PRELIMINARY REPORT

Open Pit Mine Model

MEQB REGIONAL COPPER-NICKEL STUDY

A PRELIMINARY REPORT
DETAILS OF THE OPEN PIT
MINE MODEL

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This open pit mine model is a hypothetical example of one type of mining operation producing 11.333 x 10^6 metric tons per year of crude ore averaging 0.45 percent copper and 0.15 percent nickel. The truck and shovel method selected to remove blasted rock is applicable to similar operations ranging in size from 8 to 20 x 10^6 metric tons of ore per year; however, it is not limited by these boundaries and may indeed be used over a much wider range of production.

Many factors contribute to the selection of an open pit mining method. Probably the most significant considerations are: 1) the geometric configuration of the orebody; 2) the geologic and physical nature of the deposit and surrounding rock; 3) economic factors; and 4) company experience and preference. The truck and shovel loading and hauling method selected for this mine model is only one approach that can be used in open pit mining. Other methods may be justifiable but no comparisons were made at this time.

Certain areas of an open pit operation were not dealt with in this report, such as energy requirements, water management, and long term reclamation. These areas will be covered in detail at a later date.

As this report is preliminary, it is subject to and will be revised as found necessary. All comments from reviewers will be appreciated and carefully considered in these revisions.

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EXECUTIVE SUMMARY

This report creates a model for a hypothetical open pit copper-nickel sulfide mine located near the basal contact of the Duluth Complex in northeastern Minnesota (St. Louis and Lake Counties). The mine model was developed by the Regional Copper-Nickel Study in order to determine approximate mining costs and the requirements of the mine in terms of equipment, land, and manpower.

As a basis for discussion, several assumptions for the hypothetical mine model were made. Based on available assay data, cut-off grade and average grade were set at 0.25 percent copper and 0.45 percent copper, respectively. Average nickel content was assumed to be 0.15 percent. A stripping ratio of 1.3 to 1.0 was used. Using existing geological information, final pit slope angle was assumed to be 45^{o}. The mine was designed to produce 11,333,000 metric tons (mt) of ore per year over a mine life of 20 years. Final pit dimensions are 1737 by 914 meters (5700 by 3000 feet) at the rim of the pit; 1189 by 366 m (3900 by 1200 ft) at the bottom of the pit; and 274 m (900 ft) deep.

Drawing on state-of-the-art techniques, drilling is to be carried out by three rotary drills with 15 inch (in.) diameter bits. The blasting agents used consist of ammonium nitrate fuel oil (ANFO) and aluminized slurry. A normal blast will loosen from 500,000 to 600,000 short tons (st) of rock. A truck-shovel mining method was selected for this open pit mine model. Three 18 cubic yard (yd^3) electric shovels will load the broken rock into 150 st trucks. Eight trucks will be available for hauling ore to the primary crusher and 12 will be assigned to hauling waste rock and lean ore to stockpiles outside the pit. Average haulage distances for ore and waste rock/lean ore are 9750 ft and 15,735 ft, respectively.

The open pit mine model made it possible to obtain specific land requirement figures. The pit and contiguous land will require 400 acres, the 20 m high stockpiles another 1920 acres, and the shops, offices, and storage yards will require 40 acres.

A work force of about 580 persons will be necessary for the mine to function at a production rate of 11,333,000 mt/year.

Operating costs in $/st of copper-nickel ore mined are summarized below.

	$/st of Cu-Ni ore mined	%
Drilling	$0.2171	9.0
Blasting	.1751	7.3
Excavation and Loading	.1487	6.2
Haulage	.8340	34.7
Labor*	.7695	32.0
Auxiliary Equipment	.0631	2.6
Miscellaneous Services	.1960	8.2
Total	2.4035	100.0

*includes all employees except for those operators directly associated with the unit operations of drilling, blasting, excavating and loading, and haulage.

INTRODUCTION

The copper-nickel mineral resources of northeastern Minnesota can be removed from the ground by open pit mining, underground mining, or combinations of both methods. The selection of a mining method is a decision that is made during the later stages of the exploration program. At that time, the various exploration methods should provide enough information to the property owner so that an intelligent preliminary mining feasibility study of the mineral deposit can begin. Morrison and Russell, in the SME Mining Engineering Handbook (1973), have made a list of the factors that affect the selection of a mining method. These factors are as follows: 1) the spatial characteristics of the deposit (size, shape, attitude and depth); 2) the physical (or mechanical) properties of the mineral deposit and surrounding rock; 3) ground water and hydraulic conditions; 4) economic factors, including grade of the ore, comparative mining costs and desired production rates; and 5) environmental factors, such as the preservation of the surface overlying the mine, and the prevention of air and water pollution. Of these factors, the spatial characteristics of the deposit and the physical (or mechanical) properties of the mineral and surrounding rock are fixed and limit the methods that can be employed in mining it. The last three factors are subject to change and are not as restrictive.

Open pit and underground mining of the Duluth Complex are being investigated as part of the Regional Copper-Nickel Study. Open pit mining (or surface mining) will be examined in this first report. To do this, several assumptions must be made in lieu of actual exploration data. These assumptions form the basis for the engineering and economic calculations that make up the portion of this preliminary report identified as the Open Pit Mine Model Development Report. It must be stressed that only a hypothetical model has been developed and that the accuracy of the entire Open Pit Mine Model Development Report depends on the validity of the assumptions made.

ADVANTAGES OF OPEN PIT MINING

An open pit mine has several advantages over an underground mine. The advantages most often recognized as major factors for selecting an open pit mining method over an underground mining method are economy, recovery of ore, grade control, flexibility of operation, safety, and working environment. As will be seen in the following summary, all of these advantages are interrelated.

Economy

The economic advantage of open-pit mines is due primarily to their high level of productivity.

> Surface mining today (1973) has productivity rates of 100 tons per man-shift for small ferrous and nonferrous mines and up to 500 for large coal and industrial mineral operations, including all waste and ore handled. In comparison, underground mining achieves but 10 to 60 tons per man-shift, or about one-tenth that for surface methods. (SME Handbook, 1973, p 17-7)

Productivity differences as they affect mining costs are shown in Table 1.

Table 1. Average excavation cost per cubic yard (yd^3) of material excavated (crude ore and waste) for 1967.

	Cost, $\$/Yd^3$	
Material	Surface Mining	Underground Mining
Metals	0.88	4.34
Nonmetals	1.94	6.78
Coal	0.15	4.00

SOURCE: Table C.4 of Operations Research, Inc., Report for the U.S. Bureau of Mines.

Since open pit mines are generally larger than underground mines there is an economy of scale associated with surface mining. Large equipment with high utilization can be used in place of labor intensive underground mining methods. Because of the area involved in open pit mining there is a less constricted flow of materials and mining equipment. Temporary delays in one of the mining unit operations can usually be worked around without a loss of production; whereas in underground mining the haulage system or hoisting can be a bottleneck for the mine and cause production scheduling problems. Distributing the work force into a number of small underground work spaces is less efficient and more difficult than putting everyone to work in one large, noncongested open pit mine.

Recovery of Ore

The term "recovery of ore" can be defined as the percentage of the ore grade material within the mining limits that is mined out and made available for treatment and upgrading. In an underground mine as much as 50 percent of the ore must be left in place in order to keep the ground in a stable condition so that the mined out areas will remain open for a sufficiently long time. Recovery of ore in this case could never be greater than 50 percent. But in the case of open pit mining recovery of ore is 100 percent for all practical purposes (there will always be some losses involved with transportation of ore). Because of the high recovery and the economic advantages of open pit mines as discussed in the previous section, the physical boundaries of the orebody can be extended further into the lower grade ore bearing material. This makes possible the removal of otherwise uneconomic metal values, increases the total extraction of metal contained in the deposit, and provides for the maximum use of natural resources.

Grade Control

Once an open pit has been developed and mining has begun, there is always a substantial amount of surface area exposed in all three dimensions which can be examined and sampled prior to mining. It can then be decided whether it is economical to mine this area of the pit, and if so, how and when. This feature allows a high degree of grade control at open pit mines. By recording ore grades and ore trends occurring throughout the mine and planning mine production accordingly, a constant grade of ore can be provided to the processing plant. The advantage of an open pit mine in terms of grade control is the ability to better predict the tonnages and types of materials avilable for mining and to plan most logically for the removal of each.

Flexibility of Operation

The size of an open pit and the fact that it is on the surface provides a number of alternatives to removing the ore. Drilling, blasting, and loading can be concentrated in one section of the mine or they can be operated separately from each other. One unit operation need not interfere with any other because of limited space. There are always several haulage routes available so that a steady, nonconflicting flow of material is possible. If a certain type of ore is desired for feed into the processing plant on a given day, the mine can schedule production in order to provide an ore with the proper qualities. Often, a higher grade ore is mined from one area of the pit and is combined at the plant with ore from a lower grade section of the pit, resulting in a more uniform feed. The degree of flexibility that a mine operator has to work with allows him to select the combination of equipment that will optimize the mining of the orebody.

Safety

Injury experience in the mineral industry in the United States is compiled in statistical form by the Office of Accident Analysis, U.S. Bureau of Mines. Table 2 points out that injuries in underground mines are more numerous and more severe than those which occur in surface mines, and Table 3 shows the major causes of disabling injuries in metal mines for the year 1970.

