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GUIDELINES FOR OPEN-PIT ORE PASS DESIGN

VOLUME I - FINAL REPORT

**Contract J0205041
Engineers International, Inc.**

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15. Abstract (Limit 200 words) The objective of this investigation was to develop a manual for design, construction, and maintenance of vertical and inclined ore pass systems for use in underground haulage from open pit mines. The work included an extensive information search, novel analyses in rock flow theory, cost and time estimation for shaft and raise construction methods, and development of design criteria for the ore pass proper, dump point, and draw point.				
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FOREWORD

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The work on the project was significantly aided by the contribution of Mr. Tu of the Department of Mining and Fuels Engineering, University of Utah, who provided very useful material on open pit ore pass systems utilized in the Peoples' Republic of China.

In addition to the authors, EI staff who participated in major portions of the project were Messrs. Ben Wheelis and Amitabha Mukherjee, Mining Engineers, Mr. Mark S. Ma, Geotechnical Engineer and Mr. Francis S. Kendorski, Engineering Manager - Mining and Tunneling.

This project was carried out under the technical supervision of Dr. Madan M. Singh who acted as Project Manager, while Mr. Douglas F. Hambley was Project Engineer, and Dr. William G. Pariseau of the University of Utah was subcontractor.

Dr. Pariseau wrote Section 3.0 Rock Flow Parameters in its entirety, which includes original analyses dealing with hangups and piping prevention.

We also wish to thank the many companies and private individuals who furnished information.

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EXECUTIVE SUMMARY

Since most of the minerals production of the world comes from open pit mines, methods of moving it more efficiently must continually be examined. A method that requires less energy consumption is the use of ore passes, since these are dependent upon gravity for moving the materials. Ore passes also provide much needed ore storage capacity at a mine, which serves as a buffer between mine stoppages and ore shipment. Mining engineers that are primarily involved with surface mineral exploitation, however, are not familiar with the design techniques used for these structures. A manual outlining one design approach has been prepared. This report discusses some of the features of the manual.

Prior work in the design and use of ore passes in open pit mines includes the work done by Pfleider and his associates. International Mining Consultants (1978) conducted a survey of such operations under a Bureau of Mines Contract (No. JO275032). Extensive use of open pit ore passes has been made in the Peoples Republic of China (Panzhihua Mines Co., 1979; Li et al, 1980). On the North American continent, large open pit mines that have used ore passes include Carol Lake in Canada and Cananea in Mexico.

Unless ore passes are designed properly a number of problems arise as to their smooth operation. Primary among these are hang-ups, piping (also termed rat-holing or chimneying), wear, and stability. Analyses indicate that hang-ups and piping are primarily dependent upon the relative size of the ore pass and the broken ore fragments. In this report Pariseau has presented original analyses which may be used to calculate pass/ore size ratios. In general, ratios of 5 or larger are quite acceptable. Hang-ups may be caused by wedging or interlocking of boulders or accumulation of fines causing cohesive arch formation. Hence the amount of fines needs to be restricted. Wear is generally not a problem in ore passes, but chute brows should be provided and plate liners may be needed near loading facilities and knuckles (or bends). Uniform wear could be beneficial, since this may smoothen the pass walls and enlarge its size. Ore pass stability is a function of the rock strength and local geology, but does not present a serious difficulty in most situations.

Location of ore passes depends upon haulage distances, ore pass depths, the material to be handled, number of

passes required, location of existing openings, and geological/rock conditions. To a large extent the precise location will be determined by the topography, pit geometry, and operator's preferences. The collars of ore passes within the pit may need to be lowered, reducing storage capacity and possibly disrupting production. If this is not done and the pass is in ore, minable reserves will be reduced. Siting the passes outside the pit limit may enhance haulage costs. The number of ore passes planned depends on storage requirements, desired throughput, waste handling methods, and drawpoint tonnage handling capacity. Haulage distance minimization procedures used could entail nonlinear programming, centroid determination by static moments, or other techniques. Geologically unfavorable zones should be avoided.

Ore pass design aspects that need to be considered include the support system, dump point, draw point, location, and size. The support could be rock bolts with or without shotcrete or cast concrete. The system selected will depend upon the quality of the rock, i.e. its lithology, strength, joint pattern and frequency, joint filling, and in-situ state of stress. The main considerations for dump point design include the number of trucks to be dumped simultaneously, truck configuration, and their dimensions. It is advisable to construct a bumper block to prevent the trucks backing into the ore pass. The type of foundation, liners, and other auxiliaries used will depend upon whether it is intended to lower the collar or not during the life of the ore pass. As with the ore pass proper, the design of the draw point should minimize the possibility of hang-ups and piping. The draw point could have a hopper and feeder, with a longitudinal or transverse chute. Open draw points would be seldom used. Various types of gates may be employed and the feeder should be selected carefully for the required throughput. Some ore passes may have branches, bends, or knuckles. These provide flexibility, but may be difficult to construct and maintain.

Ore passes may be constructed by the various shaft sinking methods currently in use, including conventional sinking, raising and slashing, drop raising, vertical crater retreat (VCR) (which is patented in Canada), and boring.

The method adopted would be the one that is most economical, and depends upon the cross-sectional dimensions, configuration, depth, rock strength, and availability of an access to the bottom of the proposed ore pass. This report

provides general data which could be used to estimate the cost, so that decisions as to the economic feasibility of the scheme can be made. The precise costs are of course, site-specific and need to be calculated for the case in point or obtained from local shaft contractors.

Ancillary facilities that need to be incorporated in any ore pass system include dust control devices (such as sprays, collectors, and air curtains); water handling systems to drain water used to release hang-ups, flooding due to rain, or freezing due to the cold; ore level sensors or inspection openings; and safety measures. The need and nature of these facilities is discussed, in the manual (which forms Volume 2 of this report) but detailed design procedures are not provided. These are readily obtained elsewhere.

This manual also briefly discusses operating procedures - both routine and non-routine. Regular operations require at least one operator at the draw point and either another at the dump, or some remotely operated device to control dumping at the upper end. Systematic maintenance is essential for trouble-free operation, and special attention needs to be paid to junctions with finger raises, knuckles bends, and other direction changes, variations in size or shape, and the brow over the draw point. Major non-routine occurrences include hang-up clearance and lowering of the collar. The techniques and procedures used for these vary significantly from mine to mine, and hence only very general guidelines can be provided.

This report ends with a brief summary of the major considerations discussed in this document.

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ENGINEERS INTERNATIONAL, INC.

1.0 INTRODUCTION

Open pit mining accounts for the majority of the tonnage of minerals mined throughout the world, and in order to be cost effective, very large quantities of materials are handled. Moving materials such as ore or waste involves an expenditure of energy, which in most open pit mines would be either diesel fuel or electricity. As the cost of fuels and energy constantly increases, alternative cheaper haulage methods must be found that expend less energy.

A research study of continuous underground haulage from open pit mines (International Mining Consultants, Inc., 1978) indicated that ore passes, chiefly utilizing gravity as an energy source, could be cost effective in certain cases. However, open pit mine managers expressed concern over the unavailability of design information for estimating or evaluating ore pass installation. Ore passes have traditionally been the province of underground mines and ore pass design has generally been based on experience so that published data are quite site-specific.

Accordingly the objective of this study was to develop a manual for the design, construction, and maintenance of vertical and inclined ore pass systems for use in underground haulage from open pit mines (Figures 1 and 2).

1.1 Scope of Work

The work was divided into five segments. The first segment consisted of an information search comprising a literature search, visits to mines, and contacts with design engineers and other knowledgeable individuals.

The second segment concerned behavior of material within ore passes. The major effort in this section was directed toward establishing criteria for the avoidance of arching and piping based on the theory of flow of bulk solids.

The third segment dealt with criteria for the siting and design of ore passes and was divided into two parts: location criteria and design criteria. Location criteria are based mainly on the ore body geometry, haulage method,

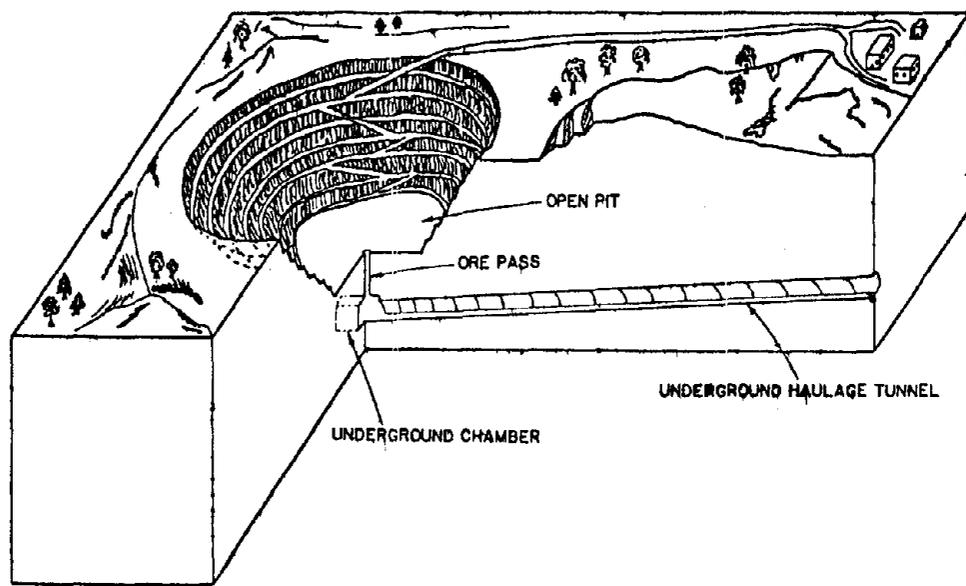


Figure 1 - Open pit with an ore pass and tunnel haulage system (idealized)

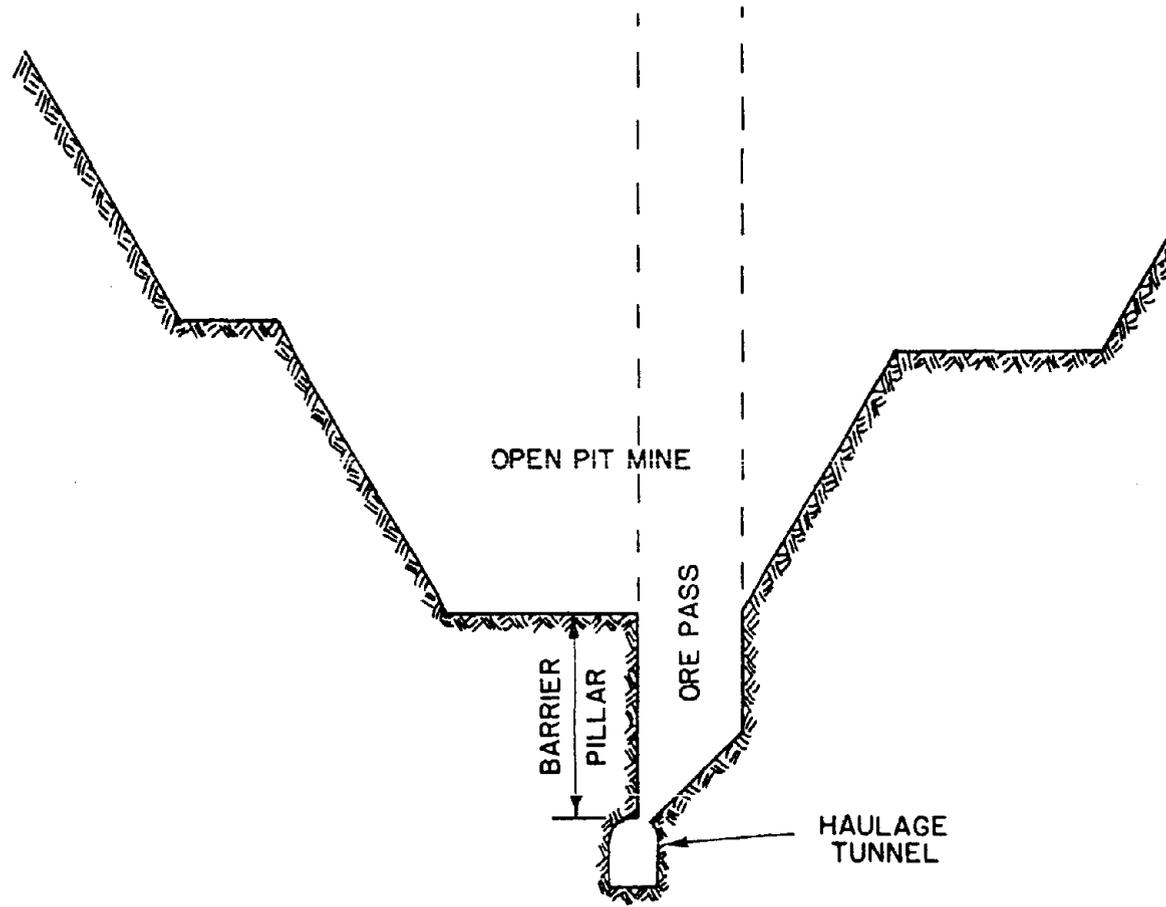


Figure 2 - Section of an open pit mine with an ore pass showing the haulage tunnel and barrier pillar

and the geology of surrounding rock mass. Design criteria are concerned with the physical properties of the muck and wall rock and the tonnage to be handled within a given period of time.

The fourth segment concerned construction methods and costs. This entailed investigating all the various shaft sinking and raising methods, analyzing the work cycle in each case, and estimating advance rates and costs for each method. The construction method chosen will have some bearing on the design of the ore pass chiefly with respect to the type of lining and the shape. Of far greater importance to an operator, however, is the cost and the time required since knowledge of these two considerations will allow him to assess the economic feasibility of an ore pass and determine the most cost- and time-effective method. In order to make such decisions, the planner needs to be provided with realistic cost and advance rate information. The scarcity of this information necessitated performing detailed cost and advance rate estimates for shafts and raises over the complete range of possible sizes.

The fifth, and final, segment comprised preparation of the design manual.

1.2 Prior Work

The concept of using underground haulage from open pit mines is not new - glory-hole mining, which consists of breaking rock in an open cut into raises connected to an underground haulage system, was a common method in the first half of this century. Ore passes have been used in pit mines as long ago as 1930 (Alenius, 1930).

The first, and only, large (production over 100,000 tpd) mining operation to use ore passes is the Carol Lake, Labrador operations of the Iron Ore Company of Canada Limited. Significant discussions on ore pass design (Pfleider and Dufresne, 1961; Selleck and Pflëider, 1968) are concerned with the design of the first ore pass used at this operation.

International Mining Consultants, Inc. (1978), under Bureau of Mines Contract No. JO275032, studied continuous haulage from open pit mines, surveyed a number of open pit mines where ore passes were used, and described their characteristics in some detail.

Mines in the Peoples' Republic of China have made extensive use of ore passes in open pit mines, especially those located in rugged or mountainous areas (Panzhihua Mines Co., 1979). Based on this experience, they have developed a number of design guidelines (Li et al, 1980).

1.3 Open Pit Mines Suitable for Ore Pass Haulage

The relative attractiveness of the ore pass haulage option in an open pit mine depends on the configuration and physical setting of the orebody. International Mining Consultants, Inc., in their report (International Mining Consultants, 1978), identified several types of orebodies where ore passes would be an attractive alternative to truck haulage, including:

- narrow, laterally constrained orebody
- orebody with extensive depth
- orebody with an elevation disadvantage relative to the processing plant

Ore passes might also be attractive in other cases, for example mines where diesel fuel and electricity are expensive or in short supply.

2.0 INFORMATION SEARCH

The information search segment of the project involved collecting information on ore pass systems, design practices, rock flow parameters, construction methods, and operating problems and remedies. The information was collected by performing a literature search, visiting mines, and contacting knowledgeable individuals.

2.1 Literature Search

In the course of the literature search, all pertinent sources including mining and tunneling journals, handbooks, technical conference proceedings, and society transactions were examined. Extensive reference material was obtained from the Engineers International, Inc. library and other libraries in the Chicago area. Other valuable sources were the personal files of the authors, the University of Utah library, and mining industry colleagues and contacts.

In Section 1.2 of this report reference was made to the pioneering ore pass design work carried out by the Carol Project of the Iron Ore Company of Canada Ltd. in conjunction with the University of Minnesota. The several articles (Pfleider and Dufresne, 1961; Selleck and Pfleider, 1968; Pfleider, 1973) which have been written based on those efforts constitute the benchmark papers in open pit ore pass design. These articles are similar in content, the later ones being merely updated versions of Pfleider and Dufresne (1961).

Several months after the commencement of the project, copies of a paper (Li et al, 1980) and a report (Panzhuhua Mines Co., 1979) were obtained which documented the design practices employed at the more than 30 open pit mines in the Peoples Republic of China where ore pass and tunnel systems are used. There are significant differences in scale and economics between their open pit mines and those in North America; however, the design principles which they elucidate, were very similar to those based on North American and European sources.

On the subject of gravity flow of broken rock, two works are of interest. These are Storage and Flow of Bulk Solids (Jenike, 1964) and Bins and Bunkers for Handling Bulk Materials (Reisner and Eisenhart-Rothe, 1971). In this present report, the theory contained in the aforementioned publications has been extended by Pariseau.

The AIME Mine Plant volume (Tillson, 1938) provided a large number of chute layout examples from which the principles of chute design were discerned.

A large number of mining case histories were consulted from which data bases were compiled for ore pass systems (in both open pit and underground mines), and shaft and raise excavation techniques and advance rates.

2.2 Visits to Mines

Project personnel visited 8 mining companies as part of the information search. At each, detailed information was collected on the design of the ore passes, type of haulage (upstream as well as down stream of the ore pass), operating conditions, and difficulties.

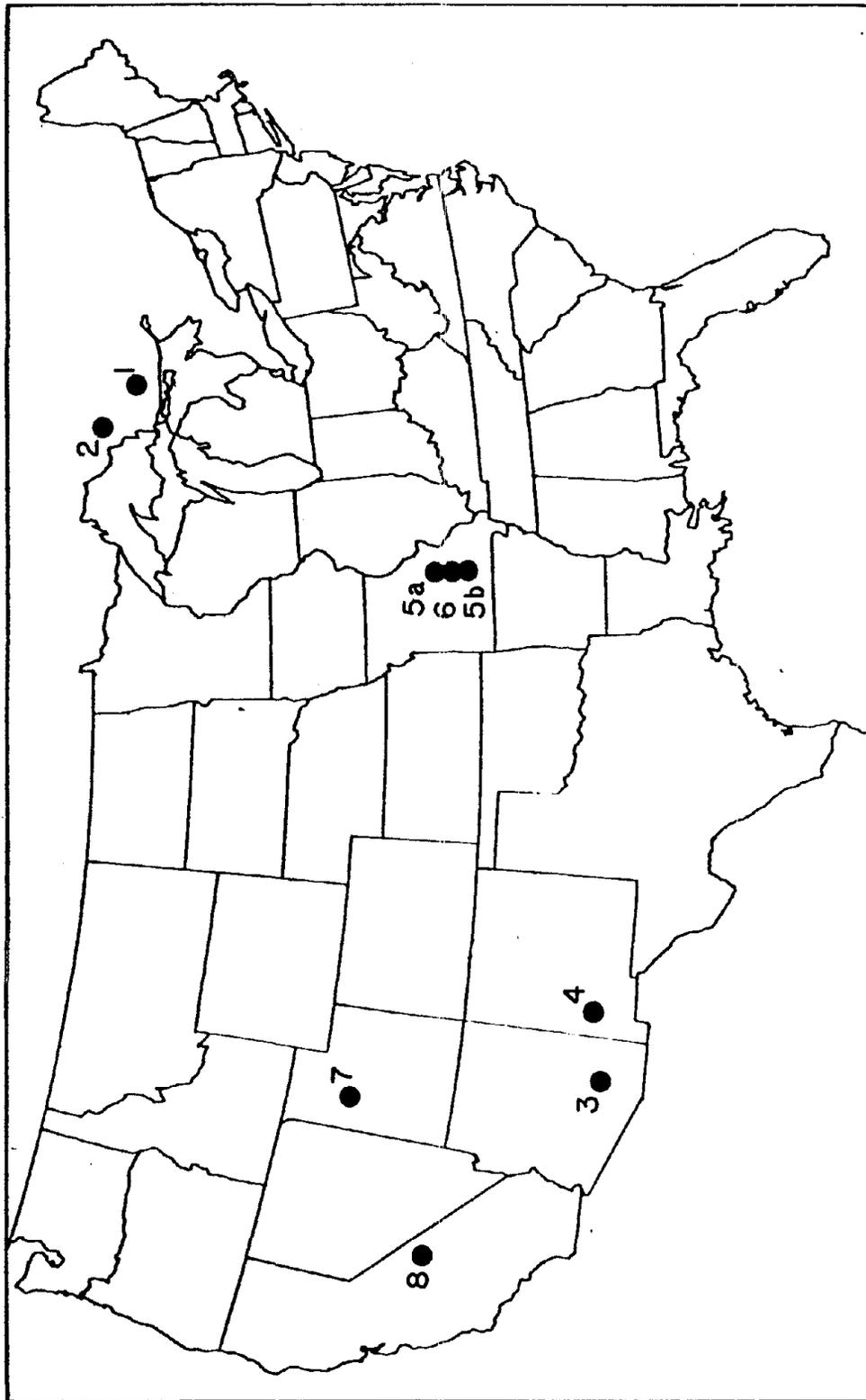
2.2.1 Selection Process

In order to ensure that the mines visited were representative, suitable, and cooperative a pre-selection process was employed. The selection criteria employed were

- open pit mines with ore passes (underground mines for remainder)
- large tonnage underground operations (> 4,000 tpd)
- wide variety of mining methods represented
- broad geographic sampling but try to keep two to three mines close together to minimize travel costs
- mines to be within the North American Continent
- ore passes in underground mines be relatively long (> 200 ft)

It was decided to visit two mines in Ontario, 2 mining organizations in Missouri, and one mine in each of Arizona, California, New Mexico and Utah (Figure 3).

Another factor which influenced the final selection of mines to visit was the fact that the copper industry in the Southwest was on strike at the time visits were to be made - thus some mines that were otherwise cooperative and suitable felt that a visit during the strike was not feasible.



● LOCATION OF MINE VISIT

Figure 3 - Location of mines visited

At each mine, a day was spent collecting information and discussing ore passes with personnel. Names or other identifying information concerning the mining companies will not be disclosed in this report, since some of the information obtained is confidential.

2.2.2 Mine 1

This mine which was visited in August 1980 is located in Ontario, Canada. The ore consists of uraniferous quartz pebble meta-conglomerate reefs which are interbedded with quartzite. The dip of the reefs is generally between 10 and 30 degrees.

The ore is mined by room-and-pillar methods using trackless equipment. Main haulage to the shafts is by a network of run-of-mine ore conveyors located in the footwall. A number of ore passes connect the conveyors with the mine workings. The dump point at each ore pass is equipped with a grizzly and hydraulic rock breaker.

The ore passes vary in length from 30 ft to over 500 ft. One particular ore pass, 525 ft long and inclined at 57° with the horizontal, was chosen for study as a hang-up of more than 60 ft in length had occurred near the end point of the ore pass which had taken 14 months to remove. A number of methods were tried including diamond drill Airtrak and down-the-hole drillholes, and an aluminum buggy mounted on bicycle wheels. In the down-the-hole drillholes, a special blast was designed and supervised by an explosives company's engineers. This blast only served to compact the hang-up and was later found (when the hang-up was finally dislodged) to have created a three-foot diameter, two-foot deep crater in one wall of the ore pass.

Eventually they drove a vertical raise using an Alimak raise climber and then sub-drifted over into the bottom of the hang-up. They installed a high pressure air and water spray and dislodged 30 feet of hang-up by undermining the material on the footwall. At this point the air and water blast became ineffective, so they drove a 50 foot inclined open raise from the vertical Alimak raise and subdrifted over to the hang-up again. They reinstalled the air and water blast and this time, freed the remainder of the hang-up. The Alimak raise was timbered and equipped with ladders and air and water lines in order to be ready should another such hang-up occur.

Other methods which were suggested but not tried were a howitzer or field piece and a "kamikaze monkey". The location of the hang-up was originally determined by sending a helium balloon up the ore pass. The hang-up consisted of chunks of ore contained within a matrix of wet fines and was eventually so well cemented together that when they subdrifted over, they were into the hang-up without realizing it.

It was felt by mine personnel that the angle of inclination of 57° was too flat for it was also stated that cohesive hang-ups were a problem every time material was allowed to accumulate in that ore pass such as during a weekend shutdown.

The mine also has several ventilation raises which are between 15 and 30 ft diameter and, from 600 to 2,800 ft long. These raises have been driven by raising followed by slashing. For the pilot raises both raise borers and Alimak have been used.

2.2.3 Mine 2

This mine which is located in Ontario, Canada was visited in August 1980. The particular reason for visiting this mine was the fact that they used to operate an open pit with an ore pass.

The ore mined consists of siderite contained within the Iron Formation and underlain by meta-diorite and andesite. Mining in the presently-operated underground mine is carried out by open stoping and room-and-pillar methods, at a rate of 8,000 long tons per day.

The collar of the open pit ore pass was lowered with each 65 ft bench. The method of lowering consisted of filling the ore pass to near the top and blasting the area around the ore pass using the ore pass as a slot. Extra blast holes were drilled around the ore pass to ensure good fragmentation. Once the area was mucked out, the bumper block, which consisted of a large (>18 inch diameter) cedar log was secured with pins to the floor of the bench.

One interesting feature of the underground system is the fact that the chutes are supported on horizontal beams so that a loader can get in underneath to clean up spills. (The chutes are located in the center of the haulageway rather than on one side as is commonly practiced elsewhere).

2.2.4 Mine 3

Mine 3, which is situated in southern Arizona, was visited in September 1980. This operation, which was on strike at the time of the visit, is a large block caving operation.

The ore body consists of a vertical stock in granite which is mineralized with chalcopyrite. The ore is caved down to an undercut level and is transferred to the grizzly level through a draw raise. After being reduced in size (where necessary) to pass a grizzly having 14-inch openings, the ore then falls down a transfer raise to the haulage level whence it is trammed to a production shaft. Haulage drifts are concreted except at the chute stands where they are timbered. Each chute stand contains two chutes - one on each side of the drift and an operator's platform to one side of the chutes.

Operating suggestions learned at this mine concerned the guillotine chute gates, and the replaceable clip-in 90-lb rail used to line the ore pocket. Another point that was emphasized was that much less wear occurs on the walls if the ore pass is kept full. The mine is converting from using men with 16-lb hammers to using mobile rockbreakers for breaking oversize at the grizzlies.

2.2.5 Mine 4

Mine 4 which was visited in September 1980 is located in an old mining district in western New Mexico. Mining is carried out by both open pit and underground methods. However, there is no connection between the open pit and underground mine although shallow old stopes mined by previous owners of the property are, from time to time, intersected by the open pit.

The ore body is a chalcopyrite replacement deposit contained within a hornfels and garnet country rock which is overlain by quartzites and shales. The ore body dips at 20° and has a 90 ft thickness. Mining methods in the underground mine are room-and-pillar and open stoping.

The main points learned from this visit concerned wear in the ore pass. The major wear was in the first 32 ft below the offset point (direction change) at an intermediate level. Significant wear has also occurred where the cross-section of the ore pass expands from circular to a D-shaped, horseshoe cross-section.

The lining consists of steel and concrete - the original lining consisted of rails bolted to steel sets which were embedded in concrete. When this lining wore out, 1/2-inch steel liner plates were installed. These liner plates wore out after about 3 to 4 years and in 1980 another steel liner consisting of 1/4-inch steel plate was added. It was felt by mine staff that steel rail provides a superior lining to steel plate. The ore pass is generally kept empty.

2.2.6 Mines 5a and 5b

Mines 5a and 5b, which were visited in October 1980, are located in southeastern Missouri. They are listed together since they are both owned by the same mining company and constituted one visit.

2.2.6.1 Mine 5a

The ore body consists of a fine rhyolite porphyry containing 1/16 inch diameter phenocrysts. The ore is magnetite and is mined by several methods including open stoping with retreating pillar recovery, sublevel open stoping and sublevel caving. Both ore and waste are hard and abrasive and the ore tends to be chunky.

Wear in the ore pass occurs mainly at branch raise entries. There was also a triangular trench, 8 ft wide and 6 ft deep in the middle, worn out of the brow over the crusher. This was repaired with poured concrete, while wear elsewhere is normally repaired with rock bolts and gunite.

Also of interest is the fact that they plan to install sonic level indicators in the pocket at the bottom of the ore pass.

2.2.6.2 Mine 5b

The orebody is located in the so-called Viburnum Trend (New Lead Belt). The ore comprises galena, sphalerite, and chalcopyrite and is mostly confined to a barrier reef-like zone in the Upper Cambrian Bonnetterre Formation (brown dolomite). Mining is carried on using the room-and-pillar mining method.

The dump consists of a 7 ft by 5 ft concreted steel grizzly, the steel bars of which are replaced every 3 to 4 years. The ore pass itself is unlined and wear on it is not severe although there is a knuckle near the top.

2.2.7 Mine 6

This mine which was visited in October 1980, is located in southeast Missouri in the New Lead Belt.

The ore body is a flat lying stratabound deposit contained within the dolomite of the Bonneterre Formation. The ore consists of galena, sphalerite and chalcopryrite and is mined by room-and-pillar methods.

Of interest at this mine is the fact that some of the ore passes are excavated by the drop raise technique using an Airtrak machine to drill the holes.

The main difficulties encountered at this operation are related to overfilling of muck cars resulting in the cars being jammed at the chute and to derailed cars hitting parts of the chute and chute supports. Damage to the chute as a result of blasting hang-ups in the chute is another common problem.

2.2.8 Mine 7

This mine, which was visited in October 1980, is located near Salt Lake City, UT. It is a new mine which began producing in 1980 after several years of development, although certain zones of the orebody have been mined by others since late in the last century.

The ore which consists of chalcopryrite is found in garnetite limestone. The hanging wall consists of quartzite while the footwall is limestone. The mining method is Vertical Crater Retreat (VCR) open stoping. Broken ore contains about 40% fines and is very sticky - it sticks even to quite small surfaces such as the drop rail in the Swedish dumps. As a result of this stickiness, cohesive hang-ups occur often and problems occur if broken ore remains in the ore pass for even a few hours. A hang-up in the main ore pass occurred 180 ft above the haulage level which took three weeks to dislodge. It was finally removed by drilling down from the level above and blasting. Some thought was given to using a "kamikaze monkey" to plant explosives. This suggestion was not really taken seriously but reveals the desperate situation which results when the main ore pass is plugged for an extended period of time.

This was the only mine visited which had an engineer on staff who was familiar with the theory of flow of material in ore passes. The engineer in question was trained in

eastern Europe where engineers use a more theoretical approach to mining engineering than is common in North America. This individual mentioned that in Europe a "Mine Mortar" is used to fire explosives into the bottom of a hang-up.

This operation was also the only mine visited which had carried out a detailed investigation of blast fragmentation and muck size distribution. This practice is highly recommended.

2.2.9 Mine 8

This mine which was visited in October 1980, is located in the Sierras in California about 200 miles south of Reno, NV. The orebody, which is reached via a 2-mile long rail adit into a mountain, is unusual in that all the workings are above adit level (which is at 8,000 ft above sea level). The mine has been worked continuously since 1938 and intermittently before that from 1916.

The ore consists of scheelite contained within granite and quartz monzonite porphyry. The mining methods are sublevel caving, open stoping and shrinkage stoping. The topmost level of the mine is at an elevation of 11,600 ft.

The mine has a large number of ore passes, the longest of which is 2400 ft long. The mine has large water inflows and the ore passes are no exception. The chutes at the haulage adit have a steady flow of 6 to 8 gpm coming out which can be up to 10-12 gpm at times. The mine as a whole makes 6,000 gpm and has made up to 9,000 gpm of water. With all this water, hang-ups from cohesive arching and rushes of waterborne muck are very real problems.

Chute gates are replaced annually and the chute frameworks last about 10 years. The USBM Hang-up Clearance Module was tested at this mine and was successful in bringing down hang-ups from distances up to 50 ft away (Bureau of Mines, 1981).

Raises up to 1,200 ft in length have been bored at this mine using a Raise Borer.

2.3 Contacts With Engineers

To supplement the information obtained from the literature search and mine visits, telephone conversations were held with a number of individuals conversant with ore

passes and the problems associated therewith, including planning engineers at Cananea in Mexico and Carol Lake in Labrador.

2.3.1 Cananea, Mexico

This mine was contacted in order to obtain specific detailed information on operating parameters related to ore passes in open pit mines, including the procedure for lowering the dump point and the type of bumper block used at the dump point.

It was learned that a 3-ft high steel and concrete bumper block was found satisfactory with 120-ton trucks. Bumper blocks consisting of 12-in. x 12-in. timbers had been tried but were not adequate - 2 trucks had gone into the ore pass (Moreno, 1981). The configuration of the dump point was such that two trucks could dump at once.

The dump point was lowered bench-by-bench, though occasionally with every second bench, until the dump was about 100 ft above the crusher. The pit bottom when operations ceased in December 1980 was about 250 ft below the elevation of the dump point and for the last years of the operation trucks hauled up to the ore pass. In addition, a pillar about 170 ft wide was left between the pit wall and the crusher chamber.

The ore pass was unlined and with time was eroded to about 40 ft square from the original 20 ft square cross-section. The maximum block size was about 54 in. with most of the muck being about 24 in. Hang-ups were not much of a problem nor was wear on the knuckle as the level of muck was kept above it using radioactive level indicators.

The procedure for lowering the dump point was as follows:

- remove the steel portion of the bumper block
- drill the normal blasthole pattern around the ore pass but without any subgrade drilling
- blast muck with controlled blasting techniques using the ore pass as a slot.

The concrete portion of the dump point was allowed to go down the ore pass and was scalped at the crusher. This was not considered to be a problem.

Ore passes are not being used in the Kino and Colorado pits now in operation. Long range plans have provision for twin steel-lined ore passes which would feed gyratory crushers. However, they will not be implemented for another 20 to 30 years.

2.3.2 Carol Lake, Labrador

These operations of the Iron Ore Company of Canada, Ltd. were contacted since a number of years had passed since articles concerning this operation had appeared in the literature. It was especially desired that information be obtained concerning the dump points and the lowering of same.

The annual production of 55 million long tons (155,000 tpd) is obtained from 4 pits - Carol East, Carol West, Smallwood and Humphrey. The largest blocks have 6 ft sides but most of the muck is between 2 and 4 ft³ in size.

The bumper blocks at the ore passes (three are in use) are 3 to 4 ft high steel pads which have been found sufficient to prevent the 120- and 170- ton trucks in use from falling into the openings (Garg, 1981).

The dump points of the ore passes have not been lowered as mining progresses downwards. In fact, the most recent ore pass which is 57 ft in diameter was constructed outside the mining area in order to make it unnecessary to lower the dump point. The reluctance to reduce the lengths of the ore passes is caused by the desirability of retaining storage capacity which totals 82,000 long tons in the three ore passes. (This storage capacity represents about 1 1/2 shifts production).

The original ore pass which is described in Selleck and Pfleider (1968) is still in use after 17 years.

The ore passes were driven by the raise and slash method and are unlined. The dump points are single-truck facilities i.e. only one truck can dump at a time. The two older ore passes incorporate doglegs at the bottom while the newest one is strictly vertical. Neither wear nor hang-ups have constituted a problem in any of the ore passes.

2.3.3 Other Contacts

Contacts were made with engineering personnel at several large Canadian mining organizations to obtain the benefit of their experience with ore passes in widely varying ground conditions. It was learned that, at one mine under development, tensioned cable bolts were used to stabilize the main ore pass in the location where it traversed a fault zone. Mention was made of a published paper concerned with ore pass stabilization (Singh, 1973).

One individual contacted who has extensive experience as a mine inspector furnished information on the problems encountered in shafts and ore passes at a wide variety of mine operations. He affirmed that arching of wet fines is commonly a severe problem in ore passes and mentioned several examples. He also mentioned difficulties due to faulting and other adverse geology including a shaft at a gold mine which intersected a fault which changed direction - the timbers in the fault zone had to be adjusted on a weekly basis (Banasuik, 1980).

In general, the information supplied by these individuals corroborated the data which had been collected by other means. The frequent reference to cohesive arching and problems related to geology served to emphasize their importance.

2.4 Summary of Information Search

Data concerning the ore passes at the mines which were visited are given in Table 1. As can be seen, ore pass inclinations exceed 55° from the horizontal with angles commonly between 55 and 75 degrees. This is consistent with normal practice underground. Hang-ups from arching of oversize blocks is more common than cohesive arching of fines; however, the latter causes more severe difficulties and results in more downtime than the former.

Wear is not normally a problem unless weak zones or fault zones are encountered or unless the wall rock is less abrasion resistant than the material being handled. Wear can actually enhance rock flow since it results in a larger opening.

Ore pass cross-sections are either circular or square. The methods employed to construct the ore passes which were studied included all shaft sinking and raising methods except blind shaft boring. At the mines which were visited, conventional raising and Alimak were the most commonly employed methods with the latter being used for the longer openings.

