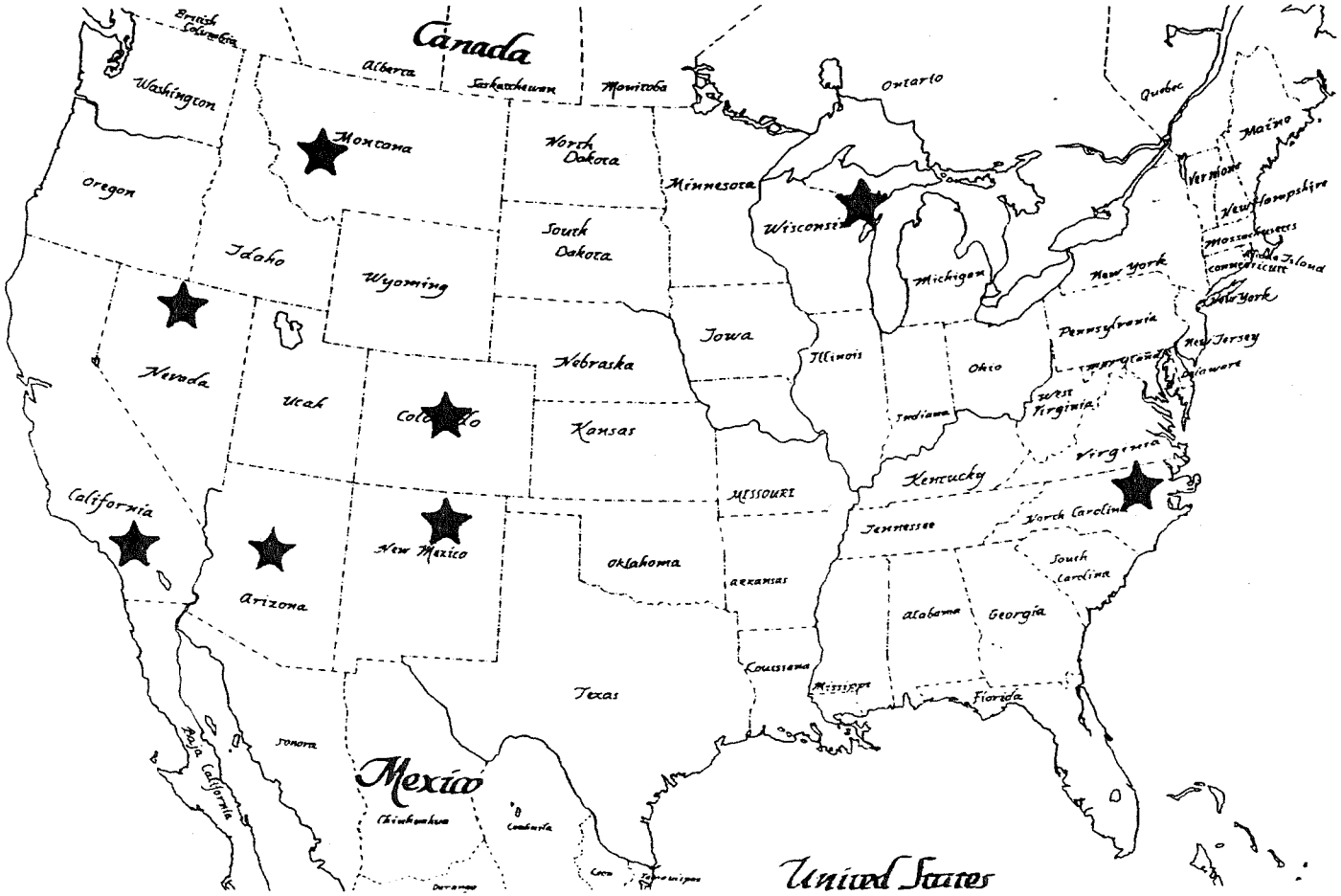


Figure 4-49 *Illustration Of The Geographic Distribution Of The Mining Companies Which Provided Feedback On Crushing and Underground Continuous Haulage.*

- Flexibility to adapt to changing pit configuration is seriously impaired.
- General ground instabilities. Limitations to steepness of pit slope angle is imposed due to potential wall instability problems.
- Belief that conveyor systems are not reliable enough.
- High capital costs and associated risks inherent in the "eggs-in-one-basket" approach.
- Lack of adequate information on cost and upkeep procedures for ore passes.
- Shortage of funds required for the sizeable initial capital investment.

Many of the companies had investigated and evaluated various combinations of crushing and haulage schemes in the course of making design decision for current operations. The most significant unknown factor in the minds of all concerned related to the potential impact technology might play over the course of time. One company, for example, had studied the use of in-pit crushers and conveyor haulage in the early 1960's, but opted to use a truck fleet due to the lower capital cost. Reviewing that decision today shows that continuous haulage was in fact the less expensive approach from both the unit and the total cost standpoint--inflation and energy having become much more important cost factors since that time. The impact of events like this on the mine planning process has been the development of a dollar-time matrix in the design stage which attempts to model the potential long-term influence of anticipated technological changes.

4.3 Comments on Ore Passes

The subject of ore passes generally raised more questions than could be satisfactorily answered in a one-day session with each mining company. The immediate design considerations concerning their use were for the most part site specific factors and focused on items such as:

- Advantages and disadvantages of underground crushing chambers.

For example, illustration of the mobile crusher system (see Section 2.2.7) provoked many cost related questions on the trade-offs between mobility and fixed installation. Of particular concern to mine operators was the issue of how to select a specific size of crusher and still allow for sufficient extra capacity to cope with expansion of future pit operations. One concern related to whether a 1,000 tph system could be mechanically modified with minimal effort to provide a 1,500 tph capacity versus having to purchase a new piece of equipment.

The overall consensus among the U. S. operators that were contacted appears to be negative on the idea of underground crushing. A number of reasons can explain this:

- These personnel had limited field experience with this kind of installation. An Idaho dam site (see Section 2.2.6) represents one of the first cases in this country where underground crushing fed by an open-pit was attempted.
- Fear of the inherent inflexibility which is automatically locked into the mining system. Should management want to alter the mining plan at some later date, the underground crusher becomes a liability.
- Concern for the higher capital cost associated with placing the crusher underground.

More exposure to the concept of underground crushing will be required before mining companies in this country become proponents of this method.

4.5 Comments on Continuous Haulage

This subject provoked the sharpest debate among mine managers and engineers since virtually every company had wrestled with the issue of truck haulage versus some other technique during the mine planning effort. The most frequently considered alternatives were:

- Conveyor (horizontal and inclined)
- Rail (horizontal and inclined)
- Skip (vertical and inclined)
- A combination of the above

Generally speaking, aboveground transport was the preferred mode of operation with underground haulage receiving interest only after other alternatives had been exhausted.

As is the case with moveable crusher systems, lack of experience in the U. S. with underground and/or continuous haulage has kept mine management from more aggressively pursuing conveyor systems as a viable alternative to truck fleets (although there has been some progress in this area). Suggestions on conveyor haulage made by mining personnel during the presentations centered on the need for:

- More definitive performance data (both technical and cost), especially on foreign mining operations.
- A reliable method for tagging different ore grades and waste.
- Fail-safe installation procedures for securing conveyor systems to the pit wall. Monitoring equipment to check for possible movement.
- Cost and performance comparisons between:
  - Conveyors anchored to the pit wall
  - Open trench cuts from pit bottom to the surface for a conveyor system
  - Underground inclined conveyors
- Site specific information on conveyor mobility for relocation within the pit.
- Research on:
  - Steeper angle belts
  - High speed transfer points
  - More reliable components such as idler belts
  - Operating costs for low- and high-speed belts
  - Faster belts

The overall conclusion drawn from the above comments is that the high initial capital cost of continuous haulage has

been the overriding factor working against its acceptance. This is related both to the limited amount of investment funds available to management during start-up and to the projected life of the mining operation (that is, whether it can hope to recover these sunk costs). In addition, problems with pit wall stability have an important secondary influence on the assessment of continuous haulage systems. Both the required pit slope (which in turn determines the stripping ratio) and the precautions taken in designing the anchoring system are the economic factors of consequence which ultimately affect management's choice of haulage methods.

Looking toward the future, the pending restrictions imposed by ecological and environmental mandates may well shift the cost picture to favor the high up-front capital investment.

4.6 General Comments and Recommendations

Mine operators were keenly interested in the pieces of information on foreign mining companies that were brought to light through this program. However, more precise data is badly needed if the U. S. is to seriously consider implementing these new haulage and crushing practices. For example, there was significant controversy about the added cost required to install an underground crusher due to the need for a ventilation system and other safety factors associated with underground excavation. These uncertainties can be dispelled only through an in-depth review of past experience with similar systems. The best prospect for obtaining detailed cost information of this kind is to approach certain state-controlled mining companies in foreign countries where the data of interest are more readily available.

4.7 Concluding Remarks

To conclude on an optimistic note, the potential for the future implementation of new crushing and haulage schemes is on the increase due to an important set of changing circumstances. The present generation of mining company decision makers rose through the ranks during the "truck-and-shovel" boom era which started in the 1950's. The upcoming generations of mine engineers and shift supervisors is not quite so attached to truck haulage due to the energy and ecology problems which have forcefully impacted on mine operation during this decade. As a consequence, senior mining personnel have become more aware of the alternative operating methods that some of their junior personnel are claiming deserve attention. For the sake of maintaining productivity and holding down costs, management is now more willing to listen to these new concepts, ideas and theories on how to improve efficiency of operation. This bodes well for the future prospect of the crushing and continuous underground haulage schemes discussed in this report.

5. COMPARISON OF HAULAGE SYSTEMS

Haulage systems comparison is the final step in the evaluation of the continuous haulage elements and it is necessary to tie in all the variables that are generally found in an evaluation.

The "Research Study of Continuous Haulage from Open-Pits" is basically a study of surface versus underground haulage methods with the emphasis placed upon the continuous haulage methods from underground. By definition in this study, hoisting by skips is also considered to be a continuous haulage method.

5.1 Continuous Versus Non-Continuous Haulage from Open-Pit Mines

While the contract for this study restricted the scope of investigation to primarily underground haulage systems, a brief discussion of the other alternatives is warranted. It is generally accepted that the tremendous increase in surface mining experienced in the last three decades is a result of the superior productivity of surface mining over underground mining. And without question, the remarkable advances in excavation and haulage technology characterized by huge haul trucks (up to 350 ton capacities) have been astounding.

The diversity of equipment types and capacities and the multitude of combinations emphasize the care with which the operators have tailored the excavation and haulage systems to their particular production levels and orebody configurations. One has only to look at the older surface mines to see the progression of excavation and haulage systems to larger and larger unit capacities. And, it has been the North American mining industry which has led the way.

The economics of large scale open-pit mining in the past have resulted in the vast majority of open-pit mine operations adopting truck and shovel haulage systems. These systems generally utilize compatibly sized electric shovels and truck

fleets. The ideal case being that the number of trucks, haul distance/cycle and the shovel capacity can be so coordinated that neither the trucks nor shovel set idle. Since this is rarely achieved the primary excavator, the shovel, is periodically idled resulting in a "non-continuous" excavation and haul system. Other interruptions to the smooth flow of material out of the pit occur with truck bunching, mechanical failures and clean-up in the loading area.

The coordination of the haul units, belts, trucks, rail, etc., to the excavating unit then is the criteria by which this report defines haulage system as non-continuous or continuous. The impact on availability of the prime excavator by the haulage system removing ore/waste from the pit is greater for cyclic systems such as trucks and ore trains than say for an adequately sized crusher/conveyor. Please note that the combined load/haul cost dictates the superiority of continuous over non-continuous systems, not just the efficient utilization of the prime excavator.

Haulage systems viewed as being non-continuous when exclusively applied to removing material from the pit either strictly on the surface or through tunnels are:

- Truck haulage systems
- Rail haulage systems
- Combined systems-such as waste removal by truck and ore removal by rail
- Dozers - for short distance and low capacities
- Scrapers

Whereas haulage systems viewed as being continuous would include:

- A minimal truck fleet feeding a stationary in-pit crusher feeding an in-pit conveyor or skip system. And, which elevates ore and waste on independent systems.
- A loading unit feeding directly into a mobile crusher that feeds an in-pit conveyor.
- Bucket-wheel excavators with conveyors.
- Dozers or scrapers feeding in-pit conveyors.
- A minimal truck fleet feeding ore/waste passes with crushing either on the surface or underground.

The ore passes acting as surge bins to feed a:

- Horizontal transport system such as underground trucks, rail systems or conveyors that feeds vertical skip hoisting.
- To inclined slope conveyors or inclined skip hoists.

Each and every haulage system will have applications which favor it over all others. This study has identified the most continuous haulage system as being crusher fed conveyor systems and the least continuous as being pure truck haulage. Therefore, a comparison of the advantages and disadvantages of each are here presented in the following pages.

5.1.1 Trucks Versus Conveyors

Large off-highway trucks are and have been the prime haulage units for a large majority of open-pit mines throughout the world for many years. They have been developed from the 20 ton units in the 1950's to the present 350 ton model. Generally most of the units are up to the 200 ton range in the big open-pits.

It is believed that this trend in truck capacity has leveled off and that increasing capacity can no longer be relied upon to compensate for increasing fuel, labor and repair costs. On the other hand conveyor technology appears to be growing vigorously. And, conveyor haulage has demonstrated superiority in known applications.

In general truck advantages over conveyors are:

- Low total costs on short-to-medium haul distances
- Lower capital costs at shallow depths
- Versatile--haul to and from several locations with different products
- Rock does not have to be crushed until it is out of the pit
- Only small percentage of production is lost when a single unit is down for maintenance instead of 100% when the conveyors are not running

The advantages of conveyors over trucks are:

- Lower total cost for long horizontal and vertical haul distances
- Can be designed for life of pit
- Less sensitive to inflation - no replacement
- Can convey material at steep angles
- Less labor intensive than trucks
- Can be easily automated
- Energy efficient
- Less hydrocarbon emissions than trucks
- Has lower noise levels
- Belt conveyors are normally more durable and have less maintenance requirements than trucks

5.1.2 Belt Conveyors Versus Hoists

In Section 3 there is a discussion of belt conveyors and hoists, which summarizes their merits for elevating material from underground.

Advantages of belt conveyors over skip hoists are:

- They can have higher production rates
- Total costs lower at shallower depths
- Generally lower operating and maintenance costs
- Safer

Advantages of vertical skip hoisting:

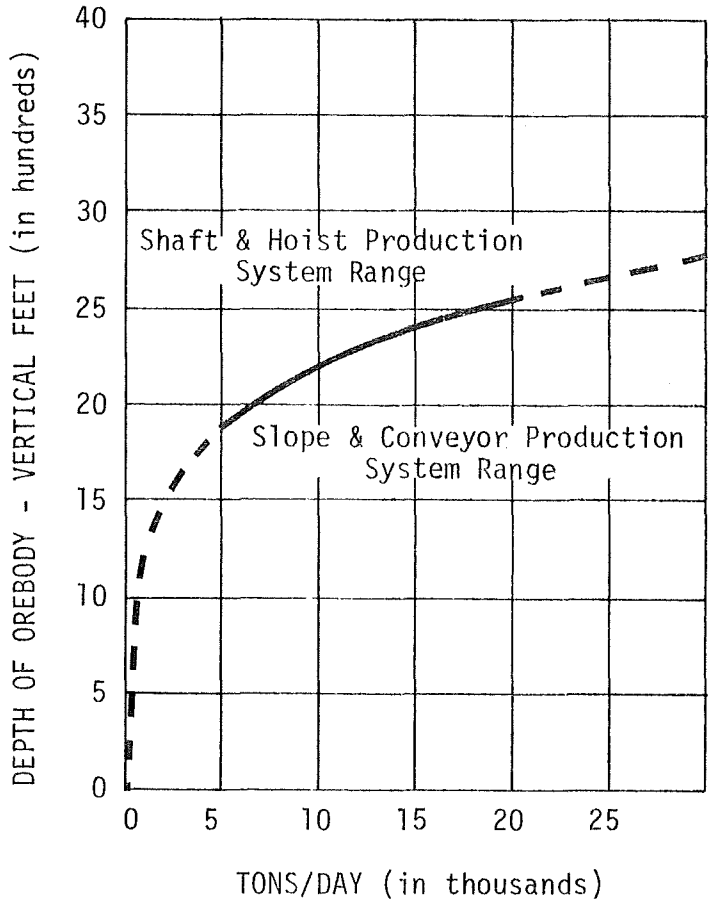
- The shaft can be located nearer the pit
- Capital costs lower at greater depths (Figure 5-50)

5.1.3 Belt Conveyors Versus Rail Haulage

The advantages and disadvantages of belt conveyors and rail haulage will be about the same whether they are located on the surface or underground.

Generally rail haulage is cheaper over long hauls, (excess of 10 miles), and with low vertical grades (1 - 3% max.). The rock does not have to be crushed as fine before transporting

MINIMUM CAPITAL COST RANGE FOR PRODUCTION SYSTEM ESTIMATING*



*Calculations based on 20 ft. dia. shaft and 240 ft^2 slope @ -15°

Figure 5-50 Comparison - Skip Hoists Vs. Belt Conveyors Production Range - Depth Vs. Production (Ref. 5.1)

to the processing plant. Rail has been used successfully in many underground mine haul applications, though the present trend now is away from its use in underground haulage.

Belt conveyor's big advantage is that it is more economical for large, short conveying distances. Belts can also convey at steep angles, thus decreasing the total length required underground. Generally the maintenance costs underground are higher than for belts on the surface.

A detail study of rail haulage was contractually excluded from this study, though it can be a viable method with the proper geometric relationship, such as the iron mine in Canada.

5.2 Inclined Shaft (Slope) Versus Sheltered Conveyor Slot

Another element that must be compared in the study on continuous underground haulage is the advantages and disadvantages of inclined shafts versus sheltered surface conveyor slots. This comparison is not necessary for most underground haulage methods, but is applicable to contrasting surface to underground conveyors and therefore must be discussed. The most important consideration is the excavation costs, which can outweigh all the other considerations in comparing the two methods:

Inclined shaft for hoisting ore advantages:

- Protected from weather
- If placed properly, will not need to be moved
- If conveyor is used in place of hoist, it can be suspended from the roof

Inclined shaft for hoisting ore disadvantages:

- Ground water could be a problem
- Difficult and costly to move
- High rope wear if skips are used
- Crushing may be required
- Dust in the confined areas underground
- Limited number of loading points
- Ventilation requirements

Sheltered surface conveyor slot advantages:

- Possible lower initial capital investment
- Can be loaded at several points
- Major components can be replaced easily and lower maintenance

Sheltered surface conveyor slot disadvantages:

- Might have to be moved as pit expands
- Stability and alignment problems if pit walls have movement
- High rope wear if skips are used
- Crushing may be required
- Climatic conditions

5.3 Continuous Underground Haulage Advantages

Section 3 presented a detailed description of the parameters which combine to make an underground continuous haulage system a viable haulage alternative. The utilization of underground systems is an advantage when:

- Laterally constricted orebodies can achieve a more favorable stripping ratio, achieving a greater pit depth by reducing bench widths (no haul roads)
- The haulage out of the pit would have extremely adverse truck profiles or be too steep for conveyors or hamper pit expansion.
- Errors in orebody delineation prevent expansion of the pit to accommodate surface haulage.
- The topography between the pit and the plant adversely affect surface haulage such as making haul roads prohibitively long or slopes excessively steep or lifts too great for belts and slurry lines.
- Environmental considerations such as degradation of scenic views, haul road dust, population centers and noise rule out surface haulage.
- The plans for developing the orebody include underground extraction in the foreseeable future.

5.4 Limiting Factors For Underground Haulage Open-Pit Ores

Even when underground haulage systems are the most

economic alternative, their inherent limitations and disadvantages must be recognized. This section identified the following as adversely affecting the viability of continuous underground haulage:

- Initial capital costs for underground haulage systems is higher than a comparative surface system because of excavation costs and greater difficulty in installation.
- The rock strengths (rock mechanics) of underground chamber and tunnel design are not easily projected so that estimated excavation and maintenance costs are subject to greater risks.
- Due to restricted access, maintenance cost for underground equipment and structures is greater.
- Once an underground haulage system is installed major design changes are difficult, costly and often require suspended operations. Thus "cradle-to-grave" mine design is more critical to underground installations. The overall flexibility of the operation is decreased with permanent underground facilities.
- It is difficult to haul more than one product from single ore pass. Therefore, ore grade and type segregation or ore/waste segregation often requires multiple ore/waste passes and more complex underground haulage systems.
- The more continuous the underground haulage system (i.e. conveyors), the greater the impact of haulage failures on pit production.
- Mine-life underground installation faces greater risk of eventual technological obsolescence and is less flexible to changes in capacity requirements.

5.4.1 Health and Safety Consideration

An important consideration with any mining system is health and safety. The accident rate of any underground mine is always higher than surface installation, and this will vary according to the mining method being used underground. The statistics show the greatest number of accidents occur in the mining areas where the rock was recently blasted.

It is believed that the accident rate with a crusher facility on the surface and the one underground is minimal since

the underground men are working in protected areas.

The noise in underground facilities is normally above the approved decibel limits, so the operators must be in isolated control rooms while the crushers are operating.

Dust is another problem with underground crushers, but it can be controlled by different types of collectors and water. Dust control is required for health and safety reasons. Besides, with the lower dust level, the maintenance costs will be lower.

Any crushing facility must have dual access. The ore pass cannot be considered the secondary escape way, but a conveyor slope could be. Besides access for personnel, the larger crusher parts must be brought underground at an added cost over a surface facility. All entries must consider the problem of material and man movement.

In non-gaseous mines, a single conveyor slope is permissable, provided the normal conveyor safety features are included. Though it isn't a MSHA requirement, a fire-resistant belt would be a very good idea. The ventilation in the non-gaseous conveyor slope can be in either direction.

5.5 Risk Areas

As with any mining method, all of the risk areas must be considered, and weighed against the positve points. Previous sections elaborate in great detail all the advantages of underground haulage from open-pits. The following are a few risk areas that could affect the proposed system:

- Geological conditions different than anticipated such as soft or harder rocks, excessive water, heavy ground, change in reserves, etc.
- Equipment technology change
- Change in company policies, ROI's requirements, management changes, etc.
- Governmental regulations
- Labor problems
- Price and product demand changes in product provided

5.6 Economic Comparison

In the previous sections of this study, State-Of-The-Art, Elements Of Continuous Underground Haulage and U. S. Mine Managers' Critique, it was found that because of the many variables involved within any haulage method it would be impractical to attempt a general economic comparison covering all the possible variables.

However, it was determined that a basic consideration, (the cost to elevate the rock out of the pit), could be evaluated under some hypothetical conditions. The cost elements for loading horizontal haulage in the pit and crushing were not considered since these cost elements should be common to all methods of haulage. Haulage considerations outside the pit limits were not addressed. Items such as climate and environmental impact sometimes do have major impact upon the comparative viability of haulage systems.

A hypothetical application of truck haul, underground slope conveyors and vertical skip hoisting at a specific vertical distance and capacity is evaluated in the following subsection (5.6.1). This hypothetical comparison serves only one purpose, and that is to demonstrate a method by which the comparison costs may be calculated. The relative merits of these three haulage systems with regard to capacity and lift have long been established:

- Shallow lifts and low to intermediate capacities favor truck haulage.
- Intermediate lifts and intermediate to high capacities favor slope conveyors.
- Lifts from greater depths and low to intermediate capacities favor vertical hoisting.

Any numerical cutoff of lift depth or capacity is site specific, and to calculate those cutoff depths and capacities for a hypothetical model would not generate representative values.

5.6.1 Haulage Cost Comparison At 1600 ft Of Vertical Lift

Because of the many variables involved in selecting a haulage method it would be impractical to recommend any single haulage system for all open-pit mining operations. However, a hypothetical haulage model was developed to indicate the possible economic advantages of underground haulage methods within a well defined application.

The basis of this haulage comparison is the cost to elevate the rock out of the pit. The operating and capital costs for ore passes, haul road construction and road maintenance are minor and so were not included.

Specifically, access roads into the pit and their maintenance are required with surface and underground haulage systems. The costs associated with properly designed and safe access roads are not appreciably different from haulage roads in as much as the same equipment would utilize both.

In regard to ore passes the lift requirement of all the haulage systems at 1600 vertical feet is such that the ore passes are of minimal lengths and as a consequence a very minor capital consideration.

Conventional surface truck haulage, inclined conveyor and vertical skip hoists were evaluated based upon the following parameters:

- Haulage rate 75,000 tpd
- Hourly rate 3,600 tph
- Vertical lift 1,600 ft

The comparison between the skip hoists, belt conveyors and trucks are listed in the following areas:

	Skip Hoists	Belt Conveyors	Trucks
Horsepower/hours/ton	2.70	2.05	6.48
Manpower/day	8	4	24
System Requirement	2-hoist	60" belt	16 trucks
Grade	Vertical	18° slope	7.5%
Distance	1600 ft.	5178 ft.	21,400 ft.

The summary of haulage costs are based upon a vertical lift only to obtain comparable costs. The following three methods would all need primary crushing somewhere in the system, and the horizontal distance hauled on the bottom of the pit would require the same number of trucks hauling to in-pit crushers or ore passes, or with trucks out of the pit. Thus, horizontal pit bottom haul distance costs would remain the same.

SUMMARY - VERTICAL HAULAGE COSTS

	Capital Cost	Capital & Operating Costs/Ton	Manpower Required
Incline Underground Belt Conveyors	$6,460,000	$ 0.07	6
Vertical Shaft Skip Hoist	12,000,000	0.126*	12
Incline Truck Haulage	13,000,000	0.3835	24

* (excludes horizontal transport cost from ore pass to vertical shaft)

As can be seen with these parameters, belt conveyors have the lowest cost for hoisting 75,000 tpd.

The truck system was based upon using 170 ton trucks hauling an 8% grade. Sixteen trucks would be required for hauling 247 tons/hr., with an ownership cost of $25/hr., and operating cost of $70/hr. Spare trucks were included in the capital estimate to compensate for a 75% availability factor.

The slope costs were estimated on a 12 ft. x 20 ft. opening on an 18° slope. The estimated cost for driving this slope is $2,400 per vertical foot. The 60 in. belt cost was estimated to be $506 per linear foot.

The hoisting costs were estimated on a 28 ft. diameter-vertical shaft at a cost of $1,650 per vertical foot. The capital costs include money for excavating, concrete, tower, electrical and mechanical equipment, but excludes horizontal conveyance or excavation costs from the ore pass to the hoisting shafts.

The power costs were based upon 2¢/kw hr. The labor costs were based upon $20,000/yr. per employee.

Detail cost calculations are included in Appendix F.

6. CONCLUSIONS AND RECOMMENDATIONS

The first and second phases of this contract study gathered and evaluated data on the current and past continuous haulage systems while giving special consideration to evaluating continuous underground haulage systems. The third phase requested industry comments concerning the current U. S. operating philosophy and suggestions regarding continuous haulage again emphasizing underground systems. From the first three phases conclusions have been drawn with respect to the relative merits of haulage systems and recommendations prepared. These conclusions have precluded the recommendation of any single haulage system or class of systems (i.e. non-continuous or continuous underground) for open-pit mining due to the large number of variables found in ore deposits and overall mine plant requirements. The following variables all affect the haulage system selection:

- Geometric relationship
- Production requirement - large, medium or small
- Location of deposit - arctic, desert, mountains, flat lands, etc.
- Type of minerals - copper, lead, zinc, iron, limestone, etc.
- Ore values and market fluctuation

6.1 Conclusions

The State-of-the Art study established the viable uses of continuous underground haulage systems in both past and present applications. These systems have proven their technical and economic feasibility under the site-specific conditions of each property where applied.

The principle conclusion determined from the State-of-the-Art study and evaluation of continuous haulage elements was that an evaluation of whether an underground or surface haulage method should be used for an open-pit depends to a great extent

upon the geometric relationships of the ore deposit.

The geometric relationship in this report is defined as including the following key parameters:

- Nature of the orebody
- Local topography
- Climate
- Proposed mine plant layout

The geometric relationships that could favor underground haulage from open-pits are:

- Laterally constricted orebodies which can achieve a more favorable ore/waste ratio by reducing bench width (no haul roads) or achieve a greater pit depth to extend the surface mining life.
- That the plans for developing the orebody include underground extraction in the foreseeable future.
- Haulage out of the pit would have extremely adverse profiles or be too steep for conveyors, or that surface skips or conveyors would hamper pit expansion.
- Errors in orebody delineation prevent expansion of the pit to accommodate surface haulage.
- The topography between the pit and the plant adversely affect surface haulage making haul roads prohibitively long or slopes excessively steep, or lifts too great for belts or slurry lines.
- Environmental considerations such as degradation of scenic views, haul road dust, population centers, noise, extreme elevations, and arctic climates, rule out surface haulage.

The State-of-the-Art study provided the following observations on underground continuous haulage:

- Large tonnage can be handled from beneath open-pits using underground haulage existing capacities in excess of 100,000 tons/day.
- Large tonnage underground rail haulage systems can be automated.
- Vertical hoisting of ore was more economical than the use of inclined conveyors in some mines; that is, mines that have been in operation for many years and invested in the facilities for skip hoisting when this was the predominant mode for vertical haulage.

- Several products can be handled by underground haulage methods.
- The ore passes can be used for storage and surge capacities.
- Underground haulage can be used for both surface and underground portions of the mine.
- The pit slopes can be steeper where haul roads are eliminated, thus decreasing the stripping ratio or improving the extraction ratio of ore to waste.
- The old surface ore passes can be used for transfer of backfill material underground after the completion of the open-pit.
- The ore passes can be vertical, inclined, or located underground with trucks hauling to ore passes.
- The surface and underground mine sections can be worked at the same time using the same continuous systems.
- Hoisting systems, belt conveyors, and rail all can be viable haulage methods from underground.
- Continuous underground haulage from underground can reduce the horizontal and vertical truck haul.
- Underground haulage for short life quarries can be an economical method.
- The primary crushers for continuous haulage can be located on the surface or underground.
- Truck haulage versus belts or skip hoisting.
  - more sensitive to inflation
  - costs increase exponentially with increase lift height
  - dependent on a larger skilled maintenance force
  - less efficient energy users
  - more sensitive to diesel fuel allocation
  - long up grade hauls adversely affect truck availability
  - higher hydrocarbon emissions

Ore/waste passes are the one common element of all continuous underground haulage systems. As is shown in the State-of-the-Art and Mining Industry Acceptance Tasks, it is the least understood but one of the most critical elements.

The risks in ore passes can be high especially when all design variables are not considered properly.

Important design considerations are location, size, depth, angle, storage capacity, loading and discharging areas, maintenance considerations, and economics. The design should also include the study of various methods for ore pass development. The correct method would greatly improve the maintenance and economic considerations.

The U. S. Mine Managers' Critique Section of this report included several negative opinions regarding continuous underground haulage. The following negative conclusions came from several operators who had discarded continuous haulage as a viable method for their properties:

- That selective mining of multiple ores and waste was incompatible with continuous haulage systems.
- Investment analysis had shown that where a large truck fleet was required for stripping the trucks were most efficiently utilized by handling ore and waste.
- Underground continuous haulage seriously impairs the flexibility to adapt to changing pit configurations and market fluctuation.
- Limitations on pit slope angle due to general ground stability prevented any realization of improved ore/waste ratio for underground haulage.
- That the high capital costs and inherent risks of "all the eggs-in-one-basket" for underground haulage offset potential advantages.
- That adequate information for construction and upkeep of ore passes is unavailable for accessing costs.
- That the funds for the sizable initial investment for underground systems were not available.

These conclusions came out of studies conducted in the 1960's and early 1970's; and, it is important to note that recent events have created renewed interest in underground haulage. This interest was reported to stem from:

- Concern that the large size trucks (350-ton) will impose severe limitation on the flexibility of operation and be mismatched to present pit equipment.

- Concern regarding the impact that escalating fuel costs and inflation are having on haulage costs.
- Fear of fuel shortages.
- An increased emphasis on environmental considerations such as air pollution and noise (hydrocarbon emissions, dust, noise levels, etc.).

It is also important to note that existing underground haulage systems have shown many of the negative opinions to be in error or simply peculiar to the property for which the systems were evaluated.

Section 5 of this report, Comparison of Haulage Systems, details the observed advantages and disadvantages of non-continuous surface haulage, continuous surface haulage and continuous underground haulage. Since this research study was restricted by contract to investigate continuous underground haulage, engineering/economic evaluations comparing surface to underground systems fell outside the scope of work.

However, many investigations have been conducted in that area and conclusions have been drawn. It is generally accepted in the mining industry that:

- Shallow lifts and low to intermediate capacities as in the early years of most open-pit operations favor truck haulage or some other non-continuous surface haulage.
- Intermediate lifts, from approximate pit depths of 500-800 vertical ft., and intermediate to high capacities, say to 40,000 tph, favor conveyor installations.
- Lifts from greater depths, say in excess of 2,000 vertical ft., and low to intermediate capacities favor vertical skip hoisting.

Of course, any numerical cut off in capacity of the lift is site specific. For example, the haulage cost comparison at 1600 ft. of vertical lift (subsection 5.6.1) is a hypothetical haulage model based only on vertical lift out of the pit bottom at 75,000 tons per day. This comparison shows an inclined underground slope conveyor to have superior economics to vertical skip hoisting or trucks.

Obviously had a shallow lift or much lower capacity been the criteria, the alternatives would rank differently.

As previously noted, this study has concluded underground continuous haulage systems to be technically viable where the proper geometric relationships exist. The applications of underground continuous haulage now in operation have demonstrated not only system viability; but, that within the economic and physical constraints of the properties, these underground haul systems are the optimum (or most economic) haulage method. The evaluation process which established the optimum haulage system goes beyond the technical viability of the system. Thus a distinction between technical viability and economic viability must be made.

This study therefore concludes that the economic viability of underground haulage is:

- Site-specific
- Dependent upon mineral commodity price fluctuations effect upon mine capacity and operating flexability requirements
- Dependent upon the overall mine costs and mine planning-anticipated mine life
- Critically influenced by the handling of escalation and inflation rates on all alternative haulage systems
- Critically influenced by the assessment of technologic trends in all alternative haulage systems
- Dependent upon the existence of favorable geometric relationships.

6.2 Recommendations

Returning briefly to a review of this study's conclusion, it was found that:

- There are definite geometric relationships which favor continuous underground haulage from beneath an open-pit.

- Where rock properties allow underground tunneling, ore passes and rock chambers at reasonable cost, the time value of money for the front-end capital invested can be favorably offset by the reduced operating and maintenance costs on continuous underground haulage systems.
- Environmental and climatic considerations can override all other criteria depending on properties location.
- There is a distinct difference between risk as defined as acknowledging the limits of engineering estimates and risk which is simple lack of knowledge of the alternative methods.
- The ultimate decision to utilize continuous underground haulage must usually be based on economic evaluations which incorporate all known physical requirements, technologic trends, detailed engineering design, the time value of money, inflation and escalation rates and risk assessments not biased by lack of knowledge.

For the many reasons previously mentioned, this study cannot recommend flatly that underground continuous haulage be applied in all open-pit mines; nor can this study present any single haulage system as being the most viable method of haulage based on pit depth or orebody configuration or any other single design parameter.

This study can recommend continuous underground haulage systems be evaluated for all open-pit mines. If the geometric relationships (such as pit depth, topography, distance from pit to mill, and elevation differential) are favorable or marginally favorable, underground haulage should be considered. Other considerations such as extreme elevation, adverse climate, the proximity of population centers and environmental constraints must enter into the in-depth engineering and economic analysis which ultimately determines the optimum haulage system. Every effort must be made to "fit" the haulage system to the orebody and overall mine plan, not to make the mine "fit" preconceived notions of what the haulage system should be. It is important to caution haulage planners that a lack of experience or knowledge about underground excavation design and costs is not "risk." These can be as successfully estimated as any other open-pit cost element with true risk quantified on the basis of availability of adequate design criteria.

This study provides extensive material on the application, engineering criteria, limitations, capacity, technologic trends and design risks of the elements which make up continuous underground haulage systems. This information and the evaluation procedures employed should be applied in assessing the viability of continuous underground haulage for specific properties.

In preparing this study the major obstacle encountered was the refusal of operating companies to provide proprietary information on the operating costs of haulage equipment. It was soon realized that if a hypothetical orebody and mine plant design was utilized to evaluate the economic viability continuous haulage, the very nature of the assumptions would load the results in favor of specific systems. Therefore, it is recommended that one, and preferably three real orebodies with or without existing operating mines be selected and used as a basis for an in-depth economic evaluation of haulage systems. If more than one site is evaluated, the sites should not be similar in climate, topography, and orebody characteristic while still being acceptable candidates for underground continuous haulage.

An area of concern in which additional research outside the scope of study for this report, was identified in Section 4 U. S. Mine Managers' Critique. Little design information appears to be available for estimating or evaluating the installation of ore passes. Ore passes and the behavior of materials in and through them has traditionally been an area of concern for underground mines. These mines have generally established the design criteria empirically, through experience, such that the published data is quite site-specific. Thus, it is recommended that research be undertaken to establish estimating and design procedures for ore passes where only one or a very few will be installed as in the case of below pit haulage systems.

Additional recommendations for research were presented by mining personnel during the U. S. Mine Managers' interviews.

Recommended was research to provide:

- More definitive performance data (both technical and cost), especially on foreign mining operations.

- A reliable method for tagging different ore grades and waste on the belt conveyors.

- Fail safe installation procedures for securing conveyor systems to the pit wall. Monitoring equipment to check for possible movement.

- Cost and performance comparisons between:
  - Conveyors anchored to the pit wall
  - Open trench cuts from pit bottom to the surface for a conveyor system
  - Underground inclined conveyors

- Site specific information on conveyor mobility for relocation within the pit.

- Research on:
  - Steeper angle belts
  - High speed transfer points
  - More reliable components such as idler and belts
  - Operating costs for low and high-speed belts
  - Faster conveyor belt speeds (from 800-1200ft/min.)

- Portable crusher systems/Mobile crushers.

- Advantages and disadvantages of underground crusher chambers.

- Engineering design and cost criteria for chambers specific to underground crusher chambers.

APPENDIX A

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International Mining Consultants, Inc. wishes also to thank the following mining companies and their management for assistance during our State-of-the-Art study, and U. S. Mine Manager's Critique tasks. Without their considerable cooperation this research study would not be possible. The foreign mining companies are listed on the following page including the individuals contacted and their addresses.

United States Companies

AMAX, Inc., Climax Mine, Colorado
American Smelting and Refining Co., Sacaton Mine, Arizona
Anaconda Corp, Berkeley Pit, Montana
Anamax Mining Company, Twin Buttes Mine, Arizona
Cleveland Cliffs Corp, Empire Mine, Michigan
Dravo Corp., Dworshak, Dan & Quarry, Idaho
Duval Corp., Sierrita Mine, Arizona
Kaiser Steel Corp., Eagle Mountain Mine, California
Martin-Marietta Aggregates, North Carolina
Molycorp, Questa Mine, New Mexico
Newmount Mining Corp, Carlin Mine, Nevada

MINE MANAGERS NAME AND MINE MAILING ADDRESS

MINE REPORT	CONTACTS	MINE MAILING ADDRESS
Kiruna Mine SWEDEN	Mr. Kjell Lidin-General Manager Mr. Knut Sorensen-Director Long Range Planning Mr. Lars Hermansson-Mining Engineer	Kiruna Division Luossavaara-Kirunavaara AB S-981-01 Kiruna SWEDEN
Pyhasalmi Mine FINLAND	Mr. Eero Erkkila-Mine Manager Mr. Tapani Kilponen-Mine Superintendent	Outokumpu Oy Pyhasalmi Mine 86900 Pyhakumpu FINLAND
Cananea Mine MEXICO	Ing. J. Ruben Velasco-Secretario General Mr. Edward Foller-Mill Supperintendent Mr. Ricardo Cornejo-Asst. Mill Superintendent Mr. Alvin J. Fenn-Superintendent del Dento De Precipitatacion	Compania Minera De Cananea, S.A. Cananea, Sonora, MEXICO
Nordcement Hannover's Alemannia Works WEST GERMANY	Plant Visit Mr. Erich Mulisch-Director Mr. Karl Heinz Eliax-Project Engineer Mr. Dieter Pfeilschifter-Engineer Mine Visit Mr. Eckhard Frhr. VonDerHorst - Krupp Sales Mr. Harold Koymode-Foreman	Freid, Krupp GMBH Krupp Industries-Und Stahlbau 4100 Duisburg 14, GERMANY Nordcement Alemannia Works Hannover, WEST GERMANY
Svappavaara Mine SWEDEN	Mr. Benzt Aaro-Mine Manager	Kiruna Division Luossavaara-Kirunavaara AB S-981-01 Kiruna SWEDEN

APPENDIX B

STATE-OF-THE-ART

B.2.1 The Kiruna Mine--Sweden

Kiruna stands out not only as a model of modern technological innovation in underground hardrock mining, but perhaps more as the best example of the complexities confronting mine management in timing the switch over from open-pit to underground operations. Following the decision to go underground in 1937, development progressed to the point where by 1954 the majority of ore production could be obtained underground. However, a marked increase in world ore prices at that time influenced management to keep the pit active for an additional 17 years. Moreover, had management forseen the technological improvements to come in the 1960's with large scale equipment, they might have postponed this decision indefinitely. Nonetheless, the Swedes have persevered to create one of the largest and most productive underground operations in the world, with tonnage output that rivals most open-pit mines.

The changeover from open-pit to underground mining was certainly one of the most important decisions in Kiruna's history. Determining how to time the transition required not only a long-term view of the forthcoming technical and economic developments, but perhaps even more a sense of the changing perspective in employee relations and environmental preservation. Kiruna may have unwittingly averted much of the contemporary environmentalist activity by going underground and, for this reason, it is important to review their current analysis of this far-reaching decision.

The following sections describe in detail the mining and ore haulage operations, both past and present, and conclude with a comparative evaluation by mine management on the pros and cons of underground versus surface activity. Figure B-1 gives a panoramic view of the iron ore mountain, while Table B-1 lists the principal characteristics of present mining and ore haulage activity.



Figure B-1 Panoramic View of the Iron-Ore Mountain at Kiruna Looking Along the Length of the Pit

© LKAB - Sweden, September, 1978

TABLE B-1

FACT SHEET ON THE KIRUNA MINE

- Owner - Luossavaara-Kirunavaara AB (LKAB), a Swedish Government-owned company
- Location - In northern Sweden 90 miles north of the Arctic Circle, 110 miles southeast of Narvik, Norway
- Ore - Iron Ore
- Pit Operations Started - 1892
- Underground Operations Started - 1938
- Pit Operations Ceased - 1971
- Current Production Rate - 18,000,000 tpy
- Full Production Potential - 30,000,000 tpy or 130,000 tpd
- Potential Mine Life - 50 to 100 year
- Operating Personnel - 1,200 to 1,500 underground
- Equipment:
  - LHDs tend raises/chutes and feed ore into a rail system ahead of the crusher circuit.
  - Crushed ore is hoisted to the surface by vertical skips.
- Operations:
  - Largest underground mine in the world. Highly automated operation. Planning effort under way for hoisting methods.
  - Open-pit mining fed the present underground system until 1971.
- Problems:
  - Despite good productivity, operation is still losing money due to depressed world market for iron ore.

B.2.1.1 General Mine History and Orebody Characteristics

The Kiruna Mine is situated in northernmost Sweden, 90 miles north of the Arctic Circle, and is principally owned and operated by the Swedish government's mining and rail conglomerate, LKAB. The orebody is one of the world's largest deposits, exceeding an estimated 1.8 billion tons, and contains iron-ore of varying concentrations interbedded with low and high phosphorous as listed below:

Ore	Iron Content	Phosphorous Content
A-ore	68%	max. 0.030%
B-ore	67%	max. 0.10%
C_1-ore	66%	max. 0.45%
C_2-ore	64%	max. 0.80%
D-ore	60%	max. 0.80%

As shown in the plan view of Figure B-2 the Kiruna ore extends for almost 8 km as a series of slabs or lenses. The

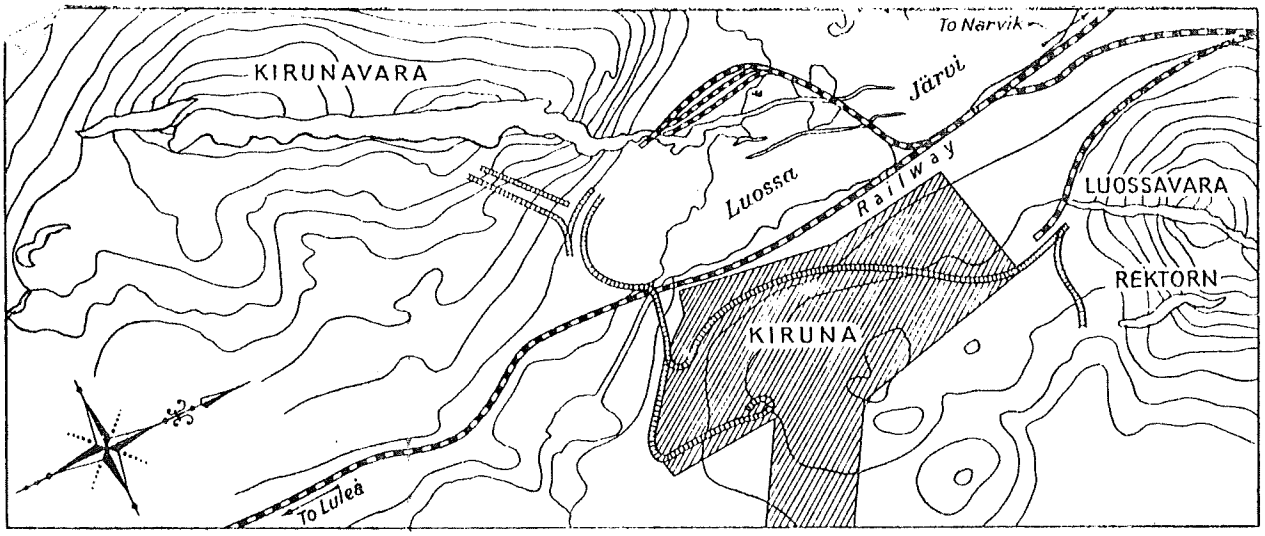


Figure B-2 Plan View of the Kiruna Orebody Overlaid on a Contour Elevation Map of the Region (Ref. D.1.1)
©Ashire Publishing Ltd., 1956

main orebody, which originally lay exposed along the crest of the Kirunavaara ridge, continues underground as two separate lenses: one in a southerly direction and the other northwards for a km, reappearing as an isolated body at the top of Luossavaara. A parallel deposit to the latter is the Rektorn ore with extremely high phosphorus content. The main Kiruna deposit is a huge tabular orebody striking nearly north-south over a 5-km length, averaging up to 100 m in thickness, and dipping steeply to the east at 40° to 75°. Exploration drilling has indicated that the ore extends to a depth of at least 1,800 m. Down to 1,000 m, the orebody shown no tendency to diminish in length, but there is a slight decrease in width. The ore area totals 3.5 million sq ft, giving 1.4 million tons of ore per vertical yard *(Ref. D.1.2)*.

The ore mineral is magnetite and is generally dense or very finely crystalline. To a small extent--chiefly in the southern part of Kirunavaara--it is mixed with hematite where the ore is particularly porous and infiltrated to a degree with quartz material. Apatite is the most important gangue mineral from a quantitative as well as an economic viewpoint.

Since mining operations started in Kiruna in 1898, over 500 million tons of ore and 350 million tons of rock have been mined. After the decision to move underground was made in 1939, development work took 10 to 15 years before any significant production was achieved. The chart in Figure B-3 shows how dramatically underground mining escalated between 1950 and 1970 until stabilizing at the current level of 15 to 18 million tpy. Actual design capacity of the currently active 540-m level is 25 million mtpy; however, annual production from the level has never topped 22 million mt and more has been in the range of 15 to 18 million mtpy.

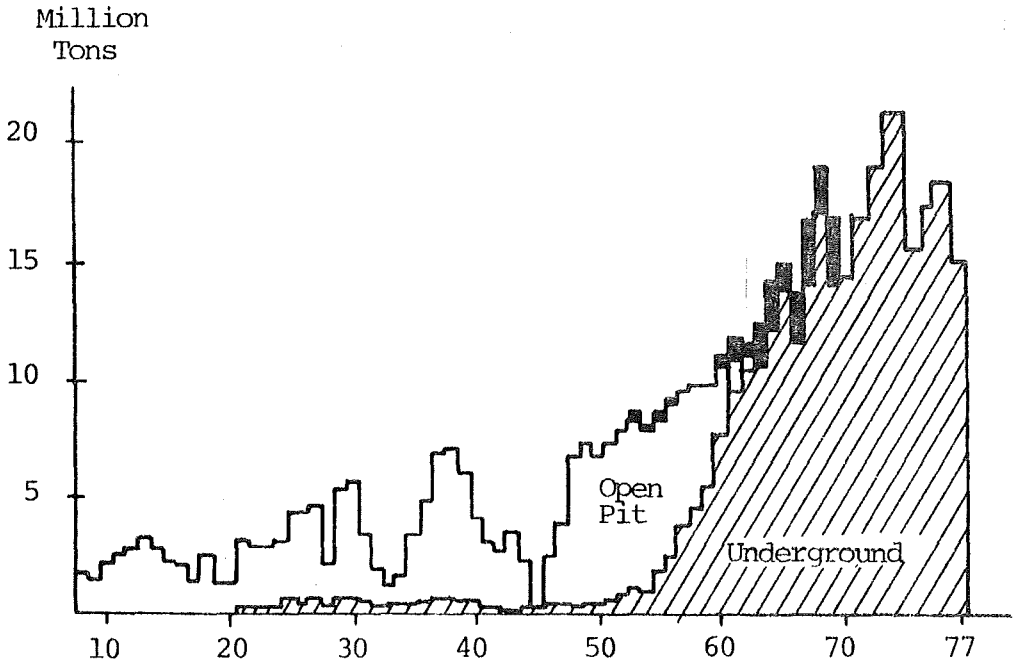


Figure B-3 Ore Production at Kiruna Over the Past 70 Years (Ref. D.1.3) © Mining Congress Journal, 1972

At present, mine management is rounding out construction of a new highly automated haulage level at a depth of 775 m, scheduled to come on line in late 1978. Design capacity for the 775-m level, which will take several years to reach full production, is 30 million mtpy. Unfortunately it has not been a propitious time to bring expanded capacity on stream. In 1977, LKAB lost more than $100 million, and by year-end held more than 10 million tons of iron ore in stockpiles at Kiruna and four million tons elsewhere.

B.2.1.2 Mine Layout

Excavation of the Kiruna ore started over 80 years ago in the form of open-pit mining. This continued through the 1950's, when production from underground mining started up and lasted until final phasing out in 1971. (Figure B-4 shows a

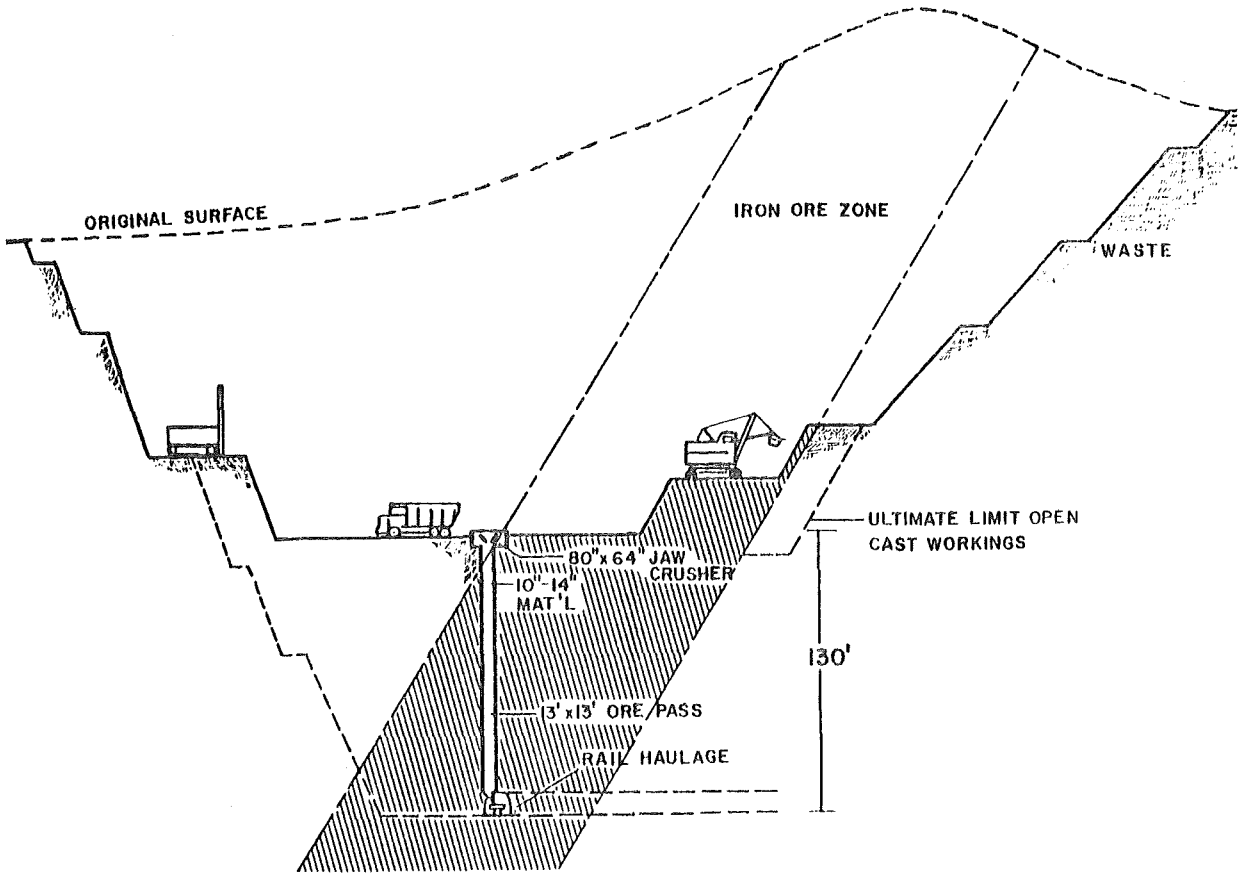


Figure B-4 Schematic of Pit Operations Feeding the Underground Ore Haulage System (Ref. D.1.3) © Mining Congress Journal, 1972

schematic of pit operation with concurrent underground ore haulage.) Operations were conducted on benches having a height of 15 m and in certain exceptional cases as much as 30 m. The length of the finished open-pit ended up being more than 3 km, the width measured .5 km, and the depth reached about 350 m.

In the mid 1950's, with the waste-to-ore ratio in the pit at 1:1, an economic calculation (based on the use of recently developed larger scale pit equipment) showed that the optimal ratio had increased to 3:1. Practically this would have meant moving the dividing line for underground 150 to 200 m deeper. Consequently, it was decided that the most northerly part of the orebody would continue as an open-pit, while the remainder went underground. Figure B-5 shows an elevational cross-section of the final dividing line between pit and underground.

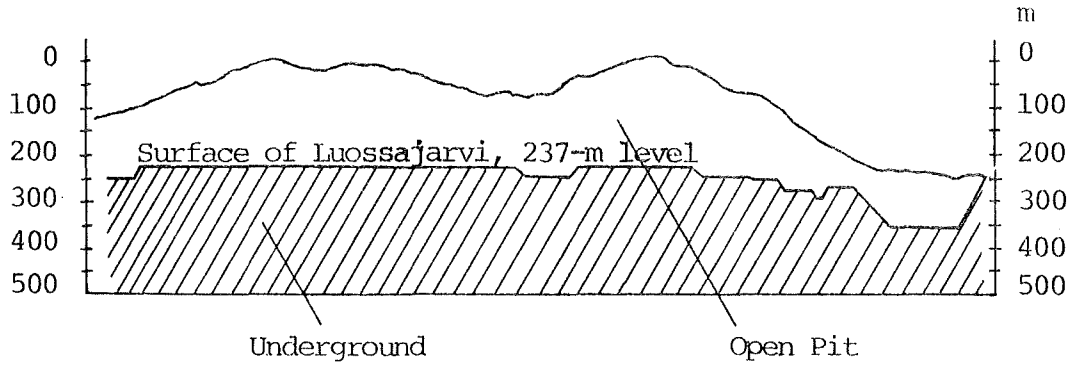


Figure B-5 *Longitudinal Cross-Section through Kirunavaara Iron Ore Mountain Showing the Elevation of the Underground Interface (Ref. D.1.3)* © *Mining Congress Journal, 1972*

As early as 1930, some underground mining was done (see Figure B-6) in order to obtain A-ore. The mining method used at that time was shrinkage stoping, and thus no selective mining could be practiced. Sub-level caving has since replaced shrinkage since it permits selective mining of the different ore qualities for the same mining cost. Several inclines in the footwall provide access to the workings for men and material, with buses transporting personnel to underground work sites. The ramp is surfaced with a hard-topped two-lane roadway that grades from 8 to 10%.

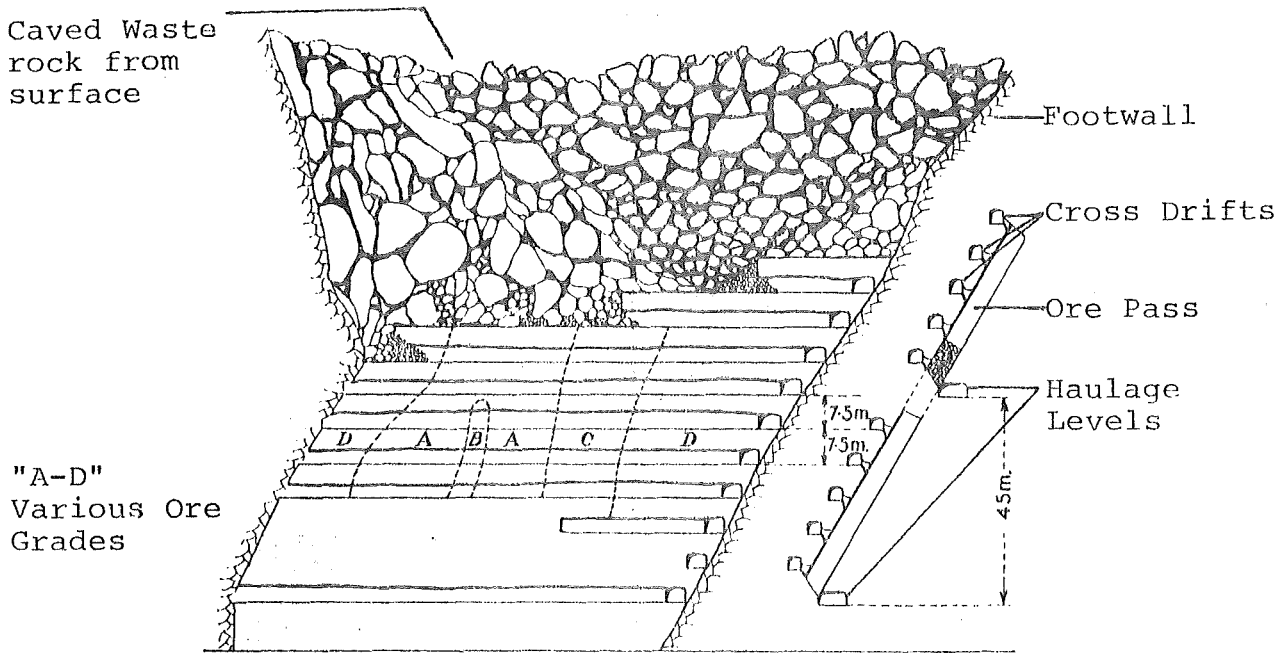


Figure B-6 *A Cross-Section of the Earliest Underground Workings for Obtaining A-Ore (Ref. D.1.4)* © *Ashire Publishing Limited, 1956*

The orebody proper has been divided into blocks between the main levels spaced almost 75 to 80 m apart vertically in the new level. Mining proceeds downward by means of sublevel "slices" developed at 11-m to 12-m vertical intervals across the full length of the mine. Sublevels are developed first by driving horizontal drifts 5.5 m wide x 4.2 m high in the footwall parallel to the strike. From these drifts, cross cuts are driven through the ore on regular 11-m horizontal centers. Crosscuts, typically 5 m wide x 3.8 m high, are offset horizontally from one level to the next.

Chutes are spaced at 200- to 250-m intervals along the horizontal drift in the footwall to provide ore passes to the next main level. At the two main levels currently in operation (540-m and the soon to be completed 775-m) rail haulage

moves the ore to underground primary crusher stations. Feed hoisting systems located in 10 separate shafts then transport the ore to the surface for secondary crushing and processing. An eleventh shaft is used exclusively to hoist waste rock. Figure B-7 provides a schematic view of the overall operation.

