1a

Information Circular 8663

Hydraulic Sandfill in Deep Metal Mines

By Lewis M. McNay and Donald R. Corson
Spokane Mining Research Center, Spokane, Wash.

UNITED STATES DEPARTMENT OF THE INTERIOR
Rogers C. B. Morton, Secretary

Jack W. Carlson, Assistant Secretary—Energy and Minerals

BUREAU OF MINES
Thomas V. Falkie, Director

ii

As the Nation's principal conservation agency, the Department of the Interior has responsibility for most of our nationally owned public lands and natural resources. This includes fostering the wisest use of our land and water resources, protecting our fish and wildlife, preserving the environmental and cultural values of our national parks and historical places, and providing for the enjoyment of life through outdoor recreation. The Department assesses our energy and mineral resources and works to assure that their development is in the best interests of all our people. The Department also has a major responsibility for American Indian reservation communities and for people who live in Island Territories under U.S. administration.

This publication has been cataloged as follows:

McNay, Lewis M

Hydraulic sandfill in deep metal mines, by Lewis M. McNay and Donald R. Corson. [Washington] U.S. Bureau of Mines [1975]

63 p. illus., tables. (U.S. Bureau of Mines. Information circular 8663)

Includes bibliography.

1. Hydraulic filling. 2. Mine filling. 3. Mining engineering. I. U.S. Bureau of Mines. II. Corson, Donald R., jt. auth. III. Title. (Series)

TN23.U71 no. 8663 622.06173

U.S. Dept. of the Int. Library

For sale by the Superintendent of Documents, U.S. Government Printing Office
Washington, D.C. 20402 – Price $1.25
Catalog No. 128.27:8663
Stock No. 2404-01721

ie

CONTENTS

ILLUSTRATIONS

ILLUSTRATIONS--Continued

TABLES

HYDRAULIC SANDFILL IN DEEP METAL MINES

by

Lewis M. McNay[1] and Donald R. Corson[2]

ABSTRACT

The Bureau of Mines and industry (mining and private) have studied the placement of hydraulic sandfill in deep-vein mines to identify existing problems and to develop techniques to strengthen the support characteristics of the sandfill. The hydraulic sandfill system, physical properties of typical classified mill tailings, and properties of sandfill modified by various techniques are discussed. Computer analysis has defined the support capability of various sandfills using actual mine field data as a reference base. High-modulus sandfill can provide adequate support in deep-vein mines to alleviate rock and pillar stresses and to reduce rock-burst incidence.

INTRODUCTION

Artificial support techniques in metal mines have undergone significant evolution because of increased depth of mining, increased mining costs, desire for total ore extraction, greater emphasis on improved safety in mines, and technology advancement. Up to the middle of the 19th century, metal mining was primarily restricted to the various surface mining techniques, although limited underground metal mining was practiced. Underground mines were small; many times not being much more than mere openings or passageways through which miners could crawl and dig along the ore vein. Because ground conditions were often poor, miners would occasionally drag timber through the small openings and prop the timber against the overlying rock for support.

By 1850, great interest was brought forth to discover and develop the "thought-to-be" fabulous "mother-lode" deposits within the earth's depths. In this era of early deep mine development, mining was accomplished in unsupported or crudely supported tunnels, drifts, stopes, and rooms. The need for new methods of ground support magnified as ground conditions continued to deteriorate and roof and rib rock falls intensified. For many years, mine shoring and rock support were comprised of placing and bracing increasing numbers of timbers in the mined areas (29).[3]

---

[1] Geologist.
[2] Supervisory mining engineer.
[3] Underlined numbers in parentheses refer to items in the list of references at the end of this report.

A major advancement in ground support technology occurred in 1860 with the development of the timber square-set. The square-set was designed and first used by Philip Deidesheimer at the Comstock Lode, Nevada (27). The design of this support system is still widely practiced today with timber sometimes being replaced by concrete and steel components. Later, waste rock (commonly referred to as gob) was placed in mine openings along with square-sets to provide additional support. Some mines continue to practice this support technique.

The next significant advancement in underground support has been hydraulic transport of solids. This technique has been practiced in the mining industry for 120 years. The concept of hydraulic movement was conceived as a result of the necessity to recover greater quantities of metals from alluvial materials in dredging and placer mining. Hydraulics were first used to place mine waste underground at Shenandoah, Pa., in 1864 (3). A priest successfully devised a scheme to place crushed coal waste in the mined area underlying his church to prevent land subsidence. By the late 1880's, hydraulic fill was used in anthracite mines to control underground fires and support heavy ground. At this time, European engineers became interested in hydraulic transport of waste and coal. Hydraulic transport in mining operations has had its greatest application in European and Indian coal industry.

The classification of mill tailings and the hydraulic or pneumatic transport of the coarse sand fraction has been used in metal mines. The use of mill sand as a support media has been employed as gold mines in South Africa since the early 1900's. Originally, the sandfill was used to support weakening pillars around shafts and level stations. Later, the sandfill was placed in mining areas to support heavy ground. One of the first metal mines in North America was the Matahambre in Cuba, where sandfilling was initiated in 1928. The Homestake mine, South Dakota, successfully initiated sandfilling to replace gobbing practices in the late 1930's (28).

Hydraulic filling of mine voids has been expressed by several idioms, dependent upon local terminology. The terms "hydraulic sandfill" or "sandfill" will be used to identify the hydraulic placement of fill in underground mines; other synonymous terms are "hydraulic stowing," " backfill," "tailings fill," and "slime fill."

The rapid and effective sandfilling with the coarse sand fraction coupled with other technological developments, such as the rock bolt, has resulted in dramatic improvement in mines utilizing cut-and-fill mining methods. The principal advantages of hydraulic sandfill over the traditional timber square-set and waste-rock filling methods are as follows:

1. Improved ground control.

2. Faster development, advancement, and extraction of ore from stopes due to the rapid filling with sand.

3. A more effective and economical method of transporting a ground-support material underground.

4. Fuller extraction of ore due to the improved support methods and the reduction in ore loss.

5. Improved ventilation control.

6. Decreased fire hazard.

An additional benefit, which has only recently been appreciated with the utilization of mill waste for underground fill, is the significant reduction in the volume of mill waste requiring impoundment behind surface tailings dams.

However, along with the improvements offered by hydraulic fill techniques over timber setting and gobfill in stopes, several principal support problems and disadvantages are yet to be solved. Among the important drawbacks that result from hydraulic backfill are the following:

1. The large volumes of water used to transport tailings must eventually be pumped out of the mine.

2. Haulageways and drainage ditches are fouled and filled with fine slimes that decant from the filled stopes with the transport water.

3. Spillage of fill from stopes due to piping (erosion) induced by imperfect sealing of the stope, high hydrostatic head, or malfunction of the hydraulic system creates additional maintenance and cleanup costs.

4. During the ore mining process, ore may become mixed with valueless sandfill material creating a finite volume of dead-load material in the ore stream.

5. Even though hydraulic fills are a substantial improvement over waste rock fill or open stoping, present hydraulic fills are not providing adequate support in many mining operations. This is evidenced by the development of rock bursts in ore pillars and/or instances of additional support (timbers) requirements in order to assure full extraction of the ore. Failure to obtain full extraction promotes poor conservation of valuable mineral resources.

The mining industry has made gradual improvements in its filling systems, such as cementation or capping of fills to reduce dilution of ore. In general, however, the filling systems continue to exist as they were in the 1950's and 1960's. At the same time, mine support problems have increased due to the greater mining depths and/or effect of large increases in mined ground in each mining area. It is becoming increasingly obvious that improved support, to restrict induced ground movements, will be required in many active metal mines as the depth and extent of the operations become even greater. This improved support must come from an improved sandfill that will be capable of supporting load essential to control excessive ground movement; thereby reducing the incidence of rock burst and other ground control problems.

The Bureau of Mines has been involved in a research program to study hydraulic sandfill and its support potential since 1961. Results of some of this research have included the determination of the physical properties of sandfill, how these properties vary with the use of chemical additives and compaction methods, the influence of various factors on water percolation rates, and field measurements of stope wall deformations and pressures within the fill. Other research included related rock mechanics studies at several mines in the Coeur d'Alene district.

It is the purpose of this investigation to draw together pertinent, current information pertaining to hydraulic sandfilling at metal mines and to discuss its capability to function and perform as a competent artificial ground support medium. The discussion will be directed to practical investigation, both in the laboratory and the field, and to the development of analytical techniques to conduct theoretical modeling of requirements and effectiveness of hydraulic sandfill.

MINING TECHNIQUES USING HYDRAULIC SANDFILL

Hydraulic filling with sand, tailings, coal waste, ground rock, or slag has developed into a highly useful means of ground support in many mining operations throughout the world. Fill applications are today found in all types of underground coal, metal, and nonmetal mines. In the United States and Canada, hydraulic sandfilling has, historically, been associated with its use in stoping operations. The introduction of fill material, however, has had wide application in European and Asian coal mines.

According to 1971 Bureau of Mines statistics, over 40 metal and nonmetal mines in the United States introduce hydraulic sandfill to some extent (table 1). Of these, 25 operations completely fill the openings or voids produced by ore excavations. The greatest number of mines using 100-percent sandfill are the nation's prime producers of gold, silver, lead, and zinc. These mines produced in 1971 the following percentage of the total U.S. production of gold, silver, zinc, and lead, respectively: 35, 50, 10, and 12.

In the United States, hydraulic sandfilling is used primarily in stope mining (27-28). Stoping is a method of mining in which the ore is excavated in a series of steps or floors along a steeply pitching to vertical ore body. Each series of steps is driven horizontally along the strike of the vein. Many variations of stope mining are practiced: sublevel, shrinkage, overhand, and underhand. The following statements, however, will briefly describe one type of stoping referred to as "cut-and-fill" stoping.

TABLE 1. - List of mines using cut-and-fill method of mining

Mine	Location	Principal commodity
(1)	Alaska	Gold.
Bruce	Arizona	Zinc.
Copper Queen	.....do	Copper.
Magma	.....do	Do.
Yucca Gravel	California	Talc.
Black Cloud	Colorado	Lead.
Bulldog	.....do	Silver.
Rico Argentine	.....do	Zinc.
Admiral	Idaho	Do.
Bunker Hill	.....do	Lead, zinc.
Crescent	.....do	Silver.
Dayrock	.....do	Lead.
Empire	.....do	Copper.
Galena	.....do	Silver.
Highland	.....do	Zinc.
Lucky Friday	.....do	Lead.
Star-Morning	.....do	Zinc.
Sunshine	.....do	Silver.
Illinois	Illinois	Fluorspar.
Cadgie-Tay	Montana	Silver.
Commanche	.....do	Do.
Drystal	.....do	Do.
Mtn Con	.....do	Copper.
True Fissure	.....do	Zinc.
Bristol	Nevada	Copper.
Cahill	.....do	Mercury.
Mina Mercury	.....do	Do.
Costerling	New Jersey	Zinc.
Bonney	New Mexico	Copper.
Tungsten	North Carolina	Tungsten.
Unknown	Oregon	Copper.
Unknown	.....do	Gold.
Homestake	South Dakota	Do.
Callaway[2]	Tennessee	Copper.
Burgin	Utah	Lead.
Butterfield	.....do	Silver.
Mayflower	.....do	Gold.
Plasterco	Virginia	Gypsum.
Knob Hill	Washington	Gold.
Rox	Wyoming	Uranium.
Seismic	.....do	Do.

[1]Little Squaw Gold Mining Co.
[2]Research on sandfill has been conducted; sandfill has not been a traditional operational phase.

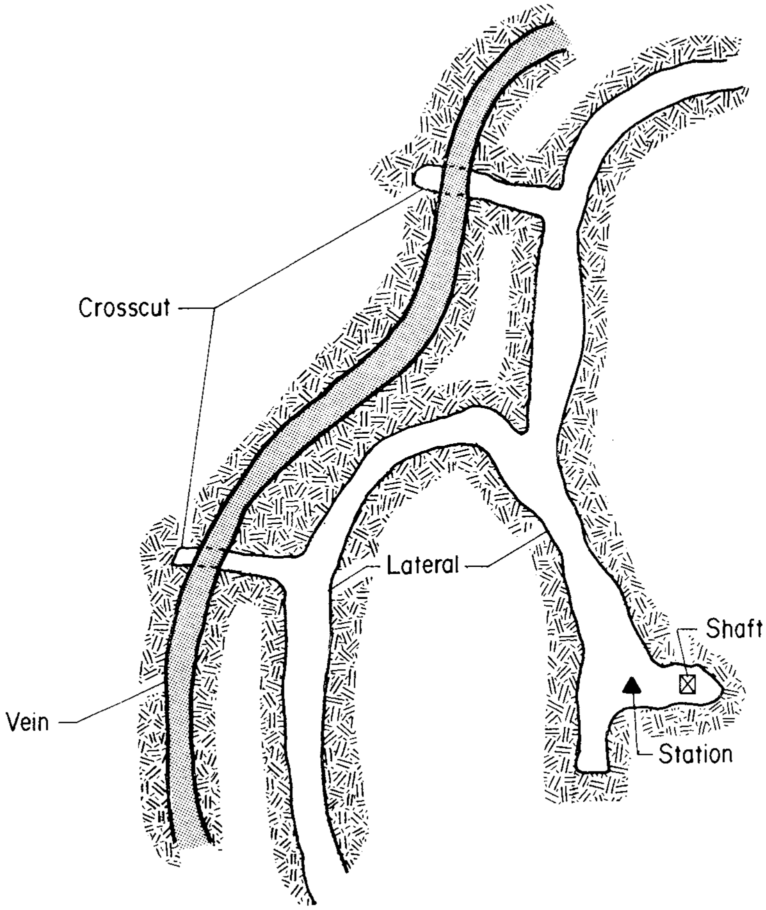

FIGURE 1. - Generalized view of a development level in a cut-and-fill stope mine.