TABLE 1 - Mine Ore Pass Data From Visits

Ore Pass Attribute	1	2a	2b	3	4	5a	5b	6	7	8
Length (ft)	525	280	771	68	300	150-200	40	20-100+	770	up to 2400
Dimensions	7 ft x 7 ft	11 ft x 11 ft	8 ft x 8 ft	4 ft x 4 ft	8 ft dia.	9 ft dia.	7 ft x 5 ft	6 ft x 6 ft	6 ft-8 ft dia	5 ft dia 7 ft x 7 ft
Lining	-	-	-	armored cribbing	steel & concrete	none or gunite	-	-	-	-
Inclination	50°	70°	65°	62.5°	90°	70°	90°	90°	73°	55°
Frequency of hangups	often	occasional	occasional	often	infrequent	occasional	infrequent	often	often	often
Causes of hangups	cohesive arch	block arch	block arch	block arch, packing	packing	block arch, packing	block arch	block arch	cohesive arch	cohesive arch
Wear	slight	heavy*	heavy	medium	heavy	medium	medium	slight	slight	medium
Excessive Difficulties from hangups	Yes	No	No	No	No	No	No	No	Yes	Yes
Max particle size (underlined indicates that grizzled)	<u>24 in.</u>	not known	<u>12 in.</u>	<u>14 in.</u>	<u>17 in.</u>	<u>36 in.</u>	<u>24 in.</u>	<u>24 in.</u>	<u>30 in.</u>	<u>24 in.</u>
Driving method	Alimak	CR	Alimak	CR	SS	R&S	CR	DR, CR	RB	CR, RB
Haulage Method to Ore Pass	LHD	truck	LHD	slusher	rail	LHD, rail	truck LHD	truck, LHD	LHD rail	LHD

*On knuckles when in Basement Rock

Block arch	= block arching	CR	= conventional raise	LHD	= load-haul-dump
cohesive arch	= cohesive arching	RB	= Raise Borer		
packing	= packing of initial material dumped	R&S	= Raise & slash		
		DR	= drop raise		
		SS	= shaft sinking		

3.0 ROCK FLOW IN ORE PASS SYSTEMS

Ore passes are simply vertical or steeply inclined tunnels. Their primary purpose is to provide a conduit for transporting broken ore and waste rock from one level in a mine to a lower level. Gravity is the driving force and consequently the cost of operation is relatively low and somewhat inflation proof, although the capital cost associated with excavation is relatively high. Ore passes also provide, without additional cost, storage capacity which is of considerable importance to some operations. However, the potential advantages of an ore pass system over alternative materials handling approaches cannot be realized even under circumstances favoring its adoption if the broken rock in the system does not flow when desired because of "hang-ups" or if the flow is uncontrolled because of "flooding" at the outlet. Proper functioning of an ore pass system should be assured for the operational life of the system, so that wear is also a major design consideration.

3.1 Ore Pass Geometries

Ore passes like tunnels are generally long compared to their cross-sectional dimensions. The usual shapes are rectangular, square, and circular. Circular ore passes are becoming more popular with the advent of raise boring machines and the ability of such machines to bore stronger rock. Cross-sectional areas vary considerably in underground mines but seldom exceed 50 ft² in area. However, much larger ore passes are used in conjunction with open pit haulage systems. Circular ore passes over 30 ft in diameter and 600 ft in length have been mentioned (Pfleider and Dufresne, 1961). A vertical 20 ft by 20 ft square ore pass and an inclined 12 by 8 ft rectangular one, the latter over 1100 ft long, have been mentioned in addition to a number of large vertical circular ore passes in a recent review of open pit ore pass systems in China (Li and others, 1980). Thus although ore passes used in open pit mine haulage systems are much larger than conventional underground mine ore passes, lengths are much greater than cross-section dimensions in both cases.

Of course, the larger ore passes in open pit mines are dictated by the much larger volumes of material handled but more importantly by the much larger particle sizes produced in open pit mine blasting in comparison with underground mine volumes and particle sizes.

However, it is important to note that if d is a characteristic particle dimension handled in an ore pass and D is a typical cross section dimension of the ore pass, the ratio D/d is very nearly the same in open pit and underground mine ore passes. A ratio of 5 or so is typical. This feature of ore pass geometry has important consequences for design.

3.2 Prevention of Interlocking Arches

Interlocking arch formation occurs as a result of large size boulders becoming wedged together mechanically to form an obstruction to flow. Interlocking arches are chance occurrences of stable arrangements of the relatively few large size fragments of rock in the ore pass. Their probability of occurrence depends in some way on the percentage of large fragments in the material handled, on the size of the particles relative to the size of the ore pass and outlet, on the shape of the rock fragments, on the velocity profile across the flowing ore, and so forth. However, a probabilistic approach to the description of interlocking or particle arch formation has not been developed.

Empirical rules based on the ratio of ore pass diameter to particle dimension D/d or outlet dimensions to particle dimension D_o/d are therefore used instead. Such rules are based on a variety of sand model type of experiments in the laboratory and on field experience. Peele (1941) cites as rule-of-thumb that clogging is prevented by maintaining an outlet width three times greater than the maximum diameter of the largest particle to be handled in the ore pass. Jenike (1961) has developed a chart in which the outlet dimension ranges from 3 to 6 times the intermediate dimension of the fragments for a reasonably uniform material. Aytaman (1960) indicates that an outlet width of 4.21 times the largest mesh size of particle will insure free functioning of the outlet with respect to interlocking. Zenz and Othmer (1960) in summarizing the work of a number of investigators, indicate that an outlet to particle diameter ratio between 4 and 6 will insure a no-interlocking condition. The work of Kvapil (1965) indicates that a ratio of 5 will suffice in most cases. Idealized experiments by Pariseau (1964) indicate that a 3.6 to 4.5 ratio of diameters is sufficient to insure free flow. More recently, Li et al. (1980) have applied a ratio of ore pass diameter to maximum particle size of 6 to insure against interlocking arch formation. They also state that if the ore pass is not "fully packed," that is heavily compacted, then a ratio of 4 to 5 will suffice.

Generally, the outlet to an ore pass is restricted and of smaller size than the ore pass proper for practical operational considerations, although feed arrangements that draw over the entire ore pass cross section are possible. A distinction between ore pass dimension and outlet dimension, however, is usually necessary. In addition, it seems prudent to recognize, as a number of investigators have, a range of behavior from almost certain interlocking to almost certain flow depending on the ratio of ore pass or outlet dimension to particle dimension and the many other variables such as particle shape and feeder arrangement that may influence the likelihood of interlocking arch formation. Table 2 summarizes the empirical evidence for design against interlocking arch formation in terms of the ratio of ore pass dimension (D) or outlet dimension (D_o) and the frequency of arch formation. Table 2 provides design guidance with respect to the prevention of interlocking arches. In order to use the information in this table, one must decide which ore pass or outlet dimension and which particle dimension should be used to form the ratio D/d or D_o/d . In the case of circular ore passes and outlets handling roughly equi-dimensional rock fragments, the situation is unambiguous. In the case of rectangular or square ore passes and outlets, arching is apt to occur across the smallest dimension. However, such an arch may not stop flow over the entire area of the ore pass or outlet. In fact, slot outlets are considered to be more active than circular outlets because arching can only occur in one direction with slot outlets while two mutually perpendicular directions are possible over circular outlets. As a practical matter, the dimensions of rectangular ore passes are unlikely to differ so much as to be considered slots, so that the least dimension of a rectangular ore pass or outlet should be used as D or D_o in Table 2. This is a slightly conservative approach.

In the case of rock fragments that are slab shaped rather than equi-dimensional, a very conservative approach would be to use the largest dimension of the fragments as d in Table 2. Li et al, (1980) take this approach. Jenike (1961) recommends the use of the intermediate dimension of slab shaped particles on the basis of an argument concerning the velocity profile across the ore pass. According to Jenike (1961) the higher velocity in the center of an ore pass will cause a slab-like particle to rotate so as to bring its long dimension into parallelism with the direction of flow. The particle will thus slide through the outlet provided its intermediate dimension is not too large relative to the outlet width. The argument while plausible lacks substantial experimental justification. Indeed, the necessary rotation of the particle may not occur before

TABLE 2 - Interlocking Arch Formation

<u>Ratio of Ore Pass or Outlet Dimension to Particle Dimension</u>	<u>Relative Frequency of Interlocking</u>
D/d or $D_o/d > 5$	very low, almost certain flow
$5 > (D/d$ or $D_o/d) > 3$	often, flow uncertain
D/d or $D_o/d < 3$	very high, almost certain no-flow

D = ore pass diameter

D_o = draw point (outlet) diameter

d = maximum size of muck

arrival at the outlet in which case arching becomes highly probable. Careful consideration must, therefore, be given to the costs and benefits associated with the prevention of interlocking when slab shaped rock fragments are involved. In this regard, large slabs tend to be relatively strong; this is why they are above average in size. Had they contained weakening defects such as cracks and fissures, breakage would likely have occurred along such defects during blasting. Thus, comminution of large size fragments during transit in the ore pass cannot be relied upon as a mechanism for significantly reducing the particle size d presented to the outlet.

In characterizing particle size by the dimension d , one must consider the distribution of particle sizes as well as the shape of particles. The simplest approach is to simply use the characteristic dimension of the largest rock fragment in the material being handled. If interlocking of large particles is unlikely, then interlocking of smaller particles is even less likely, so that the use of d of the largest particle is a conservative approach. Most rock after blasting shows a natural distribution of particle sizes that ranges from very few very large particles to very numerous very small particles. The number distribution by size is, of course, quite different from the distribution of size fractions by weight. It is the number of large particles and their frequency of appearance at the outlet that largely determine the probability of interlocking. Thus even if interlocking of particles of size d is prevented, there is no assurance the interlocking of larger size particles will not occur when handling ordinary blasted rock.

The use of a grizzly, however, provides a measure of control over the largest size particle entering an ore pass. A grizzly is a very coarse screen positioned at the top of an ore pass or wherever broken rock enters the ore pass. Grizzlies at dump points also provide a measure of the size of rock fragments being handled. Rock fragments in the material delivered to the ore pass that are too large to pass the grizzly are broken at the grizzly by such means as impact breakers. The more closely spaced are the grizzly bars the greater will be the amount of secondary breakage, but interlocking in the ore pass and at the ore pass outlet will be correspondingly less. Thus, the spacing of the grizzly bars is, as a practical matter, the particle dimension d that should be used for design against interlocking. If a grizzly is not used, then the dimension d of the largest size rock fragments may be estimated by inspection of the blasted rock. Construction schedules will almost always allow such an inspection, but

in any case an experimental blast for the purpose of estimating fragmentation should be given serious consideration because of the importance of proper ore pass design and subsequent functioning in the open pit mine ore pass system.

Sampling of development muck for strength tests of the fine fraction should also be given serious consideration before final specification of ore pass dimensions, grizzly bar spacing, and selection of materials handling equipment such as feeders and so forth.

3.3 Prevention of Cohesive Arches

Cohesive arch formation occurs as a consequence of the tendency of fine particles to stick to one another. The reason for this is that fine size particles have large surface areas relative to their volumes, so that surface forces dominate their behavior. Large particles are dominated by the force of gravity. The distinction between large and small is somewhat arbitrary. Particles that pass through a 200 mesh screen ("silt" and "clay" size, 72 microns or 0.00284 inch) are generally considered to constitute the fine fraction, although somewhat larger size particles also exhibit cohesion. Coarse sand and gravel size particles (retained on the 65 mesh screen, 210 microns or 0.00828 inch) are considered cohesionless.

Table 3 shows three particle size classification systems used in soil mechanics. For materials handling purposes Jenike (1964) suggests that particles smaller than 0.01 inches (254 microns) be considered as fines. Thus, while coarse particles resist motion relative to one another because of interparticle friction, fine size particles exhibit a cohesive component of resistance in addition to friction. The strength of the fines may be sufficient to form a stable arch across the ore pass or ore pass outlet. Such an arch is a cohesive arch in contrast to an arch formed by interlocking.

The stresses in a cohesive arch depend in some way upon the span of the arch. If the span is sufficiently large, then the stresses will be sufficiently high to overcome the strength of the fines and flow will occur. Design against the formation of cohesive arches depends therefore on a knowledge of the stresses in the arch as a function of the span and on the strength of the arch forming material. A design rule for cohesive arch prevention would indicate the span required to overcome arch strength.

3.3.1 Continuum Cohesive Arches

If the discrete size and distribution of sizes of rock fragments in an ore pass are overlooked and the material handled is modeled as a continuum, then it is possible in principle to calculate the stresses in the ore pass material during filling and drawdown initiation within the context of plasticity theory. Pariseau and Nicholson (1979) have performed detailed calculations of such for vee-shaped hoppers; application to ore passes requires only a change in geometry.

A serious complication that arises in such analyses is the fact that stress and strength interact. If the material handled is sensitive to compaction, that is, if cohesion of the fines changes significantly during transit in the ore pass, then the functional form of the dependency of cohesion on stress must be known, either from experimental tests or physical reasoning or a combination of both. In this regard, Jenike (1961) assumes that cohesion is linearly dependent upon the magnitude of the major principal stress the material has experienced. Cohesion is constant under lesser stress, but increases if the major principal stress exceeds the previous maximum reached. The material thus has a virgin consolidation curve similar in many respects to that used in soil mechanics. Jenike (1961) is primarily concerned with the flow of fine industrial materials in bins and hoppers and further assumes that stress fields in the neighborhood of the outlet regions of bins and hoppers all approach the same special distribution known as radial stress. The calculation of the radial stress field thus serves for all bins and hoppers and in fact is the basis for the Jenike design charts and "flowfactor" plots. Application to cohesive arch analysis is somewhat tenuous because of the hypothetical nature of the arch geometry, applied loads and stresses. However, this is the case with all such arch analyses.

If a flat arch forms, tensile stresses arise and are likely to lead to redefinition of the arch shape to that of a dome sustaining compressive stresses if the strength of the material is relatively high in compression compared with its tensile strength. This is usually the case for cohesive frictional materials. Failure and therefore flow then requires that the stresses in a cohesive arch tend to exceed the unconfined compressive strength of the material because the bottom surface of the arch is stress free as illustrated in Figure 4. The fundamental criterion for

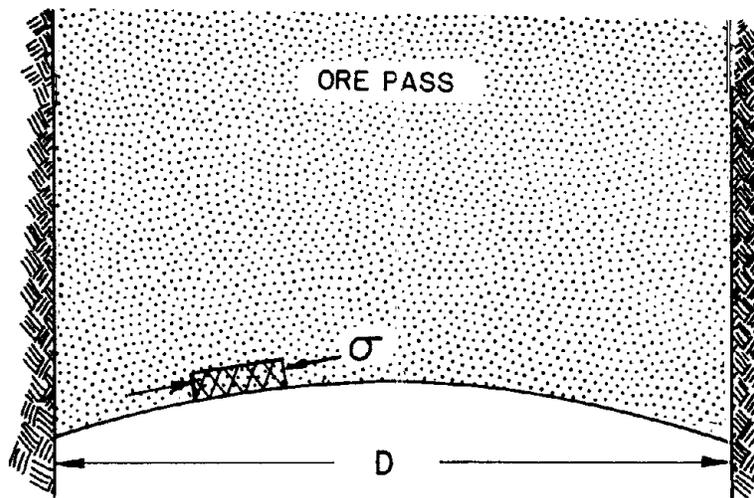


Figure 4 - Hypothetical cohesive arch of fines
in an ore pass

cohesive arch prevention under these circumstances is simply $\sigma_1 > C_0$ where σ_1 is the stress acting tangential to the arch and C_0 is the unconfined compressive strength of the material in the arch. Strictly speaking, stress cannot exceed strength, but it is helpful to pose the criterion for flow as an inequality. This criterion while certainly reasonable masks calculational difficulties that are not easily overcome. These are: estimating the stress σ_1 and estimating the strength C_0 .

In the absence of a detailed analysis of stress during ore pass filling and drawdown initiation, progress can only be made by introducing assumptions concerning the unknown data. A sequence of simplifications leading to a manageable equilibrium analysis of continuum cohesive arch, that is, one without regard to the discrete size of particles, is shown in Figure 5. The first simplification is the assumption of simple geometry as shown in Figure 5a. The next simplification is to replace the unknown distributions of tractions acting on the arch free body diagram (Figure 5b) by forces. The final simplification is to neglect the tractions acting on the upper surface of the arch, thus requiring the arch to fail under its own weight if flow is to occur (Figure 5c). For flow to occur, then

$$W > T$$

where W = weight of the arch

T = vertical shear force at arch abutments

that is,

$$\gamma Ah > \tau hP.$$

where A = cross sectional area of ore pass or outlet

h = arch thickness measured vertically

P = arch perimeter

γ = unit weight of arch forming material

τ = average shear stress acting at arch abutments

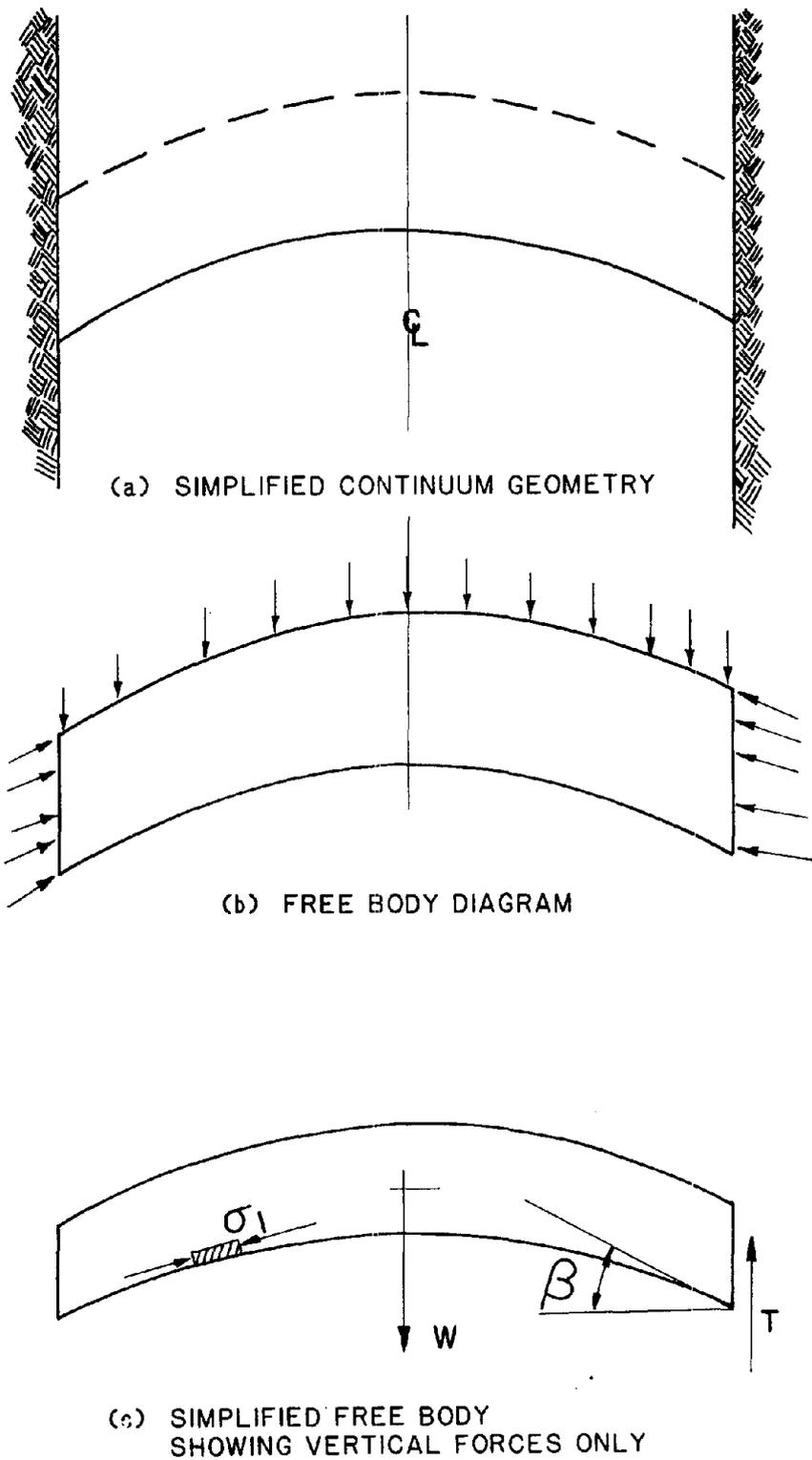


Figure 5 - Steps towards arch analysis. Arch is three dimensional and symmetric. W = arch weight, T = vertical force over periphery of arch

Hence, if

$$A/P > \tau/\gamma,$$

then the dimension defined by A/P is sufficient to prevent cohesive arching. The ratio A/P is a "hydraulic radius;" in the case of circular and square cross sections $A/P = D/4$ where D is the diameter or length of the side of the ore pass.

At the limit to equilibrium, stress must be equal to strength. There are three possibilities: (1) failure of the ore pass material only occurs, (2) failure occurs at the ore pass wall and ore pass material interface, and (3) both strength of the material and strength of the interface are overcome. In all cases the shear stress is related to the major principal stress σ_1 through the stress equations of transformation, thus

$$\tau = (\sigma_1/2) \sin 2\beta$$

where β = arch angle

Thus, the criterion for flow becomes

$$A/P > (\sigma_1 \sin 2\beta) / (2\gamma).$$

If failure occurs in the arch forming material, then $\sigma_1 = C_0$, or if failure occurs at the interface, then $\sigma_1 = C_0'$ are the unconfined or uniaxial compressive strengths of the arch and interface, respectively. The compressive strengths can be stated in terms of cohesion and friction angle once a specific form for yield is assumed. For flow then,

$$\text{Case (1)} \quad A/P > (C_0 \sin 2\beta) / (2\gamma)$$

$$\text{Case (2)} \quad A/P > (C_0' \sin 2\beta) / (2\gamma)$$

These criteria are entirely equivalent to the original fundamental criterion of $\sigma_1 > C_0$. We could, for example, require equilibrium of the hypothetical arch in Figure 5, so that $W = T$ and hence

$$\sigma_1 = (A2\gamma) / (P \sin 2\beta).$$

Applying the flow criterion

$$\sigma_1 = C_0$$

one obtains $(A2\gamma) / (P \sin 2\beta) > C_0$

that is,

$$A/P > (C_0 \sin 2\beta) / (2\gamma)$$

as before in Case (1).

The angle β is as yet unspecified. The sine cannot exceed one, hence a conservative approach consists of setting $\sin 2\beta = 1$. The choice between Case (1) and Case (2) would appear to be dictated by the larger of C_0 and C_0' . However, if C_0' is larger than C_0 , the use of the Case (2) criterion may be more conservative than necessary because the satisfaction of the Case (1) criterion assures arch failure over the entire span including a region very near the interface. Even though slip at the interface between arch and ore pass walls does not occur, shear of the arch material adjacent to the interface constitutes flow.

The design criterion for the prevention of continuum cohesive arch formation thus becomes

$$A/P > C_0 / 2\gamma.$$

If D is the diameter of a circle, side of a square or least dimension of a rectangle with long dimension L , then in all cases

$$D > (1 + 1/r) (C_0 / \gamma) \quad (B)$$

assures prevention of continuum cohesive arch formation where

$1/r = 1$ for a circular or square cross section

$1/r = D/L$ for a rectangular cross section

$1/r = 0$ for a slot cross section (L very large)

as shown in Fig. 6.

The case (i) and (ii) formulas can be refined somewhat by assuming a specific form for the failure of material and interface. If the Mohr-Coulomb criterion is adopted then the normal and shear stresses acting on a potential failure surface and related by

$$\tau = \sigma \tan \phi + k \quad \text{and} \quad \tau' = \sigma' \tan \phi' + k'$$

where the prime refers to interface conditions and

k = cohesion of arch forming material

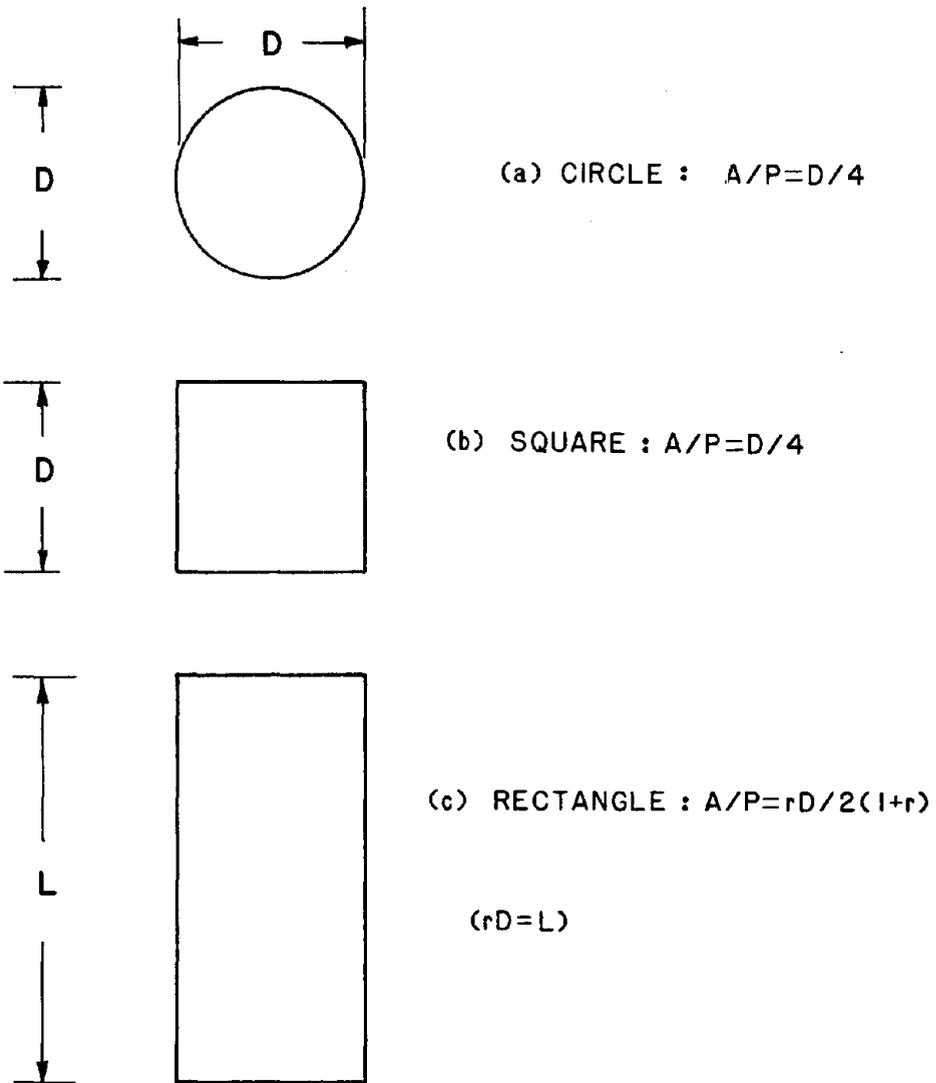


Figure 6 - Cross-section dimensions

ϕ = angle of internal friction of arch forming material

k' = adhesion of arch material to ore pass wall (interface cohesion)

ϕ' = angle of friction between arch material and ore pass wall.

The Mohr-Coulomb criterion and Mohr circle representing uniaxial compressive strength failure are shown in Fig. 7. Under the Mohr-Coulomb failure criterion, the uniaxial compressive strength is related to the cohesion and friction angle, thus

$$C_o = \left(\frac{2k \cos \phi}{1 - \sin \phi} \right) \quad \text{and} \quad C_o' = \left(\frac{2k' \cos \phi'}{1 - \sin \phi'} \right).$$

The use of k and ϕ in place of C_o is sometimes more convenient if strength test data have been obtained by direct shear rather than triaxial test procedures.

The angle between the direction of the major principal stress σ_1 and the normal to the failure surface is $\pi/4 + \phi/2$. If the wall coincides with a failure surface of the material, then $\sin 2\beta = \sin(\pi/2 + \phi) = \cos \phi$ and the criterion for flow becomes

$$\begin{aligned} A/P > (C_o \sin 2\beta) / (2\gamma) &= (C_o \cos \phi) / (2\gamma) \\ &= (2k \cos \phi) \cos \phi / (1 - \sin \phi) \end{aligned}$$

that is, $A/P > (k/\gamma) (1 + \sin \phi)$

which for a circle or square is

$$D > (4k/\gamma) (1 + \sin \phi).$$

This result is depicted graphically in Figure 8. Any value of D above the line corresponding to ϕ for a given k/γ will prevent cohesive arching over circular or square cross sections. The coincidence of the arch material failure surface with the ore pass wall is the "rough" wall case. The rough wall case provides an upper bound to the arch angle β ; instead of the criterion (B) one has

$$D > (1 + 1/r) (C_o \cos \phi / \gamma) \quad (C).$$

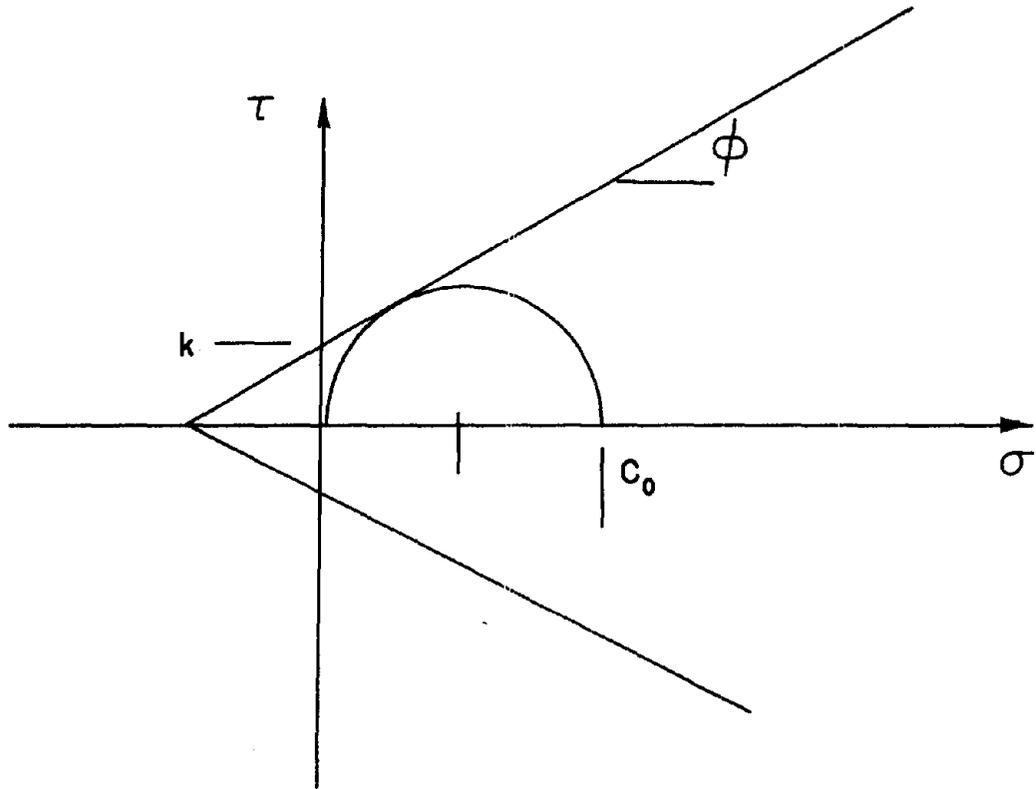


Figure 7 - The Mohr - Coulomb failure criterion and uniaxial compressive strength

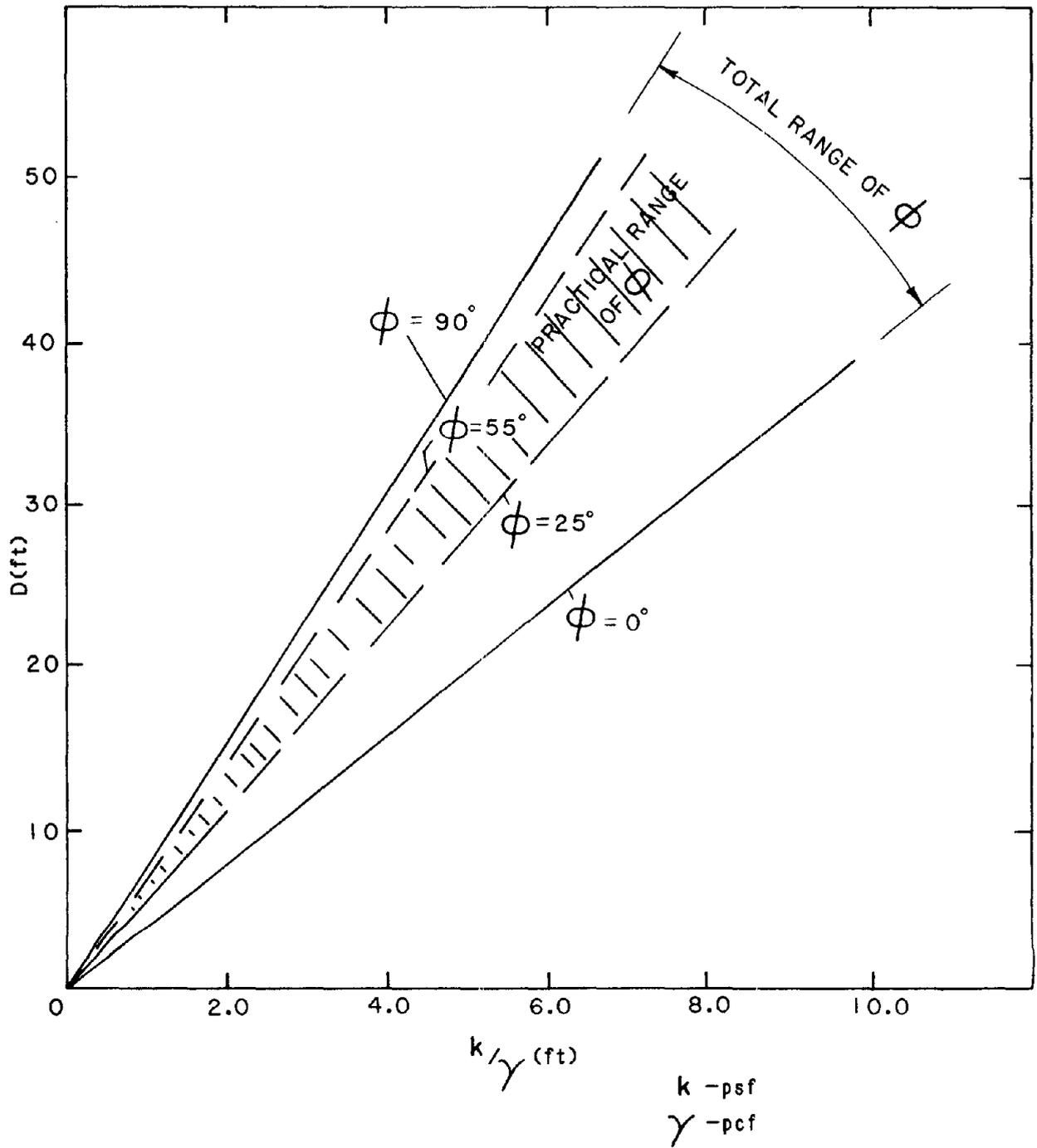


Figure 8 - Cross-sectional dimension required for the prevention of cohesive arches. D must be above the line corresponding to the material friction angle, ϕ

The degree of conservatism assumed in (B) by setting $\sin 2\beta = 1$ can be assessed in view of (C). If (C) is considered the base and one assumes a reasonable value of ϕ , say $\phi = 35$ deg, then criterion (B) requires an outlet 22% greater than (C). If $\phi = 45$ deg, then (B) requires an outlet 41% greater than (C). The difference may therefore be substantial; (B) may be overly conservative.

The lower limit to the arch angle is dictated by the requirement that tension be absent according to the original assumptions concerning arch formation. A flat arch, for example, would behave as a beam and generate tensile stresses (a flat arch corresponds to $\sin 2\beta = 0$). In any case, flow requires that largest potential arch be prevented. Criterion (C) serves this purpose.

3.3.2 Cohesive Particle Arches

An alternative approach to the analysis of cohesive arch formation is to recognize the discrete nature of the rock fragments in the material forming the arch before developing an arch prevention criterion from the requirement for equilibrium. Consider, for example, a single cubical particle of dimension d cemented to adjacent particles as shown in Figure 9a. Equilibrium requires that

$$W = K,$$

that is, $\gamma d^3 = 4kd^2.$

Thus for flow

$$d > (4k/\gamma).$$

With reference to Figure 9b equilibrium of the array of particles requires

$$W = K$$

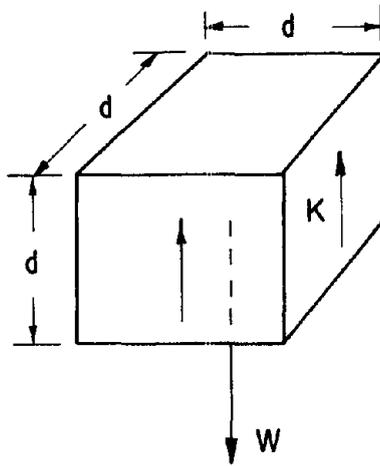
where now W is the weight of the entire array and K is the total cohesive force acting over the periphery of the array. With the notation shown in Figure 9b, one has

$$m\gamma d^3 = 2(m + n)kd^2.$$

Hence flow occurs if

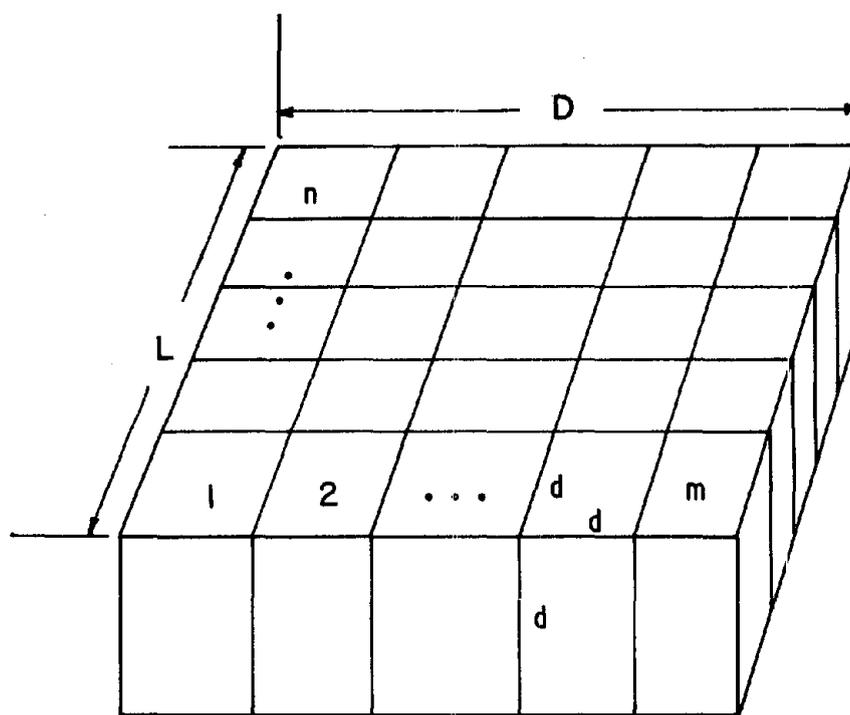
$$md > (1 + m/n)(2k/\gamma).$$

Let $m < n$, so that $md = D$, $nd = L$, $m/n = D/L = 1/r$, then the criterion for flow becomes



W=weight
K=cohesive force

(a) SINGLE PARTICLE
d=EDGE LENGTH OF THE CUBE



$D=md$
 $L=nd$

(b) ARRAY OF PARTICLES
d=EDGE LENGTH OF THE CUBE

Figure 9 - Cohesive particle arches; single and multiple arrays

$$D > (1 + 1/r) (2k/\gamma) \quad (D)$$

The criterion (C) in the previous section is in terms of the cohesion k

$$D > (1 + 1/r) (2k/\gamma) (1 + \sin\phi). \quad (C)$$

Comparison of (D) for the prevention of cohesive particle arches with (C) for the prevention of cohesive continuum arches shows that the former is more optimistic because a lesser cross section dimension D is required for flow. However, if compressive horizontal forces were acting, then some frictional resistance to flow would be mobilized in addition to cohesion. The optimism present in (D) therefore is not justified.