Table 2. Injury statistics, U.S. Mineral Industry--Five year averages, 1966 through 1970.

	Average Number Injuries per year		Frequency Rate		Severity Rate	
Mining Operations	Fatal	Non-Fatal	Fatal	Non-Fatal	Fatal	Non-Fatal
Metal Mines						
Underground	49.2	2,607	0.84	44.56	5,046	2,305
Surface	14.2	550	0.31	12.19	1,888	622
Mills	6.0	609	0.12	11.69	692	645
Nonmetal Mines						
Underground	20.0	651	1.09	35.58	6,554	1,409
Surface	63.6	3,899	0.34	20.69	2,025	1,007
Mills	27.4	3,359	0.15	18.68	914	904
Coal Mines						
Underground	209.4	8,609	1.19	48.82	7,125	2,662
Surface	28.8	1,175	0.59	24.27	3,569	1,032
Preparation plants	7.6	550	0.41	29.32	2,431	1,531

$$\text{Frequency} = \frac{\text{Number of disabling injuries (and/or illnesses)} \times 1,000,000}{\text{Total man-hours of work exposure}}$$

$$\text{Severity} = \frac{\text{Total days of disability} \times 1,000,000}{\text{Total man-hours of work exposure}}$$

Table 3. Major causes of disabling injuries in the United States Mineral Industry for the year 1970.

	Metal Mines					
	Underground Mines		Surface Mines		Mills	
Injury Causes:	Fatal	Non-Fatal	Fatal	Non-Fatal	Fatal	Non-Fatal
Falls of roof or back......	29.5%	10.8%	--	--	--	--
Falls of face or side......	--	5.9	33.3%	1.1%	--	--
Pressure bumps or bursts...	--	0.1	--	--	--	--
Inrush of water............	2.3	0.0*	--	--	--	--
Sliding or falling materials or objects......	--	5.2	--	3.3	12.5%	3.2%
Slips or falls of persons..	11.4	16.3	6.7	22.6	--	26.3
Handling material..........	2.3	22.3	13.3	23.5	12.5	28.9
Handtools..................	--	4.0	--	4.3	--	3.7
Stepping or kneeling on sharp or loose objects....	--	3.2	--	3.3	--	2.6
Striking or bumping against objects..........	--	0.4	--	0.5	--	1.9
Haulage....................	27.3	10.4	6.7	21.2	25.0	4.9
Explosions of dust, gas....	--	0.1	--	--	--	--
Explosives.................	4.5	0.9	--	0.3	--	--
Electricity................	4.5	0.2	20.0	0.6	--	0.7
Machinery..................	11.4	17.1	20.0	15.8	50.0	14.7
Suffocation................	6.8	0.4	--	--	--	2.1
Mine fires.................	--	0.0*	--	0.2	--	0.3
Miscellaneous causes.......	--	2.6	--	3.3	--	10.7
Pneumoconiosis.............	--	0.1	--	--	--	--
	100%	100%	100%	100%	100%	100%

* Injuries were reported but the percentage of occurrence was less than 0.05%.

Working Environment

The working environment in a mine has a direct affect on the safety of the workers and their productivity, which in turn determines the cost of mining. Good lighting, fresh air, reasonable noise levels, comfortable temperatures and humidity, and ample work space are all important if a worker is to consistently perform his duties at an average rate of production. The more favorable working conditions in open pit mines are partly responsible for their high level of productivity. Obtaining a work force with the necessary skills is easier in the case of open pit mining since many of the operations are related to the construction industry. Simply finding people willing to work underground can be a problem. In the past, turnover rates for underground mines have been higher than for open pit mines. In Minnesota, most of the underground labor force would have to be trained since there has been no underground mining in Minnesota for nearly 15 years.

DISADVANTAGES OF OPEN PIT MINING

Environmental Impacts

The disadvantages of open pit mining relate to the environmental impacts of the open pit, stockpiles and construction activities which take place in the mine area. Air quality can be affected by the combustion of fuels and the generation of dust from construction, mining, and vehicular activity. Blasting and the operation of large mining equipment can create noise and vibration problems. The wildlife and plant life that inhabits an area where mining takes place will be disturbed to a greater extent by open pit mining than underground mining because of the larger surface area involved. The impact of open pit mining on local ground water levels and regional water quality are other potential problems.

Along with the environmental impacts of open pit mining are other concerns such as the visual impacts and land use problems brought about by the excavation of an open pit mine and the stockpiling of waste rock. The excavation of a pit often involves land which cannot be reclaimed to its original form. Open pit mining also produces more waste rock than underground mining so more land is required to stockpile this material.

Exposure to Weather

Open pit mine operators must contend with the vagaries of the weather. In the working environment of an open pit, inclement weather, even if it is not severe, can endanger personnel, damage equipment, and cause production delays. In northeastern Minnesota a variety of weather conditions can arise which are cause for concern--wind, fog, rain, hail, snow, and tornadoes--but the biggest problem is the cold. Both men and machinery suffer from the affects of the cold and protective measures must be instituted during the winter season to keep both operating efficiently.

MINE PLANNING

Assuming that a preliminary mining feasibility study indicates that the Duluth Complex copper-nickel deposit can be economically mined and that an open pit mine is selected as the most suitable mining method, then preparation of a report similar to the following Open Pit Mine Model Development Report is commonly the next step taken by the interested mining company. An actual Development Report is for the company's internal use.

From the information obtained through the exploration program and the preliminary mining feasibility study, and with a knowledge of mining equipment, the following factors can be determined: 1) cut-off grade; 2) stripping ratio; 3) pit slope angle; 4) pit dimensions; 5) total ore reserves; 6) mine life; 7) production rate; 8) bench height; 9) road grades; and 10) size limitations on mining equipment.

Once the first three factors are established, the other seven can be determined. These factors are dependent on both economics and the physical nature of the orebody. Their determination is as important as, and probably more vital than, the selection of the mining method. The rest of the Mine Model Development Report is based on these initial figures, but once actual mining has begun, changes can be made in any or all of these factors as more is learned about the orebody or as economic conditions change.

With the absence of detailed exploration data and the objective to prepare a "regionally generic" mine model, the following assumptions have been made.

1) Cut-off grade is defined as the minimum grade at which the value of the product recovered will pay for its mining and treatment and yield the minimum acceptable profit. The cut-off grade will be stated in terms of percent copper, not a combined copper-nickel percentage. For the proposed open pit mine model, the cut-off grade limit will be set at 0.25 percent copper. It is assumed that the ore will average 0.45 percent copper and 0.15 percent nickel.

2) Stripping ratio is defined as the tons of overburden and waste rock that must be removed to gain access to one ton of ore. A stripping ratio of 1.3 to 1.0 will be assumed for the open pit mine model.

3) Pit slope angle is the angle that a line extending from the rim of the pit to the toe of the lowest bench forms with the horizontal. For safety reasons it is dependent on surface and ground water pressures, rock type, geologic features such as jointing patterns and fault structures, and the size and weight of the mining equipment using the benches. A pit slope angle of 45^{o} will be assumed. However, if the basal contact of the Duluth Complex is intercepted by the excavation of the pit and the dip of the contact is less than 45^{o}, the pit slope will be adjusted to conform to the angle of the contact since it is assumed that the contact is the boundary of ore mineralization.

Knowing the cut-off grade, stripping ratio, and the ultimate pit slope angle, the pit limits can be defined and the design parameters of the pit established. The mine planner attempts to remove the maximum amount of ore at the lowest cost and in the safest manner.

4) The dimensions of the proposed model are: 1737 by 914 meters (m) or 5700 by 3000 feet (ft) at the rim of the pit; 1189 by 366 m (3900

by 1200 ft) at the bottom of the pit; and 274 m (900 ft) deep.

5) A pit of this size and with a 1.3 to 1.0 stripping ratio will produce 226,660,000 metric tons (249,850,000 short tons) of ore and 294,660,000 mt (324,800,000 st) of waste rock and lean ore in its lifetime.

6) A 20 year life of the mine will be assumed.

7) Annual production will be 11,333,000 mt (12,492,000 st) of ore and 14,733,000 mt (16,240,000 st) of waste rock and lean ore.

From a list of 28 new copper mines being developed in North and South America (excluding Canada) that are scheduled for completion in the years 1975-1982, the average production in tons of metal content per year is 48,464 mt (53,420 st), with a standard deviation of 38,900 mt (42,880 st) (Mining Journal Annual Review 1976, p. 38). As shown by the standard deviation, production varies considerably from one mine to the next, the range being 3,000 to 154,000 mt of copper metal per year. The proposed Duluth Complex open pit copper-nickel mine model will make available about 51,000 mt (56,200 st) of copper per year.

8) The bench height will be assumed at 15 m (50 ft).

9) Maximum continuous grades on the haulage roads in the pit will be limited to 8 percent

10) The mining equipment will be limited to not more than 10 m (33 ft) wide.

A summary of data on the hypothetical open pit mine and the waste rock and lean ore stockpiles associated with this mine is listed in Table 4.