Cut-and-fill stoping occurs when the ore is excavated along horizontal floors or slices, working upward at the completion of each floor. The generalized development of a cut-and-fill stope is as follows (fig. 1):

1. Tunnellike passages are driven from each level station to the ore vein. Depending on the location of the heading with respect to the vein, the openings are termed laterals, drifts, or crosscuts. In some mines the primary openings are driven on the vein along the footwall with crosscuts to the vein, or crosscuts from the shaft to the vein.

2. At predetermined intervals (normally 200-foot distances), a raise is developed in a vertical direction along the ore body. The interval between raise areas is often determined by fault or fracture patterns which interrupt the ore body. A raise may consist of as many as four compartments: a manway, a timber chute, and two ore chutes to service stopes on either side (fig. 2). Three compartments are usually installed. The initial raise may be developed along a vertical distance of approximately 20 to 25 feet before the actual stoping operation begins. This allows ample storage in the ore chutes to store the excavated ore. In different types of stoping operations, the raise may be developed the entire distance between levels prior to mining.

3. Using the initial raise development as a platform, the ore will be drilled, blasted, and slushed[4] to the ore chute. The ore is drilled and blasted in approximately 10-foot rounds. This sequence is continued in a horizontal direction to a distance of about 100 feet (approximate effective slushing distance) or as determined by the structural features of the ore body. The final stage of the mining phase is directed toward scaling the ore

[4]Slushed connotes the use of a slusher, which is a small scraper or bucket of 1/2 cubic yard capacity moved by cables to transport broken ore to the ore chute.

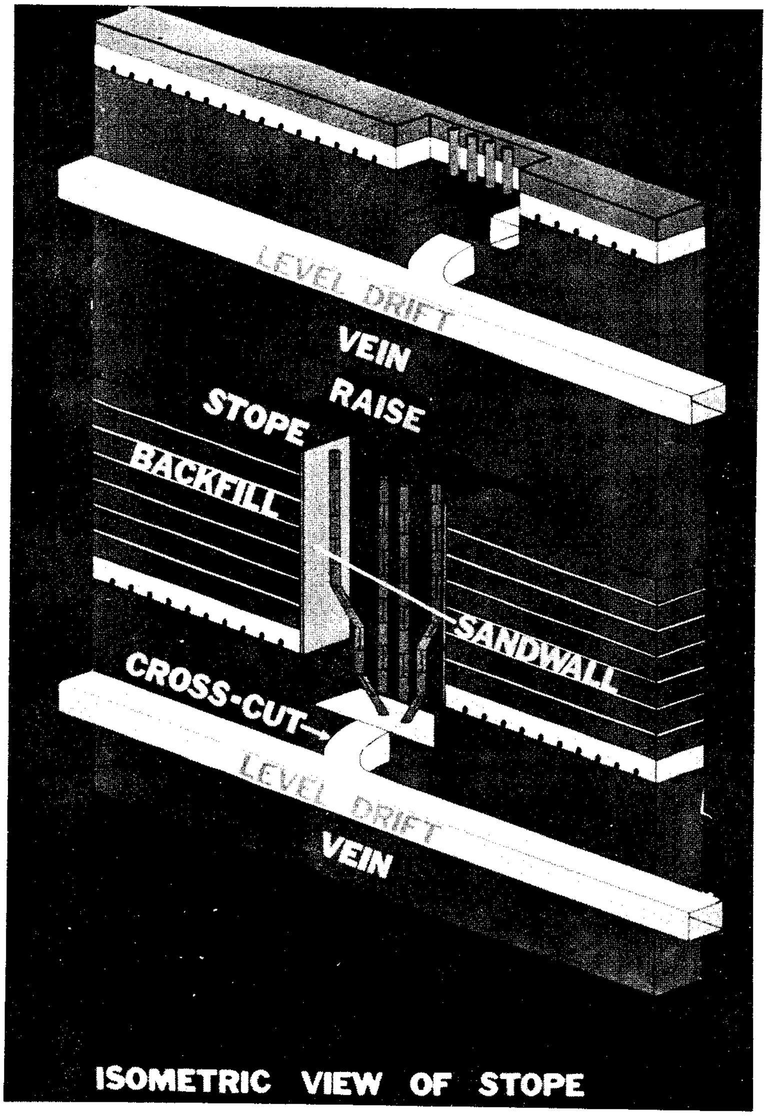

FIGURE 2. - Cross-sectional view of cut-and-fill stope mine.

remaining in the rib (sides of the stope) and small ore stringers running into the rib.

4. After the stope is cleaned of all ore, the stope is prepared for filling. A wooden, 8-foot-high bulkhead (sandwall) is constructed across the stope and covered with burlap. If the stope is mined from two different raises, a sandwall will also be placed so as to separate the two areas. Sand is transported hydraulically from the ground surface to the stope until approximately 6- to 7-foot thickness of sandfill has been placed in the stope. The fill surface is covered by a cemented sandfill cap that serves as a platform from which to mine the next floor. The first cut along the bottom of the stope (referred to as the intermediate or I drift) is usually completely filled with a cemented sandfill after lagging or stulls are placed to facilitate subsequent mining below the stope.

5. The final phase of the mining sequence involves preparing the raise area to permit mining of the next floor. First, 10- to 15-foot drill holes are placed in the back. Next, wooden sets are placed in the raise are to permit reentry after the raise is blasted. Following blasting and upon removal of the broken ore and the wooden sets, the four raise compartments are raised approximately 8 feet. Sandfill is poured around the compartments, and the mining equipment is placed in the raise area, and mining of the next cut is started. A cemented sandfill is placed around the compartments in a few mines.

Hydraulic filling has been utilized in situations other than stoping operations. European and Asian coal mines have placed fills for support for many years. In the United States, the use of hydraulic backfill as an integral phase of mining has been limited in active coal mines. Many coal companies, however, have been considering placing coal waste underground because of more stringent safety and environmental legislation.

Hydraulic filling has, however, been used extensively during projects for mine land subsidence control. Such projects have been completed in the coal fields of Pennsylvania and Wyoming. Two basic approaches are employed, dependent upon the accessibility of the mined area. "Controlled filling" is used when the project personnel are able to enter the abandoned mine workings and direct the flow of backfill material to the proper areas. "Blind flushing" or filling is used when the mined area is flooded, heavily caved, or hazardous to enter. In this situation, backfill material is flushed through boreholes until the holes will not receive any more material.

Mine filling costs vary from operation to operation, depending upon type and source of fill, transport distance, cost of sand walls and drainage system, and labor costs. Construction costs for placing burlap sand walls or fences have averaged $0.46 per square foot; costs for concrete bulkheads would be substantially higher. Placement of classified tailings has generally cost less than quarried sand and/or gravel. Other mines have placed smelter slag with the sand fill. Hydraulic sandfill costs have ranged from $0.56 to $4.49 per ton of sand (28).

HYDRAULIC SANDFILL

Although hydraulic sandfill has been placed in metal mines for many years, it has only been in the past 15 years that mining companies, educational institutes, and research agencies have been engaged in basic or applied research in this field. To a great extent, however, the majority of articles and presentations have described the physical plant facilities for preparing and transporting the fill material, the preparation of mined openings for sandfill, and general effectiveness of the backfill material at individual mines. Actual research has been primarily directed toward investigating the physical properties of hydraulic backfill and methods of densifying or modifying the sandfill to increase its support capabilities. To complement this laboratory research, field investigations have been undertaken to develop background data pertaining to physical changes experienced in the mine before, during, or following ore extraction. In addition, but to a more limited degree, research has evaluated the factors influencing the hydraulic system used to place the sandfill. Most recently, theoretical modeling of cut-and-fill stopes has enabled researchers to evaluate the effectiveness of various types of sandfill. With additional refinement, this analytical technique should provide a means for predicting the support requirement necessary to alleviate heavy ground conditions associated with stope mining.

A typical flow diagram of hydraulic backfill system is shown in figure 3. The diagram is very generalized and does not attempt to illustrate the various layout concepts, which are designed by engineers to satisfy specific mine requirements. As shown on the diagram, the hydraulic sandfill system begins at the time the raw tailings are separated from the ore concentrate at the mill. The tailings are pumped to a primary classification unit (usually a cyclone) where the coarser size fraction of the tailings are separated from

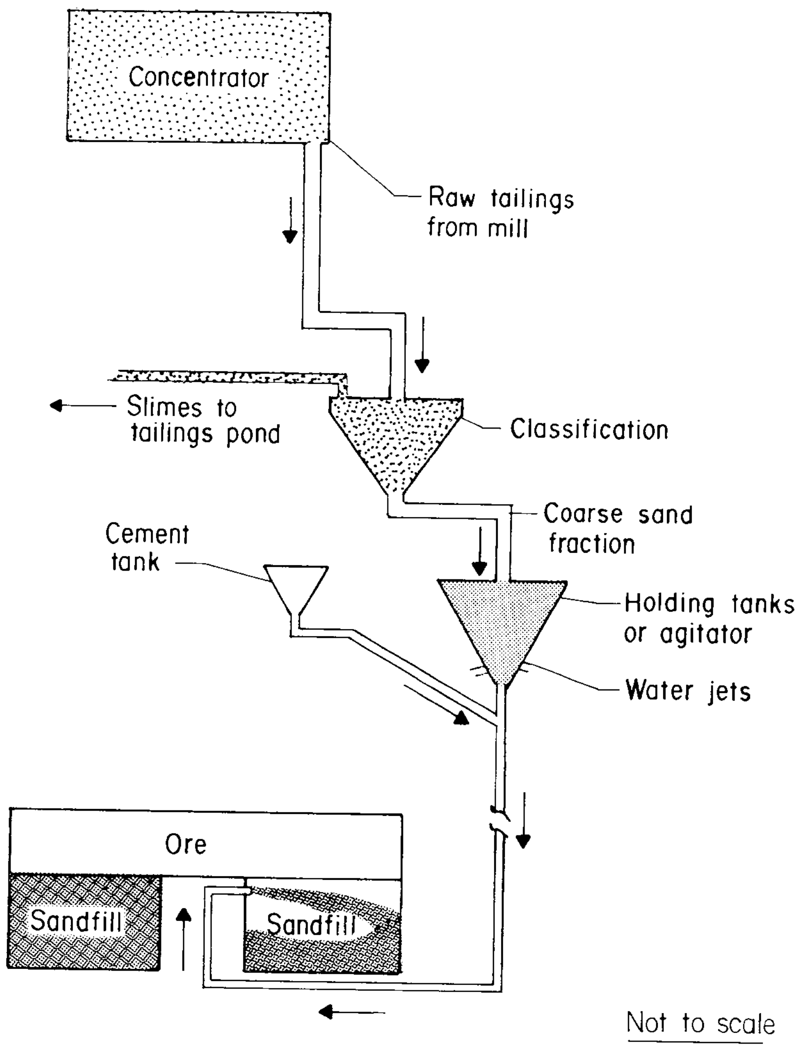

FIGURE 3. - Flow diagram of a hydraulic sandfill system.

the finer size fraction. The coarse fraction is transported to holding sand tanks; the finer fraction is sent to surface tailings ponds.

Depending upon the desired size fractions, the tailings will be passed through a series of classifiers, and often the tailings will be reworked at a second classification station. Coarser size fraction of the raw tailings are referred to as classified tailings. The classified tailings are stored in holding tanks until the sand is required underground. Upon demand, the sand is mixed with a predetermined quantity of water using series of jets at the base of the holding tank and transported underground to a specified stope. An alternative to the latter procedure would be to transport the sand from the holding tanks to an agitator, where the sand would be mixed with a prescribed amount of water and then transported underground. Cement, additive, or other modifier can be combined with the sand-water mixture in or just following the agitator tank. The final step of the hydraulic fill system is the placement of the sandfill in the mined area.

The following review will discuss research activities as related to hydraulic backfill: hydraulic transport, physical characteristics of sandfill, and methods of strengthening sandfill.

Hydraulic Transport of Sand

A hydraulic fill system at a mine generally consists of the three categories: classification, transportation, and placement of sand. The design of the system must include not only the numerical analyses for the hydraulic transport of the sand, but also other factors such as the nature and source of the sand and the most satisfactory material for placement. The classification of sand will be discussed under the section entitled, "Physical Characteristics of Sandfill."

The hydraulic sandfill system has gained acceptance basically because of the substantial savings realized over the square setting and gobbing practices. The most perplexing problem has been and continues to be the installation of the hydraulic system to transport the sand from the mill to the mined stope. If the hydraulic lines are designed and installed with accuracy, little problem will arise. However, if the system is makeshift, vertical sections too long and without orifices, or the pipe sections are not plumb, serious operational problems will eventually occur.