The purpose of examining the effect of the discrete nature of the arch forming material is to reveal the influence of particle size, if any, on flow. In this regard the flow criterion for cohesive particle arches developed here can be written as

$$d > (1/m + 1/n) (2k/\gamma). \quad (E)$$

The criterion (E) shows that the larger is d the greater is the flowability of the cross section with respect to cohesive arch formation. A large average particle size will enhance flowability according to (E), other factors being equal. If $m = n = N$, then (E) assumes the form

$$d > (4k/N\gamma).$$

The prevention of arching thus requires that the cross section dimension D satisfy both criteria: (A) given in Table 2 and (C). The discrete nature of broken rock arches does not change the form of the cohesive arch criterion, but indicates that a larger average particle size will promote flow other factors being equal. Although based on empirical observations and analyses of hypothetical arches, equilibrium requirements are satisfied and the major variables of the problem are conceptually related in a simple way.

3.4 Scale Effects

The vast majority of ore pass systems are found in underground mining operations; very few open pit mine ore pass system combinations are available for study in the United States. Consequently almost all practical knowledge of ore passes is based on experience with the relatively

small ore passes used in underground mines. Ore passes contemplated for use in open pit mines are much larger, so that a question arises concerning the effect of scale on ore pass design.

The question concerning the validity or reliability of extrapolating experience obtained from relatively small underground mine ore passes to relatively large open pit ore passes can be posed in practical terms by simply asking whether or not rock blasted to a much larger size in an open pit mine will flow as readily in a larger ore pass as the same rock blasted to a smaller size and handled in a smaller underground ore pass system.

If d is a dimension of blasted rock fragments and D is a dimension of the ore pass cross section, then the ratio D/d should exceed, say, 5 in either case in order to prevent hang-ups caused by interlocking. Thus with respect to interlocking arch prevention, there is no difference between open pit and underground ore pass design.

However, if k is the cohesion of the fine fraction of the material handled and ϕ the angle of internal friction, then the material has the same cohesive strength in either case and consequently the larger open pit ore pass system should be less likely to experience cohesive arch formation because of the larger span required. The effect of scale is therefore favorable to the functioning of the larger ore passes anticipated in the open pit ore pass system.

A conceptual model of why this is so is shown in Figure 10. In Figure 10 small particle (a cube) of unit weight, γ , linear dimension, d , and volume, V , is cemented to an adjacent particle by cohesion, k , acting over the interface area $4A$. Equilibrium requires that the weight of the particle be supported by the cohesive resistance at the sides, thus $W = K$ or $\gamma AD = 4kA$, that is, $\gamma d = 4k$, for equilibrium. The arch fails if $W > K$, that is, if $\gamma d > 4k$ or equivalently if $d > 4k/\gamma$. The larger the fragment dimension, d , the more cohesive must the material be in order to maintain arch equilibrium. However, the cohesiveness of the material is associated with the fine fraction and is the same in either case. Hence the larger d (and D) expected in open pit mines implies greater ore pass flowability.

An additional stress σ_v on top of the particle in Figure 10 requires additional strength at the interface. This can only arise through the mobilization of interface friction by a normal force acting across the interface. If the additional stress, σ_v , is caused by a column of

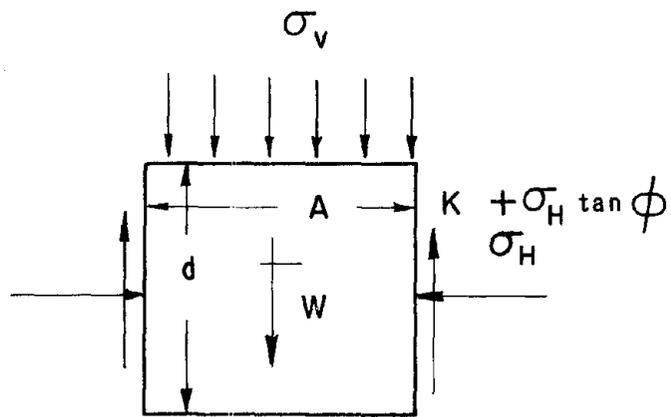


Figure 10 - Particle of size d in a conceptual particle arch

material D high then $\sigma_v A = \gamma DA$ is the additional vertical force that must be equilibrated by side friction $N \tan \phi = 4A \sigma_h \tan \phi$ and therefore $\sigma_v = \gamma D = 4\sigma_h \tan \phi$ for equilibrium. If $\sigma_v > 4\sigma_h \tan \phi$, then flow occurs. This last relationship is independent of D and d and thus independent of scale.

If one considers a smaller underground ore pass as a model and a larger open pit mine ore pass a prototype structure, then a dimensional analysis reveals the effects of scale changes between the two.

The pertinent variables are those that enter from:

- | | | |
|-----|-------------|----------------------|
| (1) | equilibrium | σ, ρ, g, a |
| (2) | deformation | ϵ, u, v |
| (3) | material | E, Y, η, γ |
| (4) | geometry | l |

where

σ	=	stress
ρ	=	density
g	=	acceleration due to gravity
a	=	acceleration
ϵ	=	strain
v	=	velocity
u	=	displacement
E	=	elastic moduli
Y	=	strength moduli
η	=	viscous moduli
l	=	length
γ	=	specific weight.

If the material handled is isotropic then two moduli are required for each category of material properties, that is, two elastic moduli (Young's Modulus and Poisson's ratio for instance), two strength moduli (cohesion, k , and angle of internal friction, ϕ), and two viscous moduli (shear and bulk viscosity). Since the flow of broken ore in ore passes is inelastic and not highly viscous but rather is mainly cohesive and frictional, one need only consider k and ϕ of the material and the adhesion k' of the material to the ore pass walls and the friction ϕ' of the material flowing on the ore pass walls.

Although the actual relationship amongst the variables that enter ore pass design for flow may be unknown, it may nevertheless be represented symbolically as

$$F(d, D, D_0, H, k, \phi, k', \phi', \gamma, \epsilon, u, v, a, g, \rho, \sigma) = 0$$

This relationship must be dimensionally consistent and can therefore be represented as a function of product of powers of its arguments, that is, as

$$F(\pi_1, \pi_2, \pi_3, \dots, \pi_n) = 0$$

where π_i -term, π_i , is the i th product of powers of the original variables. The π_i -terms are dimensionless and so the function F expresses a relationship independent of scale. The π_i -terms are thus similitude requirements.

The π_i -terms are determined in the usual way by first forming the matrix of coefficients,

	d	D	D ₀	H	ϕ	k	ϕ'	k'	γ	ϵ	u	v	a	g	ρ	σ
M	0	0	0	0	0	1	0	1	1	0	0	0	0	0	1	1
L	1	1	1	1	0	-1	0	-1	-2	0	1	1	1	1	-3	-1
T	0	0	0	0	0	-2	0	-2	-2	0	0	-1	-2	-2	0	-2

where M, L, and T refer to the fundamental units of mass, length, and time respectively. The rank of the coefficient matrix is 3. There are thus 13 independent π_i -terms possible. Such a set is

$$\begin{aligned} \pi_1 &= D/d \\ \pi_2 &= D_0/d \\ \pi_3 &= H/d \\ \pi_4 &= \phi \\ \pi_5 &= k/\sigma \\ \pi_6 &= \phi' \\ \pi_7 &= k'/\sigma \\ \pi_8 &= \gamma d/\sigma \\ \pi_9 &= \epsilon \\ \pi_{10} &= u/d \\ \pi_{11} &= v^2/dg \\ \pi_{12} &= a/g \\ \pi_{13} &= \rho g/\gamma \end{aligned}$$

which by inspection contains only independent terms. (Each term contains a variable not present in any other term and so cannot be obtained as a combination of any of the other terms).

The first pi-term has been anticipated in the discussion of particle arching as a result of interlocking while the last term is simply Newton's second law of motion ($F = ma$) applied to a unit volume of material at the surface of the earth resulting in the force of gravity $\gamma = \rho g$.

Any dimensionless product of the variables considered is a scaling requirement and may be given a physical interpretation. For example,

$$\pi_8/\pi_5 = \pi_{14} = (\gamma d/k)$$

may be interpreted as the ratio of stress caused by gravity (weight of a unit volume of material d units high) to strength k . As a similitude requirement π_{14} must hold in model and prototype, thus

$$(\pi_{14})_m = (\pi_{14})_p$$

where m and p refer to model and prototype, respectively. Hence

$$(\gamma d/k)_m = (\gamma d/k)_p$$

and

$$(k_p/k_m) = (d_p/d_m)(\gamma_p/\gamma_m).$$

The specific weight of the ore will be the same in either case, so that

$$(k_p/k_m) = (d_p/d_m).$$

Thus if larger ore passes are viewed as prototype structures of smaller model ore passes, then the cohesion must be increased in the prototype in direct proportion to the length scale factor (d_p/d_m). However, the cohesion associated with the fine fraction is the same in either case.

Alternatively, one could say that scaling from small to large ore passes handling larger particles reduces the cohesion. However, a model relationship cannot exist between small and large ore passes in any case.

From another perspective, the π_{14} term $\gamma d/k$ cannot be constant because the body forces of gravity characterized by γdA grow faster with increasing d than the surface forces of cohesion characterized by kA . In fact this result motivated the conceptual view of the cohesive particle arch d units thick presented in the first part of this section.

If the discrete nature of the material handled in ore passes is overlooked or viewed as a continuum then d does not appear in the analysis. The same view may be obtained from

$$(\pi_1) (\pi_8) / (\pi_5) = \pi_{16}$$

$$(D/d) (\gamma d/\sigma) (\sigma/k) = (\gamma D/k) = \pi_{16}.$$

Thus $(k_p/k_m) = (D_p/D_m) (\gamma_p/\gamma_m)$, and again since $\gamma_p = \gamma_m$, similitude requires

$$(k_p/k_m) = (D_p/D_m)$$

which again rules out a model relationship between small and large ore passes.

The lack of a model relationship between large and small ore passes can be seen from a more general viewpoint. Consider the requirement obtained by

$$(\pi_{13}) (\pi_8) = \pi_{15} = (\rho g d/\sigma).$$

For similitude then

$$(\sigma_p/\sigma_m) = (\rho_p/\rho_m) (g_p/g_m) (d_p/d_m).$$

However, if the model and prototype handle the same material then they have the same density and

$$(\sigma_p/\sigma_m) = (g_p/g_m) (d_p/d_m).$$

But also from the π_5 term

$$(\sigma_p/\sigma_m) = (k_m/k_p) = 1$$

since the materials handled are the same and thus have the same cohesion. It follows that

$$(g_m/g_p) = (d_p/d_m)$$

if the same material is used in model and prototype. This means that the gravitational acceleration in the model must be greater than in the prototype in the same proportion as corresponding lengths are scaled. In laboratory gravity models, the usual way of accomplishing this is by centrifuging.

Obviously g is the same in underground and open pit mine ore passes. Again no model relationship can therefore exist between the two. This result is favorable to the extrapolation of experience obtained from underground mine ore passes to the design of open pit ore passes in the sense that the larger dimensions of open pit ore passes favor flowability. Design of large ore passes handling large size material based on experience with small ore passes will be conservative with respect to the prevention of arch formation.

3.5 Prevention of Piping

Piping refers to the formation of a stable roughly cylindrical opening directly over the outlet of an ore pass as illustrated in Figure 11. Pipe formation obviously reduces the live storage capacity of an ore pass and greatly increases the probability of arching. Pipe formation also presents a hazard to operations because of the potential for sudden collapse of the pipe walls. The evolution of the stress field in a pipe of ore pass material viewed as a continuum is quite complex because of indeterminate impact loading, compaction and cementation, moisture content variations, wall adhesion, and friction. However, an estimate of the critical stress acting tangential to the inside pipe wall and the strength of material there can be made on the basis of equilibrium and testing for material properties over the range of stresses expected in the ore pass. This is the same approach adopted in the analysis of arching and the development of arch prevention guidelines.

The inner surface of the pipe wall is traction free, so that the formation of a stable pipe implies that the unconfined compressive strength C_0 of the pipe wall material is not exceeded by the major principal stress σ_1 which acts tangential to the pipe wall. The basic criterion for pipe prevention, that is, for breakdown of the pipe and flow is then

$$\sigma_1 > C_0$$

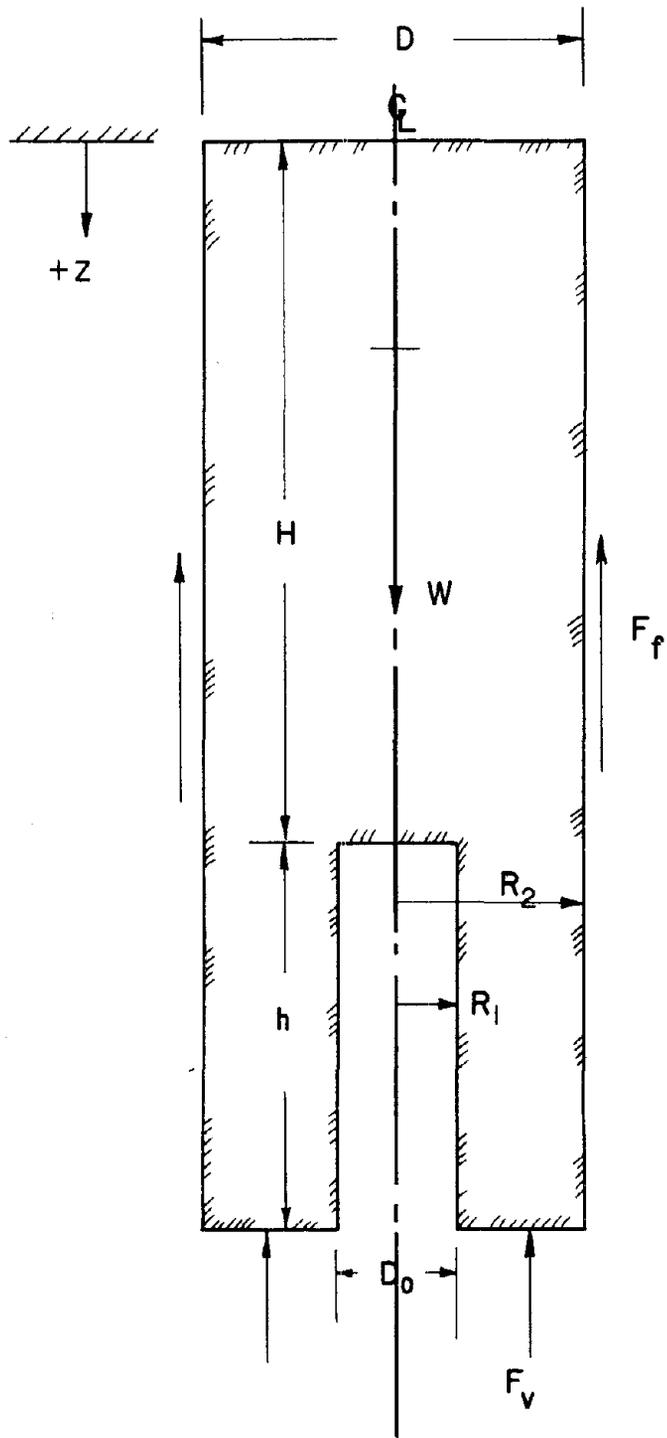


Figure 11 - Equilibrium of ore pass material with a "pipe"

which is identical to the basic criterion for the prevention of cohesive arches.

In the extreme case, the pipe extends to the top of the ore pass material as shown in Figure 12. Analysis of the pipe shown in Figure 12 leads to an expression for the maximum height of a stable pipe. Greater heights result in pipe breakdown and flow. Three cases of increasing complexity and increasing height of a stable pipe are identified:

- Case (1). negligible wall friction and adhesion
- Case (2). wall friction present
- Case (3). wall friction and adhesion present.

The criterion for flow in all Cases remains $\sigma_1 > C_0$. In the first Case, the walls of the ore pass are considered to be so remote from the inner pipe wall that even if friction and adhesion are present the remoteness of the outer wall makes their influence on the inner pipe wall negligible. Alternatively, if friction and adhesion are in fact negligible, then the position of the outer wall relative to the inner wall is of no consequence. The latter is unlikely except in the presence of large amounts of water in the ore pass which would also preclude piping and lead to the opposite situation, that is, "flooding." In general friction and adhesion are present, and the stability of the pipe depends on the nearness of the outer wall relative to the inner pipe wall.

If h is the limiting height of the pipe, D is the diameter or least dimension of the ore pass, and D_0 is the pipe dimension (diameter of the empty region over the ore pass outlet of dimension D_0), then

$$F(h, D, D_0, C_0, \phi', k', \gamma) = 0$$

is the functional relationship amongst the variables of importance to the formation of a stable pipe. In dimensionless form

$$F(h/D, D_0/D, C_0/\gamma D, k'/\gamma D, \phi') = 0$$

hence

$$(h/D) = f(D_0/D, C_0/\gamma D, k'/\gamma D, \phi')$$

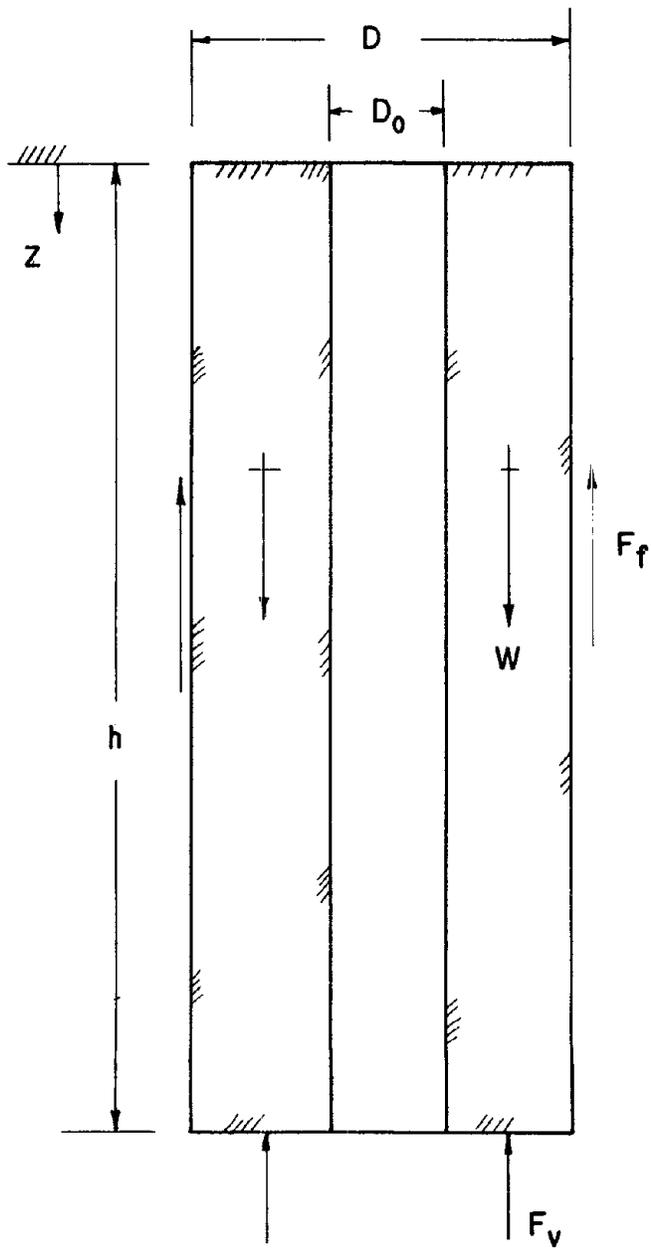


Figure 12 -- Extreme case of piping

where

- C_o = the unconfined compressive strength of the pipe forming material
- ϕ' = the friction angle between pipe material and ore pass wall
- k' = the adhesion between pipe material and ore pass wall
- γ = the specific weight of the pipe material.

If the wall is remote so that its influence is negligible then

$$(h/D) = f(C_o/\gamma D).$$

The simplest form of the function f is a constant, that is,

$$(h/D) = (C_o/\gamma D) (\text{constant}),$$

which can be simplified further by setting the constant in the second parentheses on the right equal to one. Thus

$$(h/D) = (C_o/\gamma D)$$

gives the limiting height of a stable pipe in Case (1) where wall friction and adhesion have a negligible influence on pipe stability.

This result can also be obtained directly by simply noting that the vertical stress in the pipe wall $\sigma_1 = \gamma h$, and at the limit of equilibrium $\sigma_1 = C_o$. Thus,

$$\gamma h = C_o,$$

which is identical to the expression obtained from the preceding dimensional analysis. For flow in the Case (1), then

$$(h/D) > (C_o/\gamma D) \quad \dots \text{Case (1) - pipe prevention criterion.}$$

The same procedure leads to a similar pipe prevention criterion in the general case. Analysis of equilibrium gives explicit form to the symbolic relationship amongst the variables of interest that arises in a dimensional analysis.

With reference to Figure 13, equilibrium requires that the sum of the forces acting in the vertical direction be zero, thus

$$F_z = 0,$$

that is,

$$0 = (\sigma_v + \Delta\sigma_v) A - \sigma_v A + \tau P \Delta z - A \gamma \Delta z$$

where:

σ_v = average vertical stress in the pipe

τ = average stress at the pipe ore pass interface

A = cross-sectional area of pipe material acted on by σ_v

P = perimeter of the ore pass

h = pipe height

z = vertical coordination from the pipe top

γ = specific weight of pipe material.

If slip at the wall impends, then the maximum support of the pipe material by wall friction and adhesion obtains, and

$$\tau = \sigma_H \tan \phi' + k'$$

where

ϕ' = ore pass material ore pass wall interface friction angle

k' = adhesion of ore pass or pipe material to the ore pass wall

σ_H = average horizontal stress acting normal to the ore pass wall interface.

Moreover, under this condition the vertical stress σ_v is functionally related to the horizontal stress σ_H , as shown in Figure 14. From the geometry of the Mohr's circle representing the stress state in the ore pass material at the ore pass wall

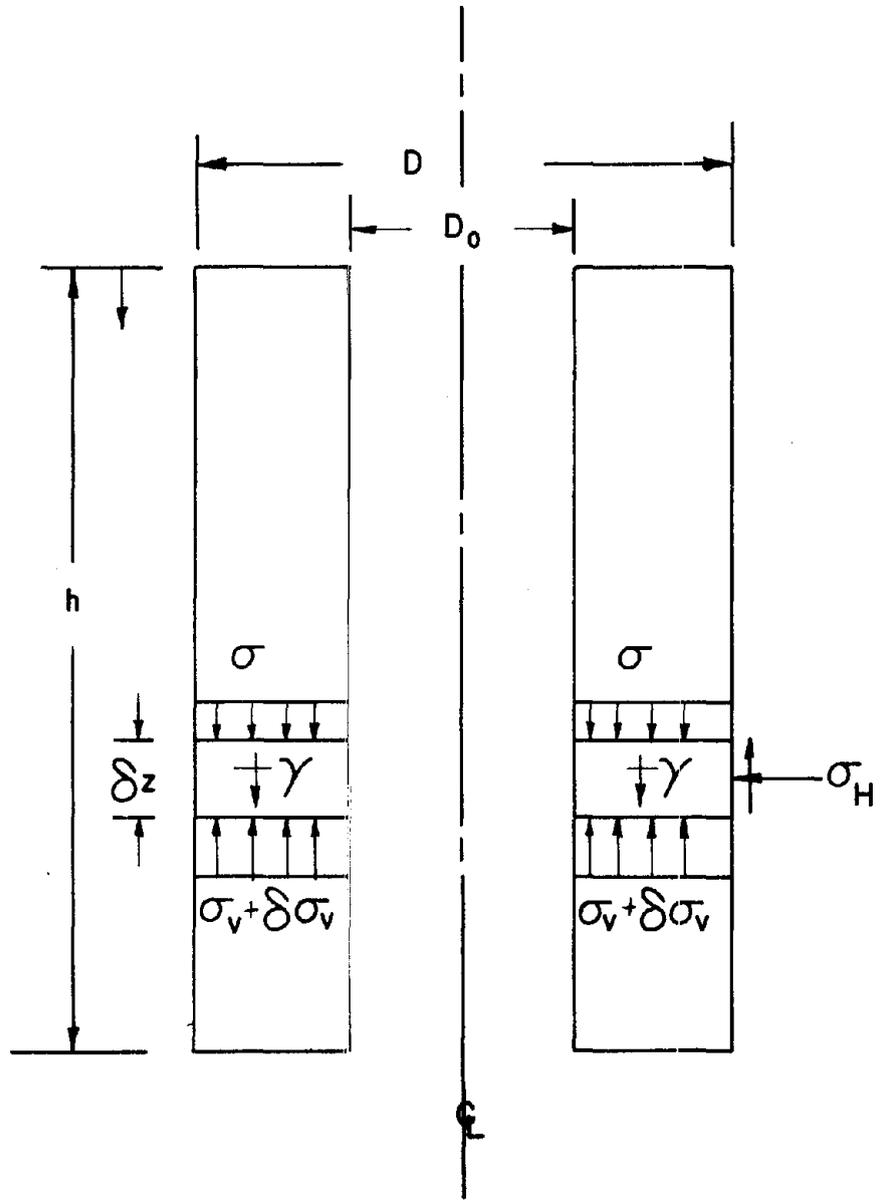


Figure 13 - Stresses in a "pipe"

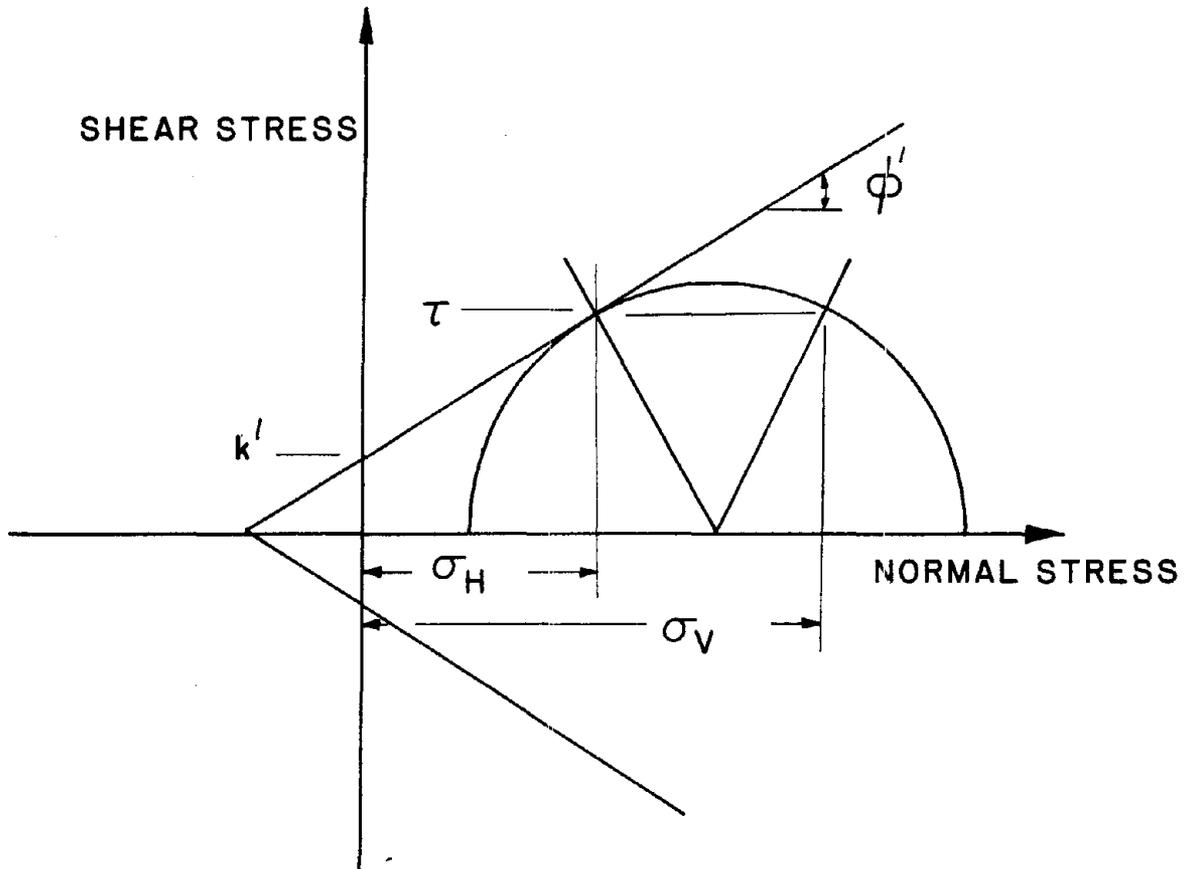


Figure 14 - Interface slip and stresses τ , σ_h , and σ_v

$$\sigma_H = \sigma_V - 2\tau \tan\phi',$$

and

$$\tau = (\sigma_V \tan\phi' + k') / [1 + 2(\tan\phi')^2].$$

Equilibrium thus requires

$$d\sigma_V/dz + (P/A) (\sigma_V \tan\phi' + k') / [1 + 2(\tan\phi')^2] - \gamma = 0.$$

After integrating this equation

$$\sigma_V = (C_2/C_1) + C_3 \exp(-C_1 z)$$

where:

$$C_1 = (P/A) M \tan\phi'$$

$$C_2 = \gamma - (P/A) M k'$$

$$M = 1 / [1 + 2(\tan\phi')^2]$$

exp = exponential function, $e^{(\dots)}$.

The constant of integration C_3 is determined by the condition that at $z = 0$, $\sigma_V = 0$, thus

$$C_3 = -C_2/C_1$$

and therefore

$$\sigma_V = (C_2/C_1) [1 - \exp(-C_1 z)].$$

At the pipe bottom $z = h$ and

$$\sigma_V = \left(\frac{\gamma R - M k'}{M \tan\phi'} \right) [1 - \exp(-M \tan\phi' h/R)] \quad (F)$$

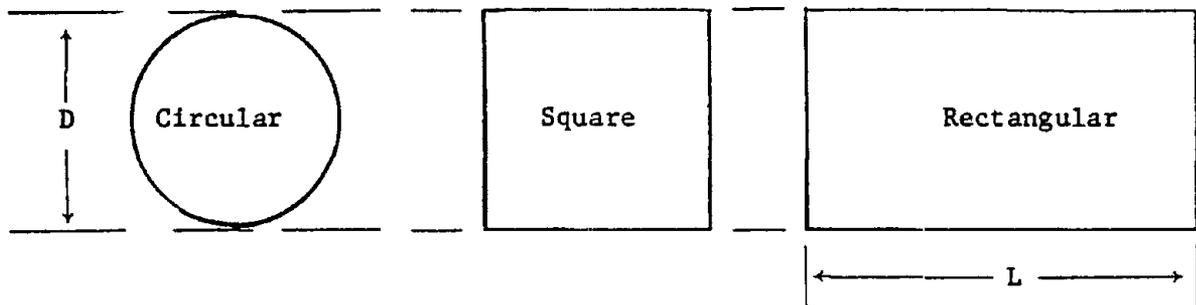
where $R = A/P$ is a pseudo-hydraulic radius.

Expressions for R for circular and rectangular ore passes containing a circular pipe are given in Table 4. With reference to Table 4, it is clear that as the ore pass diameter, D , becomes large relative to the pipe diameter (outlet dimension), D_p , that Equation (F) becomes indeterminate ($\sigma_V = (\infty)(0)$). However, the limit is readily evaluated with the aid of l'Hospital's rule. In the limit

Table 4 - R values (R = A/P) *

Circular ore pass:	$(D/4)[1 - (D_o/D)^2]$
Square ore pass:	$(D/4)[1 - \frac{\pi}{4}(\frac{D_o}{D})^2]$
Rectangular ore pass:	$(D/2(1 + \frac{1}{r}))[1 - (\pi/4r)(D_o/D)^2]$

*A = pipe area, P = ore pass perimeter, D = ore pass diameter, side of square section, least dimension of rectangular cross section, L = long dimension of rectangle, r = L/D, D_o = pipe diameter.



$$\sigma_V = \lim_{R \rightarrow \infty} \frac{\frac{C}{dR}(1 - e^{-C_1 z})}{\frac{d}{dR}(C_1/C_2)} = \gamma z.$$

This is the result that one would expect intuitively. It also shows that the influence of wall friction and adhesion diminishes greatly as the width of the ore pass becomes large relative to the outlet dimension or pipe diameter.

If R is expressed as

$$R = DR'$$

then Equation (F) assumes the form

$$\sigma_V = \left(\frac{DR' - k'M}{M \tan \phi'} \right) [1 - \exp(-M \tan \phi' / R') (h/D)].$$

At the limit of equilibrium $\sigma_1 = C_o = \sigma_V$ at the inside wall of the pipe. If Equation (F) is used to estimate σ_V , then the height of the pipe at the limit to equilibrium is given by

$$(h/D) = -(R'/M \tan \phi') \ln \left\{ 1 - \left[\frac{(C_o / \gamma D) (M \tan \phi' / R')}{1 - (Mk' / \gamma DR')} \right] \right\} \quad (G)$$

This result is symbolically

$$(h/D) = f(C_o / \gamma D, k' / \gamma D, D_o / D, \phi')$$

and could have been obtained by a dimensional analysis. However, an explicit form is needed for calculations, and this is obtained from the requirement of equilibrium. The requirement for flow is $\sigma_1 > C_o$ and since $C_o > 0$ the term in parentheses in Equation (F) must be positive (the term in brackets is always positive and less than one for finite h).

Thus

$$\gamma R - Mk' > 0$$

or

$$\left(\frac{PMk'}{\gamma A} \right) > 1$$

This can be interpreted as a requirement that the cohesive force $k'PM$ be less than the gravity force γA acting on a slab of unit thickness for flow to occur. Alternatively, if the wall adhesion is sufficient to support each vertical foot of pipe, then $\sigma_V = 0$ and no vertical load is transmitted from one slab to another and no limit to the pipe height is

reached. This condition is translated into the requirement that the term in square brackets in Equation (G) be non-negative and less than one for a critical pipe height to be reached above which flow occurs.

In this regard, the effect of wall adhesion is almost always neglected in somewhat similar studies of the flow of processed materials in bins, silos, bunkers, and so forth. However, the effect may be of importance to the handling of cohesive materials in ore passes. If the adhesion is so great that slip at the ore pass wall does not occur, then it is likely that flow will occur in a narrow zone of ore pass material immediately adjacent to the wall. In this case the appropriate wall friction and cohesion parameters are those of the material itself. Shear of the material on itself near but not at the wall is equivalent to the equality $C'_o = C_o$ and $\phi = \phi'$ where C'_o is unconfined compressive strength associated with k'^o and ϕ' . If the unconfined compressive strength C'_o is associated with wall cohesion and friction through a Mohr-Coulomb criterion for slip, then

$$C'_o = (2k' \cos \phi') / (1 - \sin \phi').$$

The preceding analysis is based on equilibrium of forces and therefore average stresses. In fact there is a distribution of stress within the pipe that qualitatively must be such that the vertical stress is higher on the inside of the pipe wall than the outside near the ore pass wall pipe interface. If the ore pass is wide relative to the height of material in the pipe, then the influence of the wall is small regardless of the wall friction and adhesion, and the height of the pipe near the outlet is dictated by first case conditions. The height of the inside of the pipe is therefore not without a limit. However, a progressive build-up of highly cohesive material at the ore pass wall is possible if the adhesion is relatively high. Thus pipe prevention requires that

$$(h/D) > -(R'/M \tan \phi') \ln \left\{ 1 - \left[\frac{(C_o/\gamma D) (M \tan \phi' / R')}{1 - (M k' / \gamma D R')} \right] \right\} \quad (H)$$

where:

- h = pipe height
- D = ore pass diameter or least dimension
- ϕ' = friction angle between pipe material and ore pass wall

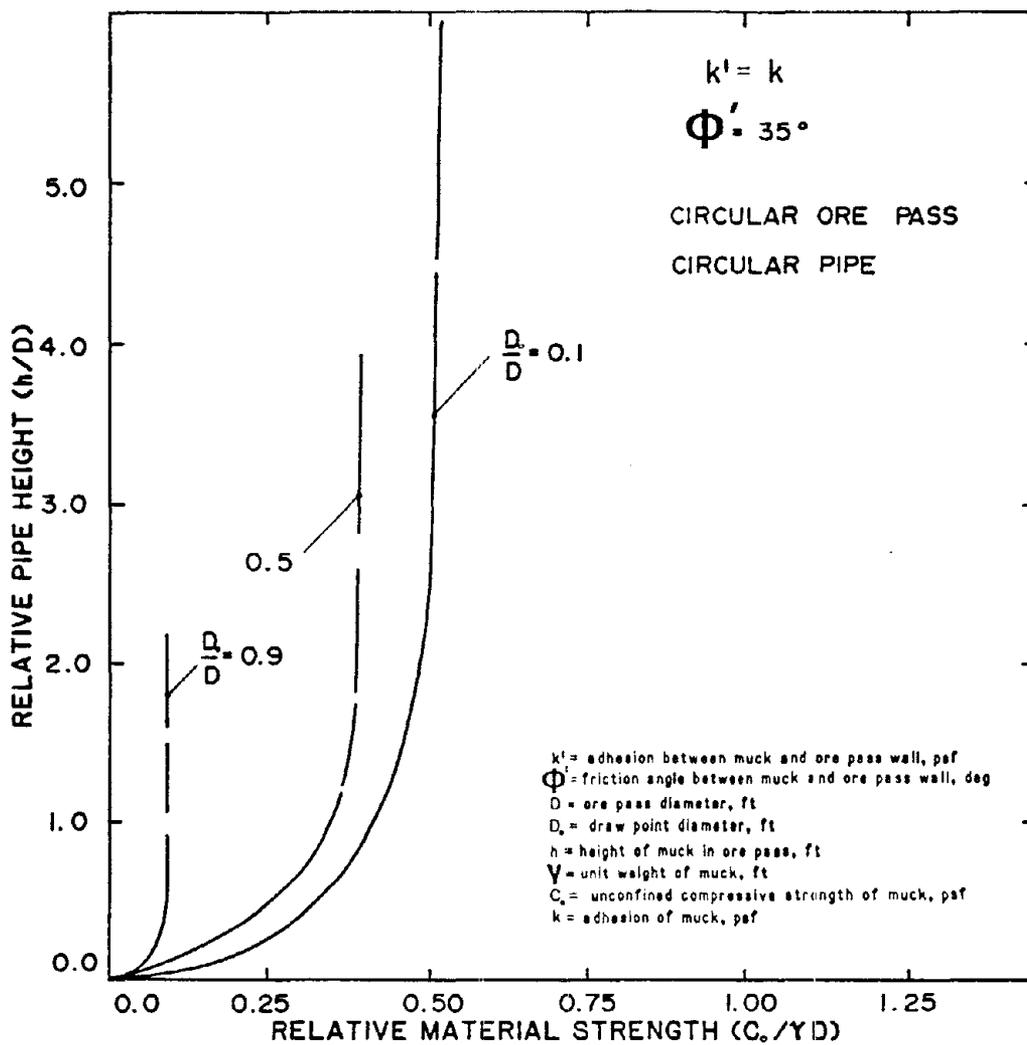


Figure 15b - Relative pipe height (h/D) as a function of relative material strength ($C_0/\gamma D$); $k' = k$.