The specific gravity of the rocks in the Duluth Complex is approximately equal to 3.0. Broken rock occupies more volume than rock in place. This volumetric increase in called swell and is assumed to be 50 percent in this model. Swell factor is equal to 100/(100 + percent of swell) or 66.7 percent. For broken rock then, the bulk specific gravity becomes 2.0.

Table 4. Open pit mine model data.

	Units	
	Metric	English
Cut-off grade	0.25% Cu	0.25% Cu
Annual ore production rate	11,333,000 mt/yr	12,492,000 st/yr
Annual waste rock + lean ore extraction rate	14,733,000 mt/yr	16,240,000 st/yr
Stripping ratio	1.3:1.0	1.3:1.0
Mine life	20 years	20 years
Tons of material produced over a 20 year mine life		
Ore	226,660,000 mt	249,850,000 st
Waste rock + lean ore	294,660,000 mt	324,800,000 st
Pit slope angle	45^{o}	45^{o}
Pit dimensions		
Depth	274 m	900 ft
Rim of pit	1737 x 914 m	5700 x 3000 ft
Bottom of pit	1189 x 366 m	3900 x 1200 ft
Approximate waste rock + lean ore volume	147,400,000 m^3	5,205,000,000 ft^3
Approximate surface area required for waste rock + lean ore Slope of 14^{o}; average height of 20 m (66 ft)	7.77 km^2	3.00 mi^2 (1920 acres)

MINING OPERATIONS

Scheduling

The mine will operate 51 weeks per year, 20 shifts per week. This results in 8160 scheduled hours of operation per year and will require four working man-shifts. At the desired rate of production, the mine must extract ore at an average rate of 1389 mt per hour (1531 st per hour) and waste at an average rate of 1806 mt per hour (1990 st per hour).

Development

After work commences at an open pit mine there is an interval of time during which development takes place but little or no ore is removed. At this time ore is being exposed, equipment is being purchased, the buildings for shops and offices are being erected, and if planned for, the mill is under construction. These activities are scheduled so that the start up of the mill and continuous ore production (at the rate desired at this stage) occur simultaneously.

Clearing and Stripping--Before ore can be mined by open pit methods the surface must be cleared and the overburden removed. Overburden is defined as barren or non-ore material that overlies, and must be removed to gain access to, minable-grade material. For the mining company, the ultimate aim of stripping the overburden is to remove it at the lowest possible cost. The selection of a stripping method is dependent on the size of the orebody, the distribution of metal values within the orebody, the nature of the overburden, the life and production rate of the mine, the haulage distances to each disposal area, reclamation plans, and the future use of the stripping equipment. These considerations will usually narrow the choice of stripping methods to one or two possibilities. A detailed cost analysis can then be used to make the final selection. A listing of the attributes of the various types of equipment available for stripping operations (as well as for mining) will outline the selection possibilities (Surface Mining 1968, p 167-169).

Excavators

Shovels:

1) Can give high production.
2) Can handle all types of material, including large blocky material.
3) Are limited to fairly rigid operating conditions.
4) Require supporting equipment for waste disposal except in some strip mining.
5) Have limited mobility.

Draglines:

1) Have the ability to dig well above and below grade.
2) Can function under less rigid operating conditions than shovels.
3) Are only 75 to 80 percent as efficient in production as a shovel of comparable size due to less precise motions.
4) May or may not require supporting waste-haulage equipment.
5) Are normally used for handling unconsolidated and softer material, but larger units can handle blasted rock.

Scrapers:

1) Have excellent mobility.
2) Are limited to fairly soft and easily broken material for good production, although they can handle broken material up to about 24 inches in size.
3) Usually require pushers to assist in loading.
4) Usually are operated without supporting disposal equipment where the distance to the dump area does not exceed one mile.

Bucket-Wheel Excavators:

1) Must be operated under very rigidly engineered conditions.
2) Have very high capital cost.
3) Are limited to fairly easy digging.
4) Are capable of high production rates.
5) Require auxiliary disposal systems.

Haulage Equipment

Bulldozers:

1) Are economically limited to a fairly short operating radius of about 500 ft.

Scrapers:

1) Require good roads to minimize tire costs.
2) Are fast but are economically limited to an operating radius of approximately one mile.

Trucks:

1) Require good roads to minimize tire costs.
2) Can negotiate steep ramps.
3) Usually are limited by economics to an operating radius of about three miles.
4) Are very flexible.
5) Can handle coarse, blocky material.

Trains:

1) Are high-volume, long-distance, low-unit-cost carriers.
2) Track requires careful conformity to engineering specifications.
3) Have a high initial capital cost.
4) Cannot handle adverse grades much greater than 3 percent.
5) Can handle coarse, blocky material.

Conveyors:

1) Are high-volume, long-distance, low-unit-cost carriers.
2) Are difficult and costly to move.
3) Have a high initial capital cost.
4) Can handle steep adverse grades (up to about 40 percent).
5) Require material broken into fairly small pieces for good belt life.
6) High maintenance cost.

> If analysis points to only a small difference in costs between two or more proposed stripping systems, the consideration of post-stripping equipment uses may point to the better choice. Most open pits strip concurrently with mining operations. Definite savings in maintenance and repairs are realized when the variety of equipment used is held to a minimum. Thus, if an alluvial overburden can be removed as cheaply with trucks and shovels as it can be with rippers and scrapers, but the orebody itself will require the use of trucks and shovels, the truck and shovel combination will obviously be the better choice for moving the overburden as well.

The shovel-truck stripping method will be used to remove the overburden in the present copper-nickel mine model. This method is advantageous for the following reasons: 1) the combination provides a high degree of flexibility; 2) the method is very mobile, even in small work areas; 3) the trucks can negotiate steep ramps; 4) trucks are best suited for the haulage distances involved; and 5) the same equipment can be used later on during the mining operation.

The overburden in northeastern Minnesota consists of unconsolidated glacial deposits of sand, clay and gravel which range from 0 to 25 m (82 ft) in thickness with an average thickness of 9 m (30 ft). Peat bogs cover about one-fourth of the area. Loamy soil, a friable mixture of varying proportions of clay, silt, and sand, is the dominant soil type. All soil material that can be used in the future for the restoration of ore stockpiles and tailings areas should be segregated and stored.

Land clearing and overburden stripping are activities which increase erosion potential in the area. Factors contributing to soil loss and, ultimately, water contamination from clearing and grubbing operations are:

1) Failure to install perimeter control measures prior to the start of clearing and grubbing

2) Exposure of soils on steep slopes

3) Clearing and grubbing too far ahead of the pit, exposing the soil for an excessive length of time

4) Improper placement and/or protection of salvaged and stockpiled topsoiling material

5) Creation during clearing and grubbing operations of a soil surface that impedes infiltration and/or concentrates surface runoff (for example, leaving ripper marks or dozer cleat marks that run up and down the slope rather than along the contour) (Erosion and Sediment Control 1976, p 6)

Haulage Roads and Mining Benches--Development of the mine continues with the construction of haulage roads and the establishment of mining benches.

The haulage road width is determined by the type and size of equipment using the road and the desired speeds. The road should be wide enough to allow faster vehicles to overtake slower moving vehicles so that the pace is not set by the slowest vehicle. A haulage road width of 80 ft should provide adequate space for two-way travel of 150 st rear dump trucks.

Thoughtful design and continual maintenance of roads are very important since tire repair and replacement can account for 20 to 40 percent of the haulage operating cost. Curves should be as gradual as possible and care should be taken in the design of the superelevation of the curves. Construction of the road's subbase and surface should be in accordance with the results of a soil survey and the anticipated traffic loads. In metal mines, blasted mine waste rock often provides a good base and coarse mill tailings or crushed slag can be used for surface material. Drainage ditches, culverts, safety berms and the roadway crown should all be considered in the design of the haulage road.

> Water trucks should be utilized to wet the surface of the road for several reasons. First, the watering provides an effective dust control which is important in assuring the safety of the haulage truck drivers, company personnel, and public using the road. The dust control measures subsequently cut haulage expenses by saving on truck repairs--wear on bearings and engines caused by the dust. In addition, the watering aids in the compaction of reworked road surfaces.
>
> Incorporation of calcium chloride, lignosulfonates, or other such chemicals into the road surface will aid mechanical stability and give a firm, hardpacked wearing surface with smoothness and riding qualities approaching those of more permanent, higher-cost roads. Treatment with chloride salts keeps roads moist in dry weather, since the salts retard evaporation and attract moisture from the air. Continued periodic applications result in a relatively dustless surface that is readily shaped to proper cross section and easily maintained with good riding surfaces. In some metal mines, particularly copper ore operations, certain chemicals on roads within the orebody may adversely affect metallurgical treatment (Surface Mining 1968, p 81).

Sanding the roads may be necessary at times during the winter season.

Roadways are a major source of fugitive dust and sediment release. Fugitive dust will be generated from all unpaved roads. The quantity of dust emitted varies with soil particle size, soil moisture content, and vehicle speed.

Sediment action will only be a problem when the sediment generated is not contained within the mine and finds its way into public watercourses. The roads involved in this case will be the access roads and the length of the haulage road from which the sediment will not be transported into the confines of the pit.