The field of hydromechanics, and in particular hydraulics, has been the subject of considerable investigation. The mechanics of water flow is well defined both empirically and theoretically. The basic water flow theory was derived through Torricelli's work, which stated that the flow of water from an opening could theoretically be determined by the following:

$$\nu = \sqrt{2gh}, \tag{1}$$

where ν = velocity of flow, feet per second,

g = acceleration of gravity,

and h = head on opening.

Thus, the equation of continuity, which stated that for steady flow of liquids, equal masses or weights of the liquids must pass all cross sections of the pipe in equal time. The laws of steady flow through pipes was further advanced by Bernoulli, and later the influence of friction was integrated into the steady-flow equations, thereby developing the fundamental laws for calculating the flow of fluids in pipes.

The utilization of these basic theories have proven successful for determining steady-state flow of fluids through pipes and has received wide application. These flow theories have proven cumbersome, however, with their application to the transport of sand or other solids. The problems of hydraulic sand transport can be summarized as follows:

1. Most theoretical liquid flow equations are primarily based on water flow, but hydraulics of a sand slurry involves a two-phase flow situation.

2. The determination of frictional head loss is a drawback because of the various flow patterns possible in a slurry. For example, design should account for turbulent or laminar flow and the manner in which these flow petterns influence the drag force of the slurries.

3. The slurry concentrate (solid-to-water ratio) determines the energy demands on the system.

4. The design of the system regulates the distribution or flow of solids through the pipelines, and improper dispersion induces excessive wear and tear on the pipeline.

5. Maintenance and upkeep of a hydraulic transport system is often neglected.

Hydraulic placement of sand in mined areas is further complicated by the fact that the gravitational forces[5] are capitalized upon to transport the media. Design of the gravity transport system is often determined by the hydraulic profile concept. The fundamental consideration of the concept is to maintain an adequate hydraulic gradient to deliver the sand to the stope. This concept can be developed graphically through a schematic diagram of the pipeline system from its source to site of sand placement (1, 13). The basic layout consists of straight transport pipe sections (oriented vertically, horizontally, or inclined) and the pipe elbows. Frictional losses or losses resulting from pipe bends are partially accounted for by increasing the length of the pipe sections. Full flow through pipes can be maintained if the hydraulic profile does not intersect the established flow head. Full flow may also be controlled by decreasing the pipe diameter with increasing transport distances or placement of orifices in the vertical lines.

Vertical or steeply inclined descent in gravity flow is essential to produce a sufficient head and thereby maintain flow through the pipeline. Vertical drops of several thousands of feet is not uncommon in many deep, vein-type mines. With this long, vertical drop, several problems occur within the system. The speed of the transported material may exceed the capacity of the system and force open the pipe at weak unions or joints. Another problem is the excessive scouring or abrasion of the pipe (particularly at elbows) which eventually leads to pipe failure. Thirdly, entrapped air in the system may create a vacuum and thereby cause extensive plugging in the pipe, surging, or excessive vibration.

These transport problems can be partially controlled by modification of the pipe system. The transport speeds and pipe pressures can be stabilized by placing ceramic orifices along the vertical sections of pipe, or where the mine layout would permit, the system may be designed with the necessary number of pipe bends and horizontal pipe sections through mine levels to control transport speeds. Pipe abrasion has been reduced by use of rubber-lined pipes and proper design and placement of bends and joints.

The following short discussion is prepared as a summary of the many factors affecting the flow of a two-phase media through pipeline. These factors influence the efficiency of the transport system. The nature of the sand material and their hydraulic characteristics should also be determined prior

[5]Gravity flow is generally employed once the classified sand leaves the holding or agitating tanks and begins its descent into the mine. Reciprocating or centrifugal pumps, individually or in a series, often are used to transport the sand from the mill to the sand tanks.

to designing the system because the support capacities of the sandfill will be determined by the type of material placed in the mined area.

The factors which affect the efficiency of two-phase flow through a pipeline are type, size, and shape of grain particles; viscosity; friction losses; critical velocity; and weight-percent and volume-percent of solids (32-33). The first of these factors will be discussed later under the subsection, "Physical Characteristics of Sandfill." It will suffice to mention that the nature of the sand material and characteristics of the sand when placed in a hydraulic transport environment should be determined prior to designing the system.

Viscosity defines the flow behavior of a fluid. Because of the attraction between particles, a drag is created in the fluid and develops a shear resistance. Only apparent viscosity of slurries are determined because slurries are not homogeneous. Viscosity is influenced by temperature, type and geometry of material transported, and density of slurry.

Friction losses are influenced by the viscosity of the slurry and are expressed as the energy required to overcome the resistance to flow. Although frictional loss is not a major concern in hydraulic systems using gravity flow, such as used in many deep-vein mines, frictional loss may be a significant problem where hydraulic sandfill is pumped to surface or underground holding tanks or if the hydraulic profile approaches its maximum limit.

Critical velocity of a slurry is defined as the amount of flow velocity required to maintain a constant slurry flow, as opposed to the critical velocity in hydraulics, which indicates that transition period between turbulent and laminar flow in fluids.

Percent of solids by weight and/or volume are often erroneously exchanged during discussions on hydraulic sandfilling. Both terms are dependent upon the specific gravity of the transported solid. Most installations maintain a percent of solids by volume below a maximum of 49 percent; above this range the possibility of plugging the pipe will rapidly increase.

Physical Characteristics of Sandfill

For the past 40 years or more, civil engineers involved with soil masses for engineering purposes have sought to improve their techniques and practices in the field known as soil mechanics. Soil mechanics has now become an invaluable tool to designers of all types of construction problems. These problematical situations center on construction through natural or in situ soil masses or over manmade deposits such as roadway fills, dams, airports, etc. One question facing design and construction engineers deals with what will be the most competent material to satisfy the load-bearing support requirement. Since each construction site offers different local materials, a standard series of tests were needed to evaluate the suitability of that soil or sediment to meet the design parameters. Standardized testing procedures have been established, and although future refinements are inevitable, a universal testing technique now exists (2).

Hydraulic sandfill used as a support in underground mines is also amenable to testing procedures used in soil mechanics. The only difference being that hydraulic sandfill is artificially created through the separation of metal values from the ore, but soil materials used in civil engineering are predominately composed of natural materials.

The physical characteristics of sandfill has been determined by almost every researcher involved in the placement of sandfill in mines. A knowledge of grain shape, grain size, and mineralogical composition enables the investigator to analyze and determine the moisture content, permeability, relative density, shear strength, and consolidation characteristics of the sandfill. Once these parameters are obtained, the support capability of the sandfill is closely defined, and the sandfill can then be further modified as necessary to meet ground control requirements.

Grain Shape

Individual grain configuration in natural deposits is dependent upon the mineralogy of the particle, method of weathering or disintegration, duration, type of transportation, and depositional environment. Grain shape or "angularity or roundness" is defined by a "measure of the sharpness of the corners" (30, p. 22), and is referenced predominately to hard mineral types. Grain size is categorized through a series of terms from angular to well-rounded. Another important shape is flat or flaky, which generally describes the mica and clay mineral groups.

Although sandfill is an artificial soil, similar factors affect their grain shape; consequently, similar support characteristics can be drawn. The grain shapes of sandfill are a product of crushing and concentration of an ore and the transportation of the sandfill to its final disposal site. Because crushing creates highly angular particles, and the transportation distance is relatively short, the sandfill particles are unusually angular. Microphotographs of several sandfill materials have been shown (5, pp. 7-8). The total angularity, however, can be somewhat reduced if a high percentage of flaky minerals is present.

Grain Size

Particle size is also dependent upon the method of crushing necessary to comply with mill circuit demands and the degree to which the raw tailings are classified. Grain-size distribution is determined by conventional wet and dry screen, with the finer particles going through careful hydrometer analysis. A better equipped laboratory may use a neutron particle analyzer to determine the fine-size particles. Because of the usual fine crushing and grinding of ores, particles greater than 30 mesh are usually less than 5 to 10 percent. Four basic types of grain size distribution are shown on figure 4. A most desirable sandfill would have a slope similar to that of a well-graded material. A well-graded material yields a denser structure because all particles tend to fit together and have a smaller void ratio. As shown in figure 5, typical size analysis of mine sandfill does vary. It is noted that most sandfill grain size slopes fall within the range of a well-graded to uniform soil slope.

565-423 O - 75 - 3

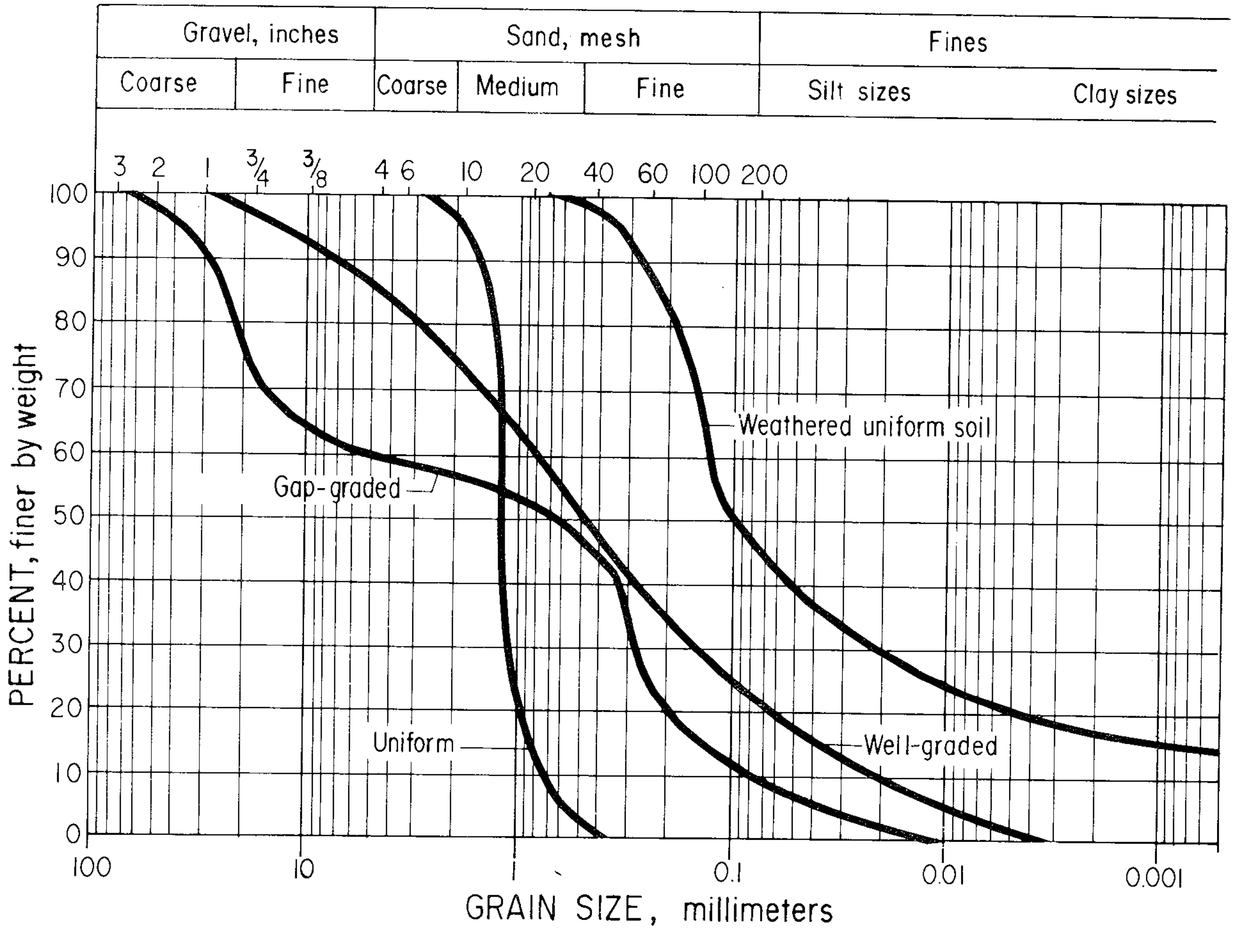

FIGURE 4. - Grain size chart of four typical soils (30, p. 20).

Another technique to characterize sandfill can be accomplished by determining the coefficient of uniformity (C^u) of material using the following formula:

$$C^u = D^{60}/D^{10}, \tag{2}$$

where D^{60} = diameter of 60 percent on the grain size curve,

and D^{10} = diameter of 10 percent on the grain size curve.

A uniformity coefficient of approximately 5 has optimum physical properties for a mine fill. Higher values tend to yield a more cohesive material and reduce permeability; lower values signify materials that are uniform.