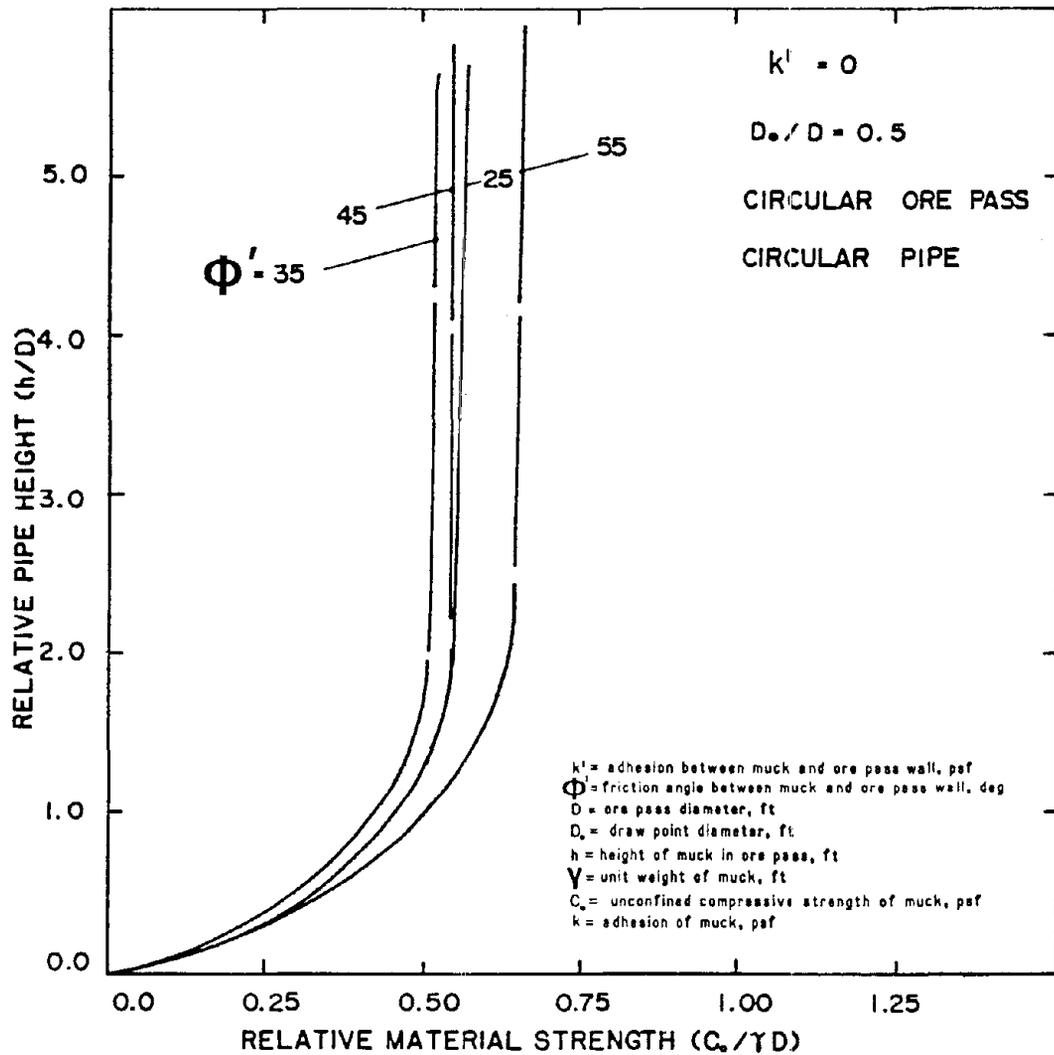


Figure 15c - Relative pipe height (h/D) as a function of relative material strength ($C_o/\gamma D$); Φ' as a parameter.

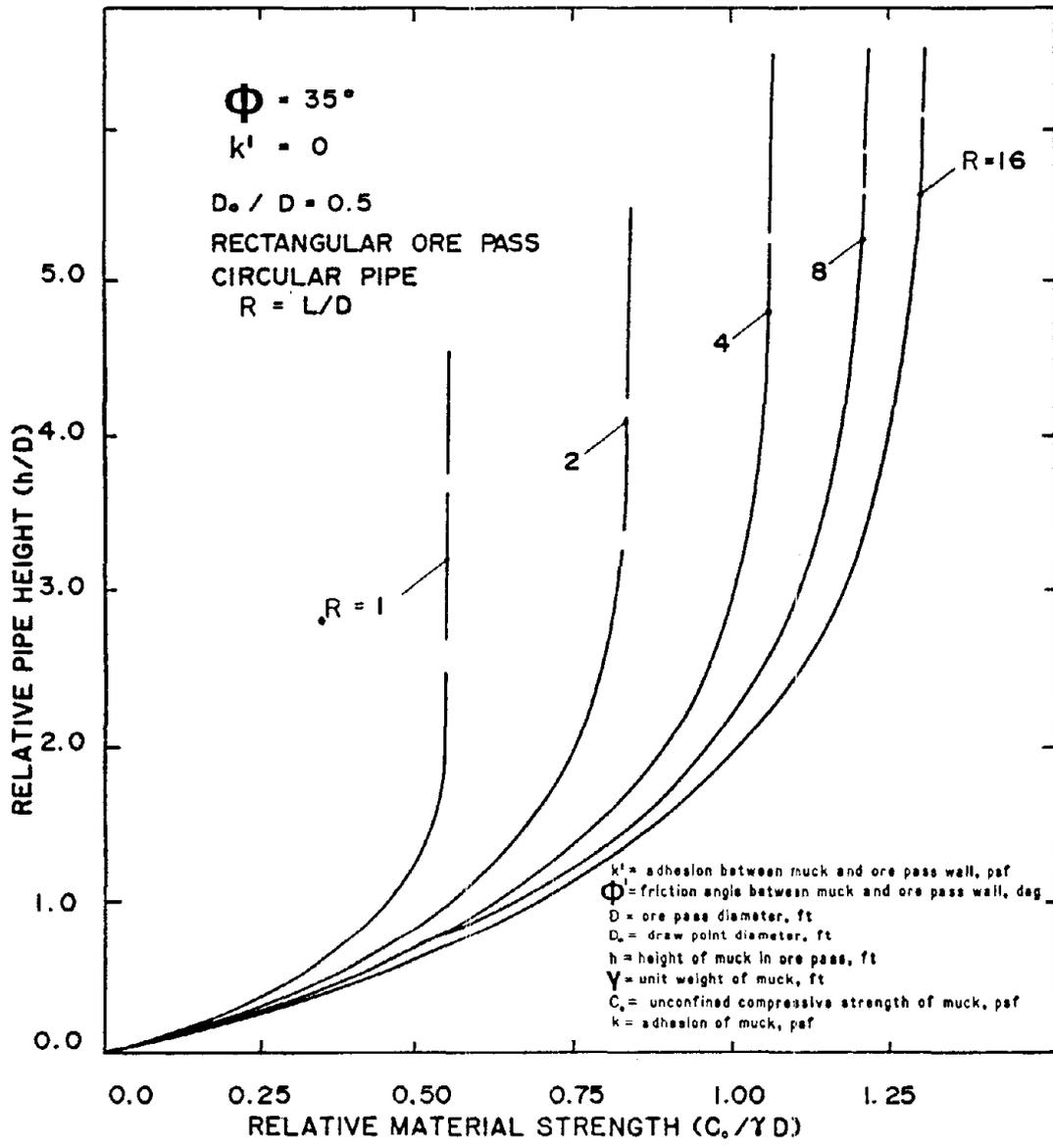


Figure 15d - Relative pipe height (h/D) as a function of relative material strength ($C_o/\gamma D$); r as a parameter - rectangular ore passes.

k' = adhesion between pipe material and ore pass wall

γ = specific weight of pipe material

C_o = unconfined compressive strength of pipe material

$M = 1/[1 + 2(\tan\phi')^2]$, a constant

R' = a geometric term that depends on the ratio D_o/D

\ln = natural logarithm.

The geometric factor R' is related to the ratio of pipe cross sectional area A to pipe (ore pass) perimeter P , $DR' = A/P$. The term $R = DR'$ is a pseudo-hydraulic radius. Expressions for R' are:

$$\text{Circular Ore Pass: } R' = (1/4)[1 - (D_o/D)^2]$$

$$\text{Rectangular Ore Pass: } R' = [1/2(1 + 1/r)][1 - (\pi/4r)(D_o/D)^2]$$

As shown, in the limit as the ratio D_o/D goes to zero with increasing D , the right hand side of criterion (F) goes to $C_o/\gamma D$. Thus in the limit as the ore pass becomes wide relative to the pipe height, the influence of the walls becomes negligible and the first case obtains. The first case represents a lower limit to stable pipe height; wall adhesion and friction add to this limit.

Plots of (h/D) as a function of $(C_o/\gamma D)$ for a range of values of the parameters D_o/D , ϕ' and k' are presented in Figure 15. Relative pipe heights h/D above the line in Figure 15 insure pipe breakdown and flow.

With reference to Figure 15, the curves divide the plane of the plot into regions of flow and no-flow. For a given material strength, C_o , of specific weight, γ , in an ore pass of dimension, D , that is, for a given value of $C_o/\gamma D$, flow occurs provided h/D is above and to the left of the appropriate curve. If the ore pass is drawn down to a height h such that h/D falls below or to the right of the curve, then a stable pipe may form because the weight of the overlying material is no longer sufficient to shear the material at the pipe bottom near the outlet.

Figure 16 shows a hypothetical piping sequence that may also lead to doming at the outlet. In this sequence drawdown of ore is initiated at the outlet under a high material head and uniform flow. As the head is reduced the

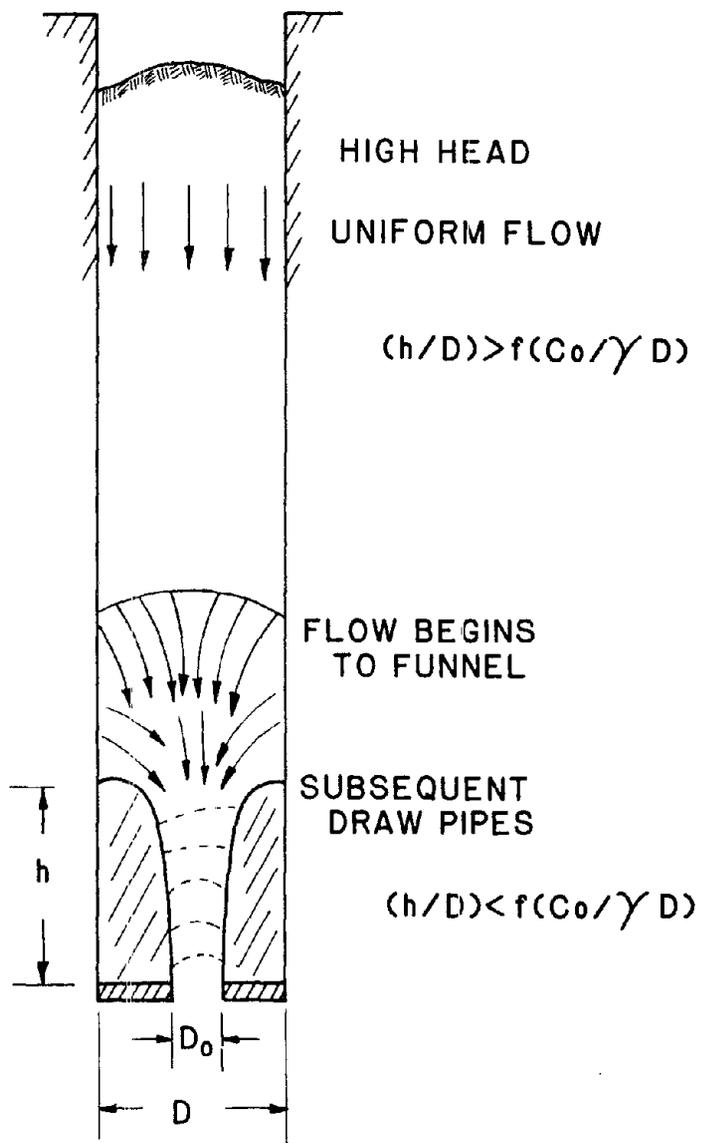


Figure 16 - Hypothetical piping and doming sequence.

flow begins to funnel. If the draw is stopped at this point and then reinitiated without additional material entering the ore pass, piping may occur because of the relatively low head and high material strength. Doming is also possible because of the greatly reduced region for flow. The likelihood of interlocking of large particles is especially high within the pipe.

The conclusion is that ore passes should be left full and operated under high heads. This also applies to wear. Dumping into relatively full ore passes greatly reduces wear and compaction from impact as well. However, these advantages may be negated if the material tends to cement and build up strength with time. In any case, keeping the ore pass active through frequent draws is preferable to low draw frequency and therefore longer draws that almost empty the ore pass.

3.6 Additional Considerations

Although the desired flow of broken rock in ore passes is primarily a question of controlling the forces of friction and cohesion resisting flow and the force of gravity causing flow by proper dimensioning of the ore pass, a number of qualitative as well as quantitative factors influence the balance of forces needed for a properly functioning ore pass system. These include the type and condition of the material handled, the size distribution of the muck, stresses on the ore pass walls, and differential velocities in the ore pass proper.

3.6.1 Type and Physical Condition of Material Handled

The type and physical condition of a material being handled in an ore pass determines its strength and abrasiveness and, therefore, its suitability for being handled in an ore pass system. However, only qualitative relations exist between rock type, physical condition and ease of handling material in ore passes. For example, highly siliceous ores when blasted are generally expected to produce strong, abrasive, brittle rock fragments while calcareous ores are expected to produce relatively soft and not particularly abrasive muck. The same rock type may produce significantly different quantities of fines when blasted and handled because of differences in the degree of local fissuring, microcrack density, and related geological features. A systematic study of such

features of rock in situ for the purpose of assessing the strength and abrasiveness of the blasted product and its suitability for handling in ore passes is difficult and time-consuming and seldom carried out at operating mines. There are also site and system specific factors that have an influence on the physical condition of the muck.

Moisture content, height of drop and frequency of draw or time at rest are examples of factors besides rock type and structure that have a significant influence on the suitability of broken rock for handling in ore pass systems. As a general rule, moisture content, height of drop, and time at rest should be minimized in order to avoid strength buildup and consequent difficulties in handling, and of course, ores that tend to oxidize (mainly sulfide ores with a high pyrite content) or to cement at particle contacts should not be kept stored in an ore pass for prolonged periods of time.

Although moisture additions usually increase cohesiveness, a large amount of water in an ore pass has the opposite effect. Large amounts of water may destroy cohesion by bringing the fines into a slurry state and at the same time may reduce frictional strength through a reduction in the effective normal stress between particles which mobilizes frictional strength. In the extreme case, a full hydrostatic head may develop and a dangerous mud rush ensue.

In any event, quantitative determination of the strength properties of broken rock requires testing under the conditions expected in operation. Direct shear testing in a conventional soil mechanics shear box test is illustrated in Figure 17 which also shows the data obtained in such a test and the strength parameters C_0 (unconfined compressive strength) and T_0 (unconfined tensile strength) as well as ϕ (angle of internal friction) and k (cohesion). The fines and the percentage of fines in the muck determine the cohesiveness of the material being handled while the angle of internal friction of the sand sized fraction is likely to be close to that of the large sized fragments. Therefore it is probably sufficient for engineering purposes to determine ϕ by testing only the sand sized fraction even though very large sized fragments are present in the muck.

The relationship between cohesion of the fines and the cohesion of the muck is less direct. Only if the percentage of fines is high enough to suspend the coarse particles in a matrix of fines is the cohesion of the fines equal to the cohesion of the muck. Otherwise, the

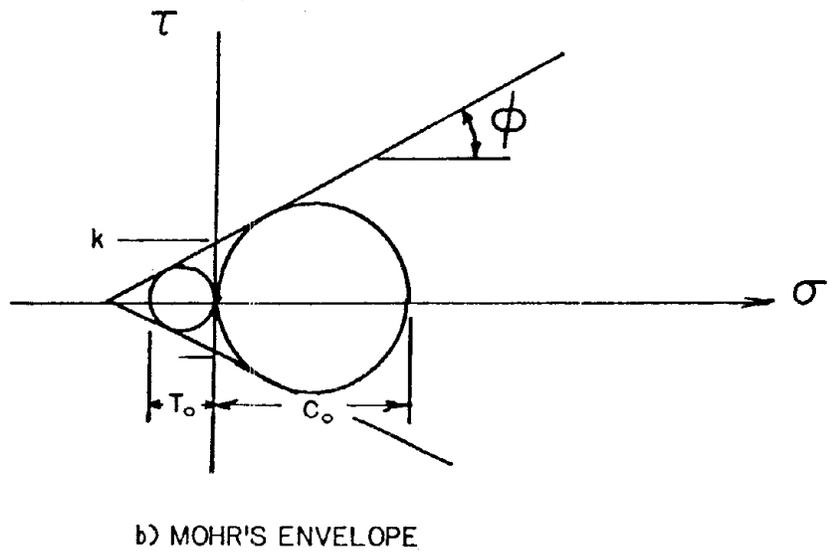
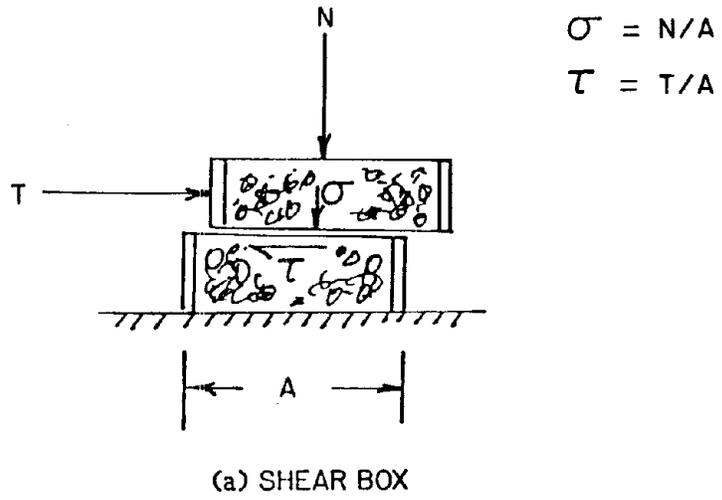


Figure 17 - Shear box test and Mohr's envelope

cohesion of the bulk material is less. In this regard, it is important to avoid segregation of the fines. Segregation may lead to the accumulation of the fines and constriction of the ore pass, especially at changes in inclination and at junctions. Build-up of fines and constriction of the ore pass makes the ore pass susceptible to interlocking arch formation at the constriction and perhaps to eventual cohesive arch formation.

3.6.2 Size of Rock Handled

The distribution of particle sizes that results from blasting is determined by the complex interactions between the rock insitu and the blast design. Well blasted rock contains neither an excessive amount of fines nor an excessive number of oversize boulders that require secondary breakage. A uniform distribution of particle sizes is desirable from the materials handling view while a skip-graded material (one that lacks intermediate size particles) is least desirable. The percentage and size of the largest fragments and the percentage of fines are the most important parameters with respect to handling broken rock in ore passes.

The maximum size of rock that enters an ore pass is advantageously controlled by the use of a grizzly. A properly constructed grizzly will moreover help to straighten the flow and thus reduce ricochets and wear of the ore pass walls. Although some method of secondary breakage must be used in conjunction with a grizzly, the expense is generally justifiable because of the improved reliability and performance of the system. Size distribution of the broken ore or waste rock may be determined in the field by inspection.

3.6.3 Stresses on the Ore Pass Walls

The stresses exerted on the walls and the strength of the walls of an ore pass determine its structural stability and wearability. The current state of stress at any point in the neighborhood of the wall of an ore pass is in turn determined by the mechanical behavior of the wall rock, the pre-excitation stress state, and the type of interaction between the wall rock and the material being handled in the pass. The interaction may be static, quasi-static, or dynamic depending upon the motion of the muck. Between draws the muck is stationary and the interaction is static; during drawdown the interaction is quasi-static, but during dumping and filling, transient loads are generated by impact and the interaction is dynamic.

Structural stability is not likely to be a problem except in instances where the ore pass crosses a fault or zone of exceptionally weak ground because generally the strength of rock is considerably greater than the static and quasi-static stresses exerted on the walls of a pass. Unlined ore passes are subject to plucking and polishing by the flow of muck, but structural failures only occur under unusual circumstances which almost certainly imply adverse geology.

Consider for example, a circular vertical ore pass sunk in rock where tectonic stresses are absent. The vertical pre-excavation stress is then $S_v = \gamma H$ where γ = unit weight of overburden, H = depth to the considered point, and the horizontal pre-excavation stress $S_h = K_o S_v$. A representative value of K_o is 0.33. After excavation the radial stress which acts normal to the empty ore pass walls is zero while the circumferential stress at the pass walls $\sigma_\theta = 2K_o \gamma H$. The vertical stress σ_v after excavation remains equal to γH . Under these circumstances, the factor of safety FS at a point near the ore pass walls where the unconfined compressive strength of rock is C_o is

$$FS = C_o / \gamma H.$$

Even if the rock at some depth H , say $H = 1,000$ ft, is relatively weak, say $C_o = 6,000$ psi, the factor of safety against compressive failure is 6. (No tensile stresses are present).

When the ore pass contains muck, the net result is to increase the local factor of safety of the wall rock through the effect of confinement. The local factor of safety of the wall rock with muck in the ore pass is

$$FS = C_p / \sigma_1$$

where

C_p = the compressive strength under confinement

σ_1 = the major principal stress acting at the wall (a compression)

Generally $C_p > C_o$. The situation is illustrated in Figure 18 which shows a Mohr-Coulomb strength envelope for the wall rock that is characterized by cohesion, k , and internal friction angle, ϕ , and an interface strength envelope depending on adhesion, k' , and friction angle, ϕ' between muck and wall rock. In Figure 18 the strength

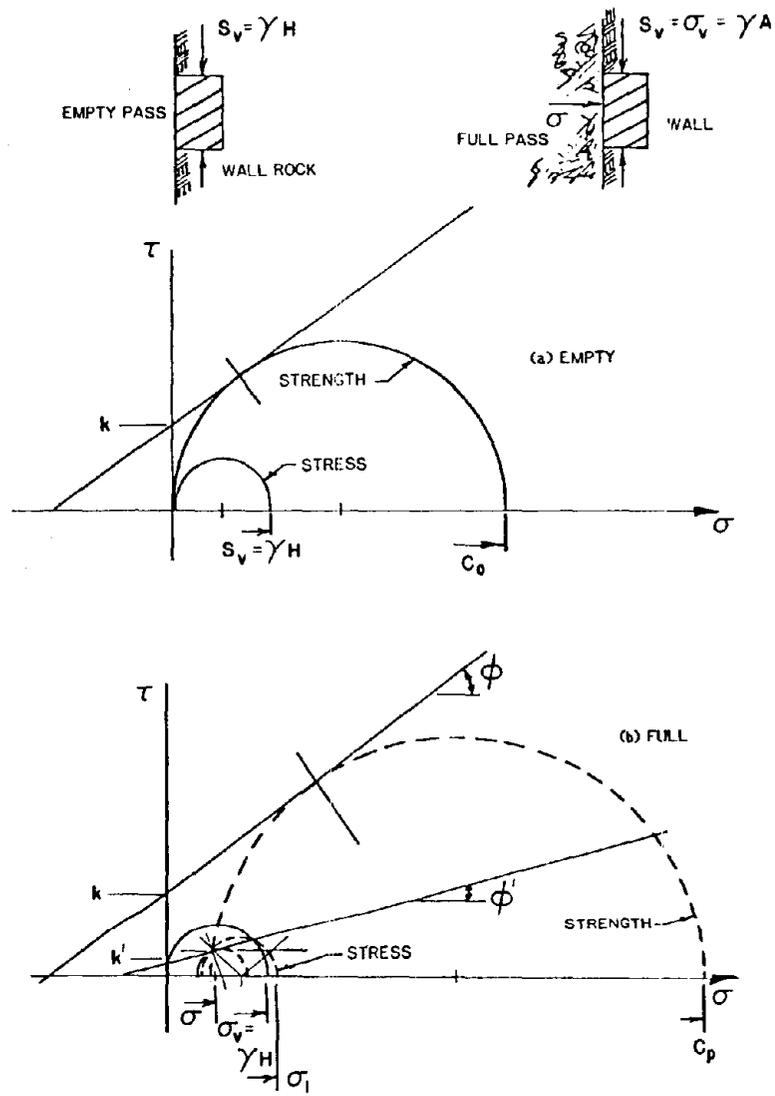


Figure 18 - Stress and strength of ore pass walls
 (a) empty
 (b) full

and stress circles that represent C_0 and S_v (γH) of the wall rock of the empty ore pass are shown in solid line while the dashed line circles which give C_p and σ_1 represent strength and stress when the ore pass is filled to the collar. The dotted circle represents the state of stress in the muck adjacent to the wall rock when slip between muck and wall impends.

Stresses generated by impact of rock fragments on the walls of an ore pass during fall of muck down the pass are not calculable as a practical matter. However, some insight concerning the dynamics of falling muck in ore passes can be obtained by trajectory calculations of individual rock fragments. Such calculations also provide some insight concerning wear of ore pass walls to the extent that wear is related to impact abrasion. These calculations are original to this report and enable one to estimate points of impact and to assess the influence of ore pass width or diameter and inclination on rock fragment trajectory. Trajectory here is used in a broad sense and includes the calculation of the acceleration, velocity, position, linear momentum, and translational kinetic energy of a typical rock fragment mass center during its fall down the pass.

Suppose the muck is delivered to an empty ore pass by an end dump truck and slides from the tilted truck box at an angle α from the vertical with a speed v into an ore pass inclined from the vertical at an angle β as shown in Figure 19. The trajectory of a freely falling rock fragment is described parametrically by the functions $x(t)$ and $z(t)$ which give the coordinates of the particle's position at time t measured from the instant the fragment leaves the truck box and enters the ore pass.

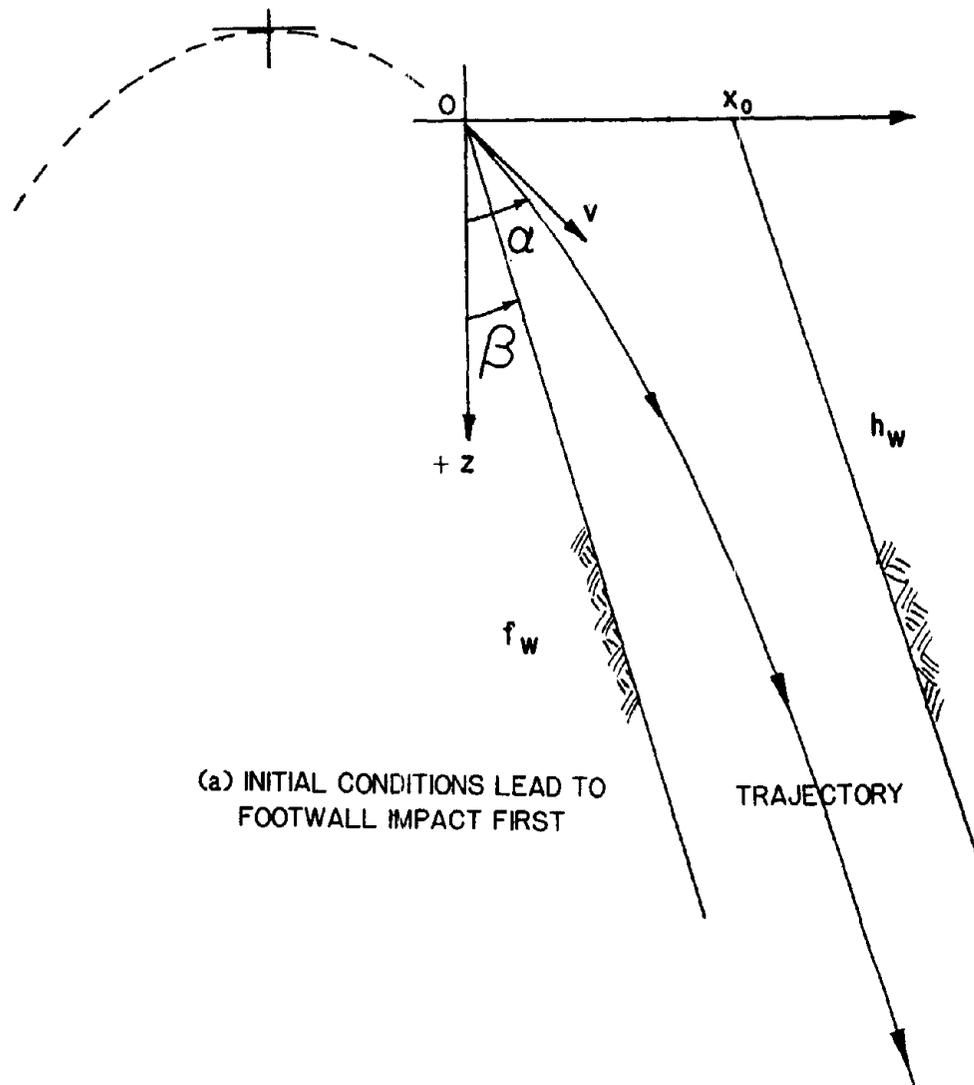
From Newton's second law of motion

$$\begin{aligned} \ddot{x} &= 0 & \ddot{z} &= g \\ \dot{x} &= v \sin (\alpha) & \text{and} & \dot{z} = gt + v \cos (\alpha) \\ x &= [v \sin (\alpha)]t & & z = gt^2/2 + [v \cos (\alpha)]t \end{aligned}$$

where: v = speed of the particle as it enters the ore pass

t = time measured from the instant the particle enters the ore pass

α = angle from the vertical of the fragment velocity at the moment it enters the pass



(a) INITIAL CONDITIONS LEAD TO FOOTWALL IMPACT FIRST

TRAJECTORY

Figure 19(a) - Rock fragment trajectory down an inclined ore pass following dumping on the footwall (fw) side. Hangingwall is hw

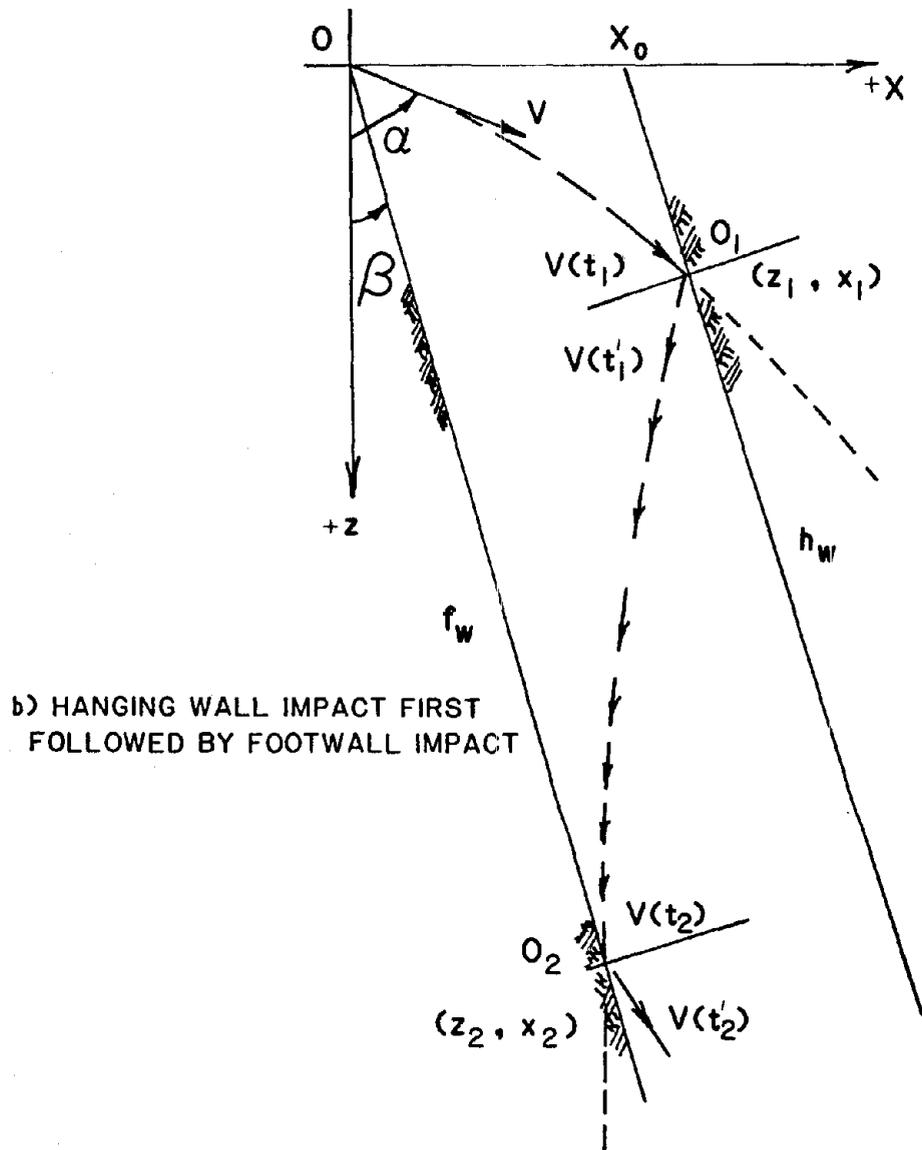
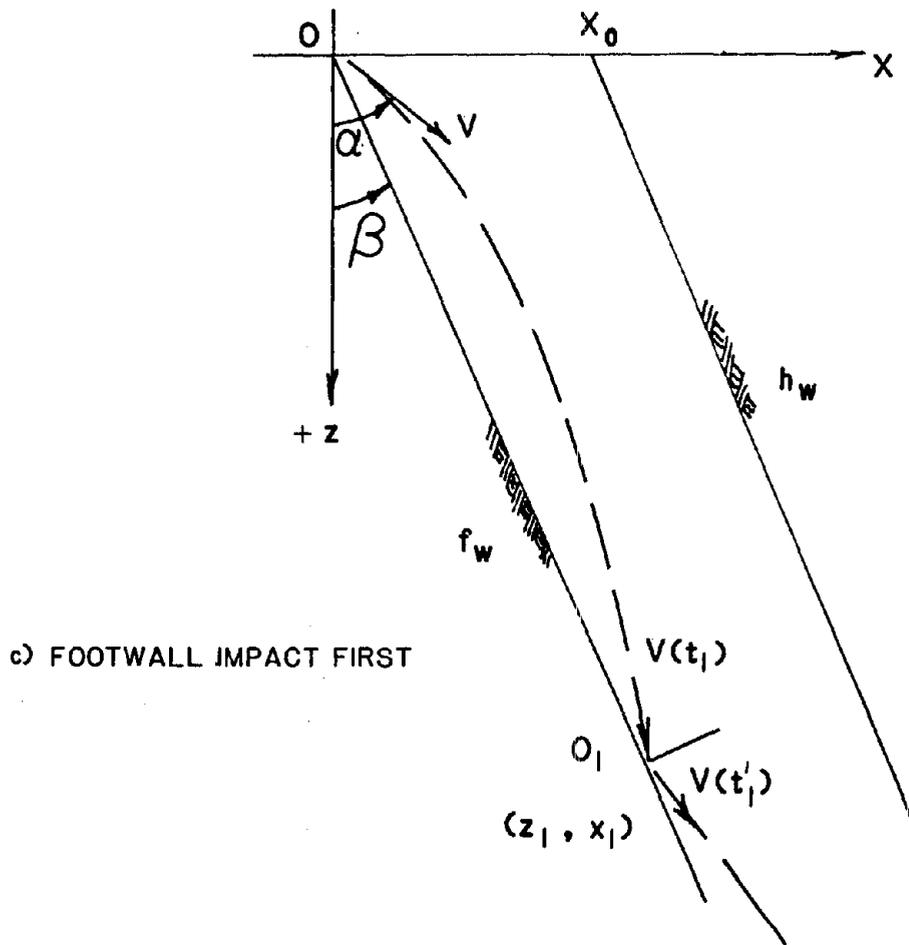


Figure 19(b) - Rock fragment trajectory down an inclined ore pass following dumping on the footwall (fw) side. Hangingwall is hw.



c) FOOTWALL IMPACT FIRST

Figure 19(c) - Rock fragment trajectory down an inclined ore pass following dumping on the footwall (fw) side. Hangingwall is hw.

g = acceleration due to gravity

\ddot{x}, \ddot{z} = horizontal and vertical accelerations

\dot{x}, \dot{z} = horizontal and vertical velocities

x, z = horizontal and vertical coordinates of a rock fragment during fall down the ore pass.