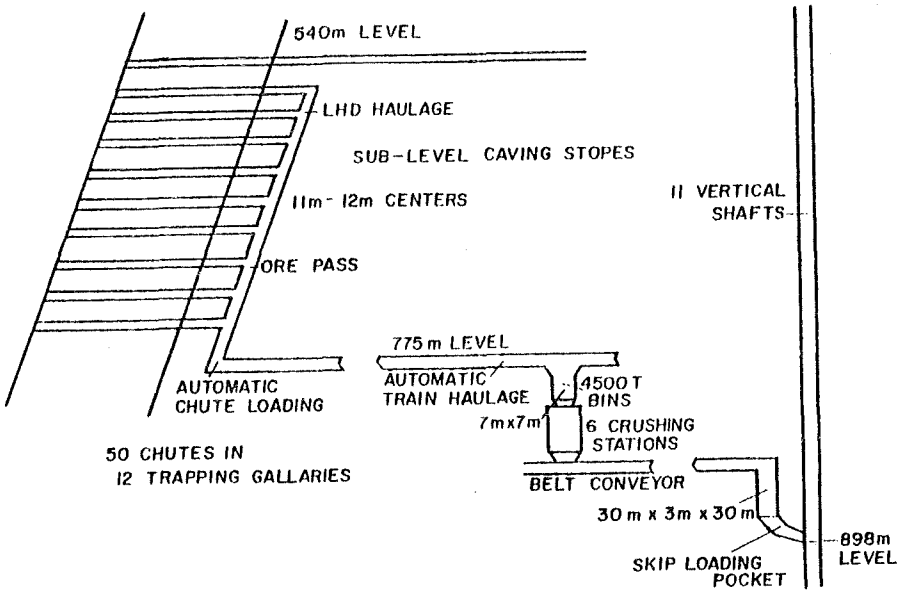


Figure B-7 Schematic showing the general layout of crusher stations, hoisting shafts and rail haulage level at Kiruna (Ref. D.1.5) © *EM/J, 1978*

The Kiruna headframe, which rises 50 m above the ore dressing plant, is about 130 m long, and 11 m wide, and houses 11 friction hoisting systems for the side-by-side shafts. The shafts measure about 9 x 9 m and have been sunk on 12.7-m centers. Skip loading takes place at a depth of 666 m for ore from the currently active 540-m main haulage level. The system is being deepened to allow loading at a depth of 898 m for ore from the new 775-m main haulage level.

B.2.1.3 Mining Operation

Sublevel caving has proven to be a very practical approach for Kiruna with its major requirement for selective mining. Numerous working faces are always in progress simultaneously which allows for large-scale mechanization in the fairly straight forward routine of development, fan drilling, and mucking out. Moreover, sublevel caving is used without exception throughout the mine, thereby affording standardization of equipment and procedures.

Ore haulage between working face and crushing station is accomplished by LHDs feeding ore passes which in turn empty into rail haulage cars. The new 775-m main haulage level is the fifth to be developed with each new level (275 m, 320 m, 420 m, and 540 m) entailing further evolution of haulage control. At the 775-m level, train operation is fully automated, including loading operations, and is controlled from a central room as shown in Figure B-8.

Trainloads of ore will be sampled automatically during loading at the tapping galleries at the 775-m level, and samples will be sent through pneumatic tubes for analysis while the train is en route to the crusher station, reducing the time required for a round trip. On the 540-m level, trains stop at a laboratory for sampling before being sent to a crusher station.

Extensive use of computer modeling in planning out the 775-m level helped determine the crushing/hoisting configuration and operating procedure. Six crushing stations were established along with individualized crushing bins, each having a volume of 4,500 mt to achieve planned capacity. Crusher stations were designed to allow each crusher to accept feed simultaneously from two trains. The conveyor systems from crushers to skips have been engineered to allow the integrated control system to direct ore from any crusher to any hoist. Skip loading

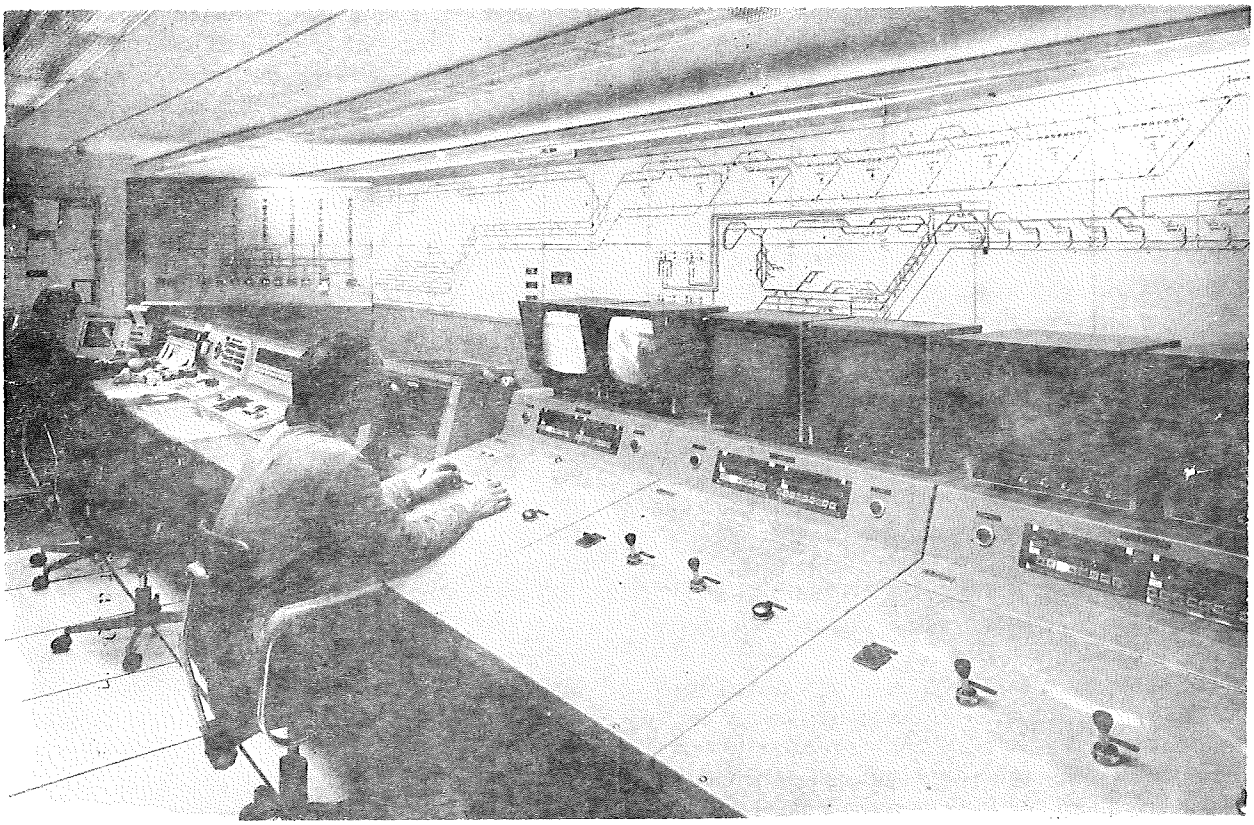


Figure B-8 Fully Automated Operator Control Desk for Track Haulage at the 540-m Level in the Kiruna Mine

and hoisting is fully automatic, with skips taking on 40-ton loads in 14 seconds. Load cells weigh the skips and control the skip loading operation, including stopping and starting the loading conveyor.

Because of the varying phosphorus content of the Kiruna ores, meticulous sampling is required to sort blasted ore in 9 types, which are blended in the course of transport

crushing to produce four ore grades for hoisting. Undiluted direct-shipping ore produced in the course of development accounts for about 20% of Kiruna production.

B.2.1.4 Mining Equipment

Kiruna utilizes rubber-tired diesel driven equipment in the drifts and crosscuts to achieve excavation and haulage. Currently there are 85 LHDs, 27 drifting rigs, and 18 fan drilling rigs in operation. Typically, 78 of the LHDs are in use at one time in ore drawing, while the remainder are used in other work.

A new ore wagon has been developed for the track haulage operation on the 775-m level which will increase train capacities without increasing train length. The new cars have only two axles instead of four, and they are shorter and deeper than cars previously used. On the 540-m level, 40-ton locomotives pull 12-car trains having a capacity of about 370 mt each. On the 775-m level, 65-ton locomotives will pull 18-car trains having a capacity of about 460 mt. Currently, 14 operatings trains service the 540-m level.

Six underground jaw crushers provide primary reduction, and feed hoisting systems for ore transport to the surface where secondary crushing is accomplished prior to grinding. Two gyratory crushers are also utilized underground in single stage crushing circuits and feed directly, via hoisting, into grind circuits at the surface beneficiation facility.

The massive hoisting operation designed to handle 30 million mtpy, takes place in 11 side-by-side shafts, 10 used for ore and one for waste. Automatic loading of the skip takes

place underground by conveyors running between the crushers and the hoisting shafts. A schematic of the skip loading arrangement is shown in Figure B-9.

The hoisting plant was modernized to handle expanding production from greater depths by installing new friction hoists. Shafts 1 through 8 are equipped with single skips and counterweights, while the 9 and 10 shafts have two skips in balance. Hoists 1 through 8 are designed for a net load capacity of 40 mt, winding at a maximum speed of 17 m per second.

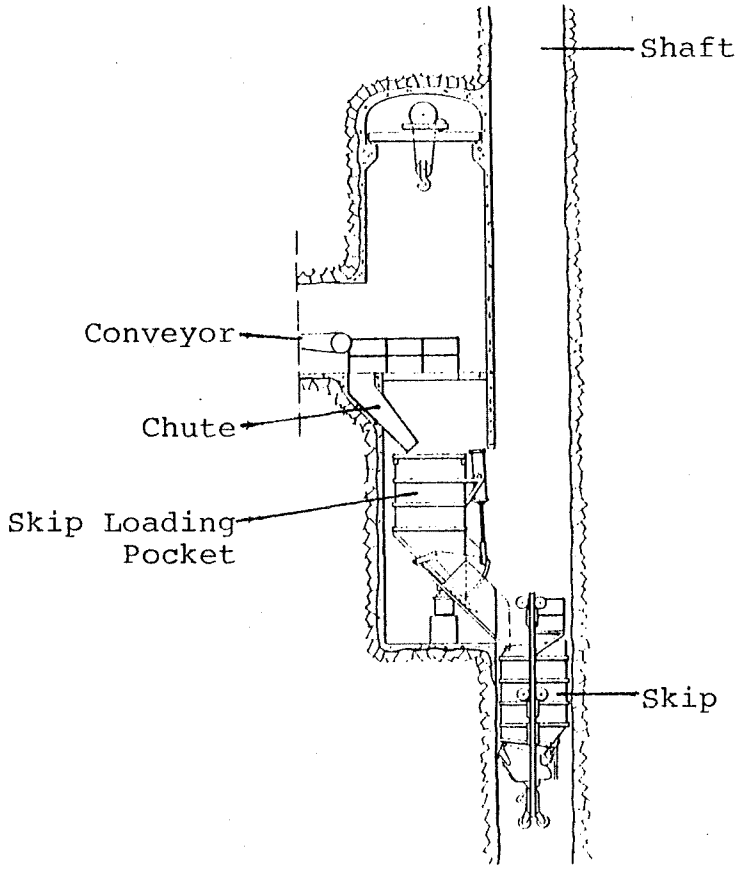


Figure B-9 Schematic of a Skip Loading Station at the Kiruna Mine (Ref. D.1.4) © Ashire Publishing Ltd., 1956

Hoist motors are rated at 4,300 kw and operate at 100 rpm on a thyristor converter drive system. In shafts 9 and 10, skips in balance each have 25-ton capacities and are hoisted at a maximum speed of 15 m per second. The hoists have been designed for 10-rope operation, but are currently equipped with 6 ropes. Drum diameters are 3.25 m, with 170 mm between ropes. Rope diameters are 34 mm on the single-skip hoists and 32 mm on the double-skip hoists.

Ventilation system capacity is being expanded to meet the demand of rising production. The volume of air circulated through the mine will rise from 3.7 million cfm in 1975 to 4.9 million cfm by 1980. Installed fan power for the general system will be doubled from 9,000 kw to 18,000 kw, while fanpower in the local system will increase from 7,000 kw to 8,000 kw. Fresh air heating capacity will expand from 50,000 to 70,000 kw. At the expanded capacity, the air intake system will have 9 fan stations with heating facilities. The fresh air is fed to working areas through a system of 25 distribution shafts. Used air accumulates in a main collection tunnel and is exhausted to the surface through about 25 shafts.

Air circulation is also monitored remotely. The control system has been in operation since 1975, reducing considerably the time required to locate defects and generally improving overall mine ventilation.

B.2.1.5 Durability, Maintenance, Manpower and Energy

Referring back to Figure B-3, which shows the annual production output at Kiruna over time, it is apparent that underground operations had little difficulty keeping pace with and ultimately overtaking all surface mining from a productivity standpoint. This result was achieved through both the installation of modern machinery and basing the entire underground layout on the same basic assumptions that had been made for the open pit, that is, trackless transport of ore feeding raises down

to the various working levels where crushing and hoisting to the surface were accomplished.

From a cost standpoint, the underground mining involved initially a doubling of the production costs, but through successively increasing efficiency, the cost difference in relation to the old open-pit has diminished dramatically. Intensive mechanization and automation over the past 20 years have resulted in a substantial rise in productivity among the underground work force. In fact, during a period when Kiruna's production of crude ore was tripling, the number of underground personnel remained constant. Figure B-10 illustrates the above result in graph form.

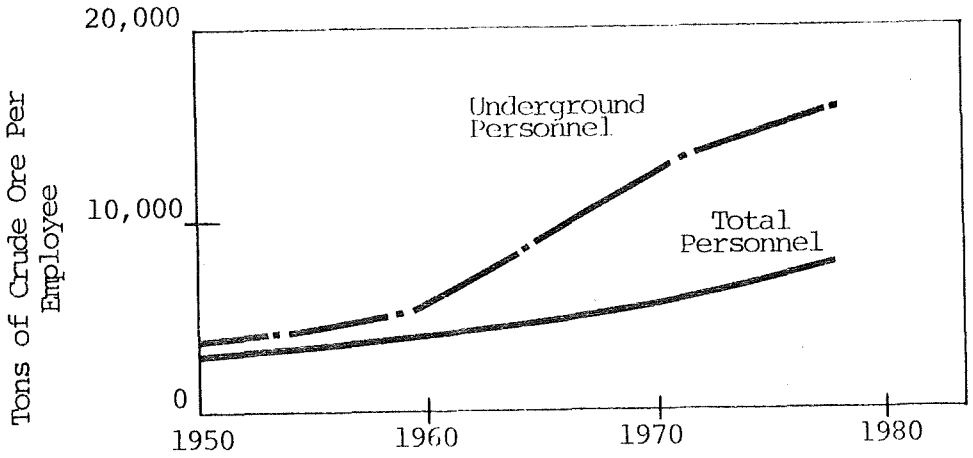


Figure B-10 Illustration of Productivity Increase per Number of Personnel Employed (Total and Underground) at the Kiruna Mine (Ref. D.1.5) © EM/J, 1978

Reliability of equipment operation has taken on increased emphasis at Kiruna due to the already high capital costs required for purchase of new machinery. Moreover, management fully expects such costs to continue to increase sharply as a result of requirements for more sophisticated and automated machinery, environmental requirements, and better maintenance

programs. At present, all maintenance, with the exception of motor rewind and major diesel overhaul, is accomplished underground in a centralized facility.

B.2.1.6 Risk Areas and Technology Gaps

The most complicated operation is that of selectively grading the ore to particular shafts and bins. Because this problem is site specific, it would not be a consideration in the design of continuous transport of material from most open-pit or underground mines. However, it is a factor which comes into play with increasing frequency in new and expanding hard-rock mines. As evidenced by Kiruna's success, coping with this problem underground does not appear to create any added difficulty over procedure that would be employed in an open-pit.

The economics of large-scale mining in today's world of rapidly fluctuating market prices for ore is perhaps the greatest risk facing management. State-owned LKAB, for example, has been slower than most privately held companies in making belt-tightening moves which are needed to cope with the widespread depressed market for iron ore. Needless to say, LKAB has yet to gain any return on the more than $200 million investment made in its new fully automated track haulage facility at the 775-m level and 11-shaft hoisting facility.

The most dramatic example of technological advancement and world economics combining to impact on the Kiruna operation came in the 1950's when the decision to go totally underground was postponed. At that time, the ore in the open-pit was being trucked to 12 permanently installed jaw crushers that were positioned over 5 x 5 foot ore raises that were 150 feet deep. As the pit was deepened further, truck ramps had to be built up to the crushers in order to dump the ore.

An important development that occurred at Kiruna almost 10 years ago was the startup of monthly salaries for miners. Although management had come to recognize that the human stress factor in an underground mine was more devastating than in an open-pit, it was not until a very serious strike in the mine around the end of 1969 that monthly salaries were introduced for all miners. The effects of this change are not yet fully apparent, although there has been an expected lowering of productivity from 5 to 10%. What concerns management is whether lower productivity will become an irreversible trend as incentive to work diminishes.

B.2.1.7 Rationale for Mining System Selection--Constraints and Economics

The decision to change over from surface to underground mining was made at the end of the 1930's when an acceptable level of gangue-to-ore was considered to be a 1:1 ratio. In the subsequent 40 years, the open-pit technique developed at a faster rate compared to underground, especially with respect to size and production capacity of the mining machinery employed. Given these new circumstances, iron ore mining in Sweden today would consider a 6:1 waste-to-ore ratio as the dividing line between continuing open-pit versus going underground. Even in the 1950's when 3:1 was the guiding ratio, calculations showed Kiruna management that it should have returned to open-pit, but technically this was impossible to do while mining was in progress.

The above recount shows how the parameters affecting the initial decision changed significantly with time. This all leads to the question of whether underground mining will ever be competitive with pit operations from a profitiability standpoint. The Swedes have investigated this matter extensively

and the following synopsizes their most relevant thoughts *(Ref. D.1.6)*:

> Swedish ore mining is concerned mainly with iron ore, most of which is mined underground. With the intensive increase in world production, the iron ore trade has lost its regional character and has moved to an international plane. This has happened, among other reasons, because of the exploitation of new deposits, many of which are far from the consumer regions. As a rule, open-pit mining of such new deposits is advantageous. But this does not affect the question as to which method is preferable--open-pit or underground mining. It is simply a fact that management emphasis has been on high production and open-pit mining right from the start.
>
> The above-mentioned attitude is further enhanced by the fact that it is easy to embark on open-pit mining since this method is relatively simple and the equipment is usually available for purchase. Underground mining requires greater knowledge, more specialized equipment, etc. Young engineers are tempted by advertising in the mining journals which abound in colorful illustrations of attractive machines for open-pit mining. Anyone standing at the edge of such a mine and watching the activity below will gain an impression of order, flexibility and productivity. It is far more difficult to form a general idea of the activity in an underground mine. Hence, open-pit mining has a psychological advantage. The negative attitude encountered with respect to underground mining frequently emanates from a lack of knowledge of the circumstances in which underground mining may be preferable.

LKAB has worked the following example to illustrate the comparative productivities of open-pit to underground mining, given today's most advanced equipment *(Ref. D.1.6)*:

> Ore production at Leveaniemi (located at Svappavaara--see Section B.2.9), an open-pit iron ore mine in Sweden, started in 1965. At that time, output per shift was about 3,500 tons per working unit (one excavator and three 40-ton trucks). Eight years later, output exceeded 5,000 tons owing to the fact that larger units were in use. This corresponds to an increase in productivity of about 5% per annum.

> In the early 1960's, Kiruna was restricted to scraper-loaders and electric shuttle cars of low capacity. Gradually, the electric shuttle cars were replaced by diesel trucks and in turn by front-loaders. Output per machine shift has increased over a period of 10 years from about 700 to about 1,700 tons. The actual increase in productivity is greater than these figures suggest, since the scraper-loader system required two men. Bearing this factor in mind, the productivity per man shift has realistically increased by about 10% per annum during the period 1960-1970.

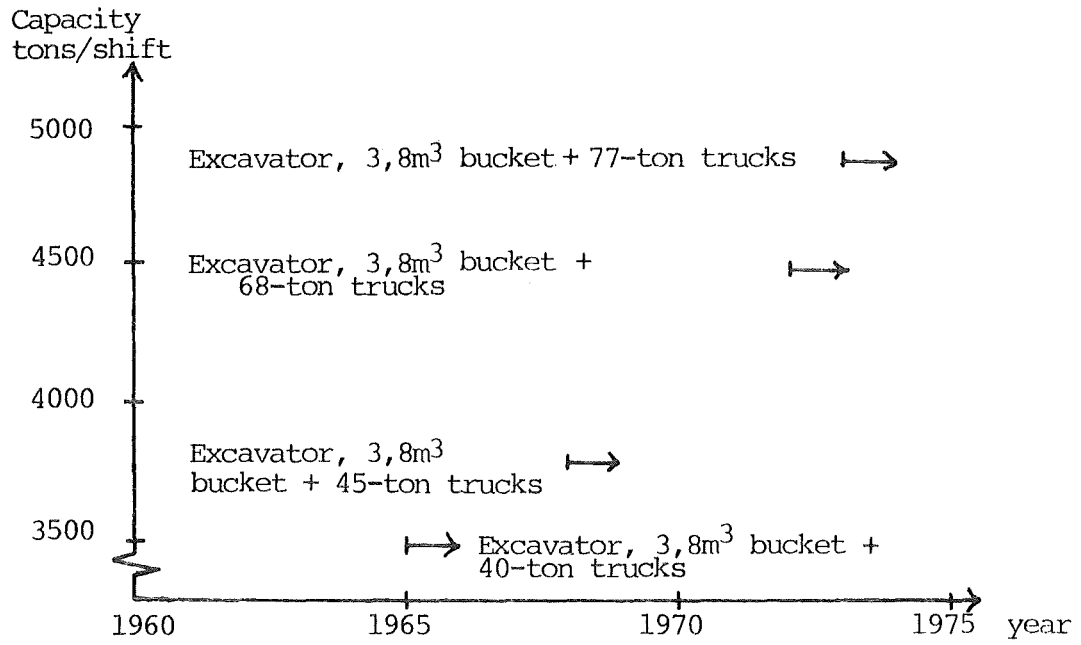
Figures B-11 and B-12 graphically display the above findings.

Underground ore haulage at Kiruna has also undergone extensive change over the years to the point where it is a more efficient and productive system compared to the Leveaniemi operation *(Ref. D.1.6)*:

> At the beginning of the 1960's, trains had a capacity of 200 tons. Twenty years later train capacity will reach 500 tons. This amounts to an increase of 5% per annum. Even larger units could be employed, but this becomes uneconomical because the time spent at the terminals will constitute too large a proportion of the total cycle time. The economical advantages of increasingly larger units therefore cannot be realized beyond a certain limit.

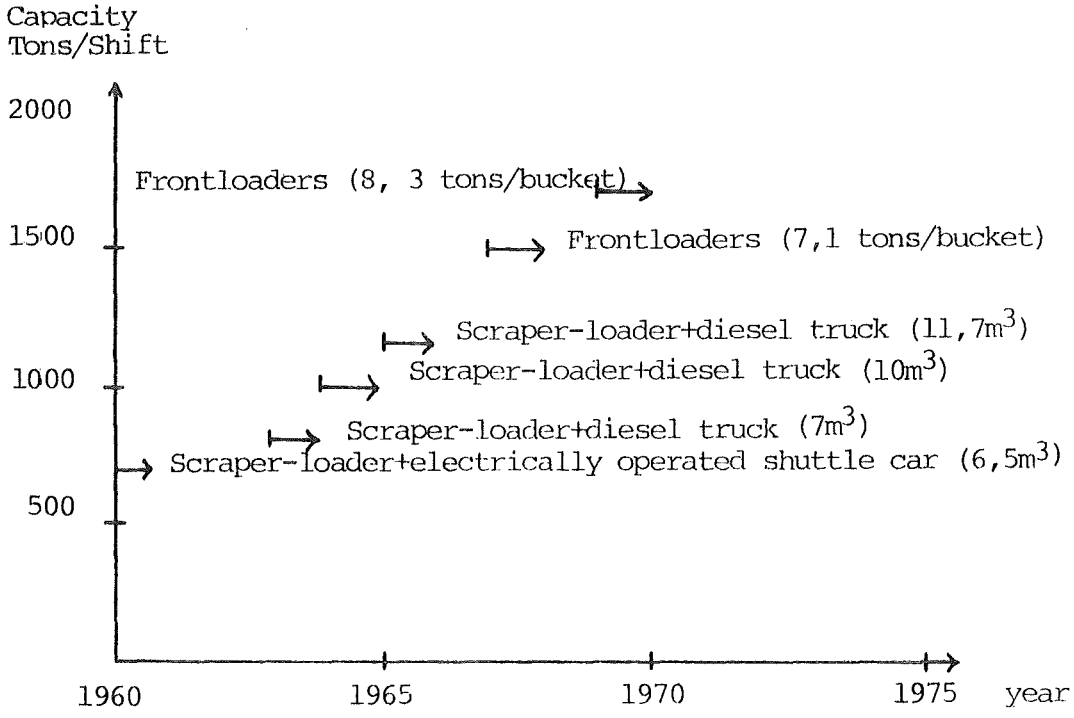
The LKAB studies have led to the conclusion that underground productivity has kept pace with open-pit operation due to the major improvements in loading and haulage at Kiruna.

Figure B-13 shows the development costs at the Leveaniemi and the Kiruna mines as a function of time. At the open-pit mine, the costs have risen since 1965 at a rate of about 3.5% per annum. The corresponding rate for the underground mine amounts to 4.0% per annum. In spite of the considerable increase in productivity at both mines, the mining costs per ton of ore have still increased. This is principally due to the fact that capital costs for mobile equipment have increased substantially at the same time. In conclusion, although the rate at which underground mining costs have increased is somewhat



Haulage distance about 1800m - Three transport units per excavator

Figure B-11 Leveaniemi Open-Pit Mine (Ref. D.1.6)



Haulage distance about 100m - One transport unit per loader

Figure B-12 Kirunavaara Underground Mine (Ref. D.1.6)

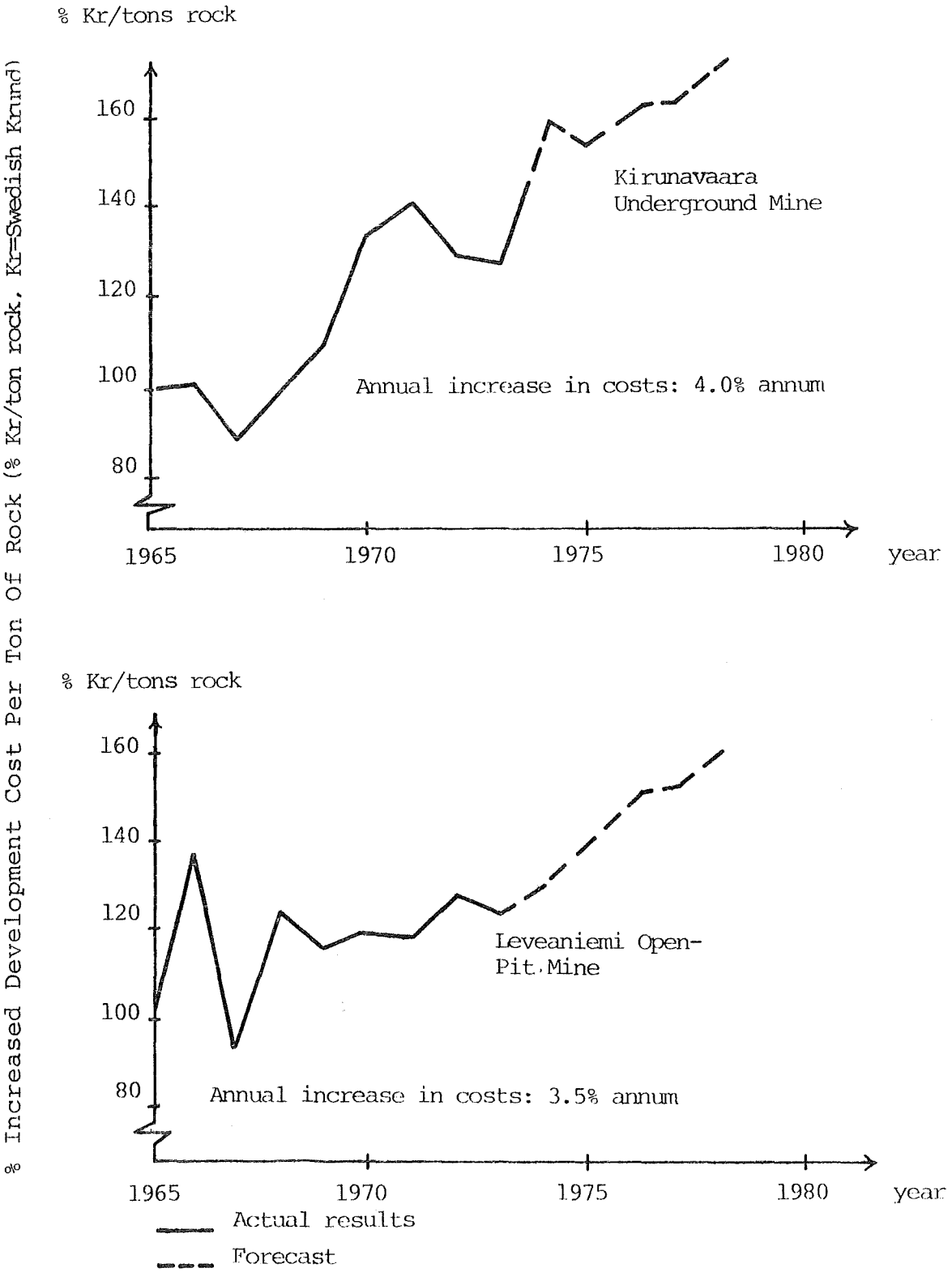


Figure B-13 Ore Mining Costs at the Leveaniemi Open-Pit Mine and the Kirunavaara Underground Mine (Ref. D.1.6)

higher than with open-pit, LKAB feels that the difference is not significant.

B.2.1.8 Conclusion

The Kiruna operation has demonstrated three important points to the mining world.

- Large tonnage can be handled from an open-pit through underground.
- Large tonnage can be automated for underground rail haulage.
- Vertical hoisting of ore was more economical than the use of inclined conveyors in deeper mines (>500m) of this kind. (that is, mines that have been in operation for many years and invested is the facilities for skip hoisting when this was the predominant mode for vertical haulage.)

The Kiruna case history clearly illustrates how even the most careful of planning efforts may not anticipate the technological changes to come in spaces of time that are relatively short compared to a large mine's expected life, especially with regard to an underground operation. In an open-pit mine, for example, there are no restrictions on the introduction of increasingly large machines. Underground operation, on the other hand, is always limited by narrow workings. However, anyone predicting 20 years ago that LKAB would have the capacity to produce 30-plus million tons of rock underground at Kiruna would have sounded ridiculous. This would also have been true of anyone suggesting that front-loaders with 15-ton and larger buckets would be possible. Some imagination in the planning stage is therefore essential and perhaps even more so with underground mining.

Indicative of the Swedes' attention to a thorough planning effort in preparation for future operations is the current effort that is underway. The long-range planning group is

faced with the problem of new haulage and mining methods, as the ore above the present skip bottoms is mined out. They will need a decision by 1981 in order to develop the mine with enough lead time for the 1990's production. They are now in the process of deciding which haulage technique to use--train, belt, inclined skips, or inclined haul road (with electric or diesel trucks).

Based upon past data analysis for the 775-m haulage level (see Table B-2), one might think they would favor some sort of electric trolley system supplying power to a truck haulage fleet. Equally important, though was the fact that neither they nor any similar mine had any practical experience in any of the new systems. To base the entire production on a system that had not been technically proven over a long period would be dangerous. The choice therefore fell on conventional track haulage.

TABLE B-2

1975 COST COMPARISON OF ALTERNATIVE UNDERGROUND HAULAGE METHODS FOR THE 775-M LEVEL IN THE KIRUNA MINE*

(Figures are written as percentages. Baseline is track haulage investment cost set at 100%.)
(Ref. D.1.7)

	Trains	Belts	Inclined Skip	Trucks on Inclined Ramp
Distance between levels (m)	250 m	250 m	250 m	100 m
Life (yrs)	10	10	10	4
Investment (%) Baseline 100	100	110	70	25
Capital Cost (%)	48	51	36	28
Operating Cost (%)	42	53	50	60
Maintenance Cost (%)	10	6	6	8
Total	100	110	92	96

*Inflation over time has been included in the above calculation

Returning to the planning effort now under way, to aid the evaluation process they have constructed a small inclined hoist to mine out a small orebody in 1½ to 3 years. The mined ore passes over a grizzly with a stinger (rock breaker) and vibrating feeder ahead of a jaw crusher. The material is then fed to an ore pass and an inclined skip hoist. This whole system is operated by one man at a control panel with a closed circuit T.V. monitor.

Returning in conclusion to the more general question of converting from surface to underground mining, it appears that the decision to change should be deferred as long as possible. Moreover, the choice of underground mining method must also take into account environmental and health aspects, since the costs of these measures are becoming an increasingly significant part of mine operation.

B.2.2 Pyhasalmi Mine--Finland

The key point to note about the relatively new and modern Pyhasalmi Mine is that it was originally designed as a combined surface/underground mining operation. Transition to total underground activity (as of 1975) gradually took place during the first 13 years of production. Ore crushing and haulage have from the outset been accomplished underground.

Figure B-14 presents an aerial view of the current mining operation. Open-pit activity has virtually ceased, except for the backfill operation with underground mine waste which is first hoisted to the surface via the auxiliary shaft shown on the left). Ore is hoisted from underground in the main shaft (shown to the right of center in the figure) and processed on site in the milling facility prior to final shipment.

The following sections highlight the key operating features of the Pyhasalmi Mine and conclude with a discussion of the decision making process which adopted the integrated surface/underground approach. Table B-3 briefly summarized some of the significant facts on the mine.

B.2.2.1 General Mine History and Orebody Characteristics

The Pyhasalmi Mine is located in central Finland, 150 km northwest of Kuopio (see Figure 2-10), and is owned by the Outokumpu Company, a government-sponsored organization. The ore content averages 38% sulfur, 33% iron, 2.15% zinc, and 0.8% copper. Mine construction was initiated in 1958 with startup of ore production taking place in 1962 at the rate of 600,000 tpy. In the early years, over 75% of the ore production came from pit operations and the remainder through sublevel caving. Ore production in 1977 exceeded 900,000 tons and resulted from a combination of sublevel caving, sublevel stoping and shrinkage (open-pit mining having been discontinued after 1975).

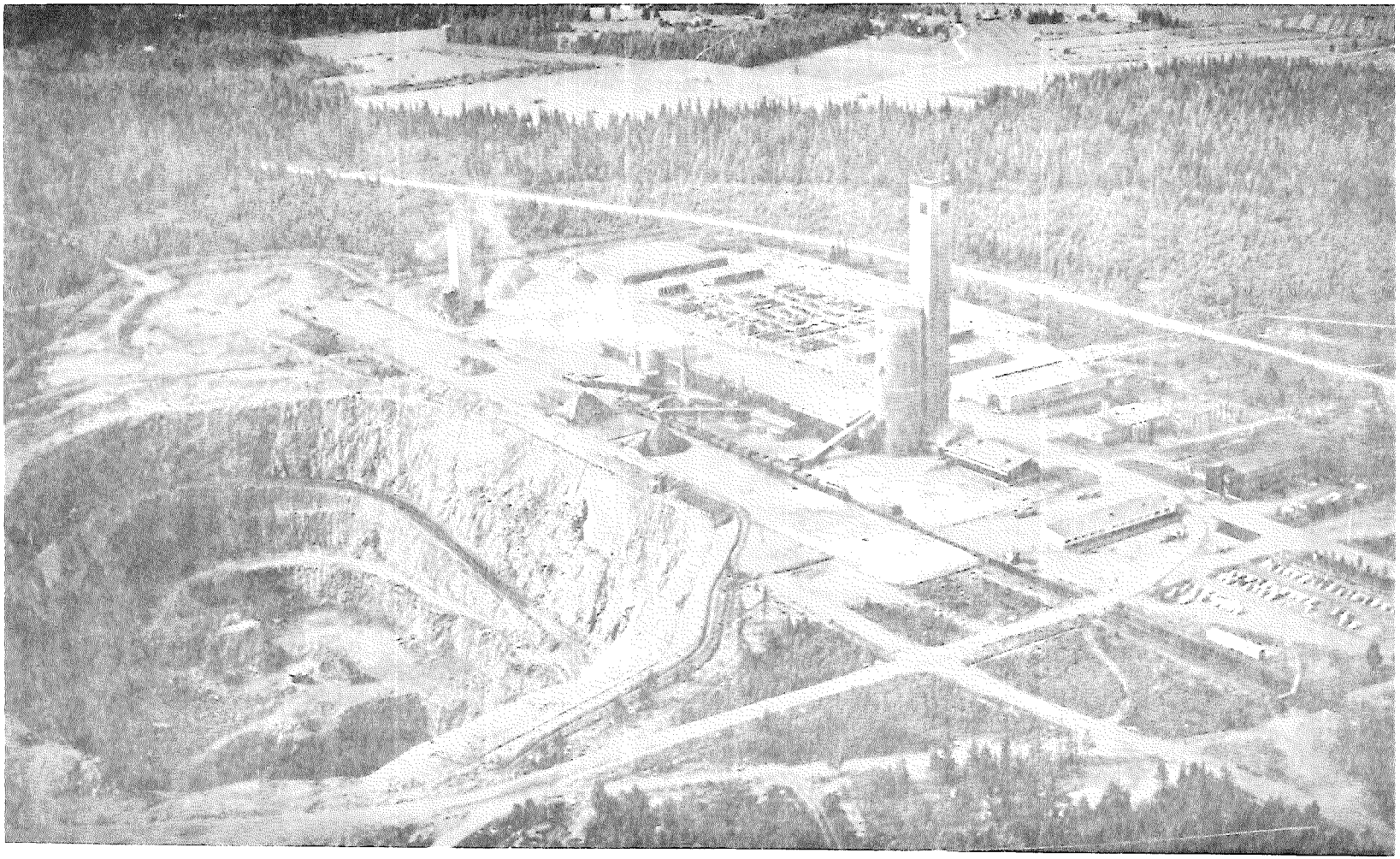


Figure B-14 Aerial View of the Pyhasalmi Mine in Finland

TABLE B-3

FACT SHEET ON THE PYHASALMI MINE

- Owner - Outokumpu Oy (Finnish Government)
- Location - 150 km northwest of Kuopio, Finland
- Ore - 0.7% to 0.8% Cu., 2.15% Zn, 33% Fe, 38% S
- Mineralization - chalcopyrite, pyrite, sphalerite
- Pit Operations Started - 1962
- Joint Underground Operations Started - 1965
- Pit Operations Ceased - 1975
- Production - 900,000 tpy
- Potential Mine Life at Present Production Rate - 20 years
- Operating Personnel - 500
- Operations:
    - Open-pit (now defunct) and underground working faces feed ore through raises to drawpoints.
    - Ore hauled via LHD from drawpoints to jaw crusher, then via conveyor to ore skips in #1 shaft.
    - #2 shaft handles hoisting of waste rock and provides ventilation air.
    - Sloped ramp at 8^{o} to the underground is used to transport large equipment and materials.
- Equipment:
    - Two-stage jaw crusher and vibratory feeder system is automated and monitored by television camera.
    - Friction hoist employs a single 10.8-ton skip running at 8.5 m/sec for 530 m. Men and material are hoisted in the same shaft.
    - 5 cu m LHDs haul ore from ore pass drawpoints to the crusher.

The orebody is of breccia type composed principally of coarse-grained pyrite (with the grains joined together with a sphalenite - chalcopyrite - gangue mix) and extends to a depth of over 500 m, the dip being essentially vertical except at the northern part (50^{o} to 70^{o}). As shown in Figure B-15. the orebody length is approximately 650 m and the width at surface level is on the order of 75 m, but narrows to a thickness of around 25 m at a depth of 115 m. Due to the orebody's extensive depth, initial calculations showed that while open-pit mining might be productive at best for 10 to 15 years, the known total reserves were sufficient to last for decades using underground mining techniques. This consideration led to the ultimate decision to initiate both surface and underground operations at the outset and gradually effect the transition to a total underground operation early in the mine's life.

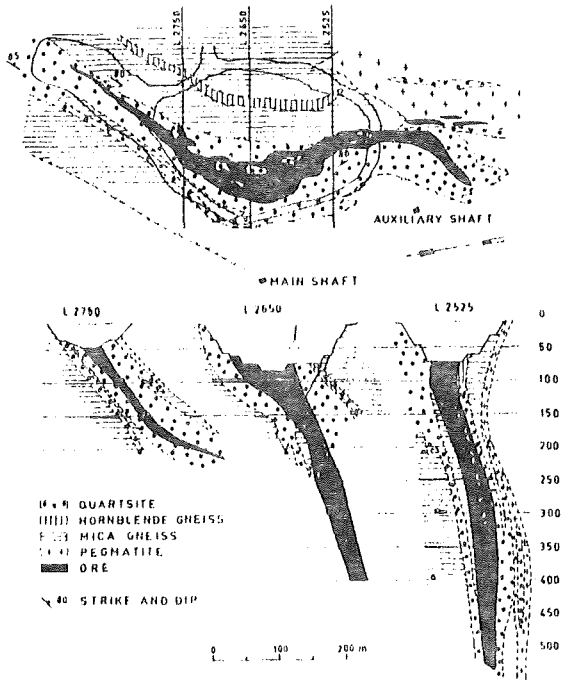


Figure B-15 Geological Map and Cross-Sections of the Orebody at the Pyhasalmi Mine (Ref. D.2.1) © *AIME, 1969*

B.2.2.2 Mine Layout

The open-pit is 800 m long, 225 m wide, and 115 m deep. The pit walls are unusually steep (averaging 60°) and bench height is approximately 12.5 m. Access to the underground mine is through two shafts and an 8° incline for vehicular travel. The main shaft, with a depth of 500 m, and the auxiliary shaft, with a depth of 537 m, are used for ore hoisting service transportation and ventilation waste hoisting respectively. Figure B-16 illustrates the overall layout.

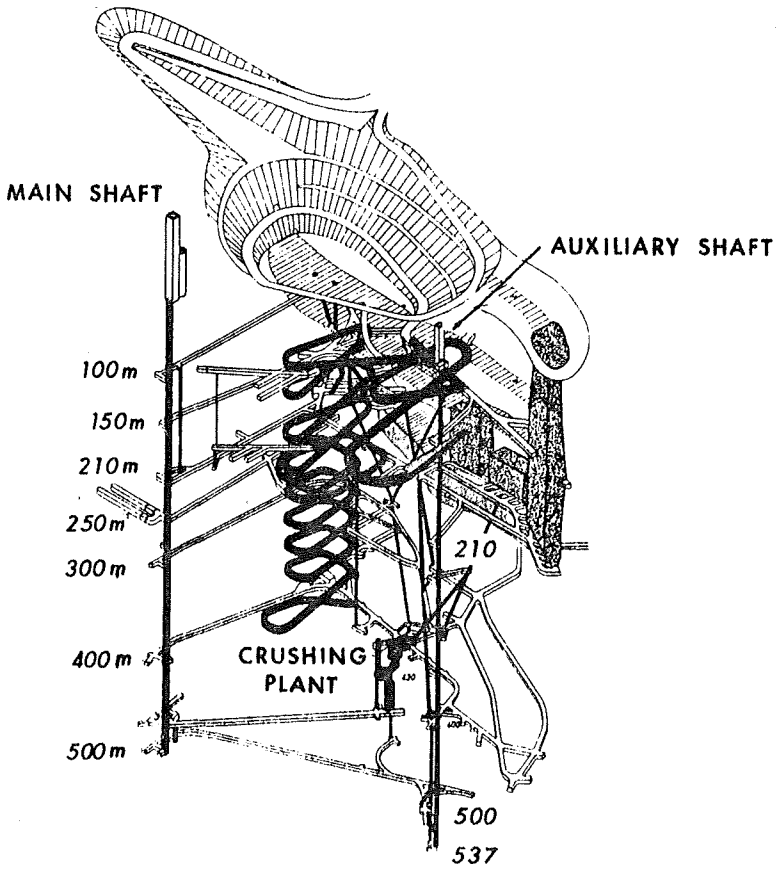


Figure B-16 Isometric View of the Pyhasalmi Mine (Ref. D.2.2)

Initially a one-stage underground crushing plant was located at the 200 m level. A 200 m-long belt conveyor was installed at the 250 m level from which the hoisting of the ore took place. The current two-stage crushing plant has been constructed at the 400 m level with another conveyor belt at the 470 m sublevel. The ore is fed into the crusher by LHD which enhances water separation from the ore (as opposed to the slushing technique used to feed the one-stage plant).

Transversal sublevel shrinkage stoping is being utilized on the ore above the 400 m level as illustrated in Figure B-17. Stoping commences at the center of the orebody. Temporary pillars have been left in the vicinity of the open-pit, the ore passes, and directly below the 200 m level.

B.2.2.3 Mining Operation

During mine startup, open-pit operation produced approximately a 0.8:1 waste-to-ore ratio with the ore being transported by LHD to one of several ore passes situated at the pit bottom. (The ore passes are of square configuration, 2.5 m x 2.5 m, set at an angle of 60° to 70°.) At the same time, overburden and waste were loaded into 20 ton trucks and transported to a dump on the surface for later use as mine fill.

Crushing during the open-pit phase took place in the single-jaw crusher at the 200 m level and was supplemented with the two-stage crushing operation at the 400 m level where it is crushed; waste is similarly passed to a collection bin at the 512 m level where it is hoisted to the surface without crushing. The waste-to-ore ratio is approximately 1:10.

In the present underground operation, the ore is fed to the primary crusher by loaders. The primary discharge enters the secondary crusher, and then goes to a 3,200 ton surge bin. From the surge a conveyor (1.0 m wide x 200 m long)

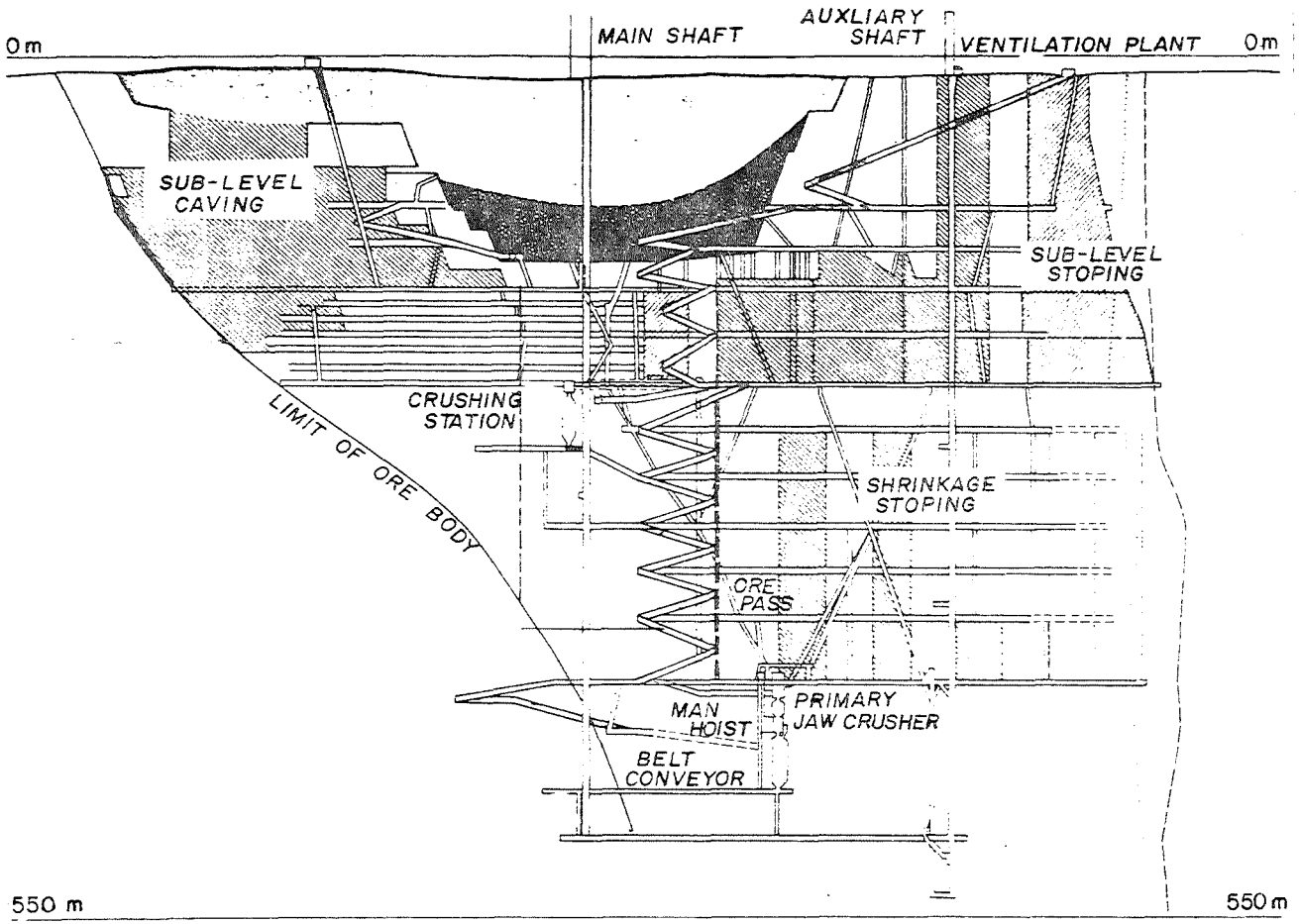


Figure B-17 Schematic of Mining Development Plan and Ore Haulage Network at the Pyhasalmi Mine

transports the crushed ore to a 10.8 ton measuring pocket adjacent to the transport shaft in preparation for skip loading and hoisting.

Beginning in 1969, the overburden and waste located in the surface disposal area were used for stope fill and were transported underground via bore hole. At present, the waste hoisted up the auxiliary shaft from underground is being trucked back into the pit or raises where it is passed back down into old stopes for fill immediately following excavation.

B.2.2.4 Mining Equipment

From the outset, the underground operation was planned for trackless mining equipment. All drifts are driven by heavy self-propelled drilling jumbos and large rubber-tired LHD's. AS mentioned earlier, the LHD's also handle haulage of the ore from the ore pass drawpoints to feed the crusher. The specially constructed underground rampway (inclined at 8^{o}) permits transfer of the heavy equipment between levels.

The ore is crushed underground in two stages starting at the 400 m level, conveyed 200 m to the main shaft, and brought to the surface via automatic friction hoist in a 10.8-ton skip for further crushing and milling. Although the crusher arrangement at the shaft bottom was not unique, it was well laid out. The entire crusher/feeder system was automated and monitored by a single man located in a control room near the secondary crusher. Television cameras provided contact with critical points in the system.

The mine ventilation is completely automated through centralized computer control. Total system capacity is 500,000 m^3/hr. Remotely-actuated motor-driven ventilation doors and remotely-monitored air-flow meters in various parts of the mine regulate the air flow through all mine passages. In winter, the down cast air is also heated.

B.2.2.5 Durability, Maintenance, Manpower and Energy

In characterisitic Scandinavian fashion, Pyhasalmi has employed the latest technology available to the European mining industry. For example, the conveyor belt running from the two-stage crusher to the main shaft is controlled by a metal detector which has the capability to stop belt movement and remove tramp iron. Another case in point is the automated man shaft installed alongside the 50 m high crusher plant to permit quick and easy visual inspection and ready access for maintenance.

The recognized importance of proper equipment maintenance and repair is clearly reflected in the centralized location of the underground maintenance facility which appeared well-equipped to handle all levels of maintenance activity.

Attention to minimizing ore haulage requirements is evident in the use of *inclined* ore passes which feed conveniently located drawpoints tended by LHD's.

This particular approach to ore transfer is but one of the many carefully laid out underground operations at Pyhasalmi which represents the level of planning and testing that preceded actual construction. For example, the absence of erosion problems in the ore passes was established early in the mine development effort when detailed material analysis was performed on the host rock.

A strong selling point contributing to the effective manpower utilization and superior equipment maintenance that we witnessed is the incentive system used by management. High productivity levels on most shifts related directly to bonuses which were awarded for surpassing daily quotas. This was the only mine in our entire travels that operated on such a basis.

B.2.2.6 Risk Areas and Technology Gaps

The one significant risk area that came to light during the on-site inspection is the potentially explosive pyrite dust which has on occasion undergone spontaneous ignition. For this reason, all miners are evacuated from the underground during blasting which is now controlled from the central monitoring facility at the surface. Otherwise, ventilation air movement has proved sufficient to keep dust levels down and prevent the further occurrence of this potential catastrophe during normal mining activity.

Another risk area worth mentioning is the proper selection of a stoping method. Each method reflects a planned response to the anticipated ground control characteristics which can vary within a mining district or within a given mine. Pyhasalmi mining method currently employs a combination method of shrinkage and sublevel stoping which to date has provided an effective extraction method in the ground conditions presently encountered.

Being the most advanced of the mines visited, Pyhasalmi by all rights was not plagued with technological gaps. However, ground control is the chief problem in any underground operation and Pyhasalmi is no exception. Improved equipment and procedures for monitoring ground movement and even the advent of new mining techniques are needed. Given the present situation, it is apparent that a significant quantity of waste material is handled to provide sufficient backfill--a characteristic of many hardrock operations. What might help matters in general is a more direct path and the related equipment for recycling waste material back into mined-out stopes.

B.2.2.7 Rationale for Mining System Selection-- Constraints and Economics

The rationale for selecting the combined surface/ underground mining operation has been touched on throughout this section, and is accurately synopsized in the following statement *(Ref. D.2.1)* © *AIME, 1969* :

> Since the orebody was covered by overburden only 2 to 3 m thick, open-cast mining was the obvious method, at least at the commencement of activities. However, since the orebody extended fairly deep, it was equally evident that underground mining would be started later. It was calculated that for open-cast mining there was ore for 10 years at the most, but the ore reserves elsewhere were known to be sufficient for several decades. It was decided to develop the underground facilities immediately to serve the surface operations and to allow early testing and transition to underground mining. To avoid building two crushing plants, it was decided to install the crushers under the open pit at sufficient depth. In this crushing plant, ore from the open pit as well as from the underground stopes could be treated. This solution also had the advantage that the ore from the open pit could be dropped into the crushing plant, thus reducing the transport of the ore in and from the open pit. Moreover, it was cheaper to hoist ore through the vertical shaft than to truck it along the spiral roadway up to the surface.
>
> Waste from the open pit was to be used as fill. A sufficiently early commencement of underground stoping would allow the dumping of waste from the lower part of the pit directly into the stopes. The establishment of an experimental stope under the open pit, the stability of which could be observed and which might be used later for testing of the filling operation, was also important. Thus, there would also be enough time for the development of underground stoping.

During our visit we attempted to gain more detailed information on the mining economics at Pyhasalmi, but the management was not at liberty to discuss this beyond giving annual production figures and number of personnel employed. Using our best judgment after a first-hand look at their operation, we would estimate that an equivalent mine setup in the U. S. would have little difficulty being profitable.

B.2.2.8 Conclusion

After inspecting the Pyhasalmi operation our collective impression as experienced mining engineers was unanimous on the model-like nature of what we had seen. Not only had the latest mining equipment and techniques been selected, but also top quality managers had been brought in to both plan and run the operation. A number of different mining methods had been practiced to accommodate the varied geological distribution of the deposit. In keeping with a well-planned effort, an underground trial stope was initiated early on after commencement of pit operation to establish the feasibility of longitudinal sublevel stoping without pillars. In addition, a combination of hydraulic and cemented backfilling were employed to aid in ground control.

Another example of the foresight used in carefully planning the mine operation is the dust control procedure for the ore passes, a potential source of high dust levels. High ventilation air counter pressures are maintained at the ore pass drawpoints to force air movement back up into the boreholes, thereby facilitating a more gradual settlement of the dust laden air.

In conclusion, we would compare the Pyhasalmi mine to a well-engineered, finely-tuned machine. Without a doubt, it is the most noteworthy and innovative of the mines visited during this program and is living testimony to the inestimable value of a thorough planning effort.

B.2.3 Open-Pit Mine - Spain

Problems of significant magnitude currently plague the combined pit/underground operation at this Spanish Mine. Pit design, handling problems related to the wet ore, and excessive rail transport distances are some of the major difficulties impacting on mine productivity. Future world interest will focus on the effectiveness of the corrective measures currently being implemented by mine management, especially with respect to the use of inclined conveyors and rail transport for continuous underground haulage.

There are three separate mine workings at the Spanish Mine Complex. One open-pit mine is abandoned; the second is a very old combination open-pit/underground operation; and the third is a relatively new open-pit. This section will deal principally with ore haulage at the second open-pit and to a lesser extent with the newer open-pit. Figure B-18 provides a view of the extensive pit workings at the former while Table B-4 lists some of its principal characteristics.

B.2.3.1 General Mine History and Orebody Characteristics

The mine complex is located in southwestern Spain, northwest of Seville (see Figure 2-10). Both open-pit and underground activity date back to over 2,800 years ago when Phoenician and Roman mining operations were extensive as indicated by the 20 million tons of slag left behind. It is the oldest active mining district in the world. Between 1875 and 1890, it grew to become one of the largest mining districts in the world. Total ore excavation to date totals an estimated 250 million tons of pyrite and 5 million tons of copper. Present ore reserves are approximated at:

- 10.9 million tons pyrite from open-pit.
- 13.5 million tons copper ore from open-pit and underground.

- 44.0 million tons of other minerals (lead, zinc, etc.) from open-pit and underground.
- 22.0 million tons Roman Slags.

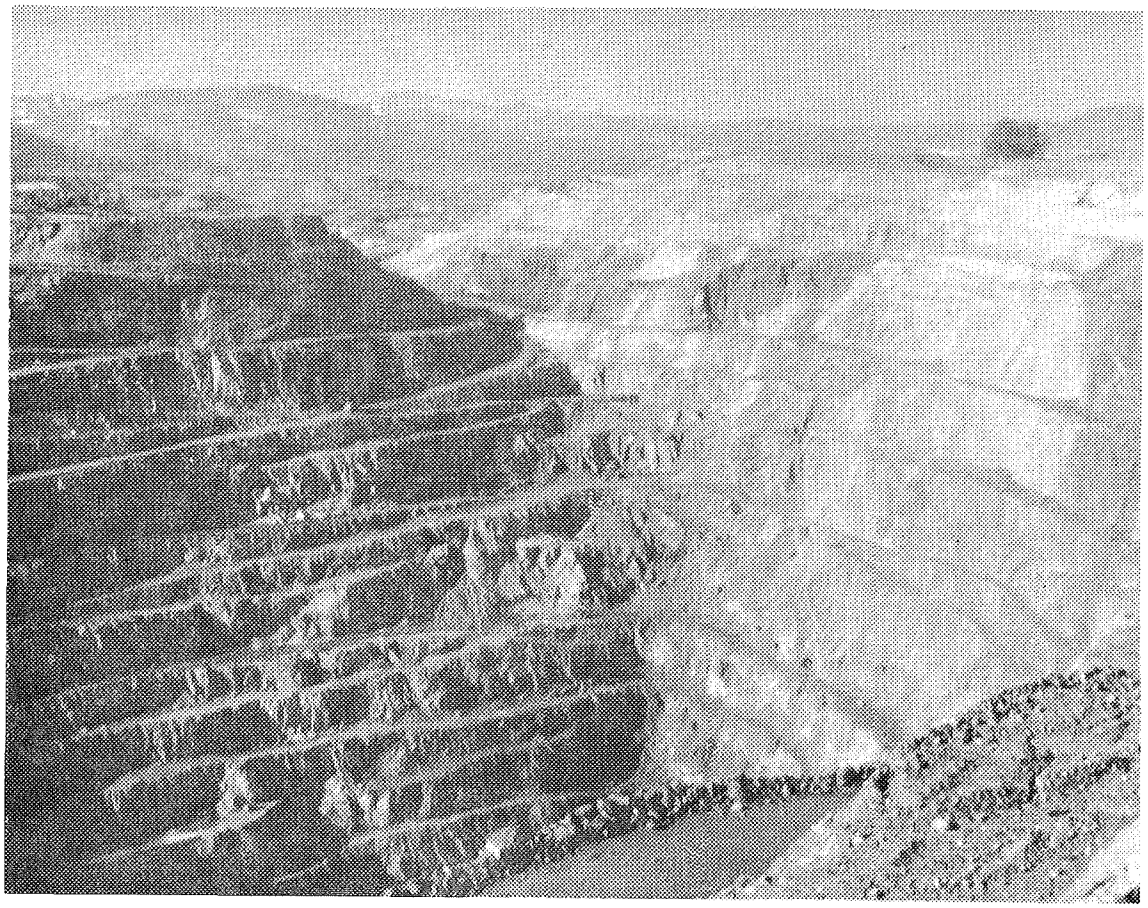


Figure B-18 Overhead View of the Impressive Combined Open-Pit/Underground Mine in Spain

TABLE B-4

FACT SHEET ON THE SPANISH MINE

- Location - northwest of Seville in southwestern Spain
- Ore - Pyrite and .25 to .7% Cu -- Leaching of mill tailings produces .19% Cu
- Mineralization - Chalcopyrite and calcocite
- Pit/Underground Operations Started - Roman times
- Combination Open-Pit/Underground Operations Started - 1940
- Production - 6,000,000 tpy by 1980
- Operating Personnel - 262
- Potential Mine Life at Present Production Rate - 20 years
- Operations (Open-Pit):
    - Pit ore is trucked into an adit at the pit bottom to a raise that feeds a crusher
    - Underground ore is fed via ore pass to a second crusher
    - A long inclined conveyor system from the crushers feed track haulage units
    - Lengthy horizontal rail network to pyrite stockpile (5.5 km) and mill (7 km)
- Problems:
    - Size of the adit is limited to small trucks (35 ton)
    - Equipment maintenance is high because of the acid nature of the ground water
    - Ore production cost is high since track haulage system cannot run at full capacity
    - Track haulage route is excessively lengthy

The primary underground ore is found in a relatively unaltered series of welded tufts. This gangue has a very high silica content and is extremely abrasive in a pulp stream, cutting out pipes and feed boxes quite rapidly in the mill. Chalcopyrite is the copper mineral and is found in veinlets and stringers in association with pyrite, quartz, and chlorite. Groups of these veinlets are rich enough to be mined as ore. There is no regular or predictable pattern to the shape and limits of economic mineralization.