Roadways serve to intercept, concentrate, and divert surface runoff, resulting in increased soil loss from roadway surfaces, ditches, cut slopes, outslopes, and safety berms. Additionally, the overall increase in the rate of runoff resulting from the construction of a relatively impermeable roadway surface, the clearing and steepening of slopes, and the interception and concentration of sheet runoff from upland areas will accelerate erosion within natural drainageways. Accelerated onsite and offsite erosion will continue to be a source of water contamination well beyond the life of the mine if, when the mine is abandoned, measures are not taken to stabilize exposed surfaces permanently with vegetation and to minimize disruption of the natural drainage system.

Factors contributing to soil loss from roadways and offsite areas affected by the roadways are outlined below:

1) Poor location of the roadway, resulting in one or more of the following adverse conditions:

-The presence of excessively long or steep grades contribute to erosion by concentrating runoff and increasing its flow velocity.
-Disturbance, either by filling or excavation, of unstable slopes or areas having a high ground water table may result in landslides, muddy roadbed conditions, and revegetation problems.
-Failure to preserve vegetated buffer (filter) areas along waterways allows the movement of sediment from the roadway into the waterway.
-Creation of unnecessary, or unsuitable, stream crossings, contributes to erosion of the banks and bed of the affected waterways.

2) Improper construction of the roadbed:

-Rutting and saturation of the roadway results from failure to provide adequate bearing capacity and/or subsurface or surface drainage. These conditions are conducive to gully erosion and landslides.

-Failure to provide a surfacing material, such as clinker or crushed stone, or good compaction seal on suitable material, exposes the soil to the erosive action of water, wind, and traffic damage.

3) Improper layout and construction of drainage structures:

-Failure to properly size, shape, and stablilize ditches: Improper sizing and shaping can result in increased flow velocities, which increase soil loss and the ability of the runoff to carry sediment into adjoining waterways. Lack of adequate stabilization with structures and/or vegetation makes the channel more susceptible to erosion and also provides for increased flow velocity.
-Improper handling and disposal of concentrated runoff: Failure to properly install culverts, or other conduits, to carry concentrated flow beneath the roadway can result in gully erosion within the ditch, flooding, and subsequent saturation of the roadway. In some instances (especially where sidehill fills are present), landslides may result. Disposal of concentrated flow, such as the runoff discharged from culverts, can cause severe gully and stream-channel erosion if stabilization and energy dissipation measures are not used.

4) Poor maintenance practices:

-Failure to control dust during dry periods: Dust particles deposited in ditches, on the roadbed, and uphill from the roadway was washed readily into adjoining drainageways during rainfall events.
-Pushing soil into the ditch when performing maintenance grading.

5) Inadequate stabilization of cut and fill slopes:

-Construction of excessively steep slopes: Slopes steeper than 2:1 or 50 percent are difficult, if not impossible, to stabilize adequately with vegetation. Excessively steep slopes also increase the likelihood of landslides.
-Failure to establish vegetation properly: Improper selection and application of plant materials, soil supplements, and mulches along with negligent maintenance practices result in only partial protection of steep slopes.

6) Failure to protect safety berms:

-Shaping the roadbed to allow runoff to concentrate and flow along the berm: Berms located along crowned roadways and constructed of loose overburden that is devoid of vegetative cover are particularly vulnerable to erosion.
-Absence of stabilized outlets, or improper placement of outlets, along the berm: Failure to provide stabilized breaks at periodic intervals along the berm will contribute to an increase in flow velocity and erosion (Erosion and Sediment Control 1976, p 7-9)

The width of the working mining benches is selected after careful consideration has been given to all of the types of activity that will take place on any bench or level at any one time. A working bench width of 150 ft will be assumed for the open pit mine model. This should provide sufficient space for the efficient operation of the shovel-truck loading arrangement.

Waste Rock and Lean Ore Stockpiles

In any open pit mine a substantial percentage of the material removed is barren, waste rock and material which is too low in grade to be treated economically at the present time, but which must be removed to expose underlying ore grade material. The lean ore and the waste rock should be segregated so that the contained copper and nickel metal values can be recovered more easily in the future. As new technology is developed and existing resources are depleted and copper and nickel prices increase, lean ore stockpiles may convert from major copper and nickel resources to copper and nickel reserves. The mine waste rock may contain large quantities of aluminum which possibly can be recovered by developing technology which uses sulfuric acid generated as a by-product of smelting.

The two types of material will be piled into one or several stockpiles located outside the pit limits and as near to the mine's exit as is possible. Proper care should be taken so that the stockpiles are not placed in areas where they may interfere with future mining activity or other important land uses. However, the double handling problem created by dumping within pit limits must be weighed against loss of productivity, and maintenance and safety considerations if the conditions are such that avoidance of the double handling problem results in substantially increased haul distances and/or sharp curves.

The open pit copper-nickel mine model will produce 294,660,000 mt of combined waste rock and lean ore over 20 years of production. This material should swell approximately 50 percent and will occupy a volume of about 147,400,000 m^3 (5.205×10^9 ft^3). In order to get an idea of how much land area would be required for stockpiling waste rock and lean ore it will be assumed that the stockpiles will be piled 20 m (66 ft) high and have a slope of 14^o (1:4). One large stockpile (divided into a waste rock area and a lean ore storage area) based on this design would cover 7.77 square kilometers (km^2) (1920 acres) after 20 years of mining. If a number of smaller stockpiles were incorporated, the total surface area involved would increase.

To reach the stockpile, mine trucks will have to travel an average of 760 m (2500 ft) from the point where they exit from the pit.

The rocks in the waste rock and lean ore stockpiles have a large surface area exposed to air and water. Because the rocks do contain metal sulfides there is the possibility of the metals being leached out and entering surface and ground waters. Factors which tend to increase the leaching potential of lean ore stockpiles are:

1) Failure to discriminate between ore grade and below ore grade material, thereby placing ore grade material in with the waste rock/lean ore

2) Failure to install perimeter control measures to contain runoff

3) Failure to isolate the stockpiles from ground water

4) Leaving a depression under the stockpiles which serves as a trap for surface waters

5) Placing the stockpiles on top of pads constructed of porous material or directly on porous overburden

6) Failure to properly revegetate the stockpiles as they are built up

Equipment Selection

As a rule, a mine as large as the open pit copper-nickel mine model will benefit economically by using large equipment. This is primarily due to the reduction in labor force (both machine operators and repair personnel) when a few pieces of equipment can be used in place of a larger number of units of smaller capacity. Also, organizational and scheduling problems are minimized by using larger equipment.

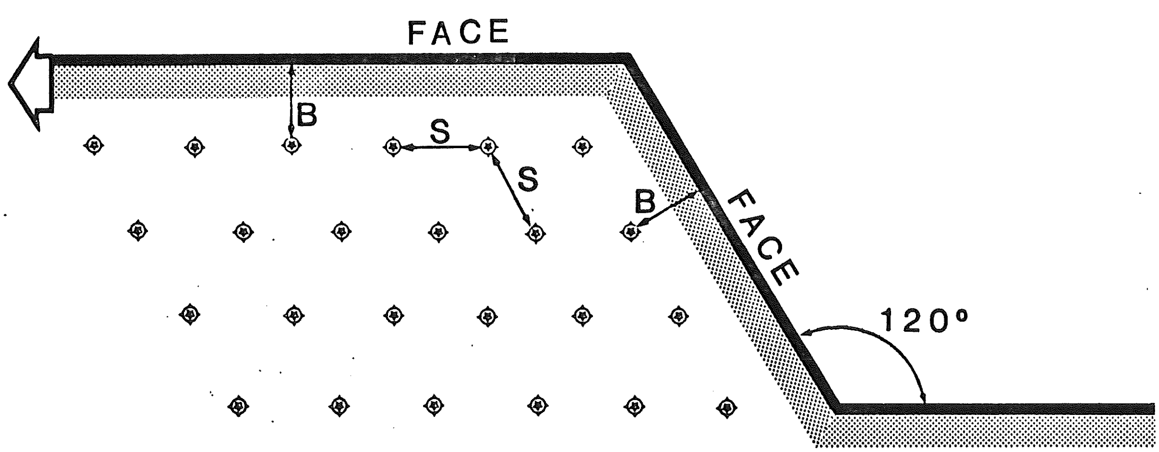
However, at some point the advantages of scaling up equipment size disappear. Large initial capital outlays, vulnerability to breakdowns, and proportionally higher costs for energy, spare parts and maintenance begin to outweigh the savings of using fewer pieces of equipment.

In order to select the optimum equipment sizes to incorporate in the open pit mine model, a cost analysis which compares the more reasonable choices must be made. This has been done for the unit operations of drilling, blasting, excavating, and hauling. Costs are reported in $/mt and in $st of copper-nickel ore mined. All costs are from the first quarter of 1977. English units are used in most of the engineering calculations since equipment specifications are presently stated in English units.

Drilling and Blast Design--Open pit mines most commonly use rotary drills with bit diameters of 6.5 to 15 in. to drill blastholes. As bit diameters increase, greater pull down pressures can be used and the drill rigs and the components used in them become bigger, heavier, and stronger. This results in nearly equivalent penetration rates for rotary drills of varying sizes. However, because of the larger diameters, fewer holes are required to achieve the proper rock fragmentation.