The linear momentum and translational kinetic energy of any freely falling rock fragment are

$$\begin{aligned} P_x &= (W/g)\dot{x} & P_z &= (W/g)\dot{z} \\ ke_x &= (W/2g)(\dot{x})^2 & ke_z &= (W/2g)(\dot{z})^2 \end{aligned}$$

where

P_x, P_z = horizontal and vertical momentum components

ke_x, ke_z = kinetic energies associated with the horizontal and vertical components of velocity

W = weight of the considered rock fragment.

After eliminating t from $x(t)$ and $z(t)$ one obtains $z(x)$, thus

$$z = (g/2) [x/v \sin(\alpha)]^2 + x/\tan(\alpha).$$

Hence,

$$\begin{aligned} \left[x + \frac{(v \sin(\alpha))^2}{g \tan(\alpha)} \right]^2 &= z \left[\frac{2(v \sin(\alpha))^2}{g} \right] + \\ &\left[\frac{(v \sin(\alpha))^2}{g \tan(\alpha)} \right]^2 \end{aligned}$$

which is a parabola with axis parallel to the vertical z -axis as shown in Figure 18. This may be better seen by setting $(v \sin(\alpha))^2 = g$ and $\alpha = 45^\circ$, the trajectory then is

$$(x + 1)^2 = 2z + 1$$

or

$$x = -1 + (2z + 1)^{1/2}$$

which is clearly a parabola with vertex at $x = -1, z = -1/2$.

The trajectory of the rock fragment may or may not carry it to the hanging wall of the ore pass. The equations of the hanging wall and foot wall shown in Figure 19 are

$$x = z \tan (\beta) + x_0 \quad \text{and} \quad x = z \tan (\beta)$$

where β is the ore pass inclination (+ counterclockwise from vertical) and x_0 is the ore pass width or diameter. If the hanging wall line intersects the trajectory then hanging wall impact occurs as illustrated in Figure 20. This graphical result is equivalent to the requirement that the system of equations describing the hanging wall and particle trajectory have a real solution.

Thus,

$$z = \left\{ \frac{g}{2[v \sin (\alpha)]^2} \right\} (z \tan (\beta) + x_0)^2 + \left[\frac{(z \tan (\beta) + x_0)}{\tan (\alpha)} \right]$$

must have a real, positive root when rewritten as a quadratic in z , that is, the equation

$$az^2 + bz + c = 0$$

where

$$a = (g/2) \left[\frac{\tan (\beta)}{v \sin (\alpha)} \right]^2$$

$$b = \left[\frac{gx_0 \tan (\beta)}{(v \sin (\alpha))^2} + \frac{\tan (\beta)}{\tan (\alpha)} - 1 \right]$$

$$c = \left[x_0 / \tan (\alpha) + \frac{gx_0^2}{2(v \sin (\alpha))^2} \right].$$

must have real, positive roots.

Alternatively,

$$\left[\frac{x - x_0}{\tan (\beta)} \right] = (g/2) \left[\frac{x}{v \sin (\alpha)} \right]^2 + \frac{x}{\tan (\alpha)}$$

that is,

$$ax^2 + bx + c = 0$$

(d) FOOTWALL IMPACT FIRST

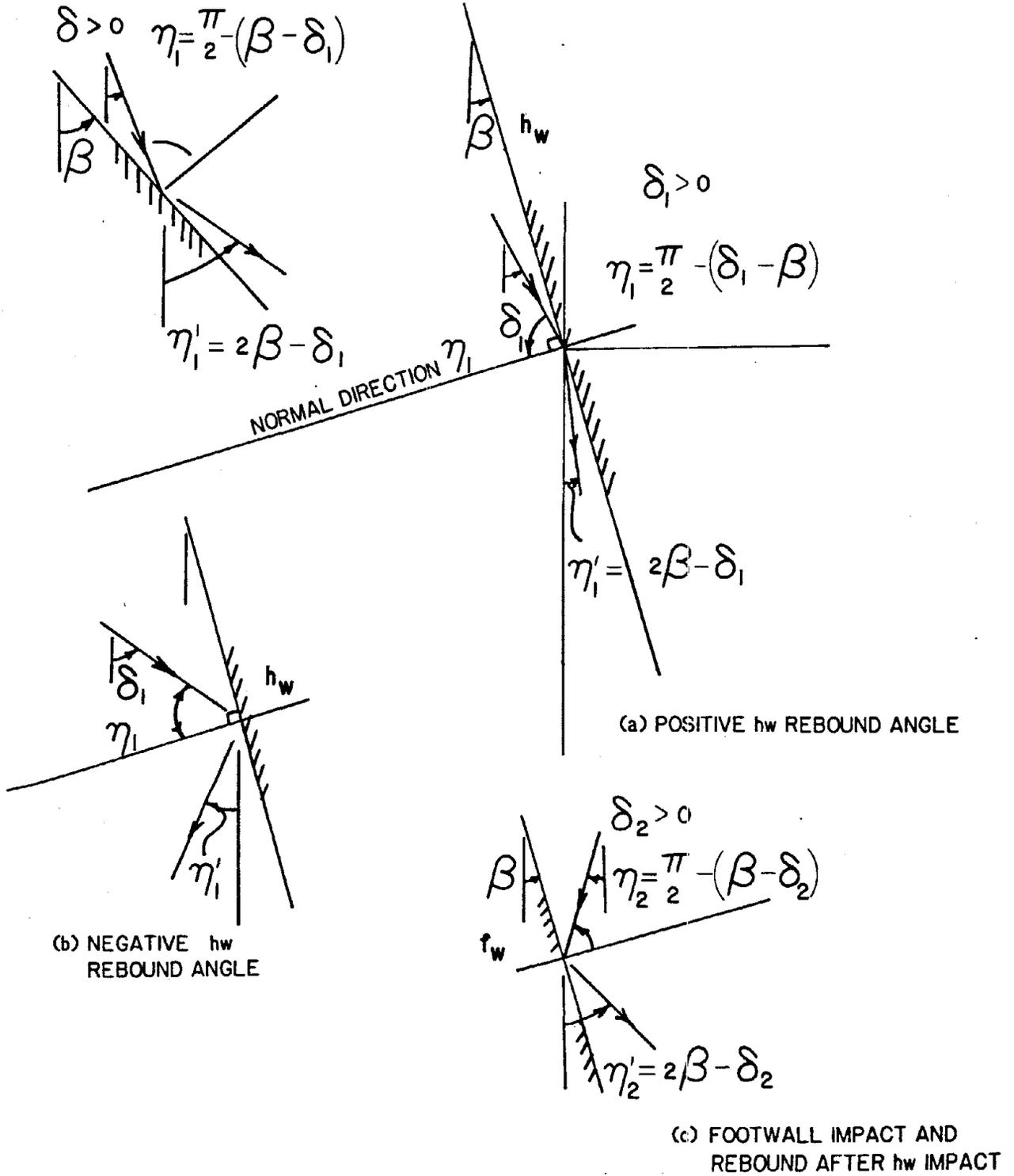


Figure 20 - Impact and rebound angles

where

$$a = (g/2) \left[\frac{1}{v \sin(\alpha)} \right]^2$$
$$b = \frac{1}{\tan(\alpha)} - \frac{1}{\tan(\beta)}$$
$$c = \frac{1}{\tan(\beta)}$$

must have real, positive roots.

If hanging wall impact does occur, say, at $x = x_1$ and $z = z_1$, then the time of fall is

$$t_1 = \frac{x_1}{v \sin(\alpha)}$$

and the angle of impact with respect to the vertical is

$$\delta_1 = \tan^{-1} \left[\frac{v \sin(\alpha)}{gt_1 + v \cos(\alpha)} \right]$$

The angle of impact relative to the normal direction to the hanging wall is

$$\eta_1 = (\pi/2) - (\delta_1 - \beta)$$

as shown in Figure 20. If the angle of rebound relative to the normal direction is equal to the angle of impact (this conserves linear momentum and kinetic energy), then the direction of rebound relative to the vertical is

$$\eta_1' = 2\beta - \delta_1$$

which may be positive or negative (counterclockwise angles are positive). The horizontal and vertical components of velocity upon rebound are

$$\dot{z}_1' = v \sin(\alpha) \cos(2\beta - \delta_1) / \sin(\delta_1)$$

$$\dot{x}_1' = v \sin(\alpha) \sin(2\beta - \delta_1) / \sin(\delta_1)$$

where the prime (') indicates the rebound side of the time of impact t_1 .

After the first impact (in this case, against the hanging wall after dumping from the footwall side):

$$\begin{aligned}
 \ddot{x} &= 0 & \ddot{z} &= g \\
 \dot{x} &= \dot{x}_1' & \dot{z} &= gt + \dot{z}_1' \\
 x &= \dot{x}_1' t & z &= (gt^2/2) + \dot{z}_1' t \\
 P_x &= (W/g) \dot{x} & P_z &= (W/g) \dot{z} \\
 ke_x &= (W/2g) (\dot{x})^2 & ke_z &= (W/2g) (\dot{z})^2
 \end{aligned}$$

where x , z , and t are now measured from the instant and place of the first impact. The new trajectory followed by the considered rock fragment is

$$\left[x + \left(\frac{\dot{x}_1'}{g \tan(2\beta - \delta_1)} \right) z \right]^2 = z \left[\frac{2(\dot{x}_1')^2}{g} + \left[\frac{(\dot{x}_1')^2}{g \tan(2\beta - \delta_1)} \right]^2 \right]$$

If hanging wall impact is indeed the first impact, then the second impact is necessarily against the footwall. The footwall relative to the new origin (point of first impact) is

$$x = z \tan \beta - x_0.$$

The time between first and second impacts is t_2 , and the location of the second impact is at (z_2, x_2) where

$$z_2 = (gt_2^2/2) + \dot{z}_1' t_2, \quad x_2 = \dot{x}_1' t_2.$$

The total vertical drop of the rock fragment is $z_1 + z_2$.

The angle of second impact in this case is

$$\delta_2 = \tan^{-1} \left(\frac{\dot{x}_1'}{gt_2 + \dot{z}_1'} \right)$$

measured from the vertical. The sign of δ_2 may be positive or negative depending on the sign of \dot{x}_1' . The angle of impact relative to the normal direction to the footwall is

$$\eta_2 = (\pi/2) - (\beta - \delta_2)$$

and the angle of rebound relative to the vertical is

$$\eta_2' = 2\beta - \delta_2.$$

The analysis can now be restarted with the origin moved to this last point of impact; the new trajectory can be determined and the equations for foot and hanging walls rewritten. If a real solution to the system of trajectory and hanging wall equations exists, then hanging wall impact is indicated next, and the process can again be repeated from that point as before.

However, if the first impact is not against the hanging wall, then first impact is necessarily a footwall impact. The calculation of the point of first impact (z_1, x_1) then proceeds as for hanging wall impact with the exception that $x_0 = 0$. The angles of impact and rebound are given by the same relationships as for footwall impact second, only in this case $\beta > \delta_1$ always as shown in Figure 20.

Thus the complete trajectory of a free falling rock fragment as it ricochets down an ore pass can be calculated provided the velocity, v , at entrance is known.

The velocity at entrance can be estimated by considering the circumstances of the dumping operation. Suppose the truck box is inclined at some angle to the vertical such that the broken rock in the truck slides from the truck into the ore pass. The equation of motion for a sliding rock fragment is

$$(W/g)a = W \cos (\alpha) \tan (\phi)$$

where

W = weight of the considered fragment

a = acceleration parallel to the truck box

α = inclination of the truck box from the vertical

ϕ' = angle of sliding friction between truck box and rock

g = acceleration due to gravity.

Thus,

$$a = g \cos (\alpha + \phi') / \cos(\phi')$$

$$v = at + v(0)$$

$$s = at^2/2 + v(0)t + s(0)$$

where

v = velocity of the rock fragment

s = distance traveled.

If the truck box is angled, then two applications of the above procedure are needed to obtain the velocity with which a rock fragment enters the ore pass.

The preceding analysis of rock fragment trajectory assumes that dumping takes place from the footwall side of the ore pass. Fortunately, there are no essential differences in the analysis procedure if dumping occurs at the hanging wall side. Of course, if dumping occurs at the hanging wall side of the ore pass, then there is no question but that first impact occurs against the far side, that is, the footwall side of the pass. An interesting possibility of updip rebound as shown in Figure 21.

In any case, the impact forces exerted on the ore pass walls are associated with the change in momentum of the falling rock fragments during impact against the walls. If the angle of impact and rebound are equal, then no change in velocity parallel to the ore pass wall occurs. Only the momentum normal to the wall changes; the change is twice the normal momentum component carried by the considered fragment at the instant of impact. The impulse is

$$\int_{t_1}^{t_1'} F_n dt = \int_{t_1}^{t_1'} (W/g) dv_n$$

so that

$$\bar{F}_n \Delta t = 2(W/g) v_n$$

where

\bar{F}_n = average force acting normal to the ore pass wall during impact and rebound

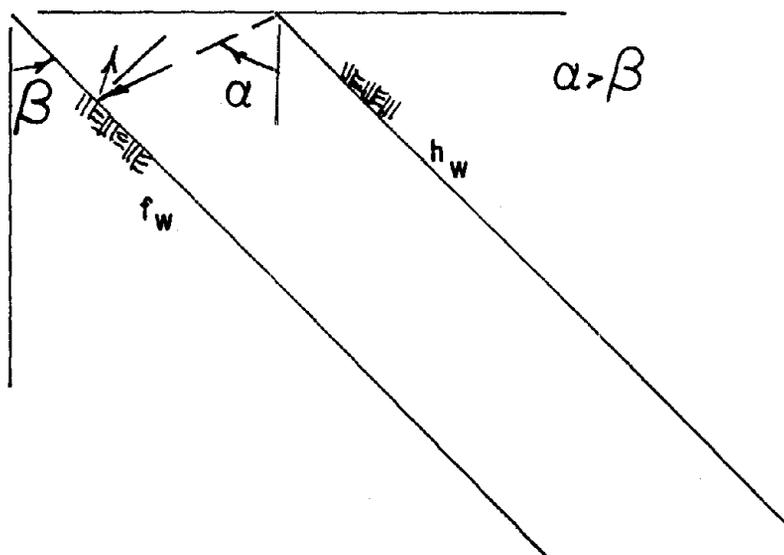


Figure 21 - Hanging wall dumping leading to updip rebound following footwall impact

Δt = duration of the impact and rebound process

v_n = velocity component normal to the ore pass wall prior to impact.

The normal component of velocity is

$$v_n = v \cos (\eta) = v \sin (\beta - \delta)$$

where

$$v = [(\dot{x})^2 + (\dot{z})^2]^{1/2}$$

The average impact force is therefore directly proportional to the rock fragment weight and velocity and inversely proportional to the duration of impact. The stress associated with F_n would be inversely proportional to the area of impact. Thus a rock fragment that strikes flat against the ore pass wall generates a relatively low stress compared with a nearly point contact impact which may generate stresses sufficiently large to cause local failure and abrasion of the wall and fracture of the fragment as well. The transient stresses, waves generated at impact, are rapidly dissipated away from the ore pass wall; local reflections would not be expected to cause tensile slabbing or structural failure of the wall. Impact-induced wall failure is therefore generally restricted to very localized crushing which comes under the classification of wear.

In the paper by the Panzihua Mines Company Planning Department the comment is made that vertical ore passes are more impact wear prone than are inclined ore passes. However, the only information made available in this paper is a diagram showing the eroded ore pass wall profiles of a vertical and an inclined pass. The former is symmetrical while the latter is asymmetrical and shows more wear on the footwall than on the hanging wall. The maximum wear appears to be about the same, so that there appears to be no dramatic differences between the two. In addition, the wear profiles are what one would expect from abrasion by sliding of the muck along the walls. It could well be that sliding dominates the wear process under some conditions while impact is dominant under other conditions. If wear control is important, then clearly a better understanding of the wear process would be helpful. However, wear that does not threaten the structural safety of the ore pass walls may not be of concern, and indeed may promote flow in the pass by enlarging and smoothing it.

Although each set of conditions represents a different case and the analyses presented here serve only to outline the steps necessary to the determination of rock fragment trajectories in ore passes, some general observations are still possible at this early stage of investigation:

- (1) wall impact can always be prevented by keeping the pass sufficiently filled with muck,
- (2) in vertical ore passes, impact against the wall opposite the dump side is sure to occur if the ore pass is empty and sufficiently deep,
- (3) in vertical ore passes, the distance between impacts and the angle of impact (measured from the normal) increases with fall distance,
- (4) in vertical ore passes, the severity of impact measured by normal components of momentum and kinetic energy is independent of fall distance, but is directly proportional to the weight of a freely falling rock fragment,
- (5) in inclined ore passes, footwall impact which may be first or second always occurs in empty ore passes sufficiently deep,
- (6) in inclined ore passes, if the first impact is not a hanging wall impact, then no subsequent hanging wall impacts will occur from footwall rebound, provided the ore pass inclination is less than one-half the entrance angle ($\beta < \alpha/2$),
- (7) in inclined ore passes, even if the first impact is a hanging wall impact (implies footwall dumping), hanging wall impacts will eventually cease in an empty, sufficiently deep ore pass.

Further analysis would no doubt reveal more generalities, but the procedure presented here is adequate for design insight because it reveals the basic relationships between the variables and parameters involved including those under the control of the design engineer, for example, ore pass width x_0 .

3.6.4 Differential Velocities in Ore Passes

Although measurements of velocities in actual ore passes have not been made, small laboratory scale sand models (Pariseau, 1966) indicate that velocity is uniform across the ore pass away from the outlet, provided the height of muck in the ore pass is relatively high. In the vicinity of the outlet, the flow becomes non-uniform and as drawdown proceeds, a point is reached where the flow is likely to funnel. Funnel flow carries with it a risk of pipe formation and complete drawdown removes the protective cushion of muck usually present over the loading chutes. Excessive drawdown should therefore be avoided.

A crude method of estimating the lowest height of muck that should be left to reduce the chances of funneling is to assume that the influence of the outlet extends to a height above the outlet between one and two times the ore pass diameter D . Muck height H left in the ore pass should therefore be between D and $2D$ as shown in Figure 22a. An alternative rule of thumb that accounts for the effect of the outlet width D_o is shown in Figure 22b where

$$H = (D - D_o) (3/4)^{1/2} \text{ if } \phi = 30 \text{ deg.}$$

The velocity of the muck flow must increase near the outlet in order to conserve mass in the process. Under steady flow conditions, the average mass flow rate across any section of the ore pass must be constant, that is

$$\dot{m} = \rho AV = \text{constant}$$

where

$$\dot{m} = \text{mass flow rate}$$

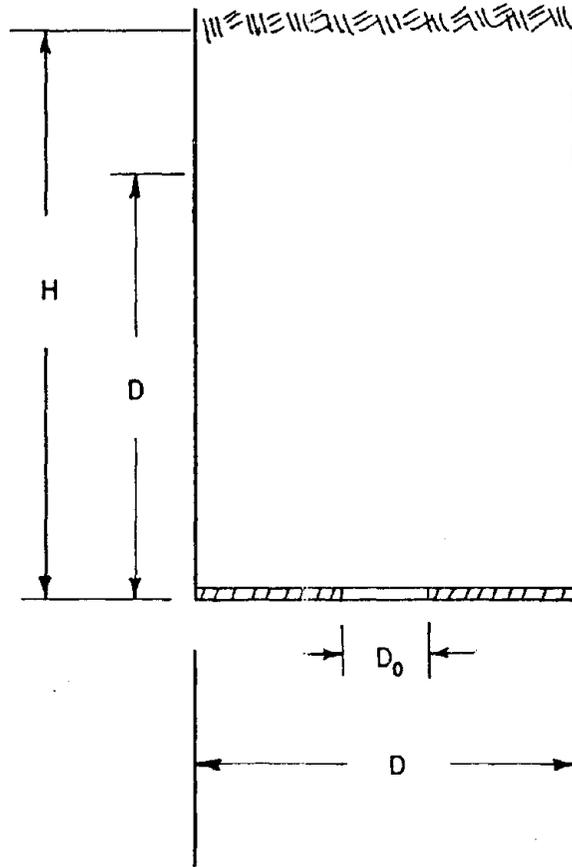
$$\rho = \text{mass density (of the bulk muck)}$$

$$A = \text{cross sectional area of the ore pass}$$

$$V = \text{average velocity normal to } A.$$

Hence the decrease in cross sectional area available for flow that usually occurs with approach to the outlet results in an increase in velocity. This increase may be augmented somewhat by a decrease in the density of the muck as it "loosens" with approach to the outlet. Although it is unlikely that the bulk density of the muck will increase in the outlet region during drawdown, the loosening effect may be impeded if the feed rate from the ore pass is less than the unimpeded gravity flow rate achievable in the ore pass.

(a)



(b)

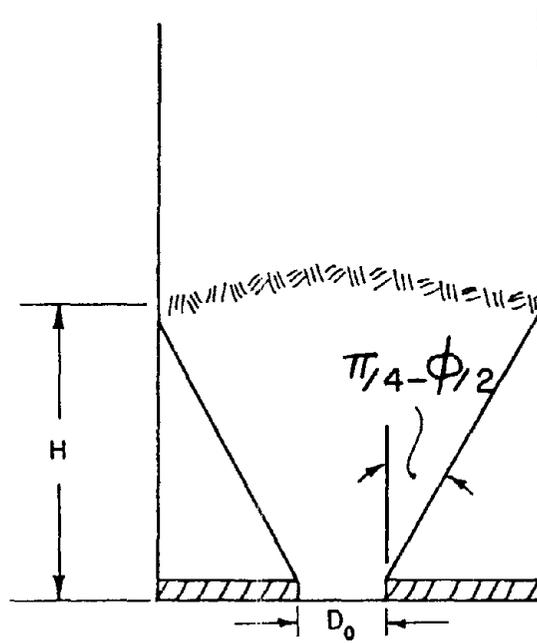


Figure 22 - Minimum drawdown height H to prevent funnel flow.

The increase in average velocity with approach to the outlet is generally accompanied by the development of velocity gradients across the ore pass as the muck in the central region of the ore pass accelerates more than the muck adjacent to the ore pass walls where the frictional retardation is larger. Differential velocities are also likely to develop at ore pass junctions, branches and bends as well as in the inlet and outlet regions where changes in the average flow rate must occur because of changes in cross section.

Consider the ore pass junction shown in Figure 23a. If the three branches have the same cross sectional area, then for practical purposes the average velocities must be related by

$$V_1 + V_2 = V_3$$

which quite clearly shows that the flow in branch 3 restricts the flows in branches 1 and 2. Conceivably the flow in branch 1 could cease ($V_1 = 0$) while flow proceeds with $V_2 = V_3$. It is also possible that flow in branch 2 could cease ($V_2 = 0$) while flow from branch 1 proceeds with $V_1 = V_3$. However, the latter requires a sharp change in the direction of flow and is therefore associated with a potential for segregation and fines build-up.

If the cross sectional areas are not equal, then

$$V_1 A_1 + V_2 A_2 = V_3 A_3.$$

In Figure 23b the flow branches from 3 to 1 and 2. This arrangement is usually conducive to flow and hang-up avoidance in the main pass 3.

A change in either the magnitude or direction of the average velocity of muck flow almost certainly involves the development of velocity gradients and therefore differential velocities over the cross-section of interest. Definitive data are lacking in this regard, but it nevertheless seems likely that gravity flows of muck in ore passes are characterized by an increase in speed on the inside of an ore pass bend and a decrease on the outside relative to the average velocity as illustrated in Figure 23c. The slowdown on the outside of the bend or "knuckle" where the hanging wall becomes

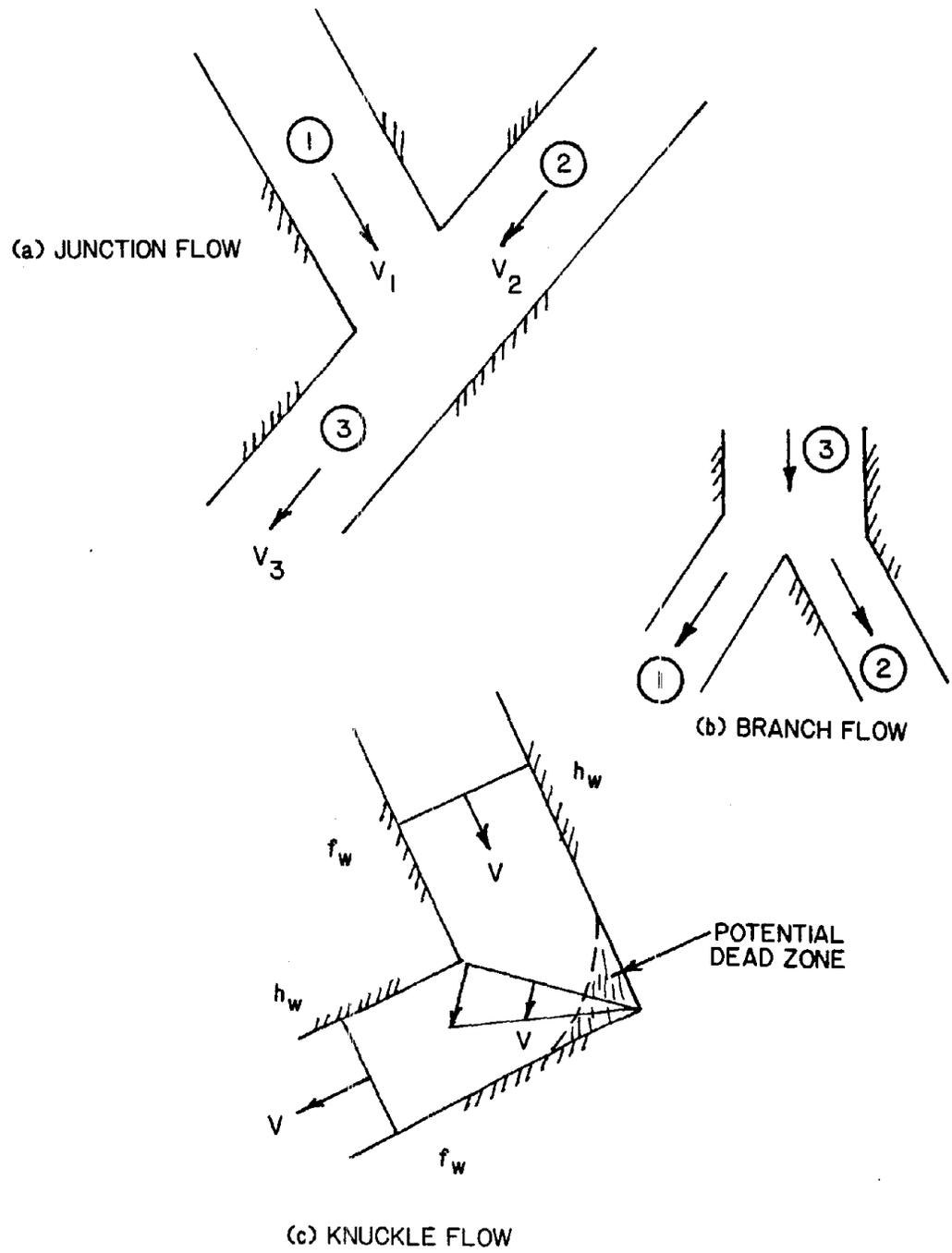


Figure 23 - Velocity changes at junctions, branches and bends

the footwall and the footwall becomes the hanging wall may lead to dead zone formation with a fines build-up and narrowing of the ore pass and a consequent increase in the probability of arch formation either by continued fines buildup or boulder interlocking.

Dead zones are not always detrimental. In fact, well planned dead zones may reduce wear and maintenance in areas of potentially high impact and abrasion. Of course, the physical properties of the material must be suitable for dead bed utilization. For example, very cohesive muck may continue to build-up beyond the design limits of the sought dead bed and become an impediment to proper functioning of the pass. Knuckles also serve to reduce impact and loads on chutes. Unfortunately observations concerning the flow of muck in ore passes, around bends and in the outlet region are only qualitative. The lack of experimental data precludes quantitative design analysis that would permit a more reliable dimensioning of features such as knuckles and dead beds in ore pass layouts.

3.7 Field Studies and Observations

This section is concerned with comparisons between ore pass design theory and actual practices observed in operating mines. The primary purpose of these comparisons is to test the design criteria developed previously for omissions and oversights and thus to ascertain the effect on design of rock flow in vertical and inclined ore passes. The situation is, of course, circular in that design criteria are developed within a context of experience and knowledge of ore passes in the first place. It is therefore not surprising to find that design criteria intended to insure reliable flow of broken rock in ore passes address the major impediments to flow, namely, hang-ups and piping. In fact, the essential features of ore pass design for flow and storage are qualitatively largely a matter of common knowledge.

Hang-ups occur because of interlocking of large boulders and the stickiness of the fines. The accumulation of fines at bends or on dead beds also leads to hang-ups and reduced storage capacity as well. The obvious way to avoid hang-ups or at least to reduce their frequency is to use wide ore passes. Moisture may increase the cohesion of fines relative to the dry state but large amounts of water may lead to mud rushes and flooding. Water entering the ore pass should therefore be minimized. Wear is often

greatest at the point of first impact near the dump site where impact frequency is high; while leaving a cushion of muck over loading facilities at the bottom of an ore pass provides a measure of protection against long muck falls down empty ore passes.

Quantitative ore pass design is quite another matter because the mechanics of rock flow in ore passes which forms the basis for design is complex. Equilibrium must be satisfied always, but the interactions between particle size, scale, and cohesion are not well understood. Very little experimental work has been undertaken in this regard, so that the design criteria developed in this report are a mixture of equilibrium analyses and empirical observations. These criteria still require some knowledge of the muck being handled, namely strength properties, size distribution and weight.

Material properties testing, however, is well beyond the scope of the project. The field studies and observations presented here are therefore qualitative, but nonetheless serve to illustrate ore pass usage and practice. It should be noted in this regard that almost all the ore passes available for study are in underground mines whereas the project objective relates to the use of ore passes in conjunction with open pit mines that generally produce larger quantities of muck containing much larger rock fragments after blasting. Fortunately, analysis of the scale-up question presented by this situation shows that the larger muck in the larger open pit ore passes should be more flowable. The basic reason for this is that the gravity forces causing flow increase faster than the cohesive forces resisting flow as the dimensions of the system are increased.

This section contains a broad sampling of ore pass operating data that are presented in table form. The tables in the first part of this section relate to ore pass practice in underground mines; the second part of this section pertains to open pit mine ore passes. Specific sources of table data are given in the reference list at the end of this report in Section 8.0

3.7.1 Underground Ore Passes

Data relating to ore pass practice in underground mines are presented in Tables 5 and 6. Table 5 contains data obtained during visits to operating mines, while Table 6 contains additional data gleaned from articles in the technical literature. The range of mine parameters including commodity, mining method, and so forth that is encompassed

TABLE 5 - Ore Pass Data - Underground Mine Visits

Mine	Ore Pass						Notes	
	(1) Commodity (2) Method (3) Production (tpd)	(4) Dimensions (ft) (5) Inclination (ft) (6) Length (ft)		(7) Lining (8) Grizzly or (in) Max Chunk Size	(9) Bends or Knuckles (10) Branches		Operator Comments	
U, R & P. 10,000 (tpd)	7' x 7'	57°	525'	no	12" x 24" G	no	no	14 month hang-up, need steeper and larger pass
Fe, R & P. 9,600	8 x 8	65	771	no	12 x 12	yes	fingers	some oversize hang-ups, knuckle wear
	11 x 11	70	225	no	12 x 12	yes		
Cu, Bl. Cav. 65,000	4 x 4	62	55	armored cribbing concrete and rail	14 x 14 G	yes	fingers	some oversize hang-ups
	8.5 x 20	55	150		14 x 14 G	yes		
Cu, R & P. 3,000	8' Diam.	Vert.	300	concrete and steel plate	17 x 17 G	yes	at levels	wear at shape changes
Fe, R & P. 6,300	9 Diam.	≥ 70	200	no	36 x 36	yes	fingers	some oversize hang-ups and runs of wet fines
Pb, Zn, R & P. 6,500	≥ 6 x 6	Vert.	100	no	24 x 24	no		some hang-ups from boulder interlock
Pb, Zn, R & P. 4,000	≥ 7 x 7	Vert.	70	no	24 x 24 G	no		some arching or boulders
Cu, VCR. 9,000	6 to 8' Diam.	73	771	no	18 - 24	no		high fines, cohesive arching with sticky material
W, O.S. & Shr. na	5' Diam. 7 x 7	55 to Vert.	1,500	no	24 - 36	yes	yes	wet, sticky ore, fines hang-ups

na = not available

- (1) U = uranium, Fe = iron, Cu = copper, Pb = lead, Zn = zinc, W = tungsten
- (2) R & P. = room and pillar, Bl. Cav. = block caving, C & F = cut and fill, VCR = vertical crater retreat, O.S. = open stope, Shr. = shrinkage stope, O.P. = open pit
- (3) tpd = tons per day
- (4) cross-sections are rectangular including square as a special case, Diam. = diameter when circular
- (5) measured from the horizontal
- (6) generally refers to vertical drop
- (7) lining excludes rock bolting and gunite, refers to heavy liners
- (8) G = grizzly
- (9) excludes consideration of chutes where a bend is always present, refers to a deliberate bend or knuckle ahead of a chute or to
- (10) a junction of an ore pass as in finger raises and gathering raises or when several levels feed a main ore pass.
- (11) references cited in Section 8.0.

TABLE 6 - Ore Pass Data - Underground Mine Literature

Mine	Ore Pass			Notes
(1) Commodity (2) Method (3) Production (tpd)	(4) Dimension (5) Inclination (deg) (6) Length (ft)			(11) References
Cu, Bl. Cav. 57,000	5' x 5' 6.5 x 6.5 16.4' Diam.	55 ^o - 59 ^o	200'	gathering and transfer ore passes, main ore passes, offset every 197' (59,61,19)
Ag, Pb, Zn, C. & F. 9,000	6' Diam. 10 x 12 8 x 8	67 - 70	350	ore passes, access raises, and ventilation raises, low fines (33,13)
Cu, Bl. Cav. 23,000	5 x 5 6.5 x 6.5	55 58, 60	56 500	maybe concrete lined, 30" grizzly, branch raises and long raises (18)
Cu, Zn, Bl. Cav. na	8 x 8	62 - 66	670	separate passes for Cu and Cu-Zn ores, 70 ^o circular passes planned to reduce hang-ups of high sulphide ores (8)

(11) References cited in Section 8.0

by Tables 5 and 6 is considerable. Although perhaps not statistically representative, the data in these tables are indicative of operating practice in underground mines. As pointed out previously (Pariseau, 1966), cross-sectional shapes are usually square, rectangular or circular and only occasionally exceed 60 ft² in area. However, there is a trend toward larger ore passes in large mechanized underground mines using load-haul-dump equipment. Lengths vary considerably depending on the level interval, typically 150 ft, and the number of levels served by a main ore or rock pass. Inclinations are usually over 60°. Linings are not generally used, although there are noteworthy exceptions. However, reinforcement and wear plates are almost always required in places, for example, at the brows of chutes. Interestingly, hang-ups caused by oversize are often noted where the ratio of ore pass dimension to maximum chunk size is noticeably less than five. However, the most troublesome hang-ups occur where a large percentage of cohesive fines are present.

Ore passes are generally straight, although knuckles and dead bed offsets, which serve the same function, are sometimes used at level dump points feeding main ore passes. Finger and gathering raises in block caving systems also involve a decided knuckle effect in that sharp changes in direction of flow are required at their junctions.

3.7.2 Open Pit Mine Ore Passes

Data relating to ore pass practice in open pit mine haulage systems are presented in Tables 7 and 8. Table 7 contains examples of operating practice presented in the technical literature including some examples presented previously in the benchmark papers by Pfleider and Dufresne (1961) and Selleck and Pfleider (1968). Table 8 is taken from a recent paper by Li et al (1981) on the subject of open pit mine ore pass haulage systems that are used extensively in China.

Ore passes used in conjunction with open pit mine haulage are much larger than those used in underground mines. Linear dimensions are two to three times as great; cross-sectional areas are accordingly four to nine times larger. Muck size is also larger in open pit mine ore pass systems; boulders up to 5 ft in run of the mine rock are handled routinely. The largest such ore passes are usually circular, vertical and unlined. Hang-ups appear to be confined to the vicinity of the loading facilities where

TABLE 7 - Ore Pass Data - Open Pit Mines

Mine	Ore Pass						Notes
(1) Commodity (2) Method (3) Production	(4) Dimension (ft) (5) Inclination (ft)	(5) Inclination	(6) Length (ft)	(7) Lining (8) Grizzly or (in) Max Chunk Size	(9) Bends or Knuckles (10) Branches	Comments (11) References	
Fe, O.P. 55,000 (tpd)	27.5' x 27.5'	Vert.	260' - 800'	no 60" x 60" G	yes	wear at knuckle, oversize hang-ups of feeder, 100' crown pillar (53,57)	
Cu, O.P. 20,000	20 x 20	Vert.	600	no 60" x 60"	no	initial fines packing near feeder and rough walls inhibited flow, used level probe in pass (17)	
Stone, O.P. na	20' Diam.	Vert.	450	no na (est. 48" x 48")	no	25' of loose minimum left in bottom, sonar monitor- ed, 60' crown pillar (16)	
Pb, Zn, O.P. na	10 x 10	65°	650	no na	yes	inspection raise access to main ore pass (60)	
Fe, O.P. 40,000	13 x 13	Vert.	130	no 14" x 14"	no	haulage at ultimate pit bottom (6)	
Cu, Bl. Cav. na	60 x 60	Vert.	240	no 12" x 12" G		cave block from surface of open pit (66,30)	
Cu, O.P. 3,600	8' Diam.	Vert.	na	concrete grizzly		two ore passes used (56)	

(11) References given in Section 8.0

TABLE 8 - Ore Pass Data - Open Pit Mines in China*

Mine	Ore Pass		
(1) Commodity (2) Method (3) Production (tpd)	(4) Dimension (ft)	(5) Inclination	(6) Length (ft)
Fe, O.P. 18,000	16.4' Diam	Vert.	1,467'
Fe, O.P. 21,000	12.5' x 7.9'	70°	1,089
Fe, O.P. 5,200	19.7 x 19.7	Vert.	210
Cu, O.P. 4,100	18.7' Diam	Vert.	591
Lm. St., O.P. 6,850	13.1 Diam. 16.4 Diam.	Vert.	1,155
Lm. St., O.P. 4,930	19.7 Diam.	Vert.	1,082
Lm. St., O.P. 2,200	9.8 Diam.	Vert.	459
Lm. St., O.P. 830	9.8 Diam.	Vert.	295

*References (2,38). The latter states that in China over 100 open pit mines use ore pass haulage.

convergence and narrowing of the ore stream into chutes generally occurs. The lack of hang-ups in the ore pass proper is likely due to the more careful planning, design, and operational control associated with large ore passes compared with small, underground mine ore passes. Costs also appear to favor vertical circular ore passes, which very much resemble shafts, in open pit mine ore pass systems.