The ore trend extends over an area 4 km long and 2 km wide. Ore findings have been made up to 260 m below the present pit bottom. Underground the hanging wall is composed of rhyolite and the footwall of slate.

The primary underground ore is capped by a leached gossan in which the ancient miners tunneled and burrowed for gold and silver. Most of the barren material (including ancient dumps) is stripped and discarded as waste. There are, however, several million tons of gossan assaying 0.08 ounce gold and 1.3 ounces of silver per ton. The gold is very finely disseminated throughout the gossan while the silver is concentrated in a thin blanket near the base of the gossan. Lead occurs as anglesite ($PbSO_4$). The ore grade portion of the gossan is stockpiled as it is stripped for future milling.

B.2.3.2 Mine Layout

Past mine sites have included two underground operations, two open-pits, a copper-gold-silver mill and a pyrite processing facility. At present, two open-pits, and one underground mine are supplying the surface facilities.

The old open-pit is steep and narrow with dimensions running approximately 1,400 m x 1,000 m and a depth of 250 m; it lies adjacent to the remaining underground operation. Bench heights run 12 m and pit walls slope at 45^{o} on the north side, but only 36^{o} on the south side where the slate is.

Figure B-19 gives an abbreviated picture of the haulage system and its relative position to the open-pit. A truck adit runs horizontally for 150 m from the pit bottom to an underground ore pass 30 m in depth. Trucks feed the pit pyrite down the raise to an underground crusher. Cross-sectional dimensions of this vertical transfer raise are 4.5m x 4.5m.

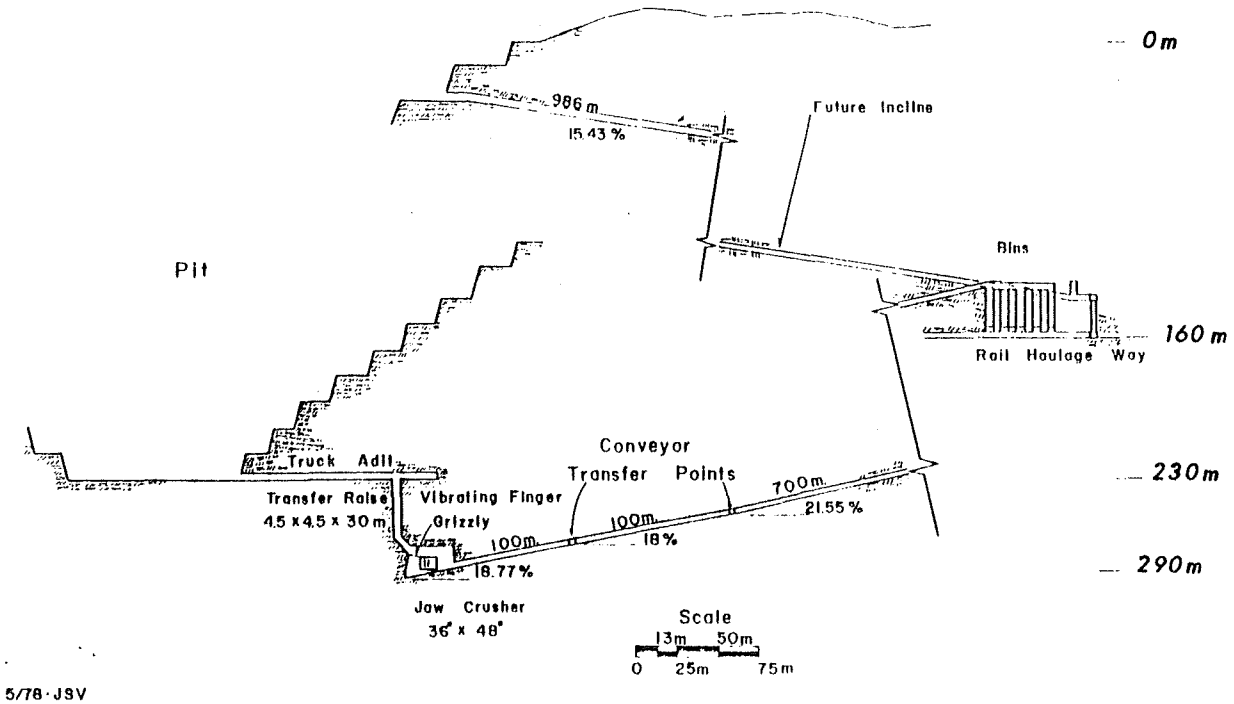


Figure B-19 Layout of the Underground Mine Adjacent to the Open-Pit Operation in Spain

Underground mining operations feed a combined pyrite/copper-lead-zinc mixture to a second crusher installation located 100 m below pit bottom. A long inclined conveyor network links the crushers to a rail haulage load point, from where the ore is shipped 5.5 km to the pyrite stockpile and a further 7 km to the copper mill. The circuitous ore path is illustrated schematically in Figure B-20.

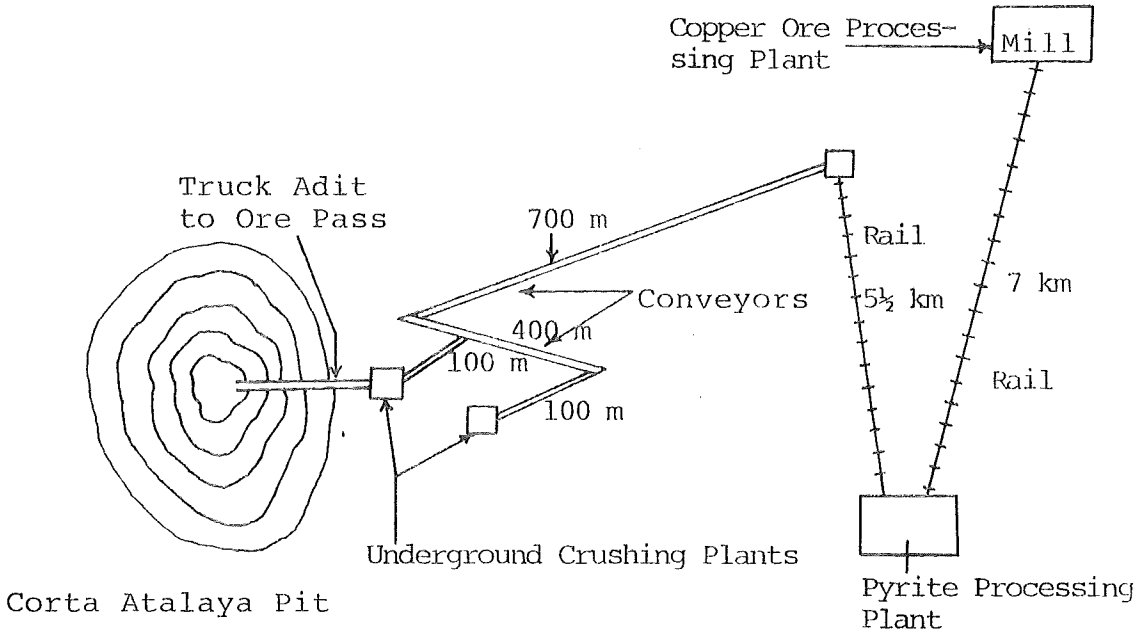


Figure B-20 Plan View of the Ore Haulage Path From Crusher to Mill at the Spanish Mine

Future plans call for a major expansion program which would move pit boundaries into the underground workings as shown in Figure B-21. In addition, a third crusher facility would be created at a depth of 250 m to service the future underground operation, while the crusher installation closest to the pit bottom would most likely be phased out.

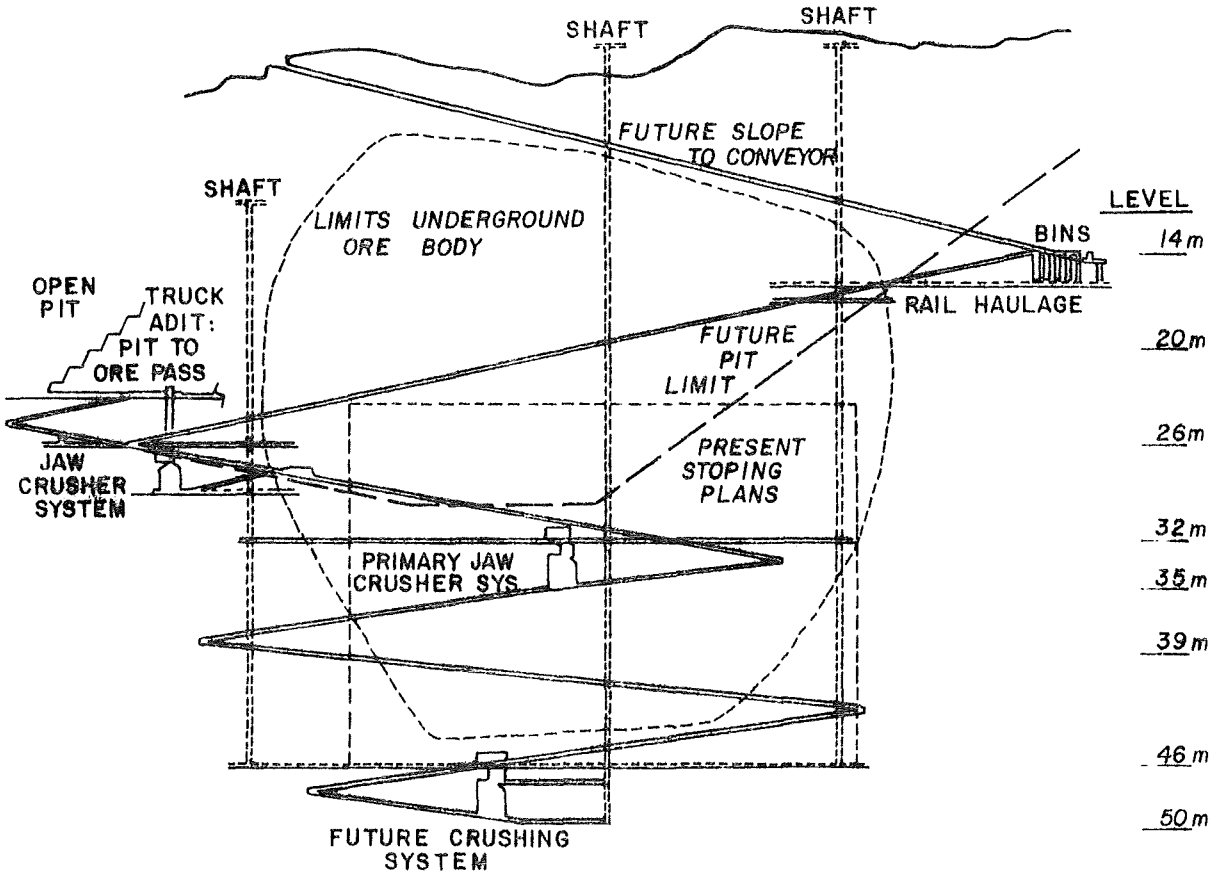


Figure B-21 Expansion Plans for Open-Pit and Underground Operations at the Spanish Mine

B.2.3.3 Mining Operation

In the mines pyritic ores from the open-pit and a complex copper, lead, zinc ore from the underground. The open-pit mine uses the underground haulage system for transportation of the ore to the surface. The movement network consists of:

- Dumping the ore from a 35-ton rear dump trucks into an ore raise above a 36" x 48" jaw crusher which is fed by a vibrating feeder with a finger-type grizzly section.
- The crushed ore (-7") is transported to holding bins via a three-section inclined conveyor system, ranging in slope from 18% to 22%.

- The ore is loaded into narrow guage 10-ton ore cars and hauled 5.5 km to the pyrite stockpile and an additional 7 km to the secondary crusher (copper ore only).

The open-pit mine uses conventional loaders, trucks, and shovels. The waste is hauled to the surface via trucks. The ore is transported to the lowest level in the pit and then into a tunnel for about 500 yards where the dump pocket is located.

Current annual production is running at:

- 3,000,000 tons chalcopyrite and calcocite ores
- 1,750,000 tons of gossan

From the above ore, besides copper and pyrite, they produce:

- 315 tons of gold
- 28 tons of silver

B.2.3.4 Mining Equipment

Specifications on the equipment currently in use are presented below:

- 35-ton dump trucks for open-pit operation
- Two vibrating finger grizzlies
- Two jaw crushers - 3' x 4", 200 hp, 180 mm discharge, 250 tph capacity
- Conveyor network designed for 500 tph capacity. Series of four inclined conveyors running to chutes for rail loading.
- Rail - 12.5 km track length, 72 10-ton cars

B.2.3.5 Durability, Maintenance, Manpower, and Energy

Due to the problems at the open-pit (mentioned in the introduction to this overall section on Rio Tinto and discussed in greater detail in the section after this), extensive

long-range planning and on-going modification to the current operation were quite evident during our visit. Acknowledging that expansion and construction programs were in progress, the "housekeeping" at this location was admittedly very poor. For this reason, it is difficult to make an intelligent assessment of the mine on the issues normally discussed in this section.

The new open-pit on the other hand, is a different case altogether. If this open-pit operation is an indication of what the management can accomplish with proper planning, there is little doubt that the old open-pit will once again become a productive model of combined pit/underground activity. For example, some of the more notable features at Cerro Colorado include the following *(Ref. D.3.1)* © *World Mining, 1971:*

- When the new mine complex manager was mining superintendent from the start of the mine planning until June, 1971, he made several long inspection tours to the porphyry pits of the United States, South Africa, and Canada. He studied and observed equipment and operational methods at many pits and he incorporated many of the best practices in the new mine complex.
- "A centralized motor pool operation services all diesel- and gasoline-powered equipment--fueling, greasing, changing oil, tire inspection, etc.
- "The garage is sectionalized for separate truck, dozer, loader, and drill repairs.
- "All major repairs, such as engine and motor rebuilding, are done at factories or specialized shops away from the mine.
- "The garage facility has been built large enough and high enough to service 150-ton trucks if they are ever purchased.
- "During the early phases of mine development, all maintenance was contracted to the distributor of the new equipment for which he was paid a fixed fee per operating hour to keep each machine in good mechanical repair.

- "All trucks, bulldozers, wheel loaders, and road graders are Caterpillar. A separate spare parts warehouse is stocked and staffed by Caterpillar employees. Rio Tinto Patino pays only for parts taken from this warehouse and doesn't have to advance capital to maintain its own inventory. When a part which is not in stock is needed, it can be located by Caterpillar's computer and immediately shipped from the closest available supply depot.
- "The use of outside suppliers extends to tires which are repaired, not recapped, by the manufacturer.
- "ANFO blasting agent is purchased on an in-hole loaded basis from Union Espanola de Explosivos S.A., which drives its own mixing and metering truck right into the mine.
- "Extensive training courses were held for local men, many of whom had never operated heavy equipment."

In many respects, the old open-pit has the same potential to become a smooth and highly productive operation, but the facelifting effort will be long and involved.

B.2.3.6 Risk Areas and Technology Gaps

The most pressing problems for the old open-pit at the time of our visit were the following:

- The steep inclination of the haul roads combined with the moisture and slate surface over a portion of the roadways inhibited efficient operation of the truck fleet.
- The high moisture content of the open-pit ore results in "bulking" and "hang-ups" of the material in the ore passes and surge pockets. This also affects the loading of the haulage trains and subsequently increases the cycling time.
- The current rail system is not delivering ore to the mill in an efficient manner. In addition to the problem of wet ore, the rail system from underground actually transports the ore away from

the mill at first before looping back, thereby necessitating twice the track mileage than normally called for (see Figure B-20).

- The present haulage system actually has more tonnage capacity than the mine can produce.
- The acid-like nature of the ground water produces a high rate of wear and tear on the rubber-tired pit equipment as well as necessitating high track maintenance underground.

Planning is already underway to solve the more serious problems as follows:

- Modification of the underground conveying system between the crusher and the rail load point which would bring the ore directly to the surface for a short, horizontal rail haul.
- Possible use of overland conveyors directly to the mill.
- Redesign of the pit in regard to haul roads and with consideration of pit enlargement for future operations.
- An increase of 100% in mill capacity to accompany underground modifications already underway.

B.2.3.7 Rationale for Mining System Selection--Constraints and Economics

The nature of the ore deposit configuration (pyrite close to the surface, copper-lead-zinc mix further down) led to the combination open-pit/underground mining system now in use. The overall waste-to-ore ratio has been running at 2.5:1 with production costs for pyrite running at approximately $11/ton, which is only slightly within the pits economic guideline for a profitable operation. Future expansion and facility upgrading plans (such as the relatively new inclined conveyor) should improve operations considerably.

As for the decision to maintain a combined open-pit/ underground operation, the narrow, constrained nature of the pyrite deposit was the determining factor when the present underground haulage system was installed several years ago.

B.2.3.8 Conclusion

This Spanish mine is clearly and example of the type of ore deposit which lends itself readily to underground crushing and haulage. Once improvements have been made to the current operation, it will be a facility well worth visiting to review the economic potential of continuous underground crushing and haulage.

B.2.4. Lead-Zinc Mine - Ireland

An open-pit mining operation was the only activity envisioned when planning efforts commenced after a limited exploration effort. It was not until after surface facilities had been commissioned and erected that the full extent of the irregular shift in the lie of the orebody was realized through a detailed investigation of the normal fault on which the host rock was situated. Physical limits to further pit development (due particularly to the proximity of the tailings pond) dictated the development of an underground operation for the deeper ore reserves. Under these circumstances, a thorough analysis of the potential mining alternatives was undertaken, resulting in the current economical operation which uses underground crushing and vertical skip haulage.

Open-pit activity ceased as of 1973 except for a limited backfilling operation. The underground mine waste, after separation from the ore in the surface mill facility, is redirected through the pit bottom ore passes to the active stopes. Figure B-22 provides a panoramic view of the pit as it exists now.

Table B-5 summarizes the key features of the current mining operation and points out the major problems experienced in converting from surface to underground. The following sections provide a detailed account of the mine development activity and conclude with a discussion of the evaluation process which selected the current underground mining scheme.

B.2.4.1. General Mine History and Orebody Characteristics

The Lead-Zinc Mine belongs to Irish Base Metals Ltd., a subsidiary of Northgate Exploration Ltd., and is situated in County Galway, 115 miles due west of Dublin (see Figure 2-10). Current production, which comes totally from underground, is 600,000 tons per year of ore grading 7-8% combined lead-zinc and one oz silver per ton, along with minor copper. Mill tailings are sold for recovery of barite. Sublevel stoping is the only mining method being used.

The orebody is emplaced in a limestone host rock on

TABLE B-5

FACT SHEET ON THE LEAD-ZINC MINE-IRELAND

- Owner - Irish Base Metals, Ltd.
- Location - 115 miles west of Dublin, Ireland
- Ore - 3.89%, 3.75% Zn, .18% Cu, .9 oz Ag. Barite in mill tailings
- Pit Operations Started - 1965
- Underground Operations Started - 1968
- Pit Operations Ceased - 1973
- Production - 600,000 tpy
- Potential Mine Life at Present Production Rate - 2½ years (as of Jan. 1978)
- Operating Personnel - 346 (includes 121 underground miners)
- Operations:
  - -Single ore pass from surface to crusher (7 ft. diameter, 500 ft. deep)
  - -Multiple ore passes from underground mining sections to crusher
  - -Skip hoisting of crushed ore to surface
- Equipment:
  - -Ore haulage via LHDs to crusher
  - -Ore and waste are hoisted to surface by 10-ton skip
  - -Unique design for a dump gate at the raise to protect both the trucks from falling in as well as collar caving
- Problems:
  - -Ore would hang up in raise due to its soft, easily compacted nature
  - -Vertical drop of ore damaged chutes at bottom of raise; required conversion to draw points without chutes



Figure B-22 View of Lead-Zinc Open-Pit and Entrance to Underground Mine

the hanging wall side of a limestone/sandstone normal fault, as shown in Figure B-23. The upper part of the two main ore zones is shallow and secondary in nature, occurring in a residual form originating from the weathering of a heavily mineralized limestone. The deeper primary ore consists of sulphide mineralization in a favorable 'reef' limestone horizon. Maximum fault displacement is 700 m, occurring in the location of the orebodies.

Originally, the orebodies consisted of three main classifications: secondary sulphides, secondary oxides, and primary sulphides. The secondary sulphide and oxide zones lay beneath a 15- to 50-foot mantle of glacial till and boulders, and was mined by open-pit methods (completed in 1973).

The primary sulphides are located beneath and adjacent to the open-pit, with proven current reserves amounting to

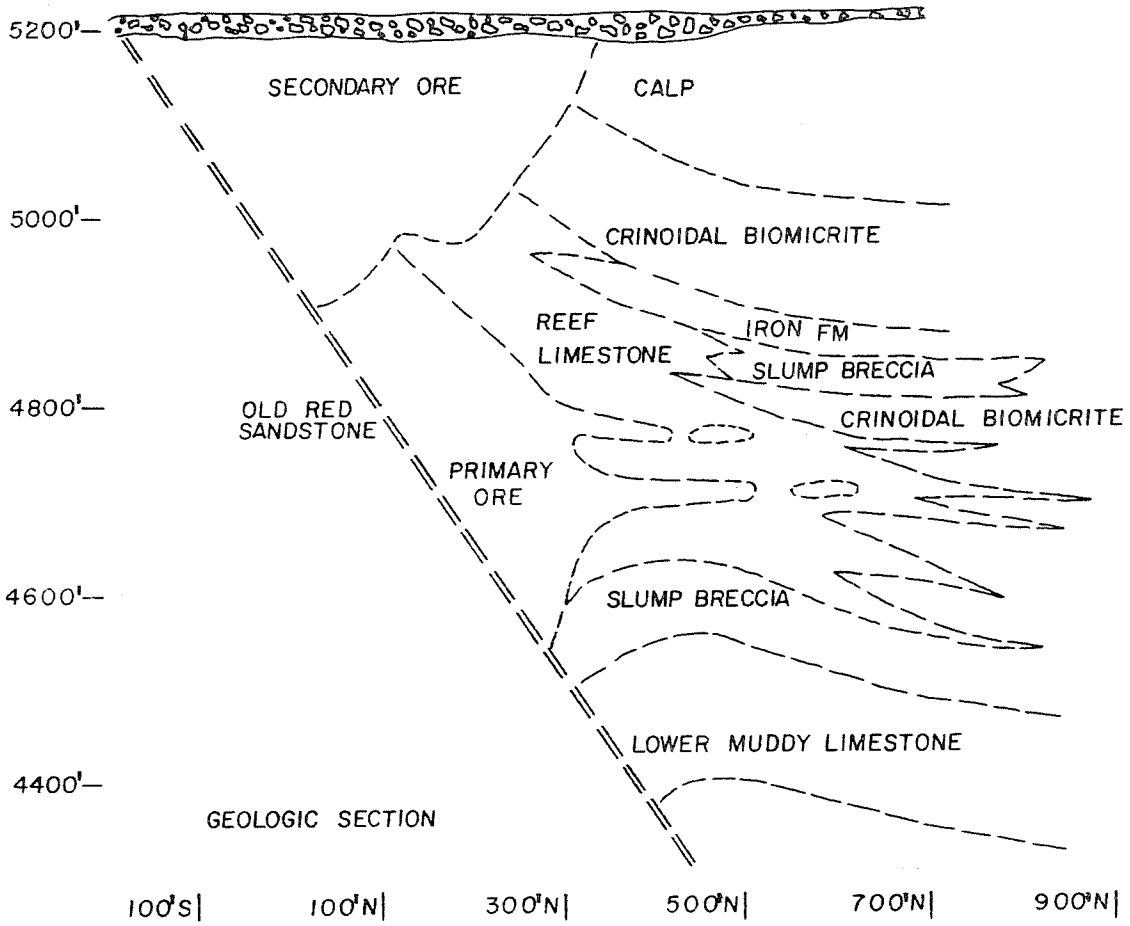


Figure B-23 Geological Cross-Section of Lead-Zinc Orebody

1.5 million tons. The final output of the underground mining operation will just about equal the 4.5 million tons of secondary and primary sulphide ore mined in open-pit.

B.2.4.2 Mine Layout

The now defunct open-pit is 2,500 x 500 feet in plan and 350 feet deep. Access to the underground workings is provided by a 15% decline which is also used for the transport of men, material and trucking development waste to the surface. Ore is handled by means of an underground, crushing, conveying and hoisting system. The main shaft is a 7 foot diameter raise, 1,750 feet in depth, employing a skip and counterweight installation operated by a fully automatic hoist.

In 1968 underground development commenced in the pit bottom with the excavation of the decline portal (15 x 9 feet in cross-section, 3,500 feet long, on a maximum grade of 15%). Over a four-year period, the ramp was taken to a depth of approximately 750 feet below surface. All development was performed within the limestone hanging wall (outside the orebody) in extremely competent rock. Figure B-24 illustrates a cross-section of the overall layout.

Upon completion of underground development, a 7 foot diameter raise was drilled on a 68^{o} angle from the open-pit to a depth of 500 feet to provide a raw ore pass. A single-stage, 36 x 48 inch primary jaw crusher was installed at a depth of 700 feet and fed through a special 7 foot diameter raise connected to the 600 foot level where the drawpoint for the raw ore pass was being serviced by LHDs.

The underground mining operation is serviced by three drifts at depths of 300, 400, and 500 feet below surface. There is a main track haulage drift at a depth of 500 feet and a scram loop at 600 feet. Crosscuts at right angles to the three upper drifts are driven to the orebodies (all are 15 x 9 feet in

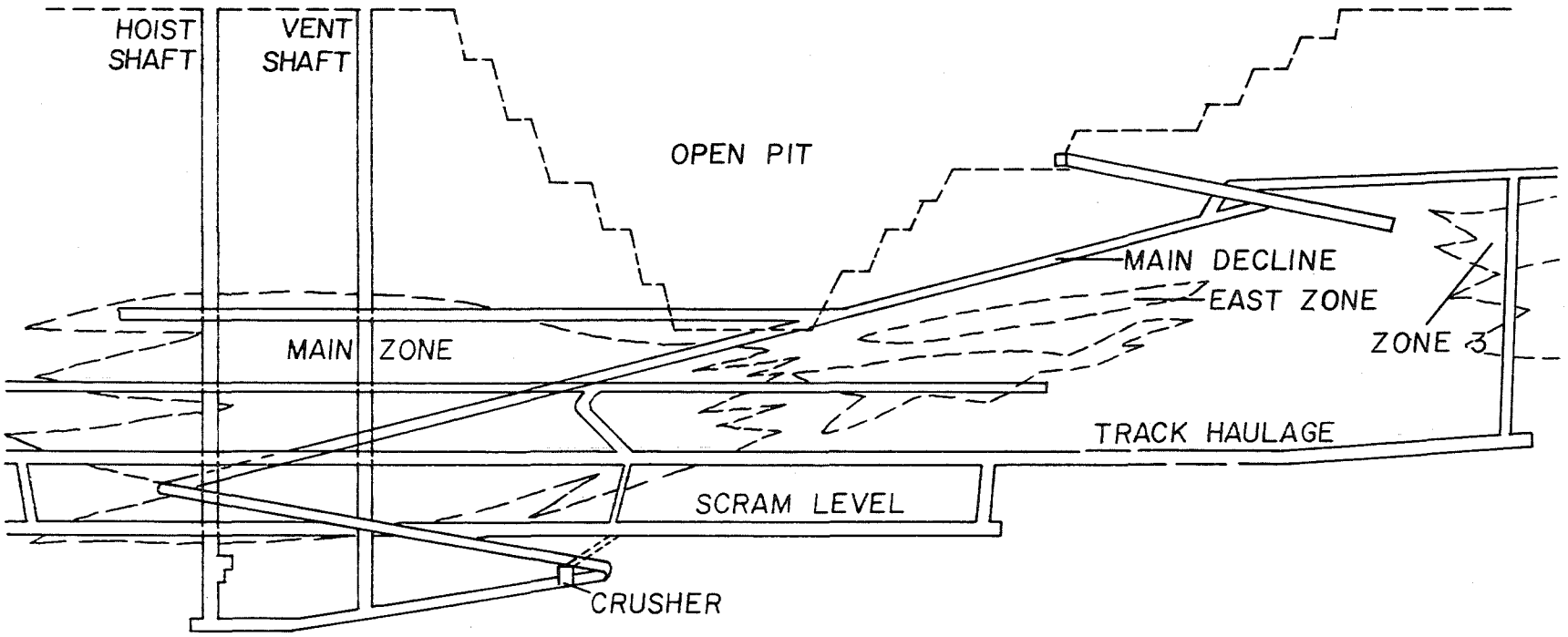


Figure B-24 Cross-Section of the Overall Mine Layout - Ireland

cross-section). The stopes run east-west, paralleling the service drifts, are 100 feet between levels, and up to 150 feet square. Seven-foot diameter slot raises connect the levels to provide for ore passage.

To accomplish ore haulage to the surface, an 800 foot deep shaft was raise bored to 7 foot diameter and equipped with a single skip. The loading pocket is 715 feet below the dump point. The system is supported by a 115 foot high steel headframe. The underground ore movement network is illustrated in abbreviated form in Figure B-25.

In addition to the production shaft and decline, a second 750 foot deep, 7 foot diameter shaft has been installed for services and as an emergency exit.

B.2.4.3 Minning Operation

Open-pit mining started in 1965 with the objective of retrieving all secondary zone ore. This provided sufficient reserve to keep the mill in full operation through 1971 when primary ore was needed. The resulting total output was 4.5 million tons by 1975. Conventional truck and shovel operation accomplished the open-pit mining until startup of underground production during the transition phase in 1972. By 1975, all activity had moved underground. Figure B-26 provides a complete layout of the underground operation, with ore passes.

The underground orebodies have been grouped into three distinct ore zones for mining purposes. The principal zone is mined by a transverse sublevel blasthole stoping method with the ore being handled on the haulage level by LHD units which dump directly into the underground crusher. Other smaller flatter zones are mined by random room and pillar mining, again using LHD units. The third zone is remotely situated and, although LHD equipment is used locally, the method of transport from this section to the underground crusher is by conventional track haulage over a maximum distance of 2,800 feet.

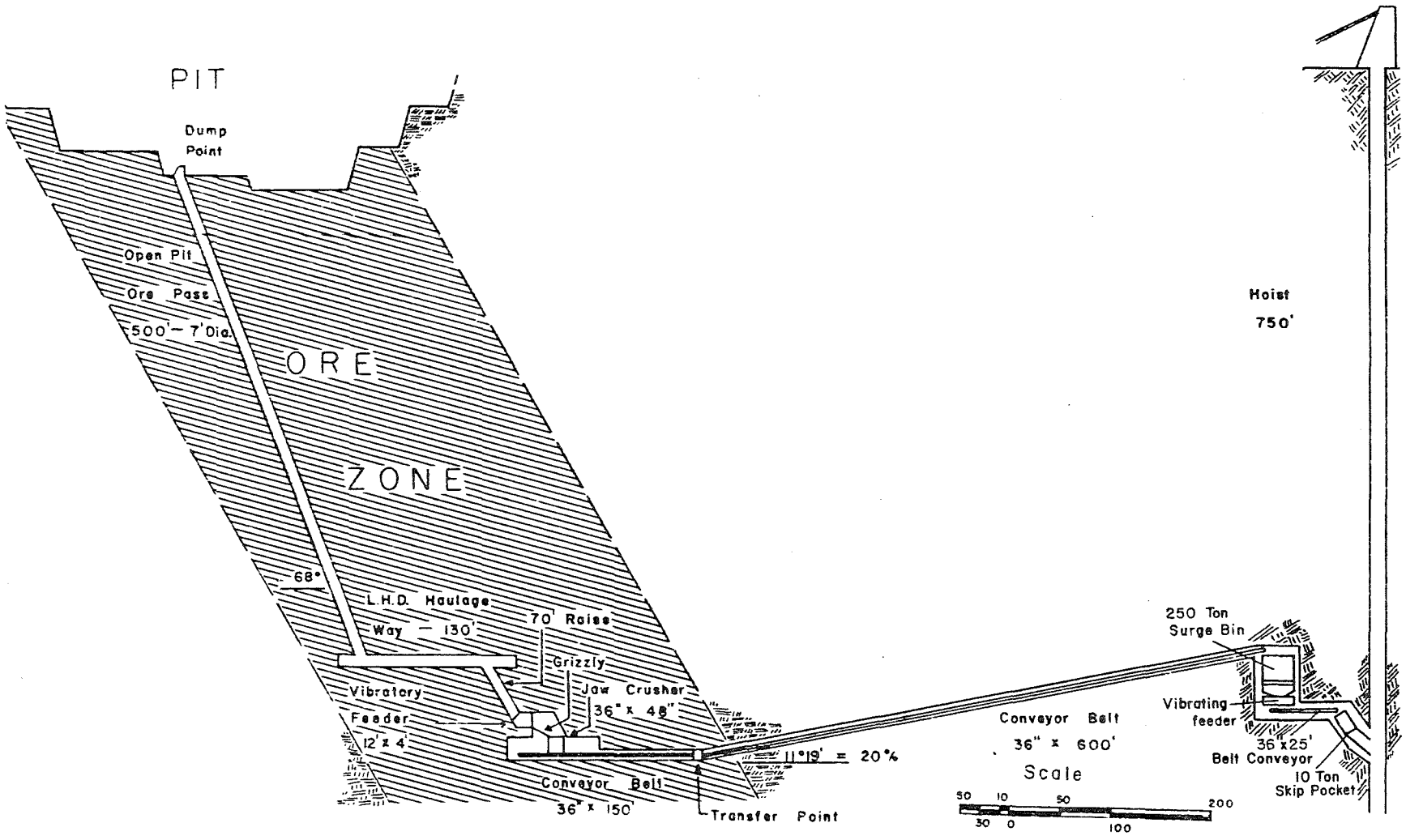
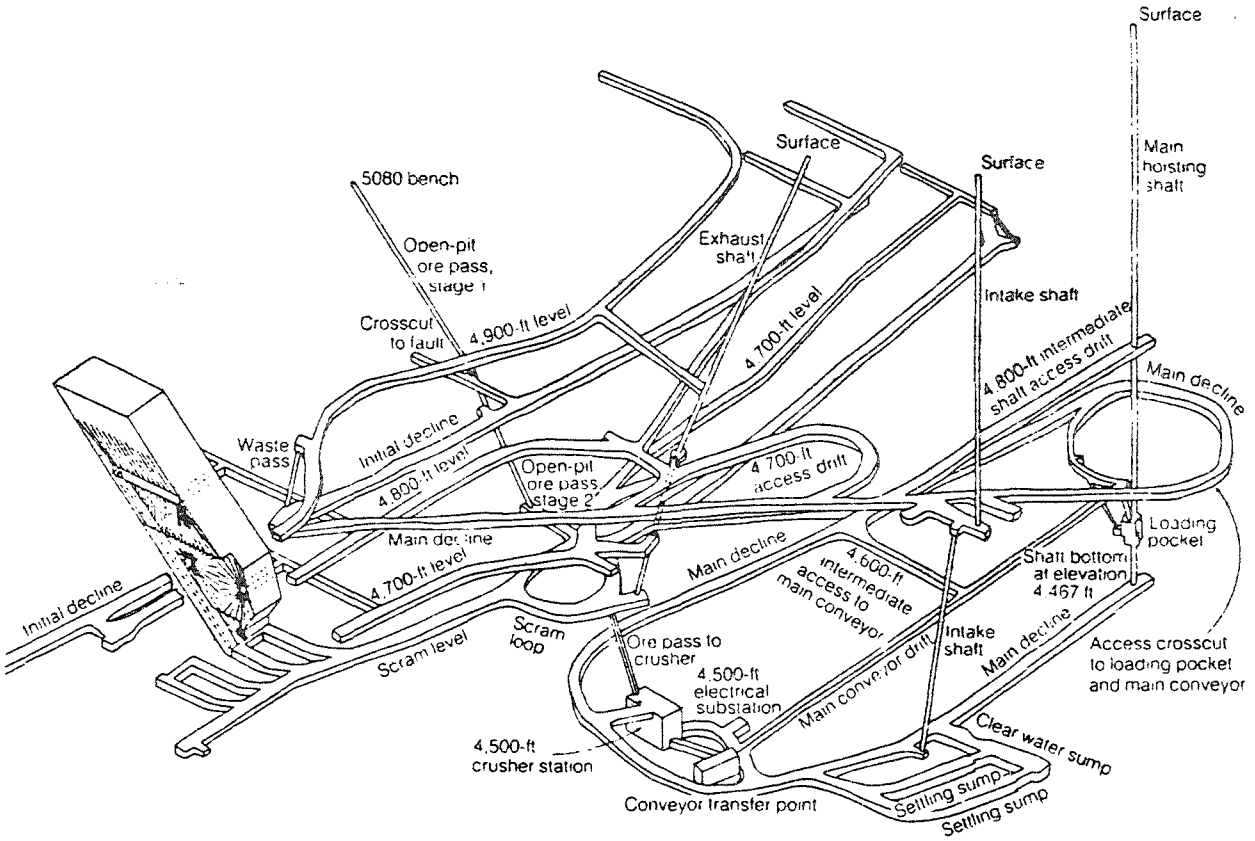


Figure B-25 Cross-Section of Crushing and Haulage From Open-Pit



(Note. Elevations are based on a vertical coordinate system that puts the surface, 200 ft above sea level, at elevation 5,200 ft.)

Figure B-26 Isometric Projection of Mine Development at Lead-Zinc Mine - Ireland (Ref. D.4.1)© EM/J, 1977

Both the ore and the development waste are commingled through the underground haulage and hoisting system and separated only at the dense media separation plant at the surface. After excavation, the mined-out material is either moved by LHD or is side-dumped from rail cars into a 7 foot diameter x 70 foot long ore bin discharging onto a mechanical vibratory feeder ahead of the jaw crusher. From the crusher, the material is conveyed over a 36 inch wide, 750 foot belt to a 250 ton capacity storage bin. The bin loads a vibratory feeder discharging to a belt and then a 10 ton measuring skip pocket. The skip, mounted on load cells, loads the skips in the production hoist which takes the material 715 feet to the surface.

Waste from the media separation plant is returned to the open-pit and passed back underground for use as fill in the stopes.

B.2.4.4 Mining Equipment

Horizontal haulage equipment includes LHDs, rail cars, and dual 750 foot conveyors between crusher and hoist mentioned in the preceding section. The LHD fleet consists of eight 915's, two 912's, and two ST 8's. On each production shift, five loaders are assigned to development headings and three to the stopes. (Six headings are in development at any one time.) Rail equipment consists of two locomotives and thirteen cars. One locomotive and 10 cars are used per shift.

Vertical haulage is accomplished by a single drum skip hoist with 10 ton capacity and bottom dump capability. Hourly throughput has averaged 300 tph. Major features of the hoist include a 500 hp motor drive and a 7 ton counterweight. The maximum unbalanced load is 14,100 pounds. Rope diameter is 41 mm and hoist speed is 1,200 fpm.

Underground crushing is handled with a 36 x 45 inch jaw crusher which discharges a nominal 6 inch product. Two gyratory crushers at surface level further reduce the feed to 1½

inches for the heavy media separation plant. Daily throughput capacity has been on the order of 1,800 tons.

All shaft bottom facilities including the crusher station are equipped with dust collection equipment or water sprays.

Mine ventilation air (400,000 cfm), taken in through the decline and production service shafts, is evenly split between the production zones. The air is exhausted by two fans located at separate return air raises to the surface above the same zones.

Underground water is a significant problem, requiring a 400,000 gallon capacity mine sump below the crusher station equipped with two 500 gpm and one 1,500 gpm 10 stage centrifugal pumps.

A noteworthy innovation by this management was the installation of a special swing gate arrangement at the entry to the pit ore pass (see Figure B-27). Not only did it provide a solid barrier which the haulage trucks could back up against, but it also kept the ore pass sealed when dumping was not taking place.

B.2.4.5 Durability, Maintenance, Manpower and Energy

Maintenance of equipment is done above ground. Mobile equipment is serviced in a shop near the portal. Because the mine has limited life expectancy at this point, the maintenance effort on the underground facilities, as opposed to the equipment, is virtually nil--certainly not the best of situations from a safety standpoint.

For the same reason, the overall operation does not lend itself to ready analysis of manpower and energy. The mine management is expending the minimum effort to keep the underground activity functional. Barring any potential catastrophe, the operation will undoubtedly be profitable, although not under the healthiest of circumstances.

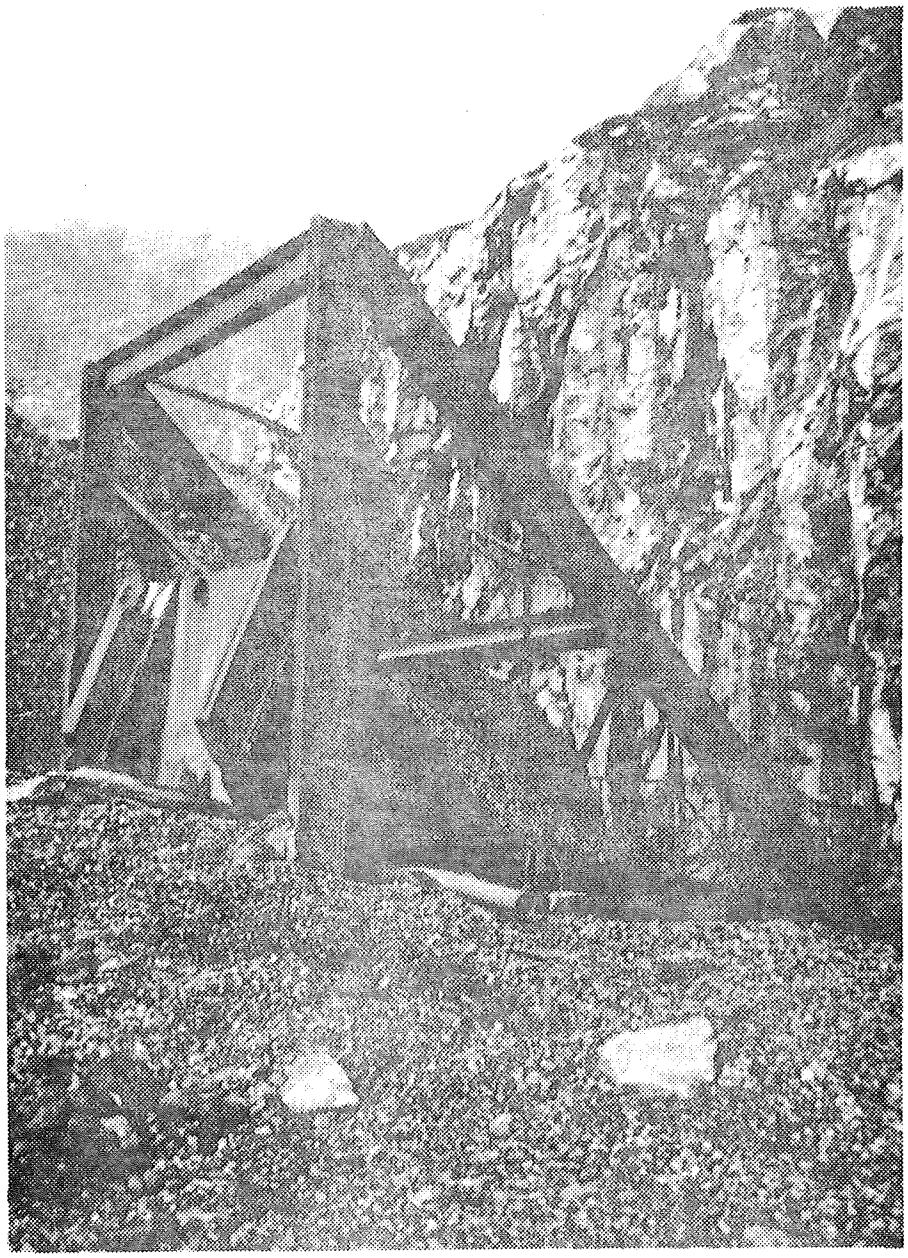


Figure B-27 Specially Designed Ore Pass Barrier Gate in The Bottom of the Open-Pit at the Lead-Zinc Mine - Ireland

B.2.4.6 Risk Areas and Technology Gaps

This mine experienced major problems with the ore passes, one of which related to the chute arrangement at the bottom. Because the raise was straight, the drop of the material would damage the chute. In addition, if ore were allowed to remain in the pass, it would easily compact due to its softness and hang up, thereby requiring blasting. The problem was overcome by removing the chute, allowing the material to collect on the haulageway floor and using an LHD for transport of the ore to the crusher station. According to management, if the project were to start today, they would insist on a dogleg in the ore pass to break the fall of ore before it reached the bottom. They felt the diameter of 7 feet was adequate.

Efficient milling of the ore was adversely impacted when the open-pit and the underground ore were mixed. This was due to hardness differences between the two ore types and did not result from the particular mining system employed.

Perhaps the most important risk area to single outat this mine is the fact that an inadequate planning effort prior to any mine development activity ultimately forced the operation underground. The extreme distortion in the geological deposition of the orebody was not appreciated until after the open-pit was in full production. Nonetheless, given this unfortunate circumstance, mine management was quick to evaluate alternatives with a more methodical approach than before. Before a full capital development program was committed to an underground mining operation, further detailed examination of the ore bodies had to be made and bulk samples collected for metallurgical test work. This established the possible mill recoveries and the alterations that were required in the existing surface plant to make it suitable for the treatment of primary ore.

Gaining the required exploratory information became possible through construction of the initial decline to the future underground workings, which was laid out to give

quick access to the ore zone in question. Subsequently, a program of underground diamond drilling was carried out from these workings; the required bulk samples were obtained for metallurgical test work; and a detailed sampling/assaying program was carried out to determine the best sampling method and to cross check the grades obtained from the earlier surface diamond drilling.

Once the decision to go underground had been made, another risk area that had to be confronted right away was the lack of underground mining expertise throughout the entire staff. Consequently, an arrangement was made with a Swedish contracting firm to supply an underground foreman and 12 miners to form the nucleus of the underground crew and to train the local men as miners. This training program considerably slowed up initial progress on the development as well as adding to the expense. Consequently, it took nearly two years before all the Swedish mining instructors were phased out. This mine, however, now has a full complement of well-trained miners from the locality and the turnover of workers is very low.

B.2.4.7 Rationale for Mine System Selection-- Constraints and Economics

Given the choice between closing the mine down altogether, and exploring various expansion options, management elected to do a detailed evaluation of four possible alternatives:

- A deeper extended open-pit operation.
- An underground mining operation without crush- and utilizing a main shaft with skip hoisting of ore and cages for servicing. Underground transport to be by track haulage with loading of ore in the scrams to be handled by slushers.
- A decline equipped with a conveyor belt to be used for both access and hauling ore to surface. Ore handling underground is to be by LHD.

- A decline to be used for access and a shaft for skip hoisting of ore. LHD units would be to handle ore underground.

A deeper open-pit was rejected because of the high stripping ratio required, the inability to mine all the ore zones, and the associated costs of relocating the surface facilities. The main points favoring the fourth alternative were:

- The initial capital cost was less than the other systems.
- By driving an initial decline, quick access to the ore zones would be possible. This would allow a full examination of the ore zones before the major part of the capital investment was committed.
- The method allowed for a high degree of mechanization. This was an important factor as no skilled miners were available in the area.
- An early start could be made on training underground miners.

Although the management was unable to provide us with the economic background to the decision-making process for reasons of confidentiality, we assume that profitability of the overall operation was maintained. Our reasoning relates to the fact that an extensive capital development program of this magnitude would never have been undertaken otherwise.

B.2.4.8 Conclusion

This Lead-Zinc Mine is an excellent example of a mining operation that converted from surface to underground and remained profitable, despite adverse circumstances such as lack of qualified manpower skills. Moreover, although the total mine life will have been relatively short when operations are discontinued in 1980, the return on investment has justified an extensive underground development effort, including the acquisiton of a raise boring machine to construct the shafts and ore passes.

Given the disadvantages of inexperience combined with heavy capital outlay, we found this Lead-Zinc Mine an encouraging case in point for justifying the potential success of open-pit operations combined with underground development, ore crushing and haulage.

B.2.5 Cananea Mine--Mexico

In contrast to the four mines already discussed, the Cananea Mine was a long-standing underground operation which started to convert to open-pit in 1944, but retained the use of an underground crushing and continuous haulage system during and after the transition. Many years of underground experience handling relatively coarse muck through long ore passes had suggested the idea of placing the crusher underground beneath the ultimate bottom of the pit, and using an inclined conveyor to transport the ore to the surface. In addition, investigations into low-cost ore transport and lifting of crushed rock produced in the surface crushing plants had convinced management that conveyors were the most economical haulage medium for their particular needs.

Although most of the open-pit development and underground equipment at Cananea are relatively new, maintenance problems have inhibited the operation from reaching full production potential. But even with recognized design flaws in mine layout added to the maintenance difficulties, tonnage output has still exceeded by 20% the estimated level that would have been achieved with truck haulage. This is strong justification for the decided advantages which continuous haulage can bring to open-pit operation.

Figure B-28 shows an aerial view of the present pit. Table B-6 summarizes the major features of the current operation. This section amplifies further upon the information contained in the table and gives detailed background on the decision-making process leading to the selection of underground crushing and conveying (see Section B.2.5.7).

B.2.5.1 General Mine History and Orebody Characteristics

The Cananea open-pit mine is located at an elevation of 5,200 feet in the north-central part of the state of Sonora, Mexico, about 30 miles south of the United States

TABLE B-6

FACT SHEET ON THE CANANEA MINE

- Owner - Compania Minera de Cananea, S.A. (51% - Mexican Government, 49% - Anaconda Corporation)
- Location - Sonora, Mexico, 150 miles south of Tucson
- Ore - .76% Cu
- Mineralization - breccia pipes, granitic intrusive rocks
- Underground Mining Started - 1760
- Open-Pit Mining Started - 1944
- Production - 20,000 tpd milling; total mining capacity 80,000 tpd
- Potential Mine Life - 75 years
- Operations:
  - Ore from the open-pit goes down through a raise to an underground crusher system, and then via inclined conveyors to the concentration mill.
- Equipment:
  - 120-ton dump trucks in the open-pit
  - Underground jaw crusher
  - Combined underground-overland inclined conveyor network
- Problems:
  - High maintenance costs for the underground facilities
  - Underground facilities located in rich section of ore deposit
  - Underground crusher located too close to pit bottom
  - Erosion in the ore pass due to soft rock
  - Initial high acid water seepage in ore pass and crusher room.

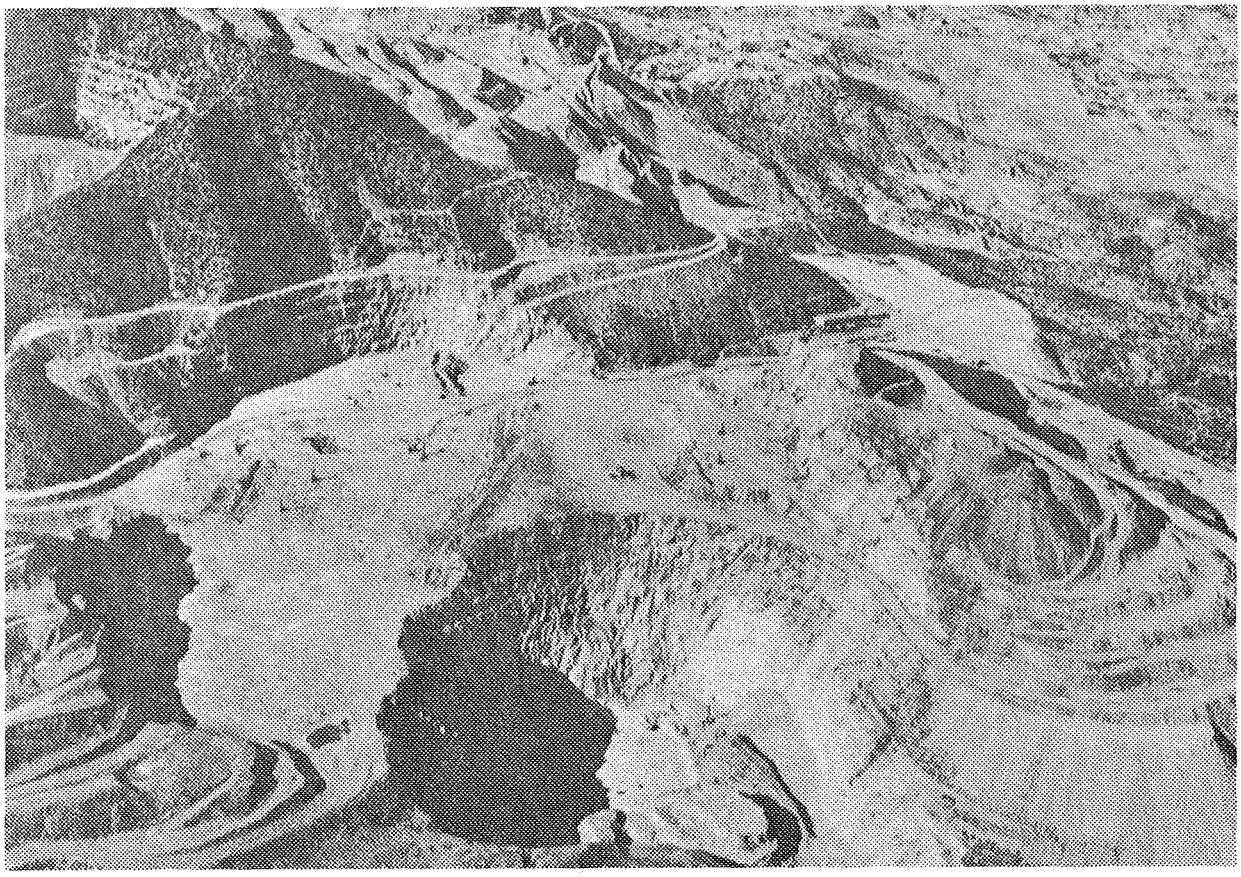


Figure B-28 Panorama of the Cananea Open-Pit Mine (Ref. D.5.2)
© AIME, 1969

border, and 150 miles south of Tucson, Arizona (see Figure 2-11 The Compania Minera de Cananea S.A., a Mexican government-controlled organization, owns 51% of the mine (Anaconda Corporation owning the other 49%) and generates electric power for its own installations, as well as supplying both electricity and water to the town. Fuel for the power plant, smelter, and town is natural gas, supplied through a pipeline from the United States.

The mineralized district is approximately 6 miles wide and 15 miles long, and the orebodies are associated with breccia pipes and granitic intrusive rock. In the early days, many small mines operated within the area on high-grade orebodies. Through the period from 1900 to the beginning of World

War II, high-grade orebodies of variable size were mined, the largest being the Colorada which produced more than 7 million tons of ore containing about 7% Cu and 0.4% Mo. By the start of the war, these high-grade orebodies were nearly exhausted.

A large low-grade body of ore was known to exist above the Colorada pipe. Toward the end of the war, a new 12,000 tpd concentrator was built and the low-grade orebody above the pipe was prepared for mining by open-pit methods. The expanded facilities for the recovery of copper proved that the low-grade ore could be mined economically and pointed the way to the current large-scale operation.

The present mining and milling operation extracts an average .76% grade copper ore, with cutoff for the mill set at .4%. A leaching operation on the waste ore (average grade .15% Cu) produces a 60-65% grade copper.

B.2.5.2 Mine Layout

The total Cananea complex consists of:

- A large open-pit mine capable of moving 80,000 tpd of material with a shovel-truck, underground crusher, and belt-conveyor haulage system.
- A 30,000 tpd concentrator utilizing four stages of crushing, rod mill/ball mill grinding with cyclone classification, followed by flotation in conventional machines.
- A dump leaching process producing one million pounds Cu per month in a tin can scrap precipitation plant.
- A two-reverberatroy, 6-converter smelter which can make 140 tpd of blister copper.
- Supporting departments, running the gamut from housing to power generation, which service these operating groups.

Figure B-29 shows a schematic of the present ore haulage network from mine to mill. A vertical ore pass, 30 feet x 40 feet in cross-section (originally 20 feet x 30 feet), connects the open-pit to the underground crusher room. The pass was initially 600 feet in length, but mining activity over the past decade has reduced the drop to only 70 feet as shown in Figure B-30.

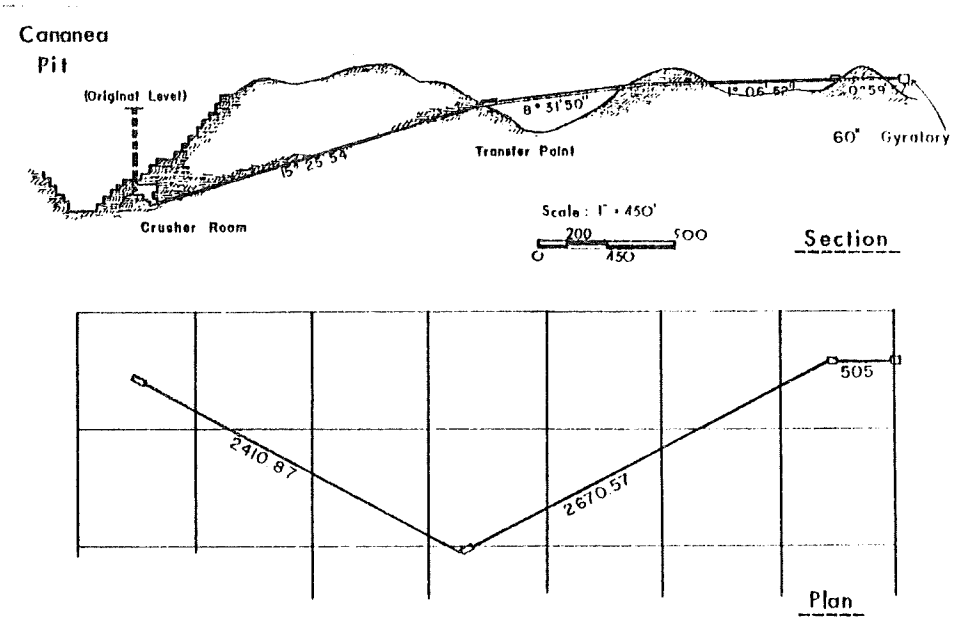


Figure B-29 Layout of the Ore Haulage Network from Open-Pit Mine to Mill at Cananea (Ref. D.5.2) © AIME, 1969

The minimum-sized crusher room was designed to stand open without concrete or steel support. A rock-bolted arched roof proved sufficient. The schematic section in Figure B-31 shows the room's critical dimensions of 125 feet in length, 26 feet in width, and 125 feet in depth. An interesting account of the construction of this facility is described in *Ref. D.5.1. © AIME, 1969*

The underground 48" conveyor running between the crusher and the surface is slightly over 2,400 feet long and inclined at

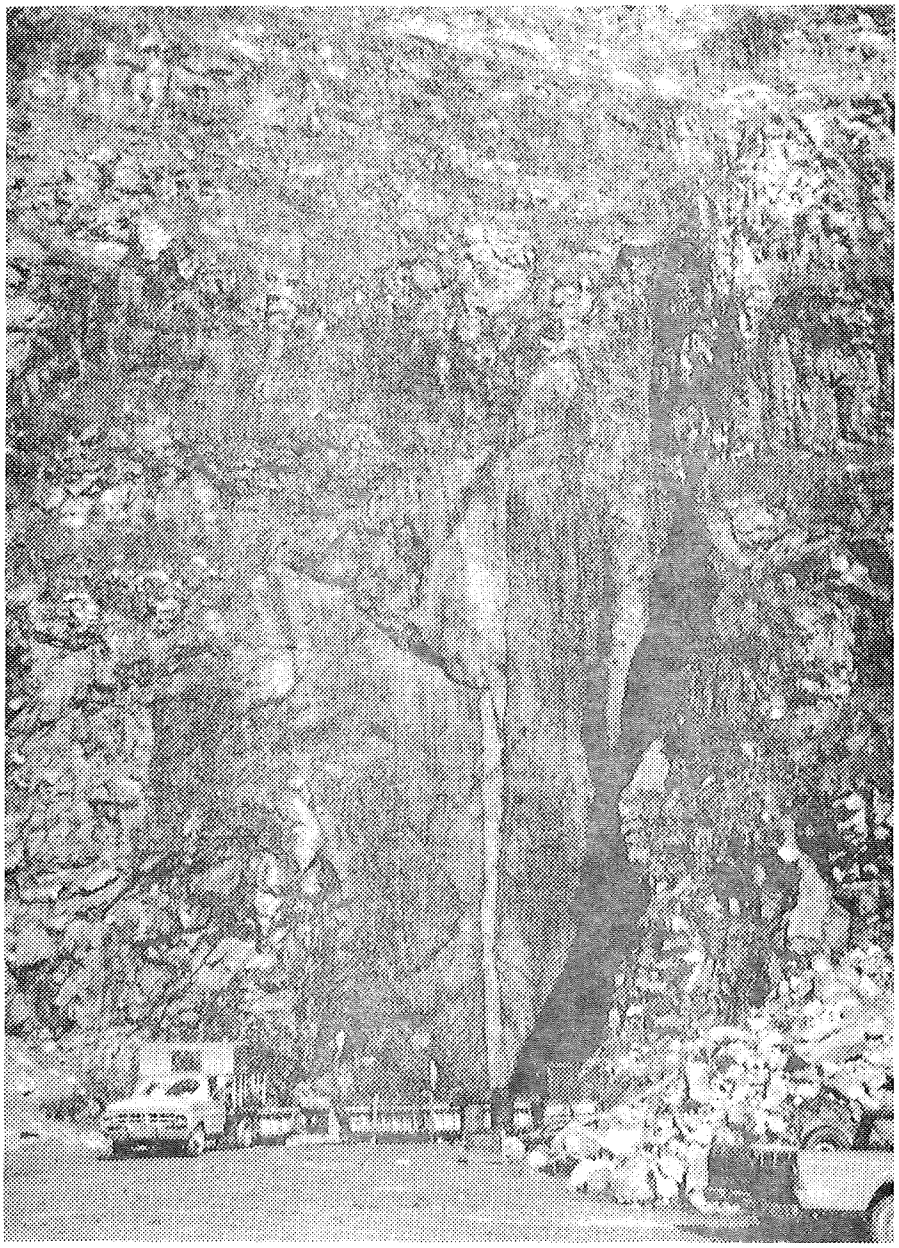


Figure B-30 Cut Away Ore Pass Running from Open-Pit Bottom to Underground Crusher Room at the Cananea Mine, Originally 600 Feet Long, Now Only 70 Feet.