Mineralogy

The composition of a sandfill is representative of the country rock and ore type extracted at each particular mine (5, pp. 5-6). Most mine fill materials contain quartz and feldspars with minor amounts of pyrite, dolomite, chlorite, and other minerals. The type of mineral present in the sandfill and how these minerals are affected by the chemical reagents, transportation of particles, and mineral alteration once placed in the stope influences the

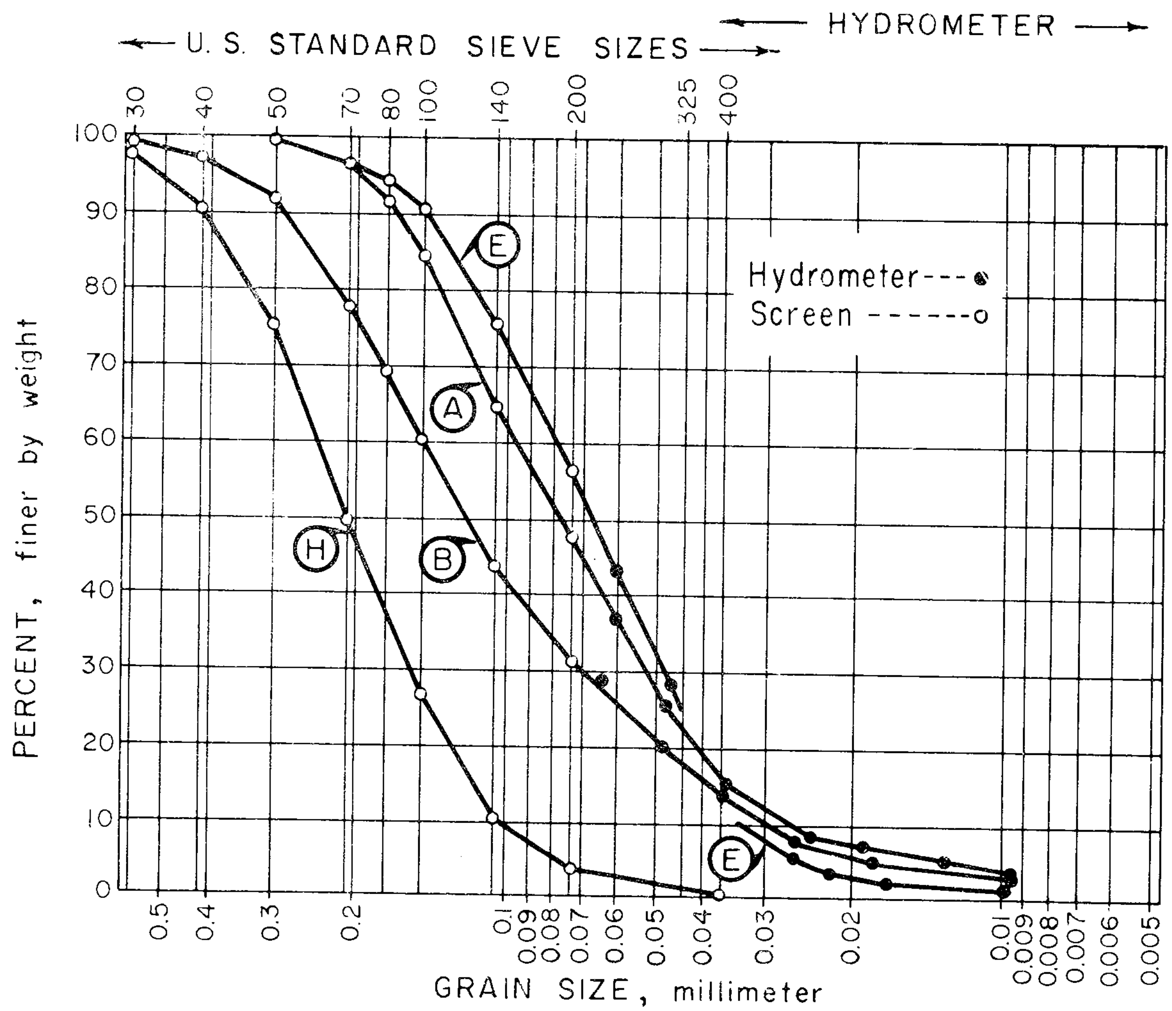

FIGURE 5. - Grain size chart of four sandfills from different mines (22, p. 6).

quality of the sandfill. For example, hard grains of silica-based minerals (except micas and clays) will undergo little or no chemically induced alteration. Most other mineral types can be dissolved, experience swelling or shrinkage, or undergo chemical transformation. The composition of the grains influences the specific gravity of the material and in situ densities.

The mineral composition of sandfill further influences whether the material will be cohesive or cohesionless. A cohesive soil adsorbs mo[illegible] by the presence of clay and clay-type minerals. This cohesiveness [illegible]es the particle attraction. Such soil masses deform plastically, d[illegible]ing upon the water content, and they tend to deform readily and are compressible under static load. A cohesionless material is composed of bulky grains such as quartz or feldspar. These soils do not adsorb water readily, and as such, will not yield under load as readily as a cohesive material. It should not be misinterpreted, however, because cohesionless materials will exhibit some grain attraction when moisture is present. This is, in a large part, caused by the surface tension formed between the grain particle and the liquid.

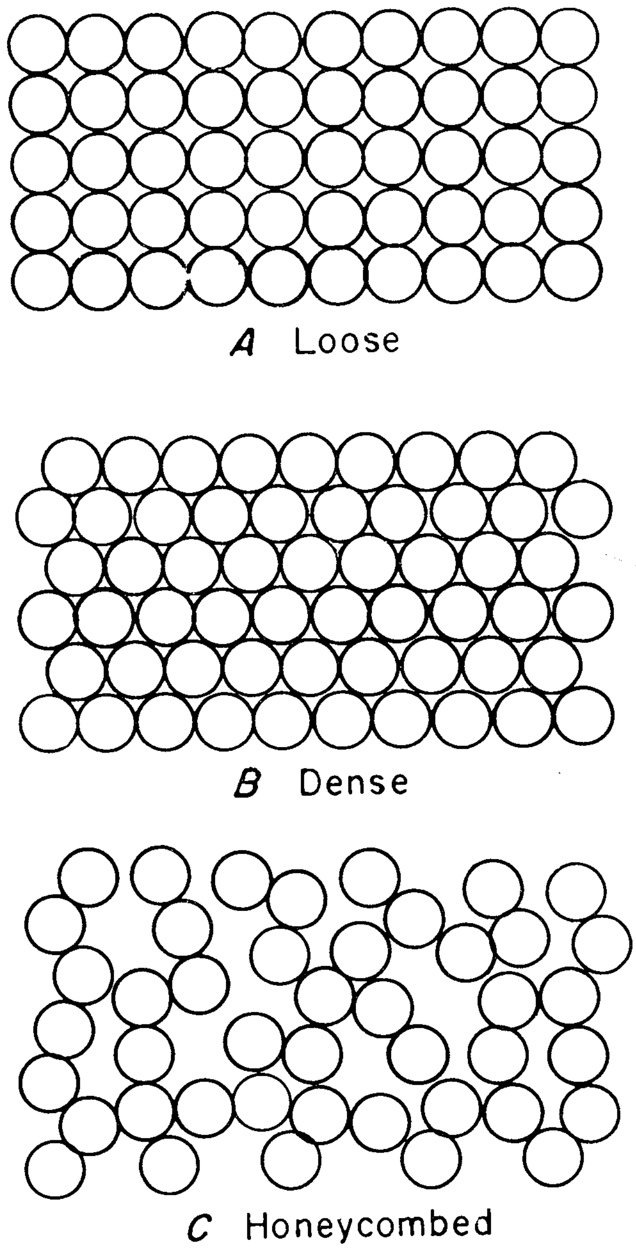

FIGURE 6. - Three structures (dense, loose, and honeycombed) common to sandfill.

The structural arrangement of sandfill particles are similar to structures found in cohesionless soils. The three possibilities are dense, loose, and honeycombed (fig. 6) (22, p. 29; 30). An ideal sandfill structure would be a denser structure. The grain-particle arrangement undergoes minimal consolidation upon application of a load. If the sandfill structure existed at maximum densities, it is possible that vein-wall closure in stopes could be reduced. The latter two structures are more likely to occur in hydraulic fill because of the amount of water required to transport the solids and the manner in which the sandfill is placed. To alter the loose structures to a more dense one, additional modification through vibration, tamping, or chemical-physical alteration would be required. Honeycombed structures are, oddly enough, capable of supporting significant loads. This phenomena exists because the grains are arranged in the form of arches. The structure can probably carry an increasing load without causing failure if other outside influences do not occur. Should a horizontal force become active or vibratory motion occur, the honeycombed structure would fail. Failure would continue as long as the outside force existed or until a dense structure was created.

Effect of Water on Sandfill

The soils engineer is aware of the influence that water will have on the effective strength of a soil. The presence of water either at the time of or following construction can significantly reduce the load-bearing capacity of the material supporting a foundation. The soils engineer must further consider what type of material will best support a structure under this potential condition. These same theories can also be used to analyze the impact that water may have on the total strength of a hydraulic sandfill. The total stress (σ) applied to a substance is equal to the total load (Q) divided by the total area (A):

$$\sigma = \frac{Q}{A}. \tag{3}$$

The load on any one grain in the sandfill is difficult to determine because of the irregularity of the grain shape ans size and the existing void ratio. As might be expected, the load is greatest at the grain contact points (30, p. 94) and the stress is not equally distributed throughout the mass. Water in the sandfill must also be considered. The pressure within the water particles or pore pressure is referred to as the neutral stress (μ). Since water cannot accept a static shear stress (18, p. 241), the vertical and horizontal stress components (μ_v and μ_h) are the same and they are equal to the neutral stress:

$$\mu_v = \mu_h = \mu. \tag{4}$$

The neutral stress is equal to the product of the height of the water column times the specific gravity of water or load placed upon the water because the neutral stress may support an applied load.

The grain-to-grain stress (effective normal stress, σ'), which would exist in an undrained material, would be equal to the total normal stress (σ) minus the neutral stress (μ):

$$\sigma' = \sigma - \mu. \tag{5}$$

Once the neutral stress becomes equal to the normal stress, the effective stress equals zero, and failure of the grain-to-grain stress will occur. This failure is determined by the following equation:

$$\bar{p} = p - \mu, \tag{6}$$

where $\bar{p}$ = effective stress at failure (or $\bar{\sigma}$ at failure),

and p = normal stress on failure plane.

How do these equations pertain to hydraulic sandfill? If it could be assumed that a sandfilled stope would retain all transport water, the sandfill would exist in a saturated state. Should an event (blasting, etc.) occur in the stope area to create an abnormally high neutral stress, a rapid failure of the grain-to-grain stress could occur. A "quick" condition would be created in which the sand is unstable and liquefies. Excessive pressure could be placed on the bulkhead causing it to collapse. The outcome would be tons of fluid sand flowing into the raise area, and down the ore chutes, and out through the main haulage level, not to mention the possibility of incurring hazardous working situations.

Water Percolation and Drainage

Water in filled stopes originates from three main sources:

1. The majority of the water is the hydraulic transport water for the fill. If a typical stope fill were 100 feet long, 8 feet wide, and 8 feet high, and the sand averaged 100 pounds per cubic foot or 20 cubic feet per ton), then approximately 320 tons of sand would be required to fill the stope. An average sand-to-water ratio is assumed to be 60 to 40 by weight; then approximately 51,000 gallons of water is required to transport the sand to the stope. In addition, flush water is run through the pipes prior to adding sand and following the placement to clear the pipes.

2. An unknown quantity of ground water seeps into the sandfill.

3. Machine water for drilling and spraying down the face introduces a certain amount of water.

The removal of excessive water from the filled stopes is important, not only to prevent hazardous "quick" conditions, but also the water content will influence the effective stress. The compressibility and strength of the sand is controlled by the effective stress. These aspects will be briefly discussed in the following section.

The free water generally decants to the surface and flows toward the outlet, whereas additional water percolates through the sand fill and through the burlapped bulkhead. In a few cases, it is possible that some portion of the water seeps into the fractured country rock. An investigation (33) conducted in Australia has reported the following dewatering mechanisms:

1. Decantation of water during and following sandfill placement (primary mechanism);

2. Surface water produced by fill consolidation; and

3. Percolation through the sandfill.

Thomas (33) suggests that about 80 percent of the water may be decanted, depending upon the physical characteristics of the fill material. Water percolation is controlled by the permeability of the sand. The rate of percolation can be determined and modified if the following physical properties of the sandfill are known: bulk density, specific gravity, and grain-size distribution.

The flow of water through backfill can be calculated by using Darcy's law:

$$k = \frac{Ql}{Ath}, \tag{7}$$

where k = coefficient of permeability, inches per hour,

Q = quantity, cubic inches,

l = length of sample, inches,

A = area of sample, square inches,

t = time, hours,

and h = head of water, inches.

The percolation rate through a sand column is determined assuming that the sample length (l) is equal to the head of water (h). Therefore, the basic formula for percolation rate (P) can be written as follows:

$$P = \frac{Q}{At}. \tag{8}$$

Percolation rates of different sandfills (22) have been determined in the laboratory (table 2). The report also included analyses of the possible factors influencing the rate determinations. A proposed modification to the traditional method of determining the percolation rate has also been suggested (35). The fundamental difference is that the modified method establishes the relationship between percolation rate and void ratio of the sandfill. Percolation rates of sandfill have also been studied using multivariable least-squares regression analysis (5). The equations successfully predicted percolation rate, coefficient of permeability, and seepage velocity; accuracy is dependent upon sample and data collection techniques.