3.8 Summary and Conclusions

This section describes (i) ore pass design criteria applicable to the flow of rock in ore passes and (ii) ore pass data obtained from visits to operating mines and the technical literature. Ore passes serve two important purposes: (i) they provide an efficient method of transporting broken ore and waste rock from one level of a mine to a lower level and (ii) they provide storage capacity needed to insure a steady flow of ore from mine to mill. Rock flow in ore passes is gravity flow, so that operating costs are minimal in a properly functioning system of ore passes. Surge storage capacity is an additional benefit of the system that comes with little or no added cost.

Broken ore and rock flows in a controlled manner in a properly functioning ore pass. Hang-ups are absent, and live storage capacity is maintained. Rushes of ore and piping or rat-holing, that is, undesired dead zone formations do not occur nor is wall wear excessive in a well designed ore pass. The achievement of these design objectives is essentially a process of sizing the ore pass to the broken ore. In this regard, interlocking or wedging of large boulders is the most frequent cause of hang-ups in ore passes while the accumulation of fines and cohesive arch formation is perhaps the most difficult malfunction encountered in ore pass operation. Both types of hang-ups can be prevented by making the ore pass sufficiently wide. Calculational procedures concerning hang-up prevention and other design aspects are presented in Sections 3.2 through 3.6.

Ore pass data obtained from mine visits and the technical literature are presented in table form in Section 3.7. However, the design formulas developed in Section 3.2, 3.3, and 3.5 require a knowledge of the properties of the broken ore in the ore pass and the wall rock as well. Material properties testing is beyond the scope of the project, so that only a rough comparison between design theory and practice is possible in Section 3.7. As expected, "over-size" is the most frequent cause of hang-ups while ores that

tend to cement, sticky ores and ores containing a large percentage of fines are most troublesome. Ore passes in underground mines are relatively small and handle relatively small chunk sizes compared with ore passes and chunk sizes in open pit mines, although main ore passes in large underground mines may approach those used in small open pit operations. Structural stability and wear do not appear to be widespread problems, although incompetent rock and fault zones may require special treatment and wear plates are often required at chute brows.

The main conclusions that appear justified by the observations and analyses presented in this report are necessarily qualitative but nevertheless worthy of mention. These are:

- (i) Although the mechanics of flow of broken ore in ore passes is complex and not well understood in all details, equilibrium analyses remain valid and provide a basis for design,
- (ii) from the viewpoint of ore pass function, design for flow is accomplished primarily by sizing the ore pass to prevent hang-ups of large boulders and accumulation of fines,
- (iii) relatively large ore passes may pipe and thus not flow over the entire cross sectional area, although this type of "chimneying" that is sometimes observed in some caving operations has not been reported in ore passes,
- (iv) wear of ore pass walls in the ore pass proper is not particularly detrimental, in fact, such may be beneficial because of the enlargement and smoothing of the walls that occurs, however, wear at chute brows and in the vicinity of loading facilities is often noted and a source of maintenance expense,
- (v) structural stability of ore pass walls is not generally a problem, although adverse geology, and fault zones, for example, may require reinforcement,
- (vi) long, free falls of muck should be avoided because of the potential heavy impact damage to walls, chutes and loading facilities; keeping the ore pass filled is desirable in this regard but if the

ore has a tendency to cement, is cohesive, or prone to oxidation then draw should be frequent and only a minimal muck cushion should be left in the pass for protection,

- (vii) chutes and loading facilities are critical to the successful operation of ore passes and should be given careful design consideration.

Some of these conclusions are obviously a restatement of conventional wisdom concerning ore passes, others are based on a combination of analysis and interpretation of mine data. In this regard, the work of Chinese investigators appears to represent the largest accumulation of scientific observation extant, although details are not available at this time. Elsewhere, design guidance beyond rules of thumb follows largely from studies of the flow of bulk materials in steel bins and hoppers. Direct scientific observations and measurements of the flow of broken rock in ore passes made for the purpose of understanding and eventually controlling the phenomena involved are lacking. Ultimately improved ore pass design procedures must be based on a better understanding of "muck" mechanics, so that in view of the growing importance of the subject a modest program of experimental investigation would be justified.

4.0 ORE PASS LOCATION

There are a number of considerations related to selection of the location for an ore pass:

- rock mechanics
- minimization of haulage distances
- minimization of ore pass depth
- type of material to be handled
- number of ore passes required
- location of existing openings.

4.1 Number of Ore Passes

The first decision to be made is the number of ore passes which are required. In underground mines, the number of ore passes used depends on the mining method, ore body geometry and the handling and haulage system employed in the mine. In open pit mines the number of ore passes required will be determined by several factors including:

- requirement for waste handling
- draw point tonnage handling capacity
- storage requirements
- maximum satisfactory in-pit haulage distance.

4.1.1 Waste Handling

Underground mines occasionally have a waste pass as well, although breaking and handling of waste rock is usually avoided. Generally, when waste is broken, it is used for filling mined-out stopes as for example, in the mining systems employed at Mt. Isa in Australia, Geco in Northern Ontario, and Black Mountain (Aggeney's) in South Africa. The only other time waste handling becomes necessary is in development as current practice tends toward siting permanent openings such as levels, shafts, and ventilation raises in waste in order to minimize the stresses and displacements which occur in the walls of these permanent openings as a result of stress redistribution due to mining operations.

In open pit mines, the situation is different and the amount of waste to be handled usually exceeds the amount of ore - in some operations stripping ratios may be as much as 20:1. If an ore pass is being contemplated to reduce ore haulage costs, then, by the same reasoning, a waste pass could also be viable. Unless ore mining and waste stripping operations occur at different times of the year, as is practiced at the Knob Lake (Schefferville, Quebec) operations of the Iron Ore Company of Canada, Limited, a single "ore" pass would likely not be practical for both ore and waste.

On the other hand, waste dumps at open pit mines are often located around the perimeter of the pit and as close to the pit and the ultimate pit haul road as practicable. Thus, use of an underground haulage system for waste handling could result in increased cost and energy expenditure than would otherwise be the case, since grade and location constraints for the tunnel, through which the broken rock is carried from the ore pass draw point, would generally result in the tunnel portal being at some distance from the pit.

To determine, therefore, whether a waste pass is feasible requires computation of haulage costs for waste, with and without an ore pass and tunnel, and comparison of these costs. Haulage cost calculations will be unique to each mine and are beyond the scope of this report.

4.1.2 Drawpoint tonnage handling capacity

A very important factor in determining the number of ore passes required at an operation is the maximum rate of muck transfer at a draw point. If a feeder is used, the maximum transfer rate is about 4,000 to 5,000 tph or about 96,000 to 120,000 tpd at 100 percent utilization and availability. Assuming a combined availability and utilization of 75 to 80%, practical feeder capacity can be taken as 80,000 tpd. Pits whose production exceeds this value will require multiple draw points. This can be obtained in one or two ways:

- multiple draw points from a single ore pass
- multiple ore passes.

Although it is certainly feasible to have multiple draw points i.e. a branched bottom on a single ore pass, and

although branched bottoms result in enhanced flow characteristics, this is not common practice. Unless the tunnel and draw point are at sufficient depth that the branching occurs below the ultimate pit limit, use of a single ore pass with a branched bottom would result in several ore passes quite close together in later stages of the pit, if the collar was lowered with mining. This tends to suggest that this option would not provide much flexibility since the dump points would be close together - if there are to be multiple dump points they should be located in different portions of the pit in order to reduce truck haulage. Thus, multiple ore passes would be a more attractive alternative than a single ore pass with a branched bottom.

4.1.3 Storage Requirements

Another reason for providing multiple ore passes is storage requirements. Provided that the broken material does not contain large quantities (say greater than 20%) of wet fines it can be stored in ore passes for a period of time. This storage has a number of attractive features:

- provides feed for start-up after a scheduled holiday shutdown
- provides feed to the concentrator or screening plant in the event of a breakdown in the pit
- allows the use of continuous haulage systems from the ore pass draw point although dumping into the ore pass is cyclical.

In underground mines, it is customary to have ore storage capacity in excess of one shift's production. If this rule of thumb is applied at an open pit mine which produces 300,000 tpd, storage capacity of 100,000 tons or more is required in the ore pass system.

An ore pass having a diameter of 60 ft - the probable upper limit - would need to be at least 530 ft long, depending on the density of the broken material. This is the minimum length for the ore pass, i.e., if the ore pass is in the middle of the pit and the dump point is lowered when the benches are mined around it, the length of the ore pass cannot decrease below this value if the desired storage capacity is to be retained. This also means that if the collar is lowered until the ultimate pit bottom is reached, the ore pass must extend this distance below the pit if

storage capacity is to be retained. If this length cannot be provided below the ultimate pit bottom, there are several options:

- provide more ore passes
- do not lower the ore pass collar right to the ultimate pit bottom
- accept a reduction in storage capacity.

In some cases, it may be necessary to combine options in order to provide sufficient storage capacity because of length constraints due to tunnel alignment, topography, and pit geometry. For these reasons, the collars of the three ore passes of the Carol Project of the Iron Ore Company of Canada have never been lowered (Garg, 1981; Tuomi, 1981), in spite of plans to the contrary indicated in Pfleider and Dufresne (1961), and Selleck and Pfleider (1968).

4.2 Location of Underground Openings

4.2.1 Existing Openings

In districts having a long history of mining activity, deposits presently worked by open pit methods may have previously been mined by underground methods. Furthermore, in some instances, underground and open pit mining may be carried out concurrently; however, in such cases, the open pits are often of small tonnage.

Where shafts exist and are in good repair, one might consider hoisting ore pass muck. Mining of open pits by the "glory-hole" method with skip hoisting of muck was once quite common. However, the tonnage that can be handled in this manner is quite limited - maximum tonnage for a single skip hoist with balanced hoisting is about 25,000 tpd using 30 ton skips. Thus, shaft hoisting of muck from large open pits is impractical.

Where the underground workings have been abandoned for a period of time, they should be thoroughly inspected before consideration is given to their utilization in an ore pass system. Little reliance should be placed on old survey plans because they are often inaccurate and incomplete, especially for workings which are a long distance from the shaft used as the origin of the coordinate system.

Since no two mines are exactly alike, generalizations are dangerous and specific situations should be evaluated based on their individual merits. However, ore zones mined

by caving or by filling methods should generally be avoided. Drifts and cross-cuts more than 30 years old will probably have small cross-sections except in the case of room and pillar operations. Old room and pillar operations could be utilized as sites for ore pass draw points; however, if the same ore body was mined as is presently being mined by open pit, the old workings are either downdip from the open pit or will be intersected by the open pit. Since ore bodies worked by room and pillar methods are generally flat to gently dipping, downdip openings would likely be offset horizontally from the open pit beyond the lower limits of acceptable ore pass inclinations.

Rock surrounding blasted openings is fractured so that for reasons of stability ore passes should be kept well away from old stopes - also because of potential uncertainties in stope location as discussed above.

4.2.2 Planned Openings

There are four types of haulage which could be used to transport material from the ore pass draw point to the screening plant or concentrator:

- rail
- truck
- belt conveyor
- slurry pipeline.

The slurry pipeline alternative is practical only with soft, friable, non-abrasive materials such as coal and hematite. Slurry pipelines are uncommon and so will not be discussed further.

If trucks are used for haulage from the ore pass the tunnel location is rather flexible - trucks can climb 8% grades underload and have turning radii of about 40 to 50 ft. However, this option would reduce haulage cost only if the concentrator or screening plant elevation is lower than that of the top of the pit by an amount greater than the length of the ore pass or if surface topography is such that the haul road from the pit is long and circuitous, e.g. in mountainous regions.

Rail haulage would result in more constraints in draw point location than with truck haulage since radii of curvature for curves must exceed 600 ft and should preferably be at least 1,000. Grades should not exceed 4% (although occasionally higher grades are encountered, e.g. in

mountainous areas) and preferably should be 1% or less for best results. With rail haulage, operations can be highly automated especially if remotely controlled unit trains are used such as at the Carol Project of the Iron Ore Company of Canada, Ltd. Design details for rail haulage systems can be found in Pfleider (1968).

Another haulage alternative is the belt conveyor. Belt conveyors can generally be used for grades up to about 30% with run-of-mine ore although research has been carried out concerning steeper conveyors. The primary constraints on the use of conveyors is the fact that they must be straight - belt conveyor systems which incorporate curves have been designed but are not in common use. Thus if belt conveyors are to be used the locations must be carefully planned.

In summary, if rail or belt conveyor haulage is used the tunnel location constraints imposed by limitations in the haulage method could dictate the locus of acceptable draw point locations.

4.3 Inclination

The inclination of the ore pass must obviously be sufficiently great that material will flow. Scrutiny of a large number of case histories indicates that ore passes are either vertical or inclined at an angle of between 60 and 70 degrees from the horizontal. The prevalence of inclined ore passes is due in part to the belief, held by many engineers, that hangups are less likely in inclined as opposed to vertical ore passes (Pfleider and Dufresne, 1961). A further consideration in favor of inclined ore passes is the fact that there is less damage to the walls from the ricochet of blocks than in a vertical ore pass as material in the former will tend to slide down the footwall. Inclined ore passes do sometimes have inclinations less than 60 degrees but this is not common practice and troubles do arise when wet fines are dumped. Conversely, inclinations greater than 70 degrees may be required if the material handled contains a large percentage of wet fines.

The ore passes found in underground mines rarely have dimensions exceeding 10 ft by 10 ft. However in open pit mines having production rates in excess of 300,000 tpd, ore passes up to 60 ft in diameter would be required. In such cases, vertical or near vertical openings are easier to construct than openings inclined at 60° degrees to the horizontal. Furthermore, hang-ups in such large openings are

highly unlikely except in freezing climates - in this case calcium chloride sprays are used at the dump point to minimize this problem.

However for the purposes of evaluating locational constraints, a minimum inclination of 60 degrees from the horizontal can be used in preliminary planning.

4.4 Depth

From the standpoint of capital cost, the depth of the ore pass should be the minimum consistent with other considerations such as the elevation of the ultimate pit bottom, ore pass cross-section and storage requirements.

The minimum distance that the ore pass must extend below the ultimate pit limit is at least 75 ft - that is, there should be a barrier pillar of at least that thickness. Although this thickness is arbitrary, it is based on experience and engineering judgment. It is known that a fracture zone of up to 10 ft surrounds blasted openings, and, that in underground mines, high stress concentrations occur if the pillar between a stoping block and the level either above or below it contains 30 ft or less of solid (unfractured) rock. Thus the minimum pillar thickness should be 50 ft, and, applying a safety factor of 1.5, this pillar thickness becomes 75 ft.

This 75 ft depth together with a 60 ft diameter provides storage of only about 14,000 tons of broken copper ore. Thus, if the collar of the ore pass is carried downward as benches are mined, very limited storage would be available in the latter stages of the pit life. Possible remedies are:

- reduce the amount that the collar is lowered
- increase the depth of the ore pass
- provide additional ore passes
- do not lower the ore pass collar
- accept a reduction in desired storage capacity.

The particular option, or options, chosen will obviously be site specific and depend on the preferences of the operator and the physical characteristics of the particular mine.

4.5 Minimization of Haulage Distance

If an ore pass is to be of any value in an open pit mine, its placement must result in efficient haulage throughout the pit. It must be recognized at the outset, however, that an ore pass must be accessible and so the location will be dependent on pit geometry, ultimate haul/road location and other site-specific constraints.

In general there will be a maximum desirable one way distance when trucks haul to a more efficient and continuous haulage system such as rail. At the Carol Project of Iron Ore Company of Canada Ltd, ore passes were located at about 2750 ft intervals so that the average round trip truck haul would be less than a mile (Pfleider and Dufresne, 1961). At the Denison trackless room-and-pillar underground mine, conveyors were laid out at 3,000 ft intervals (McCutcheon and Futterer, 1960). Note that these distances are examples and are based on smaller units with smaller load capacities than are presently in use at open pit mines. The actual maximum desirable distance will be site specific.

In general, the location of a facility such as a truck dump should be such that it minimizes the cost of haulage to it, that is, the average haulage distance should be minimized. There are several types of distance that can be involved in location problems (Francis and White, 1974):

- straight-line or Euclidean distance
- rectilinear distance
- distance for a non-linear cost function.

Rectilinear distance applies if possible travelways comprise an orthoogonal set of axes, e.g. city streets. Euclidean distance applies if the cost function is linear with distance and if the distance traveled between any two points is a straight line. A common formulation of the non-linear cost function is the "gravity" problem in which the costs are proportional to the square of the straight-line distance. The gravity problem has a simple solution which is sometimes referred to as the centroid or center-of-gravity solution (Francis and White, 1974).

In an open pit mine, the distance traveled is usually straight line as much as possible so that the rectilinear solution will generally not apply. The cost function is linear with distance so the Euclidean distance applies. The problem is complicated by the fact that the existing

"facilities", i.e. shovel locations, are not fixed so that the problem is really one of minimizing a maximum distance or a "minimax" problem. These problems are nonlinear programming problems and thus, must be solved iteratively. A useful algorithm for solving these problems is given by Francis and White (1974). The procedure, as presented in their example can be significantly simplified by choosing, for the first two points, the points situated the greatest distance apart to define the diameter of the circle. An alternative procedure would be to find the centroid using the method of static moments (Church, 1981). It can be shown geometrically that the centroid of the figure lies near the point found by the method given by Francis and White (1974) and hence either method is acceptable.

In the case of a single ore pass the location procedure is carried out using the outline of the whole bench. If several ore passes are required, the bench outlines should be subdivided into a number of approximately equal-sized parts such that each part has an ore pass. The location of the ore passes shall be determined using one of the aforementioned for each subdivided area.

The number of ore passes used in the previous paragraph may have been based on considerations such as storage and throughput requirements. There may also be a desirable upper limit to the haulage distance, as discussed in previous paragraphs. If we assume straight line haulage and a rectangular pit with ore passes at the centroids of m equal-sized areas along the length and n equal sized areas along the width, the average haul can be estimated by:

$$H = \frac{1}{2} \sqrt{\left(\frac{L}{m}\right)^2 + \left(\frac{B}{n}\right)^2}$$

where

- H = haulage distance
- L = length of the pit
- B = width of the pit
- m = number of sets of ore passes along the length
- n = number of sets of ore passes along the width.

If this distance is not considered satisfactory, one could consider using more ore passes. However, the savings due to reduced haul distance must be more than the capital cost of an additional ore pass and longer tunnel together with the operating costs of the additional tunnel facilities, using discounted cash flow analysis.

4.6 Rock Mechanics

Rock mechanics enters into ore pass location in two ways:

- size of barrier pillars
- location of unfavorable rock zones.

The need for and selection of a barrier between the ore pass draw point and the bottom of the pit has been discussed in Section 4.4.

If the ore pass is outside the pit or, in the latter stages of mining, is in the wall of the pit relative to the mining benches, then a barrier is also required between the wall of the pit and the wall of the ore pass. This barrier should have a thickness of at least twice the diameter of the ore pass. However from the same considerations as given in Section 4.4, the thickness should be greater than 75 ft. The thickness of the barrier should be chosen according to the criterion which results in a greater thickness. If desired, numerical analyses could be performed to determine a site-specific barrier. Zones of weak, or altered rock, heavily faulted zones, and shear zones should generally be avoided when siting an ore pass. However, this is not always possible. Singh (1973) discusses an ore pass system which was, of necessity, located in a shear zone, as well as the reinforcement which was carried out after the instability of one section of the ore pass became severe. Similar problems have been encountered at other mines when ore pass location is unfavorable with regard to fault zones. An extreme example of the difficulties that can be encountered due to faults occurred in the shaft at the former Chesterville Gold Mine near Larder Lake, Ontario, when movement of a wedge defined by a fault which intersected the shaft, changed direction and reintersected the shaft, and necessitated weekly reblocking of the shaft timber (Banasuik, 1980).

The decision as to whether to avoid such a disturbed zone as opposed to providing reinforcement requires a knowledge of the pit geology in the suggested ore pass location. To this end, a diamond drill hole should be drilled at each projected ore pass location to the full depth of the ore pass.

4.7 Ore Pass Planning

To this point in this section, individual constraints have been discussed, but the location selection process has not been addressed. A design manual can make suggestions and provide guidelines, but it cannot wholly take into account site-specific items such as pit geometry and topography. Also, there may be a number of locations which satisfy the constraints imposed. The final location selection therefore rests with the pit operator and will reflect his preferences.

Another important consideration is whether to locate the ore pass within or outside pit limits. In underground mines ore passes are generally located in country rock away from the ore body in vein type deposits. Ore passes in flat lying deposits are generally used to transfer ore to a gathering level (haulage level) in the footwall. Since ore passes in underground mines are thus, generally in waste, the walls must be stable so that dilution is minimized. In an open pit mine, minimization of the haulage distance will generally require the ore pass to be within the pit limits. This will also mean that ore pass storage is reduced every time the dump point is lowered as a bench is removed. The alternative - not lowering the collar or reducing the amount that the collar is lowered - will result in a reduction of mineable ore reserves if the ore pass is sited wholly or primarily in ore. Siting an ore pass outside the pit limits obviates this problem but results in a longer and generally uphill haul. It should be remembered that uphill hauls under load are more costly than downhill hauls under load.

Previous discussions in this section have dealt with particular elevations - draw point and dump point. The ore pass will pass through a number of benches and one must ensure that any constraints are satisfied on these benches as well. An obvious method is to draw the projected ore pass(es) on individual plans and sections. An alternate method would be to use the block plan system often used in underground mine planning. Applying this system to open pit ore pass design results in the creation of a composite plan showing bench outlines, ore contacts and fault locations of several equally spaced elevations. By using a different color for each elevation, it is possible to show about 8 elevations on a single plan. The elevations shown should include the dump point and draw point elevations. The remaining elevations chosen should conform to toe elevations of benches. In general it will be adequate to depict every

third or fourth bench. Pre-existing constraints to dump and draw point location should be shown. The effect of dump point constraints on draw point location and vice versa, can be shown by the limit at the required elevation defined by a 60° cone from the constrained location. Locations chosen in order to minimize haulage can then readily be checked for compliance with other constraints.

4.8 Summary

Ultimately, ore pass location is dependent on the physical characteristics of the mine and mine operator's philosophy. Hence this section merely indicates some aspects to consider and some methodologies which could be used in the selection of the location for an ore pass. It must be considered that there may be several satisfactory ore pass locations.

The aspects to be considered include:

- number of ore passes required
- minimization of ore pass depth
- minimization of haulage distance
- location of existing and planned openings
- location of geologically unfavorable zones.

The number of required ore passes, in turn, depends on several subsidiary considerations including:

- storage requirements
- throughput requirements
- waste handling
- draw point tonnage handling capacity.

5.0 DESIGN

5.1 General

Design of an ore pass can be broken down into several distinct tasks: support design, dump point design, draw point design, location, and physical size selection. The requirements for location and size selection have been discussed in Section 4. This section, therefore, shall be concerned with support, dump point design and draw point design.

5.2 Support Design

The requirements for support are determined using rock mechanics. The first step is to collect information on the structural geology of the proposed ore pass location. Mine plans and sections should be scrutinized for faults, folds and zones of weakness near the projected ore pass location. Such zones of adverse structure should be avoided: however, if it is not possible to avoid them, rock reinforcement and lining systems can be designed to stabilize them. Ore passes in the underground mines of one Canadian mining company, which have been sited in heavily jointed or faulted rock have been stabilized using long grouted cables as rock reinforcement (Singh, 1973; Pakalnis, 1980).

It is imperative that detailed geological information be available along the axis of the projected ore pass. This is obtained by drilling a diamond drill hole to full length along the ore pass axis. Holes of BQ or NQ size are preferred, and the coring equipment used should be suitable for the type of rock being drilled.

Core from the drill hole should be logged as soon as possible after it is removed from the core barrel. This is especially important in formations susceptible to weathering. The drill log should include:

- lithology
- percent recovery
- Rock Quality Designation (RQD)
- jointing
- angle between joints and core axis
- type of joint surface - smooth, rough or slickensided

- amount of joint opening, if any
- type of joint filling, if any
- locations of disked core i.e. core broken into small disks with lengths less than or equal to core diameter.

Rock Quality Designation (Deere, 1964) is the percentage of a run of drill core consisting of pieces longer than 4 in. of NQ or larger core. Where a core run crosses contacts, RQD should be calculated separately for the different materials. The core log should also indicate areas where drill water was lost and where water inflows occurred. In the case of inflows, the rate of inflow should be estimated, if possible.

Disking of drill core, i.e. the breaking of core into small pieces whose lengths are less than their diameters, is an indication of high stresses perpendicular to the axis of the borehole. In a vertical hole, diskings is, therefore, an indication that high horizontal stresses are present. Although it is normally assumed that horizontal stresses underground are less than the vertical stresses, there is strong evidence that, this is not the case in many parts of North America for depths less than 4 to 5,000 ft (Haimson, 1977; Herget, 1980; Bhattacharya, 1980).

Using the core log, determine locations where joints are inclined at more than 45° from the horizontal should be determined. For vertical holes (assuming no deviation), this would be all joints whose angle with the core axis is less than 45° . For inclined holes, the angle a 45° -joint would make with the core axis can be found by geometry. (If a core orientation device were used the azimuths of the joints could also be determined).

For all joints with an inclination greater than 45° , the joint cohesion and angle of friction should be determined in a shear tester and the safety factor against sliding determined. This is dealt with in two ways depending on whether the rock under consideration is near the surface or at depth. For near surface joints, it is first assumed that joints are through-going and equally spaced. One then determines the depth for which a wedge would slice on one of the joints. For joints at depth, the thickness can be taken as 10 ft since blasting generally produces a loosened zone of up to 10 ft thick around openings.

The height of the wedge would be the thickness of the rock stratum under consideration. From the condition of limiting equilibrium between the sliding and resisting forces (see, for example, Hoek and Bray, 1977).

$$SF = \frac{W \cos\alpha \tan\phi + cA}{W \sin\alpha}$$

where

- W = weight of the wedge
- α = angle of inclination of joint plane
- ϕ = angle of friction of joint
- c = cohesion
- A = area of joint surface
- SF = safety factor.

If the safety factor is greater than 1.0, the joint surface is stable and sliding is unlikely to occur. Otherwise either rock reinforcement or a lining is required to restrain the rock - rock reinforcement comprising grouted rebar bolts is preferred. Bolts should be directed approximately normal to the joint and extend about 3 ft beyond it. Joints containing clay, gypsum, or calcite fillings should be bolted even if the safety factor is greater than 1.00. If grouted bolts are used, the number of bolts to use can be found by choosing a safety factor and substituting into the following formula:

$$SF = \frac{W \cos\alpha \tan\phi + 0.6N_s A_s F_y + cA}{W \sin\alpha}$$

where

- N_s = number of bolts
- A_s = cross-sectional area of steel in the bolts, in²
- F_y = yield strength of the steel, psi
- other symbols as defined previously.

The foregoing discussion has dealt with reinforcement of joints within an otherwise stable rockmass. However it is also important to know if the rock mass itself will require reinforcement. In tunneling, there are a number of rock mass classifications which have been proposed to aid in the evaluation of support requirements. These classification schemes are at present not in general use in mining and, because of differing life span requirements for mines and civil works, may be of limited usefulness. Although attempts have been and are being made to apply at least one of the classification systems to mining situations, this work is

not sufficiently general to merit its inclusion here. Fracture frequency obtained from drill core logs has been used in some mines to predict weakened zones (Mathews, 1975). Because RQD is related to fracture frequency, RQD is often used to evaluate the rock quality at a site (Deere, 1968), and hence estimate the support requirements. Table 9 gives generalized estimates of support requirements. Zones less than about 20 ft in length where the rock quality is better than that of adjacent zones should be considered as part of the rock adjacent to it.

The Snowy Mountains Authority in Australia has made recommendations for the length of rock reinforcement in tunnels and underground chambers (Alexander and Hosking, 1971) as follows:

$$L \geq 6.0 + 0.004W^2$$

where

L = bolt length, ft
W = span of opening, ft.

As an approximate guide $L = (1/3 \text{ to } 1/4)W$.

The bolting spacing can be estimated from the following rules of thumb:

excellent rock	- spot bolting only
good rock	- $(1/2 \text{ to } 3/4)xL$
fair rock	- $(3/8 \text{ to } 1/2)xL$
poor rock	- $(1/4 \text{ to } 3/8)xL$
very poor rock	- $\leq 1/4xL$

where L = bolt length, ft.

For closely jointed material, wire mesh should be installed to restrain rock between the bolts from spalling. In horizontally bedded formations, pattern bolts in shafts and raises are often inclined about 45° downward in order to reinforce bedding planes. The bolting pattern selected should also be adequate to reinforce against sliding on joints as discussed in previous paragraphs in this section.

For rock classified as "very poor" (RQD ≤ 25) a cast-in-place concrete liner is required. Although rules of thumb exist for thickness of concrete liners in tunnels as a function of opening diameter this is not so in the case of shaft linings (Golder and MacLaren, 1976). Shafts are

TABLE 9 - Support Requirements Based on Rock Quality

ROCK QUALITY	ROCK BOLTS	SHOTCRETE	CAST CONCRETE
EXCELLENT	SPOT BOLTING	LOCAL	NO
GOOD	PATTERN BOLTING WIDE SPACING	LOCAL	NO
FAIR	PATTERN BOLTING MEDIUM SPACING	4 in. (OR MORE AS REQUIRED)	NO
POOR	PATTERN BOLTING CLOSE SPACING	6 in (OR MORE AS REQUIRED) WITH PATTERN BOLTS	PERHAPS
VERY POOR	PATTERN BOLTING WITH MESH VERY CLOSE SPACING	6 in (OR MORE AS REQUIRED) FOLLOWED BY CAST CONCRETE LINER	YES

FOR DETAILS ON DESIGN, SEE TEXT

commonly designed using Huber's formulae for restrained thick wall cylinders which can be used in the following form (Galanka, 1960):

$$t = a \left(\sqrt{\frac{f'_c}{f'_c - 5.2p}} - 1 \right)$$

where

t = lining thickness, ft
a = inside radius, ft
f'_c = concrete compressive strength, psi
p = external pressure, psi.

The above equation includes a safety factor of 3 for the compressive strength, in conformance with normal practice. In order to use this formula it is necessary to estimate the external pressure which will be acting on the lining. In competent non-water-bearing rocks the external pressure will be minimal provided critical joints are reinforced. In competent water bearing rocks such as sandstone, the external pressure would equal the pore water pressure within the formation unless the shaft walls were grouted. In sedimentary sequences, weak strata, can be squeezed out and exert an external pressure if the adjacent rocks above and below are more competent, for example fine grained sandstones or limestones with intercalated shale. For weak impermeable strata such as shales, the lateral pressure may be obtained from the active earth pressure due to loosening of a wedge of material (Coates, 1970):

$$p = \gamma H \tan^2(45 - \phi/2)$$

where

p = external pressure, psi
γ = unit weight of material, psi/ft
φ = angle of internal friction of material, deg
H = thickness of weak stratum, ft.

For incompetent rocks below the water table, Ostrowski (1972) gives the following estimates for the external pressure, based on European experience:

for sandy formations: $p = (1.3-1.4) \times 0.434 \times H$
for loam and clay formations: $p = (1.6-1.7) \times 0.434 \times H$

where

H = depth, ft
p = external pressure, psi.

Above the water table, the coefficients can be reduced by about 50%. Taking the lower values of the coefficients and combining coefficients we obtain

sandy formations: $p = 0.564H$
loam and clay formations: $p = 0.694H$

p , H are as defined previously.

All the aforementioned considerations concern uniform loading of the lining. To model non-uniform loads, the following relation has been used (Weehuizen, 1960):

$$p = p_o(1 + \beta + \beta \cos 2\psi)$$

where

p = non-uniform stress
 p_o = uniform stress
 β_o = constant (usually 0.10 - 0.15)
 ψ = angle between point at which load being calculated and axis of maximum load.

This equation is not used for lining design except possibly in the case of faults known to traverse the shaft location but is used to check the resistance to non-uniform loads of a lining dimensioned for uniform loads.

5.3 Dump Point Design

5.3.1 Size of the Dump Point

The main consideration in design of the dump point is the required size. This is a function of:

- the number of trucks to be dumped simultaneously
- the truck dumper configuration -- side-, rear-, or bottom-dump
- the dumping dimension of the truck.

In open-pit metal and non-metal mines, rear dump trucks are used almost exclusively whereas, in open cast coal mines, bottom and side dump trucks may also be used. The articles on the ore pass system at the Carol Project (Pfleider and Dufresne, 1961; Selleck and Pfleider, 1968; Pfleider, 1973) mention the use of 100-ton side dump trailer trucks; however, rear dump trucks of at least 120-ton capacity have been in use at the operation for at least a decade.

For a single truck dump the width of the dump should exceed the pertinent dimension of the truck box by at least

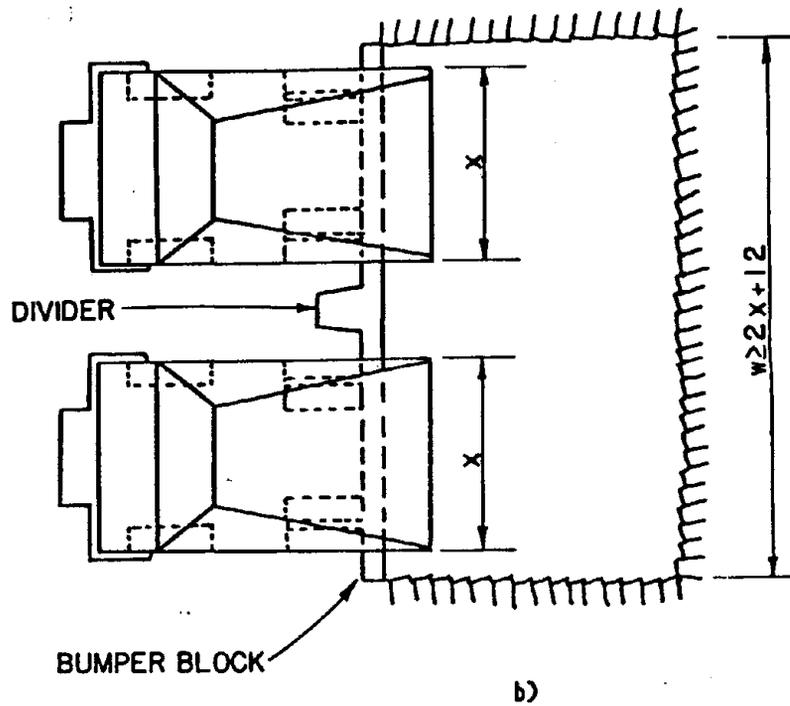
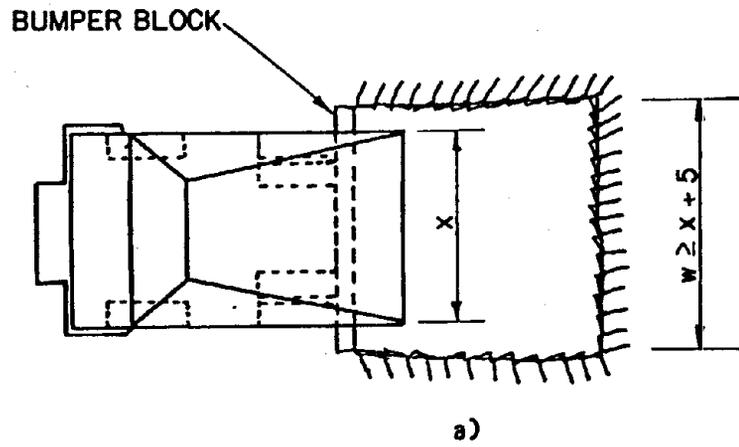


Figure 24 - Dump point width
 a) single vehicle dump
 b) two vehicle dump

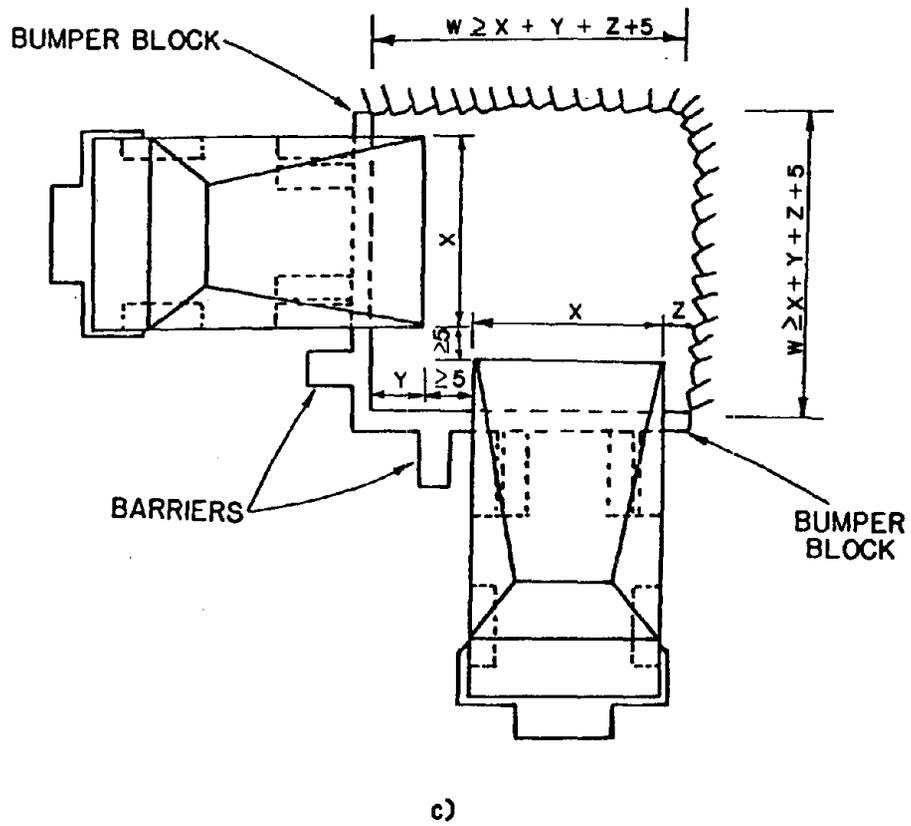


Figure 24 - Dump point width
 c) alternate configuration
 for two vehicle dump

5 ft as shown in Figure 24a. With rear dump trucks, an additional allowance should be provided due to visibility limitations when backing.