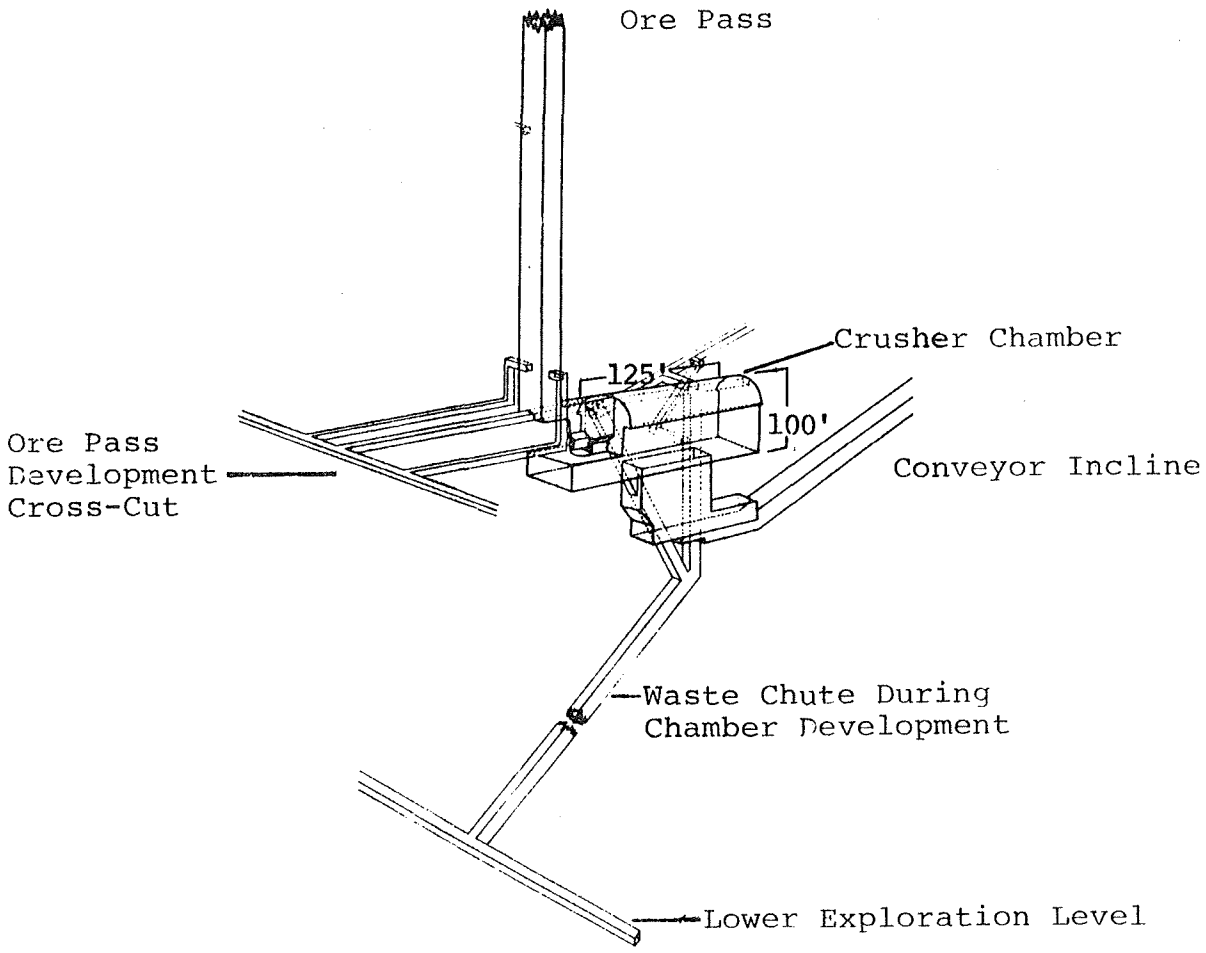


Figure B-31 Diagram of Underground Crusher Room at the Cananea Mine (Ref. D.5.2) © *AIME, 1969*

an angle of 15.5°. The conveyor's transfer point is at the top of the slope. Figure B-32 shows the point along with the maintenance skip. Two additional overland conveyor sections in series with the first connect the haulage network to a discharge point near the mill as shown in Figure B-29. Total conveyor system length exceeds one mile.

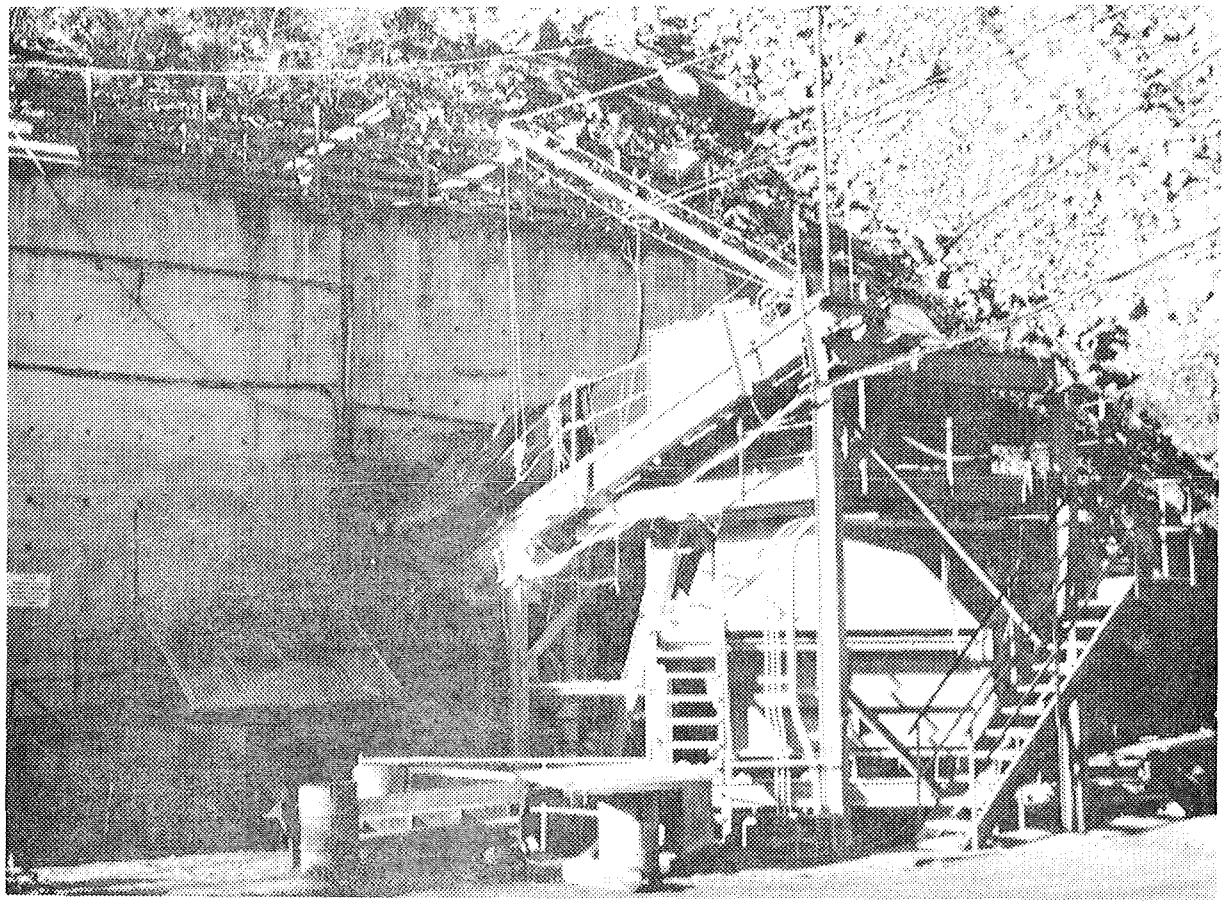


Figure B-32 Slope Conveyor Transfer Point and Maintenance Skip Beside Conveyor

B.2.5.3 Mining Operation

Ore has been mined from several pits in the sprawling mineral district around the Cananea mine which is the largest in the area and has the only underground crusher installation. Present milling capacity is 30,000 mtpd with mining operation capacity of 80,000 mtpd; both are running somewhat below design capacity due to the depressed market for copper. Estimated reserves should allow the Cananea operation to continue for another 75 years at the present mining rate.

Current open-pit activity centers on the underground crusher system which was installed in August, 1966 (see Figure B-33). The ore pass to the crusher feeds a maximum size 5 foot x 5 foot x 5 foot ROM material to a 10 foot x 26 foot variable speed apron above an 8-foot wide vibrating scalper and a 66 inch x 84 inch primary jaw crusher. Below the jaw crusher is a secondary pan feeder to a speed-up belt that feeds the 2,410-foot slope belt set at a 15° angle. The ore is then conveyed 5,584 feet to a 60 inch gyratory crusher upstream of the concentrator. The design capacity of this total system is 1,100 mtph.

Besides mining surface ore from the main Cananea Pit with its ore pass and underground crusher system, the other past and present pit are:

- Old Colorada Pit, East Area Sonora
- New Colorada Pit
- Kino Pit

B.2.5.4 Mining Equipment

Equipment currently being used at Cananea for ore crushing and haulage includes:

- 120-ton dump trucks
- Apron feeder - 10 feet x 26 feet, variable speed rate of 2 to 8 fpm
- Vibrating scalper - 8 feet wide, set at 8 to 10 inches
- Jaw crusher - 66 inches x 84 inches, capacity 1,100 tpd, 450 hp electric motor drive
- 60 inch x 30 foot pan feeder - variable speed
- Speed up belt
- #1 Belt - 48 inches wide x 2,580 feet long, 15° angle slope
- #2 Belt - 48 inches wide x 2,671 feet long
- #3 Belt - 48 inches wide x 505 feet long, discharges to surge bin
- 60-inch gyratory crusher located at the conveyor system discharge point

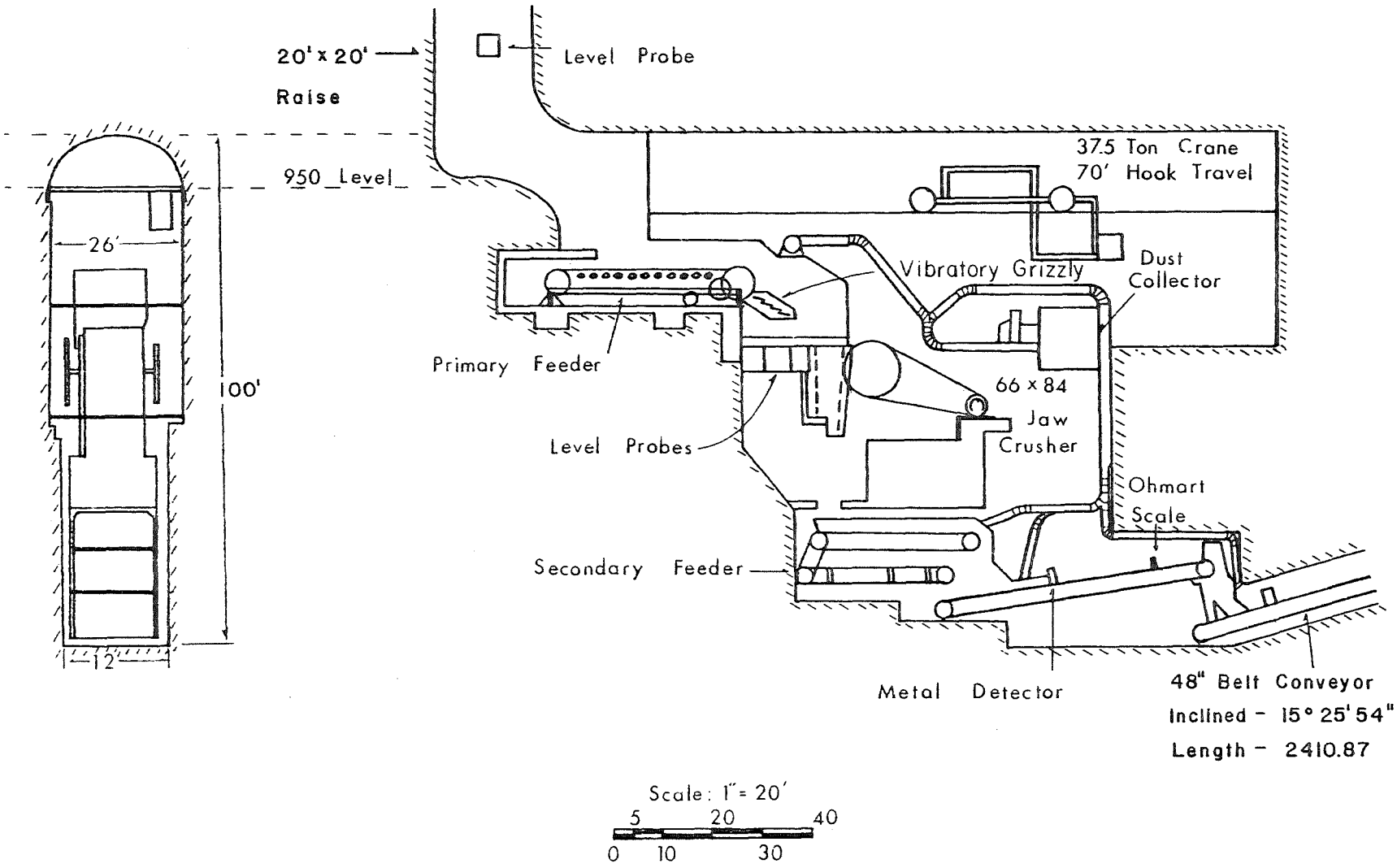


Figure B-33 Schematic of Underground Crusher Room at the Cananea Mine Mine (Ref. D.5.1) © AIME, 1969

The latest innovations of the late 1960's in interlocking safety controls and level probes have been installed, although a combination of acid water and high humidity have effectively deterred probe effectiveness, resulting in their removal. Dust collecting is of the wet type, and has purposely been overdesigned, resulting in excellent working conditions through the use of a single unit of 40,000 cfm. The major belt transfer points are unhoused and have been located above ground to avoid dust control problems. Eddy current couplings have been used on the feeders and major belts to insure smooth, even start-ups. These are a few of the small considerations integrated into the total planning effort.

B.2.5.5 Durability, Maintenance, Manpower and Energy

Maintenance appears to have been a significant problem at the Cananea Mine with management recently making a key decision to reassign responsibility for upkeep of the crusher to the Mill Concentrator Division. The high level of maintenance required to keep the underground machinery functioning smoothly has been noticeably lacking. The push is on to increase equipment availability according to management.

Compounding the above problem has been the slope into the crusher area which has a small incline hoist beside the conveyor to transport the men and supplies into the room. The hoist is very slow and takes approximately 20 minutes to travel one direction. This has lowered the efficiency of the maintenance crews for several years during which the availability of the crusher system was very poor. This situation has been improving noticeably since the crusher section was put under the Mill Concentrator Division's control.

The ore pass was grouted to decrease the amount of acid water entering the crushing area, but it later increased

in size from a 20 foot x 30 foot shaft to 30 foot x 40 foot due to erosion of the soft rock.

Underground experience had taught Cananea engineers that, when the flow of coarse rock changed direction, troubles in the form of excessive wear and stoppages could be expected. With this in mind, much thought and time were spent designing the "brow" area over and before the 10 foot wide pan feeder. Thick (36 inches) heavily reinforced concrete walls lined with 7-inch manganese jaw plates formed the main chamber, with a heavy steel beam covered with a specially designed curved manganese plate protecting the cutoff wall directly above the feeder. Even with these precautions, the brow wore considerably and had to be further reinforced with steel and concrete.

B.2.5.6 Risk Areas and Technology Gaps

As with any large-scale mining operation, the world market for ore prices can play havoc with profitability of the overall operation. In Cananea's case, the depressed copper business has raised the acceptable limits of ore grade which can be economically mined. To assess the full impact of this situation, management has determined that the optimal location for the ore pass and crusher system is a good distance from the present setup. This is clearly one of the big dangers in going underground, and must be considered during the planning stage to guarantee financial survival. In Cananea's case, the operation has made money for the time being, but the loss of flexibility in repositioning the equipment to cope with a bear market is clearly a major disadvantage.

Another critical problem looming larger on the horizon each year is the fact that the crusher is no longer positioned deep enough underground. What had originally been a 600-foot ore pass feeding the crusher has been reduced to 70 feet, necessitating the construction of a ramp for truck dumping in

order to maintain sufficient surge capacity in the raise. Figure B-30 shows the remnants of the former ore pass carved into the side of the exposed pit wall.

Perhaps the most cogent problem forcing reevaluation of crusher location is the relatively rich nature of the orebody in which the crusher facility is positioned. Management wants to take the pit deeper and the crusher lies in its way. This is certainly not part of the long-range plan from 15 years ago, but reflects how changing conditions have impacted on the earlier design decisions. Management can only hope at this point that the equipment will be used elsewhere and that the construction costs associated with crusher installation have been and will continue to be offset by the savings in operating costs.

A danger associated with underground crushing is the backing of dump trucks into the ore pass. This results in great harm to the driver and lost production time spent removing the vehicle. A better procedure for accomplishing the dumping in conjunction with a more effective retaining wall around the shaft collar is needed.

B.2.5.7 Rationale for Mining System Selection-- Constraints and Economics *(Ref. D.5.2)* © *AIME, 1969*

Planning for the major plant improvements began in the late 1950's after all underground mining had been converted to the open-pit operation with truck haulage. At that time, the new Cananea orebody was roughly 700 feet below the crusher as well as several miles away by haulage road. This entailed a long, expensive haul up adverse grades, an estimated additional investment in trucks of some $2 million, and about 10 million tons of extra waste stripping for roads.

Studies to determine a better haulage method considered the following alternatives which had already been tried in various copper pits located in the southwestern U. S.:

- Truck-rail haulage
- Truck-skip-truck haulage
- Truck-skip-rail haulage

All the above options had one undesirable thing in common: excessive material handling. Also, the skips would limit the size of the trucks dumping into them and therefore build some inflexibility into the operation.

Since the above factors added up to an unreasonably high operating cost, Cananea management reviewed its years of experience handling relatively coarse muck underground and came up with the eventual solution. This involved gravity feed of material through an ore pass to a crusher near the ultimate bottom of the pit, followed by inclined conveyor transportation through a tunnel to the concentrator. One of the key basic premises throughout was that the rock should be crushed only to the maximum size that a belt could carry.

One major advantage of this haulage system was the ore storage that a long ore pass from the pit bottom to a subterranean crusher room would provide. This was badly needed for the surge capacity essential to a flexible pit operation. Such storage capacity, initially for 10,000 tons, would be of maximum advantage in the early stages of pit life and of minimum advantage toward the end. In anticipation of the new system, revised pit scheduling based on reduced haulage requirements was worked out, and a plan of pit development focused around a dump hole within the pit was formulated. These studies indicated that the proposed system would provide faster access to ore along with an overall simplified open-pit design.

For the heart of the system, a jaw crusher was chosen over a gyratory because it took up less space both vertically

and horizontally in the limited room available underground. Cananea geologists reviewed rock types in the areas to be excavated, looking particularly at rock strength, and determined that underground openings could be held to approximately the same size as some of the small earlier stopes. As a consequence, a minimum-sized crusher room was designed to stand open *without* concrete or steel support.

Preliminary pricing and excavation estimates revealed an original machinery cost of approximately $1.9 million including installation and excavation costs of $600,000. These amounts appeared very attractive beside an estimated expenditure of about $2 million for trucks and $1 million for stripping roads to do the same job. As it turned out, however, the actual costs of the crusher-conveyor system and the attendant excavation was the same as the estimated cost of a truck system--about $3 million. This was largely because final machinery prices were higher than expected. Notwithstanding a near stand-off in capital cost, comparisons indicated that the shovel-truck-conveyor system would still cost far less than a comparable shovel-truck system owing to lower operating expense.

Commencement of full-scale operation with the newly installed system back in 1967 bore out management's expectations. Despite some serious startup problems, open-pit costs dropped the estimated 20% due largely to reduced trucking costs.

B.2.5.8 Conclusion

Without a doubt Cananea represents a step in the right direction with respect to demonstrating how a more efficient pit operation can be achieved through the use of continuous underground haulage. Even with several design flaws, the new system has provided considerable savings over the former truck, shovel and crusher operation in the pit. Moreover, the

operating cost differential of 20%, calculated during the initial planning effort, continues to be realized.

Although the system has had its share of problems, especially during startup and even now as regards maintenance, it is still proving satisfactory both in performance and availability. For these reasons, it must be considered an advancement in mining technology and a potential model for the future development of more efficient open-pit operations.

B.2.6 Quarry at Dam Site--Idaho

Severe terrain conditions coupled with the close proximity of the proposed quarry site to the dam's left abutment made convenitonal truck haulage of crushed rock an unattractive approach. Innovative planning based on a thorough study of the area geology led to the design of an underground crushing chamber positioned deep in the heart of the quarry. Continuous belt haulage of the final product through an inclined tunnel to a primary surge pile would feed the required aggregate to the on-site concrete batch mixing plant.

Of course, the above scenario has become history as over 13 million tons of granite gneiss were successfuly quarried using the underground crushing operation. This section presents background information on the quarry setup and discusses the thinking that influenced management to accept this novel approach. Figure B-34 provides a view of the completed dam project and the reclaimed quarry site which is only 600 feet downstream of the dam's left abutment. Table 2-11

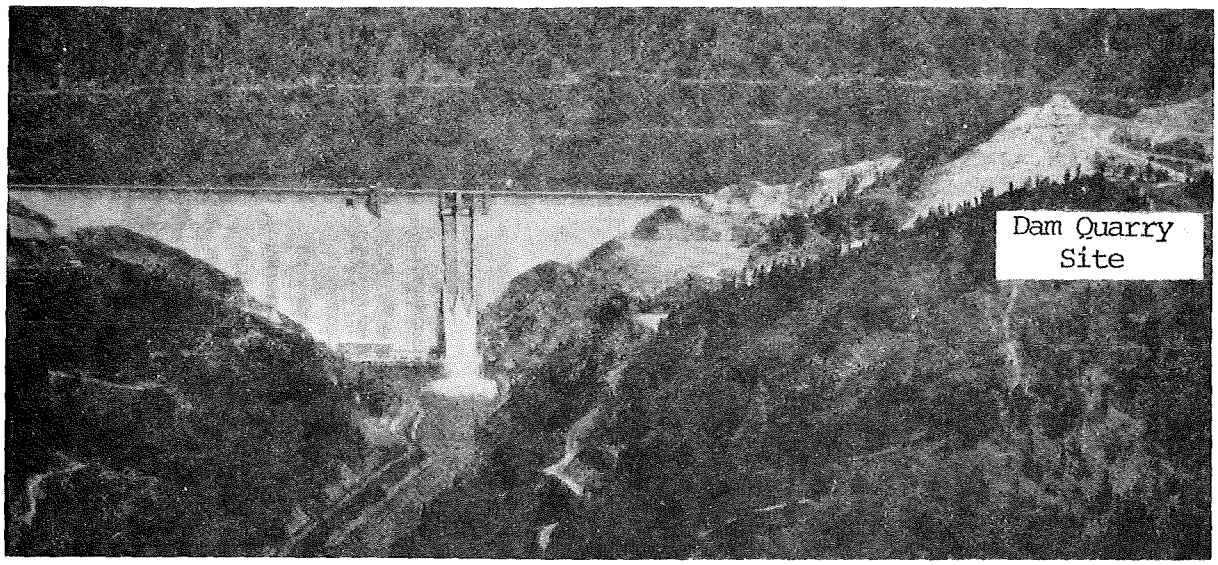


Figure B-34 Idaho Dam and Quarry Site

TABLE B-7

FACT SHEET ON THE OPERTION AT THE QUARRY AT DAM SITE

- Location - North Central Idaho on the North Fork of the Clearwater River
- Contracting Agency - U. S. Army Corps of Engineers
- Ore - Granite gneiss for concrete aggregate
- Start of Quarry Operations - 1966
- Completion of Quarry Operations - 1973
- Quantity Quarried - 13 million tons
- Opeartions:
  - Truck and shovel feeding vertical shaft on top of hillside quarry.
  - Primary and secondary underground crushing facility.
  - Inclined conveyor in tunnel discharging to outside surge pile near concrete batch mine plant.
- Equipment:
  - 54 x 80 inch primary gyratory crusher.
  - 5½ foot secondary cone crusher.
  - 48 inch wide, 800 foot long conveyor inclined at 10^{o}.

B.2.6.1 General Dam Project History and Area Geology

The Idaho Dam is a concrete gravity dam located in North Central Idaho on the North Fork of the Clearwater River, approximately one mile upstream from its confluence with the main Clearwater River. The dam spans a canyon with a crest elevation of 1,613 feet and a length of 3,287 feet. The maximum height of the dam above its base at crest level is 717 feet. Construction of the dam and powerhouse was accomplished for the United States Army Corps of Engineers by a joint venture of private companies. Work started in July, 1966, and was completed in the late fall of 1973 at a cost of approximately $170,000,000.

The aggregate source for the dam's concrete was a hillside quarry of granite gneiss, located 600 feet downstream of the dam's left abutment. The quantity of rock suited to this operation was limited and lay for the most part in bedded deposits between elevation 2,110 feet (ceiling) and 1,610 feet (floor). Approximately 2.8 million cu yd of weathered rock starting at elevation 2,250 feet had to be stripped before quarrying could begin. The river side of the quarry had a natural slope of 1.5:1 initially, but as the quarry developed, the discovery of three sets of slip planes at an average general slope of about .75:1 made it obvious that construction and maintenance of roads in the quarry face would have been a difficult problem.

B.2.6.2 Quarry Layout

A 20-foot diameter ore pass led down through the quarry floor to the underground crushing chamber. Connecting access and product discharge adits linked the chamber to the outside valley floor where dam construction activity was coordinated. This chamber was basically 82 feet long, 34

feet wide, and 102 feet high in order to contain a 10 foot x 26 foot primary feeder, 54 inch x 80 inch gyratory crusher, scalping screens, and secondary cone crushers as shown in Figure B-35.

The roof line of the chamber was located near one edge and 30 feet above the minimum allowed floor elevation of the quarry. Three simple convergence measurement points were installed on the roof line for monitoring with a seismometer located on a concrete base bolted to the rock on the side facing the quarry. Effects of the quarry shots were monitored and appropriate adjustment made to the amount of powder per delay in order to keep the seismic response of the roof below a maximum particle velocity of 10 inches per second. This was accomplished with little difficulty at a minimum of 500 pounds of powder per delay while approaching within 90 feet vertically and approximately 60 feet laterally from the chamber. Convergence points in the roof indicated a light, ½-inch raise of the roof arch attributed to removal of the burden from above with virtually no dropout of fragments into the roof mesh.

Several considerations were involved in picking the general site for the chamber. A location near the upstream wall of the quarry was desirable in order to place the access and conveyor tunnels in areas relatively safe both from quarry blasts and rolling rock from other quarry operations. The location was also designed to allow a safe rock cover over the top of the chamber while utilizing all possible rock in the quarry. The shaft was to be located as close as possible to the center of the quarry to minimize the hauling distance.

There are several methods to lower the ore passes, and the one chosen by the contractors is shown in Figure B-36. A ramp was made down to the new level parallel to the ore pass. Then when the ramp reached the new bench, the cut was turned toward the ore pass. Trucks would haul the rock up the ramp and around to the ore pass until it was reached by the shovel, which would then load directly into it. The ore pass would be

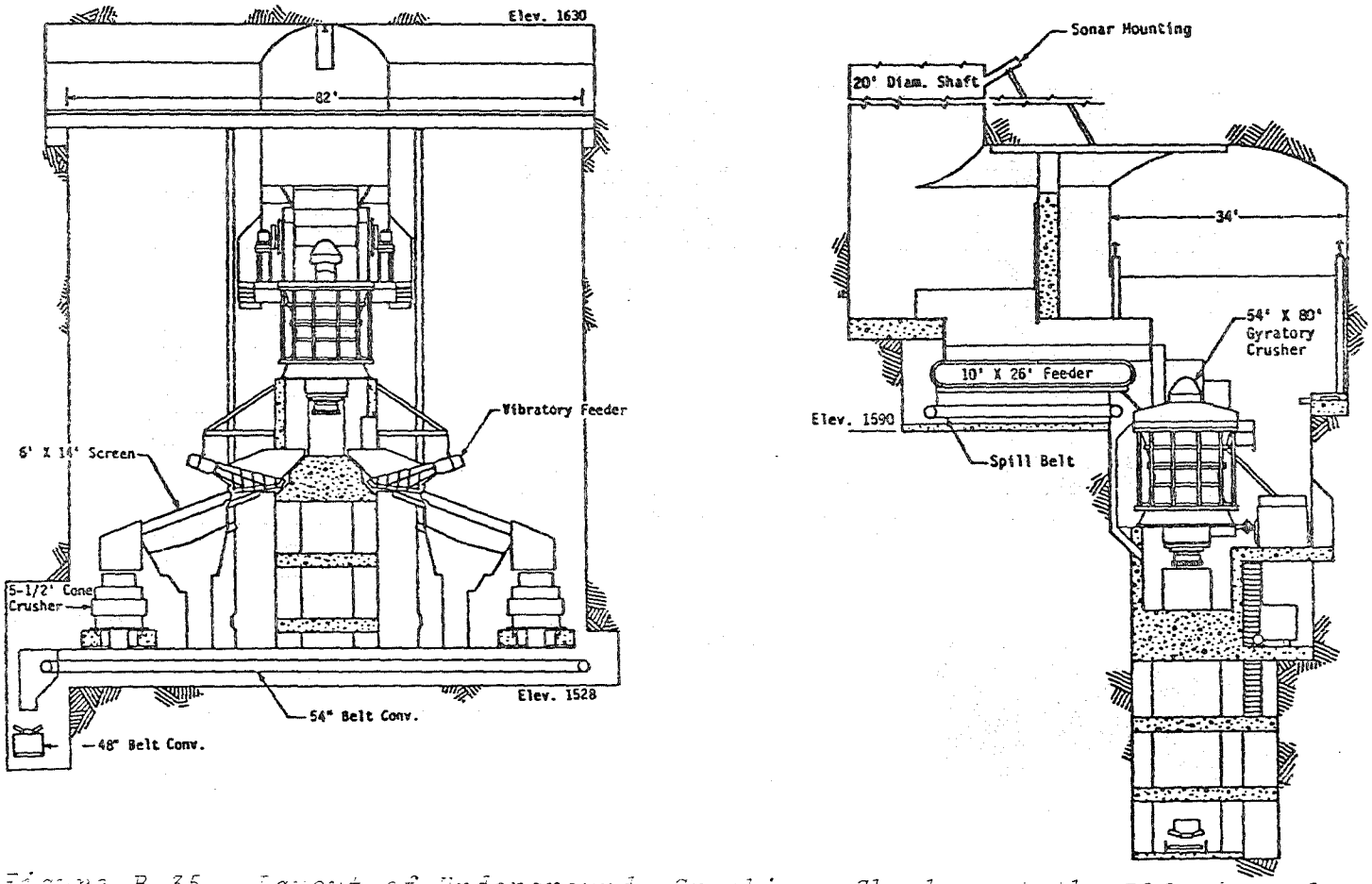


Figure B-35 Layout of Underground Crushing Chamber at the Idaho Dam and Quarry (Ref. D.6.1)© Civil Engineering-ASCE, 1991

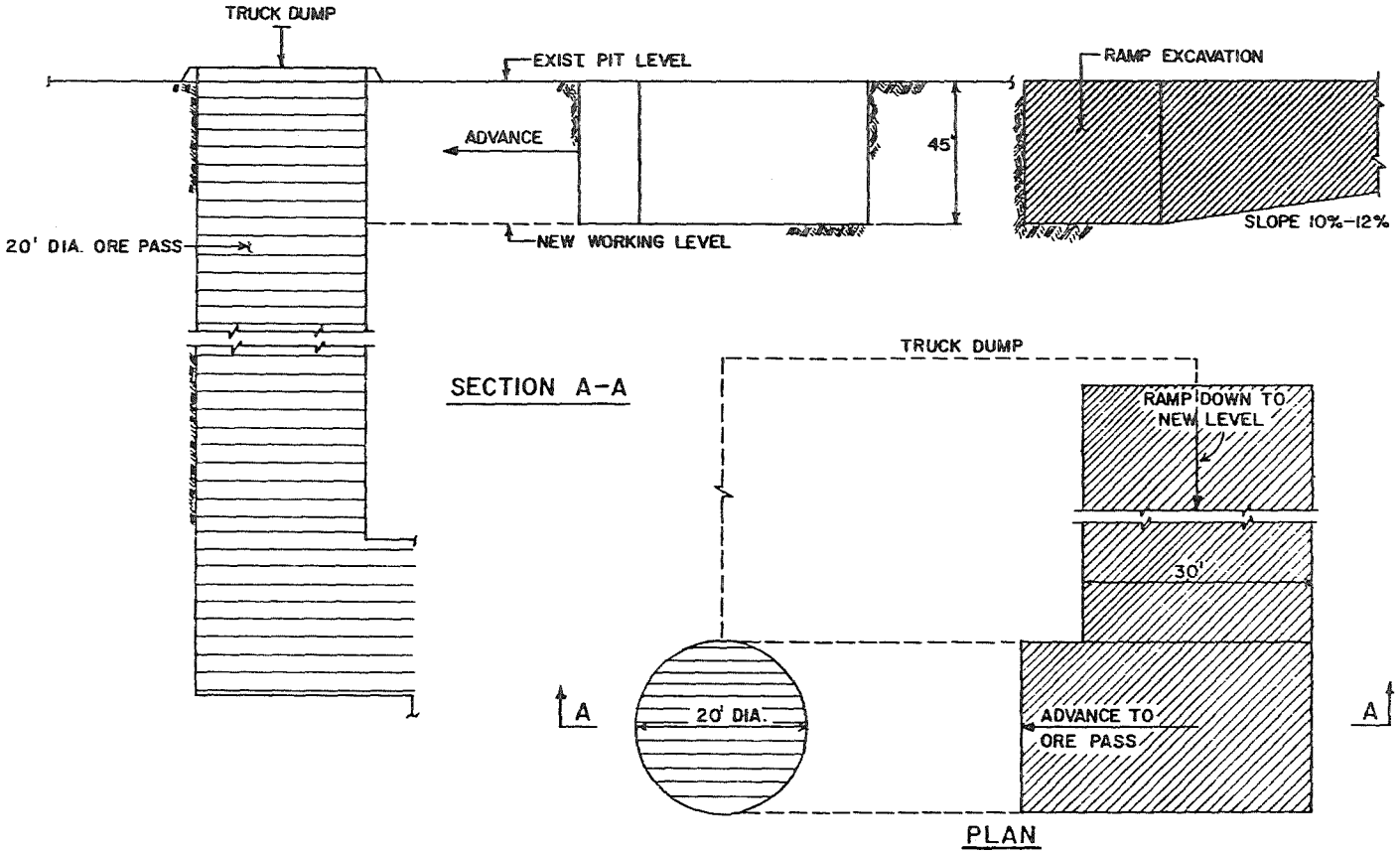


Figure B-33 Benching Down Method at Idaho Dam Quarry Operation: A Ramp Is Driven in a Slot Down to the New Bench. Rock is Trucked Up Around to the Ore Pass Until the Shovel Breaks Through to the Ore Pass.

kept full during blasting for safety. This method was chosen for safety reasons over others evaluated.

B.2.6.3 Quarry Operation

The general operating scheme in the quarry was to drill the rock with 9 inch rotary drills and shoot with ammonium nitrate prills. Excavation was handled by electric shovel and haulage by end-dump trucks to the vertical shaft. The rock was dumped into the ore pass (20 foot diameter, 450 foot depth) and carried down by gravity to the feeder. As each 45 foot bench in the quarry was removed, the shaft became shorter until, finally, it was nonexistent and the last of the rock was dumped almost directly into the feeder.

In the underground crushing chamber the apron feeder conveyed the rock under a hanging wall to a primary gyratory crusher that reduced the rock to minus 10 inches. In turn, the product of this crusher passed over feeders and scalping screens with the plus 6 inch rock being reduced by two standard cone crushers to less than 6 inches. At rates of up to 2,000 tph, the minus 6 inch rock was conveyed 800 feet through an inclined tunnel to a primary surge pile located in a natural draw between the quarry and the dam. The 10 foot wide by 8 foot high conveyor tunnel, besides accommodating the belt conveyor, provided a ventilation exhaust and possible escape route from the chamber.

A three foot dirt beam was used for the backstop for the trucks at the vertical ore pass. The shaft was kept filled at all times to protect the apron feeder and injury to the personnel. Rock was dumped only when the crusher system was running. If the shaft had to be cleaned out for maintenance of the system, then several truckloads of sand would be dumped to act as a buffer when the system was started up again. There was little erosion of the shaft for the 13 million tons of rock that were crushed.

B.2.6.4 Quarry Equipment

Pit operations utilized a 12 cu yd electric powered shovel and 110 ton end dump trucks as shown in Figure B-37. The crushing system shown in Figure B-38 was designed for 2,000 tph of minus 6 inch product, utilizing the following equipment:

- 10 foot wide x 26 foot long manganese apron feeder with spill belt.
- 54 inch x 80 inch primary gyratory crusher
- Two vibrating feeders
- Two 6 foot x 14 foot scalping screens
- Two 5½ inch standard cone crushers
- 48 inch wide, 800 foot long conveyor on a 10° inclined slope

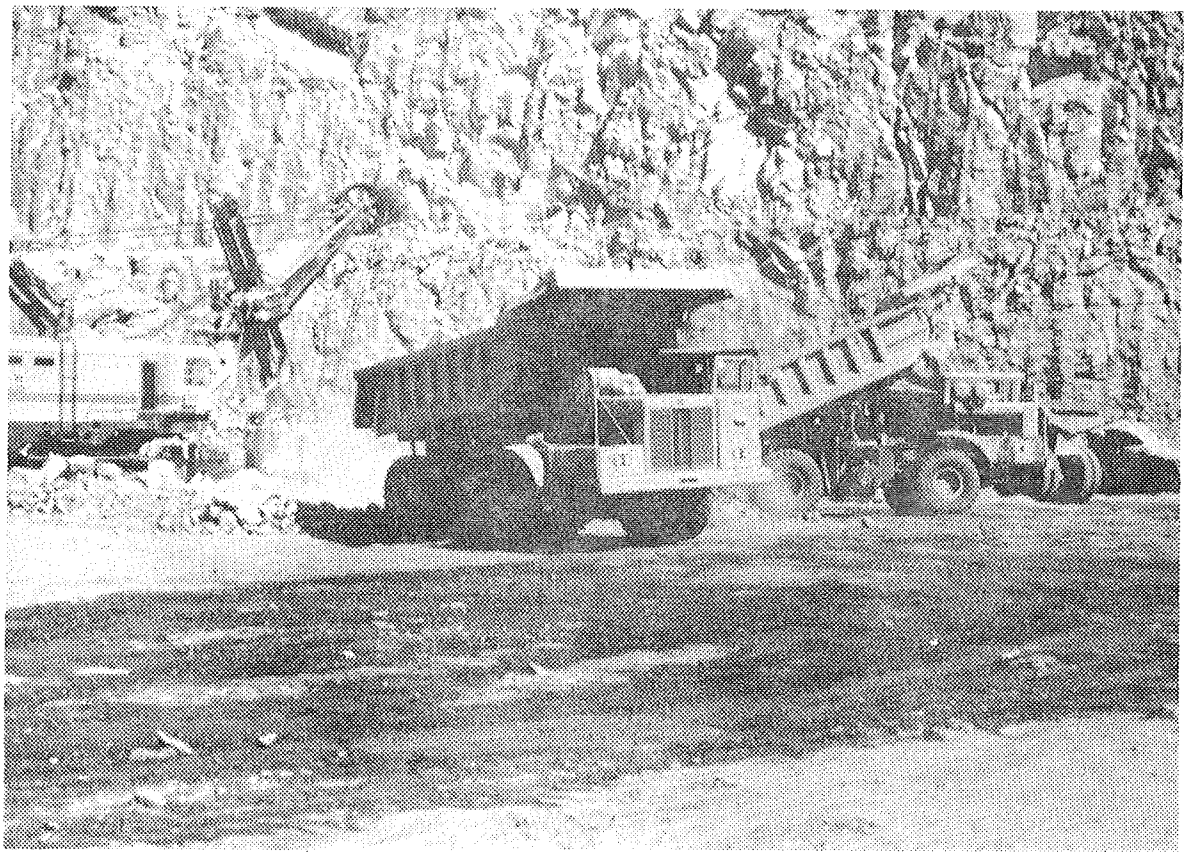


Figure B-37 12 cu. yd. Electric Shovel and 110-ton End Dump Trucks Used in Quarry Operation at Idaho Dam Site

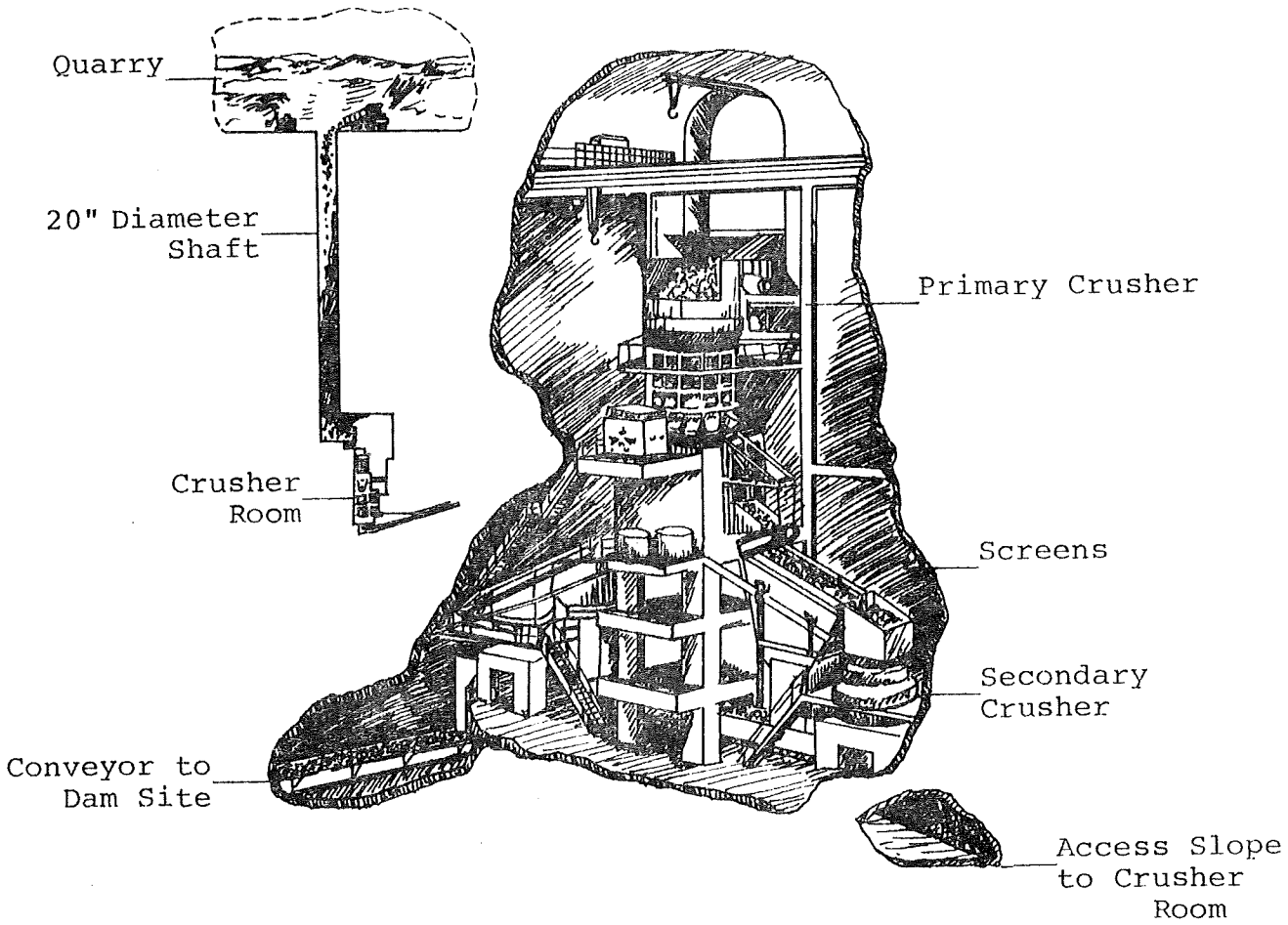


Figure B-38 Primary and Secondary Underground Crushing Facilities in the Idaho Dam and Quarry Site

B.2.6.5 Durability, Maintenance, Manpower and Energy

Considerable forethought went into the design of the underground crushing facility to insure reliability and effeciency of operation. With the dam project hinging on the availability of aggregate, crusher downtime was kept to a minimum. Some of the more notable design features included:

- Four seismometers were installed in the crushing chamber to monitor the blasting techniques so that the particle velocities would not be too high to damage the roof. Multiple hole-per-delay blasts were designed to limit the particle velocity.
- A control room in the crushing chamber gave the operator a clear view of the primary feeder and crusher. The 55 ton overhead bridge crane with

heavy erection hook and lighter utility hook was remotely controlled.

- Critical points such as the conveyor discharge at the surge pile were monitored by closed circuit TV.
- Signal lights with double circuits for reliability were set up near the top of the shaft to enable the crusher operator to keep the quarry crew posted on the status of operations in the chamber for close coordination of the two operations.
- Wear surfaces of the feeder cavity were lined with three-inch manganese steel plate to minimize maintenance work at the bottom of the shaft.
- Dust control was maintained by chemically treated water sprays combined with a 42 inch diameter ventilating system that pulled air from the access tunnel and discharged it into the top of the chamber. Dust-laden air was exhausted from critical points out through the conveyor discharge tunnel.

One design feature that would be changed on future projects relates to the benches and pockets excavated in the sides of the chamber for footings to support the bridge crane rails and other equipment. Construction costs indicate that a more economic crushing installation could be obtained by making the chamber slightly larger and supporting the crane by structural steel carried down to the floor of the chamber.

B.2.6.6 Risk Areas and Technology Gaps

Among the key potential problem areas was the danger of bridging in the ore pass, which would be difficult to clear. Related experience at a copper mining installation in northern Mexico indicated that problems from this source were confined mainly to packing when the ore was wet. The quarry shaft had nearly the same diameter as the Mexico installation, but its more granular material was expected to minimize any potential problem. As an additional precaution, rail was placed in concrete in the hanging wall of the ore pass. However, this was

a mistake, as the rail was replaced by "T-1" steel which showed very little wear by the end of the project.

Another major risk was that the bottom of the shaft might be allowed to run dry, either from the previously mentioned bridging or through operator error by feeding all the rock out of the shaft. This could result in severe damage to the equipment and injury to personnel when the rock came crashing down into the feeder-crusher chamber. A sonar device was designed and interlocked to the feeder controls with an alarm to give warning and automatically shut down the feeder when there was less than 25 feet of rock in the bottom of the shaft. However, due to the dust laden conditions in the air, the instrumentation never functioned properly.

One extremely critical risk area that was studied extensively was the safety of the crushing chamber and its ability to keep its structural integrity while the quarry lifts were successively removed. For example, several theories were advanced on the direction that the crushing chamber should be oriented to give the crown the best structural strength. It was finally decided that, with the relatively narrow 34 foot width, the direction or orientation would not be too important.

B.2.6.7 Rationale for Quarry System Selection--Constraints and Economics

The three most plausible schemes for the quarry operation were:

- Hauling excavated rock down switchback roads cut into the river face of the quarry to a primary crusher installation.
- Cutting a chute in the face of the quarry to slide rock down to the same crusher facility.
- The selected approach described earlier.

Despite the lack of former experience in the construction industry with underground crushing, several strong advantages worked in its favor:

- Some of the stripping at the river side of the quarry could be left and carried down as the quarry benches were opened up.
- The high cost of building roads to carry the rock down more than 400 feet vertically was avoided.
- The danger of delays or accidents due to rock slides on a haul road was avoided and the crushing plant itself was in a safer position.
- The capacity of the shaft would serve as a surge pile of quarry material ahead of primary crushing.
- The cost of the long haul during excavation of the higher benches in the quarry would be saved. This would be partially offset by the higher cost of putting the plant underground.

B.2.6.8 Conclusion

To summarize this successful project in a nutshell, it should be noted that a construction wage spiral, which accelerated faster than anticipated when the 6 year project was estimated, hurt the operating costs much less than if a surface crushing facility with truck haulage had been implemented. As stated before underground crushing combined with continuous conveyor haulage can provide excellent results and above average return on investment if used in the appropriate application.

B.2.7 Nordcement Hannover: Alemannia Works--Germany

While there are at present no American users or manufacturers of primary mobile crushing plants, successful units have already been built and deployed in Europe for the past 20 years. Most of these installations have been in limestone quarries with maximum crusher size to date being a 42-inch gyratory having estimated maximum capacity of 1,000 tph. Only a few of the mobile units have been made with gyratory type crushers, while the majority mount impact-type and jaw crushers. Most of these plants have been used in the cement industry, of which Nordcement has been one of the leaders pioneering this new application.

Experience to date shows that the conservative U. S. mining industry tends to discount the advantages made possible by primary mobile crushing plants with conveyor haulage.

Because of the relatively high production rates required, mine operators feel that any primary crusher installation must be both large and require an extremely high percentage of availability, which of necessity means low maintenance. Consequently, until the confidence in a truly mobile mechanism has been established, the mining industry will lean toward a semi-moveable crusher complete with certain mandatory auxiliary equipment and which need only be moved occasionally in the pit.

European technology is clearly far ahead of the U. S. in this respect. The type of mobile crusher shown in Figure B-39 has been in operation at Nordcement, Hoever near Hannover, since the early 1960's. The following sections describe the current pit setup and provide background on the decision-making process that led their management to try this approach. Table B-8 summarizes the principal features of the present quarry.

B.2.7.1 General Mine History and Orebody Characteristics

The Alemannia Works belongs to Nordcement, Hannover, A.G., and is located just east of Hannover off the E-4 autobahn

TABLE B-8

FACT SHEET ON THE ALEMANNIA WORKS

- Owner - Nordcement, Hannover, A.G.
- Location - Hoever near Hannover, Germany
- Ore - Calcareous marl for cement products (75-90% $CaCO_3$)
- Pit Operations Started - Early 1900's
- Production - Maximum pit output of 1,000 tph
- Operating Personnel - 17 equipment operators, 8 per shift
- Operations:
    - Two mobile (impact-type) crushers producing a maximum output of 500 tph each
    - Twin transfer conveyor feeds
    - Two - one meter wide cross pit conveyors
- Problems:
    - Pit floor very soft
    - Ore difficult to mine during wet periods

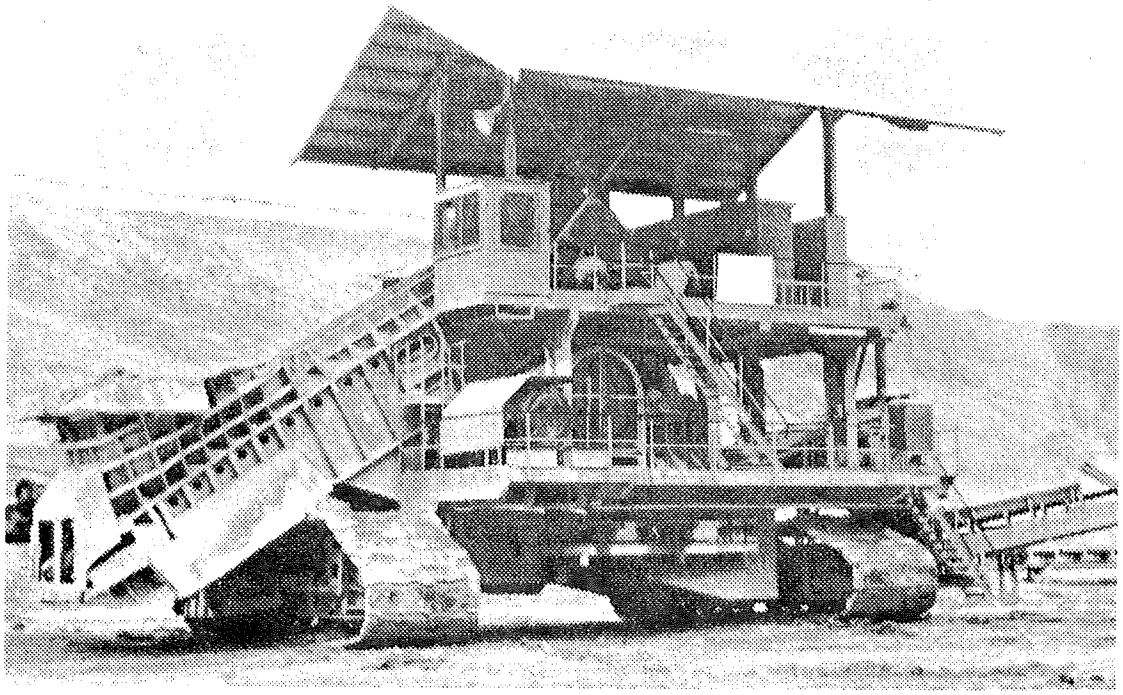


Figure B-39 Primary Mobile Crusher (500 tph capacity) Used at the Nordcement Hannover Hoever Works Quarry

(see Figure 2-10). Current production is handled by a combination of primary mobile crushers and conveyor haulage out of the pit, with shovels supplying the crushers. It was decided to employ a mobile crushing unit in the quarry because the moist soft ground did not permit the use of heavy trucks for transporting the material, and a road network would have entailed a large investment. The use of a rail system was eliminated from the beginning, since the ground would be very uneven after the quarry was developed.

From the geologic point of view, the deposit lies under a very moist surface and belongs to the Turon (Upper Chalk)

formation, which is soft on the surface and medium to very hard farther down. Between the 15% sloping banks of chalk (with a calcium carbonate content of 75 to 90% and running from west to east), there are numerous beds, up to 3 m deep, which are rich in alumina and iron oxide but contain only 50 to 60% $CaCO_3$.

B.2.7.2 Mine Layout

The quarry material, Calcareous marl, was loosely consolidated or decomposed and quite uniform in particle size with relatively few large boulders. A considerable amount of water was present on the quarry floor, apparently natural seepage, creating a relatively unstable working surface with low allowable bearing pressure. They used to blast the material for many years until the proximity of the village prohibited continuation of blasting with respect to safety regulation and environment protection. They will probably revert to blasting as soon as the working face is out of bonds. Therefore, it was decided that excavation would be handled with a crawler-mounted tractor equipped with a ripper.

B.2.7.3 Mining Operation

Bulldozers push the soft rock to electric-powered shovels which load directly onto the two mobile crushers. A 1.6 m apron feeder is set at a 30° inclination to feed the double-shaft hammer mill that is driven by electric motors. The rock is discharged into a discharged belt conveyor which feeds the slewing conveyor and the self-contained transfer belt ahead of the main pit conveyor.

The quarry operates two shifts per day with eight men per shift. The crews have six operators, one mechanic, and one foreman per shift. The crews do most of their own maintenance and have an availability of 75% which includes delays for lunch, maintenance, travel and plant delays at the discharge of the belts.

B.2.7.4 Mining Equipment

During normal operations D-9's, bulldozers, or equivalent, are used to rip and bulldoze the material to two electrically powered shovels with 2.5 cum buckets for charging the mobile crushing units. Because of variations in the deposit, primarily differences in clay content and composition, two identical track-mounted crushers with attendant equipment, as shown in Figure B-40 are utilized to provide proper quantities of each material for blending to meet plant feed requirements. The crushers, supplied by Krupp (see Figure B-41) operate in an open circuit with a capacity of 500 tph, a maximum feed size of 1,250 mm x 1000 mm x 800 mm (49 inches x 4 inches x 31 inches) and a product size of 0-100 mm (0-4 inches). Data pertaining to these units is listed on page 287:

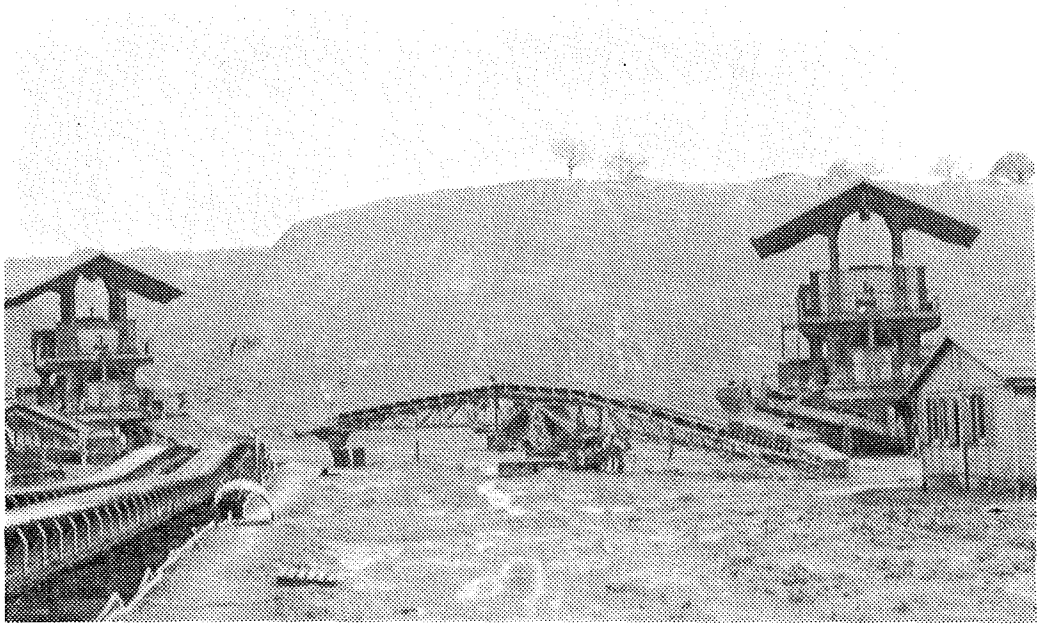
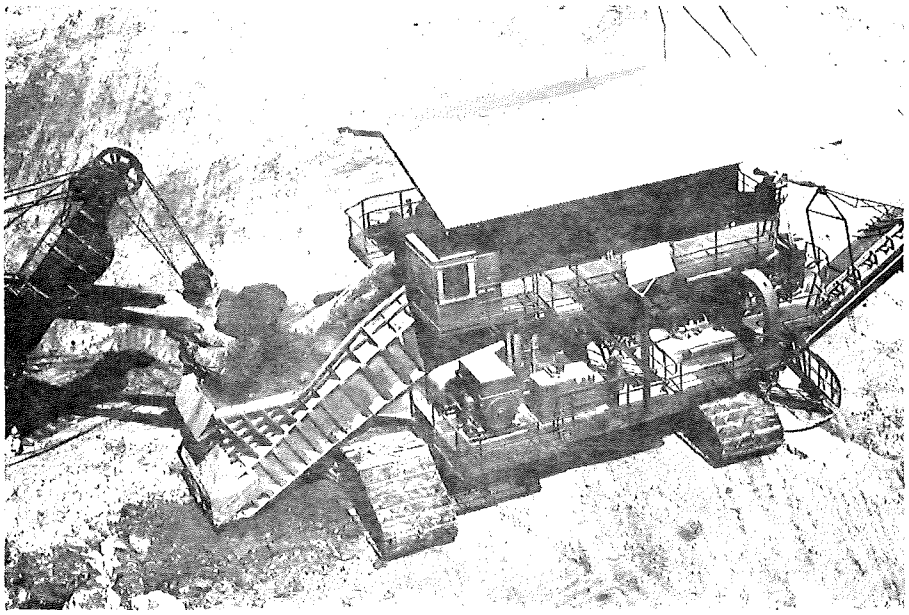


Figure B-40 Two identical track-mounted crushers with attendant equipment.