In selecting the blast hole diameter (D_e) to be used in the open pit model, the economics of 12¼ and 15 in. blastholes were compared. The blasting design is as follows (based on the formulas of R.L. Ash):

Figure 1. Plan view of a mining bench showing the drilling and blasting pattern, burden (B) and spacing (S).



For 12¼ in. holes the burden (B) is 24 ft and the spacing (S) is 28 ft. For 15 in. holes the burden is 28.5 ft and the spacing is 33 ft. The other design parameters which are listed below are less dependent on blasthole diameter.

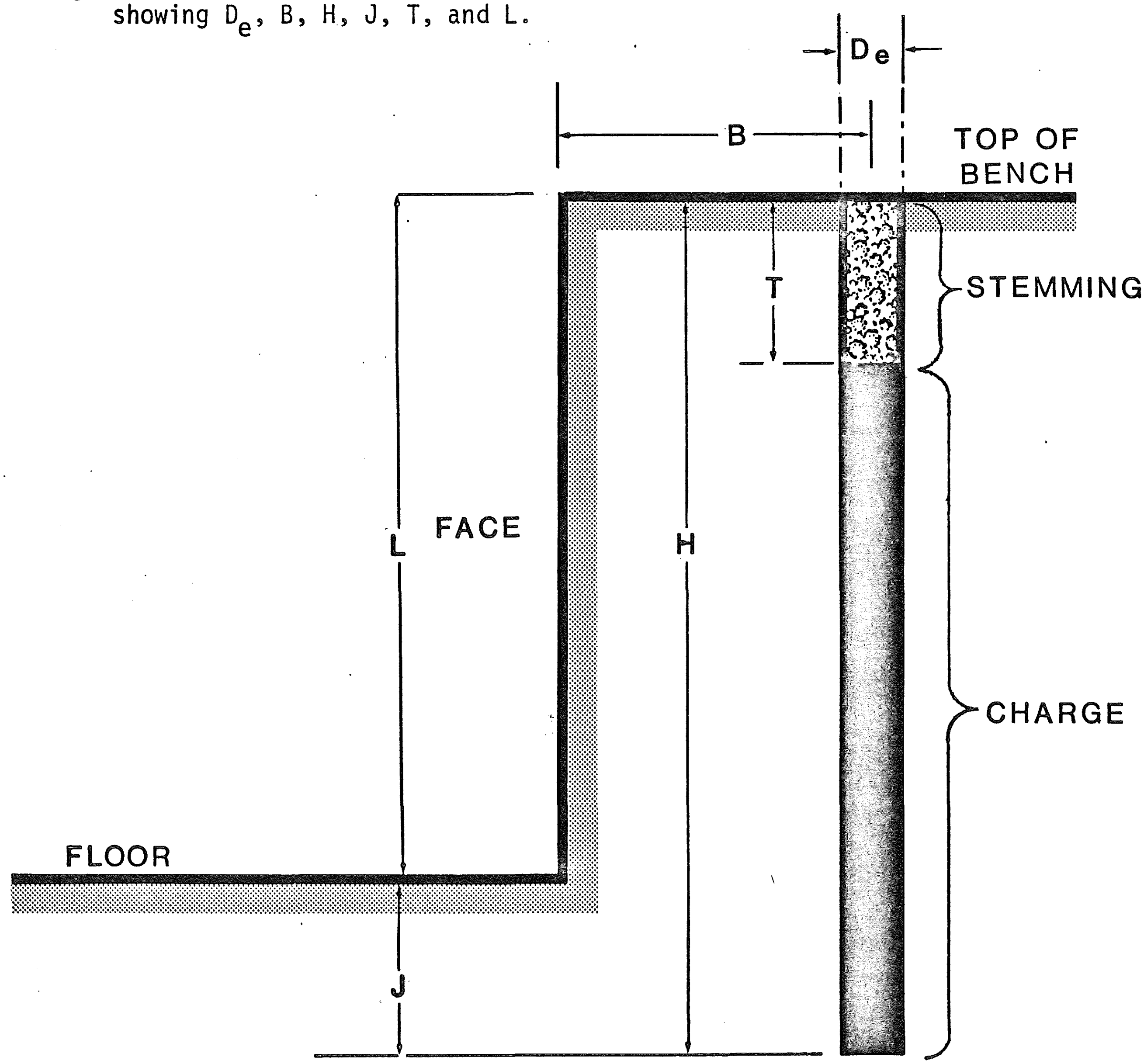
L = Bench height = 50 ft

J = Subdrilling = 5 ft

H = Hole length = 55 ft

T = Collar distance = 20 ft (stemming)

Figure 2. Bench cross section view showing D_e, B, H, J, T, and L.



The powder column is 35 ft high. The explosive mixture is approximately two-thirds ammonium nitrate fuel oil (ANFO) and one-third aluminized slurry, so each hole averages 23 ft of ANFO and 12 ft of aluminized slurry. Stemming material will fill the top 20 ft of each hole. This stemming material helps confine the gases produced upon explosive detonation and thus helps reduce airblast. The gases contained are then available to do further work in moving the rock.

The density of ANFO is 50 lb/ft^3.

The density of aluminized slurry is 70 lb/ft^3.

Pounds of explosive per 15 in. blasthole:

$$\text{ANFO} = \frac{3.14}{4} (15/12)^2(23)\ ft^3 \times 50\ lb/ft^3 = 1400\ lb$$

$$\text{Slurry} = \frac{3.14}{4} (15/12)^2(12)\ ft^3 \times 70\ lb/ft^3 = 1000\ lb$$

Pounds of explosive per 12 in. blasthole:

$$\text{ANFO} = \frac{3.14}{4} (23)\ ft^3 \times 50\ lb/ft^3 = 900\ lb$$

$$\text{Slurry} = \frac{3.14}{4} (12)\ ft^3 \times 70\ lb/ft^3 = 645\ lb$$

Tons of rock broken/week = 12,492,000 st/year x 2.3/51 weeks/year = 563,400 st/week

Volume of rock broken/week = 563,400 st/week x 10.68 ft^3/st = 6,017,000 ft^3/week

There will be one blast each week.

Bench height is 50 ft.

$$\text{Area of pattern per blast} = \frac{6{,}017{,}000\ ft^3}{50\ ft} = 120{,}300\ ft^2$$

<u>15 in. blastholes</u>: four rows will provide sufficient working space

D = depth of pattern = 33 ft (sin 60^0) x 4 rows = 114 ft

L = length of pattern = $\frac{120,300}{114}$ = 1055 ft, $\frac{1055}{33}$ = 31,97, use 32 holes per row

32 holes in each row x 4 rows = 128 holes/pattern = 128 holes/week

Actual rock broken per blast = Area x 50 ft = 32 holes/row x 33 ft spacing x 114 ft x 50 ft = 6,019,200 ft^3 (563,600 st)

Total footage to be drilled per week = 55 ft/hole x 128 holes = 7040 ft

Use a penetration rate of 25 ft per hour; 200 ft per shift.

Availability is 85 percent.

$$\frac{7040 \text{ ft}}{\text{week}} \times \frac{1 \text{ drill-shift}}{200 \text{ ft}} \times \frac{1 \text{ week}}{20 \text{ shifts}} = 1.76 \text{ drills}; \frac{1.76}{0.85} = 2.07 \text{ drills}$$

Use two drilling rigs with 15 in. diameter bits, plus have one spare.

<u>12¼ in. blastholes</u>: five rows will provide sufficient working space

D = 28 ft (sin 60^0) x 5 rows = 121 ft

L = $\frac{120,300}{121}$ = 994 ft, $\frac{994}{28}$ = 35.5, use 36 holes per row

36 holes in each row x 5 rows = 180 holes/pattern = 180 holes/week

Actual rock broken per blast: Area x 50 ft = 36 holes/row x 28 ft spacing x 121 ft x 50 ft = 6,098,400 ft^3 (571,000 st)

Total footage to be drilled per week = 55 ft/hole x 180 holes = 9900 ft

Use a penetration rate of 25 ft per hour; 200 ft per shift.

Availability is 85 percent.

$$\frac{9900 \text{ ft}}{\text{week}} \times \frac{1 \text{ drill-shift}}{200 \text{ ft}} \times \frac{1 \text{ week}}{20 \text{ shifts}} = 2.48 \text{ drills}; \frac{2.48}{0.85} = 2.91 \text{ drills}$$

Use three drilling rigs with 12¼ in. diameter bits, plus have one spare.

Blasting is to occur once a week, as weather conditions permit. To loosen the necessary rock with the preceding blast pattern, 128-fifteen inch holes would have to be drilled each week, or 180-twelve and one-quarter inch holes.

When drilling and blasting costs for 15 in. holes and for 12¼ in. holes are compared, the results indicate that 15 in. diameter blastholes are cheaper than 12¼ in. diameter blastholes.