With rear dump trucks, two trucks could be dumped side-by-side as shown in Figure 24b. The divider should be large enough to provide an effective separation between the trucks. The allowance of 12 ft given in Figure 24b is an absolute minimum and a greater allowance is preferable. Another configuration for a two-truck dump is given in Figure 24c, in which case the width of each dump must exceed the pertinent dimension of the truck by at least 12 ft.

In any case the dimensions of the dump point must be equal to or greater than those in the ore pass proper. If the ore pass is vertical or near vertical ($\theta > 85^\circ$) a flared lip on the dump side is desirable in order to assure the stability of the truck dump(s). The flared lip should be inclined at no less than 60° from the horizontal to ensure that muck landing on it flows into the ore pass. If the ore pass is itself inclined, then no flared lip is necessary.

5.3.2 Bumper Block

To prevent a truck from backing into the ore pass a bumper block is required at the dump point. Bumper blocks should be designed as retaining walls for which design procedures can be found in civil engineering or soil mechanics textbooks and handbooks. The bumper should be of sufficient height and strength to fulfill its stated purpose. The bumper block must adequately be anchored to the rock beneath it.

At a small open pit mine where 35-ton rear dump trucks dumped into an ore pass, a tree trunk, of about 18 in. diameter, which was pinned to the rock was found adequate. At a medium sized open pit mine where 120-ton trucks dumped into a 30-ft by 35-ft ore pass, the bumper block consisted of a 3-ft high steel plate backed by concrete. At this operation, other types of bumpers including 12-in. by 12-in. timber were tried unsuccessfully - at least two trucks ended up in the ore pass. At a large scale iron ore producer which uses 120-ton and 170-ton rear dump trucks, the bumpers consist of 3 to 4 ft high steel pads (Garg, 1981).

5.3.3 Lowering the Dump Point (Collar)

If the ore pass is located within pit limits, it will generally be desirable to carry the collar of the ore pass downward as mining progresses, since to leave the dump point

at a higher elevation than current mining would tie up ore reserves in barriers and in the ramp to the dump point.

If the ore pass dump elevation is to be lowered regularly, then the dump facilities should be designed to facilitate this; otherwise considerable down time could result. That is, bumper blocks and guard rails should be robust enough to perform their duties but should at the same time be sufficiently movable so that demolition and later re-erection at the lower level can be readily accomplished. Thus, barriers, bumper blocks, and other facilities provided for reasons of safety and security should be minimized consistent with requirements.

One other aspect to consider with regard to dump point lowering is the reduction in storage capacity that results. For example, lowering the collar of a 55-ft ore pass by 50 ft would reduce storage capacity by an amount ranging from 6,600 to 11,900 tons depending on the density of the broken material being handled.

Actual procedures for lowering a dump point are discussed in the section entitled "Operating Procedures."

5.3.4 Dump Point Foundations and Liners

The requirement for a foundation, or more properly a roadway slab, at the dump point is dependent on the bearing capacity of the rock and the maximum wheel load for a fully loaded haulage vehicle. The stress imposed on a road surface by the tire of a haulage truck is approximately equal to the tire air pressure. As a first approximation for the bearing capacity of the material, the allowable bearing capacity of the material as given in, for example, the National Building Code of the National Board of Fire Underwriters can be used.

A more rigorous procedure would require determination of the California Bearing Ratio (CBR) of the material. One would then design the slab as one would design a roadway either by the somewhat complicated methods given in highway engineering texts or by the simplified method given by Chironis (1978).

To enhance stability and to minimize erosion of the walls of the dump point a liner is recommended. Unless a liner has also been prescribed for reasons of structural integrity, the lining can be constructed of a nominal (8 to

12 in.) thickness of concrete covered by 1/2 to 1 in. thick steel plate on wear surfaces. Otherwise, the structural lining governs.

5.4 Draw Point Design

The design of the draw point, or outlet, of the ore pass is crucial to smooth operation of the system, since the draw point is the transition from the ore pass to the downstream haulage.

If the ore pass is to operate successfully, the throughput and live storage requirements must be met. To do so the draw point must be sized to minimize the possibility of:

- hangups
- piping.

These problems and their minimization have been discussed in detail by Pariseau in Section 3. In the case of block hangups, however, the criterion for their avoidance at the draw point is:

$$D_o \geq 3d$$

where D_o = width of the outlet, ft
 d^o = maximum dimension of largest block, ft.

This reduction is permissible since block hangups which occur at the draw point can be readily removed and, therefore, do not pose as great a hazard as in the ore pass proper. Similarly, if a water hose is available at the draw point which is connected to a pressurized water supply (greater than 100 psi), cohesive arches at the draw point can be easily undermined.

Depending on the type of haulage system to be employed, the draw point can be one of several types:

- hopper and feeder
- longitudinal chute (flow along tunnel axis)
- transverse chute (flow perpendicular to tunnel axis)
- open draw point in cross-cut off the tunnel.

The latter alternative is used with L-H-D vehicles and is not recommended for systems requiring large tonnage throughput. Chutes can be used with rail or truck haulage and feeders can be used with rail or belt conveyor haulage.

5.4.1 Chute Design

The suggested width of the chute has already been indicated. From examination of scale drawings of large number of mine chutes, the following ratio between chute height and width was deduced:

$$H \approx 0.8D_o$$

where

H = height of the chute ft
D_o = width of the chute, ft.

From the same examination, it was also found that the slope of the chute slide is generally between 30 to 45°.

The dimensions of the chute must also be compatible with the pertinent dimension of the railcars or trucks used for haulage. Thus, if a transverse chute is used, its width must be at least 6 in. less than the width of the haulage vehicle, and if a longitudinal chute is used, its width must be at least 6 in. less than the length of the haulage vehicle.

An important consideration in chute design is the type of gate to use. There are a number of types of gates which have been used in underground mine chutes including:

- undercut arc gates with control chains
- underhand guillotine gates
- finger gates
- bar and chain gates
- stop logs.

Stop logs are only found on very small chutes and are more or less obsolete. The other types are usually opened and shut by a compressed air cylinder and can all be used for even the largest chutes. Each has advantages and disadvantages:

- undercut arc gates with control chains are excellent for controlling fines, and good for blocks but will discharge if the compressed air supply is cut off.

- underhand guillotine gates are also excellent for fines, but can be hard to close if there are blocks and will also discharge if the compressed air supply is cut off
- finger gates are excellent for blocks, and will not be affected by a lack of compressed air but will not hold back fines
- bar and chain gates hold back blocks very well but do not work well with fines, and would probably not be heavy enough to prevent a run if the compressed air supply were cut off.

Probably the best system would be to combine a finger gate with an undercut arc gate although this option would be more expensive than those previously mentioned. Otherwise, the option to choose for a given situation basically depends on the amount of fines expected in the muck. (The amount varies, but generally over 25 percent.) The choice should be made on the basis of the advantages and disadvantages of the different gates as noted above. This only serves to emphasize that it is of paramount importance that the size distribution of broken muck be known and accounted for in design.

It was previously stated that the inclination of chute slides has been found to be between 30 to 45°, with a mean of 37°. However many of the chutes depicted in Tillson (1938) were intended to be hand operated and hence would be made overly flat in order to allow easier control of the muck flow. At one of the mines visited, chute slide inclinations were 37° and resulted in a build-up of fines - investigations showed the angle of friction for muck on steel to be 43° for their muck. Thus it is recommended that an angle of 45 degrees or greater be used for chute slides.

As stated previously chutes can either be longitudinal or transverse. The transverse chute is the traditional arrangement in most underground mines, except where scrams are used. The longitudinal arrangement is more efficient, and, if the ends of railcars overlap can be used to continuously load a slowly moving train.

5.4.2 Feeder Selection

Feeders are generally used to remove bulk materials from storage units such as bins, bunkers, silos, and hoppers. For coarse materials such as run-of-mine ore, the

best feeders are apron feeders and bar-flight feeders (Reisner and Eisenhart - Rothe, 1971).

Apron feeders can handle up to 3,000 tph whereas the bar-flight feeder can handle over 5,000 tpd. The capacity of an apron feeder is found as follows:

$$C = \frac{S\gamma w_f h_f}{33.3}$$

where

- C = feeder capacity, tph
- S = feeder speed, fpm
- γ = material unit weight, pcf
- w_f = width of feeder, ft
- h_f = depth of flow of material on feeder, ft.

Ranges for the variables are as follows:

$$\begin{array}{l} 2 < w_f < 10 \\ 0 < h_f < 5 \\ 10 < S < 50. \end{array}$$

The horsepower required for an apron feeder can be estimated as follows if a circular or square hopper is used:

$$h_P = \frac{\gamma D_o^3 S}{22,000}$$

where

- D_o = hopper outlet width, ft
- γ, S = as defined previously,

If a slot opening is used in the hopper, the horsepower required can be estimated by:

$$h_P = \frac{\gamma D_o^2 L_o S}{11,000}$$

where

- L_o = length of the hopper outlet, ft
- D_o, γ, S = as defined previously.

Feeder length should be at least 1.5 to 2 times its width and should also be greater than twice the length of the hopper. To ensure uniform flow from a hopper with a slot opening, the slot should be tapered outwards by about 5% in the direction of flow. The inclination of the walls of the hopper should generally exceed 65°.

As has been indicated, maximum practical feeder capacity is about 5,000 tph, with a utilization (including availability) of 75%. The maximum daily capacity is about 90,000 tpd. Thus for larger production rates than 90,000 tpd more than one draw point is needed. With rail haulage an ore pass with multiple draw points could be used. Otherwise, multiple ore passes are likely required.

5.5 Branches, Bends, and Knuckles

As indicated in the previous section, a branched bottom could be used if a single ore pass was desired but the required production exceeded the capability of a single draw point. A branched bottom provides some flexibility, but even more important is that it implies that production is not halted if repairs become necessary in the lower portion of the ore pass. The draw point facilities and the rock in the brow over the draw point are the most common locations requiring repairs. Branching is also conducive to rock flow as has been explained in Section 3. The inclination of the branches should exceed 60° and the cross-sections should be selected as follows:

- if draw is from one side only at a time, the branches should be the same size as the ore pass proper
- if draw is from both sides, branches may be smaller than the ore pass proper provided that the dimensions are adequate for the prevention of arching and piping. Branches could also be used in the upper part of the ore pass to provide multiple dump points. However, the junction point is generally a source of wear and additionally it is difficult to drive large openings on a 60° incline. With smaller sized ore passes (less than 15 ft diameter), multiple dump points could be quite effective.

Bends and knuckles are often used in the bottom of ore passes to provide impact protection and flow control. Bends could also be employed to bypass a known zone of weak rock, or to intersect it at a more favorable angle. The inclination of the bends should exceed 60° . In the case of knuckles, an inclination of 45° is commonly used for both parts although to enhance flow the angle should exceed 60° .

One major difficulty concerning changes of direction is the fact that they usually result in a large amount of wear. Thus, it is a wise precaution to provide wear plates at such locations.

5.6 Summary of Design Aspects

Support requirements are determined using rock mechanics. Areas of adverse geological conditions should be avoided if possible. Otherwise rock reinforcement or support is required. Rock Quality Designation (RQD) can be used to identify such adverse zones from drill core and to estimate the amount of reinforcement or support required. If a liner is required it should be constructed of concrete and have steel liner plates in the interior.

The main consideration in the design of the dump point is the required size. This is dependent on:

- the number of trucks to be dumped at once
- the truck dumper configuration-- side-, rear-, or bottom-dump
- the dumping dimension of the truck.

The cross-section dimensions of the dump point must also be at least as large as those of the ore pass itself. The dump point must be provided with a bumper block and guard rails to prevent personnel and vehicles from accidental ingress. The size of the bumper block depends on the size of the trucks used in the pit. In designing the dump point facilities, it should be borne in mind whether the dump point is to be lowered with mining and, if so, how often this would take place. The requirements for foundations and liners at the dump point are dependent on the quality of the rock; the size of such facilities should be the minimum consistent with stability, especially if the dump point is to be lowered regularly.

Design of the draw point is crucial to smooth operation of the ore pass system since it is the point of transition from the ore pass to the downstream haulage. To operate successfully, the tonnage throughput and live storage requirements must be met. Thus the draw point must be sized to minimize the possibility of hangups and piping. The two most likely draw point configurations are a chute and a

hopper and feeder. The dimensions of these facilities must satisfy both flow criteria and size requirements dictated by the type of downstream haulage employed.

Branches, bends and knuckles may be employed in an ore pass for various reasons. Direction changes near the bottom do provide the draw point with impact protection but themselves are sites of heavy wear as a result. Branches provide flexibility but for very large openings could be difficult to construct.

6.0 EXCAVATION METHODS AND COSTS

6.1 General

Ore passes can be driven from either the top or bottom of the proposed opening. Methods whereby the opening is driven from the top comprise shaft sinking, shaft boring, drop raising, and raise boring. Methods whereby the opening is driven from the bottom include conventional raising, raise climbers and raise boring. These methods are discussed individually, with each discussion including a short description of the method, cost estimates, and advance rates.

6.2 Shaft Sinking

As its name implies, shaft sinking advances an opening from the top down. A shaft sinking cycle for an ore pass consists of drilling, blasting, mucking and lining.

Drilling is done with either hand-held plugger drills or with a special jumbo. Use of the jumbo is restricted to full-face rounds. Rounds in rectangular shafts are benches of half the shaft cross-section while in circular and elliptical shafts, full-face rounds are generally used although bench rounds have been used as well. Holes are usually 1 1/4 in. diameter and the number of holes can be estimated from the shaft cross-sectional area. Based on drilling patterns reported in the literature, we obtained regression equations for estimating the number of holes for both benching and full-face rounds. For benching we determined the relationship to be:

$$N = 0.104A + 6.7$$

where

N = number of holes per round

A = shaft cross-sectional area, ft².

The relationship for full-face rounds was found to be:

$$N = 0.297A - 9$$

where

A and N are defined as before.

Blasting is performed using 75% strength ammonia gelatin dynamite, using LPV electric delay caps for firing.

Muck is removed using either skips or cylindrical buckets known as "kibbles" which are transported to the head-frame dumps in crossheads. The crossheads run on guides which are timber or wire ropes. The buckets are loaded either by a crawler mounted EIMCO 630 loader or a shaft mucker. Cryderman, and cactus grab (orange peel) types are the most common shaft muckers although other types such as the Riddell mucker and hydraulic backhoes are sometimes used.

Linings are usually framed timber sets, steel sets or concrete rings in rectangular shafts, while in circular shafts, monolithic concrete is most common. Shotcrete has occasionally been used as a shaft lining and compound steel and concrete linings have been used under severe conditions such as the Blairmore sands in Saskatchewan. With the exception of New Mexico, state regulations are not specific about the maximum distance which can be opened up between the face and the bottom of the lining. The Federal Metal and Non-metallic Mine Safety and Health Regulations (30 CFR 57) contain no regulations on this subject. On the other hand, in Canada, provincial mining acts generally allow no more than 50 ft of shaft to be unlined. For shafts which are to be used as ore passes, the most practical lining would be either monolithic concrete or shotcrete. Since abrasion from falling blocks would erode a concrete lining quite rapidly steel liner plates on the interior surface would be required as an additional measure.

There are two basic shaft sinking cycles - sink and equip concurrently and sink then equip. Since an ore pass does not need to be equipped the first does not apply except that it is the standard method of sinking a rectangular timbered shaft. A timber lining could readily be stripped, so that this method could be used to sink square ore passes up to 500 ft² in cross-section.

The overall project advance rates for a large number of North American shafts are given in Figure 25.

The overall project advance includes time required for mobilization, demobilization, head frame, sinking hoist erection, and commissioning.

Also shown in Figure 25, are estimates for the three most likely types of shafts for ore passes. So-called high speed shaft sinking refers to the method developed in South Africa, in which a multi-level Galloway stage suspended from

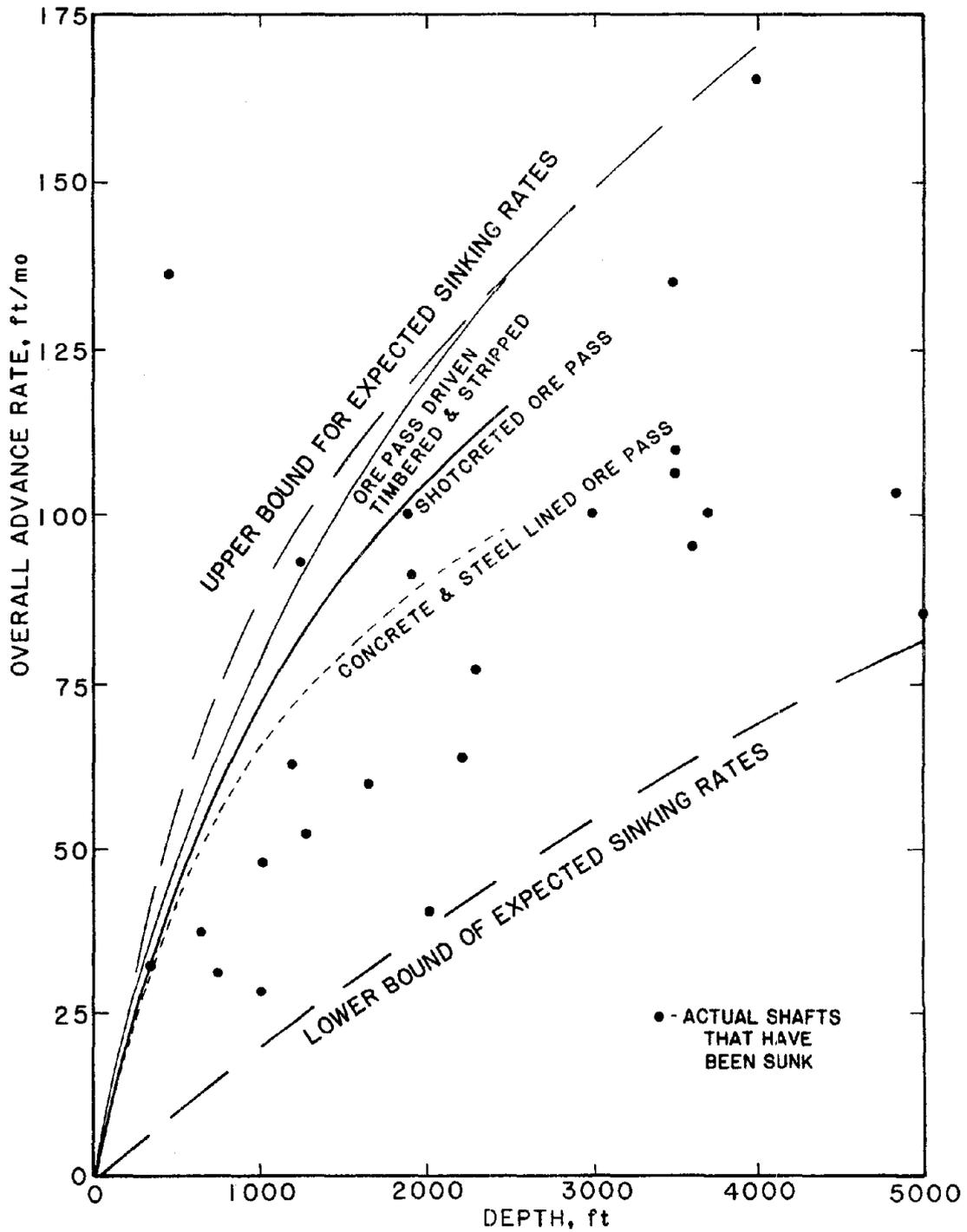


Figure 25 - Effective advance rate (over total project duration) for conventional shaft sinking for hard rock, salt and potash. [In hard rock the rates in the upper half of the envelope apply.]

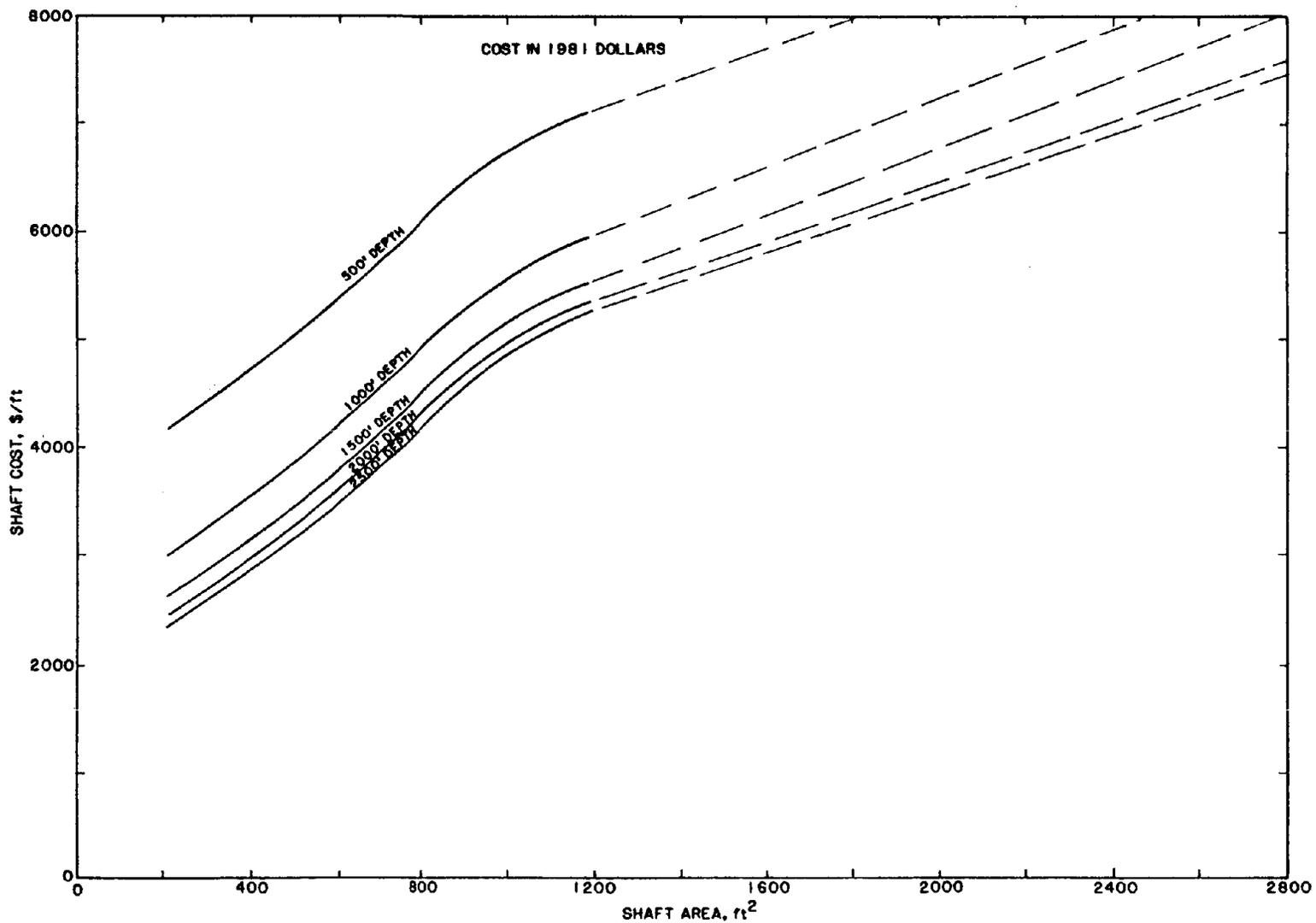


Figure 26 - Cost to sink an ore pass with concrete and steel plate lining using high speed sinking methods. Costs for freezing or pregrouting of less competent and water bearing strata are not included.

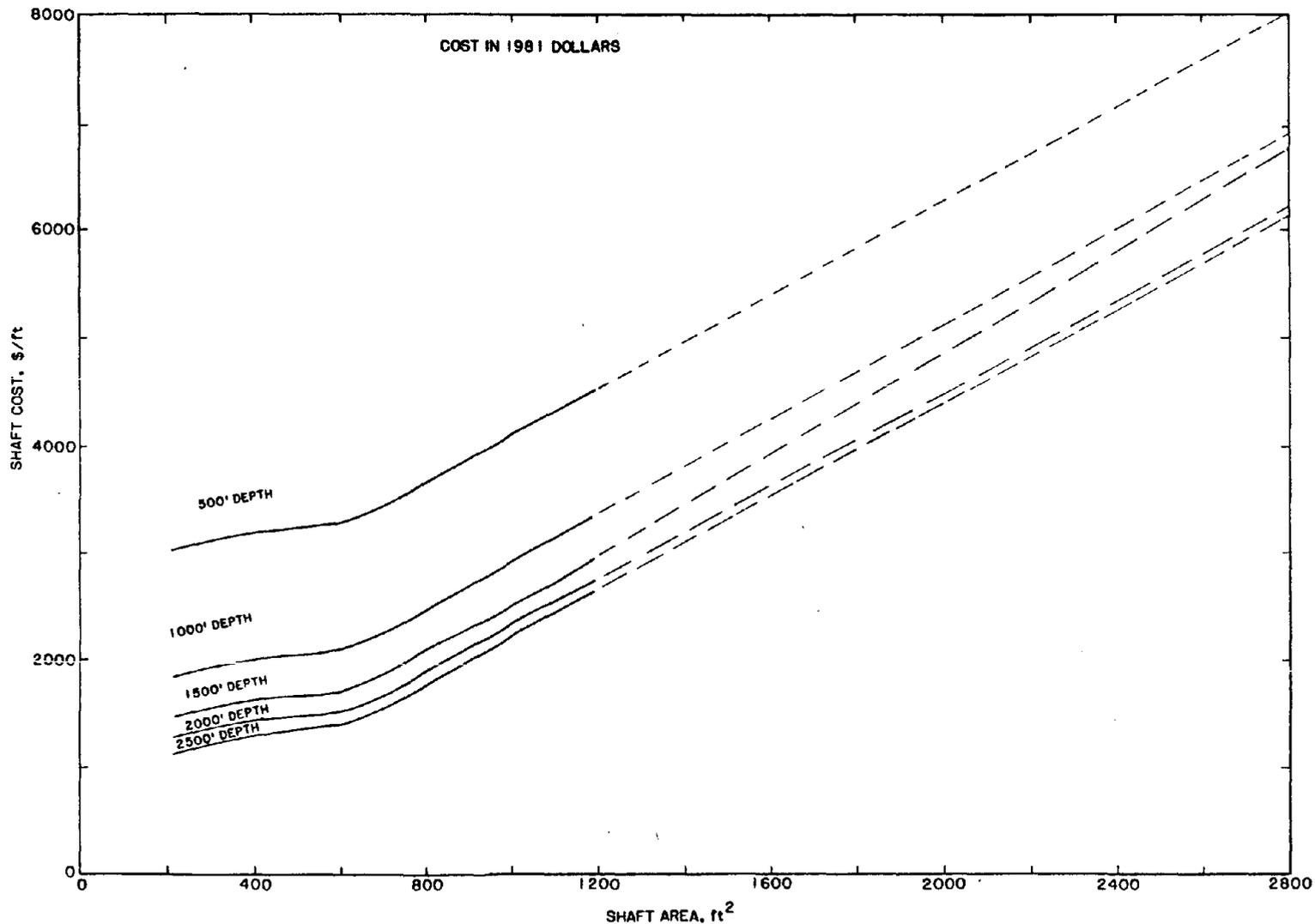


Figure 27 - Cost to sink an ore pass with a shotcrete lining using high speed sinking methods (sharp increase in cost at 600 ft² due to limitations of mucking equipment - up to that area the size of the clamshell could increase as required to reduce the mucking time and, hence, the cost)

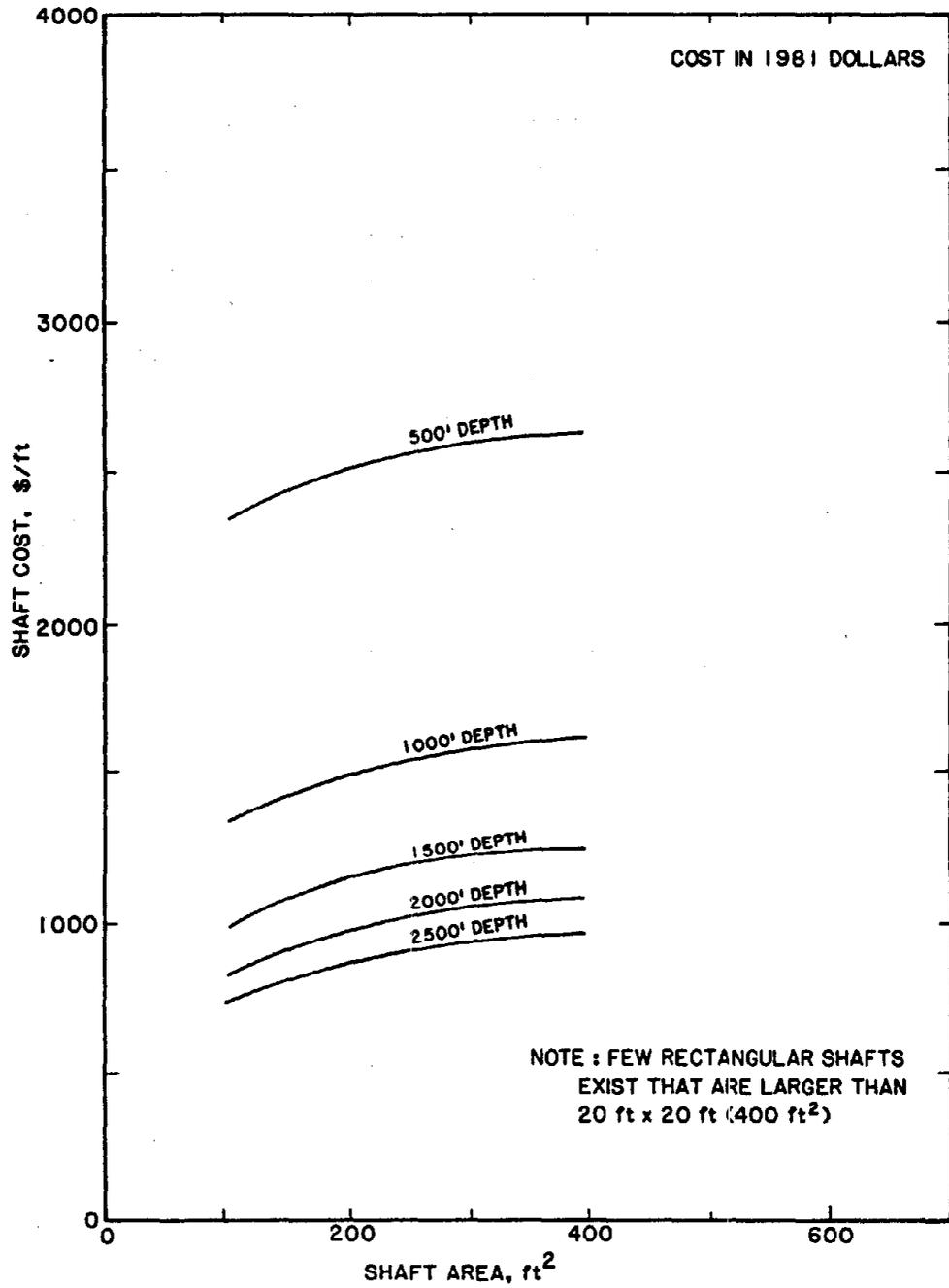


Figure 28 - Cost to sink a rectangular ore pass with a timber lining by standard sinking methods (lining later stripped)

Blair hoists is used for sinking and lining a circular shaft with equipping carried out subsequently. Mucking is done by a cactus-grab suspended below the stage.

Costs for sinking ore passes with concrete-and-steel, shotcrete, and (stripped) timber linings are given in Figures 26, 27, and 28 respectively. Shafts are generally unique entities; hence the full cost of much of the sinking plant - headframe, Galloway, buckets - must be charged to an individual project. This represents a substantial fixed cost and thus on a per foot basis, shallow shafts are costlier than deeper ones. The 2,500 ft depth was chosen as the maximum since only in exceptional cases will ore passes in open pits exceed this depth. Also, from Figures 26 through 28, it is apparent that as the depth increases beyond 2,000 ft, the effect of depth on the cost per foot decreases.

6.3 Raise and Slash

This method consists of driving a small pilot raise and then slashing to full size. Several methods could be used for the pilot, namely

- raise boring
- raise climber or lift
- conventional raising.

For the small ore passes used in underground mines these methods are used to drive the ore passes themselves; slashing being unnecessary except in the case of ore storage bins. However, for the large diameter ore passes which would be used in open pit mines, these methods alone could not produce large enough openings, and slashing would be necessary. Slashing a smaller opening is also attractive from cost and advance standpoints since slashing is a low-cost method which is also relatively rapid.

6.3.1 Raise Boring

Boring of a pilot raise is carried out in two stages - first, a pilot is drilled for the full length of the raise and second, the pilot is reamed to a larger diameter. The boring machine can be located either at the top or the bottom of the projected raise, but the former is much more common than the latter. In either case, however, the top and bottom of the projected raise must be accessible prior to commencement of drilling.

The raise borer is a rotary drill and its operation is similar to the rotary blast hole drills used in open pit mines, the main difference being the shorter mast and drill rod lengths with the raise borer necessitated by the low clearances underground as well as the much greater pulldown capacity on the raise borer. The pulldown weights on rotary and rotary-percussive blasthole drills are consistent with those used for the raise borer pilot, so an open pit operator could use one of his own blasthole drills for the pilot.

The greatest problem with raise boring is the tendency of the pilot hole to wander. This tendency can be overcome with careful drilling and for this reason it is common practice to limit the penetration rate to 10 ft per hour during pilot drilling although much greater penetrations could be otherwise achieved.

In the case of reaming, the pulldown weight is normally about 30 kips per cutter although recent papers have suggested that loadings of 15 to 20 kips per cutter result in more cost effective drilling due to longer cutter and bearing life although penetration rate is reduced (Hendricks, 1978). The number of cutters on a reamer head can be estimated as follows (Uski, 1981):

$$c = 2R - 2$$

where

c = number of cutters

R = diameter of reamer head, ft.

To estimate reamer penetration rate, the following formula developed for rotary blasthole drilling (Bauer, 1970) was employed:

$$p = \frac{(61 - 28 \log_{10} C_o) L (\text{RPM})}{300R(x12)}$$

where

p = penetration rate, fph
 C_o = compressive strength of rock, ksi
 L = pulldown weight, kips
 R = diameter of the reamer head, ft
 RPM = speed of rotation.

The rate of rotation varies for machines of different manufacture but can be assumed to be about 10 rpm for estimating penetration rate. Using this value together with $(30 \times 2R)$ for the pulldown weight - simplifying thus the expression for the number of cutters and hence pulldown weight does not significantly affect the penetration rate for raises larger than 8 ft diameter - substituting these values into the penetration rate equation and simplifying, we obtain:

$$p = \frac{(61 - 28 \log_{10} C_o)}{6}$$

all variables as defined previously.

The overall advance rate for raise boring, including both pilot and reaming, is given as a function of compressive strength in Figure 29. Based on equipment costs, machine lives, and salvage values furnished by manufacturers, rental costs were determined for raises of different lengths as shown on Figure 30. Estimates of the direct costs for raises of different sizes are given in Figure 31. These costs include pilot hole bit and cutter costs, operating labor, and maintenance costs. The deepest bored raise has been drilled to a depth of 2,298 ft, with a diameter of 12 ft. The largest bored raises which have been successfully completed were 16 ft in diameter and about 1,500 ft long (Hendricks, 1978). The usual range of raise borers' applicability are raises less than 12 ft in diameter and 1,500 ft in length (Hendricks, 1978).

The advantages of bored raises are:

- smooth circular perimeter
- no blast damage around the perimeter of the opening
- safety - no workers are exposed to the face (especially important in bad ground)
- speed - except for raises less than 50 to 100 ft in length, bored raises can be constructed significantly faster than by drill-and-blast methods.