Crusher supplied 1971



Crusher supplied 1972

Figure B-41 *Two double shaft hammer mill crushers supplied to Nordcement Hannover, Alemannia Works.*

- Feed Bin: Approximately 17 m^3 (22.25 cu yds)
- Feeder: Apron type with a width of 1,900 mm (74.8 inches): length of 11.34 m (37.0 feet); inclination of 30^o with a 45 kw drive motor.
- Crusher: Double-shaft hammer with a diameter of 1,650 mm (64 inches), width of 1,700 mm (66.5 inches) with each rotor powered by a 280 kw drive motor.
- Discharge Conveyor: Rubber belt with a width of 1,600 mm (63 inches), length of 10.2 m (34.0 feet) with an 18.5 kw drive motor.
- Slewing Conveyor: Rubber belt with a width of 1,200 mm (47 inches), length of 9 mm (30.0 feet) with a drive motor of 18.5 kw.
- Travelling Mechanism: Two crawlers with a width of 2,300 mm (7.5 feet) on 11,000 mm (35.75 feet) centers each powered with 55 kw motors.
- Travelling Speed: 6.0 mpm (2.0 fpm)
- Weight: 345 tons
- Total Power Installed: 760 kw

A slewing conveyor associated with the crusher unit discharges onto an accompanying track mounted transfer conveyor, which then feeds one of two parallel permanently fixed extendable haulage conveyors (see Figure B-42). Material received from the transfer equipment is conveyed approximately one kilometer (0.6 miles) by two parallel conveyor belts with a width of 1.00 m (3.0 feet) and belt speed of 1.5 mps (5 fps) to a transfer tower. From the transfer tower the material is conveyed on a similar belt system approximately one kilometer to storage and blending silos at the plant site.

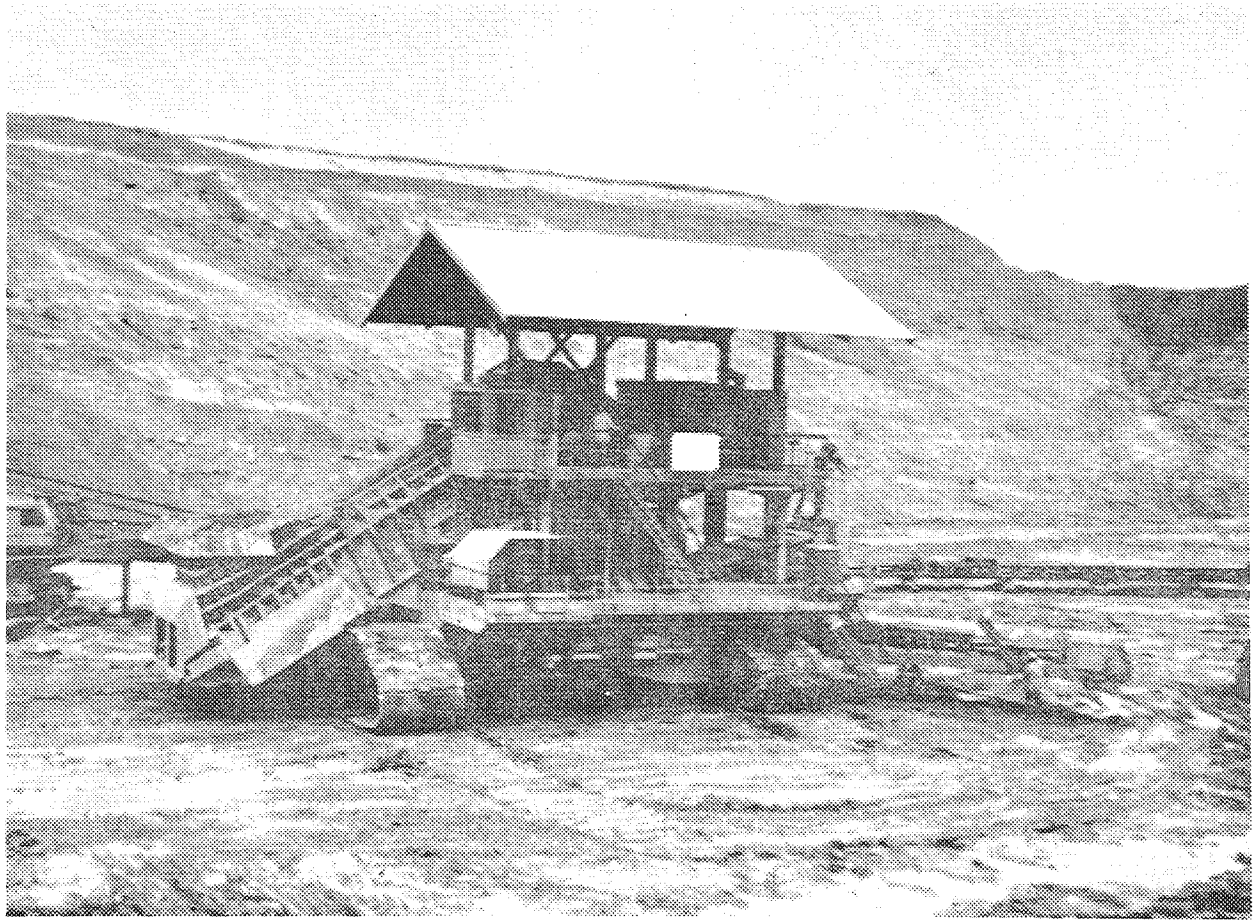


Figure B-42 Track-mounted crusher feeding slewing conveyor to transfer conveyor and two extendable pit haulage conveyors. (Note the size of operators on equipment.)

B.2.7.5 Durability, Maintenance, Manpower and Energy

In the mid 1950's, German heavy equipment manufacturers worked with cement works like Nordcement Hannover to reduce the rising haulage costs of truck and shovel operation resulting from ever-increasing distances between working face and crushing plant. High-speed belt conveyor systems offered the best solution, but necessitated prior crushing of the material to a size suitable for belt conveyor transport. The logical consequence was the development of mobile crushing units discharging onto shiftable belt conveyors which feed stationary conveyors to the processing plant. Mobile crushers follow the shovel along the quarry face under their own power.

In operations where mobile crusher-conveyor systems have been introduced, shovel performances have been increased by eliminating time losses caused by waiting and maneuvering of trucks, and through the reduction of the swing angle of the shovel by placing the mobile crusher in the most favorable position with respect to the shovel. The shovel operator may dump the total content of the bucket into the feeding hopper. He does not need to empty the bucket gradually as it is the case when loading on a truck in order to avoid shock loads and resulting damages to truck body, springs and shock-absorbers. This, too, speeds up work. The fact that the crusher-conveyor system is a continuous one also increases crusher performance.

Mobile crushers also permit increases in man-hour output and reduce power consumption and maintenance of haulage equipment. Mobile crushing plants are now working with just three operators to take care of the loading, crushing and haulage for any distance or lift. Using a 15 cu-yd shovel, outputs of 1,700 tph or 570 tons per man-hour (or 40,000 tpd using nine men over three shifts) can be obtained with a mobile crusher system.

B.2.7.6 Risk Areas and Technology Gaps

Proponents of the primary mobile crushing plant indicate that economies can be achieved by conveying material directly from the pit to the secondary crushers located in the fine crusher building completely outside the pit. Generally speaking, the aggregate industry as a whole agrees that conveying material is cheaper than trucking it, whereas the mining industry does not. This difference of opinion coupled with a commitment to very large primary crushers (up to 10,000 tph) has precluded the U. S. mining industry from seriously considering the potential advantages of a mobile crushing/conveying system.

Mining industry sentiment on what constitutes a satisfactory, moveable, primary crusher includes the following points *(Ref. D.7.1)*. © *Mining Congress Journal, 1973*

- The plant must be totally self-contained.
- An overwhelming preference for the gyratory type crusher. The consensus is that it has superior capacity, low maintenance, produces a minimum amount of slabbing and is most suitable to handle abrasive material. On the other hand, it has some inherent disadvantages in that it requires excessive head room, is excessively heavy, and is high in capital cost.

  As far as the mechanism to provide mobility for the crusher is concerned, the following types have been considered.

  - Self-propelled crawler
  - Self-propelled walking mechanism
  - Self-propelled wheeled system
  - Towable skid
  - Towable wheeled trailer
  - Self-propelled railroad car
  - Towable railroad car
  - Towable structural skid

- Little interest in a mobility mechanism of any type. An overwhelming desire for a semi-moveable primary crusher elevated on a steel frame and placed on a concrete slab.
- All electric drive
- Dust control mechanism

- Auxiliary equipment which is considered to be an integral part of the plant:
  - A feeding mechanism to the primary crusher either of the pan type or the grizzly type.
  - An overhead crane which could be used for both maintenance and removing bridges within the crusher itself.
- Optional features
  - A discharge hopper with a pan feeder under the hopper to feed the conveyor taking material away from the crusher.
  - An automatic lubrication system
  - A closed-circuit TV for observing crusher operation.

It is also interesting to note that most plant operators felt that the person in charge of operating a primary mobile crushing plant would not demand higher wages or any other type of special union considerations because of his position as operator of this rather specialized piece of equipment. However, there was a distinct possibility that union labor problems might start if a primary mobile crushing plant were introduced into an established working pit because of the decreased use of truck drivers.

B.2.7.7 Rationale for Mine System Selection-- Constraints and Economics

Extremely adverse ground conditions in the pit were in the beginning a major factor influencing the German limestone companies to investigate alternatives. Now that mobile crusher/conveyors have become standard operation, these same outfits are quick to point out the added benefit resulting from longer haul distances and lower transport costs per ton of material compared to truck haulage.

Nordcement Hannover presented the following arguments in favor of the mobile crushing/belt conveyor systems:

- Shovel productivity is increased by as much as 50% due to uniform continuous feeding of the mobile unit with no waiting time for dump truck maneuvering.
- Crusher capacity is increased as it is charged continuously and always operates with a full cavity.
- Continuous material transport from the quarry by a combination of shiftable and stationary belt conveyor systems.
- Shortening of haulage routes, since belt conveyors are able to overcome higher gradients than dump trucks.
- Reduction in energy costs for the equipment.
- Reduction in truck driving and maintenance personnel.
- Only a small dump truck repair shop is needed.
- No investment required for dump truck haul routes.
- No truck accidents.
- Mobile crushers save costs for buildings used to house permanent installations.
- Overall increase in productivity and efficiency per man and shift.

To illustrate investment and operating cost comparisons for dump truck haulage versus mobile conveyor transport, German equipment manufacturers, have worked up a fairly straight forward evaluation procedure. Referring to Fig. B-43, in the abscissa scale is the horizontal haulage distance, that is the straight line distance between the loading and discharging points. On the left ordinate scale are the transport charges per ton-mile, and on the right the investment cost. (Figures are based on 1,000 tph system.) Although the investment costs are higher for belt conveyors than for trucks, the operating costs for belt conveyors are considerably lower. In addition, when the necessary road construction and maintenance costs are added, the situation becomes even more favorable for belt transport. In this fashion the cost tradeoffs between truck and conveyor must

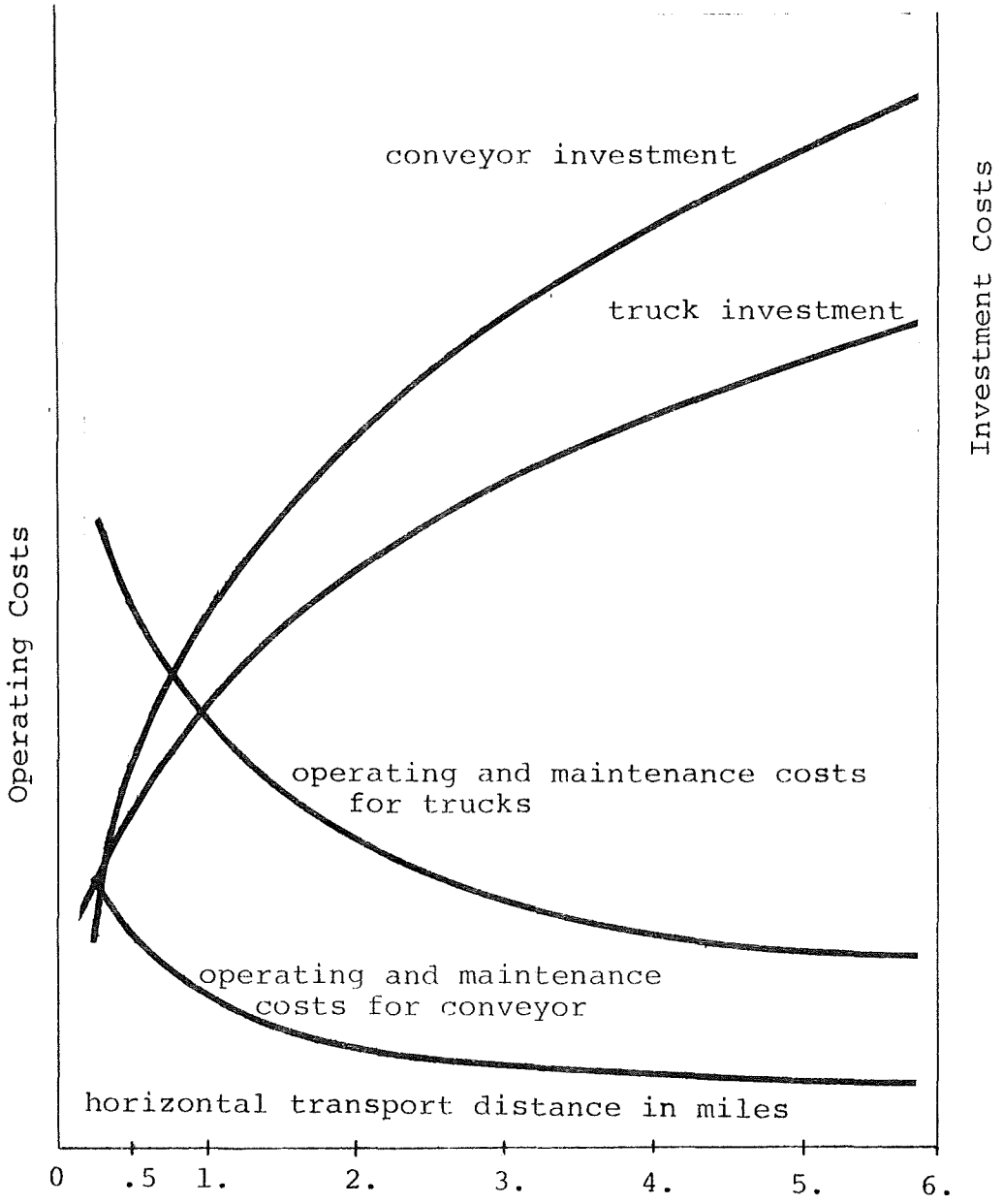


Figure B-43 Operating and Investment Cost Comparison for Truck Haulage vs. Conveyors in Horizontal Transport

be analyzed in light of the total life expectancy (composite tonnage) of both the pit and the equipment.

Taking a look at the more typical case where pit operation occurs at an elevation below that of the processing plant, it is important to note a belt conveyor can negotiate a higher gradient than a dump truck. In open-pit operations it is uneconomical for most trucks to cope with gradients of more than 10% (6°), while belt conveyors can be arranged at slopes of up to 32.5% (18°). In practice this is equivalent to a shorter haul route relatively speaking for the conveyor. Figure B-44 shows the interrelation between the investment and operating costs for roughly a 600 foot deep pit. The operation costs for belts are far below those for the dump trucks, while investment costs tend to equalize around 0.3 miles. Once again, the cost analysis must be carried through for estimated total pit output and equipment life expectancy.

The above cost comparisons do not take into consideration a very essential feature in favor of belt conveyor transport, namely, its insensitivity to weather, which is particularly advantageous in case of bad soil conditions. There are pits and quarries--especially in the cement industry--which are not passable for any vehicle during rainy seasons. To insure continuous cement production it is necessary to provide large storage capacity to ensure that there will be no interruption during periods when the open-pit operations come to a standstill. This means that the conventional equipment for quarrying limestone and transporting it to the processing plant should be dimensioned to guarantee a sufficient stockpile for such downtimes.

B.2.7.8 Conclusion

A very comprehensive assessment on the potential impact of mobile crushing/conveying for U. S. open-pit operation was made 5 years ago by Mr. Arnold P. Szaj *(Ref. D.7.1)*

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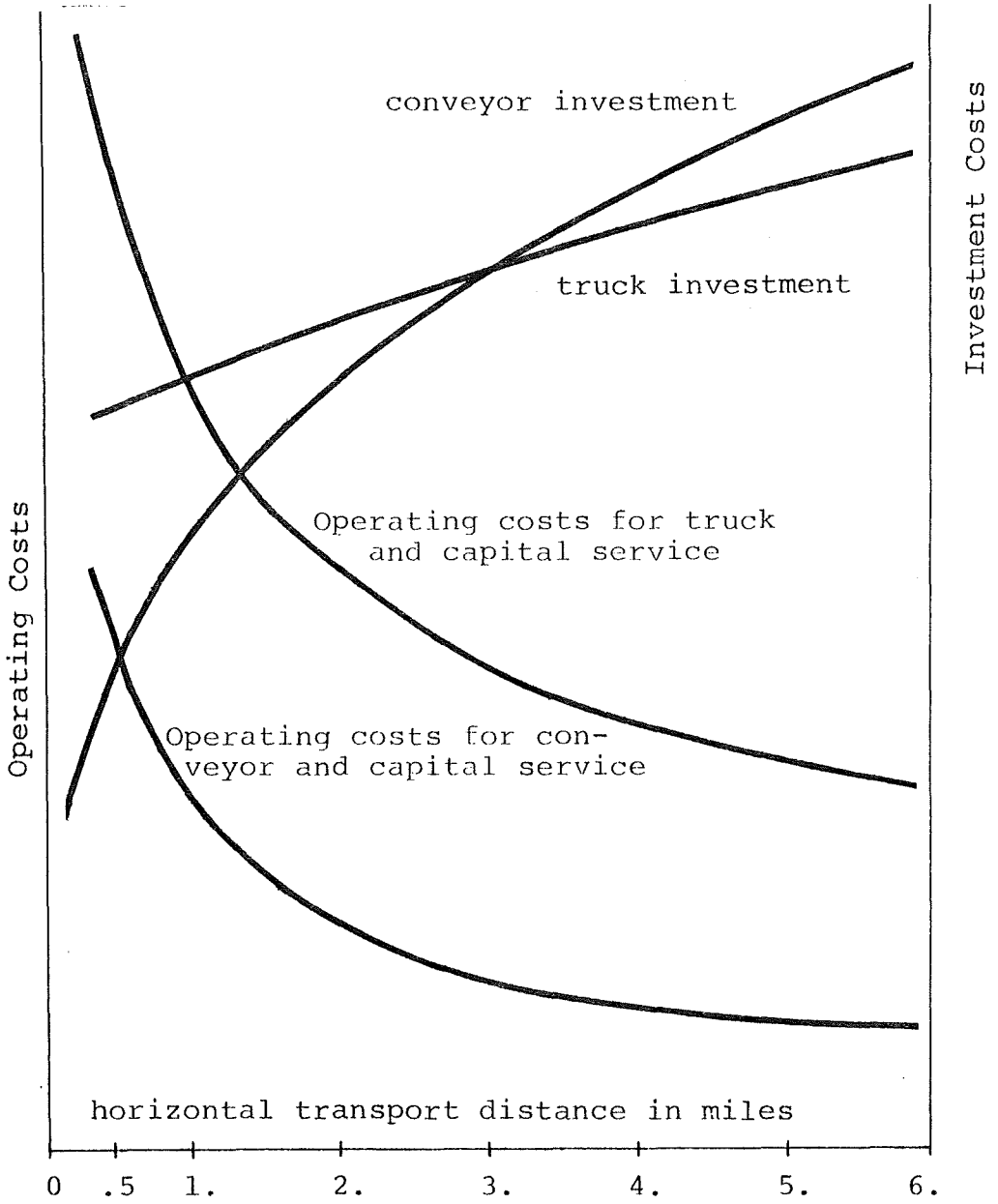


Figure B-44 Operating and Investment Cost Comparison for Truck Haulage vs. Conveyors out of a 600 ft. deep Pit

His analysis, presented below, still holds true based upon the feedback obtained during debriefings on this program with mining company officials.

> It is our opinion that the true primary mobile crushing plant in the mining industry has only limited application at the present time. It would appear that the mining industry is much more interested in the semi-moveable structure than the truly mobile type crushing plant.
>
> The mining industry is definitely leaning toward the very large type gyratory crusher in size ranges from the 54-inch opening to the 72-inch opening. They do not have the confidence that crushers of this size can be mounted on a mobile mechanism without a multitude of mechanical problems.
>
> Some companies feel that the idea of moving the crusher in order to shoot the face simply would not work out from a time requirement standpoint. They would much rather move a semi-moveable crusher periodically perhaps as frequently as every three months, but more likely on a basis of from two to ten years dependent upon the pit itself.
>
> Most mining companies felt that the problem of blending ores was disadvantageous to the concept of a primary mobile crushing plant rather than an advantage.
>
> There is considerable interest shown in the semi-moveable primary crushing plant, primarily because of what operators consider to be an economic savings in the field of using conveyors rather than truck.

B.2.8 Open-Pit Copper Mine--Arizona

The increasing interest in the development of low-grade orebodies suggests that, in the case of copper, the average future grade coming from the predominant porphyry type deposits will be on the order of .25%. One particular copper company, which looked at the potential of low-grade mining in the early 1970's, concluded that low-cost continuous haulage of large tonnage production would be essential to cope with fluctuating operating costs and unpredictable world market prices for copper. An engineering-economic analysis showed that the use of a high-capacity conveyor belt running on an incline up the pit wall with in-pit primary crushing offered the best return on investment. This system was subsequently selected and installed versus going to a conventional truck and shovel haulage operation.

This section discusses in some detail the reasons for mine management's decision to favor conveyor haulage over the use of dump trucks. Figures B-45 and B-46 give a view of the current operation, in particular the inclined conveyor rising out of the pit, and Table B-9 presents a list of principal features about the operation.

B.2.8.1 General Mine History and Orebody Characterisitics

This open-pit operation in Arizona was started in 1967 at an initial investment cost exceeding $300 million. Initial production was accomplished through the use of trucks to haul waste to the surface and in-pit crushing of the ore with conveyor belt haulage to the surface. In 1974, a study determined that a combination of truck to in-pit crushing and conveyor haulage to surface of both production ore and waste rock was justified. The proposed project was completed in May, 1977.

The ore being mined includes copper, molybdenum, and silver. The deposit covers a 13,000-acre area with two separate pits at the ore epicenters creating a somewhat figure-8 like form (see Figure B-47). The ore is mixed fairly well

TABLE B-9

FACT SHEET ON THE OPEN-PIT COPPER MINE IN ARIZONA

- Ore - Copper, Molybdenum, Silver
- Mineralization - chalcopyrite
- Pit Operations Started - 1967
- Production - 92,000 tpd of ore, 164,000 tpd of waste rock
- Potential Mine Life at Present Production Rate - 20 years
- Operating Personnel - 2,300
- Operations:
    - Conventional in-pit truck and shovel operation.
    - Inclined conveyor up the pit wall hauls crushed material to the surface.
    - Separate crushing/haulage system for ore and waste rock.
- Equipment:
    - 120 to 170 ton haulage trucks.
    - two -double 60-inch gyratory crusher systems.
    - two inclined conveyor systems.
- Problems:
    - Circuitous path of ore conveyor is not an ideal arrangement.

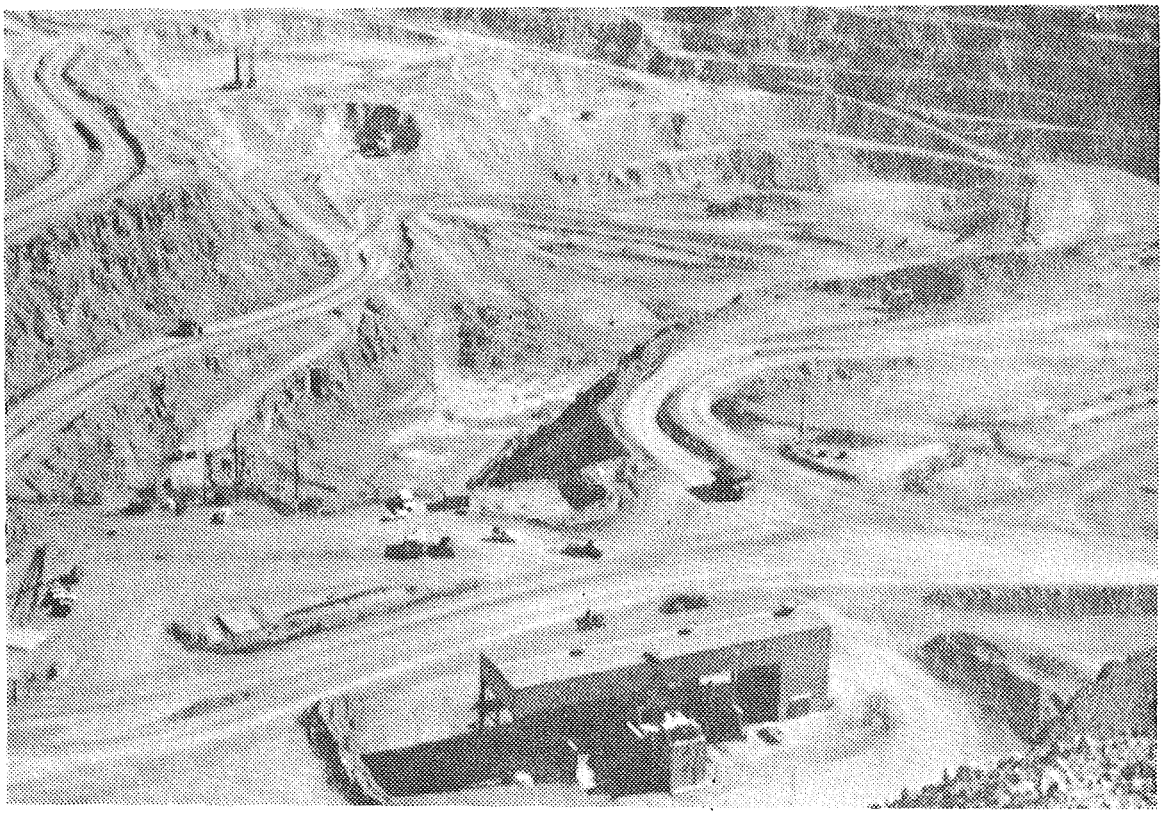


Figure B-45 Arizona Copper Open-Pit With In-Pit Crushing

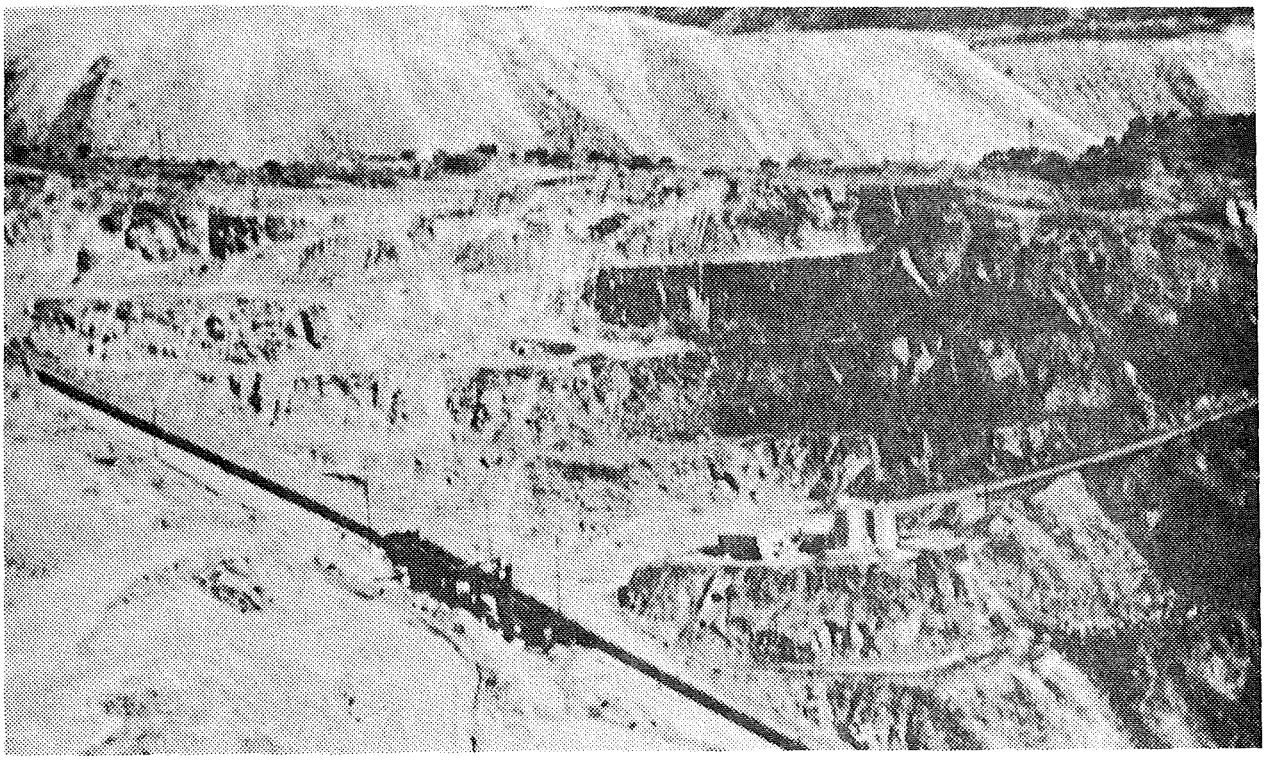


Figure B-46 Arizona Copper Pit With Conveyor Haulage

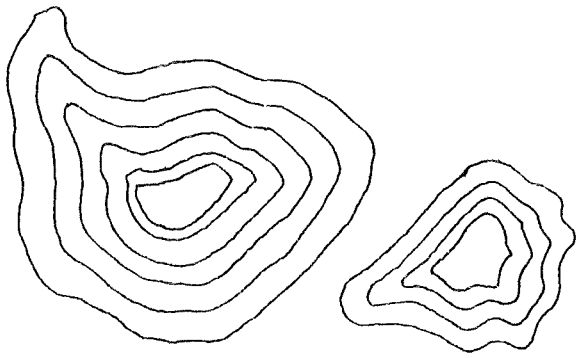


Figure B-47 Two Separate Pits For Somewhat Figure Eight Like Form

and disseminated throughout the host rock in a 1.6:1 waste-to-ore ratio. Moisture content runs on the order of three percent.

B.2.8.2 Mine Layout

The open-pit operation commenced at an initial surface elevation of 4,350 feet. In over 10 years of operation, the pit bottom has been sunk to 2,600 feet with an ultimate design elevation set for 1,850 feet (not to be reached for at least 10 or more years). Pit walls, which sloped initially at 45%, have been reduced to 38%. Bench heights are on the order of 50 feet. There have been a few rock slides in isolated areas.

Mine layout is as shown in Figure B-48 with all haulage being accomplished from the larger pit. Waste is conveyed from near-pit via a three-section conveyor to the dump area. A much longer four-section conveyor moves the ore from the pit to the surface discharge point for mill processing.

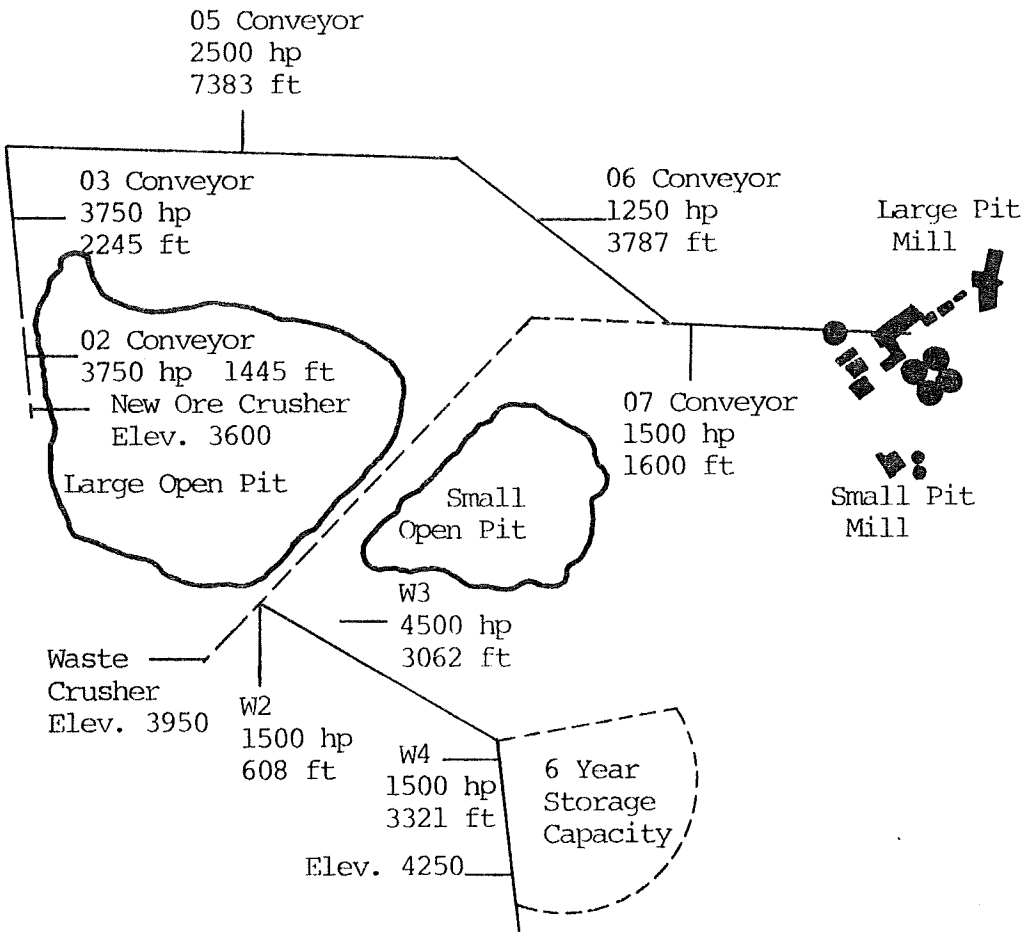


Figure B-48 Conveyor Haulage Arrangement

B.2.8.3 Mining Operation

Current operations produce 92,000 tpd of ore and 164,000 tpd of waste with equipment running at 75% availability. Total present reserves are estiamted at 500 million tons of ore, enough for at least a 20-year supply at the present production rate.

The mining operation uses 15 cu yd electric shovels to load 120- to 170-ton haulage trucks. These trucks haul waste to two 60-inch gyratory crushers where it is crushed to minus 8

inches and conveyed by three sections of 7,988-foot, 72-inch wide, steel cable belt from the 3,950-foot to the 4,350-foot elevation. Maximum uphill slope is 16%. Ore is crushed in an identical crushing system to minus 6 inches and is carried by four sections of 16,000-foot, 60-inch wide, steel cable belt conveyor from the 2,600-foot level to the 4,350-foot level.

The new ore crushing plant completed in 1977 has above-grade facilities including an enclosed shop area for maintenance work and two open maintenance bays, all serviced by an 80-ton/20-ton bridge crane. The operator's control room has a direct view into both crusher dump pockets. A small console-type control panel is centrally located in the control room from which the crushers and pan feeders are operated. Both crusher dump pockets have rock breakers set in a side wall. Each dump pocket has two bays, providing a total of four dump points for end-dump trucks. There are no grizzlies ahead of the crushers. There is a 300-ton surge bin under each crusher. Each bin discharges on to an 84-inch pan feeder which is controlled by variable frequency drives to maintain a steady feed rate on the conveyor system.

The ore conveyor system consists mainly of four new belt flights and one which is part of the old ore system. The total length of the system is about 19,000 feet, of which 16,000 feet is the new conveyor described earlier. The conveyor runs at 831 fpm with a maximum capacity of 6,400 tph. The system was designed to average 4,800 tph at a conservative operating time of 75%. There is a scheduled shutdown for maintenance for one shift a week.

The old ore crushing plant at the 3,950 elevation became the waste crushing facility in 1977, and was modified to handle 8,000 tph. Both 60-89 gyratory machines had their crushing chambers revised to provide an 8½-inch open-side setting and a capacity of 4,000 tph each. The surge bin and pan feeder arrangement is identical to that in the new ore crushing plant.

The waste conveyor system delivers material to a tripper-stacker at the 4,250 elevation. The total length of the three belts is 7,988 feet, the third section being a 3,321-foot shiftable belt with the drive at the tail. The belt will be moved every 35 days when operating at design capacity of 144,000 tpd.

B.2.8.4 Mining Equipment

The blast-load-truck operation utilizes the following equipment:

- Drilling and Blasting: 9-12½-inch rotary blast hole drills.
- Loading equipment: 15 cu yd electric shovels
- Haulage equipment: 120- to 170-ton diesel-powered trucks.

Crushing equipment in the pit is listed below:

- Ore: Two 60-89 gyratory crushers, 500 hp each, 6-inch openside setting, 3,200 tph each, 300-ton surge bin under each crusher, 84-inch pan feeder.
- Waste: Identical to ore crushers. They were the former ore crushers (pre-1977).

Ore haulage from the pit to the surface uses the following equipment:

- Ore: Four main conveyor belts, 16,000 foot total length. 11,250 total connected hp. 60-inch wide steel cable belt, maximum tension rating 3,150 piw. 831 tpm belt speed, 6,400 tph capacity. 16% maximum uphill at steepest, first conveyor out of pit.
- Waste: Three main conveyor belts, 7,988-foot total length. 7,500 total connected hp. 72-inch wide steel cable belt, maximum tension rating 2,650 piw. 831 fpm belt speed, 8,000 tph capacity. Stacker and tripper travel 3,300-foot length of shiftable conveyor.
- Conveyor Data: Load idlers 35° troughers, with 7-inch rollers. 5-foot intervals. Return idlers 10° vee spaced at 20-foot intervals. Hydraulic trainer idlers. 40-foot conveyor frames.

B.2.8.5 Durability, Maintenance, Manpower, and Energy

Mine management is fairly forward-looking, as evidenced by its recently completed major expansion program after only 10 years of operation. Moreover, they are keeping an eye to the future through active consideration of new additions to the existing system to enhance productivity and availability--for example:

- Portable or semi-portable crushers.
- A one-on-one shovel-crusher system.
- An ore pass to a below-pit crusher at the 1,700-foot level with inclined conveyor haulage to the surface.
- Crusher-grizzly with undersize hauled by conveyors; oversize by trucks.

The mining operation employs a total of 2,300 personnel on a three-shift per day basis, 364 days a year. Efforts at increased efficiency of operation through automation have resulted in reduced manpower requirements. Examples include:

- Both the ore and waste systems can be operated remotely from their respective crusher control rooms. Semi-graphic control panels display equipment status by means of colored lights.
- A computerized maintenance program keeps equipment performance statistics and controls servicing requirements.
- Component standardization was used on the material handling systems. Pulley bearings, motors, reducers and holdbacks are all interchangeable.

The result of all this planning and attention to operating detail is a relatively low cost of production.

The maintenance program deserves further mention as the low operating costs are partly attributable to this effort. In particular, there are preventative maintenance programs on all items of equipment. Computerized listings give equipment statistics vital to operation planners. Data obtained include cost per unit with production, warehousing and accounting costs.

Visual computer printouts are tied into the centralized computer operation in Houston, where all mining activities are monitored and centrally coordinated.

B.2.8.6 Risk Areas and Technology Gaps

The main obstacle confronted by mine management was the lack of experience in the U. S. with this kind of a system. Essentially the haulage method was untried when the go-ahead commitment was made. In particular, conveyor technology was not sufficiently advanced to provide the large-sized or high-speed belts required for this magnitude of operation.

One of the major risk areas that is inevitable with this constant effort to modernize is that it takes more time to get the equipment fully operational once installed at the property and to train the operators and maintenance personnel. An adequate costing system is also not available, due to changing needs. The Systems Group has the priority assignment to both build a data bank and keep the information updated at the same time. Further evolution of the equipment employed and changes in the mining layout seem to be outdistancing this data management effort for this time being.

Other risks relating to the use of belt haulage included:

- Flexibility of mine planning is reduced.
- The consequences of unscheduled shut-downs of the belt system could be financially damaging.
- Initial capital cost of the belt system is high.

A spatially fixed haulage system induces a lost flexibility in mine planning. This eventually results in increased mining costs.

The equipment in the conveyor system is in series. The unscheduled failure of any component could result in a loss of production. The probability of such loss must be estimated

and avoided by conservative design and adequate surge (stockpiles or bins) capacity.

High initial investment is required for the belt system. The risk incurred here is that of the mine ceasing operations early, or of a long-term interruption of operations. The probability of these things occurring before the project has paid out its scheduled returns must be evaluated in making the final mining system evaluation.

B.2.8.7 Rationale for Mining System Selection--Constraints and Economics

In early 1974, a study of alternative haulage methods was initiated because cost forecasts for truck haulage out of the deepening pit were excessive. Several plans for shortening truck haul distances were examined. The arrangement illustrated in Figure 2-59 was selected because it had little impact on mine planning. Other conveyor routes were more direct but obstructed access to necessary stripping.

Return-on-investment analysis showed that the crusher-conveyor system was a viable project due to the possible reduction of 12 trucks from the haulage fleet which would justify the capital investment of $30.7 million which in addition to direct cost savings, replacing trucks by conveyors had other advantages:

- It allowed the combining of two pits.
- Truck haulage is more sensitive to inflation.
- Truck costs increase exponentially with increase lift height.
- Truck haulage is dependent on skilled maintenance labor which is in short supply.
- Trucks are less efficient energy users than conveyors. Truck fuel is subject to sharp price increases and shortages that could result in rationing or being put on allocation. On the other hand, coal or nuclear-fueled electrical power is readily available for conveyors.

A significant positive impact on overall mine operation could also be achieved through use of a conveyor network. Because of the savings in operating costs with belt haulage, the profitable life of the mine should be extended. The savings would permit a lower-grade ore to be mined, with a resultant increase in reserves.

B.2.8.8 Conclusion

The major factor affecting the outcome of the engineering-economic analyses on the different mining systems was the operating cost of the dump trucks--which is very difficult to calculated as a function of increasing pit lift. Unit operating costs for crushing and conveying, on the other hand, can be closely estimated by allowing for changes in power consumption for various lifts.

The final "caveat" on this decision making process involving truck versus conveyor haulage has been well summarized by mine management *(Ref. D.8.1)*. © *AIME, 1976*

> The long-term consequence of continued truck haulage could be very serious. The possible effects on truck haulage by inflation, fossil-fuel shortage, and high cost of labor have to be considered. If haulage costs by truck continue up with lift at a geometric rate, they could abruptly shorten the life of the mine. The evaluation of these "irreducible" factors is the vital function of management in the project analysis. It is an essential part of the proper decision.

B.2.9 Svappavaara Mine--Sweden

Mine management at Svappavaara, an open-pit operation, working on the Leveaniemi deposit, has already performed an extensive study on the various alternatives concerning the future transition to underground mining. Although results have shown that, based upon currently available equipment and the prevailing world market for iron ore, pit life should be good for at least 25 years, management believes in maintaining an up-to-date comparative analysis on the underground option. The tell-tale factor has, for the time being, been expressed in terms of a cut-off grade of ore (< 45% Fe) at which point underground mining should be practiced. With this level of detailed information at their disposal, management feels confident that a timely decision, complete with trial underground mining efforts, will be made.

The long-range planning effort, characteristic of Swedish mining operations, is clearly evident at Leveaniemi (see Section B.1.3.7). Because of Kiruna's experience in the transition from surface to underground mining during a time of rampant technological development (see Section B.1.2.7), LKAB will be careful to explore its options before becoming fully committed to the changeover.

The following sections present a brief description of current operations at Svappavaara (herein referred to as Leveaniemi) and conclude with a review of management's thinking on how to properly determine the right time for changeover. Table B-10 lists the principal characteristics of the present open pit. Figure B-49 shows an artist's sketch of the fully expanded pit in the year 2000. The existing processing facilities are drawn to scale as shown. Figure B-50 shows the present pit which is covered by snow for 8 months of the year.

TABLE B-10

FACT SHEET ON THE SVAPPAVAARA (LEVEANIEMI) OPEN-PIT MINE

- Owner - Luossavaara-Kirunavaara AB (LKAB), a Swedish Government-owned company
- Location - 47 km southeast of Kiruna, Sweden (north of Arctic Circle)
- Ore - Iron Ore
- Mineralization - Magnetite, Hematite, Ore Breccia
- Pit Operations Started - 1964
- Estimated Start of Joint Underground Operations - 1990's
- Estimated Cessation of Pit Operations - 2,005+
- Production - 14 million tpy - ore and waste
- Potential Mine Life at Present Production Rate - 35 years
- Operation Personnel - 137 (plus an additional 60 belonging to subcontractors)
- Operations - Conventional Open-Pit
- Equipment - Trucks and Shovels
- Problems:
  - Weather: extreme cold in winter; wet conditions during spring thaw
  - Shortage of skilled labor

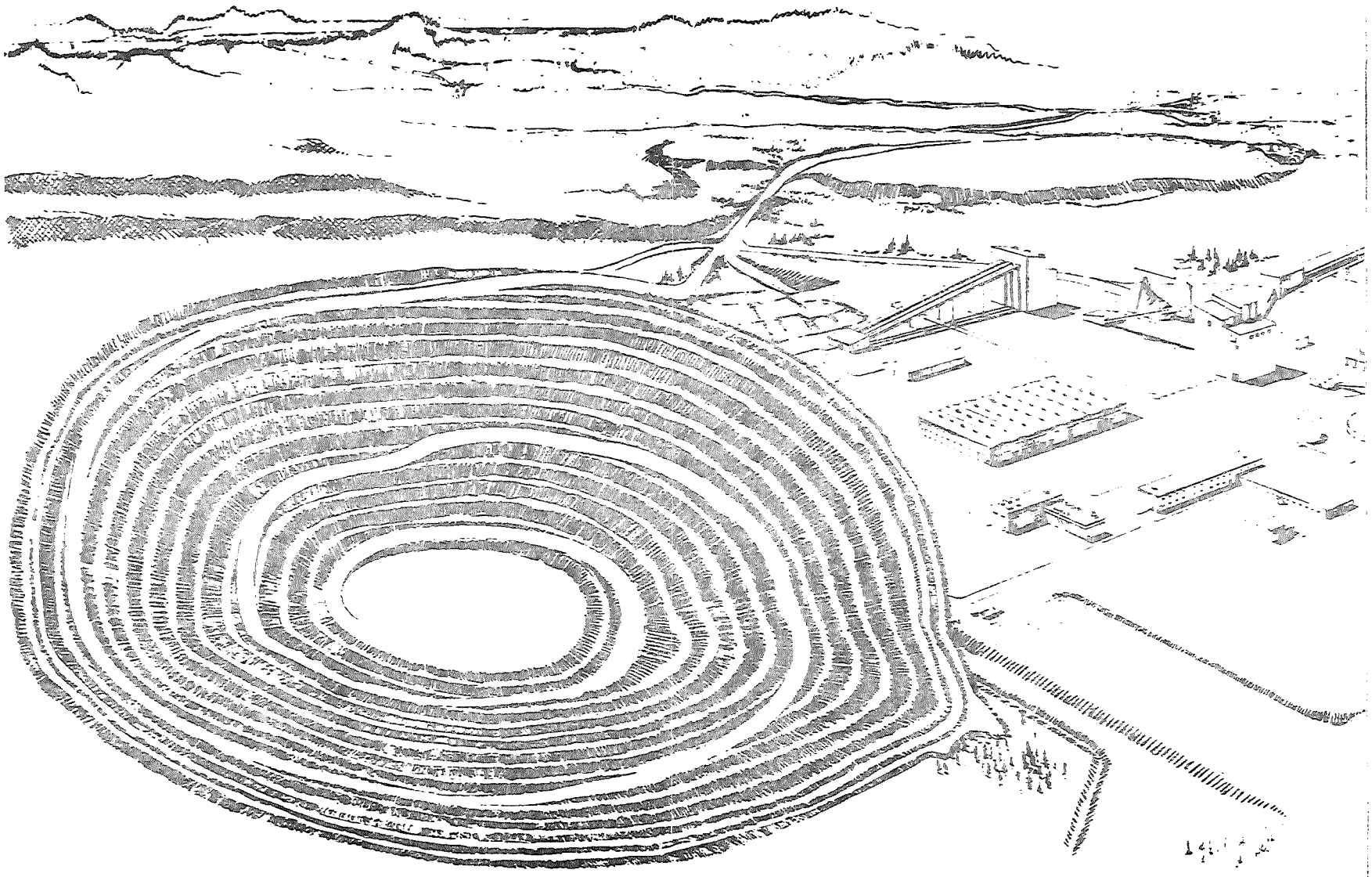


Figure B-49 An Artist's Sketch Of The Svappavaara (Leveaniemi) Mine in The Year 2000

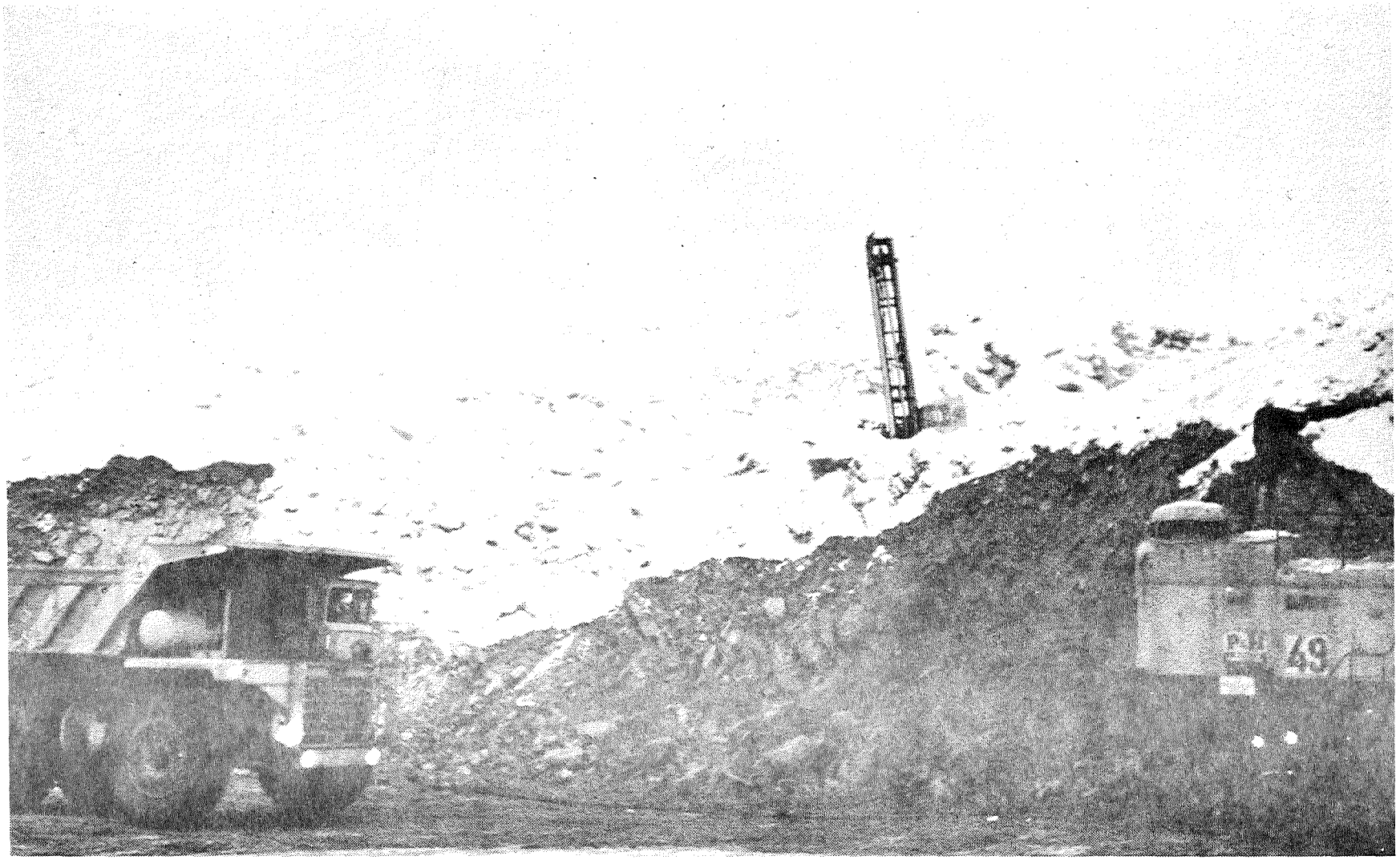


Figure B-50 The present Svappavaara (Leveaniemi) Pit Which Is Covered By Snow 8 Months Of The Year

B.2.9.1 General Mine History and Orebody Characteristics

The Leveaniemi Mine is located 47 km southeast of Kiruna, where its sister mine enjoys the reputation of being the world's largest underground iron ore operation. LKAB, the Swedish government's mining conglomerate, owns Leveaniemi as well as Kiruna, and therefore achieves certain economies for the relatively small open-pit mine by having the larger operation provide support as required.

The area containing ore at Leveaniemi extends lengthwise over about 1,300 m and has a maximum connected width of 150 m. The maximum depth from the surface is approximately 550 m and is crescent shape, as shown in Figure B-51, but the lower part with less dip creates a sail-boat hull shape. The total area of ore is approximately 130,000 sq m, containing over 300 million tons of ore and with the mineralization breakdown as shown below:

Mineral	Tonnage of Iron Ore (x 1,000,000)	Area of Deposit (sq m)
Magnetite	126	65,000
Martite and Hematite	45	25,000
Magnetite with high $CaCO_3$ content	33	5,000
Ore Breccia (25%)	103	35,000
Total	307	130,000

The ore breccia is for the greater part magnetite and waste.

From the mining point of view, two ores can be distinguished, namely martite and concentration ore. The most extensive martitization has occurred in a faulted zone in the central part of the east shackle of the orebody. The average iron content in the mining operation is 63± 1% for the martite and 45± for the concentration ore. The cut-off grade for the concentration ore is 25% Fe.

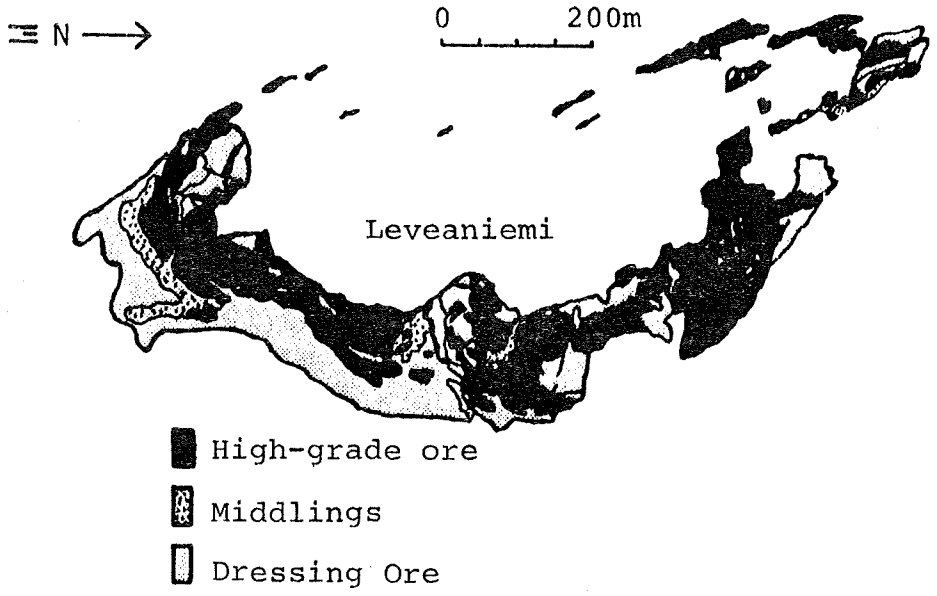


Figure B-51 Crescent Shape Of Orebody (Ref. D.1.2)

Although the present day mine fields were discovered in 1897, extensive mapping of the Leveaniemi deposit did not begin until 1957. In 1961, LKAB decided to commence operations and production of ore finally came on line in 1964. The current annual mining rate is running in excess of 14 million tpy with over 4 million tons being iron ore (waste-to-ore ratio is 2.5:1).

B.2.9.2 Mine Layout

Present and ultimate dimensions for the Leveaniemi open-pit are listed below:

	Present	Ultimate
Length	1225 m	1250 m
Width	800 m	950 m
Depth	100 m	300 m

Bench height is 10 m and will on ultimate design slope of 42^{o} to 45^{o}. Bench width runs 10 to 20 m. Two main haul roads, 10 to 25 m in width service the pit and ore inclined at slightly greater than 8%.

The location of the various plants at Leveaniemi are shown in Figure B-52. In the planning work carried out by LKAB at Kiruna, consideration was given to the location of the plants in relation to the deposit at Leveaniemi and to the exploitation of possible nearby deposits.

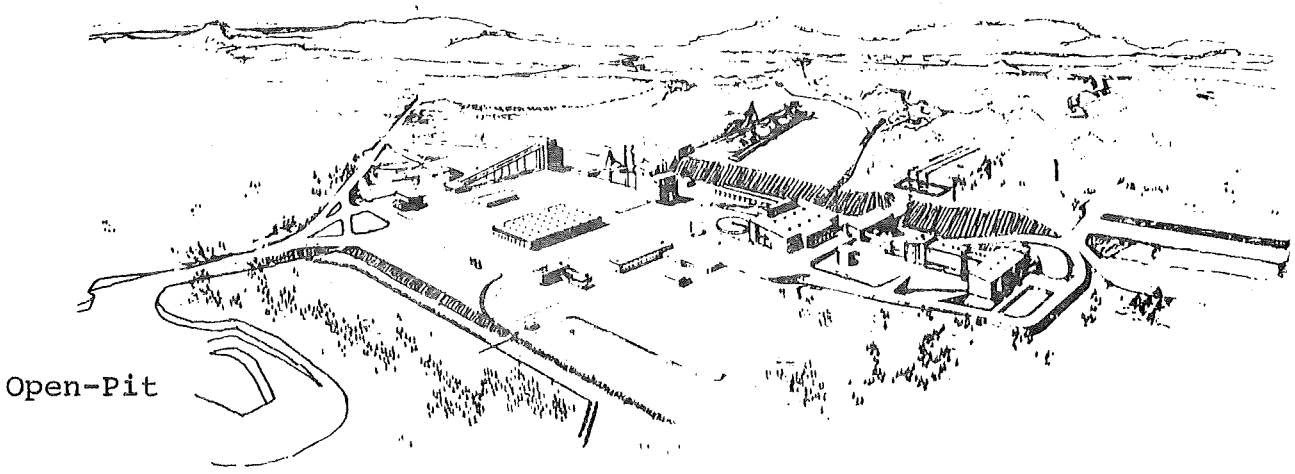


Figure B-52 Plant Layout At The Leveaniemi Open-Pit Mine

B.2.9.3 Mining Operation

Mining production history and annual tonnages for 1976 are shown below. All figures are in millions of tons.

	1976	Up To and Including 1976	Remaining in Place
Ore	4.0	41	137
Waste	10.0	53	155
Soil	0.7	11	1
Totals	14.7	105	243

Current ore production consists of:

- Direct ore (76.0% Fe, 0.45% P, crushed to 90% - 30 mm and screened into lumps and fines at 6 mm). No production in 1976.
- Concentration ore (magnetite 45% Fe, 0.6% P). This ore forms the raw material for the wet concentrating plant, (3.2 million tons) with the remaining concentration ore (0.8 million tons) transported to Kiruna for concentration-pelletizing.

Stockpile handling amounts to approximately one million tpy. Figures B-53 and B-54 shows the stockpile and reclaim facilities. Waste and soil are loaded and hauled partially by contractors (5 million tpy). Up to and including 1976 contractors have loaded and hauled about 30 million tons of waste and soil.