Water drainage of a cohesionless material such as sandfill is a function of the coefficient of permeability and thus the percolation rate of the placed sand. There are circumstantial factors that may reduce or prevent the flow of water from the stope. Many of these conditions are caused by placement techniques and mining practices at various operations.

The method of sandfill placement can produce conditions such as particle segregation, which could impare the percolation rate. Many mines utilize single-point discharge to place sandfill either from the bulkhead and then from the stope end or vice versa. Sandfill flow from a single point in a stope can be expected to behave similar to tailings placed behind an embankment. It has been established that the coarser sand fraction will settle before the slimes (15-17). Laboratory model tests also substantiated particle segregation. To illustrate this phenomena, a comparison of values for the coefficient of permeability can be used.[6] Coefficient of permeability values obtained from model tests ranged from 9.94×10^{-5} centimeters per second at the point of placement to as low as 1.21×10^{-5} centimeters per second at the decant point (16). It is noted also that the lower value was taken from a horizontal position, but the higher value was taken vertically. In the same report, grain size curves from the intake to outflow points show a marked shift of the curves toward the finer sized particles. The significance of grain segregation in a stope can be observed by the ponding of water at the stope end. This is assuming that the stope is dead ended and that the initial fill is placed from the bulkhead in the raise. The ponded water exists because fines have deposited on the fill surface and the low permeability restricts the downward flow of the water.

[6] The coefficient of permeability describes the ease with which water will flow through a particular material such as a soil or sandfill.

TABLE 2. - Physical properties of investigated backfills

Mine	Specific gravity	Percolation rate,[1] inches per hour	Average density (in-place), pounds per cubic foot	Average relative density, percent	Average void ratio (in-place)	Average porosity (in-place), percent	Average moisture (in-place), percent[2]	Degree of saturation, percent	Coefficient of uniformity[3]	Effective size[4]
A	2.89	2	100	51	0.81	45.0	15.0	53.0	3.7	0.026
B	2.83	4	104	66	.70	41.0	9.0	36.0	4.9	.030
E	2.82	4	[5]95	65	.86	47.0	8.5	27.0	2.1	.033
H	2.96	[6]25-50	[6]97	[6]23	.907	47.5	18.0	58.1	2.3	.105

[1]Percolation rates determined at 20° C, with 53.5-centimeter elevation head of water--calculated at 0.800 void ratio.

[2]Samples taken within 1 day of filling; the moisture content is affected by ventilation conditions in the stope.

[3]Coefficient of uniformity = diameter of 60-percent finer size divided by diameter of 10-percent finer size (grain size diameters in millimeters are obtained from grain size distribution curves in figure 5).

[4]Effective size = diameter of 10-percent finer size in millimeters. Obtained from figure 5.

[5]Surface desiccation appears higher than in other mines. This makes the in-place density values higher than would normally be expected for a uniform material.

[6]Highly variable.

There exists several methods to minimize the entrapment or perching of water in the sandfill:

1. Sandfill placement at the stope end would force the sand and water to flow toward the bulkhead. The water will decant and seep through the bulkhead if constructed of cribbing and covered with burlap or other pervious material. The finer sized particles will deposit against the bulkhead. A potential problem could arise if the sand should seal the bulkhead and entrap a large quantity of water adjacent to the bulkhead. Although surface water would continue to decant, an unknown amount of water would percolate toward the bulkhead through the coarser material. Should the pressures exerted by a rising phreatic surface become great enough on the bulkhead or should a "quick" condition evolve, the bulkhead could collapse. The possibility of this situation occuring is low with respect to narrow vein-type deposits; this type of failure should be realized as a potential when filling large room voids. Another problem, which is more of a nuisance than anything, is the flow of fine sand and slimes through the bulkhead. The cleanup process is then a cost to be borne.

2. Various types of drains have been installed in stopes to remove water. Drains are more commonly used in large room operations, although a few narrow vein-type mines do install drains. The most common drain is the "mouse trap," a simple wire frame or square box construction covered with a porous filter cover such as burlap. A second method would be to place sand filters along the stope floor and adjacent to the bulkhead. This type of drain is similar to a "French drain" or "conduit," and the filter material should permit the flow of water without head loss. Because of the particle size of sandfill, clogging of this type of drain filter could pose a serious problem (7, 30, p. 185). Perforated drainage pipes, both horizontal and vertical, through the stope could be used to carry off water. The horizontal drains would require sand filters or porous cover to prevent plugging, whereas the vertical stand should be burlapped. If the bulkhead were constructed of concrete, proper drain would be required. A contained sand filter could serve this purpose. Several mining operations construct permanent concrete conduits which are added on to as mining advances vertically. If the conduit is not crushed or plugged, the bulk of the water would escape and exit at a common point. Handling of water and slimes on the main levels could be eased.

3. An alternative solution to prevent particle segregation would be the utilization of a multispigoted discharge system. The placement from several points in the stope could encourage the placement of a well-graded sandfill rather than a segregated fill. The well-graded material would improve water percolation and possibly improve the overall strength of the sandfill.

Strength of Untreated Sandfill

A primary purpose of hydraulic sandfill is to place an artificial material capable of supporting the vein wall. Observation has shown that vein wall closure occurs as a slow, but continual movement following mining. This movement is caused by two mechanisms: incremental deformation of the rock strata adjacent to the stope induced by mining, and longer term failure caused by earth pressures. An outcome of this vein-wall closure can be the reorientation of stresses once distributed along the entire vein to the remaining and shrinking ore pillar. The ultimate outcome will be a rapid energy release (rock spawl or burst) when the failure strength of the ore or country rock is violated.

The reaction of sandfill in a mined stope can be interpreted by considering the theory of earth pressures at rest as used in soil mechanics. Because many deep stoping operations occur in narrow, pitching veins, the pressures exerted onto the sandfill can be assumed (fig. 7). Principal stress or load (σ_1) is applied to the sandfill by vein-wall closure or rock pressure. The other two stress components (secondary stress, σ_3) are assumed to be approximate because the sandfill is, for all purposes, confined laterally. In practice, however, the vertical stress can be relieved as attested by floor buckling or heave in the stope. Nicholson (21) applied the theories of earth pressure at rest and one-dimensional compression to evaluate the effects of compaction for several mine sandfills. In Nicholson's publication, the theories, laboratory procedure, and equipment fabrication are presented. Prior to summarizing the test results, a short discussion of typical sandfill strengths is necessary.

Federal Bureau of Mines engineers (5, 11, 22) have conducted laboratory and field investigations to define the strength characteristics of sandfill. A summary of these findings are presented in table 2. The range of in situ densities[7] was 95 to 104 pounds per cubic foot. These same sandfills were tested in the laboratory to determine their minimum and maximum density (fig. 8). The in situ densities were found to have an average relative density of 55 percent. (See footnote 8.) In other words, the sandfill placed in stopes exists at slightly more than one-half of its potential relative density.[8] Typical sandfill is then probably in a loose and/or honeycombed structure and is very responsive to any compressive loading. This loosely structured sandfill is unable to contain the pressures exerted by vein-wall closure. Consequently, consolidation within the sandfill occurs.

---

[7] Density is the unit weight of the material and is used to define the particle arrangement of a cohesionless material.

[8] The term "relative density" is defined as the ratio D_d.

$$D_d = \frac{e_{max} - e}{e_{max} - e_{min}} \times 100, \tag{9}$$

where e_{max} = void ratio in loosest condition,

e_{min} = void ratio in densest condition,

and e = in-place void ratio.

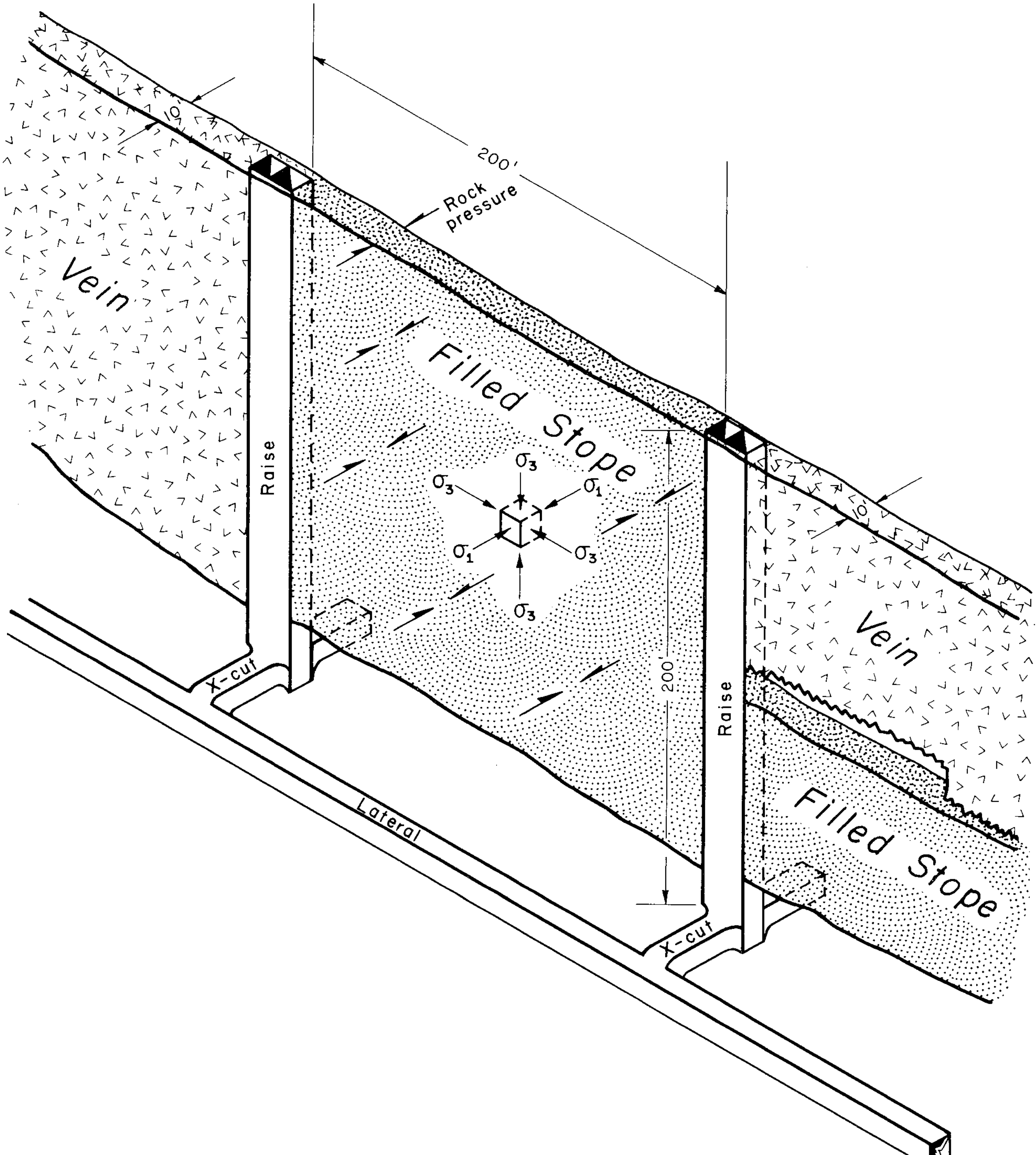

FIGURE 7. - Field pressures active on sandfill in a deep vein mine (21, p. 4).

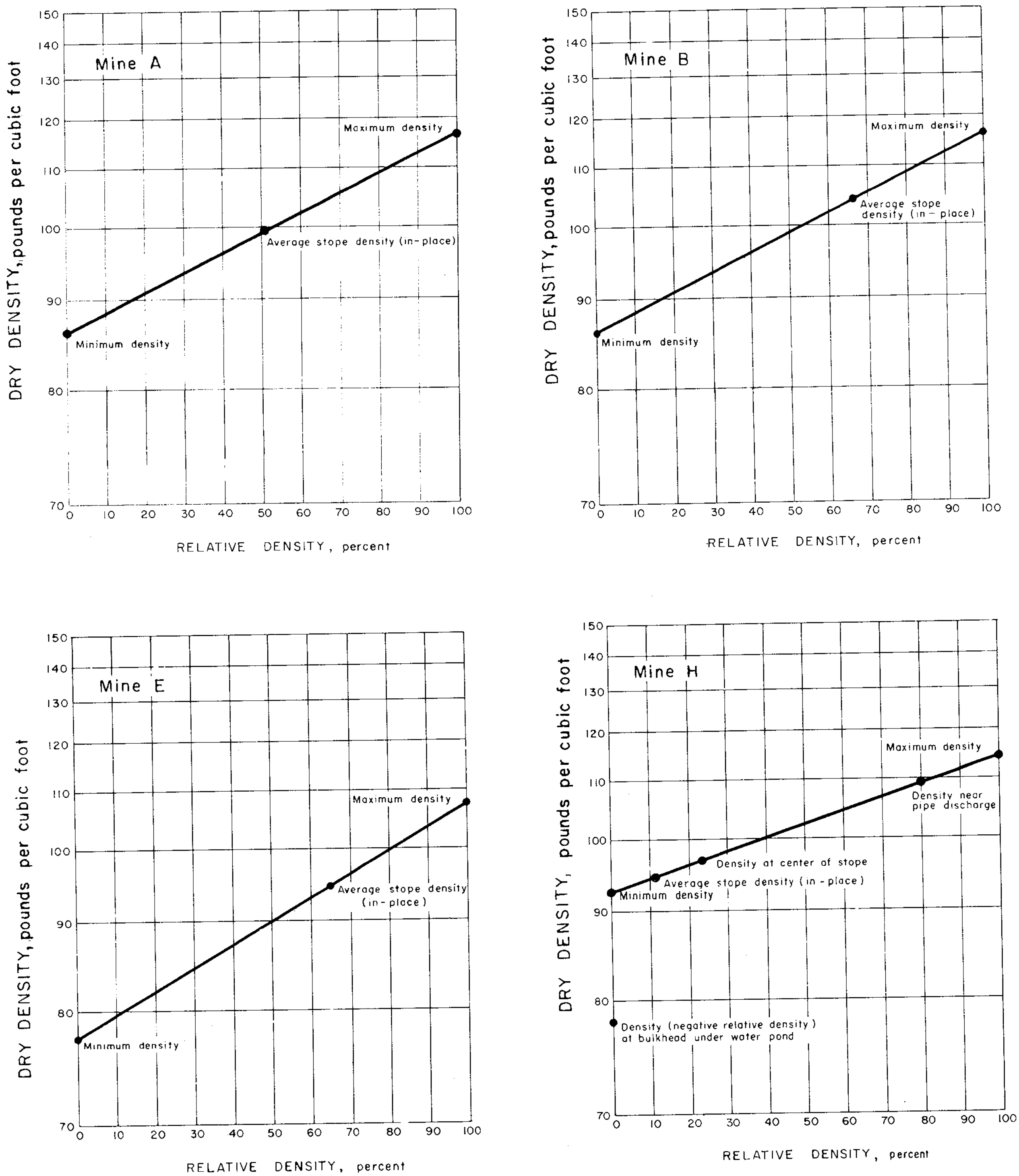

FIGURE 8. - Maximum, minimum, and relative in situ density of sandfills at four mines.