	15 in.	12¼ in.
Drilling	$.2393/mt	$.2944/mt
Blasting	.1930.mt	.1791/mt
TOTAL	$.4323/mt Cu-Ni ore	$.4735/mt
Drilling	.2171/st	.2671/st
Blasting	.1751/st	.1625/st
TOTAL	$.3922/st Cu-Ni ore	$.4296/st

The powder factor (pounds of explosives/st or rock broken) for 15 in. holes is 0.55 and for 12¼ in. holes it is 0.49.

A breakdown of the drilling and blasting costs for 15 in. diameter blastholes is listed below:

Drilling	$/ft	% of Operating Cost
Depreciation	$ .22	
Interest, Insurance & Taxes	.21	
Maintenance	1.85	27
Power	1.53	22
Oil, Filters & Grease	.25	4
Labor	.84	12
Bit Cost	2.35	35
	$7.25	100

Blasting	
Explosives & Supplies	$252.33/hour
Labor	$ 15.74/hour

Excavation and Loading--Power shovels of 15 yd^3 and 18 yd^3 capacity were compared and evaluated in terms of their applicability and costs.

The number of shovels required in order to excavate 3521 st of material per hour (1531 st of ore plus 1990 st of waste rock) on a continuous basis is calculated as follows:

Operating Efficiency = 75% (6 hours/8 hour shift)

Dipper Fill Factor = 85%

Swell Factor = 67%

Swing Factor (110°) = 0.9 (a factor that corrects for shovel swings of other than 90°)

18 yd^3 shovel

Effective load per cycle (loose density x capacity x fill factor)

$$\left(2.527\ \frac{st}{yd^3} \times 0.67\right) \times 18\ yd^3 \times 0.85 = 25.91\ st/cycle$$

Effective cycles per hour; Cycle time is 36 seconds (sec) for a 90° swing

$$\frac{3600\ \frac{sec}{hour} \times 0.75\ efficiency}{36\ \frac{sec}{cycle}\ /0.9} = 67.5\ cycles/hour$$

Hourly production

$$25.91\ \frac{st}{cycle} \times 67.5\ \frac{cycles}{hour} = 1750\ st/hour$$

$$\text{Number of units required} = \frac{3521\ st/hour}{1750\ st/shovel\text{-}hour} = 2.01\ shovels$$

The mine model will require that two shovels (plus one spare) of 18 yd^3 capacity be used.

15 yd^3 shovel

Effective load per cycle

$$(2.527 \frac{st}{yd^3} \times 0.67) \times 15\ yd^3 \times 0.85 = 21.59\ st/cycle$$

Effective cycles per hour; cycle time is 34 sec for a 90^0 swing

$$\frac{3600 \frac{sec}{hour} \times 0.75\ efficiency}{34 \frac{sec}{cycle}/0.9} = 71.47\ cycles/hour$$

Hourly production

$$21.59 \frac{st}{cycle} \times 71.47 \frac{cycles}{hour} = 1543\ st/hour$$

$$\text{Number of units required} = \frac{3521\ st/hour}{1543\ st/shovel\text{-}hour} = 2.28\ shovels$$

The mine model will require that 3 shovels (plus 1 spare) of 15 yd^3 capacity be used.

Using 18 yd^3 shovels, the loading cost for the copper-nickel mine model will be \$.1639/mt or \$.1487/st. If 15 yd^3 shovels were used in place of the 18 yd^3 shovels, it is calculated that 4 shovels would be required and the resulting loading cost would be \$.1656/mt or \$.1502/st.

Individual shovel costs can be broken down as follows:

18 yd^3 capacity	\$/hour	% of Operating Cost
Depreciation	17.47	
Interest, Insurance & Taxes	16.75	
Maintenance	36.68	59
Power	15.20	24
Oil, Filters & Grease	1.05	2
Labor	9.28	15
	\$96.43	100

15 yd^3 capacity	\$/hour	% of Operating Cost
Depreciation	14.14	
Interest, Insurance & Taxes	13.57	
Maintenance	29.69	57
Power	12.40	23
Oil, Filters & Grease	0.90	2
Labor	9.28	18
	\$79.98	100

Haulage Routes--The pit will eventually reach a depth of 900 ft. Half of the pit's volume will have been removed when production has reached a depth of 340 ft below the rim of the pit. The perimeter of the 350 ft level is approximately 11,500 ft. Average haul distance at zero percent grade on this level will be $\frac{11,500}{4}$ = 2875 or about 3000 ft.

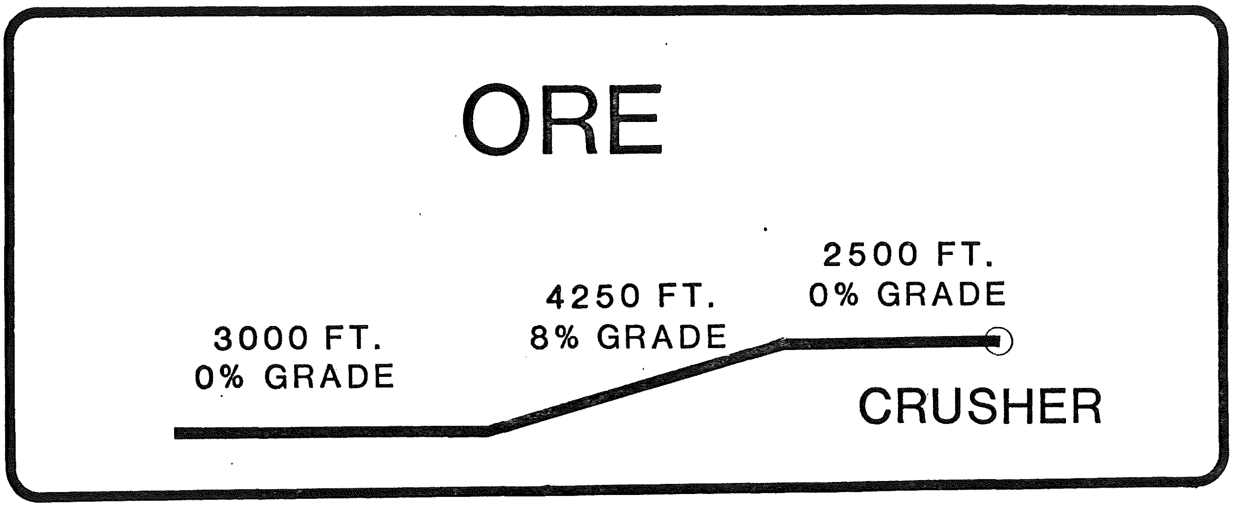
Assuming an 8 percent grade on the haulage road, the trucks will have to travel 4250 ft (340 ft/.08) in order to climb out of the pit, on the average.

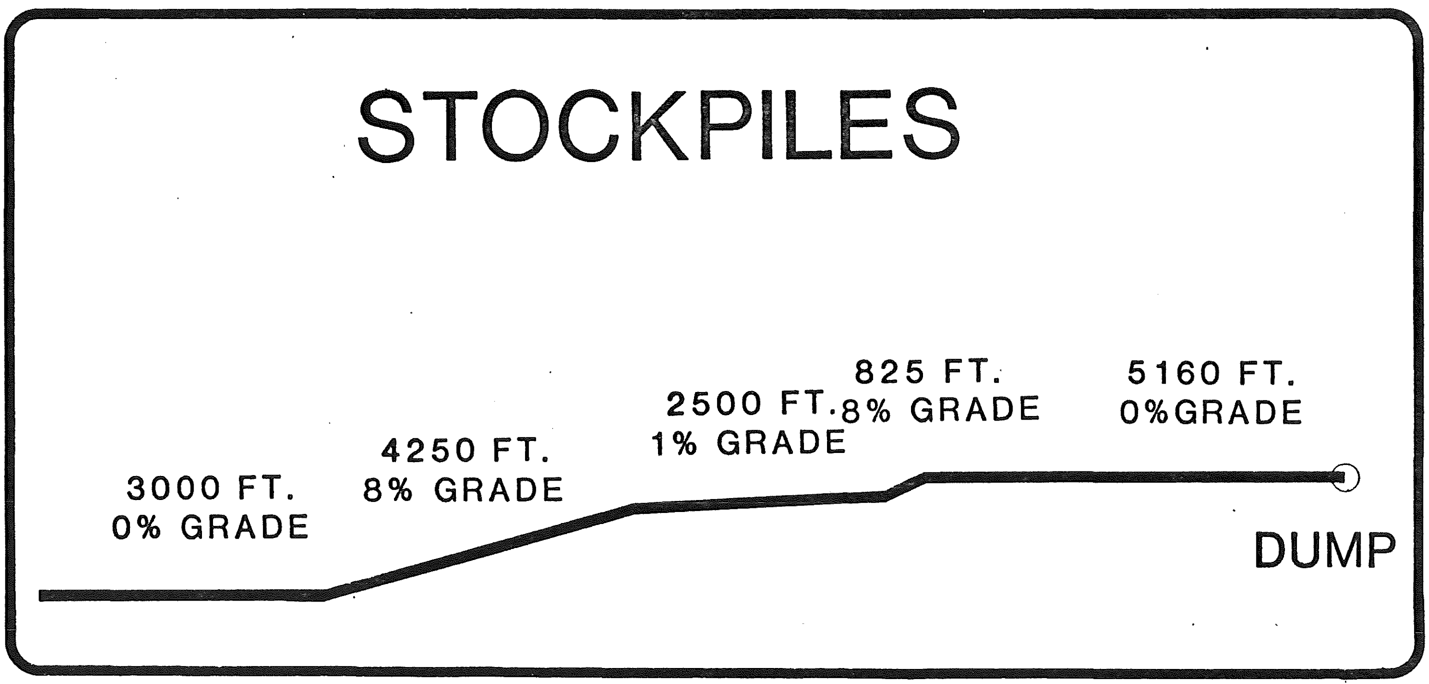
Trucks hauling ore will travel from the rim of the pit to the crusher, as assumed haulage distance of 2500 ft on level ground (zero percent).