Extensive drilling and blasting trials were carried out before production commenced at Leveaniemi. Technical and economic calculations resulted in a bench height of approximately 10 m. From the standpoint of loading and transportation, a burden of approximately 3.2 x 3.6 m in ore and slightly more in waste rock appeared appropriate. From the standpoint of rock breakage, it was found that the holes would best be drilled at a slope of 70° with a 4½-inch

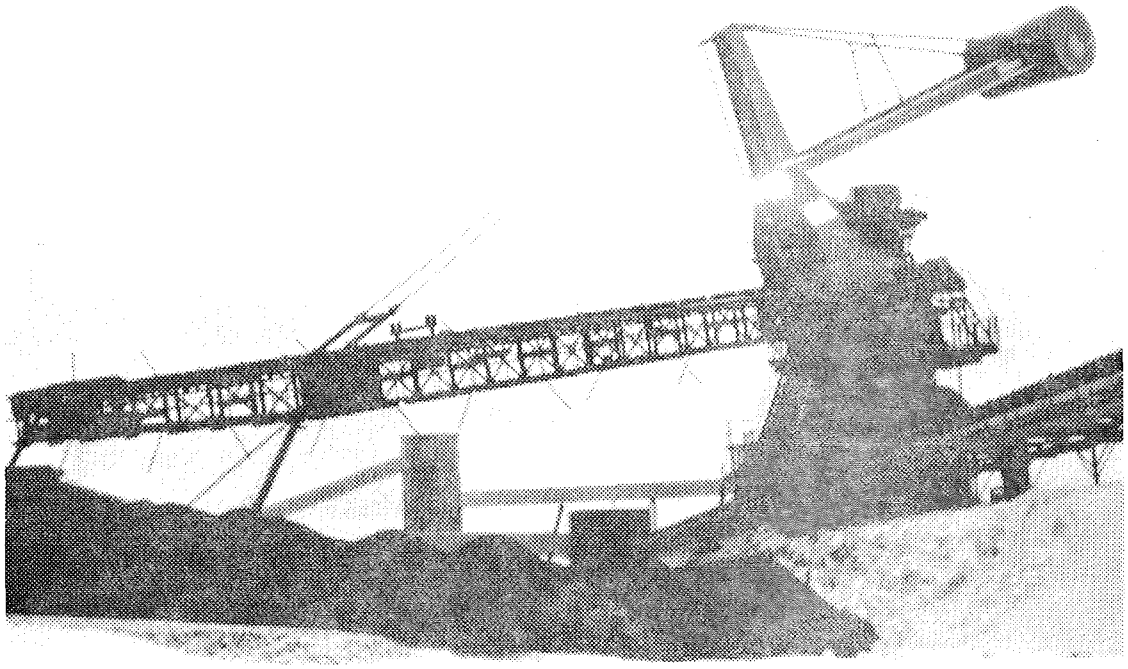


Figure B-53 Stockpile System At The Leveaniemi Open-Pit Mine

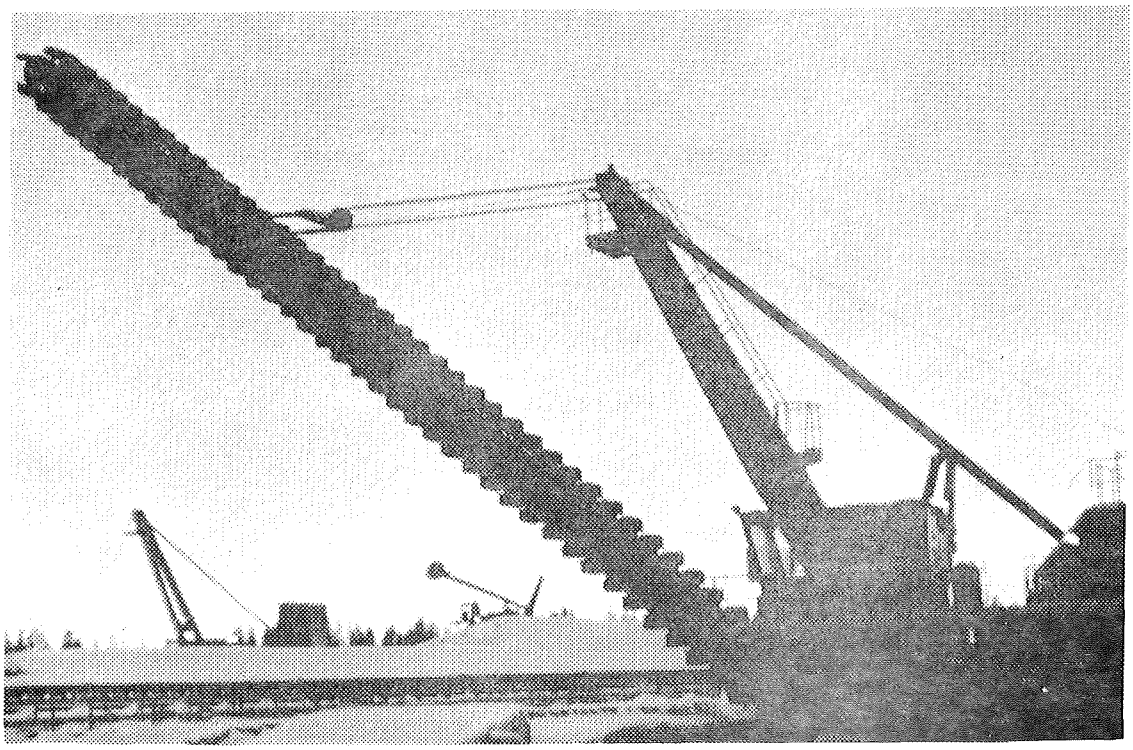


Figure B-54 Reclaiming Conveyor for Concentrating Plant At The Leveaniemi Open-Pit Mine

hole. The average rock displacement, with different species of rock, was approximately 40 tons per drilled meter, and in ore approximately 2,000 to 2,400 tons per shift per machine. The rock broken counted per kg of explosive has not been satisfactory less than 3 tons per kg of slurry.

A gelled slurry is used at Leveaniemi because of the fissured rock and the presence of water in the drillholes. The gelling agent does not act too quickly which makes it possible for air and water in the holes to pass through the slurry. Before loading, the holes are blown clean. Approximately 2% of the holes have to be redrilled prior to loading. As a rule, the blast produces approximately 20,000 tons of rock. Secondary blasting has taken place--despite the fissurous nature of the rock--only to a very slight extent, and is done with boom mounted, small-scale bench drilling machines.

Loading of blast rock is handled with shovels and front end loaders. Haulage is accomplished by large diesel-powered trucks.

B.2.9.4 Mining Equipment

Fifteen drill rigs are used to create the blast holes, 10 of which are pneumatically driven and the remaining five diesel powered. In addition, there is one diamond drill and a small secondary drilling unit mounted on a converted bucket loader. Slurry for the blasting is supplied by a 20-ton truck dispatched from the LKAB factory in Kiruna.

Loading is partially handled by five electric shovels with 5 and 6 cu yd buckets. Clean loading height is approximately 10 m. Production per machine is on the

order of 380 tph. Three rubber-tired, diesel driven front end loaders with 7 cu m buckets augment the operation with each machine also having a 400 tph production capacity. About 25% of the loading relates to the handling of the stock piles.

Haulage is done with diesel-driven trucks of 82.5 mt capacity.

B.2.9.5 Durability, Maintenance, Manpower, and Energy

Approximately 140 LKAB personnel work at the Leveaniemi Mine. Because the nearby Kiruna Mine is the parent operation for the area, the larger equipment maintenance jobs, such as engine overhaul and motor rewind, are handled there where adequate facilities exist for work of this magnitude. Local contractor support (an additional 60 people) is solicited in the area around Leveaniemi to provide additional haulage capacity for waste rock and soil removal.

For road maintenance in the pit, one road grader is used. One specially designated haulage truck is employed for miscellaneous material haulage in the open-pit mine, and one radio-dispatched, all-purpose truck is at the disposal of the supervisory staff. Radio communication is possible between all loaders, the road grader, and the haulage trucks.

Because the winter at Svappavaara is long and frequently cold (-20° to -35° C), the following procedures have been adopted:

- Drilling machines are run on thinner oil and are filled with warm oil every morning. If this technique does not help, the hydraulic tank is put under pressure.

- In loaders and trucks, thinner winter oil is used, and thinner grease is used in the excavators. The gear boxes in the loaders are provided with electric heaters.

- The roads are salted with calcium chloride. In addition, ploughing, road-grading and surfacing with asphalt have sufficed to avoid interruptions in production.

Because the iron and phosphorus contents vary within wide limits, the cuttings from test drillholes are sampled, and the results recorded on an analysis map. In order to obtain a rough control between different grades, intensive diamond drilling has taken place using 50-m deep holes in 25 x 25 m test patches.

Each loose-fired blast is sampled and sent to the laboratory for testing of the ore grade. The concentrating plant also samples their incoming material every other hour, and reports deviations to the mine for relocation of the loading machines. Sampling is the responsibility of the geological section at Kiruna which provides full-time work for two men.

B.2.9.6 Risk Areas and Technology Gaps

Mine management at Leveaniemi has been acutely aware of the attendant negative factors that come along with increased equipment size in open-pit operations. Specifically, larger machines lead to a corresponding increase in space requirements for movement, and this in turn calls for additional waste rock removal. As a consequence, the waste-to-ore ratio rises and may exceed the limit at which it is still practical to continue with open-pit operation. (Unless, of course, the increased productivity of the larger machine can offset this effect.) The inescapable changes to pit dimensions at Leveaniemi resulting from use of bigger equipment are discussed in the following paragraph.

The level in the open-pit mine is lowered by removing the waste rock using a series of successive "turn-around platforms" as shown in Figure B-55. Each bench must also handle two-way transportation. As larger haulage trucks are introduced, the bench width must also increase as shown in Figure B-56. This means that the waste rock is removed at an angle of slope which becomes progressively more acute. At Leveaniemi, this angle would be reduced from 28^{o} to 23^{o} as shown in Figure B-57.

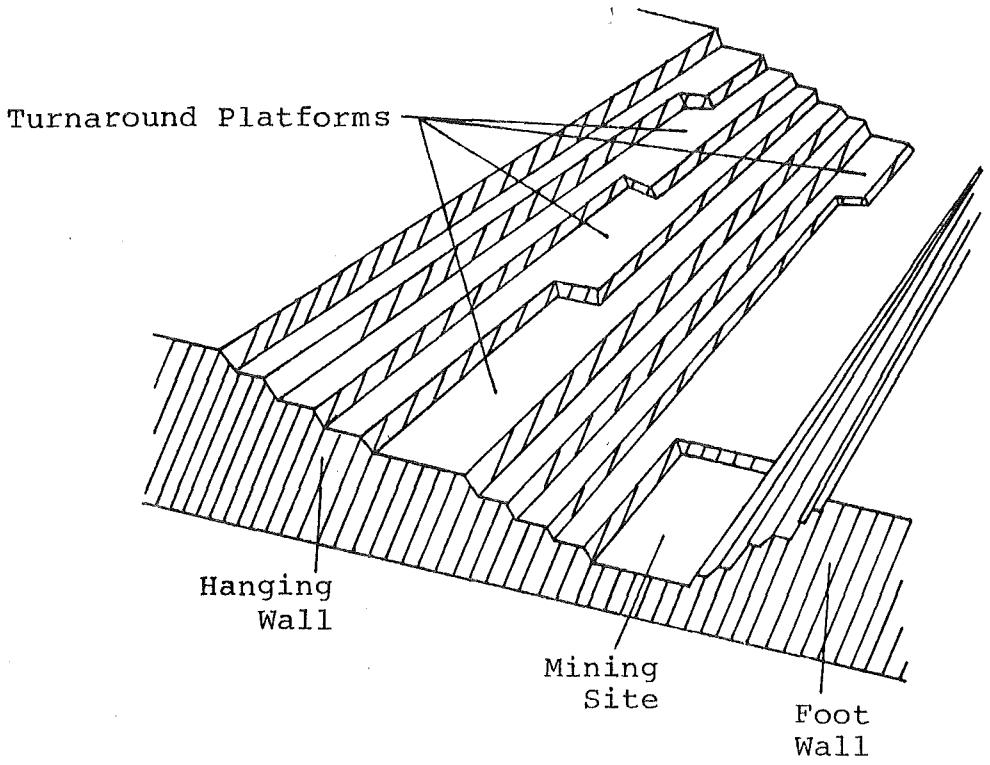


Figure B-55 *Schematic Diagram Showing The Lowering Of Levels And Successive Turnaround Platforms At The Leveaniemi Open-Pit Mine (Ref. D.1.6)*

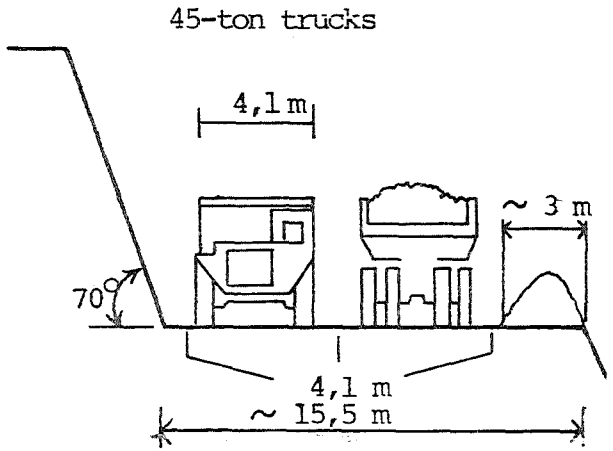


Figure B-56 *Required Bench Width As A Function Of The Truck's Size (Ref. D.1.6)*

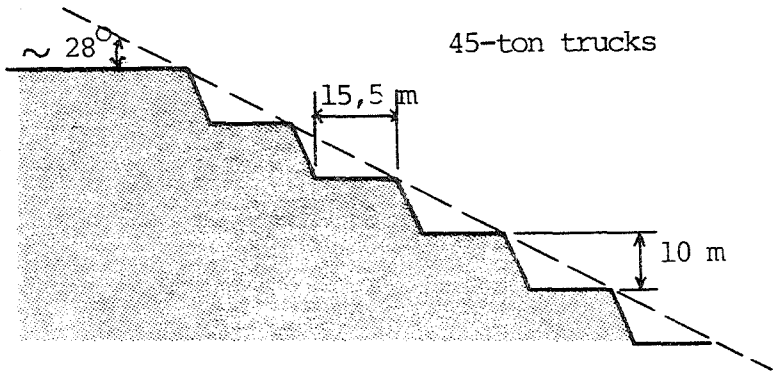
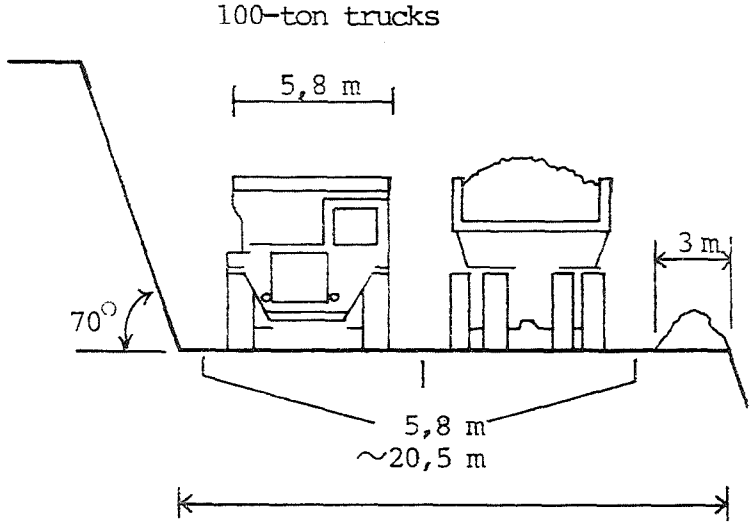
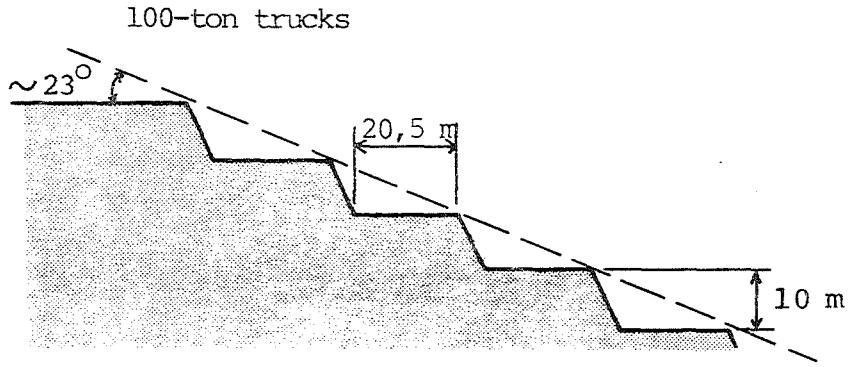


Figure B-57 *Effective Angle of Slope As A Function Of The Truck Size (Ref. D.1.6)*





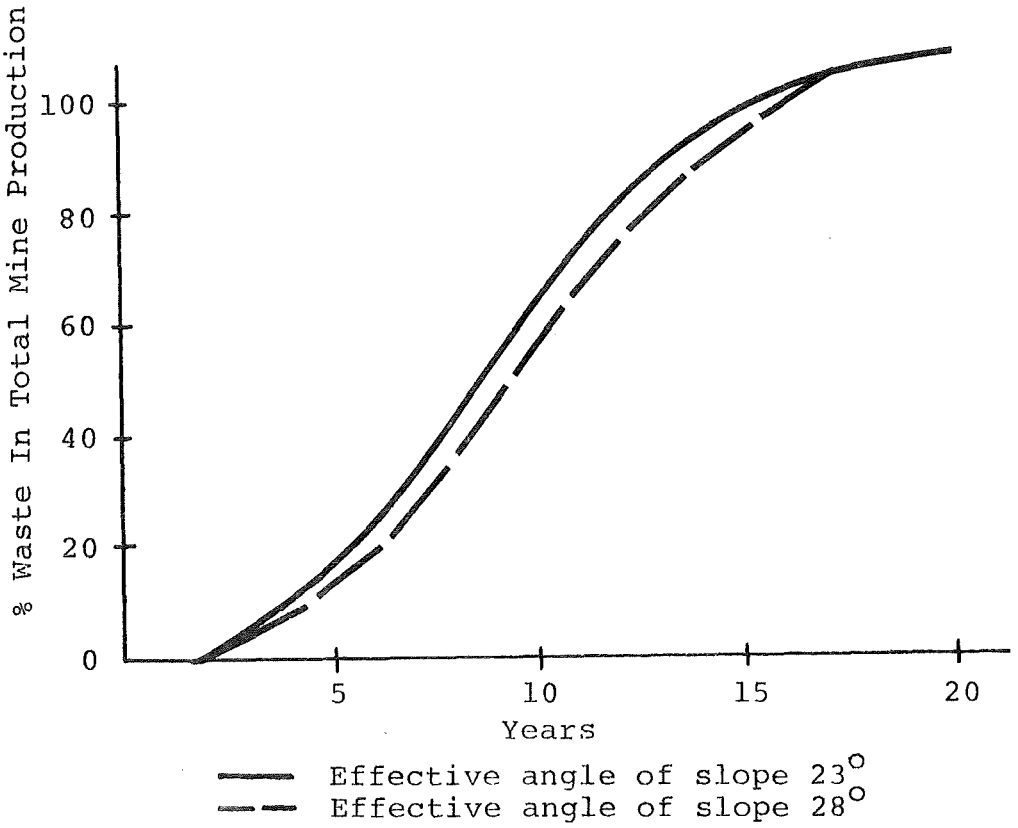


Figure B-58 Tonnage of Waste Rock Removal Required As A Function Of The Effective Angle Of Slope At The Leveaniemi Open-Pit Mine

As can be seen from above, this situation necessitates the removal of additional waste work (see Figure B-58). This corresponds to a cost of about eight cents per ton of waste rock, which is considerable. Profits due to the introduction of larger haulage units are not always sufficient to offset this increase. This point must therefore be taken into account when deciding on how large a haulage truck is economically practical.

Other points raised by Leveaniemi personnel included the following:

> The advantages of the open-pit mining are that the ore losses are small, different grades of ore can be sorted at the face, and the risk of accidents is much less than in underground workings. A disadvantage is that the deeper they get, the more

> waste rock has to be mined in order to avoid overhangs and the risk of collapse. Another drawback is that unfavorable weather conditions in winter and especially spring hamper the work. *(Ref. D.9.1)*

Pit operation in the middle of winter, for example, is done in virtual darkness. The wet ground conditions in spring create movement problems for the large equipment in the pit. Haul roads in particular require special attention.

B.2.9.7 Rationale for Mining System Selection--Constraints and Economics

With reference to the question of ultimate open-pit depth, LKAB research has suggested a number of alternatives, which can be seen in the mine's longitudinal section schematic of Figure B-59 Variables looked at included the amount of ore to be mined above and below ground, the corresponding life values of each operation, the angles of permissible slope in the pit (the average being in the range of 45^{o}), the cost of facilities required in the case of an underground operation, etc. The optimal solution recommended the intermediate alternatives (Case B in Figure B-60) and also established that if 45% of all ore was mined underground, a future underground operation would then be profitable.

This proposed solution means that about 140 million tons of Leveaniemi ore should be mined by the open-pit method, the total ore available being 275 million tons. Hence, the open-pit mine would have a life of about 30 years as shown in Figure B-61. Subsequent forecasts regarding future technical development suggest a somewhat deeper open-cast alternative than has now been decided. Nevertheless, as regards the last ton of ore to be mined by open-pit, LKAB can only afford a stripping ratio of 3:1, no more.

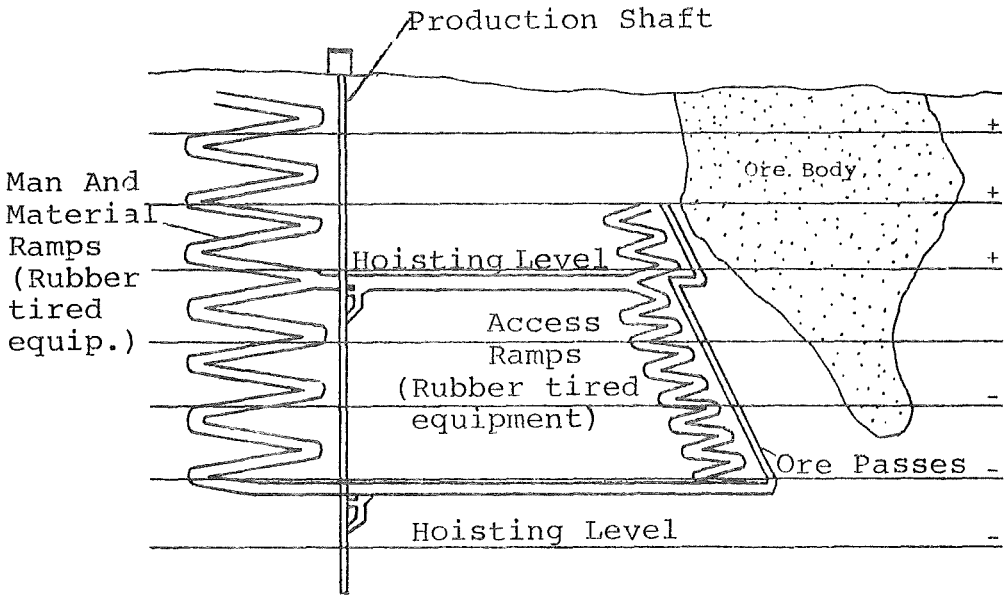


Figure B-59 *Proposed Leveaniemi Haulage System To Handle Nearby Orebody As Well As Present Open-Pit (Ref. D.1.6)*

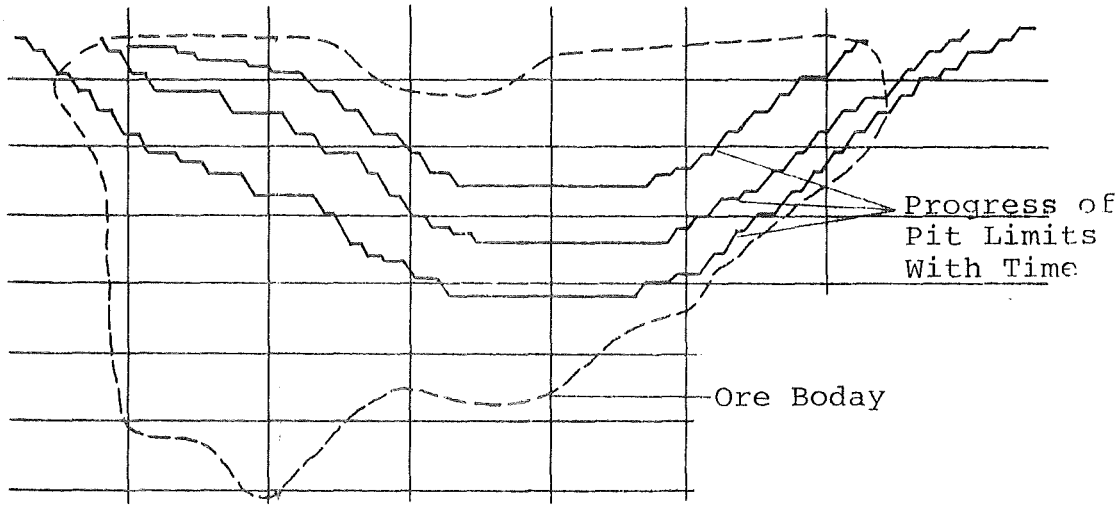


Figure B-60 *Longitudinal Section Of Orebody In Leveaniemi; Optimal Degrees Of Open-Pit Mining (Ref. D.1.6)*

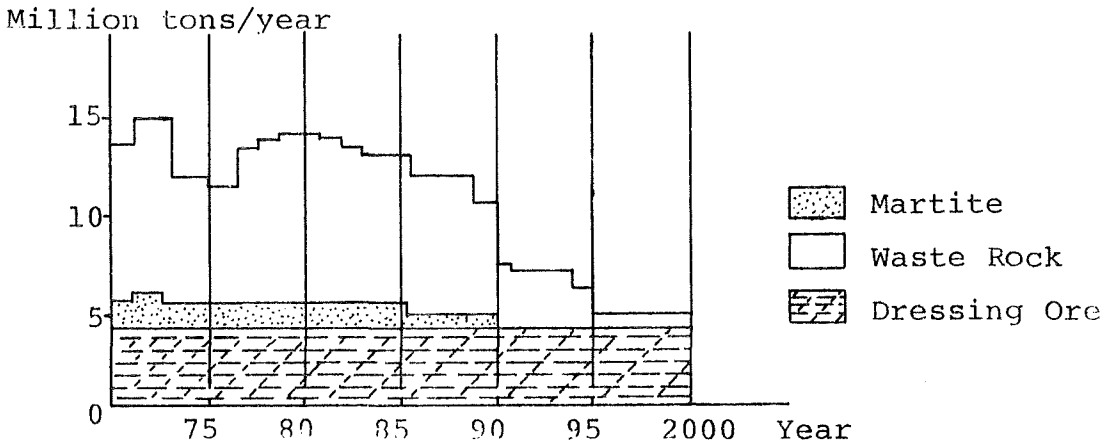


Figure B-61 Projected Plan For Open-Pit Mining At Leveaniemi To The Year 2000 (Ref. D.1.6)

B.2.9.8 Conclusion

What seemed most interesting and important from the standpoint of this program on haulage is the amount of planning and research done by LKAB to more precisely pinpoint when and how an underground operation should commence. For example, Leveaniemi management expressed one concern about development of the area directly below the present pit and the relationship of this mining plan to the presence of a second orebody nearby. Possible schemes include one to drive a slope between the two orebodies and connect the slope with horizontal haulage drifts as shown in Figure B-58. The slope would be equipped with a hoist for ore movement. However, they are very worried about hoisting of ore on an incline

because of a lack of experience using this technique. Rates of wear on rails and hoist rope are the principal unknowns.

A second proposal being actively considered for underground development is to drive an incline near the present pit and use it for haulage of open-pit ore now and later for haulage of underground ore. This proposal would only serve the main orebody, however.

In all of their plans it is obvious that LKAB plans a heavy use of diesel equipment for the movement of ore in some manner.

The advantages of the underground road include reduced stripping requirements for the present haul road, a lower maintenance effort on the haul road, and increased slope angles in the open-pit. Men and supplies alike will have access to the underground operations by way of an incline which will be large enough to provide passage for large LHDs.

In summary, Leveaniemi management plans to eventually conclude its open-pit operation. An interim step prior to full underground mining will be the installation of a below pit continuous haulage system to prolong the pit life and facilitate underground development. This move to continuous underground haulage will gain the following advantages:

- Lower stripping requirements.
- Achieve a greater slope angle on the pit walls.
- Lower maintenance on haul roads.
- Better extraction ratio for the ore.
- The new slope can also be used for future underground mining.
- Lower ore haulage costs are possible with conveyors or skips.

And finally, the haulage methods undergoing comparative cost study include:

- Truck Haulage - conventional, open air (Plan PO)
- Truck Haulage - Road underground, diesel and electrically driven (Plan NA)

- Conveyor Belt - Portable open-pit crusher (Plan PO_3)
  - Permanent underground crusher
- Skip - vertical
  - inclined

In Table B-11 there is a comparison of several of the above methods showing the final pit slope angles and the available one that can be recovered. The economics were not included, since the study had not been completed at the time of this report.

TABLE B-11

PIT SLOPES AND ORE RECOVERY FOR THE LEVEANIEMI PIT FOR THREE HAULAGE PLANS

	PLANS		
	TRUCK HAULAGE (Conventional, Open air)	CONVEYOR BELT (Portable open pit crusher)	TRUCK HAULAGE (Road underground, diesel & electrically driven)
Final Pit Slope	42°	42° (excluding crusher)	42° (underground access haulage roads)
Available Ore (Millions of Metric Tons)			
Magnetite	136	131	128
Hematite	8	8	8
Waste	150	139	59

Recent Stripping Ratio: 1:1.2

With Underground Haulage Road: 1:0.6

B.2.10 Open-Pit Copper Mine--Arizona

The fastest growing open-pit copper operation in the U. S. has selected continuous conveyor haulage as the most viable method for hauling both waste rock and ore to the surface. Key factors contributing to this decision included:

- *Large volume operation (has averaged almost 100 million tpy of waste and ore over the past 10 years)*
- *Deep pit (over 700 feet down to top of ore deposit)*
- *Extensive long-range stripping program (waste-to-ore ratio exceeds 10:1).*
- *Economics*

The evaluation of in-pit crushing and conveying versus truck haulage was the key issue in mine management's planning phase for the open-pit operation. Their assessment concluded the following *(Ref. D.10.1):* © *Mining Congress Journal, 1974*

> Conveyors, if used in the proper application, can come out on top of pure truck haulage. Deep pits, with high average lifts can, in many cases, use high capacity conveyors advantageously. Our operation has compiled empirical data on both conveyor and truck haulage that can be extremely useful in the design of material handling systems for open-pits.

Needless to say, conveyor haulage has worked well for this open-pit in its 13 years of operation. This section will provide some insight into the equipment performance analysis made by management during the planning phase and will conclude with some actual data on how well conveyor haulage has fared compared to trucks. Table B-12 provides basic information on the current open-pit operation.

B.2.10.1 General History and Orebody Characteristics

This particular open-pit operation, located about 30 miles southwest of Tucson, Arizona, involves one of the largest

TABLE B-12

FACT SHEET ON THE OPEN-PIT COPPER MINE

- Location - 30 miles southwest of Tucson, Arizona
- Ore - Cu, MoS_2, Ag
- Mineralization - Chalcopyrite, calcocite, covellite, bormite
- Open-Pit Mining Operation Started - 1965
- Annual Production Rate (1976) - 7.5 million tons ore
  76 million tons waste
- Potential Mine Life at Present Production Rate - 25 years
- Operations:
  - Truck and shovel feed in-pit crushers
  - Inclined conveyors haul material to surface
- Equipment:
  - Electric shovels - 15 cu yd buckets
  - 75- and 100-ton dump trucks
  - 54 x 80-inch gyratory crushers
  - 60-inch steel belts for main conveyor haulage
- Problems:
  - Conveyor capacity overdesigned
  - 100-ton dump trucks too large for crusher transfer chute design.

overburden stripping programs ever undertaken by a mining company. The geological nature of the orebody has required a significant departure from conventional pit design. The basic problem relates to the large amount and depth of overburden material to be removed -- more than 200 feet of waste rock and an additional 500 feet of alluvial gravel. The total lift requirement, therefore, totals over 700 feet simply to reach the top of the ore deposit. The project was to represent an initial stripping operation of 270 million tons of alluvium. The job could not be accomplished economically by truck haulage alone, so an earthmoving system was investigated.

Extensive studies at the outset of open-pit mining in 1965 showed that a system of conveyors and in-pit crushers working in conjunction with normal shovel-truck operations would be the best approach. The total system was installed and running by 1968 and has operated continuously since providing validation of the initial investment analysis.

As for the orebody itself, geological studies have focused on a persistent southeasterly trending mass of altered and mineralized quartz monzonite porphyry averaging 450 to 550 feet in thickness and buried beneath a post-mineral gravel cover which fill a valley area where the pit is now situated. Extensive drilling has revealed continuous copper oxide and underlying sulfide mineralization, principally in the flanking Paleozoic limestones, siltstones and Mesozoic clastic rocks, along and beyond the contacts of the quartz monzonite prophyry instrusive. Unlike many of the porphyry copper deposits in the area, however, the quartz monzonite porphyry contains only scattered, overall weak mineralization. Chalcopyrite is the most abundant and widespread copper sulfide mineral, but the mineral suite includes secondary chalcocite and covelite and minor bornite Molybdenite occurs in variable but economically important amounts throughout the mineralized zone. Some sphalerite and galena occur locally. Trace amounts of gold, insignificant silver and widespread pyrite occur and in places magnetite is abundant.

The two iron minerals are found mostly in the mineralized limestone and siltstone beds.

The orebody proper is actually segregated into 5 vertically positioned bands with waste hoists in between. The acceptable cut-off grade for ore has been set at .3% copper.

B.2.10.2 Mine Layout

The overall layout of the entire open-pit mining, milling, and waste handling operation is shown in Figure B-62. The pit itself is somewhat circular in nature with a circumference of over 6,000 feet and has a depth of 1,130 feet. The bottom bench is at elevation 2,220 feet, the top of the rim is at elevation 3,350 feet. Under current planning, the pit may eventually be only a few hundred feet deeper.

The orebody does not out crop but, as mentioned in the introduction, is covered among other things by up to 500 feet of post mineral gravel (alluvium) on the east side. The orebody contains very few boulders over 6 inches in diameter and is sufficiently consolidated to permit mine roads with almost vertical banks. The wall slope on the alluvium side is on the order of 60^{o} while the rock side has a much more gradual variation of 27^{o} to 45^{o}. Bench height runs on the order of 40 feet.

As the pit was deepened and expanded, it was necessary to extend and raise the height of the waste dumps which surround the pit. Several windows in the dumps have been purposely left to provide road and railroad access for sites such as the flotation mill, the new leach-LIX-electrowinning plant, shops, office, and other supporting facilities. The highest dump is at elevation 3,620 feet, so from top to bottom there is a vertical difference of 1,400 feet.

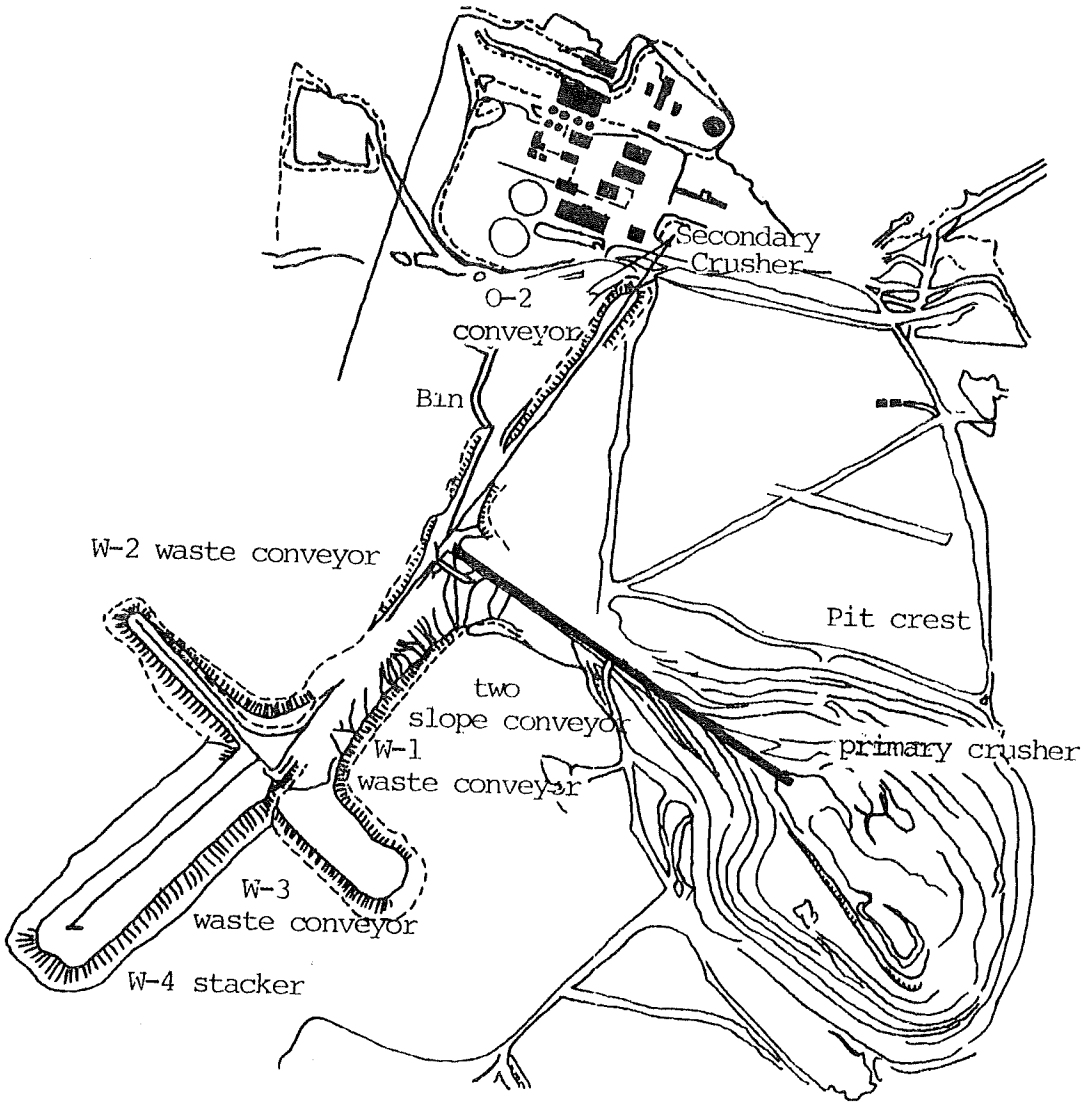


Figure B-62 *The Overall Layout of The Entire Open-Pit Mining, Milling, and Waste Handling Operation*

B.2.10.3 Mining Operation

The mining system uses 15 cu yd shovels to load 75 - and 100-ton trucks. The ore and waste rock are hauled in-pit to a mechanical vibrating grizzly with 10-inch spacings over a 54 - x 80-inch gyratory crushers. A 2 - x 16-foot secondary feeder receives the crushed product as well as the primary feeder undersize. This unit feeds a short 60-inch conveyor which in turn discharges onto the main 60-inch inclined belt conveyor (25% slope) for transport to the surface (2,520 feet).

The pit has identical twin units starting from the grizzly and continuing to the surface. This system permits the handling of mill ore, stockpile ore of various grades and rock waste to some extent simultaneously.

At the pit crest, the elevating conveyors discharge in line onto two other slope belts (5.8% gradient) which are 1,950 feet long. The head pulley of these belts is a carriage mounted on rails and snubbed to permit about 30 feet of lateral travel. A manned remote control tower near the end of these belts permits the operator to program the material flow to any one of several other conveyors depending on the type of material being conveyed:

- Sulphide ore to mill
- Oxide ore to new leaching plant
- Sulphide ore to stockpile
- Oxide ore to stockpile
- Waste to the stocker

Twenty shifts per week produce an average daily tonnage of 300,000 tons with generally a 10:1 waste-to-ore ratio. Table B-13 shows the recorder output for the time period 1965 through 1976. Waste removal is accomplished via one of three schemes (percentages shown are for 1976):

- 25% truck direct to dumps
- 25% truck to lower pit crushers
- 50% truck to twin conveyors (30% waste)

TABLE B-13

TONS OF WASTE STRIPPED AND ORE MINED AT OPEN-PIT COPPER MINE FROM 1965 THROUGH 1976

(Ref. D.10.2) © *World Mining, 1976*

Years	Tons Sulphide Ore Mined	Tons Waste Stripped
1965	---	7,140,460
1966	---	49,538,259
1967	---	89,555,730
1968	---	69,963,789
1969	2,866,202	48,383,472
1970	9,643,004	55,681,595
1971	7,975,686	73,222,262
1972	10,596,593	98,567,458
1973	12,400,572	115,231,981
1974	10,576,005	125,634,662
1975	3,047,094	146,152,549
1976	7,500,000	76,000,000
Totals	64,605,156	954,772,227

B.2.10.4 Mining Equipment

Bowl scrapers (30 - to 80-ton capacity) are used for the stripping operation. Crawler tractors are used for ripping the alluvium and pushing the scrapers during loading. Bottom- and end-dump trucks (75 - and 100-ton capacity) are used for local ore and waste haulage in the pit. Electric shovels with 15 cu

yd capacity are used to load the trucks. Dual primary gyratory crushers (54 - x 80-inches) in the pit feed twin 60-inch steel conveyor belts designed for 6,000 tph capacity. The belt passageway leading from the pit to the surface is a slot cut through the alluvium. The ore belt is roughly 7,000 feet long and the waste belt approximately 10,000 feet. The belts have 4-speed drive provided by staged 1,250 hp motors and run at 825 fpm.

B.2.10.5 Durability, Maintenance, Manpower and Energy

Sustaining the high daily volume output has required the implementation of several carefully thought out management techniques; for example:

- Each shovel is painted with a different combination of colors so that all supervisory personnel can readily identify each shovel from any part of the pit. This quick identification is important to direct empty trucks to the shovels.
- General equipment, availability has been remarkably improved by employing a 5-day mining cycle and two-day repair schedule. An extensive program of preventive maintenance has also helped to raise unit availability.
- The purchase of extra "standby" units of equipment which in turn become "replacement" units has enhanced total system availability and justified the added expense.
- The ore handling system from pit bottom to surface was not selected until the alluvium stripping conveyor had handled in excess of 20 million tons of material over a period of several months. This full-scale shakedown test was reassuring with actual operating costs some 4-5% below the original estimated costs.

A more far-reaching management decision regarding the main conveyor haulage system concerned the excavation of a slot in the pit wall rather than the construction of a slope. Analysis during the planning phase had shown that installation costs would be significantly less. Moreover, accessibility to the conveyors for repair and maintenance would be easier.

B.2.10.6 Risk Areas and Technological Gaps

The greatest risk associated with this continuous haulage method is, of course, the danger of conveyor malfunction. Because this mining system closely integrates the operation of mobile excavators and transfer haulage vehicles with belt conveyors, the familiar problem of equipment availability is compounded into a much more serious one of system availability. Should the conveyor breakdown for a short interval, all other equipment in the system is idled. When the conveyor is down for a protracted time, the delay is partially offset by routing the scrapers directly to the dumps and by moving men to other equipment standing by. Not much else can be done except to plan for such contingencies and be prepared to cope with them as the situation arises.

A couple of other unforseen problems have arisen with regard to the primary conveyor system. The belts, which were designed to handle a maximum capacity of 6,000 tph, have only been running at 4,000 tph. This has resulted in a higher wear rate and maintenance cost per ton hauled than had been anticipated.

The other problem relates to the so-called "portable" nature of the smaller conveyor sections on the pit rim. Apparently they are not relocated as easily as management had envisioned before they were purchased.

A more general observation by mine personnel regarding truck versus conveyor haulage is that the 300- to 350-ton dump trucks of the future will be competitive with the present ore/waste rock movement system.

A problem spot with the present truck operation is the oversized nature of the 100-ton dumps and the spillage that results around the transfer chutes to the primary crushers.

B.2.10.7 Rationale for System Selection--Constraints and Economics

The basic problem confronting mining company management during the planning phase related to the depth of the ore deposit and how an open-pit mining operation could be economically implemented. Complicating the situation was the fact that U. S. experience in high-capacity, high-lift conveyors was virtually non-existent at the time (mid 1960's), which left management feeling rather insecure when studies revealed that a system of in-pit crushers and conveyors would be superior to truck haulage. But this departure from conventional pit design appeared to be the only practical alternative, given the *geometric constraints* of this particular orebody; namely, the overlay consisted of over 700 feet of waste rock, oxidized material, and alluvial gravel--a formidable lift distance for any truck haulage operation. Consequently, the final decision to employ conveyors was made because the cost equivalent risk of installing this untried ore transport method was less than the readily quantifiable cost of utilizing a truck fleet.

The planning phase study on different haulage methods produced findings analagous to the data shown in Table B-14. For open-pits with over 200 feet of lift to haul out waste rock and ore, conveyors became a very attractive option. A 600 foot deep pit, for example, could experience a cost savings of almost 7 cents per ton by utilizing conveyors over trucks.

Other reasons for selecting conveyor haulage included the following:

- Environmentally, the hydrocarbon and noise emissions would be significantly less.
- A potential fuel shortage (which actually happened in 1974) would have less impact.

Some of the negative aspects of using conveyors that were investigated included:

- The cost of conveyor equipment was substantially higher than for trucks (in fact, several million

TABLE B-14

COMPARISON OF TRUCK VERSUS CONVEYOR HAULAGE
PIT WITH ANNUAL TONNAGE OF 30 MILLION
(20 MILLION WASTE AND 10 MILLION ORE TONS)

(Ref. D.10.1) © *Mining Congress Journal, 1974*

	TRUCKING COSTS		CONVEYING COSTS		ADVANTAGE (DISADVANTAGE) CONVEYOR TO TRUCKING	
Lift In Pit	Annual Capital Cost	Operating Cost Per Ton	Annual Capital Cost	Operating Cost Per Ton	Annual	Per Ton
100'	$1,889,000	$.0630	$2,252,000	$.0751	($ 363,000)	($.0121)
200'	2,477,000	.0862	2,489,000	.0830	(12,000)	(.0004)
300'	3,128,000	.1043	2,726,000	.0909	402,000	.0134
400'	3,846,000	.1282	2,963,000	.0988	883,000	.0294
500'	4,619,000	.1540	3,203,000	.1068	1,416,000	.0472
600'	5,458,000	.1819	3,440,000	.1147	2,018,000	.0673

dollars difference), necessitating a high capital outlay prior to start of operation.

- In-pit crushing and conveying would be less versatile in accommodating changes to operating layout or expansion plans.

After weighing the pros and cons of conveyor haulage, mine management went back to its analysis of truck operating costs versus increasing pit depth and decided that here was the determining factor for this particular operation. They felt that there was a strong correlation between the hourly operating costs for a truck fleet versus amount of lift in the haul cycle, a tradeoff which would have adverse impact on economy of operation the deeper the pit got.

As things turned out, the mine shut down its rock conveyors in 1973 for a period of about two weeks each in order to make modifications to the system and to change out some of the conveyor belting. At that time, the majority of rock mining activity was from relatively deep in the pit. Management noticed that the long upgrade hauls to the surface were adversely affecting truck availability and that the cost per operating hour of the trucks was climbing significantly. Truck operating costs were recorded as a function of lift height and the fuel consumption comparison shown in Figure B-63 was established. Mine management's initial planning efforts had been vindicated.

B.2.10.8 Conclusion

A broader review of why an open-pit was selected as the best mining method is in order to put the entire planning effort into perspective.

Preliminary lab work in drill cores coupled with experience gained during the pilot mill stage in the mid-1960's had shown management that the metallurgical amenability of the ore deposit was anything but uniform. Several complicating factors had to be taken into account in evaluating the geological and geometric constraints of the orebody:

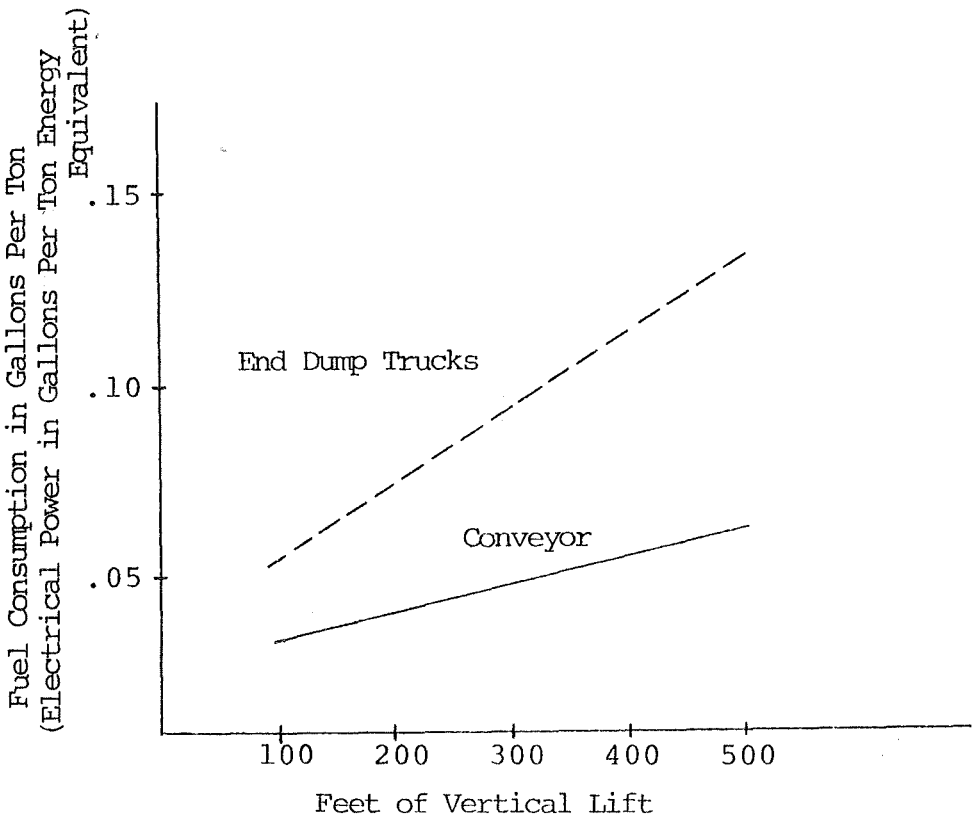


Figure B-63 *Correlations Between Average Life And Operating Cost In Hauling From A Deep Pit With 100-ton End Dump Trucks (Ref. D.10.1)* © *Mining Congress Journal, 1974*

- Diverse mineral associations
- Variable hardness and grindability
- Different amounts of oxidation
- Inconsistent quantities of slimy materials
- Fluctuating assay grade

Given these sets of circumstances, management paraphased its thinking and conclusions into the following brief statement *(Ref. D.10.3).* © *AIME, 1970*

> Because of the complex nature of the orebody it was recognized that it would be impractical to set up an ideal mining and milling combination. A truly selective underground mining method would be far too costly. Lower cost underground mining such as block caving, even if practicial, would be somewhat costly and also result in excessive dilution and admixture of various ore classes. This system might also affect mine extraction. What it all boiled down to was an open-pit mine designed to attain low unit operating costs essential to the economics of the venture.

This analysis set the stage for a good hard look at continuous haulage and resulted in its ultimate selection once the cost information (discussed in the preceding section) had been worked up. As with the open-pit operation described in Section B.2.9, similar geometric constraints pertaining to the nature of the ore deposit influenced an agressive management into implementing a continuous haulage scheme, despite the inherent risks mentioned in Section B.2.10.6. This should stand as reasonable evidence today that in-pit crushing and conveyor transport have a viable place in the movement of open-pit material.

APPENDIX C

WILLIS & PAUL CORPORATION

IN - PIT

Crusher Study
and
Conveyor Study

WILLIS AND PAUL CORPORATION

In-Pit Crusher Study

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INTRODUCTION

The purpose of this study is to develop data for approximating the capital costs of in-pit primary crushing facilities of various capacities. By utilizing this cost data for the crushing facilities along with the cost data established in the belt conveyor section of this report, one can arrive at an approximate capital expenditure required for in-pit crushing facilities of 1000 tph to 10,000 tph capacities and primary conveyors with zero to 2400 feet of lift.

Also, both surface and underground facilities have been evaluated. The main difference between the two types of facilities is the depth of the installation below the surface. In the surface facilities, trucks will dump into a rock box which feeds directly into the gyratory crushers or to a vibrating feeder in the case of the jaw crusher. The underground facilities are considered to have an ore pass between the truck dump rock box and the crusher. This ore pass would most likely be in the range of 50 to 2500 feet in depth, depending on mine plans.

DESIGN PARAMETERS

In this study we have attempted to allow for as many site specific variables as possible, and still arrive at representative capital expenditure requirements. In view of these numerous variables, a certain number of specific design requirements must be defined in order to provide a basis for estimating costs.

The first assumptions made are the following characteristics of the material being crushed:

- Material: Ore/Waste, clean and friable
- Density: 100 lbs. per cu. ft.
- Work Index: (Kw-Hr/Ton) 13 - 16
- Screen Analysis: 99% -4 ft.
  95% -2 ft.
  50% -12 inch.
- Material to be crushed to -12" maximum

This study also assumes the basic ore haulage to the crushing facility to be by conventional end dump trucks of approximately 170-ton capacity. Though variations to the size of trucks will most likely occur, the variations will have very little effect on the cost data developed herein. The cost of delivering ore to the crushing facility is not included in this study.

DISCUSSION OF FIGURES A THROUGH L

Figures A through L, in Appendix III, show the various crushing facilities analyzed for developing the cost data utilized in this report. Following is a list of these facilities:

Figure A - 1000 tph Jaw Crusher Surface Facility

Figure B - 1000 tph Jaw Crusher Underground Facility

Figure C - 1000 tph Gyratory Crusher Surface Facility

Figure D - 1000 tph Gyratory Crusher Underground Facility

Figure E - 2500 tph Gyratory Crusher Surface Facility

Figure F - 2500 tph Gyratory Crusher Underground Facility

Figure G - 5000 tph Gyratory Crusher Surface Facility

Figure H - 5000 tph Gyratory Crusher Underground Facility

Figure I - 10000 tph Gyratory Crusher Surface Facility Plan

Figure J - 10000 tph Gyratory Crusher Surface Facility Elevation

Figure K - 10000 tph Gyratory Crusher Underground Facility Plan

Figure L - 10000 tph Gyratory Crusher Underground Facility Elevation

Though site-specific conditions and customer preferance may result in variations to the facilities as shown, only major variations will appreciably affect the capital costs developed in this report.

The crushing facility capacity range of 1000 tph to 10,000 tph was selected as a range that would be applicable to the majority of open-pit mining operations in the United States. These capacities are material crushed not total throughput. However, the actual capacity of the facilities can substantially vary from these limits, depending on the amount of fines in the mined ore.

For instance, the capacity of the 1000 tph jaw crusher facility could vary from about 500 tph to somewhat over 1000 tph depending on the feed characteristics. Consideration of a single jaw crusher for a 1000 tph facility is possible only because of the assumed screen analysis of the ore (50%, -12"). If the jaw crusher is capable of crushing about 1000 tph and the feed material containing at least 50% fines which can be scalped and by-passed from the crusher to arrive at a total capacity of

1500 tph. The actual screen analysis of a particular mining operation will depend on blasting techniques, etc.

On the upper end of the capacity range, it is possible to process in excess of 18,000 tph of ore in the facilities shown in Figures I, J, K and L with relatively little modification such as a wider apron feeder. Since fines will flow through a gyratory crusher without being further reduced in size or decreasing its crushing capacity, the total capacity of the facility can be taken as the sum of the fines plus the amount of material crushed. When considering the installations with two 60 - 109 gyratories, which can crush about 6500 tph each, a feed of 13000 tph of oversized plus 5000 tph of fines gives a total capacity of 18000 tph. To aid in the proper selection of a gyratory crusher for a particular application, some manufacturers have developed a "fines factor" to compensate for the fines in the mined ore. This factor is applied to the estimated screen analysis of the run-of-mine material to help determine the actual crusher capacity.

Since the crusher is the main piece of equipment in a crushing facility, its proper application is essential for an efficient operation. Each installation must be evaluated with its own site-specific requirements and conditions to insure the proper application. The following information should be known in order to make the correct crusher selection:

Characteristics of the crusher feed:

- Type of ore/waste
- Bulk density

- Crushability or work index (Kw-Hr/Ton)
- Moisture content
- Quantity of fines (screen analysis)
- Dimensions of the largest lump
- Size of quarry and haulage equipment
- Method of feeding
- Desired capacity

Proper feeder selection is also an important part of a crushing facility. While most heavy duty, high capacity installations require apron feeders, modern technology is advancing vibrating feeders to where they are appearing more in heavy duty installations up to approximately 1500 tph capacity. Considerably less capital and installation costs of vibrating feeders as compared to apron feeders made them worthwhile considering in the proper application.

Apron feeders are still unequalled for performance in heavy duty, high capacity installations. The parameters which are utilized in selecting an apron feeder include:

- Maximum lump size to be handled
- Required capacity
- Size of feed hopper or bin opening

Apron feeders are available with capacities compatible with the crusher capacities (in excess of 6500 tph for a 60 - 109 gyratory crusher) previously mentioned. For instance, under the right conditions, a 126-inch-wide apron feeder could be capable of delivering approximately 10000 tph of 100 lbs. per cu. ft. material.

For both jaw crusher facilities, vibrating feeders have been utilized for feeding the crushers. By placing a grizzly ahead of the jaw crusher, and thus, bypassing the crusher with the fines, the capacity of the facility is increased. Also, jaw crushers operate more effectively when fed at a uniform rate.

Because of their large feed opening, high capacity and durability, gyratory crushers have been used for the high capacity crushing facilities. In the surface gyratory installations where the crushers are fed in surges from the trucks, there exists the problem of material being discharged in surges from the crusher. In order to prevent these crusher discharge surges from overloading the belt conveyor, an apron feeder is installed between the crusher discharge and the conveyor. The apron feeder also serves to absorb the vertical impact of the material being discharged from the crusher and starts the material moving in the direction of belt travel. This will increase the life of the belt and reduce maintenance.

The underground gyratory crusher installations utilize an apron feeder to regulate the flow of ore to the crusher. Since the feed to the crusher is controlled, the problem of material surges overloading the belt conveyor does not exist. However, depending on the capacity required, consideration should be given to an apron or vibrating feeder between the crusher discharge and the conveyor as a means of increasing belt life. Whichever method is used, adequate measures must be taken to insure a smooth flow of material to the belt conveyor.

Acceleration belts, as utilized in this study in all but the jaw crusher facilities, also provide a means of insuring a smooth flow of material to the primary conveyor. They accelerate material with little or no velocity in the direction of belt travel (such as ore being discharged from an apron feeder or crusher) to a velocity somewhat under the velocity of the primary conveyor. Then, by making an in-line transfer to the primary conveyor, material impact and turbulence wear on the primary belt is held to a minimum. Acceleration belts are relatively short, only long enough to allow the material to stabilize on the conveyor, and thus, belt replacement costs are minimized.

The capacities of the jaw crushers were much lower so the primary belts were run at a slower rate and acceleration belts not required.

DISCUSSIONS OF GRAPHS

The costs of the various crushing facilities described earlier have been compiled and used to formulate a graph of Material Crushed tph vs. Capital Cost for Crushing Facility in Dollars per tph. This graph, which can be found in Appendix I, covers cost data for surface and underground facilities. Special note is given to the portion of the graph which gives the tph of material crushed. These numbers represent the amount of material in the run-of-mine ore being fed to the crushing facility that will require crushing.

The following example illustrates the use of the graph along with the information necessary to use it:

Surface facility

Required capacity, 5000 tph

Product size, -12"

Amount of fines in feed, 50% -12"

5000 tph - 50% fines = 2500 tph to be crushed.

From the 2500 tph point on the horizontal axis of the graph, draw a vertical line upwards until it intersects with the surface facilities curve. From that point of intersection, draw a horizontal line to the left until it intersects with the vertical axis of the graph and read approximately $890/ tph. Then $890/ tph x 2500 tph = $2,225,000 which represents the approximate capital cost of the 2500 tph facility.

Although this is a simplified procedure, it gives results accurate enough for initial estimating purposes.

The costs represented in the Capital Cost Graph include engineering, equipment, procurement and erection of crushing installations based on the facilities previously described and as illustrated in Figures A through L in Appendix III. It has also been assumed that electrical power of adequate capacity and proper voltage is available for distribution in the vicinity of the crushing facility. Although the figures of the crushing facilities indicate concrete exterior walls on all installations, the cost of these walls is included in only the surface facilities. It is assumed that most surface facilities will require them, whereas in most underground facilities, rock conditions will be of adequate integrity so as not to require them.

The costs do not include costs for the ore pass in the underground facilities. These ore pass costs must be calculated by experienced estimating personnel based upon site specific data.

Calculations were carried out for crusher capacities of 1000, 2500, 5000 and 10000 tph. The overall results are summarized in the graph on page 358 which shows capital cost per tph as a function of plant capacity (tph). The backup calculations appear in Appendix IV. In addition, a final cost computation was run on the capital investment required for a 1000 tph jaw crusher facility to illustrate the significantly lower price compared to a gyratory plant (approximately 25% less). All cost estimates are in 1978 dollars.

Based on the curve of the Cost Data Graph, capital costs of crushing facilities per ton of ore crushed rapidly decreases as the capacities increase to about 2500 tph. From that point on, the cost does not change nearly so quickly as the tonnage rates increase.

The Horsepower vs TPH Graph can be found in Appendix II, showing the approximate total horsepower required for various capacity crushing facilities.