Laboratory triaxial tests were also conducted to further quantify the stress characteristics of sandfill of differing densities. The shear strength of test specimens are defined by the following equation:

$$S = p' \tan \phi, \tag{10}$$

where S = normal effective stress, and

p' = normal effective stress, and

$\tan \phi$ = coefficient of friction.

The results of two series of tests are shown in figure 9. It is evident that the shear strength of the denser sandfill is greater. The increased angle of internal friction (ϕ) of the denser sample should also be noted. It has been stated in soil mechanics texts (18, 30) that wet cohesionless soils should be placed in as dense a condition as possible. The aforementioned experimental results with sandfill support this position.

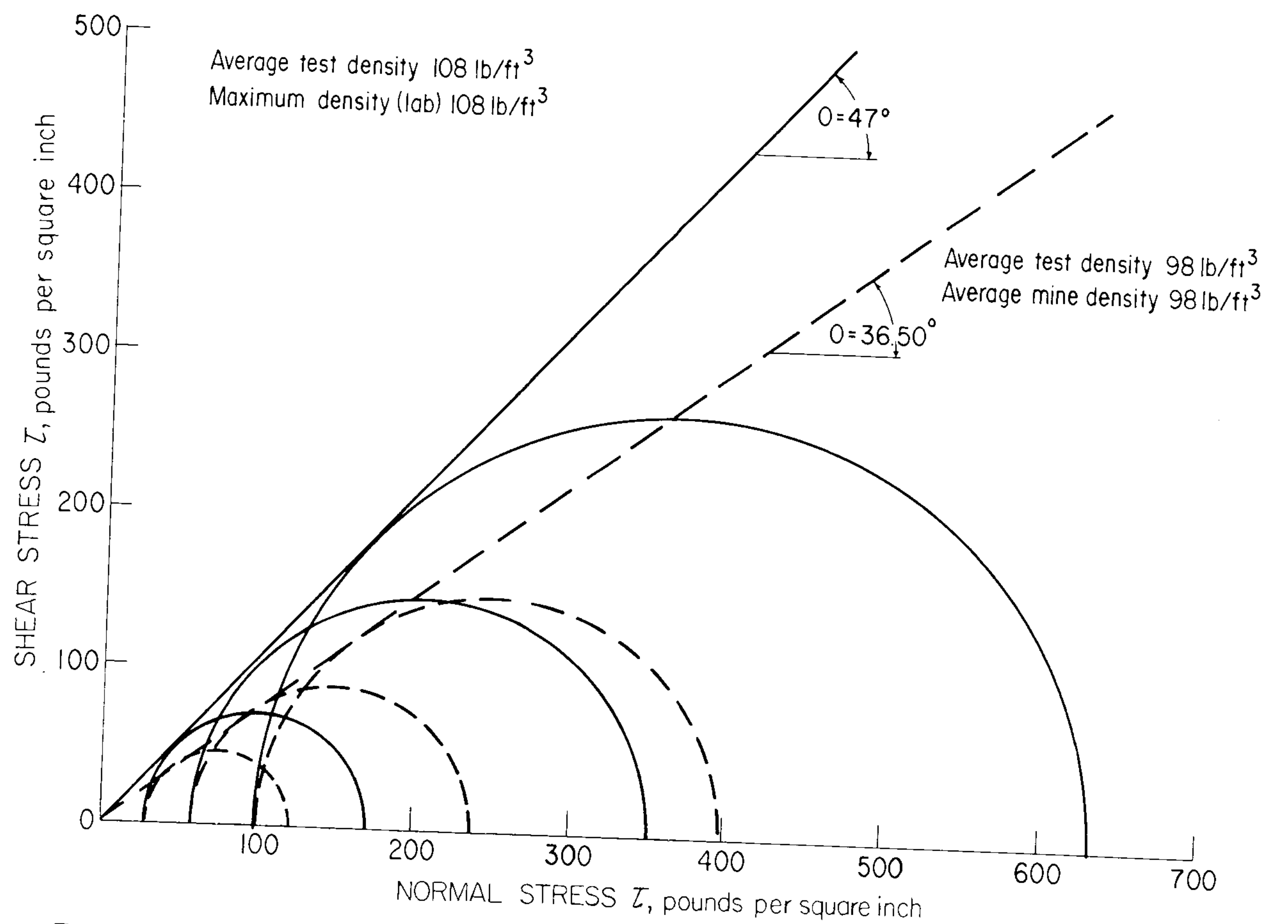

FIGURE 9. - Triaxial test plots for sandfills of two different densities (22, pp. 17, 21).

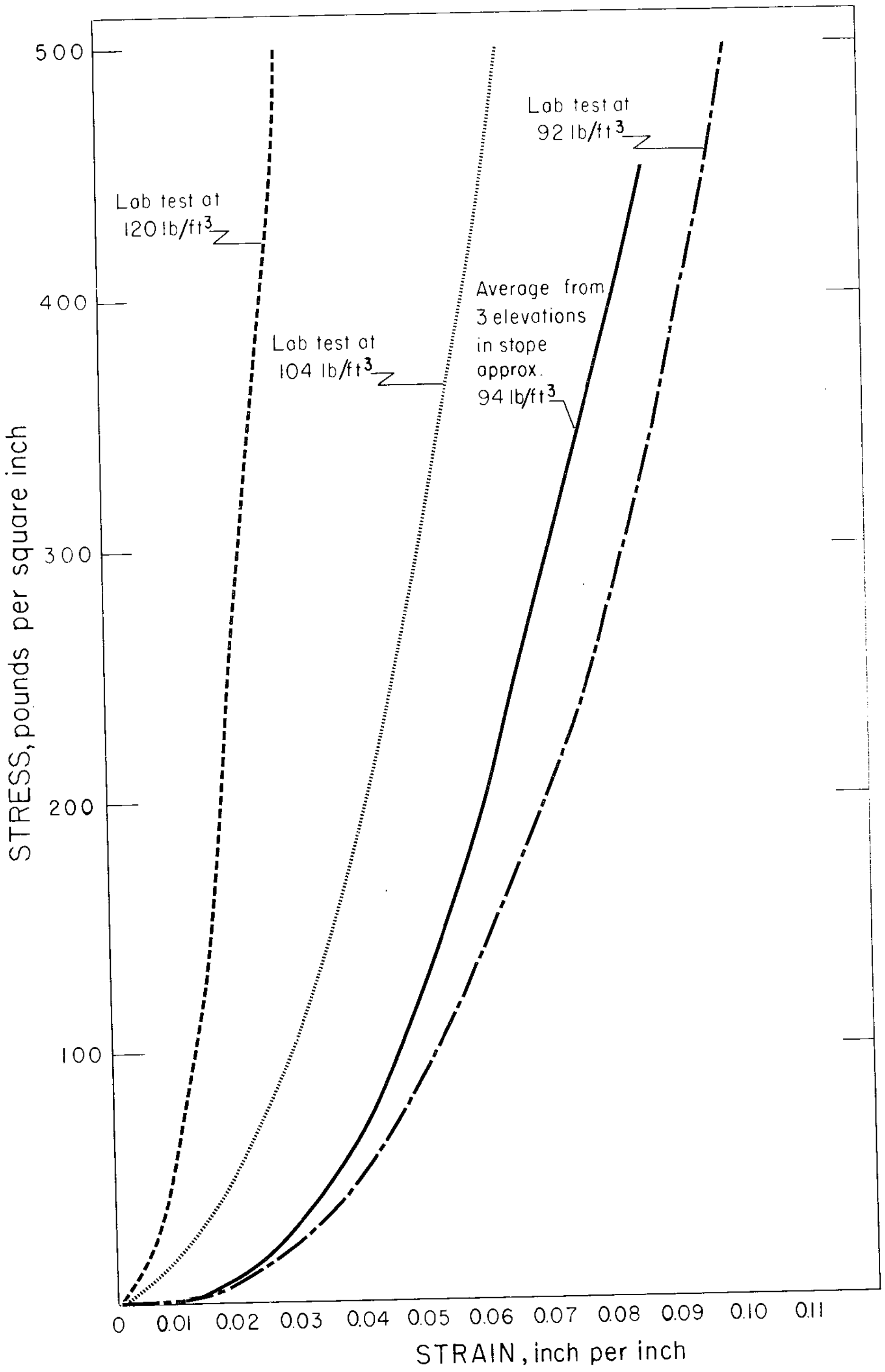

FIGURE 10. - Comparison of stress-strain relationships at various densities.

The support capability of sandfill of varying density is further illustrated by the stress-strain characteristics of the material (fig. 10). With a sandfill density of 92 and 120 pounds per cubic foot, vein-wall closure should be reduced from 11.4 to 3.4 inches, respectively (11). It could, therefore, be expected that a lower ground load would be placed on the ore pillar with the reduced vein-wall closure realized from greater fill density. The effect that wall closure control could have on pillar stress is presented in greater detail in the discussion on theoretical analysis.

Using the theories of earth pressure at rest, it was also found that a smiliar stress-strain relationship could be obtained using one-dimensional compression (fig. 11) as was obtained with triaxial testing; the basic conclusion being that

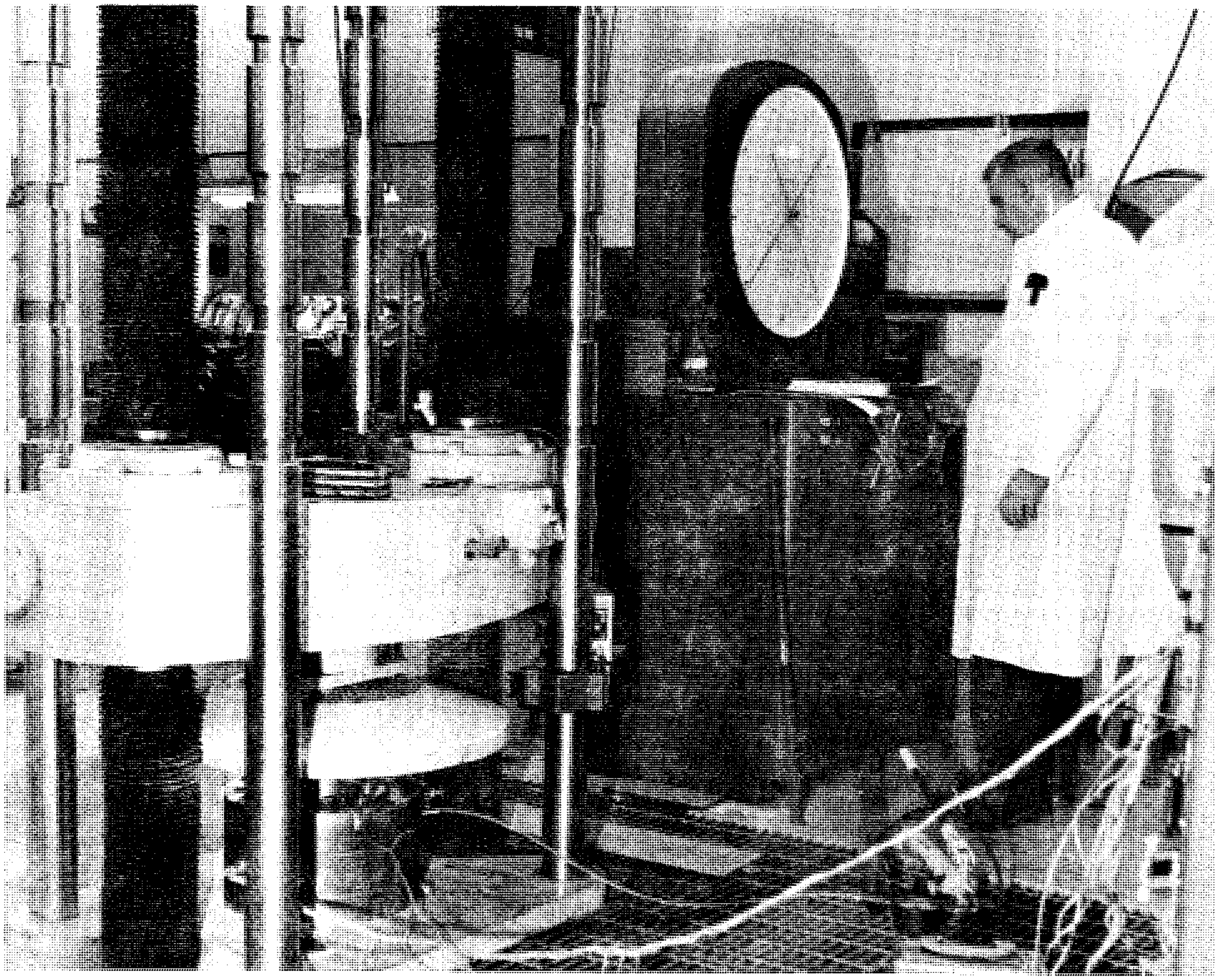

FIGURE 11. - One-dimensional testing chamber, manual hydraulic pump for applying lateral pressure, 400,000-pound Universal tester, and strain indicator.

dense sandfill with low void ratios yields a higher strength fill material and reduces strain with increasing stress. The one-dimensional compression tests indicated that an eightfold reduction in yield could be achieved between a loose- and dense-structured sandfill (21).