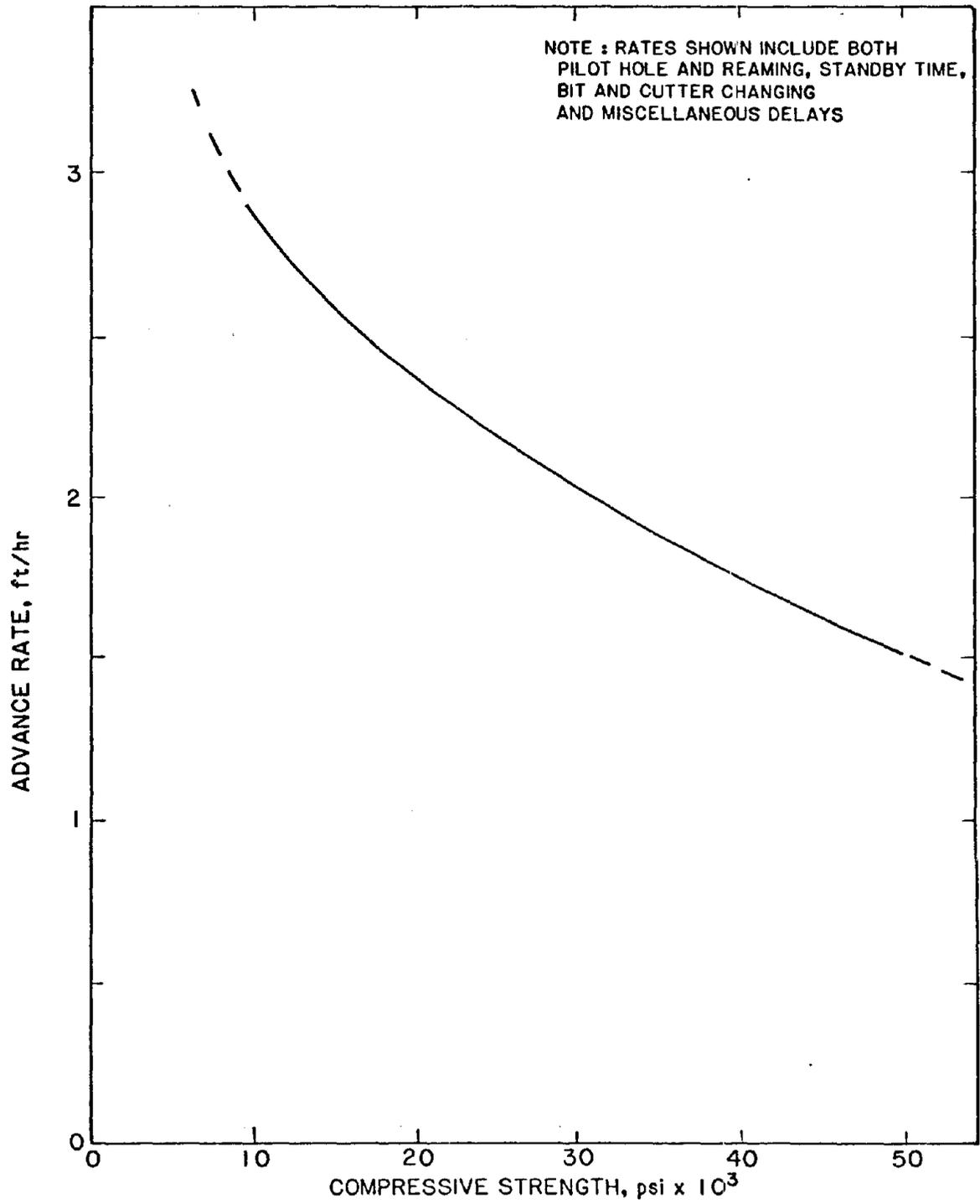


Figure 29 - Overall raise boring advance rate

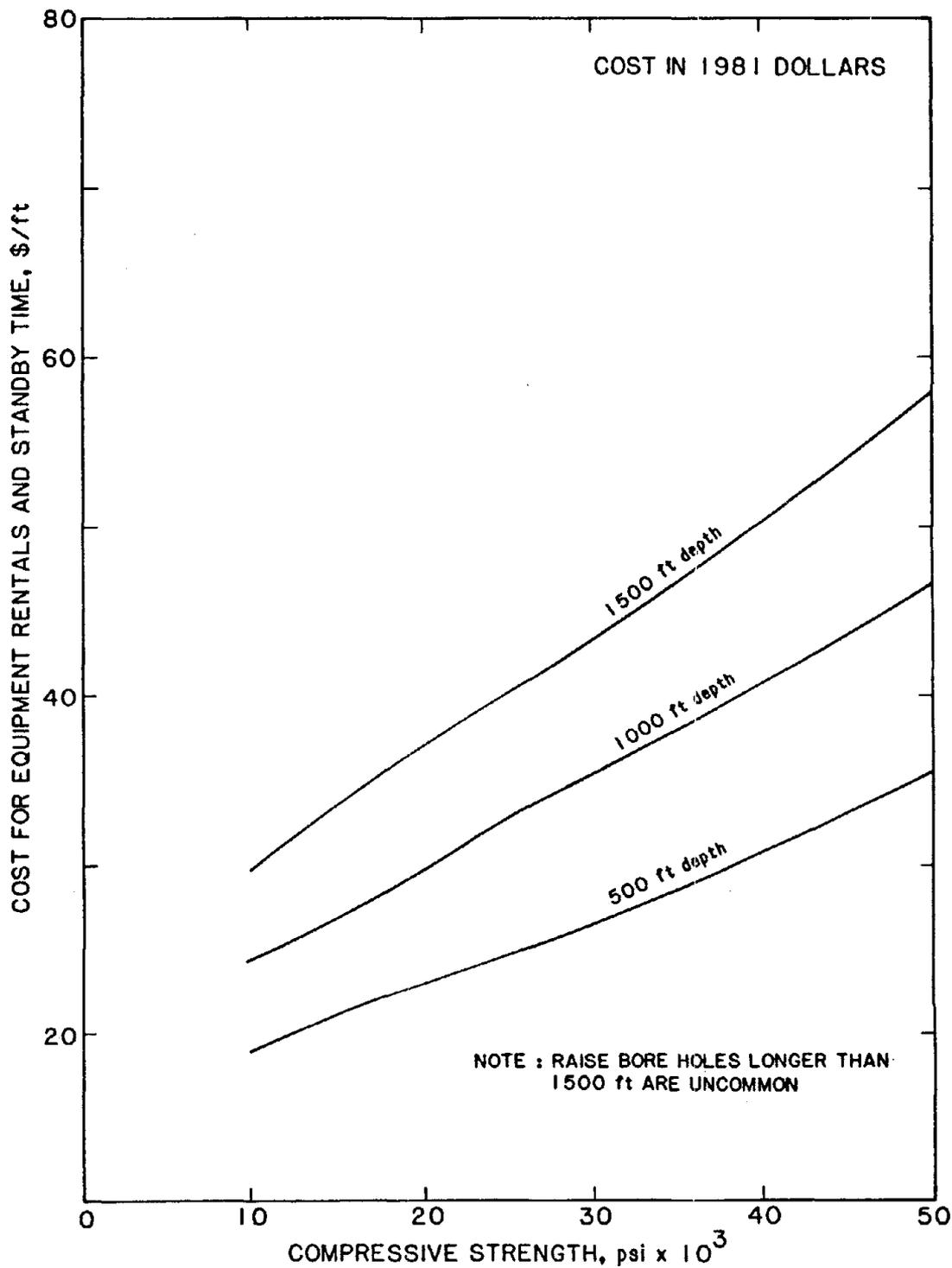


Figure 30 - Raise boring costs: equipment rentals, standby labor, and delays as a function of compressive strength (based on advance rates, bit life estimates and manufacturer supplied equipment costs)

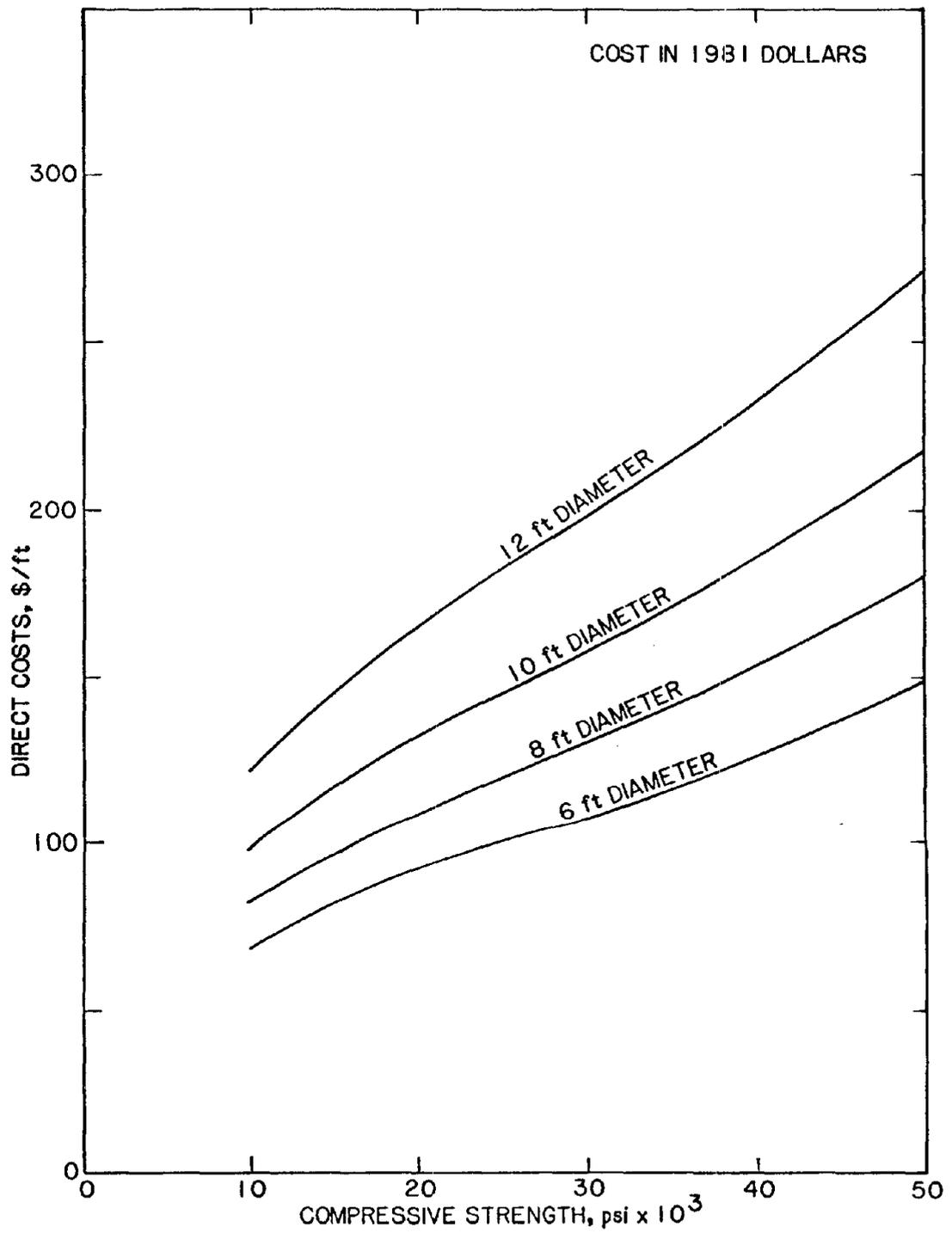


Figure 31 - Direct costs for raise boring

The disadvantages of using raise borers are:

- cost - raise-bored raises are significantly more expensive than drill-and-blast raises of a similar size
- on long raises large deviations from planned line and orientation can occur.

6.3.2 Alimak Raise Climber and Jora Lift

The Alimak Raise Climber and the Jora Lift were developed in Sweden in the 1950's by Linden-Alimak AB and Atlas Copco AB respectively. In both cases, the raise is advanced by drill-and-blast using hand held stopers - the raise climber and lift are used to transport miners to the face and to provide a working platform for drilling.

The Alimak Raise Climber (Alimak for short) consists of a small cage with a platform on top. Mounted on the cage is a small air motor which drives a pair of pinions which grip a rack on a rail bolted to the hanging wall of the raise. The rail incorporates 4 one-inch service pipes so that only short hoses are needed. For very long raises, electric and diesel powered models are available. Alimak has been used to drive pilots for large ventilation raises up to 2,500 ft in length with good success. Alimak is used for raises from 5 ft by 6 ft, to 11 ft by 11 ft in cross-section.

The Jora Lift also consists of a small cage with a platform on top. In this case, a winch mounted in the cage is used to raise the lift on a cable. The cable slung from the top of the raise through a pilot hole. The limiting factor for the length of raises to be driven with a Jora Lift is the accuracy with which the pilot can be drilled, which according to Rogert and Anderson (1975) limits raises to 325 ft.

With both Alimak and Jora lift, advance rates of 6 ft per shift are quite feasible. Costs are shown on Figure 32.

6.3.3 Conventional Raising

Conventional raising refers to raises advanced by drill-and-blast methods where access to the face is by ladder, and drilling is done from a wooden platform located about 8 ft below the face. There are two types of conventional raises: open (or "bald-headed") raises, and cribbed

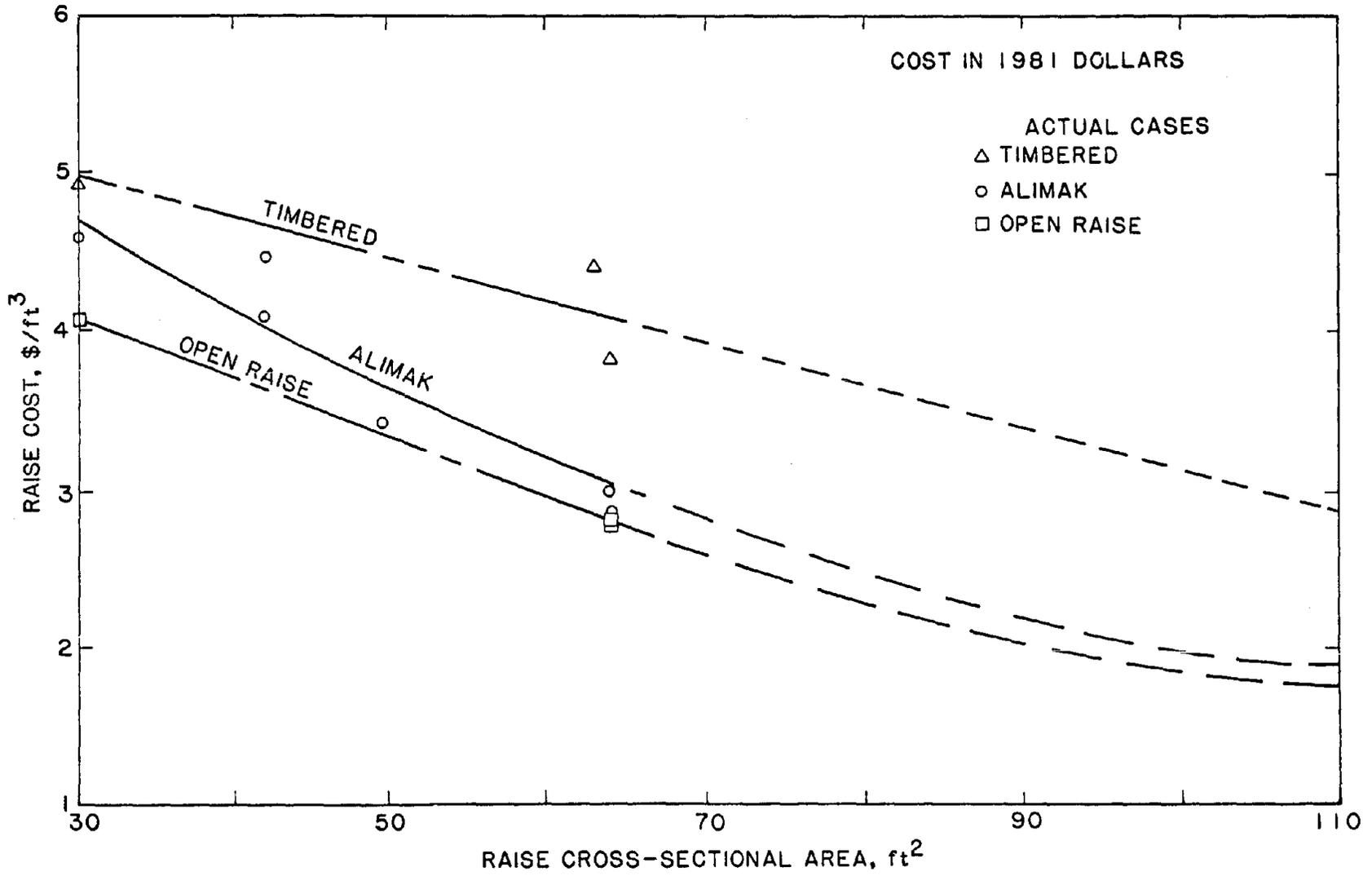


Figure 32 - Costs for pilot raising methods

raises. There are no Federal or State mining regulations in the United States which limit the use of either type of raise. In Canada, on the other hand, mining regulations in most provinces require raises inclined at more than 50° from the horizontal and longer than 60 ft slope distance, be divided into two compartments, one of which must be maintained as a travelway and equipped with ladders. Canadian regulations thus preclude the use of open raises for ore passes longer than 60 ft. Conventional raises are generally not practical for lengths exceeding 150 to 200 ft.

Costs for open raises are given in Figure 32. Advance rates of 3 to 5 ft per shift are common, with timbered raise advances being on the low end of the range.

6.3.4 Slashing the Pilot Raise

When we examined a large number of raise and slash case histories, it became apparent that there is a good correlation between the size of the pilot and the overall cross-sectional area. The relationships for conventional and Alimak pilots, and for bored pilots are shown in Figure 33. However, as it is recognized that openings of specific size would be used rather than continuously increasing areas, the regression lines have been replaced by step functions.

The area to be slashed is obviously the difference between the total area and the area of the pilot. In Figure 34, advance rates are given for slashing a pilot to full size as a function of the area of the slash. These advance rates were obtained from actual case histories.

Slashing of a pilot raise can either be carried out from the top down or from the bottom up. An example of a sinking stage for excavation from the top down is shown in Figure 35. Slashing from the bottom up is done in a manner similar to shrinkage stoping. Representative costs for slashing are given in Figure 36.

6.4 Drop Raises and Vertical Crater Retreat

Drop raising is the term applied to a method of raise excavation in which holes are drilled downward to the full length of the projected raise and then loaded and blasted in stages from the bottom upwards. Prior to the introduction of down-the-hole drills in the early 1970's, this method employed 2-1/8 in. diameter long holes drilled with an Airtrak or a bar-and-arm mounted drifter drill. Because of the tendency of the holes to wander, raises, driven by this

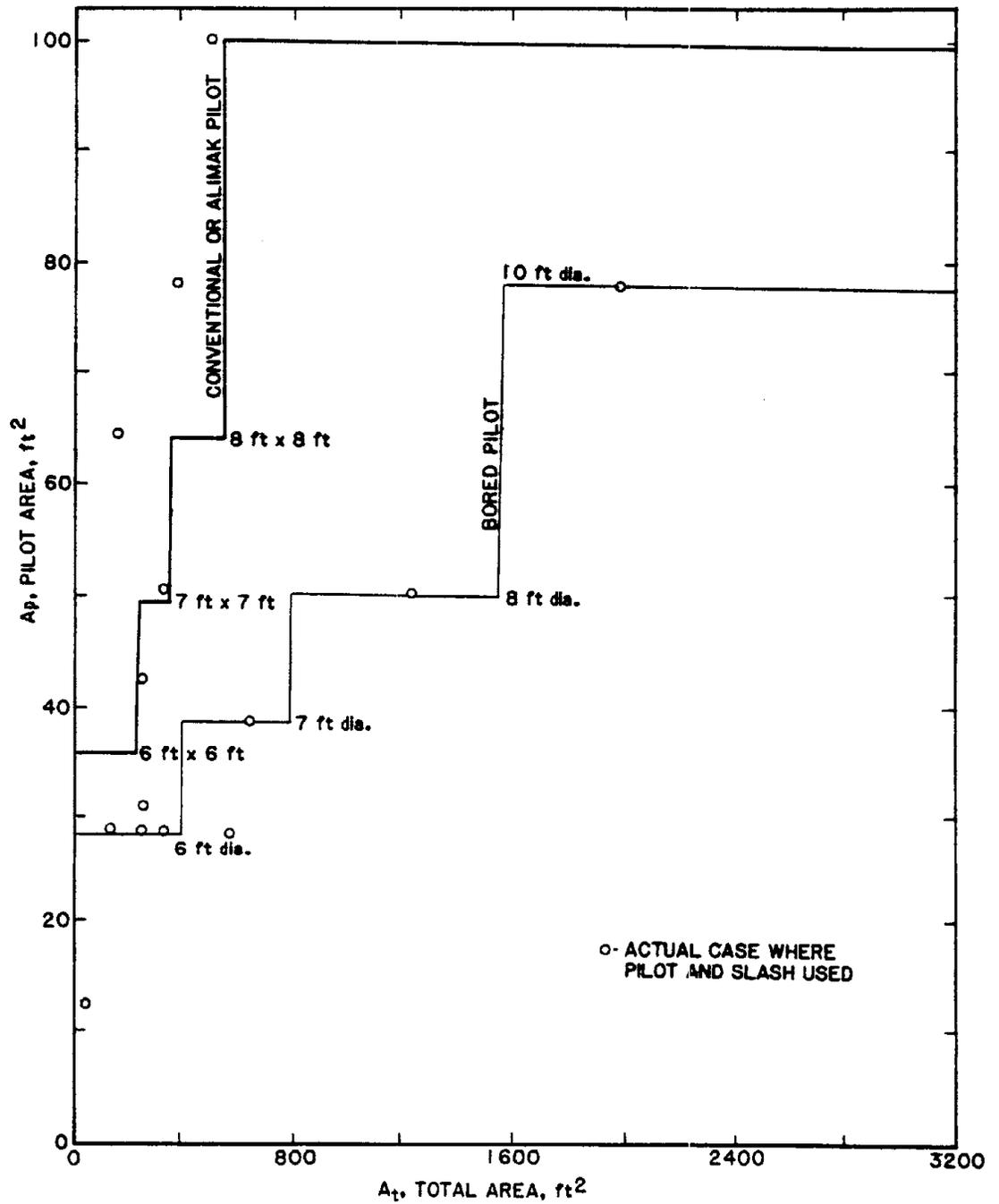


Figure 33 - Determination of pilot size for ore pass driven by raise and slash

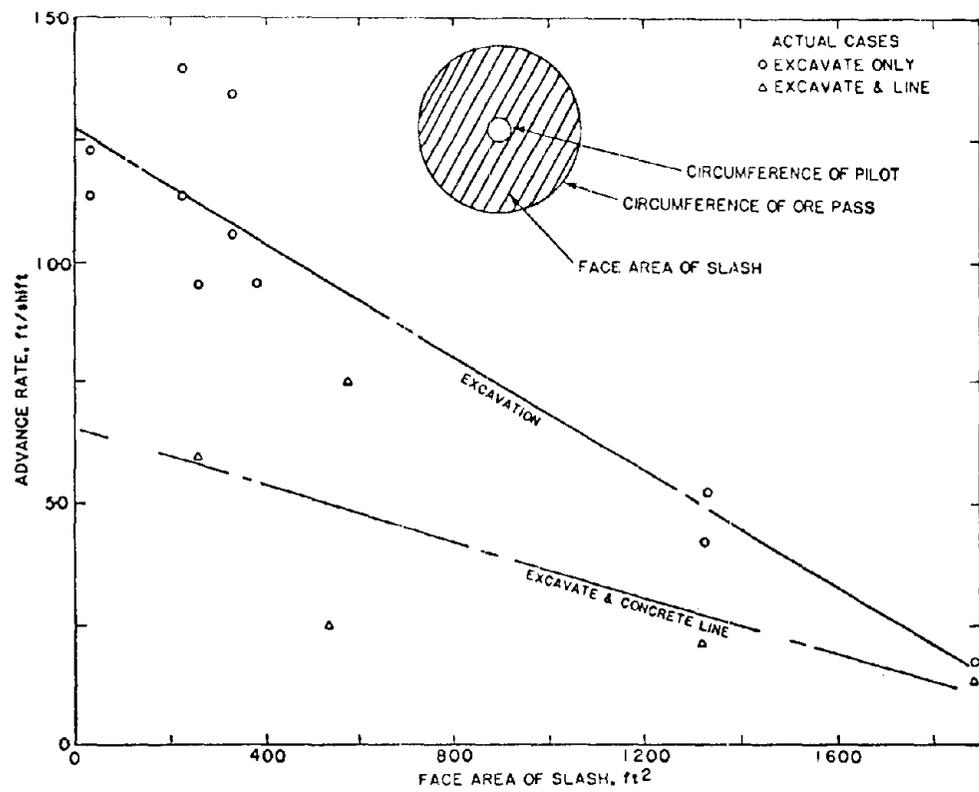


Figure 34 - Advance rate for slashing a pilot raise to full size

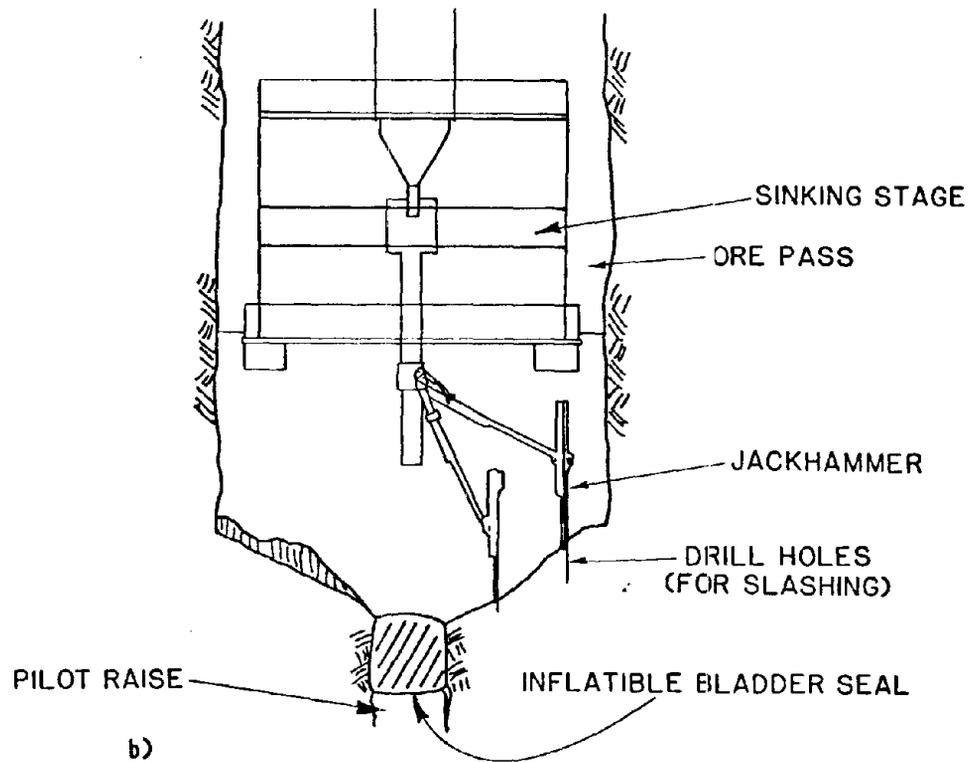
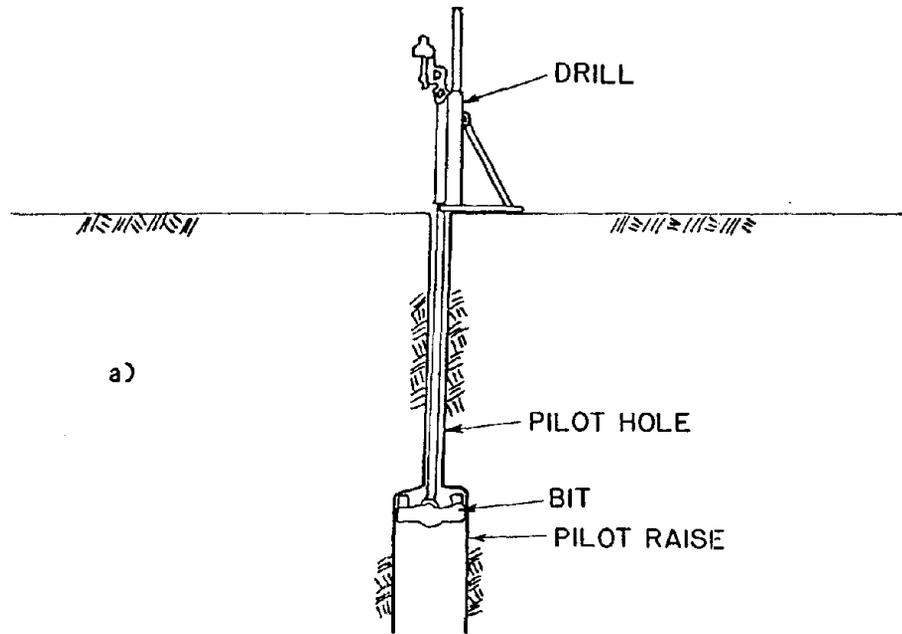


Figure 35 - Raise and slash
 a) Raise boring the pilot raise (reaming the pilot hole)
 b) Slashing to full size

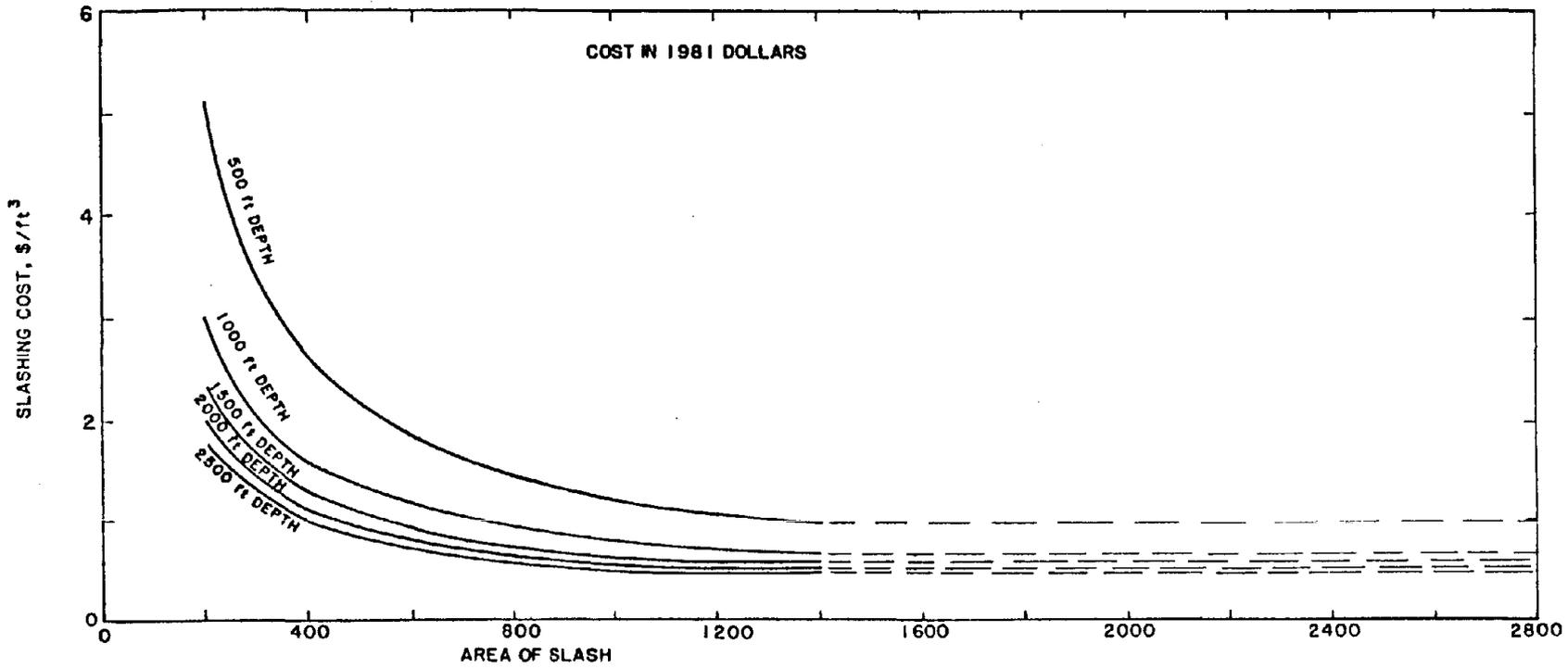


Figure 36 - Cost to slash a pilot raise to full size with a shotcrete lining

method are limited to about 70 ft in length. Use of the Atlas Copco Simba 5 raise drilling rig allows raises up to 150 ft to be driven, since its design results in less hole wander (Meagher, 1976).

The introduction of down-the-hole drills underground permitted holes up to 6-1/2 in. in diameter to be drilled. This meant considerably longer raises could be contemplated as well as blasting with spherical charges. These developments have led to a new mining method called Vertical Crater Retreat (Lang et al, 1977). This method has been used for a large shaft having a length of a few hundred feet, at a coal mine in Australia, and is potentially attractive for large diameter ore passes several hundred feet long, which would be required in large tonnage open pit mines. VCR is characterized by low costs and rapid advance rates.

A further attraction to open pit operators is the fact that their own equipment and personnel could be used to excavate ore passes by this method.

In such cases, operators would also be well able to perform their own cost estimates for ore pass construction based on drilling and explosives costs. Hence, we have not endeavored to provide cost or advance estimates.

6.5 Shaft Boring

Shaft boring has been used to sink small diameter shafts in soft rocks since the introduction of the Honigmann Method about 50 years ago. Other shaft boring methods for coal and other soft rocks include:

- rotary table drilling
- downhole drilling
- reaming.

These methods are generally limited to rock having compressive strengths less than about 10,000 to 15,000 psi.

Rotary table drilling employs oil field type equipment and can be used for shafts up to 12 ft in diameter. This method has been and is commonly used for coal mine shafts in the United States as well as elsewhere. The method is limited to shafts less than 12 ft in diameter.

Reaming refers to the Turmag shaft boring method which was developed in Germany about 1973. In this method, a conical reamer enlarges a pilot hole from the top down. A

platform on top of the reamer is used to line the shaft as reaming progresses downward. The method is limited to diameters less than 18 ft and is rather slow - advances average about 15 ft per day in German mines (Henneke and Wallussek, 1981) - compared with downhole boring. In Germany, it is cost effective for depths less than 800 ft.

Downhole drilling consists of using a boring machine in a shaft. At present, two companies make such machines: The Robbins Co., and Wirth Maschinen - and Bohrgerate-Fabrik GmbH. The Robbins machine bores holes 24.5 ft in diameter, and is designed for use in coal measure rocks, and is presently only a prototype. Wirth on the other hand has built several machines which have been used successfully in the United States and Germany for shafts up to 26 ft in diameter by the contractors Thyssen, and Deilmann-Haniel (Bruemmer and Wollers, 1976). The Wirth machines are also limited to coal measure and other soft rocks.

Recent West German literature (Grossekemper and Tonscheidt, 1980) indicates that raise boring has superseded all other methods for diameters up to 16.5 ft and lengths up to 1,600 ft in their mines.

Since shaft boring is a specialized field, it is recommended that anyone contemplating its use contact contractors engaged in this field, in order to discuss costs and advance rates for individual projects.

6.6 Summary of Construction Methods and Costs

There are a number of methods which can be used to drive ore passes, including:

- shaft sinking
- raise and slash
- Vertical Crater Retreat
- shaft boring.

For a given situation, the applicability of each method listed above is determined by a number of factors, including:

- ore pass diameter
- ore pass depth
- compressive strength of the host rock

- availability of access to the bottom of the proposed ore pass.

For the raise and slash option, the pilot raise may be excavated by one of several methods:

- conventional raising
- raise climber
- raise borer.

Data have been furnished to allow a planner to prepare cost and time estimates for all the methods except Vertical Crater Retreat and shaft boring. An open pit operator can prepare estimates for VCR based on his own drill-and-blast experience. In the case of shaft boring, this information can be obtained from a qualified shaft boring contractor.

7.0 SUMMARY AND CONCLUSIONS

The objective of this contract was to compile a manual for the design of vertical and inclined ore pass systems for open pit mines. The work performed was divided into several distinct efforts:

- information search
- development of rock flow parameters
- development of criteria for location of ore passes
- development of criteria for ore pass design
- construction methods and costs
- design of ancillary facilities
- operational considerations.

The information search comprised a literature search, visits to mines, and conversations with planning and research engineers in order to compile as large a data base on ore passes and ore pass design practices as possible.

An ore pass cannot function if the broken material in it does not flow. The three impediments to rock flow are block arching, cohesive arching, and piping, and prevention of these should result in satisfactory rock flow.

The location of an ore pass is important, and provided that there are no locally-imposed overriding constraints, should be such that in-pit haulage distances are minimized. The location that satisfies this requirement is the centroid of haulage. If multiple ore passes are required, the pit should be divided into equal-sized areas with an ore pass at the centroid of each area. There may be constraints on location such that ore passes cannot be located at the centroid, so that the actual location(s) chosen depends on the operator's philosophy and preferences.

For the purposes of design, an ore pass can be divided into three sections:

- the dump point
- the draw point
- the ore pass proper.

Design of the ore pass proper is dependent on two considerations - rock flow and support requirements. Design of the dump point is dependent on truck dimensions, requirements for barriers, and requirements for foundations and liners. Design of the draw point is dependent on the size of the downstream haulage facilities and on flow requirements. Two types of draw points can be envisaged - chutes, and hoppers and feeders, with the latter more likely.

A number of different methods could be used to construct an ore pass depending on whether or not access to the projected bottom of the ore pass is available. The possible methods are:

- shaft sinking
- Vertical Crater Retreat
- raise and slash
- shaft boring.

For raise and slash, the pilot raise could be driven by:

- conventional raising
- raise climber
- raise boring.

Advance rates, costs, limitations and advantages of all the above alternatives have been discussed.

There are several considerations which while not directly affecting the design, must be included in planning for an ore pass system. These include:

- dust control
- safety
- monitoring systems.

These are not discussed here, but mentioned in the design manual in a cursory fashion, as each of these is a subject for which requisite guidelines are available elsewhere.

For the benefit of an operator who is unfamiliar with ore pass operations, a section concerning operating procedures has been included in the manual which forms Volume 2 of this report. Routine day-to-day operations are discussed as well as the non-routine situations of hang up removal, and lowering of the dump point.

In conclusion, using the manual developed for this project, an operator has sufficient information to rationally design an ore pass system for an open pit mine.

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