STATE-OF-THE-ART REVIEW

When evaluating the high capital cost of a permanent crushing facility against risks involved such as long term interruptions in mining operations and the possibility of the mine ceasing operations early, it appears that portable crushing facilities could prove advantageous. While there are portable facilities available in the United States, they are rather limited in capacity. Further investigation into the feasibility of high capacity portable systems should be done in view of the risks involved with a permanent installation.

Also, there are what might be considered some "technical difficulties" in the crushers. One of these difficulties is the frequent need to replace the wear surfaces of the crushers. If the life of the wear surfaces were increased, considerable savings would be realized when considering the cost of the downtime, the loss in production and the man-hours required to make the change.

Another area that seems to have good potential but needs further development is the use of jaw-gyratory crushers. These crushers, as produced by at least one European manufacturer in a small capacity range, combine into one unit the ability of jaw crushers to accept large lumps and the high reduction ratio and capacity capabilities of gyratory crushers. With these operating characteristics, jaw-gyratory crushers offer a potential for cost savings in blasting operations and possibly in secondary crushing.

WORK INDEXES FOR DRY CRUSHING*

Material	Sp.Gr.	Work Index, Wi
Andesite	2.84	18.25
Basalt	2.91	17.10
Bauxite	2.20	8.78
Cement raw material	2.67	10.51
Coal	1.4	13.00
Copper ore	3.02	12.73
Dolomite	2.74	11.27
Gabbro	2.83	18.45
Granite	2.66	15.13
Gravel	2.66	16.06
Hematite	3.53	12.84
Taconite	3.54	14.61
Limestone	2.66	12.74
Nickel ore	3.28	13.65
Oil shale	1.84	15.84
Quartz	2.65	13.57
Trap rock	2.87	19.32

* From Allis-Chalmers, Milwaukee, Wisconsin

Wi is a work index or communition parameter expressed as the gross power in Kwh required to reduce one ton of material from a theoretical infinite size to 80% passing a 100 micron (about 67% passing 200 mesh) square-mesh aperture.

APPENDIX I

COST DATA GRAPH

APPENDIX I

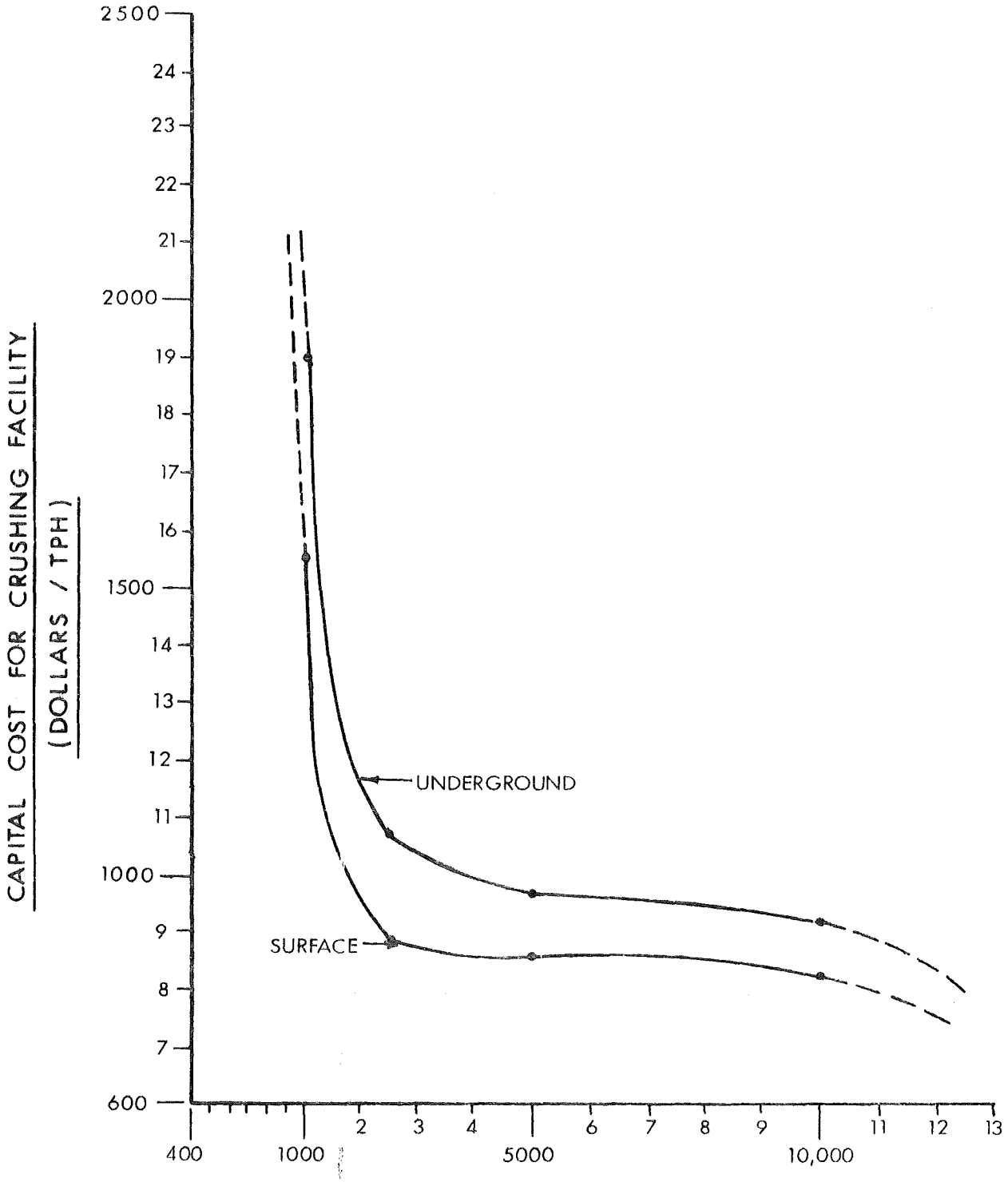
COST DATA GRAPH

This graph is used to make an initial estimate of the installed capital cost of surface or underground crusher facilities in the crushed rock capacity range of 100 - 10,000 tph. However, it does not reflect the cost of ore/waste passes necessary in underground crushing facilities.

For example, find the installed capital cost of a facility with the following requirements:

- Surface installation
- Total throughput 5,000 tph
- Product size -12"
- Amount of fines in feed, 50% -12"
- 5,000 tph - 50% fines = 2500 tph to be crushed

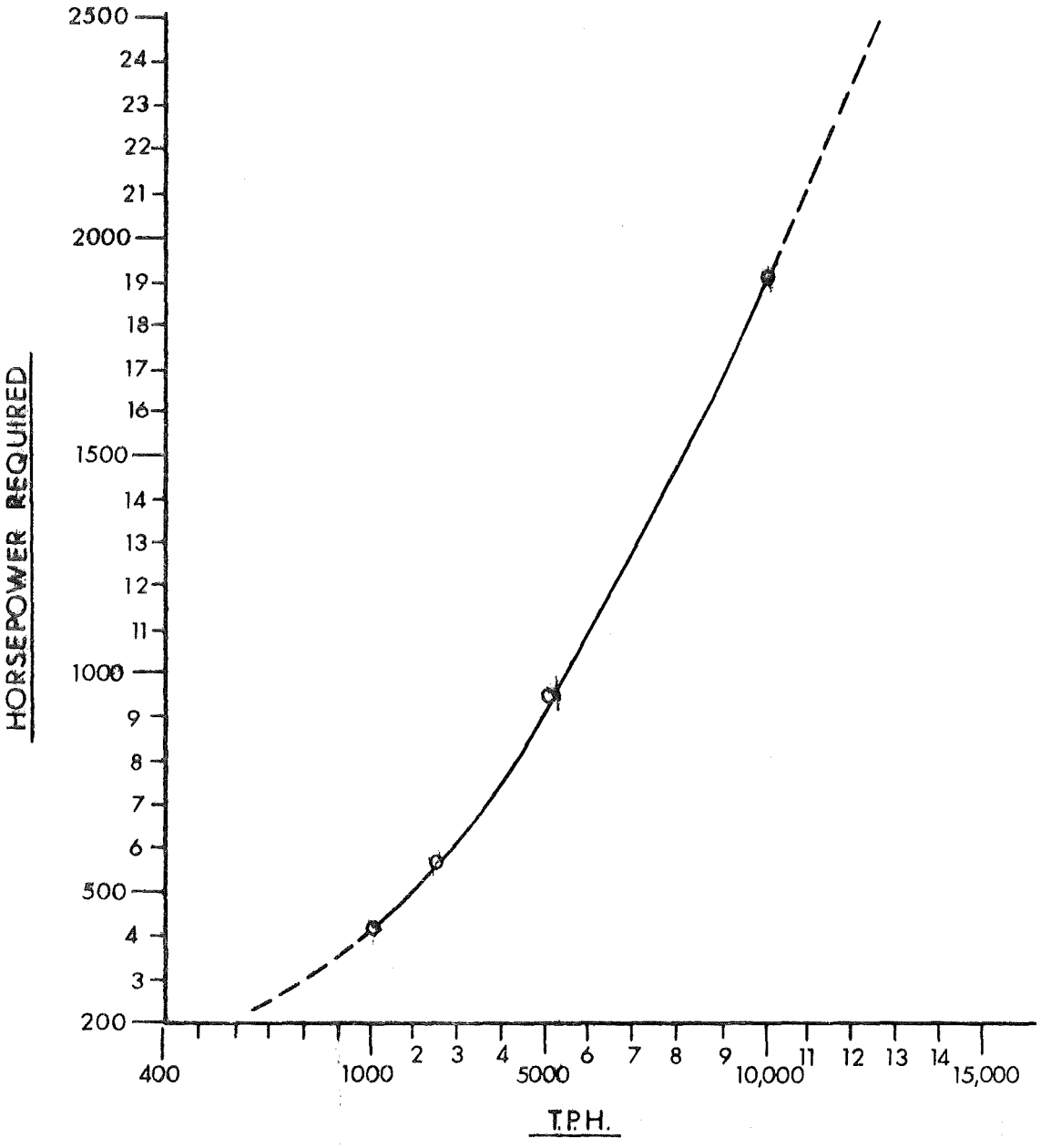
From the 2500 tph point on the horizontal axis of the graph draw a vertical line upwards until it intersects with the surface facilities curve. From that point of intersection, draw a horizontal line to the left until it intersects with the vertical axis of the graph and read approximately $890/tph. Then $890/tph x 2500 tph = $ 2,225,000, which represents the approximate capital cost of the 2500 tph crushing capacity facility.



COST DATA GRAPH
GYRATORY CRUSHER INSTALLATIONS

APPENDIX II

HORSEPOWER VS TPH GRAPH



SURFACE & UNDERGROUND
CRUSHING FACILITIES

(H.P. VERSES T.P.H.)

APPENDIX III

FIGURES A THROUGH L

CRUSHING FACILITIES

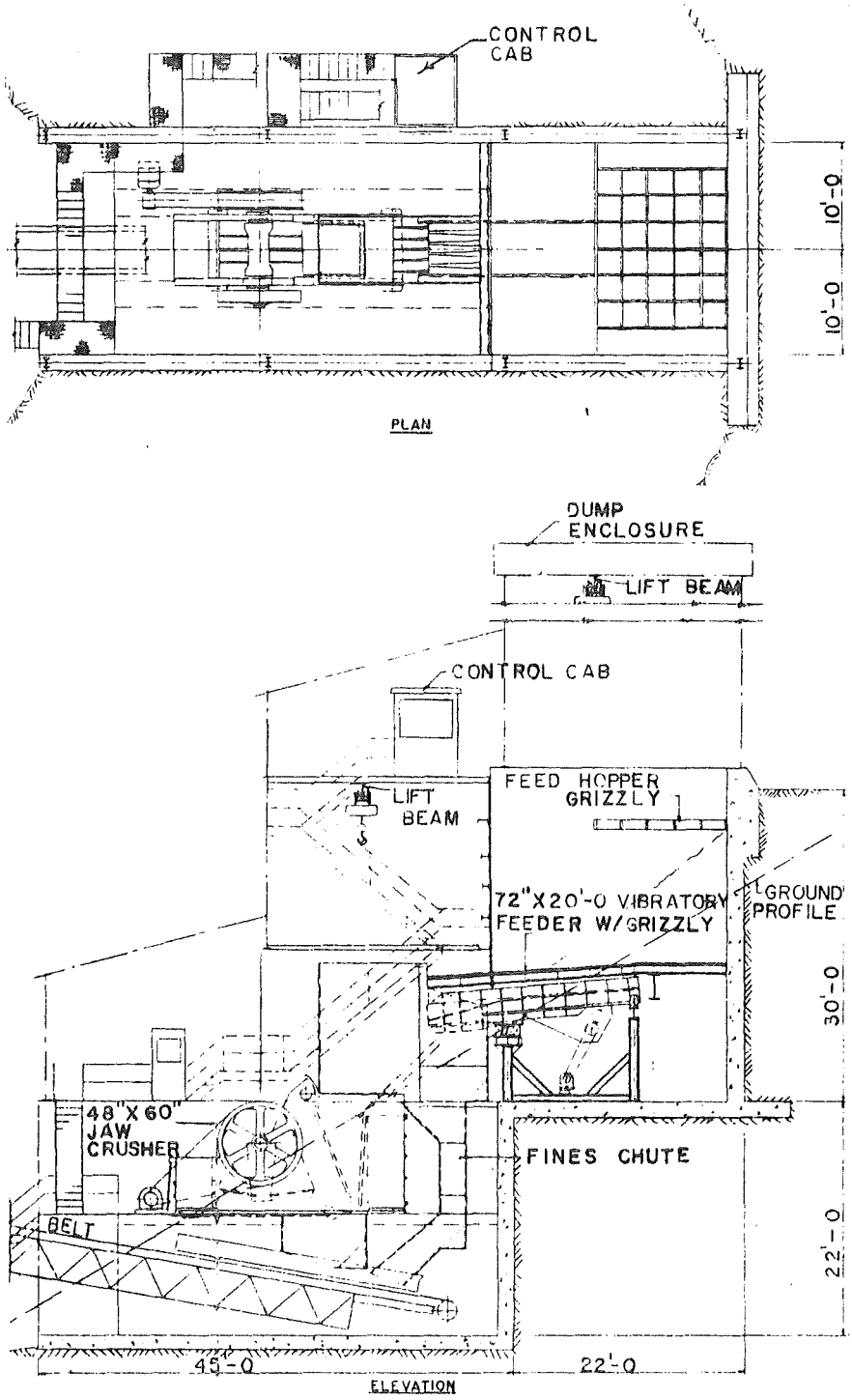


Figure A

1000 T.P.H.
SURFACE FACILITY
48" X 60" JAW CRUSHER
AND 72" WIDE VIB. FEEDER

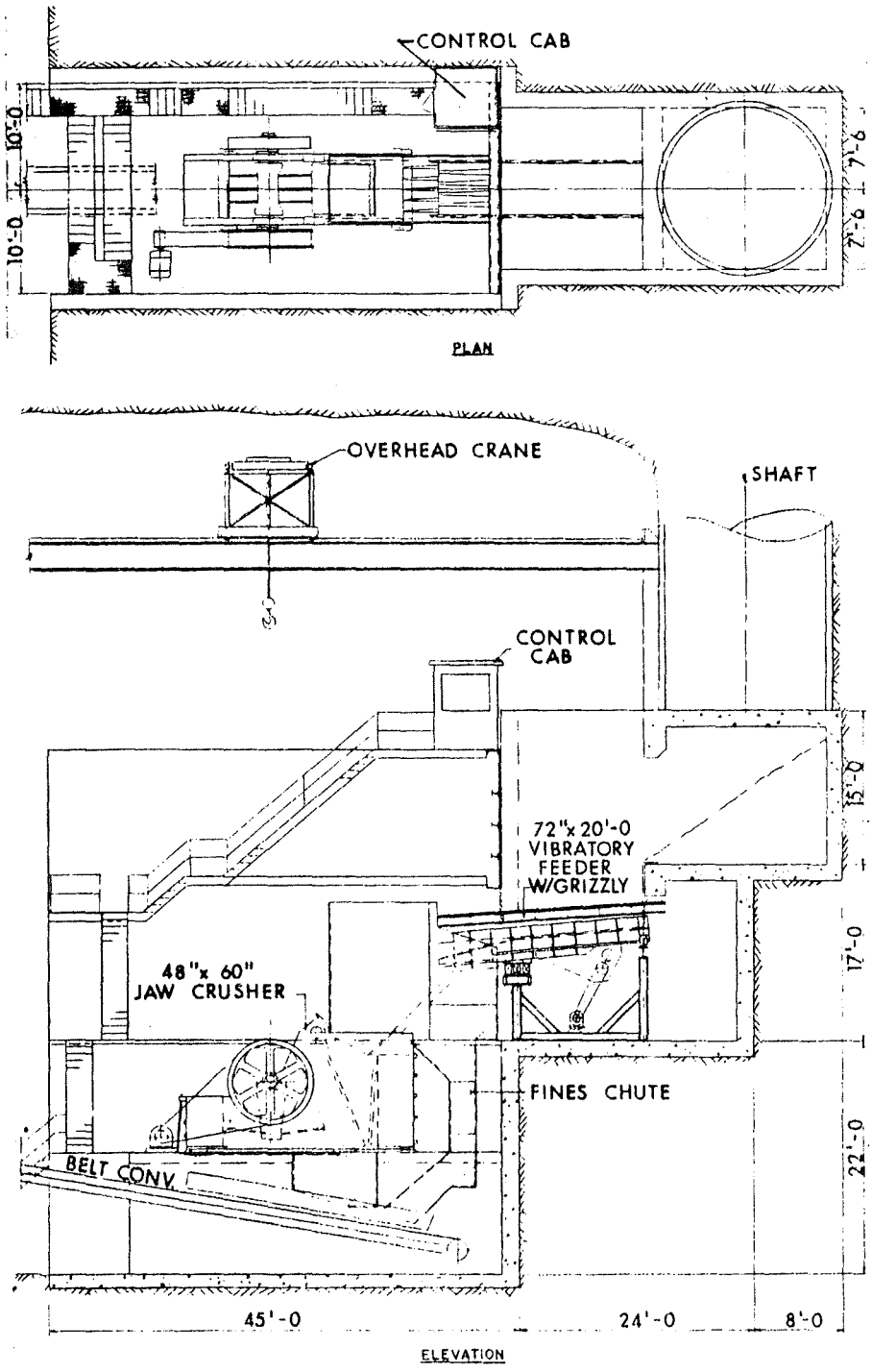
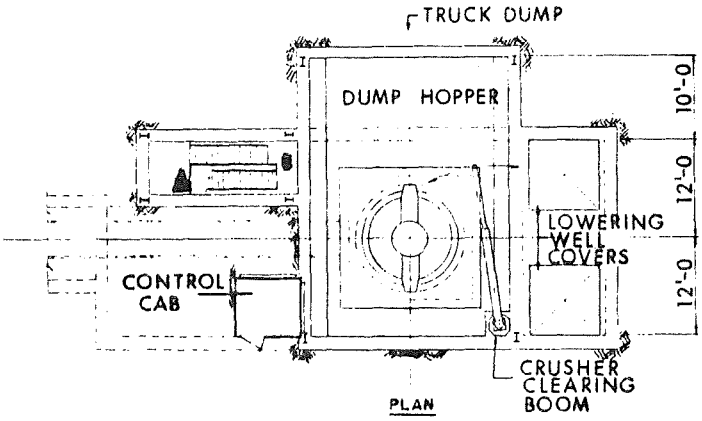


Figure B

1000 T.P.H.
UNDERGROUND FACILITY
48" X 60" JAW CRUSHER
AND 72" WIDE VIB. FEEDER



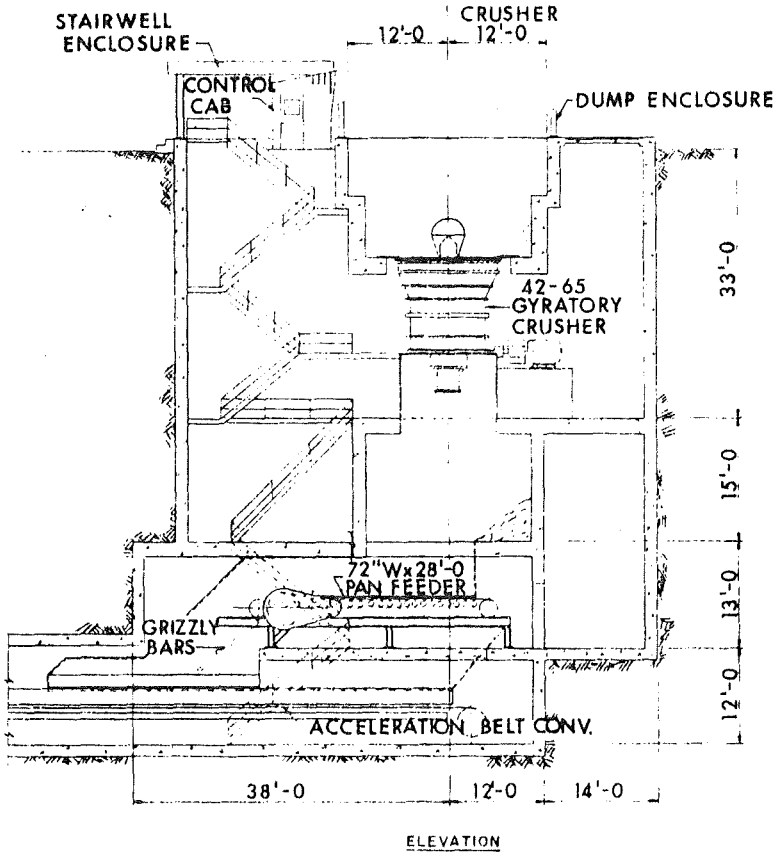


Figure C

1000 T.P.H.
SURFACE FACILITY
42-65 GYRATORY CRUSHER
DUMP TRUCK FEED

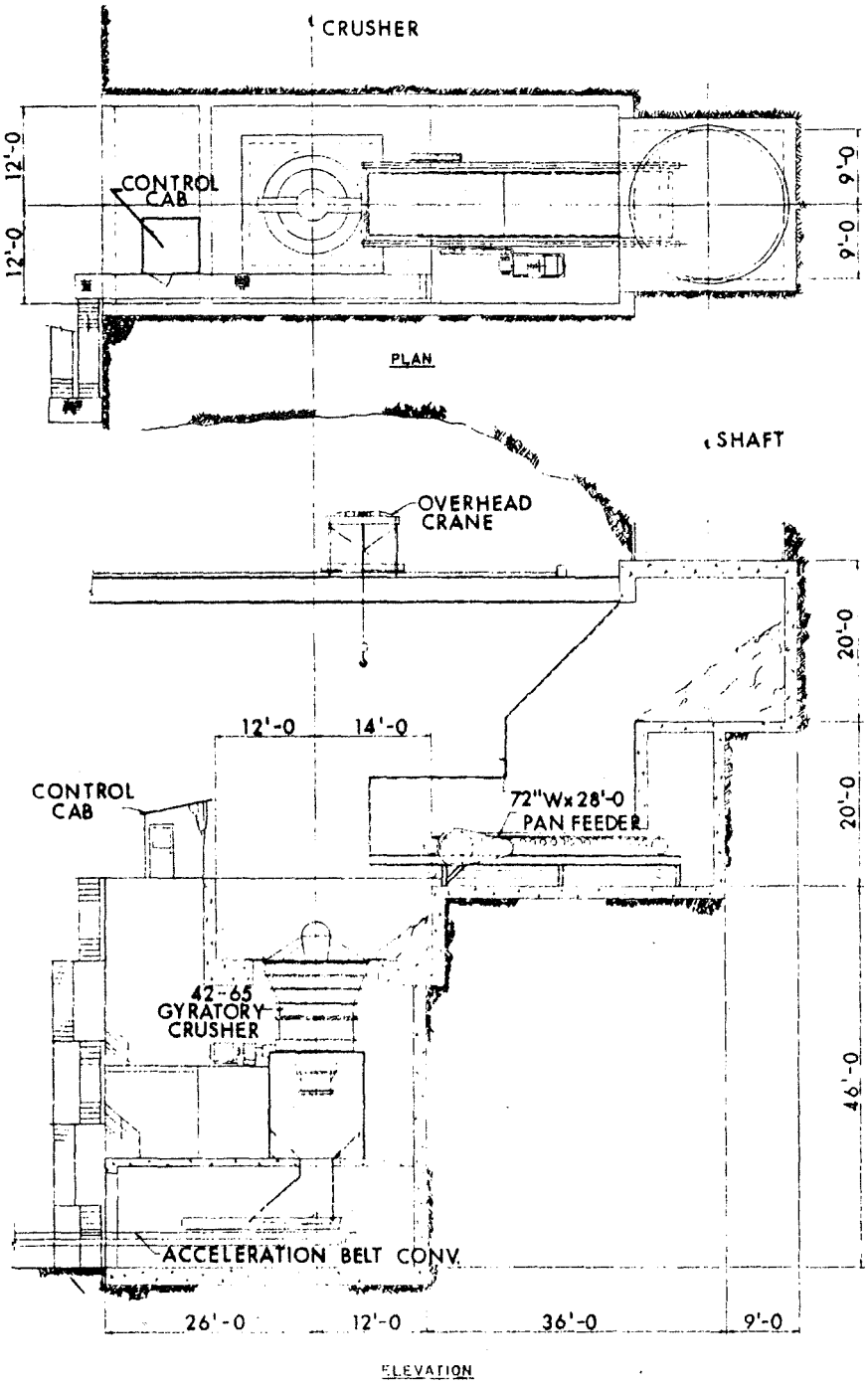
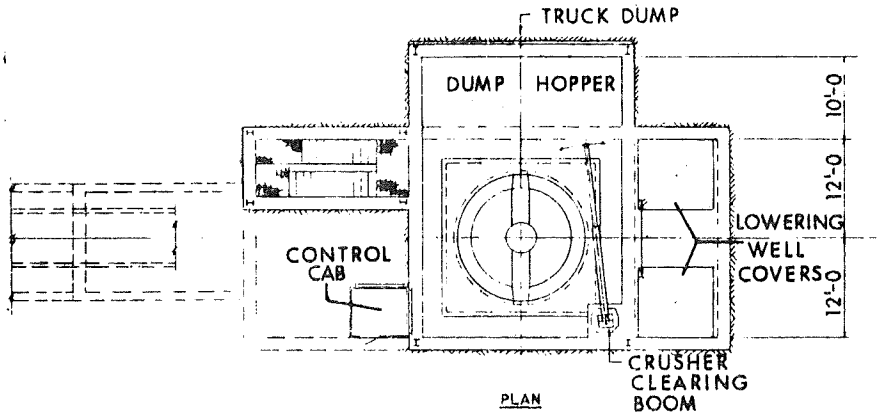


Figure D

1000 T.P.H.
UNDERGROUND FACILITY
42-65 GYRATORY CRUSHER
AND 72" WIDE PAN FEEDER



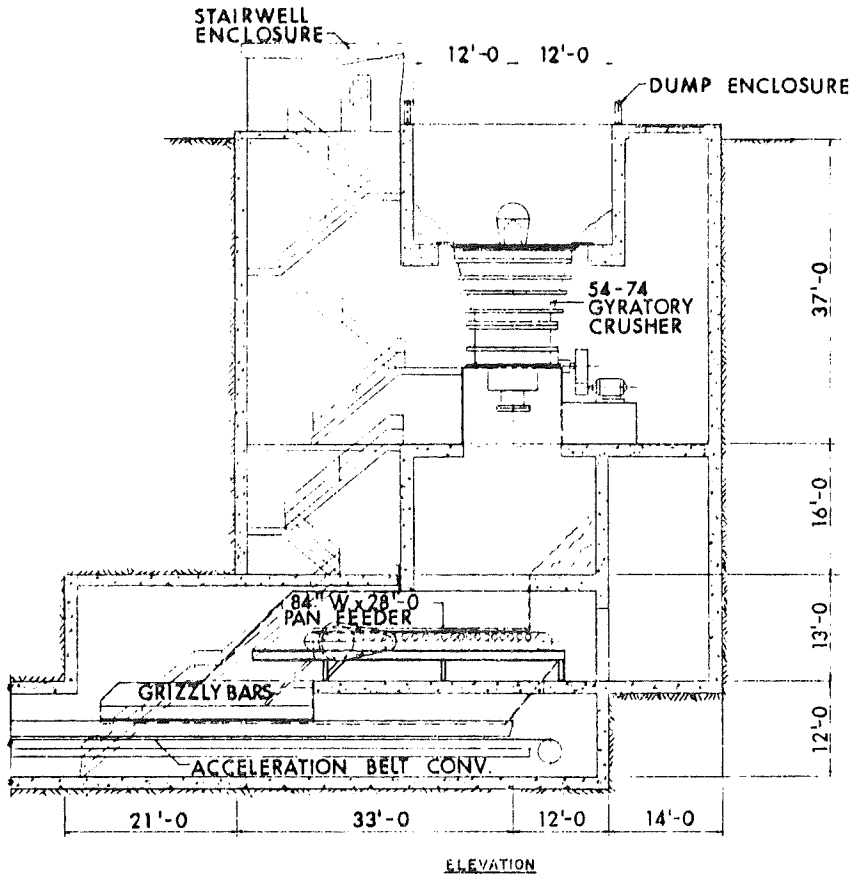


Figure E

2,500 T.P.H.
SURFACE FACILITY
54-74 GYRATORY CRUSHER
DUMP TRUCK FEED

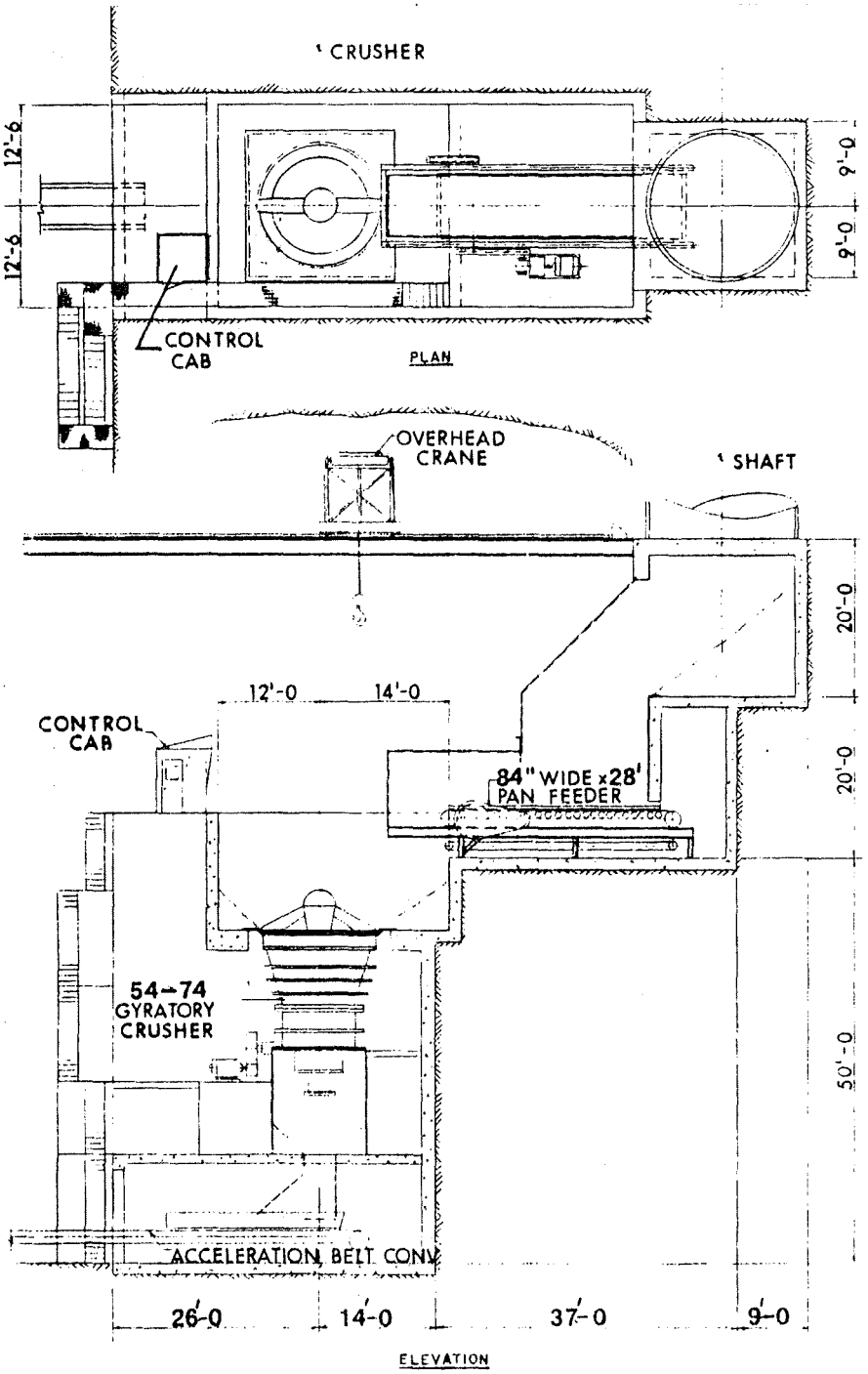


Figure F

2,500 T.P.H.
UNDERGROUND FACILITY
54-74 GYRATORY CRUSHER
AND 84" WIDE PAN FEEDER

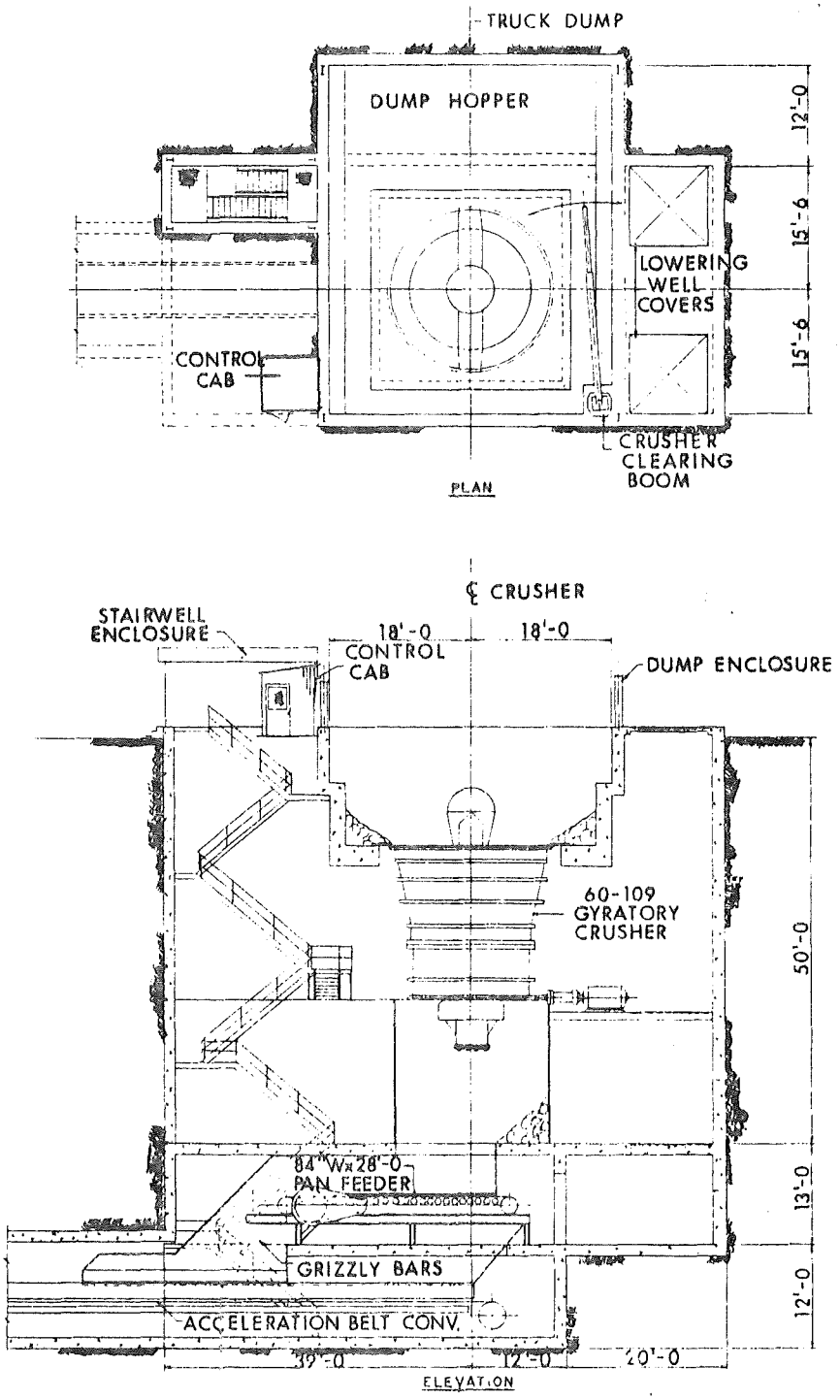


Figure G

5000 T.P.H.
SURFACE FACILITY
60-109 GYRATORY CRUSHER
DUMP TRUCK FEED

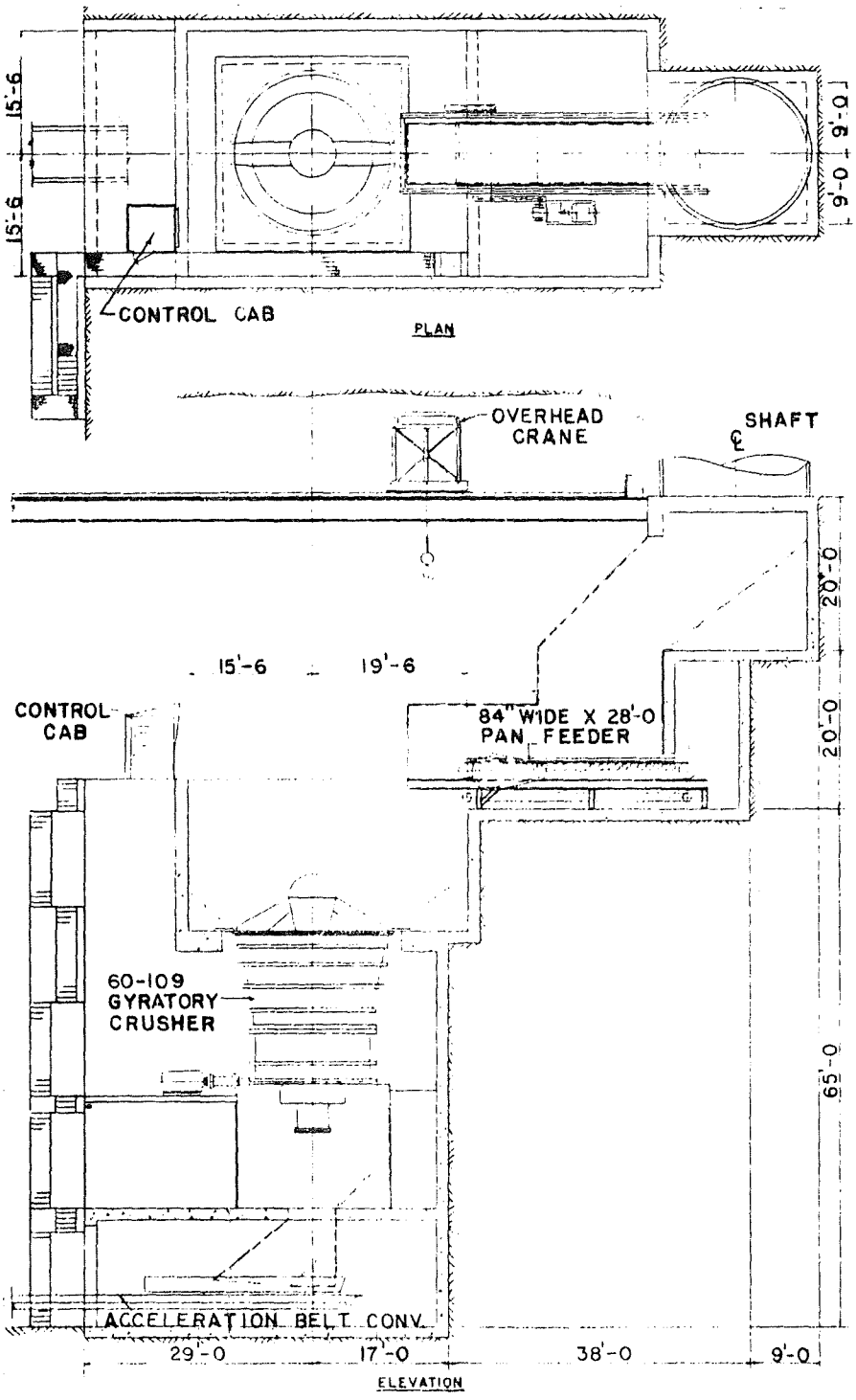


Figure H

5000 T.P.H.
UNDERGROUND FACILITY
60-109 GYRATORY CRUSHER
AND 84" WIDE PAN FEEDER

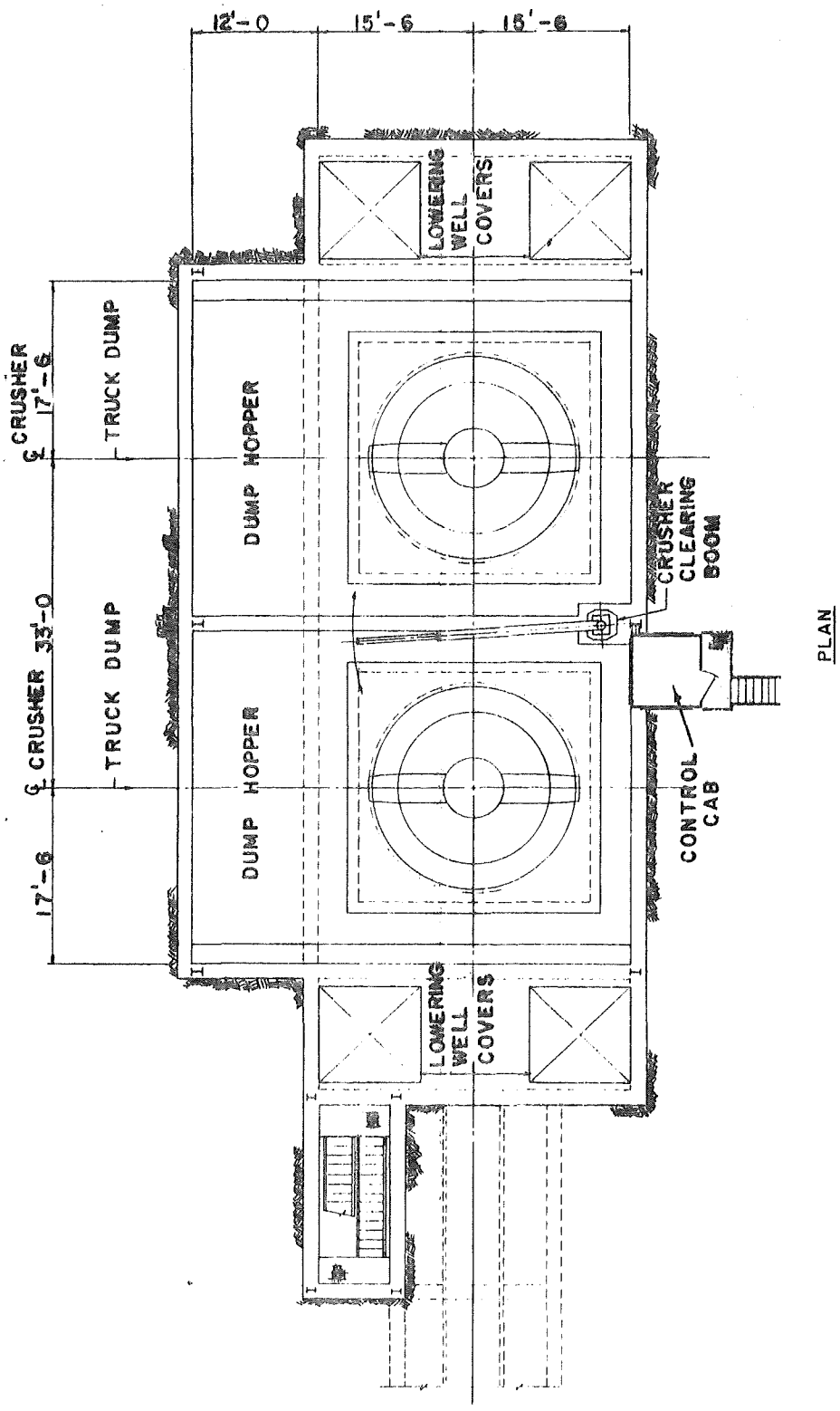
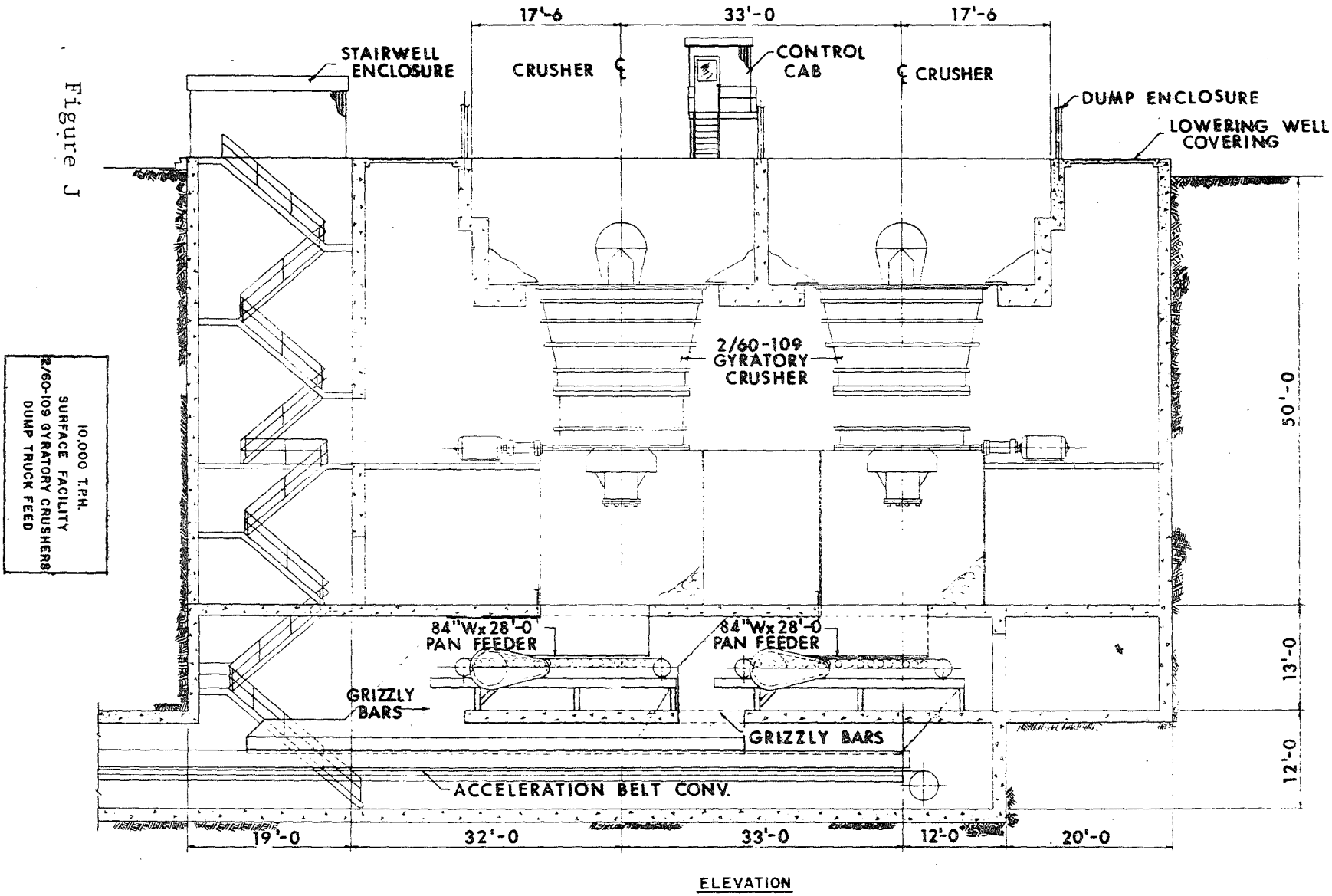


Figure I

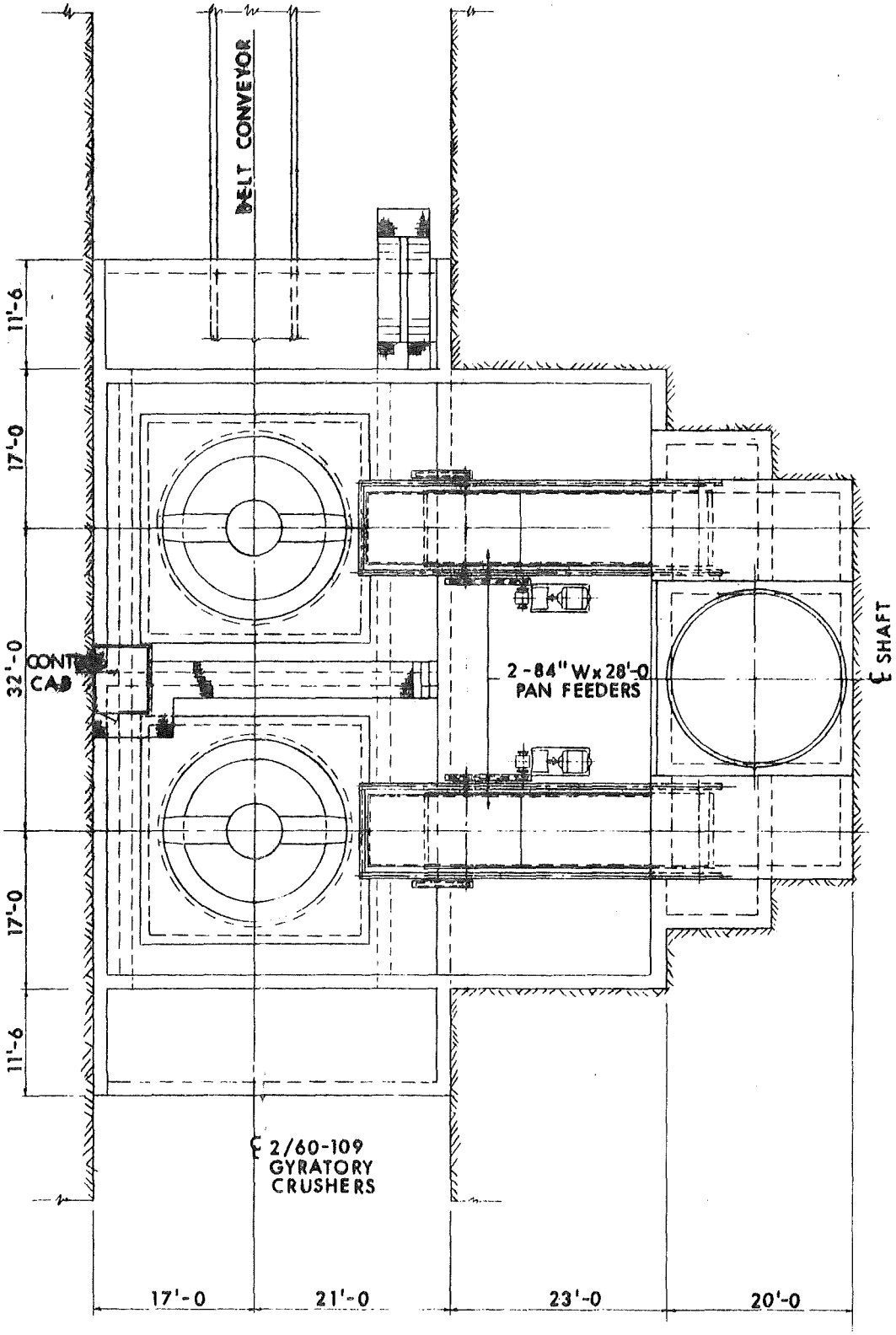
10,000 T.P.H.
SURFACE FACILITY
2/60-109 GYRATORY CRUSHERS
DUMP TRUCK FEED

Figure J



ELEVATION

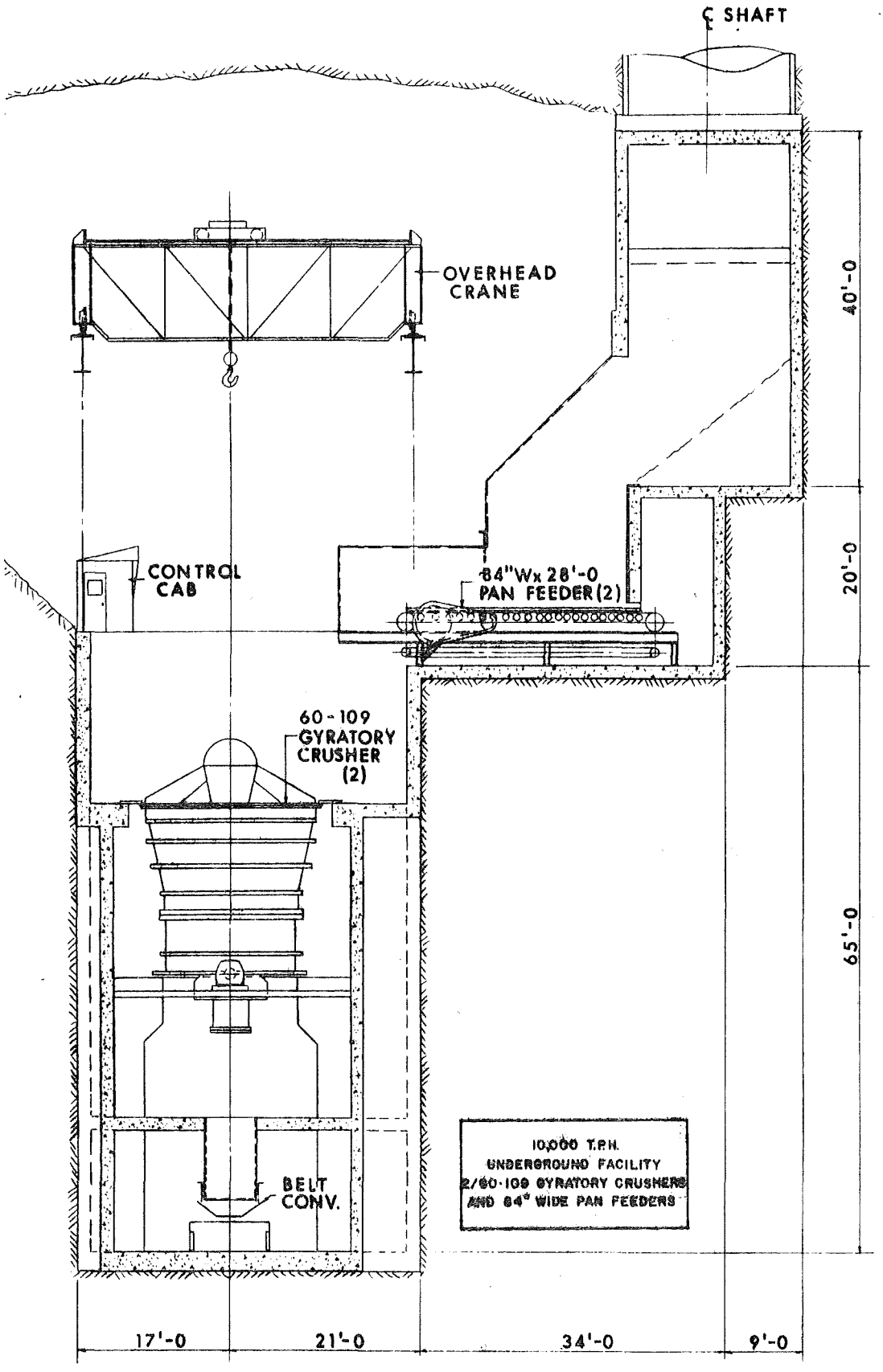
10,000 T.P.H.
SURFACE FACILITY
2/60-109 GYRATORY CRUSHERS
DUMP TRUCK FEED



PLAN

10,000 T.P.H.
UNDERGROUND FACILITY
2/60-109 GYRATORY CRUSHERS
AND 84" WIDE PAN FEEDERS

Figure K



ELEVATION

Figure L

APPENDIX IV

(COST SUMMARIES)

COST SUMMARY

PRIMARY CRUSHING FACILITY

1,000 TON PER HOUR

JAW

	SURFACE	UNDERGROUND
Engineering	$110,000	$110,000
Equipment and Steel Supply	644,900	745,000
Piping Equipment	6,600	11,000
Dust Collection Equipment	5,500	5,500
Site Work	10,500	62,700
Foundation	72,000	96,000
Architectural	9,300	--
Equipment and Steel Erection	114,700	117,100
Piping	3,600	6,000
Wiring	41,000	45,600
Construction Supervision	107,800	107,800
Subtotal	$1,125,900	$1,306,700
Total (+5% Contingency)	$1,182,195	$1,372,035

COST SUMMARY

PRIMARY CRUSHING FACILITY

1,000 TON PER HOUR

GYRATORY

	SURFACE	UNDERGROUND
Engineering	$120,000	$120,000
Equipment and Steel Supply	913,000	1,112,100
Piping Equipment	6,600	11,000
Dust Collection Equipment	5,500	5,500
Site Work	46,200	83,600
Foundation	72,000	96,000
Architectural	6,000	--
Equipment and Steel Erection	130,300	194,800
Piping	3,600	3,600
Wiring	41,000	45,600
Construction Supervision	107,800	107,800
Acceleration Belt	25,000	25,000
Acceleration Belt Tunnel	5,300	5,300
Subtotal	1,482,300	1,810,300
Total (+5% Contingency)	$1,556,400	$1,900,800

COST SUMMARY

PRIMARY CRUSHING FACILITY

2,500 TON PER HOUR

GYRATORY

	SURFACE	UNDERGROUND
Engineering	$120,000	$120,000
Equipment and Steel Supply	1,461,700	1,722,900
Piping Equipment	6,600	11,000
Dust Collection Equipment	6,600	6,600
Site Work	52,800	94,600
Foundation	90,000	168,000
Architectural	6,000	--
Equipment and Steel Erection	156,200	214,200
Piping	3,600	3,600
Wiring	51,600	57,600
Construction Supervision	115,500	115,500
Acceleration Belt	37,500	37,500
Acceleration Belt Tunnel	7,700	7,700
Subtotal	2,115,800	2,559,200
Total (+5% Contingency)	$2,221,600	$2,687,200

COST SUMMARY

PRIMARY CRUSHING FACILITY

5,000 TON PER HOUR

GYRATORY

	SURFACE	UNDERGROUND
Engineering	$120,000	$120,000
Equipment and Steel Supply	3,049,200	3,339,600
Piping Equipment	6,600	6,600
Dust Collection Equipment	11,000	11,000
Site Work	69,300	137,500
Foundation	336,000	480,000
Architectural	7,500	--
Equipment and Steel Erection	214,700	273,500
Piping	3,600	7,200
Wiring	74,400	84,000
Construction Supervision	132,000	132,000
Acceleration Belt	50,000	50,000
Acceleration Belt Tunnel	9,900	9,900
Subtotal	4,084,200	4,651,300
Total (+5% Contingency)	$4,288,400	$4,883,900

COST SUMMARY

PRIMARY CRUSHING FACILITY

10,000 TON PER HOUR

GYRATORY

	SURFACE	UNDERGROUND
Engineering	$125,000	$125,000
Equipment and Steel Supply	6,023,600	6,396,500
Piping Equipment	8,800	13,200
Dust Collection Equipment	16,500	16,500
Site Work	121,000	254,100
Foundation	672,000	912,000
Architectural	9,500	--
Equipment and Steel Erection	405,100	463,200
Piping	7,200	12,000
Wiring	120,000	132,000
Construction Supervision	220,000	220,000
Acceleration Belt	75,000	75,000
Acceleration Belt Tunnel	15,400	15,400
Subtotal	7,819,100	8,634,900
Total (+5% Contingency)	$8,210,000	$9,066,600

WILLIS & PAUL CORPORATION

Conveyor Study

WILLIS AND PAUL CORPORATION

In-Pit Conveyor Study

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INTRODUCTION

The purpose of this section is to present a current, comprehensive picture regarding the conditions, requirements and relative costs involved in the utilization of conveyors operating in open-pit mining operations.

Certain basic assumptions have been made in order to define and simplify the data and information presented herein. A more detailed description regarding the basis for these assumptions is presented further on in this section.

It should be noted that no attempt has been made to compare conveyor haulage to any other type of haulage system, whether it be truck, rail, etc.

Use of the data specified herein is subject to modification, as dictated by specific pit operations. Adjustments to the installed cost values due to any modification of parameters, as they are stated, such as belt width, conveyor support type, cut or tunneling operations, etc., are presented either in Nomograph form, as multipliers, or add-ons, to be applied to the final installed cost figure, as derived from the Nomograph.