MODIFICATION OF HYDRAULIC SANDFILL

Thus far, the discussion has focused on an analysis of hydraulic sandfill as it is placed in cut-and fill mining operations. The physical properties, effect of water, and in situ densities have been presented, followed by an abbreviated discussion of stress-strain characteristics and their effect on vein-wall closure. Evidence dictates that hydraulic sandfill is not approaching maximum density when placed in the mine. It is, therefore, inconceivable that hydraulic sandfill can serve as a competent ground support media if substantial vein-wall closure must take place before the sandfill will begin to achieve higher in situ density. Additional investigations were then initiated to seek ways to improve the support capability of sandfill in an effort to provide greater competence with sandfill and reduce vein-wall closure and pillar stress problems. Techniques that have been studied include vibratory compaction; chemical modification with cement, dispersants, flocculants, and other additives; and electroosmosis.

Vibratory Compaction

In situ densities of concrete structures and soil foundations have been significantly improved through vibratory compaction. Pneumatic or hydraulic vibration yields greater consolidation of soil grains and stiff concrete by the extrusion of entrapped air bubbles and/or water and densification of loose and honeycombed structures. Capitalizing on the successful application of vibratory compaction to structural engineering situations, laboratory and preliminary field tests were undertaken to evaluate the vibratory densification of hydraulic sandfill in mines.

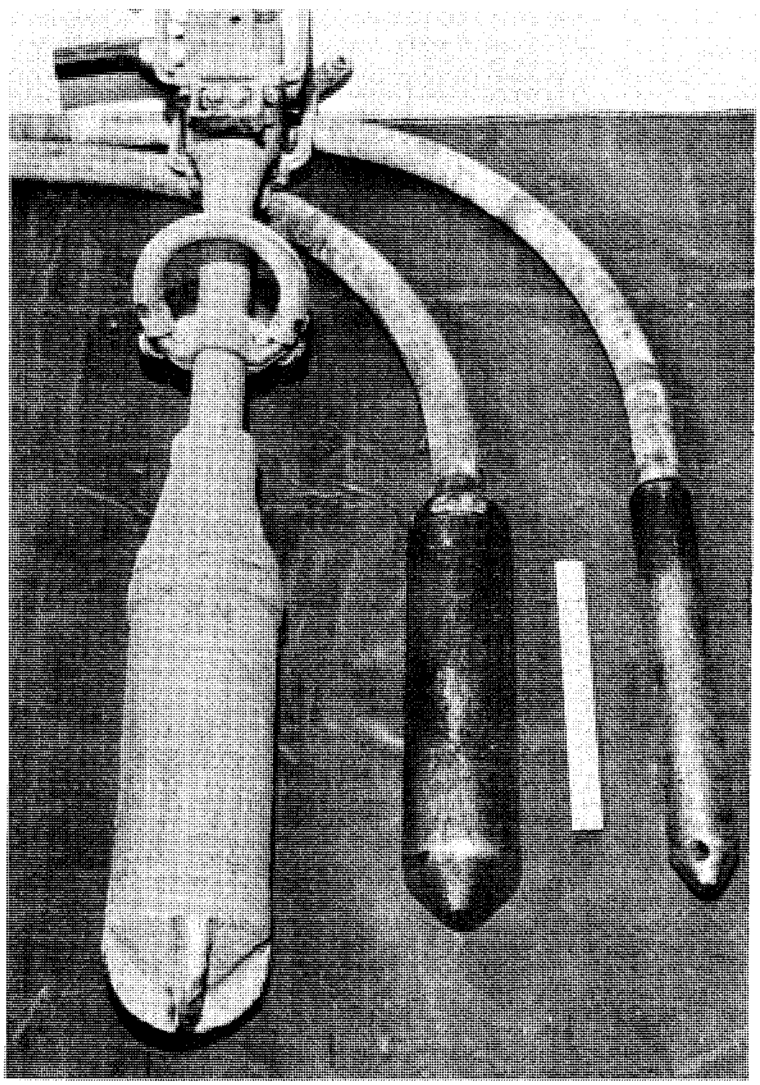

FIGURE 12. - Three types of probe vibrators used in compaction tests.

An air-activated, 2-inch concrete vibrator was tested under laboratory conditions (22). Sandfill densities as high as 117 pounds per cubic foot were achieved, dependent upon the physical characteristics of the test material. For this particular sandfill, a well-graded material, the relative density[9] was increased approximately 30 percent. Such densities would enhance the support capacity of this particular material to near its maximum without further strengthening with cement or other treatment. These laboratory investigations have further demonstrated that hydraulic backfill density is obviously dependent upon the physical characteristics of

[9]See footnote 8.

the material. The conclusions drawn from vibration of three different materials were as follows:

a. Coarse-grained material exhibited good drainage; however, because of the absence of finer size particles (200 to 325 mesh), a less dense hydraulic backfill material would be obtained. The excessive quantity of coarse-grained particles encourages the flotation and removal of the few fine-grained particles.

b. Fine-grained material (>30 percent, 200 mesh) have poorer drainage characteristics; therefore, they retain a high quantity of water. These materials have poor workability qualities.

c. Well-grained sands provide an ideal working material. They have good drainage properties, and the improved mixing characteristic which would provide a higher density fill.

Preliminary underground vibratory compaction tests were conducted to determine if backfill densities in an active stope could be improved in a similar manner and magnitude as the laboratory experiments (23). The following types of air-actuated vibrators were tested: 2-inch, 4-inch, and 6-inch probe vibrators (fig. 12), a 2- by 2-foot-square steel plate vibrator, and an aluminum plate vibrator approximately the same size as the steel plate vibrator. The dynamic characteristics and influence of vibration in hydraulic backfill are presented in table 3. The conclusions drawn from the five vibration tests were that the probe-type concrete vibrators were most effective. The effectiveness of the probe vibrator was believed to be attributed to the fact that the probe was immersed in the backfill and that a greater couple was achieved between the material and the vibrator.

TABLE 3. - Dynamic characteristics of tested vibrators

	2 inches	4 inches	6 inches	Steel plate	Aluminum plate
Vibrator weight...........pounds..	21.0	37.0	91.0	90.0	50.0
Air, pounds per square foot:					
Static.........................	90.0	90.0	90.0	90.0	90.0
Running........................	77.0	71.0	72.0	-	-
Frequency.......cycles per second..	170.0	105.0	105.0	118.0	-
Acceleration peak..........grams..	187.5	109.4	49.2	55.0	-
Velocity........inches per second..	67.8	64.0	28.8	-	-
Amplitude...................inch..	0.0634	0.097	0.0437	-	-
Approx. dynamic force......pounds..	3938.0	4047.0	4479.0	-	-
Maximum fill density pounds per cubic foot..	110.0	108.0	110.0	104.0	102.0
Maximum depth of influence...feet..	2.0	3.0	3.0	1.2	1.2
Maximum diameter of influence..do...	3.0	4.0	5.0	4.0	4.0

To substantiate the results of probe vibration pilot tests, a full-stope demonstration (10) was undertaken (figs. 13-14). Backfill pressures, stope closure, and physical property measurements were recorded in two adjacent stopes; one which was compacted while the comparison stope was placed under normal techniques and was untreated. Density measurements taken in the compacted stope did not approach maximum. Densities ranged between 98 and 114 pounds per cubic foot with an average of 104 pounds per cubic foot. These values compare less favorably than the 117 pounds per cubic foot obtained in laboratory tests. Water content of the backfill was believed to be the cause of the disappointing results. Ponded areas in the stope and areas containing water contents less than 15 percent have poor vibration characteristics.

Although density results were disappointing, the following benefits were derived from the vibration operation:

1. Excessive free water was exuded from the stopes.

2. Although backfill pressures were similar in both test stopes pressures increased faster in the compacted stope. This would indicate that the backfill was denser in the vibrated stopes, and the fill was providing greater support.

3. Backfill in the compacted stopes exhibited improved strain characteristics.

4. Compaction provided a relatively smooth surface throughout the stopes which could improve the cement cap placed, reduce amount of cement to achieve suitable platform, and reduce ore dilution by eliminating extremely thin cement layers.

Cemented Sandfill

Prior to the 1960's, cemented sandfill was placed in underground mines on a piece-meal basis and with a very limited knowledge of its potential capability. Originally, timber-planked floors were installed over backfill to serve as a working floor. The great quantities of timber required for this task created a costly, burdensome material-handling problem, not only in the stoping area but also in the shafts and haulageways. The concept of using cemented sandfill to cap a backfilled stope was intriguing because of the benefits to be derived.

Cemented sandfill was originally placed in 6- to 12-inch layers on untreated backfill to serve as a rigid working platform for slushers and large, rubber-tired vehicles. In addition to serving as a mining platform, cemented sandfill eased the costly handling and setting of plank flooring and reduced the incidence of ore-sandfill dilution. By 1965, adequate research had been accomplished to prove the use of cemented sandfill as a matted floor for undercut-and-fill operations, construction of competent bulkheads and free-standing walls in large pillar mining operations. Cemented sandfill is now a major support material and has improved mining costs and working safety in traditional cut-and-fill mines (narrow or wide vein), shrinkage stoping, undercutting and filling, pillar recovery in overcut-and-fill operations, and block cut-and-fill mining (19).

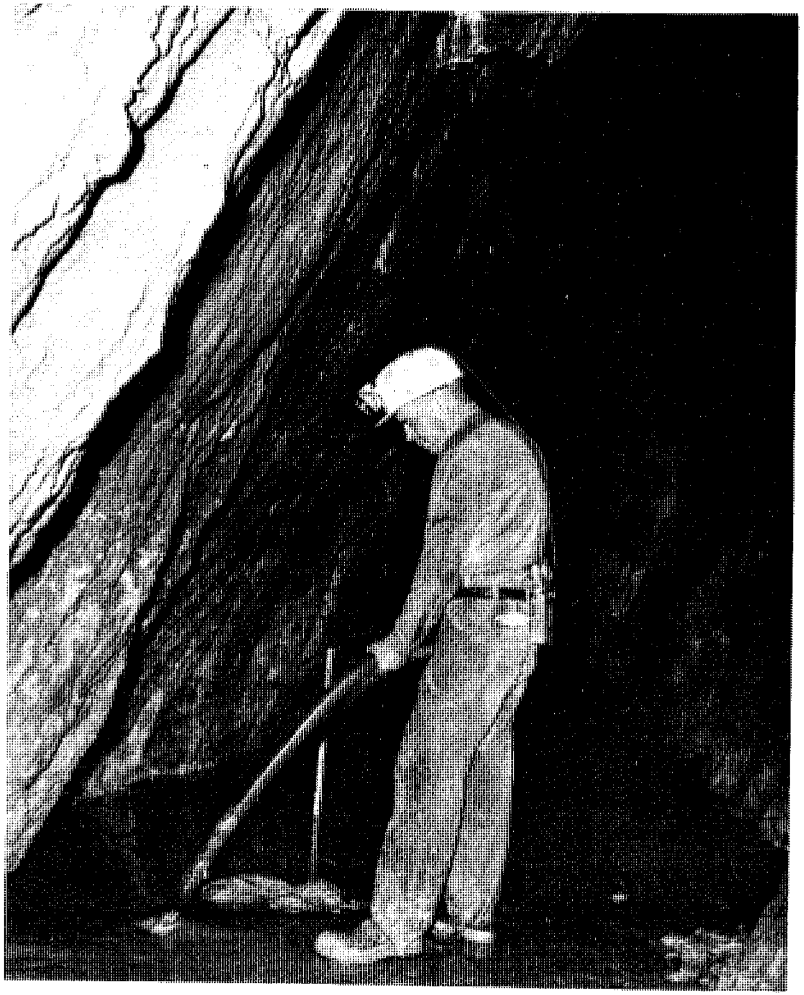

FIGURE 13. - Compaction of sandfill with hand-held vibrator.