From the rim of the pit, trucks hauling waste rock or lean ore will go to the stockpiles, an assumed haulage distance of 2500 ft on a 1 percent grade. To reach the top of the stockpile, a road 825 ft long (66 ft/.08) and at a grade of 8 percent must be travelled. The average haulage distance on top of the stockpiles will be 5160 ft. A schematic diagram of the two haulage routes is shown in Figure 3. Total length of the ore haulage route is 9750 ft and total length of the waste rock/lean ore haulage route is 15,735 ft.

Truck Haulage--The most efficient truck sizes to use with 15 and 18 yd^3 shovels are in the 120 to 170 st capacity range (Buckley and Zimmer,

Figure 3. SCHEMATIC OF HAULAGE ROUTES





Mining Congress Journal, Feb. 1974, pp. 33-34). In order to select the most economic truck-shovel combination, the various truck cycle times and fleet requirements were computed for 120, 150, and 170 st trucks. Ownership and operating costs were then applied and a haulage cost was found for each combination. Then, by adding the loading cost to the haulage cost the most economic match up was found. The results of the cost analysis are summed up below:

	$/mt of copper-nickel ore mined		
Truck Size	120 st	150 st	170 st
Haulage cost with 15 yd^3 shovel	.9560	.9482	1.0036
Loading cost with 15 yd^3 shovel	.1656	.1656	.1656
TOTAL	1.1216	1.1138	1.1692
Haulage cost with 18 yd^3 shovel	.9323	.9193	.9892
Loading cost with 18 yd^3 shovel	.1639	.1639	.1639
TOTAL	1.0962	1.0832	1.1531

	$/st of copper-nickel ore mined		
Truck Size	120 st	150 st	170 st
Haulage cost with 15 yd^3 shovel	.8673	.8604	.9105
Loading cost with 15 yd^3 shovel	.1502	.1502	.1502
TOTAL	1.0175	1.0106	1.0607
Haulage cost with 18 yd^3 shovel	.8458	.8340	.8974
Loading cost with 18 yd^3 shovel	.1487	.1487	.1487
TOTAL	.9945	.9827	1.0461

It can be seen that the economic analysis points to using 150 st trucks and 18 yd^3 shovels for the open pit copper-nickel mine model.

The following calculations are a sample of the calculations that are performed prior to determining the haulage cost for any one shovel-truck combination. The sample is for a 150 st truck.

Number of shovel passes required to fill each truck (see calculations on page 31):

$$\frac{150 \text{ st/truck}}{25.91 \ \frac{\text{st}}{\text{pass}}} = 5.79 \text{ passes or 6 passes per truck}$$

$$\text{Load time} = \frac{36 \text{ sec}}{0.9} = 40 \ \frac{\text{sec}}{\text{pass}} \times 6 \text{ passes} = 240 \text{ sec or 4 min}$$

Spot at the shovel = 0.3 min

Travel Times

Maximum allowable speed in the pit has been determined to be 24 mph.

Trucks from pit to crusher and return:

Road Segment	Length ft	Resistance %	Max Speed mph	Speed Factor	Avg Speed mph	Total Time min
1	3000	5	19.0	0.65	12.4	2.76
2	4250	13	6.5	0.80	5.2	9.30
3	2500	5	19.0	0.70	13.3	2.14
3	2500	5	27.0	0.70	18.9	1.50
2	4250	(-3)	32.0	--	24.0	2.00
1	3000	5	27.0	0.65	17.6	1.94
	19,500 ft					19.64 min

Average velocity = 11.3 mph

Trucks from pit to stockpiles and return:

Road Segment	Length ft	Resistance %	Max Speed mph	Speed Factor	Avg Speed mph	Total Time min
1	3000	5	19.0	0.65	12.4	2.76
2	4250	13	6.7	0.80	5.2	9.30
4	2500	6	15.5	0.80	12.4	2.29
5	825	13	6.5	0.75	4.9	1.92
6	5160	5	19.0	0.80	15.2	3.86
6	5160	5	27.0	0.80	21.6	2.71
5	825	(-3)	32.0	--	24.0	0.39
4	2500	4	28.0	0.80	22.4	1.27
2	4250	(-3)	32.0	--	24.0	2.00
1	3000	5	27.0	0.65	17.6	1.94
	31,470 ft					28.44 min

Average velocity = 12.6 mph

Turn, Spot, and Dump Time = 1.3 min

Truck Cycle Time; ore = 4 + 0.3 + 19.6 + 1.3 = 25.2 min
waste = 4 + 0.3 + 28.4 + 1.3 = 34.0 min

Fleet Requirements

Productivity factor equals 83% (50 min/hour)

Ore

$$\text{trips/hour} = \frac{50 \text{ min/hour}}{25.2 \text{ min/trip}} = 1.98 \text{ trips/hour}$$

st/truck-hour = 1.98 trips/hour x 150 st/trip = 297 st/truck-hour

$$\text{Trucks required for moving ore} = \frac{1531 \text{ st/hour}}{297 \text{ st/truck-hour}} = 5.15 \text{ trucks, use 6}$$

Use 6 trucks for hauling ore, plus have 2 spare trucks available.

Stripping

$$\text{trips/hour} = \frac{50 \text{ min/hour}}{34 \text{ min/trip}} = 1.47 \text{ trips/hour}$$

st/truck-hour = 1.47 trips/hour x 150 st/trip = 221 st/truck-hour

$$\text{Trucks required for moving waste} = \frac{1990 \text{ st/hour}}{221 \text{ st/truck-hour}} = 9.0 \text{ trucks}$$

Use 9 trucks for hauling waste rock and lean ore, plus have 3 spare trucks available.

A total of 20-150 st trucks will be needed.

The haulage cost of a 150 st truck is broken down as follows:

	$/hour	% of Operating Cost
Depreciation	11.90	
Interest, Insurance & Taxes	5.54	
Tires & Tire Repair	18.85	29
Maintenance	21.20	32
Fuel	12.80	19.5
Oil, Filters & Grease	4.27	6.5
Labor	8.47	13
	83.03	100

Auxiliary Equipment--The mine will need many smaller units of equipment to perform various duties around the area. A listing of the auxiliary equipment required by the proposed open pit copper-nickel mine appears in Table 5. The annual cost of all the auxiliary equipment in 1977 dollars is $788,010. Stated in terms of $/ton of copper-nickel ore mined, the cost of auxiliary equipment is $.0695/mt or $.0631/st.

Table 5. Auxiliary equipment required for proposed mine.

Unit	No.	Cost/Unit	Total Cost	Estim. Life years	Total Annual Depreciation	Annual Expense Factor	Annual Expense
Bulldozers	9	$140,000	$1,260,000	5	$252,000	.25	$315,000
Wheel dozers	5	142,000	710,000	5	142,000	.25	177,500
Front end loaders	4	120,000	480,000	8	60,000	.35	81,000
Motor graders	3	90,000	270,000	8	33,750	.25	42,190
Mobile crane	1	250,000	250,000	15	16,670	.20	20,000
Utility crane	4	34,000	136,000	8	17,000	.25	21,250
Water trucks	3	40,000	120,000	8	15,000	.22	18,300
Flat bed trucks	2	42,000	84,000	8	10,500	.22	12,810
Cable reel truck	1	56,000	56,000	8	7,000	.22	8,540
Sand truck	1	30,000	30,000	8	3,750	.25	4,690
Dump truck	1	30,000	30,000	8	3,750	.22	4,580
Welding trucks	3	12,000	36,000	5	7,200	.25	9,000
Electrician's line truck	1	40,000	40,000	8	5,000	.25	6,250
Lube van	1	60,000	60,000	8	7,500	.22	9,150
Maintenance trucks	7	25,000	175,000	8	21,880	.25	27,340
Explosives truck	1	6,000	6,000	5	1,200	.25	1,500
3/4 ton trucks	4	5,500	22,000	5	4,400	.25	5,500
1/2 ton trucks	12	5,000	60,000	5	12,000	.25	15,000
Buses	2	12,500	25,000	8	3,130	.25	3,910
Water pumps	5	3,600	18,000	5	3,600	.25	4,500
			$3,868,000		$627,330		$788,010

BUILDINGS AND SHOPS

The mine will have a centralized shop and a mine office building on the property. The shop should be located near a main traffic route so that both out-of-mine service vehicles and mining equipment can reach the shop conveniently. A location which is as near as possible to the exit from the pit without conflicting with stripping operations, waste piles, and ultimate ore limits would be the most favorable site. As this is generally the same criteria by which a processing plant site is chosen, the two facilities are often located in the same area.