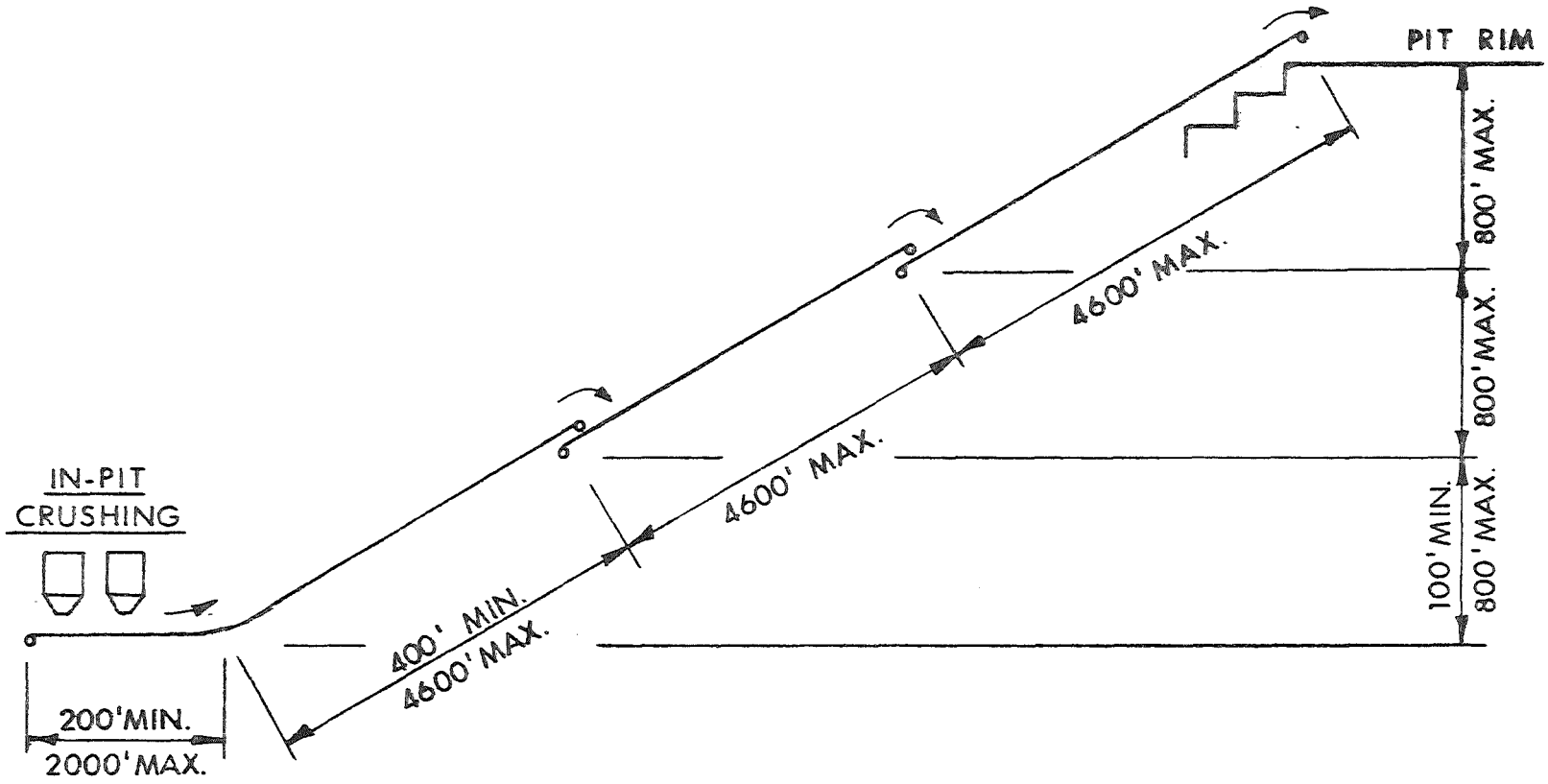
These adjustments are best applied by the mine planning and estimating personnel. These persons are most familiar with the site specific problems and requirements, and are in a position to judge with some degree of accuracy those factors which most closely meet the needs and requirements for a given mining operation.

STUDY PARAMETERS

Since this is a general description, not specific to a given open-pit mining operation, site specific data such as location, nature of the mining operation, depth of overburden, pit slope angles or specific material characteristics other than bulk density, material surcharge angle, and angle of repose, have not been considered in this study, due to the vastly complex number of site specific possibilities.

This section assumes the following set of conditions, applicable to either new or existing facilities. The conveyor or conveyors are located in an open-pit mining operation with a conveyor profile as shown on Fig. M, horizontal to incline. It is possible that it may undulate with varying numbers of concave and convex curves, as required to fit the site specific requirements. The horizontal portion may vary from 200 ft. to 2000 ft., depending upon the size and number of in-pit primary crushers being utilized to feed the conveyor, the geometry of the most recent or proposed level of the mining operation, and the type of mining method being utilized or proposed. The horizontal section may be located as required, depending on the method of feed to the crushers and the type of crusher, jaw or gyratory, being utilized.

The inclined section may vary from 400 ft. to 4600 ft. with overall lifts varying from 100 ft. to 800 ft. for any single flight. For lifts greater than 800 ft., a transfer point is required to an entirely new conveyor, for the case of the 60-



IDEAL CONVEYOR SYSTEM PROFILE

AT MAXIMUM PIT DEPTH CONDITION

FIGURE M

inch belt only.

The additional conveyor/conveyors will have a maximum lift of again 800 ft. The number of conveyors and transfer points are dependent upon the amount of lift required to convey the material from the level of the mining operation to the rim of the pit. The pit depth may vary from 500 ft. to 2400 ft. depending upon projected or existing mine planning operations.

The conveyor horsepower will vary from 100 hp to 12,000 hp, per conveyor, depending upon tph, belt speed, required lift and belt width. The conveyor profile may require open cut work in hard rock in order to maintain a maximum incline angle of 18°. Conveyor capacities will range from 1000 through 10,000 tph utilizing material with a bulk density of 100 lbs./ft.^{3}.

The conveyor belt will be a 60-inch-wide steel cable belt, except where tonnage rates range from 1000 to 3000 tph. For this range of values a 36-inch-wide belt would be required. The belt will have a maximum tension rating of 4400 pounds per inch of width, and will utilize an 80% load profile on 35° troughing idlers with an estimated surcharge angle for the load profile of 20° and an angle of repose for the material of from 30° - 35°.

The belt speed range will be from a nominal 300 ft. per minute to a nominal of 1100 ft. per minute for various conditions of lift and tons per hour. This is a result of utilizing an 80% load profile for both the 60 and 36-inch-wide belts. The results of a series of calculations, which are discussed in depth further on in this section, indicate that a 60-inch-wide belt encompasses

the majority of the variable load conditions over the broadest range of values, whereas the 36-inch belt provides an economical solution in the lower capacity range.

The method and type of feed to the conveyor is from a primary crusher with a nominal discharge of a maximum of 12" lump size. Note: The 36-inch-wide belt limits the maximum lump size to a nominal 8". The conveyor may be fed by an accelerating belt or belts, depending upon the number of stages of acceleration required.

Once the material has been elevated to the rim of the pit, the main belt may discharge to various of the following:

1. Additional conveyors providing feed to either a material processing facility or stockpiling operation.
2. Trucks
3. Rail cars

The method of haulage, once the conveyor has reached the rim of the pit is dependent upon such site specific and economic factors as:

1. Distance of the processing or concentrating facility from the pit.
2. Geometry and character of the waste material to be moved in order to make the deposit accessible.
3. Availability and location of waste disposal areas.
4. Topography
5. Overall cost of the operation

DESIGN PARAMETERS AND ASSUMPTIONS

In the development of the Study Parameters, certain assumptions regarding the design and application of belt conveyors to

open-pit mining operations have been made. Additionally, the limitations inherent in conveying equipment have been carefully considered and provide a basis for numerous of the aforementioned Study Parameters.

In the selection and design of belt conveyors, certain specific information regarding the nature of the material to be handled should be known in order to size and select the proper unit.

In order to allow for interpolation of capacities for the greatest range of material bulk densities, 100 lb./ft.^{3} material was selected rather than a specific material bulk density. The material lump size range, with values from 6-inch to 12-inch nominal, covers a representative cross section of primary crusher product output sizes most often encountered.

Conveyor tonnage rates from 1000 through 10,000 tph are compatible with economical mine planning methods presently being used. In addition, the majority of the open-pit mines in the United States have primary crusher output rates which fall within these limits.

The maximum allowable lift value of 800 ft. was arrived at due to the following conditions: a maximum allowable unit belt tension of 4400 pounds per inch of belt width, a maximum conveyor belt speed of 1200 feet per minute, and a belt loading of 80% load profile, the exception being in the case of the 36 inch wide belt. For this case specifically, where capacities are from within 1000 to 3000 tph, the lift is limited to a maximum of 1500 feet per flight of conveyor, due to the decrease in material

weight per foot of conveyor, resulting directly in lower belt tensions and, consequently, a greater lift capability. When considering the calculations of belt tensions, the limit for the 60-inch-belt becomes 264,000 lbs. total. Which is the maximum running tension; 264,000 lbs. x 1.40 = 369,000 lbs., may be sustained by the belt for short periods of time either during startup or during intermittent service.

For our purposes, we are concerned primarily with the maximum tension value. For the conveyor profiles as indicated in Fig. M, the controlling tension value is produced by the lift tension, which is the pounds of tension required to lift a given load a given vertical height. Other tension factors such as the tension to move the load horizontally, empty belt tensions, frictional resistance to the movement of the load, etc., all combine to create the effective tension (T_e). However, since for the case in question, as defined in this study specifically, we are dealing with larger than average lift requirements, the lift tension is, therefore, the controlling factor and far outweights all other contributing tensions. We have considered only the lift tensions, for reasons of brevity in calculating the maximum belt tensions incurred.

By utilization of the CEMA formula (Appendix VI) for lift tension and by applying the parameters as previously stated regarding belt speeds and tons per hour for an 80% load profile, it may be shown that the maximum lift for all conditions of capacity and belt speeds is limited to a nominal 800 ft. for any single flight of 60-inch-wide belt.

The 800 ft. maximum lift value is due specifically to the 80% load profile factor being utilized, since this is the determining belt speed factor. The same holds true in the case of the 36-inch-wide belt, except the lift is limited to a maximum of 1500 ft. The greater cross-sectional area afforded by utilization of 45° troughing idlers only offers a 6% advantage in terms of belt speed increases, but does not allow necessarily for accommodation of the assumed surcharge angle of the material.

In the case of faster speed values for the same tons per hour capacity, the material weight per foot of conveying would be less, since the equation for the material weight per foot (Wm) of conveyor has velocity (belt speed) in feet per minute in the denominator. Essentially, this means that the belt tension is less, however, the horsepower demand remains the same since the same tons per hours capacity is required on a rate basis for the same lift requirement. The only difference is that there is less material per foot of conveyor involved, which has its advantage in lower belt tensions, and allowing for narrower belt conveyors more economical drive equipment.

There is, however, a limit to the maximum belt speed with which present day conveyors and equipment are capable of withstanding. At speeds approaching 1200 ft./min., a lack of reliable test data exists with regard to the effects of high belt speeds on conveyor components. Therefore, we have limited belt speeds from a nominal 300 ft. per minute to a maximum of 1100 ft. per minute in order to be in accord with the presently available equipment designs.

In this discussion of belt tensions and lift capabilities, only single or tandem head drives have been considered. On the larger capacity units, dual pulley tandem drives would be required in order to accommodate the extremely large tension and horsepower requirements.

There is an alternative approach to the problems of excessive belt tensions, transfer point problems, etc., namely, the tire type intermediate drive.

Generally speaking, this type of drive eliminates the problem of excessive belt tensions and provides, within the limits of practicality, a theoretically infinite lift, thereby eliminating the need for transfer points and reducing maintenance and equipment replacement costs.

The tire type intermediate drive, a relatively recent addition in this country, has its limitations. Due to its present state-of-the-art design, there are various features of this drive which do not lend themselves to the handling of large lump sizes. In addition, backstop requirements, in the instance of inclined conveyors, need to be investigated since it has not been necessary to incorporate this feature until now. A modification of the existing design could provide solutions to these problems, as they apply to the handling of large lump sizes and steep incline angles.

Present drive module sizes incorporate a 200 hp drive. Research and development studies are being conducted at this time with regard to the development of 1000 and 2000 hp modules to accommodate the requirements of larger capacity and steeply inclined conveyors.

The horsepower range of from 100 through 12,000 hp per conveyor is a result of utilizing the conventional type of head drive and the basic laws of mechanics. Since horsepower is defined in terms of ft.-lbs./per minute, then clearly, the rate at which work is done in tons per hour is independent of the belt loading or pounds of material per foot of conveyor. Specifically, this means that 5000 tph delivered by a conveyor which requires 1000 ft. of lift at a belt speed of 539 ft./min. at an 80% load profile requires the same amount of horsepower as a conveyor delivering 5000 tph requiring a 1000 ft. lift at a belt speed of 1078 ft. per minute, at a load capacity of approximately 40%. In either case, 5000 tons of material must be lifted 1000 ft. within one hour's time. The means by which it accomplishes this task has no effect on the horsepower requirements, regardless of the type of drive utilized.

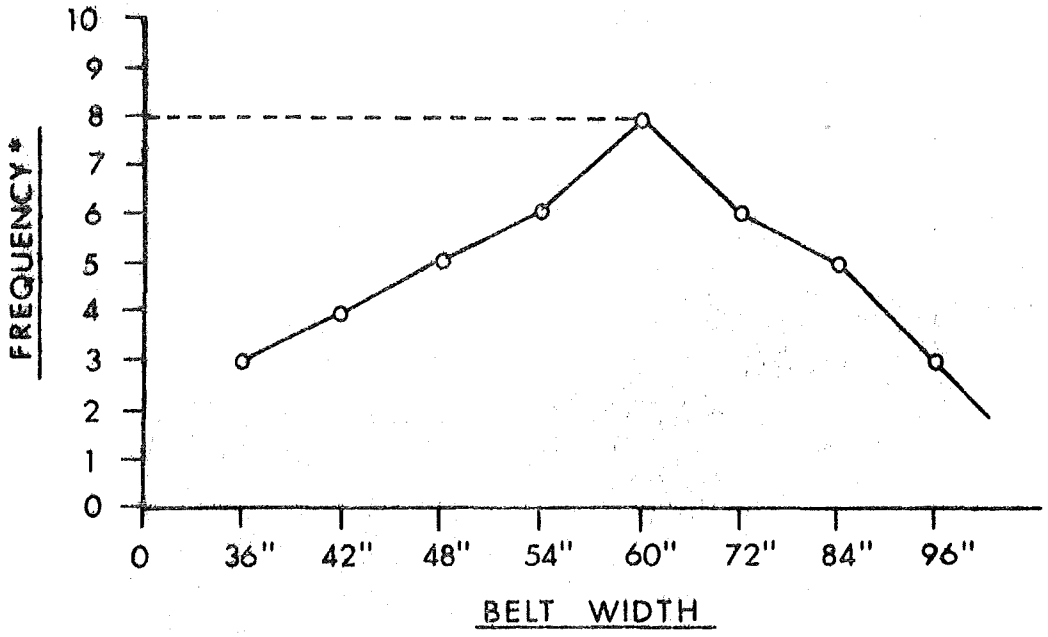
In order to allow the conveyed material to exit the pit with a relative degree of stability, a maximum incline angle of 18° was chosen. This situation would allow for the shortest overall length of the conveyor for the greatest lift requirement.

The determination of the specific type of haulage methods lies ultimately with the mine planning engineer, his estimating personnel, qualified material handling engineers and the nature of the material to be mined.

The selection of a 60-inch-wide conveyor belt as the optimum belt width for the higher range of capacities, as previously defined, is based upon the following:

1) a capacity interval of 3000 through 10,000 tph for an 80% load profile of the belt, 2) utilization of 35° idlers, 3) surcharge angles and bulk densities, as stated previously. It was found that for this case specifically, the frequency of occurences within the interval of capacities as stated, fell most often within the range of the 60 inch wide belt, except within the interval of from 1000 to 3000 tph. For the lower tonnage, the 36 inch wide belt prevails as the most practical width within the range of the parameters (see Figure N, Frequency vs. Belt Width). This frequency distribution curve is based upon the results of the information and data as shown on Figure O, Belt Speed vs. Tons Per Hour, for various widths of belt from 36-inch through 96-inch-wide.

It may be seen from these curves that the 60-inch-belt is capable of handling 3000 through 10,000 tph of material at belt speeds ranging from a nominal 300 ft. per minute to a nominal of 1100 ft. per minute. It is indicated within these curves that other belt widths may be utilized to handle 3000 tph of material, however, the possible number of combinations of belt widths, belt speeds, and tons per hour become too numerous to consider, within the scope of this study. Therefore, in order to present a representative installed cost figure for an in-put conveying system, two conveyors were chosen which encompass most of the variable conditions of capacity, maximum lump size, etc. The 60-inch-wide belt appears to be the most practical width within the 3000 through 10,000 tph range and the same being true for the 36-inch-wide belt within the 1000 to 3000 ton range.

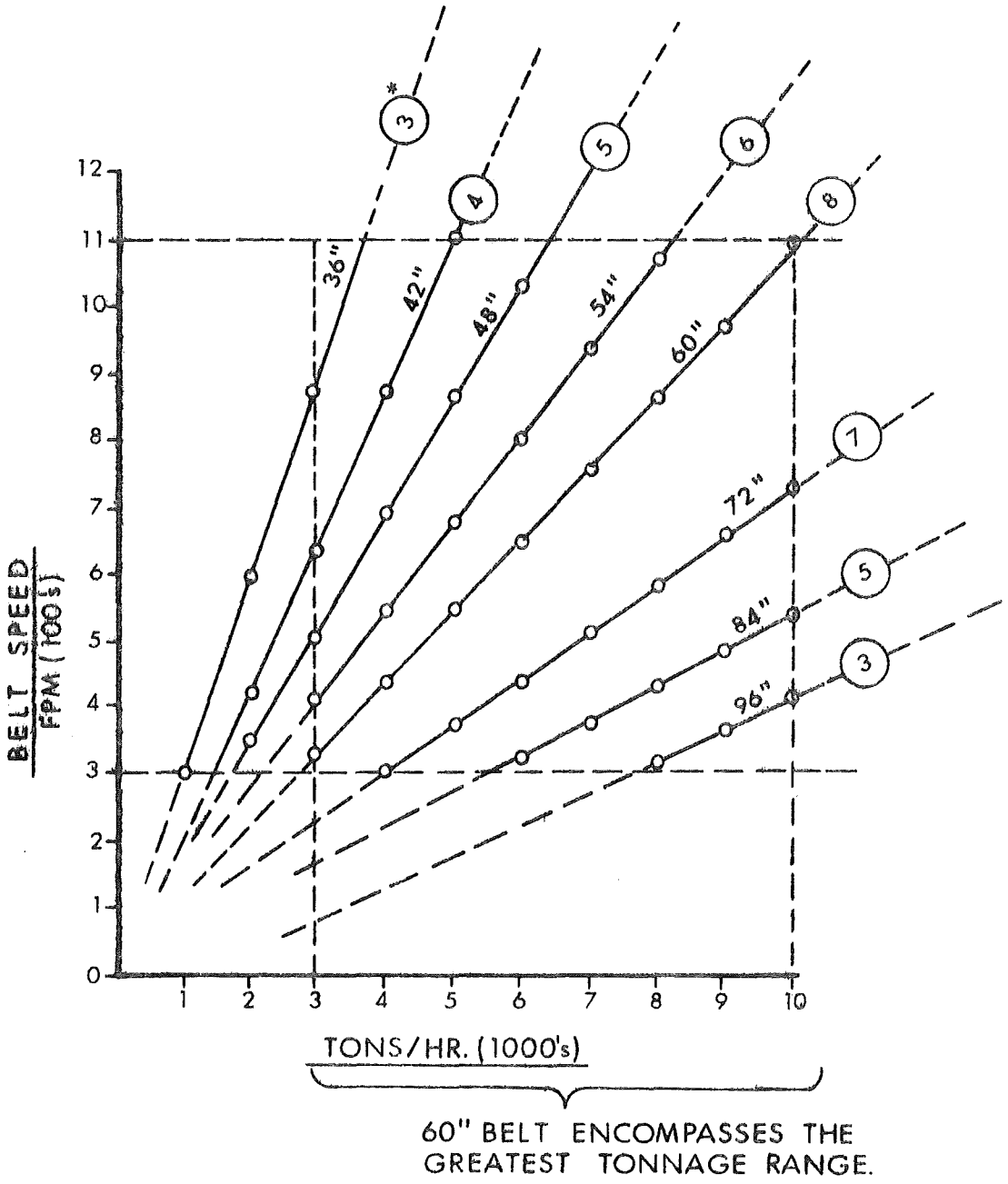


* Number of TPH rates in 1000's achievable by various belt width systems in the belt speed range 300-1100 FPM and 1000-10000 TPH capacity range.

60" BELT ENCOMPASSES THE GREATEST TONNAGE RANGE (see Belt Speed vs. Tons/Hr. Graph)

FREQUENCY VS. BELT WIDTH GRAPH

FIGURE N



*(3) Frequency of occurance in the speed range of 300-1100 FPM and capacity range 1000-10,000 TPH.

BELT SPEED VS. TONS/HR. GRAPH

FIGURE 0

One of the most critical areas of concern regarding belt conveyor application and design is the manner in which a conveyor is loaded. Improper loading of the belt may cause a number of problems, a few of which are difficulty in the tracking of the belt due to unbalanced forces at the load point and excessive wear of the belt due to the effects of high impact loading of the material on the belt. For the specific case, as defined in this study, the method of loading the main conveyor belt from the discharge of the primary crusher is effected by the use of either feeders or accelerating belts. Feeders provide a relative degree of protection for the main belt by reducing the effects of impact forces at the load point. They also allow for a consistent and properly distributed cross-section of material flow, thereby enabling the belt to operate more efficiently with fewer shutdowns and less maintenance and repair time. The accelerating belt provides essentially similar protection for the main conveyor belt. In terms of application, the basic difference is dependent upon main conveyor belt speeds and primary crusher product discharge rates. Apron feeders are utilized more often in the case of large capacity and lower belt speed conveyors. The use of the accelerating belt permits the material to approach the main belt speed, in proximity of the loading section, thereby minimizing the resistance of the material flow in the direction of belt travel. This provides for proper material flow to the main belt, enabling it to stabilize the load prior to its encountering the inclined portion of the conveyor. In either case, the application of feeders and/

or accelerating conveyor belts to loading and transfer point requirements is dependent upon site specific requirements, customer preferences, capacities and main belt speeds.

As close an attempt as is possible has been made here to provide a comprehensive and state-of-the-art basis for the Study Parameters, as previously defined.

Since the purpose of this study is to provide an installed cost figure for open-pit mining operations utilizing in-pit conveyors, it must be understood that the Design Parameters and Assumptions, although they represent the present state-of-the-art, remain to a degree, flexible, and are subject to revisions and changes, as required by the future demands and requirements of the mining industry.

Therefore, all installed cost figures as indicated on the Installed Cost Nomograph (Figure Q and Figure R) reflect only the present day industry standards and prices and are subject to modification, per the above.

Since each mining operation has many site-specific requirements, modifications to the installed cost figures, as presented in this section, are a likely possibility.

Following is a discussion of these variables and how they may be applied to the Nomograph values in order to obtain a more realistic cost figure for a number of these site specific and preferential conditions. In addition, a basic summary of costs is provided in order to illustrate and compare the various disciplinary areas of a project of this nature.

However, before a description and an analysis of the Installed Cost Nomograph is discussed, a mention of the Break Horespower requirements for the 60-inch-wide belt under various conditions of lift and capacity should be presented, in order to correlate the difference between Motor Horsepower, as indicated on the Installed Cost Nomograph, and the Brake Horsepower, as noted on this series of curves.

The Brake Horsepower, as indicated on Figure P, refers to the actual horsepower required to accommodate the various conditions of lift, belt speeds and capacities, as indicated.

The curves represent only one single flight of conveyor, with a maximum lift of 800 ft. (Note: The horsepower value as indicated represents the requirement for a maximum of 800 ft. lift only.) Horsepower requirements are cumulative per each 800 ft. of lift. Therefore, if it required 8000 bhp to deliver 10,000 tph against an 800 ft. lift, it would require 16,000 bhp to deliver the same capacity against a 1600 ft. lift. This would require two conveyors and a transfer point, due to the maximum allowable lift per conveyor. Subsequently, to deliver 10,000 tph against a 2400 ft. lift would require 24,000 bhp utilizing three conveyors and two transfer points.

Due to equipment manufacturer service factor recommendations, for the severe duty as encountered in open-pit mining operations, there is a noticeable difference between the Brake Horsepower requirements as indicated on Figure P and the Motor Horsepower as indicated on the Installed Cost Nomograph, Figure Q. The difference amounts to a nominal 1.5 service factor applied to the Brake Horsepower requirement.

A similar service factor has been applied to the thermal horsepower and torque ratings of the speed reducers utilized in the development of the cost figures for the Nomograph. The 1.5 service factor applied to the drive equipment represents a considerable increase in equipment costs over equipment of similar design being utilized under less severe operating conditions.

In the case of the 36-inch-wide belt, a similar series of curves applies, the difference being primarily in the maximum lift dimension attainable for capacities from 1000 to 3000 tph. Since the purpose of presenting these curves is to correlate the difference between Motor Horsepower and Brake Horsepower requirements, it is not necessary to illustrate the results of the 36-inch-wide belt curves.

INSTALLED COSTS

The Installed Cost Nomograph (Figure Q and Figure P) represents the installed cost for a 60-inch-wide and 36-inch-wide in-pit conveyor haulage system as defined previously. For capacities from 1000 through 10,000 tph and lift requirements from 100 ft. through 2400 ft., the following information may be obtained:

- Motor horsepower required.
- Conveyor length, exclusive of any horizontal portion, up to 2400 ft. of lift for 60-inch belts. Conveyor length, exclusive of any horizontal portion up to 1500 ft. of lift for 36-inch belts.
- Conveyor system total installed costs.

The system installed cost figure was arrived at by consideration of the following conditions: The utilization of a channel frame type conveyor structure, incorporating support points at 20 ft. intervals. Concrete piers are assumed in order to accommodate the support legs of the conveyor. At each transfer point, as required, a transfer tower structure, including foundations to allow for the support of the conveyors, their drive equipment, discharge and loading sections, is included.

All structural erection and mechanical and electrical equipment installation costs have been considered, and are given in 1978 costs.

The following is a breakdown of the various items which compromise the total installed costs for the 60 inch wide belt conveyor only.

		Cost/Ft.
1.	Mechanical equipment cost	$326.00
2.	Conveyor steel cost (includes transfer tower steel)	$ 20.00
3.	Foundations cost	$ 1.00
4.	Equipment installation	$ 84.00
5.	Foundation & steel erection	$ 56.00
6.	Electrical equipment costs	$ 95.00
7.	Electrical installation	$ 50.00
	TOTAL	$632.00

Inclusive of the previously stated conditions, an installed cost figure of $632.00 per foot of conveyor was concluded as being a reasonable average value to use, for all conditions of capacity, lift and conveyor length, up to the maximum values as indicated on the Nomograph (Figure Q). This conclusion is drawn

from the following information and data regarding erection and installation costs and their relationship to capital equipment and material costs.

For the specific case as defined by the Study Parameters, the cost per foot of conveyor for the mechanical equipment, generally, remains a constant unit cost value. This holds true for conveyor lengths from 600 ft. through 10,000 ft. However, drive component costs vary with horsepower demands, which are directly related to capacity and lift requirements and indirectly to overall conveyor lengths.

The direct relationship of horsepower requirement for a required lift and a given capacity in tons per hour, indicates that for shorter lifts or lesser capacities, the required horsepower and, consequently, the drive component size and cost is reduced in direct relation to the change. A greater lift requires more horsepower, therefore, a larger drive; consequently, a more costly drive is required. However, greater lift means a longer conveyor and since these variables are directly related, the cost per foot for the larger drive remains approximately the same due to the increased length of the conveyor.

All other major conveyor equipment items such as belting, idlers, pulleys, bearings, etc., are unit priced. They are directly related to the overall length of the conveyor when considered on a cost per foot basis, and, therefore, remain a relatively proportional value. Likewise, the conveyor structural steel and foundation costs are directly proportional to the overall length of the conveyor.

Electrical equipment and material costs are related in much the same manner to the horsepower requirement, as is the mechanical equipment. Power distribution systems, motor control centers, control equipment and conduit are all proportional to power demands and conveyor lengths, and are, therefore, able to be defined on a cost per foot basis. The figure of $632.00/ft., as it applies to the 60-inch-wide belt only, may be utilized within the limits as set forth in this section with a reasonable degree of accuracy.

In the case of the 36-inch-wide belt, as indicated on the Installed Cost Nomograph (Figure R), there exists a similar situation regarding the relative proportions and costs for the installed system equipment, material and labor. Essentially, the difference lies in the component equipment costs. The cost of the 36-inch-wide belt is approximately 40% less than the cost of the 60-inch-wide belt. The idlers and pulleys are approximately 30% less than the cost for the 60-inch-wide components. This provides us with an average reduction in cost of approximately 35% for the major conveyor components, exclusive of the drive equipment. The drive equipment requirements are as stated previously, proportional to the capacity, lift and the resulting horsepower demand, and represent an average of approximately 40% of the total mechanical equipment cost per foot of conveyor length. This may be considered applicable to the case of either the 60-inch-wide or 36-inch-wide conveyors.

The remaining major items of equipment such as the belt, idlers and pulleys comprise, in both cases, approximately 60%

of the total mechanical equipment cost per foot.

All other items such as foundation costs, equipment installation costs, electrical equipment costs, and material costs, remain approximately the same, as defined for the 60-inch-wide belt, for the reasons as stated in the previous discussion.

Therefore, an installed cost figure for the mechanical equipment of $230.00/ft. of conveyor may be utilized for the 36-inch-wide belt conveyor. A total system installed cost figure of $570.00 per foot of conveyor length may be utilized as a reasonable average value for the 36 inch wide conveyor belt, for the conditions as indicated on Figure R only. This represents approximately 90% of the installed cost of the 60 inch wide belt conveyor system, within the range of capacities, as shown.

Variations to the parameters, as previously defined, with regard to belt widths other than 60-inch, within the capacity interval of from 3000 through 10,000 tph, such as, conveyor structural designs, the inclusion of certain site specific requirements such as tunnels, etc. are to be dealt with by adjusting the installed cost for a 60-inch-wide belt.
Note: Adjustments to the 36-inch-wide belt are not required since no other belt width can economically handle the lower rnage of capacities as stated, within the limits of these Study Parameters.

The change in total cost when a belt width other than 60-inch-wide is required is as follows:

Utilizing the longest 60-inch-wide belt, per the parameters, the belt, idlers and pulleys, exclusive of the drive component

costs, represents 31% of the total installed unit cost of $632.00 per foot; therefore, for each standard incremental belt width from 42-inch to 96-inch, the following factors may be applied to the total installed cost.

BELT WIDTH	INSTALLED COST MULTIPLIER
42"	0.923
48"	0.950
54"	0.975
60"	1.000
72"	1.146
84"	1.293
96"	1.440

These values are approximate and reflect principally the difference in equipment costs for the various widths of conveyor. Any change in the material of construction costs, due to either larger or smaller belt widths, e.g., more or less structural conveyor steel, smaller or larger foundation requirements, in order to accommodate the situation, is nominal as compared to the cost of erection and/or installation costs.

In the case of conveyor frame design, if deck type truss is required, then multiply the total installed cost for the appropriate conveyor above, by a factor of 1.23.

For wire rope or cable suspended design the total installed cost for the appropriate conveyor above should be multiplied by a factor of 0.80.

There are, however, certain limitations which exist regarding the use of these variables. Where belt widths other than 60

inch wide are concerned, a review of Figure O will indicate the range of capacities in which these belt widths fall within the parameters of this study. Again, these capacities reflect an 80% load profile of the belt.

Where a deck type truss would be considered, then a careful review of the site specific topography and method of mining must be considered. This type of frame design may limit benches, access areas, etc. and could interfere with the mining operation unless careful consideration is given to its utilization.

Wire rope or cable suspended conveyors are, by virtue of their design, not as sturdy as structural steel frame supports and are, therefore, more susceptible to damage due to natural causes, mining and haulage equipment. Expansion problems exist relating to the stretching of the wire rope, due in some instances to temperature differentials, and in some instances to the resulting weight of material on the belt. Therefore, careful thought should be given regarding these conditions when considering the utilization of a conveyor of this design. Also, availability of wire rope suspended idlers and other equipment should be investigated, especially in the belt width sizes above 60 inch.

When tunneling operations, due to site specific requirements, are called for, add $500.00 per foot of elevation (vertical lift) to the installed cost of the conveyor, as indicated on the Nomograph. The tunnel assumes a 10° or 18° maximum incline angle and would be approximately 12 feet in width by 8 feet in height. The cost per foot for this add-on is based on the assumption

that hard rock mining only will be employed, utilizing hard rock mining methods to include roof bolting. For the employment of any other methods, an additional add-on cost would be required, depending on the specific operation.

Although provisions have been made to allow for the selection of belt widths other than 60-inch-wide, a thorough review of the requirements and anticipated production rates for the projected life of the mine and the application of practical economics to the selection of an in-pit conveyor should be undertaken as soon as project schedules permit.

STATE-OF-THE-ART REVIEW CONVEYORS

Specific areas such as belt engineering and technology are lacking information, test data, etc., with respect to the effects of belt speeds in excess of 1200 ft. per minute on the presently available conveyor components.

Some testing has been done by equipment suppliers and belt manufacturers on belt speeds of up to 2000 ft. per minute. At these speeds, belt to belt transfer becomes exceedingly difficult to effect without the generation of large amounts of particulates, due in part to the natural attrition of the material being handled, the nature of the operation and specific material characteristics.

Research and development in the area of high speed belts, regarding speeds in excess of 1200 ft. per minute, is necessary in order to permit larger tonnage rates at higher speeds and reduced tensions. Presently, we are limited by the belting industry to a maximum unit belt tension of 4400 pounds per inch

of width for steel cable belting presently available on today's market.

This study indicates that for the most practical conveyors selected, at 60-inch-wide and 36-inch-wide, the lift was limited in all cases to a maximum of 800 ft. and 1500 ft., respectively. This is due primarily to belt tensions approaching the limit of the 440 p.i.w. steel cable belt's maximum allowable unit tension value.

Whether research and development projects address themselves to the problem of belt tensions or belt speeds, it should be kept in mind that there is an economical and practical limit to the capacities involved in open-pit mining operations, ore concentrating facilities and material processing plants.

We already have at our disposal, a steel cable belt capable of withstanding extremely high belt tensions. It would appear that research programs directed toward the investigation of the effects of high speed belts on component equipment, its effects on the material being conveyed, load stability, etc., could prove to be very informative and provide us with solutions to the problems associated with belt tensions.

Any program directed toward the development of steel cable belts with unit tension ratings in excess of 4400 p.i.w. and the effects of high tension values, with regard to the state-of-the-art designs surrounding present day conveyor component equipment, would have to be investigated thoroughly so that an upgrading of present equipment designs could be effected in order to accommodate these higher tensions.

As was stated previously in the Design Parameters, there is an alternative approach to the problems specific to excessive belt tensions and high speeds. Within certain limits of capacity and application, the intermediate drive could possibly provide solutions to these problems. However, since it has never been employed in open-pit mining operations, and due to its present state-of-the-art design, a certain amount of redesign would be necessary in order for it to meet the severe duty requirements expected of most mining equipment. It does, however, present us with a means by which the problems associated with excessive belt tensions might possibly be eliminated.

Mining industry future trends regarding equipment capacities, processing facilities, mining techniques and methods, etc., will produce problems for present state-of-the-art equipment design. If belt designs are to remain within the limits of practicality regarding belt speeds, capacities, tensions, horsepower requirements, etc., then further studies are necessary in order to evaluate the effects of high speed belts, and larger capacities and tensions on equipment and equipment support designs. The direction which research and development programs should take now, in order to be prepared to meet these future needs, is directly related to these studies.

SUMMARY

The Installed Cost Nomograph is intended for use by management and project personnel involved in the preparation of capital equipment costs, mine planning and development budgets.

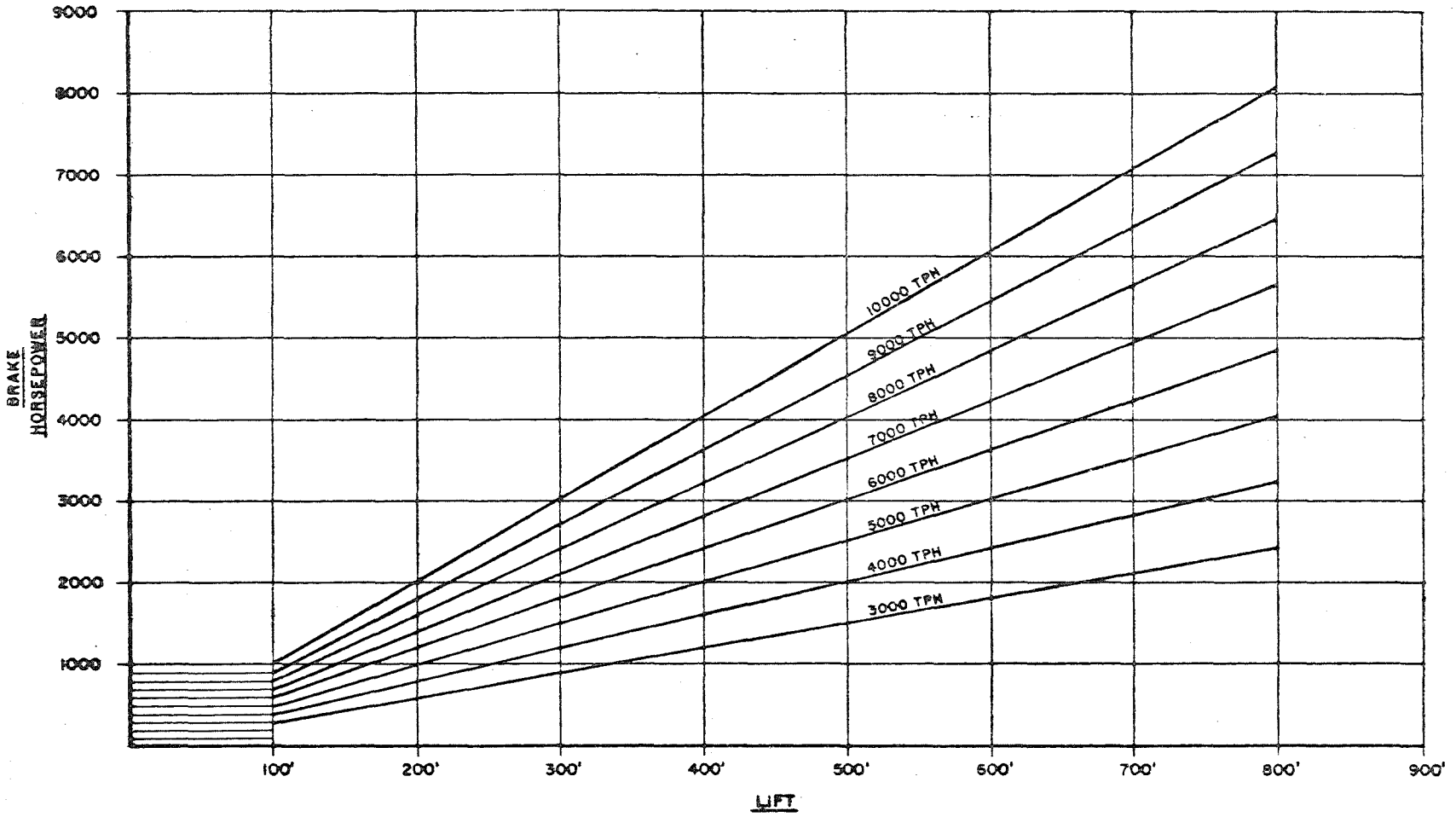
Since all possible conditions with regard to the mine site could not be taken into account, an ideal situation was selected with sufficient flexibility to allow for a number of variations.

The total installed cost is a budget figure in 1978 costs, representing the conditions as stated and the combinations of capacity and lift as provided for on the Nomograph.

In conclusion, it appears that equipment costs, both mechanical and electrical, represent on the average, approximately 70% of the total conveyor installed cost. Erection and installation costs are equal to approximately 30% of the total. No consideration to any special site preparation requirements, such as tunneling, has been given in these relative cost figures and percentages. They have been included as add-on costs per site specific requirements.

Utilizing foresight in the preliminary stages of mine planning and development, the mine planning engineer should be able to incorporate this type of system within the scope of the plan to meet present and future requirements.

In the case of existing facilities, where a system of this type would be considered, a revised mining plan, in order to accommodate the requirements of space, angle of incline, etc. could be implemented with little impact on production rates and schedules.



BRAKE HORSEPOWER vs. LIFT

FIGURE P

APPENDIX V

COST DATA

BELT CONVEYOR COST DATA

1. Mechanical Equipment Cost

Belting 5000 ft. x 64.83 $/ft	=	324,150	
*4500 ft. x 61.59 $/ft	=	277,150	

* 5% discount on purchases in excess of 5000ft.

Idlers	300 x 220 Trough	=	66,000
	950 x 115 Trough	=	109,250
	300 x 115 Return	=	34,500
	200 x 60 Return	=	12,000
	10 x 550 Impact	=	5,500
	49 x 486 Trainers	=	23,815
Pulleys		=	32,000
Motor		=	146,720
Reducer		=	234,752
Brg's, Cplg's & Misc.		=	58,688
			1,324,525
$1,324,525 x 12% Contingency		=	1,500,000

$\frac{1,500,000\ \$}{4600\ ft}$ = 326 $/ft — 326 $/ft

2. Conveyor Steel Cost (including Transfer Tower Steel)

Steel Cost $\frac{97,000\ \$}{9600\ ft}$ = 10 $/ft

Transfer Tower Steel Cost

2 Tower, Steel	22,000
Found	16,320
Erection	10,080
	48,400 each tower

$\frac{48,400\ (2)}{9600}$ = 9.87 $/ft = 10 $/ft

20 $/ft — 20 $/ft

3. Foundation Cost — 1 $/ft

4. Equipment Installation

5 man hour/ft @ $16.80/ man hour

4600 ft x 5 mh/ft x $16.80/mh = 368,000 $

$\frac{36,800}{4600}$ = 84 $/ft 84 $/ft

5. Foundation and Steel Erection

Conveyor 20 #/ft x 4600 ft = 920,000 # = 460 tons

@ 20 man hours / ton

20 (460) = 9200 man hours

9200 mh x $21.75/mh = 200,100

9200 mh x $\frac{\text{crane day}}{80 \text{ mh}}$ 500 $/cd = 57,500

257,600

$\frac{\$257,600}{4600}$ = 56 56 $/ft

6. Electrical Equipment Costs

3 Conveyors, (4) 3000 hp Motors each

3 x 4 = 12 Motors @ 3000 hp each

12,000 hp required

Electrical Cost to produce 12,000 hp

with 12 motors 95 $/ft

(Price does not include motors)

7. Electric Installation

Labor 50 $/ft

Total 632 $/ft

APPENDIX VI

GLOSSARY OF TERMS AND FORMULAS

CEMA - Conveyor Equipment Manufacturers Association

CEMA Formulas -

1. Horsepower = $\frac{Te\ V}{33000}$ Where

2. $Te = L\left[Kt(Kx + Ky\ Wb + .015\ Wb)\right] + KyLWm \pm HWm + \text{Accessories}$

Te = Total effective tension, lbs., required to move conveyor belt and load and to lift load

V = Velocity of the belt (belt speed), fpm

L = Length of the conveyor along belt line, or center, ft.

H = Height, ft., that the material is lifted

$.015Wb$ = Return belt friction, lbs. per ft. of conveyor length

Kt = Ambient temperature factor

Kx = Force, in lbs. per ft. of conveyor length, to rotate the idler rolls, carrying and return, and to cover the sliding resistance of the belt on the idler rolls

Ky = Force, lbs. per ft. of length to flex the conveyor belt and load over the idlers

Wb = Weight of the belt, in lbs. per ft. of length

Wm = Weight of the material, lbs. per ft. of belt length

$$Wm = \frac{33.3Q}{V}$$

Q = tph (tons per hour) capacity. Short tons of 2000 lbs.

V = Belt speed, fpm

WILLIS AND PAUL FORMULA

1A. $\text{Horsepower} = \frac{T\ell\ V}{33000}$

2A. $T\ell = Wm \times H$

$$\text{Where } Wm = \frac{33.3\ Q}{V}$$

$T\ell$ = Total tension, lbs.,
Q = Tons per hour (tph), Short tons of 2000 lbs.
H = Height, Ft., that the material is lifted
V = Velocity of belt (belt speed), fpm
Wm = Weight of the material, lbs. per ft. of belt length

Note: This method of calculating total required horsepower neglects all friction and flexure within a belt line. However, where applied to belt systems of high capacity and positive lifts this abbreviated formula and a 1.5 service factor produces a result which is 125% of the corresponding CEMA result without service factor.

Belts designed by CEMA standards used under adverse conditions may have a 1.15 service factor applied. If a 1.15 service factor is applied to the CEMA results, the above abbreviated formula produces with a 1.5 service factor results 109% of the CEMA formula with service factor.

Thus the abbreviated Willis and Paul formula is conservative and appreciably shorter to apply in evaluating in-pit haulage belt systems.

The Angle of Repose of a material is the angle to the horizontal which the surface of a normal, freely formed pile makes to the horizontal.

The Angle of Surcharge of a material is the angle to the horizontal which the surface of the material assumes while the material is at rest on a moving conveyor belt. This angle usually is 5° to 15° less than the angle of repose, though in some materials it may be as much as 20° less.

A Back Stop is a mechanical device that allows the conveyor to operate only in the desired direction. It allows free rotation of the drive pulley in the forward direction but automatically prevents rotation of the drive pulley in the opposite direction.

The Load Profile is the shape of the material on a moving conveyor as seen in a section view perpendicular to the direction of belt travel.

Break Horsepower is the measurement of the power necessary to elevate the conveyed material plus the power required to overcome the belt friction and flexure and idler friction.

Thermal Horsepower is the power lost in the belt drive torque converter and or gear speed reducer to heat energy and not transferred to the belt conveyor from the drive motor.

Motor Horsepower is the rated horsepower of the drive motor and must equal the brake horsepower plus the thermal horsepower. It includes all energy necessary to elevate the material, overcome idler and pulley friction, belt flexure and thermal losses in torque conversion and gear speed reduction.

Service Factor is a factor by which the specified horsepower (break horsepower) is multiplied to compensate for drive conditions.

Deck Type Truss is a framed belt structure built entirely from tension and compression members arranged in panels so as to be stable under load. Carrying idlers and return idlers are separated by flat metal plates to prevent belt spillage from damaging the return belting by being carried back to the tail or drive pulleys. Such trusses are often used to span rivers and roadways with distances up to 120 ft. between supporting piers.

Wire Rope or Cable Suspension System is a belt structure system utilizing two anchored and tensioned steel cables on which carrying and return idlers are hung. This system utilizes regularly spaced steel frames to maintain the proper distance between cables and to mount return idlers.

APPENDIX D

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INMINCO	ENGINEERING COMPUTATIONS

COMPANY U. S. Bureau of Mines
LOCATION "Continuous Haulage from Open Pits"
DESCRIPTION Costs Trucks and Shovels

Trucks (Assumptions)

Wabco	16V149TI	1600HP	170 Ton
spot and dump time		2.25 min	
load and spot time		3.8 min	
11-12 mph up grade			
30 mph down grade			

Time

50 min-hours, 7hr/shift

Truck: Time for haulage

4.05 miles/12 mph = 0.338 hr.

4.05 miles/30 mph = 0.135 hr

Subtotal = 0.473 hr

load = 3.8 min

dump = 2.25 min

Subtotal = 6.05 min or 0.101 hr

Total cycle time = 0.574 hr

Hourly Capacity: (170 tons/cycle) $\div \frac{0.574 \text{ hr}}{\text{cycle}} \times \frac{50 \text{ min}}{60 \text{ min}}$

= 246.8 Tons/hour

Shovel Capacity

1870 Tons/hour ÷ 246.8 Truck tons/hr = 7.6 trucks
8 trucks required

With 75% Availability factor then = use 10 trucks required for one shovel

20 trucks required for 75,000 tons/dry production - 2 shovels

Capital Costs/Truck $650,000 or $13,000,000 for Fleet

Estimated Truck Cap. 247 Ton/hr Inclined Haul

2

INMINCO	ENGINEERING COMPUTATIONS

COMPANY U. S. Bureau of Mines
LOCATION "Continuous Haulage from Open Pits"
DESCRIPTION Costs for Trucks and Shovels

TRUCKS

Operating Costs [2]	Cost/Hour	
Fuel	15.00	
Tires	13.00	
Maintenance (P.M.)	2.50	
Maintenance (Repairs)	30.00	
Operator	9.50	
		$70.00/hr
$70/hr ÷ 248 Ton/hr. =	28.23¢/ton	
$25/hrs Ownership Cost [2] ÷ 248 Ton/hr =	10.08¢/ton	
$95/hrs	38.31¢/ton total	
ROUND OFF TO	38.3¢/ton	

3

INMINCO	ENGINEERING COMPUTATIONS

COMPANY U. S. Bureau of Mines
LOCATION "Continuous Haulage from Open Pits"
DESCRIPTION Incline Belt Conveyor

Parameters

Production Required	-	75,000 tons/day
Depth	-	1,600 Vert. Ft.
	-	5,178 Feet on Slope at 18°
Operational Day	-	21 hours
	-	3 shifts/day, 5 days/week
Belt Size	-	60 inches

Without in line drives systems, 2 Belt Flights will be required.

Summary	Conveyor
Capacity ton/hr	3,571
Horse Power Required	7,321
Horse Power/ton	2.05
Capital Costs	$6,460,000
Operating Costs	7.0¢/ton

Conveyor Horse Power Requirements[3]

$$\text{HP empty conveyor} = \frac{C\,(L + Lo)(.03QS)}{990}$$

$$\text{HP convey mat'l horiz.} = \frac{C(L + Lo)\,T}{990}$$

$$\text{HP elevate mat'l} = \frac{TH}{990}$$

S = Belt speed in feet per minute
H = Distance maten'd is lifted = 1600 ft.
C = Friction Factor = 0.03
L = Slope Length of Belt Flight
Lo = Length Factor = 150
T = Peak Capacity in tons per/hr

Material will be loaded onto belt at a constant rate from a bin.

Q = Weight in lbs. of moving parts & belt for heavy service and material weight over 100 lb/ft^3 = 115

$$T = \frac{75{,}000}{21} = 3571 \text{ tons/hour}$$

$$\text{HP empty belt} = \frac{0.03\ (2600 + 150)\quad (0.03)\quad (115)\quad (500)}{990} = \underline{\underline{144}}$$

1

INMINCO	ENGINEERING COMPUTATIONS

COMPANY U. S. Bureau of Mines
LOCATION "Continuous Haulage from Open Pits"
DESCRIPTION Incline Belt Conveyor

$$\text{HP convey material} = \frac{(0.03)\ (2600 + 150)\ (3571)}{990} = \underline{298}$$

$$\text{HP elevate material} = \frac{(3571)\ (1600)}{990} = \underline{5771}$$

Total demand HP = (2)(144)+(2)(298) + 5771 = 6655

Service Factor = 1.10

Motor HP = (6655)(1.10) = 7321

Manpower Requirements

1 Remote operator/shift
3 Man Maintenance crew/day

Capital Costs

Conveyor Costs[4]
$632/Lin. Ft. x 0.8 (Cable Support Belt) = $506/Lin. Ft.
$506/Lin. Ft. x 5,178 Ft. = $2,620,000

Slope[5]
Size - 12 Ft. x 20 Ft.
Cost/Slope - $1,600/Vert. Ft. x (1.5 Escalation factor) = $2,400/Ft.
Slope Lift 1,600 Vert. Ft. 2,400 = 3,840,000

TOTAL COSTS $6,460,000

Cost Per Ton

Capital Cost/Yr (20 yrs) Straight Line Depreciation	$ 323,000
Labor/yr (6) @(20,000)	120,000
Power at .02¢ KWhr = (7321)(.7457)(.02)(21)(250)	573,000
HP x (0.7457 KW/hp) x Cost/KWhr x 21 hr/day x 250 days/yr	$ 1,016,000
$1,016,000 ÷ (75,000)x(250) = $0.054	$ 0.054
Maintenance: 30% of capital, labor & power costs (0.054 (0.3) = $0.016	0.016
TOTAL COST PER TON (= 7.0¢/ton)	$ 0.070

2

INMINCO	ENGINEERING COMPUTATIONS
COMPANY	U. S. Bureau of Mines
LOCATION	"Continuous Haulage from Open Pits"
DESCRIPTION	Vertical Friction Hoist

Parameters

Hoisting	-	75,000 tons per day
Depth	-	1,600 ft.
Hoisting Day	-	21 hours
Operating	-	3 shifts/day, 5 days/week

Summary

	Friction Hoist
Capacity tons/hour	1,800
Horse Power Requirements	4,852
Horse Power/ton	2.70
Man power reg/shift	4
Capital costs	$6,000,000
Costs/ton Mat. handled	12.6¢
Capital Costs 2 hoist system	$12,000,000

Vertical Friction Hoist [6]

Design for Friction Hoist:

A. Velocity 30 f.p.s.

B. Desired Capacity (t.p.h.)

$$\frac{75{,}000 \text{ t.p.d.}}{21 \text{ hours}} = 3{,}571 \text{ t.p.h. will use } 3{,}600 \text{ t.p.h.}$$

By Nordberg Process Machinery Reference Manual, it shows 3,600 t.p.h. as not feasible so will use two hoists w/1,800 t.p.h. each. Reference pg. 12-13.

C. SKIP LOAD

$$\text{Skip load (sl)} = \frac{\text{depth (in ft.)}}{\text{Velocity (f.p.s.)} + 0.4 \text{ velocity (f.p.s.)} + 12}$$

$$\frac{3{,}600}{\text{Capacity (TPH)}}$$

$$SL = \frac{\frac{1600}{30} + (0.4)(30) + 12}{\frac{3{,}600}{1{,}800}} = 38.67 \text{ tons}$$

use 39 tons

1

INMINCO	ENGINEERING COMPUTATIONS

COMPANY U. S. Bureau of Mines
LOCATION "Continuous Haulage from Open Pits"
DESCRIPTION Vertical Friction Hoist

D. ROPE SIZE

$$\text{diameter (d)} = \sqrt{\frac{\text{skip load} = \frac{\text{skip weight}}{K_1}}{\text{Safety factor} - K_2 X \text{ depth}}}$$

number of ropes

Skip Weight = Skip load x .75
= 39 x .75 = 29.25 use 29.3

Due to the fact that a 39 ton load is considered large from a hoisting standpoint, six ropes will be used. This will also reduce rope size and their drum diameter.

$$d = \sqrt{\frac{(39 + 29.3)}{\left[\left(\frac{61.6}{7.25}\right) - ((200122\ (1600))\right]}} = 1.32"$$

6

Will use six 1 3/8 inch diameter locked coil ropes.

Factor K_1, K_2 and safety come from pg. 12-14 of Nordberg handbook of North American Koepe Hoist.

E. DRUM DIAMETER

D/d ratio $\stackrel{\text{n}}{=}$ 86 (pg. 12-16 Nordberg)

Therefore, (86) (1 3/8") = 118.25 inches = 9.85 ft.
9.85 ft. diameter drum is considered reasonable.

F. CYCLE TIMES

$$\text{Cycle Time (CT)} = \frac{3{,}600}{\text{t.p.h.}} \times \text{skip load}$$

$$\frac{3{,}600}{1{,}800} \times 39 = 78 \text{ sec.}$$

Running Time (RT) = CT - rest (rest=creep time + idle time)
rest assumed to be 12 sec. = 78-12 = 66 sec.

ta = acceleration time in seconds
tr = retardation time in seconds

Acceleration assumed to be 2.5 f.p.s.^{2}
Deceleration assumed to be 2.5 f.p.s.^{2}

$ta = tr = \frac{V}{2.5} = \frac{30}{2.5} = 12.00$ sec.

G. FACTORS

Equivalent/Effective Weight (EEW) (pg. 12-18 Nordberg) = 35,000 lbs.

Skip Load (SL) Skip Weight (SW)

Rope Weight (R) = (depth)(rope wt/ft) (no. of ropes)

(1600) (4.78) (6) = 45,888 lbs. = 22.944 tons

Total Suspended Load (TSL) = EEW + SL + 2 (SW) + 2R

= 35,000 + 39(2000) + 2 (39) (.75) 2000 + 2(2000) (22,944) = 321,776 lbs.

Velocity = V (f.p.s.)

H. HORSEPOWER REQUIREMENTS

$$HP_1 = \frac{TSL \times V^2}{32.2 \times ta \times 550}$$

$$= \frac{(321,776)\ (30^2)}{(32.2)(12)(550)} = 1363$$

$$HP_2 = \frac{TSL \times V^2}{32.2 \times tr \times 550}$$

$$= \frac{(321,776)\ (30^2)}{(32.2)(12)(550)} = 1363$$

$$HP_3 = \frac{SL \times V}{550} = \frac{39\ (2000)\ (30)}{550} = 4255$$

$$HP_4 = \frac{SL \times V}{550} \times .111 = \frac{39\ (2000)(30)}{550} \times .111 = 472.3$$

3

INMINCO	ENGINEERING COMPUTATIONS

COMPANY U. S. Bureau of Mines
LOCATION "Continuous Haulage from Open Pits"
DESCRIPTION Vertical Friction Hoist

A - (peak accelerating h.p.) = $HP_3 + HP_4 + HP_1$

= 4255 + 472 + 1363 = 6090

B - (full speed) h.p. = $HP_3 + HP_4$

= 4255 + 472 = 4727

C - (total retardation h.p.) = $HP_3 + HP_4 + HP_2$

= 4255 + 472 - 1363 + 3364

HP_5 (h.p. required to accelerate motor rotor) = $\frac{.75A \times 1.2}{ta}$

= $\frac{(.75)(6090) \times 1.2}{12}$ = 456.8

HP_6 (h.p. required to retard motor rotor) = $-\frac{.75A \times 1.2}{tr}$

= $-\frac{(.75)(6090) \times 1.2}{12}$ = 456.8

D - (total h.p. required to accelerate hoist and motor) = A + HP_5

= 6090 + 457 = 6547

E - (total h.p. required to retard hoist and motor) = C + HP_6

= 3364 - 457 = 2907

Root Mean Square Horsepower for d.c. motor (self ventilated motors)

$$\text{RMS HP} = \sqrt{\frac{D^2 \times ta + B^2 \times tfs + E^2 \times tr}{.75\ ta + tfs + .75\ tr + .5\ rest}}$$

Full speed time (tfs) 66 sec - 12 sec - 12 sec = 42 sec

$$\text{RMS HP} = \sqrt{\frac{(6547)^2(12) + (4727)^2(42) + (2907)^2(12)}{(.75)(12) + 42 + (.75)(12) + .5(12)}}$$

= 4853 HP per hoist

4

INMINCO	ENGINEERING COMPUTATIONS
COMPANY U. S. Bureau of Mines	
LOCATION "Continuous Haulage from Open Pits"	
DESCRIPTION Vertical Friction Hoist	

Manpower Requirements

Entire operation will be automated
1 - Operator - Electrician on surface/shift
1 - Operator - Electrician at shaft bottom/shift
3 man shaft crew, spending 2/3 time on production shafts.

Capital Cost Hoisting[7]

1600' x 28' diameter shaft	
cost/ft. 1650' x 1600 =	$2,640,000
Concrete towers =	1,450,000
Equipment hoists	
Mechanical	500,000
Electrical	950,000
Installation	
Mechanical	300,000
Electrical	200,000
	$6,040,000
ESTIMATES	$6,000,000

Cost/Ton Hoisting 1 Year

1. Capital Cost/yr - (20 years Str. Ln.) 12,000,000 ÷ 20 = $600,000	$ 600,000
2. Labor Cost/yr 4 operators - surface and shaft both 4 shaft maintenance - 1 shift 16 men/day x $20,000/yr.	320,000
3. Power @ 2¢/KWhr # hoists x Hoist HP/Hoist x (0.7457 $\frac{KW}{HP}$) x cost/KWhr x hrs/day x days/yr. 8440 hr x .7457 x $.02 x 21 hrs x 250 days =	660,839
	$11,580,839
$1,580,839 ÷ 250 day ÷ 75,000 tons/day =	8.4¢/ton
Maintenance = 50% of capital, labor and power	4.2
	12.6¢/ton

5

INMINCO	ENGINEERING COMPUTATIONS
COMPANY U. S. Bureau of Mines	
LOCATION "Continuous Haulage from Open Pits"	
DESCRIPTION Costs for Trucks and Shovels	

[1] Bucyrus-Erie Co. letter of June 7, 1978.

[2] WABCO Truck Co. verbal communication June 1977.

[3] Equations and Calculations based on Goodyear Handbook of Conveyor and Elevator Belting

[4] Willis & Paul Corporation, See Appendix C

[5] Escalated costs from "Analysis of Large Scale Non-Coal Underground Mining Methods". USBM Contract Report S0122059.

[6] Equations and Calculation based on:

Beerkircher, Gary, "Proper Selection to fit the Application", Mine Hoist Conference 1975, c1975, The American Institute of Mining, Metallurgical and Petroleum Engineers, Inc.

[7] Verbal Communications Mr. Gary Beerkircher of Rexnord Process Machinery Division (Nordberg Machinery).