The area in which the processing plant, shops, and offices are located will involve about 100 acres; the shops and offices alone will require about 40 acres. The fuel storage tanks and the electrical substation will also be placed in this area. Space should be available in the event of possible expansion of the plant or shop buildings. Space must be provided for parking equipment before it enters the shop and after it has been repaired. There should also be an area available for storing heavy, bulky materials such as steel plates, beams, and mill liners. Near the offices, space for employee and visitor parking must be provided.

About 180,000 ft^2 of floor space will be required for the buildings containing the shops and offices. Cost of these facilities will be about $12 million. This excludes external utilities and shop tools and equipment.

MANPOWER REQUIREMENTS

A work force of about 580 persons will be necessary for the mine to function at a production rate of 11,333,000 mt/year. Total payroll will be about $11 million. Tables 6 and 7 show the personnel required for the general office and for the various mining positions. The wages used are derived from the February 1, 1977 scale of the United Steel Workers of America.

Table 6. General office manning table.

	No.	Rate	Annual Total $
General Manager	1	58,000	58,000
Manager - Public Relations	1	48,000	48,000
Assistant	1	24,000	24,000
Manager - Environmental Control	1	48,000	48,000
Environmental Engineer	2	24,000	48,000
Technicians	2	12,000	24,000
Manager - Product Sales	1	48,000	48,000
Salesman	2	24,000	48,000
Clerk-Typist	1	12,000	12,000
Comptroller	1	48,000	48,000
Accountant	1	24,000	24,000
Subtotal	14		430,000
Fringe Benefits @ 16%			69,000
TOTAL			499,000

Table 7. Mine manning table.

Administrative Services

	No.	Rate	Annual Total $
General Superintendent	1	48,000	48,000
Assistant General Superintendent	1	40,000	40,000
Administrative Superintendent	1	36,000	36,000
Chief Accountant	1	24,000	24,000
Accountant	3	20,000	60,000
Clerk	3	12,000	36,000
Timekeeper	3	12,000	36,000
Personnel and Labor Relations Supervisor	1	24,000	24,000
Personnel Assistant	1	20,000	20,000
Labor Relations Assistant	1	20,000	20,000
Safety and Security Supervisor	1	24,000	24,000
Safety Officer	1	20,000	20,000
Security Officer	1	20,000	20,000
Guards	8	15,000	120,000
Medical Aid	5	16,000	80,000
Custodians	18	5.815	218,000
Purchasing Supervisor	1	24,000	24,000
Buyer	3	20,000	60,000
Storekeeper	10	12,000	120,000
Clerk-Typist	12	12,000	144,000
Secretary	12	12,000	144,000
Subtotal	88		1,318,000

Table 7. Mine manning table. (contd.)

Technical Services

	No.	Rate	Annual Total $
Chief Engineer	1	36,000	36,000
Chief Mining Engineer	1	30,000	30,000
Mining Engineer	6	20,000	120,000
Geologist	1	20,000	20,000
Surveyor	3	16,000	48,000
Surveyor Helper	6	6.029	75,000
Draftsman	2	15,000	30,000
Chief Chemist	1	30,000	30,000
Chemist	12	6.778	169,000
Sampleman	20	6,029	251,000
Subtotal	53		809,000

Mining Operations

	No.	Rate	Annual Total $
Mining Superintendent	1	36,000	36,000
Mining Foreman	1	30,000	30,000
Assistant Mining Foreman	4	24,000	96,000
Mine Shift Foreman	4	20,000	80,000
Drilling Foreman	1	20,000	20,000
Blasting Foreman	1	20,000	20,000
Loading Foreman	1	20,000	20,000
Hauling and Dumping Foreman	1	20,000	20,000
Driller	12	6.457	(161,000)
Driller Helper	12	6.136	(153,000)
Blaster	4	6.564	(55,000)
Blaster Helper	4	6.029	(50,000)
Shovel Operator	12	7.420	(185,000)
Truck Operator	72	6.778	(1,015,000)

Table 7. Mine manning table. (contd.)

Mining Operations (contd.)

	No.	Rate	Annual Total $
Dozer Operator	40	6.564	546,000
Front End Loader Operator	12	6.778	169,000
Crane Operator	2	6.885	29,000
Grader Operator	16	6.671	222,000
Auxiliary Truck Driver	36	6.457	484,000
Helpers-Oilers	12	6.243	156,000
Secondary Breakage Driller	2	6.350	26,000
Subtotal	250		3,573,000

Mining Maintenance

	No.	Rate	Annual Total $
Maintenance Foreman	1	30,000	30,000
Assistant Maintenance Foreman	1	24,000	24,000
Shift Foreman	4	20,000	80,000
Automotive Foreman	1	20,000	20,000
Machine Shop Foreman	1	20,000	20,000
Electrical Foreman	1	20,000	20,000
Automotive Mechanic	32	7.099	473,000
Automotive Mechanic Helper	24	6.243	312,000
Maintenance Mechanic	16	7.099	236,000
Maintenance Mechanic Helper	20	6.243	260,000
Electrician	12	7.313	183,000
Electrician Helper	12	6.136	153,000
Welder	16	7.313	243,000
Machinist	4	7.313	61,000
Blacksmith	4	7.206	60,000
Plumber-Pipefitter	2	6.992	29,000
Plumber-Pipefitter Helper	2	6.243	26,000

Table 7. Mine manning table. (contd.)

Mining Maintenance (contd.)

	No.	Rate	Annual Total $
Carpenter	2	6.992	29,000
Carpenter Helper	2	6.136	128,000
Crane Operator	4	6.885	57,000
Painter	4	6.778	56,000
Service Truck Driver	4	6.564	55,000
Greaser	4	6.457	54,000
Equipment Cleaner	2	6.136	26,000
Subtotal	175		2,635,000
Salaried Total	108		1,930,000
Fringe Benefits @ 16%			309,000
			2,239,000
Hourly Total	458		6,405,000
Fringe Benefits @ 25%			1,601,000
			8,006,000
TOTAL	566		10,245,000
TOTAL from Table 6	14		499,000
GRAND TOTAL	580		10,744,000

SUMMARY

At the present time, open pit mining of the copper-nickel ores of northeastern Minnesota is a technically feasible mining method. The economic feasibility of open pit mining is something which must be evaluated for a specific mine by the interested mining company, since it involves corporate finances. An economic analysis to determine feasibility must also account, somehow, for the effects of environmental impacts, public and political attitudes and changes in governmental regulations.

REFERENCES CITED

Ash, R.L. The Mechanics of Rock Breakage, Parts I, II, III, IV. Pit and Quarry, 56(2,3,4,5). 1963.

Ash, R.L. Cratering and Its Application in Blasting. Unpublished Report for The Department of Civil and Mineral Engineering. University of Minnesota. July 1970.

Borquez, Guillermo V. Basic Elements in the Economics of an Open Pit Design. Society of Mining Engineers of the A.I.M.E., Preprint No. 77-F-8. 1977.

Buckley, R.E. and P.F. Zimmer. Open Pit Mining--A Review of the Past Five Years. Mining Congress Journal, 60(2). February 1974.

Burton, Alan K. Off-Highway Trucks in the Mining Industry, Parts 1, 2, 3, 4, and 5. Mining Engineering, 27(8,10,11,12); 28(1). 1975-1976.

Caterpillar Performance Handbook, 7th Edition. Caterpillar Tractor Company. October 1976.

Clay, J.A. Copper. Mining Journal Annual Review. June 1976.

Ditter, Donald J., Jr. Copper-Nickel Mining Feasibility Study. Duluth Gabbro Area--Minnesota, Open Pit. Department of Civil and Mineral Engineering. University of Minnesota. June 1974.

Erosion and Sediment Control, Surface Mining in the Eastern United States. United States Environmental Protection Agency. October 1976.

Hays, Ronald M. Environmental, Economic, and Social Impacts of Mining Copper-Nickel in Northeastern Minnesota. Department of Civil and Mineral Engineering. University of Minnesota. August 1974.

Lawver, J.E., R.L. Wiegel, and N.F. Schulz. Mineral Beneficiation Studies and An Economic Evaluation of Minnesota Copper-Nickel Deposit From The Duluth Gabbro. U.S. Department of the Interior, Bureau of Mines. December 1975.

Peurifoy, R.L. Estimating Construction Costs. McGraw-Hill Book Company, Inc. 1953.

Peurifoy, R.L. Construction Planning, Equipment, and Methods. McGraw-Hill Book Company, Inc. 1956.

Production and Cost Estimating Manual. Euclid, Inc. November 1974.

Pugliese, Joseph M. Designing Blast Patterns Using Empirical Formulas. U.S. Department of the Interior, Bureau of Mines. IC 8550. 1972.

Pulver, Harry E. Construction Estimates and Costs, 4th Edition. McGraw-Hill Book Company, Inc. 1969.

SME Mining Engineering Handbook. Given, Ivan A., editor. Society of Mining Engineers of The American Institute of Mining, Metallurgical, and Petroleum Engineers, Inc. 1973.

Surface Mining. Pfleider, Eugene P., editor. A.I.M.E. 1968.