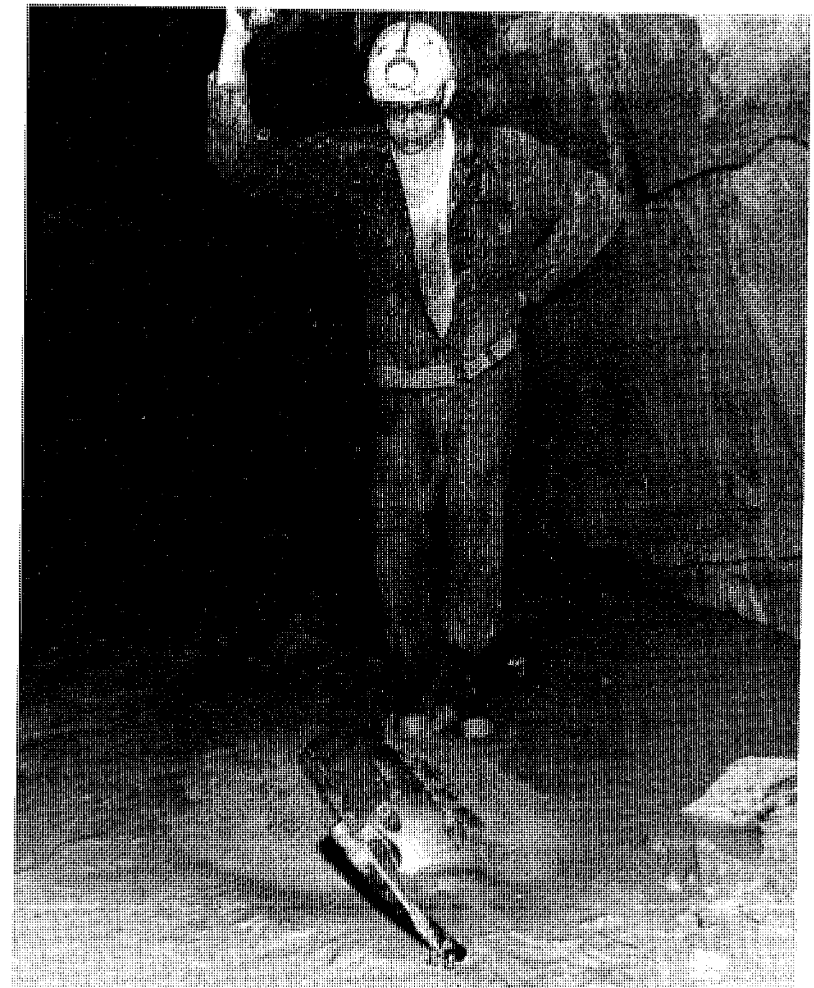

FIGURE 14. - Operation of immersion-type vibrator.

However, before management accepted cemented sandfill on a large scale in the aforementioned methods of mining, a great deal of research was necessary. The questions that required answers follow:

1. What cement-to-sand ratio should be used?

2. Does particle size have an influence on cemented sandfill?

3. How does a cemented sandfill affect permeability?

4. What pulp density is best?

5. Does curing time affect in-place strength?

6. What strengths can be achieved?

7. How should cement be mixed with sandfill?

8. How will additives influence the strength of cemented sandfill?

Similar to concrete, the important factors influencing the strength of cemented sandfill are sand-cement ratio, pulp density, mixing time, and aging. A fifth factor that has been investigated is the addition of additives, either dispersant or flocculant. Investigations into these factors have been predominately directed toward determining the strengths of sandfill as modified by varying quantities of cement, with a lesser emphasis on mixing time and aging. Researchers at the Federal Bureau of Mines, Spokane Mining Research Center (6, 8) evaluated the unconfined compressive strengths and permeability of classified tailings from various mines. The test variables were modified by varying quantities of portland cement, dispersants, and flocculants. The testing procedure was generally uncomplicated with the mixing of the desired cement-sand ratio and then adding 30 weight-percent of water. All samples were cured 7 days and were tested using an unconfined compressive testing machine shown in figure 15. The ranges and average unconfined compressive strengths are shown in table 4.

TABLE 4. - Unconfined compressive strengths of cemented sandfill, pounds per square inch

Sand-cement ratio	Ranges	Averages	Ranges (12)	Values (37, p. 991)
40:1	16.1-40.6	25.3	40	30
30:1	-	-	40-70	45
20:1	42.2-99.3	65.1	70-100	75
10:1	-	-	200	-
5:1	250.6-711.1	442.0	500-600	>500
2:1	-	-	2,500	-

Similar compression tests were conducted by engineers (12) within the Falconbridge Group of Mines in Canada with the exception that they cured their samples for 28 days (table 4). Data from the two separate investigations are

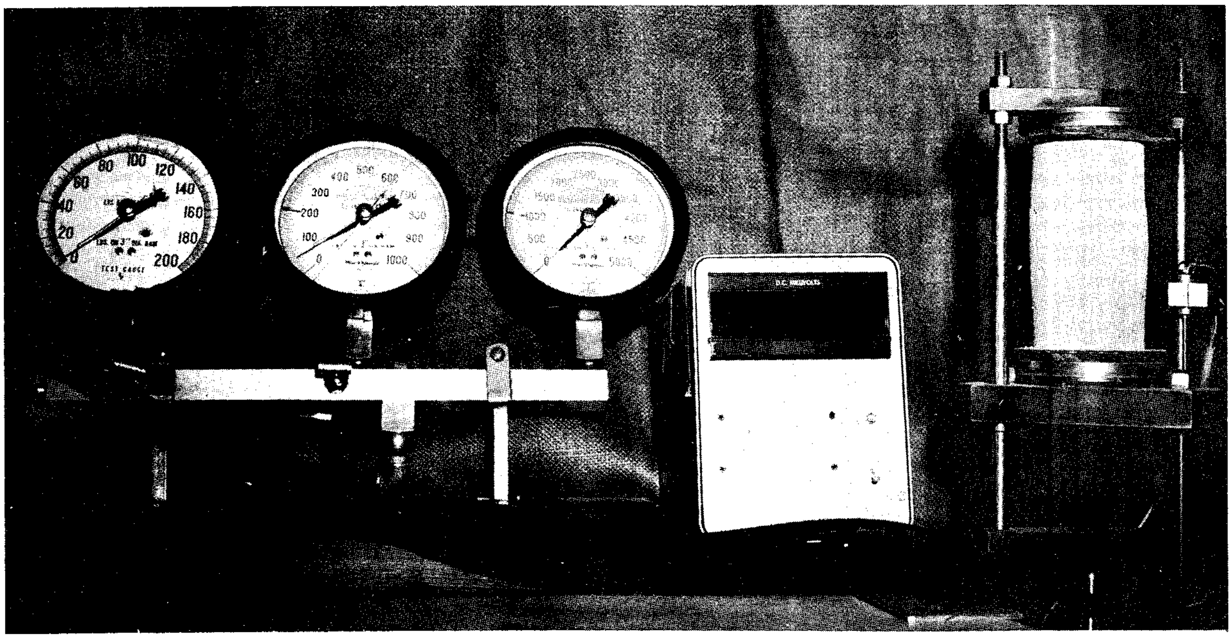

FIGURE 15. - Unconfined compression testing machine and DC DT with digital readout.

comparable. The high sand-cement ratio (2:1) is of particular interest. A compressive strength of 2,500 psi was obtained. The use of such mixtures is exceptionally unusual and is restricted to bulkheads because of the cost.

Research was also undertaken by the Canada Cement Co. Ltd. to investigate the various parameters influencing the strength of cemented sandfill (36). The results indicate that higher compressive strengths were achieved with higher pulp densities (weight-percent of solids in mixture). As a comparison and following an 80-day cure period, samples mixed with 60 and 72 percent pulp density had an approximate compressive strength of 22 and 39 pounds per square inch, respectively. The approximate strengths obtained for other sand-cement ratios are consistent with other investigations (table 4).

As discussed earlier under the section entitled, "Physical Properties of Backfill," particle size characteristics of sandfill and its distribution has a definite effect on the percolation of water through a placed sandfill. A well-graded sandfill will normally have a permeability of 4 to 5 inches per hour. With increased percentage of minus 325-mesh particles, the permeability drops in half. Cement will have the same effect as reducing the grain size characteristics of a sandfill. Laboratory tests, which were conducted to determine the influence of cement-sand ratio and curing time on permeability of backfill material (9), found that the coefficient of permeability for samples decreased as the curing time increased. As would be expected, the rate of percolation decreased with additional percentages of fines (36).

A comparison of unconfined compressive strengths on test samples from U.S. mines revealed that similar results were achieved with specimens composed of differing particle size combination (9). Specimens prepared with well-graded sand and moderately gentle slope on grain size scale yielded the higher

565-423 O - 75 - 4

average strengths. The highest strengths (5:1 sand-cement ratio) were surprisingly obtained from a sample having no particles larger than 70 mesh, uniformly graded, and with a wide range of finer particles (23 percent finer than 400 mesh). Alike specimens, however, had below average strengths at the leaner sand-cement mixes. A coarse, uniformly graded specimen (less than 4 percent minus 200 mesh) yielded the lowest strengths at all sand-cement mixtures.

A regression analysis of test data from six different fill materials of varying cement contents resulted in formulation of strength prediction equations (9). Cement content and grain-size gradation were found to be primary factors affecting sample strength. A relatively simple equation was developed to account for 96 percent of the variability of the data:

$$\ln \text{strength (psi)} = 8.271565 + 1.482918 \ln \text{cement} + 0.1129835\, C_u, \quad (11)$$

in which C_u = uniformity coefficient (D_{60}/D_{10}),

D_{60} = diameter in millimeters of 60-percent-finer size,

and D_{10} = diameter in millimeters of 10-percent-finer size.

Subsequent testing of addition fill materials indicated that the equations yielded realistic strength estimates for fills having physical and mineralogical properties within the range used in the analysis.

Investigations at the Falconbridge Group of Mines in Canada have also indicated that similar strengths can be achieved using classified tailings of differing particle size (12). It was suggested, however, that a reduced strength "up to 50 percent" was realized at mines adding coarser sand to tailings. This was attributed to "particle size distribution of cement relative to tailings backfill." Sandfill with higher cement-sand ratios exhibited layering or stratification within the sandfill material. The investigators suggested the use of a coarser cement to alleviate this situation.

One basic objective of sandfill research was to find or develop an additive to strengthen cemented fill at minimum cost. Various materials added to sandfill have included dispersants, flocculants, and fly ash. Again, similar research was initiated by soil engineering firms. The load-bearing capacity of soil beneath structural foundations has been improved by mixing cement with the soil. Further alterations of soil structures to increase its strength have been accomplished through the use of additives. Predominately dispersants, such as sodium silicate, sodium metaphosphate, and sodium polyphosphate, have been successfully employed. Dispersants will coat the soil particles thus encouraging the clay particles to repulse each other without increasing the attraction forces. The soil particles become oriented, thus improving the bonding and strength of the material.

The compressive strength of backfill modified with cement and a dispersant was determined by Corson (8). The quantities of cement and dispersant (sodium hexametaphosphate) were varied to determine the optimum mixture for achieving the highest strength. Average values for the tests are shown in table 5. Addition of a minor amount of dispersant is beneficial in increasing strength of lean (40:1 and 30:1) sand-cement ratios. An increase in strength in 40:1 sand-cement samples of approximately 50 percent was achieved with a 0.1 dispersant-cement ratio (fig. 16). As noted on figure 16, little increase in strength is achieved through further dispersant addition until a dispersant-cement ratio of 0.8 is exceeded. Using the 40:1 sand-cement mixture, the optimum cement-dispersant ratio was 1.8:1. This mixture yielded a bearing strength of 515 percent higher than strength values obtained without dispersant treatment. The richer mixes (sand-cement ratios of 10:1 and 5:1) required additional dispersant in excess of 1 weight-percent of dry materials before an increase in strength was achieved. The strength values continued to increase up to 5 weight-percent of materials. At 10-weight-percent dispersant with 5:1 sand-cement mixture, a drop in compressive strength was recorded.

TABLE 5. - Average unconfined compressive strengths of cemented sandfill modified by a dispersant, pounds per square inch

Sand-cement ratio	Dispersant, pounds per ton of tailings								
	0	2	4	6	8	10	20	100	200
No cement..	5.23	-	-	-	-	2.92	2.89	-	-
40:1.......	24.82	28.47	35.95	-	-	34.93	38.98	87.05	-
30:1.......	35.19	42.17	47.46	-	-	51.63	52.31	179.89	-
20:1.......	72.87	-	81.85	77.37	-	-	88.0	138.0	-
10:1.......	251.85	-	123.64	173.0	190.1	220.0	236.84	278.61	657.0
5:1.......	711.06	-	334.14	-	-	372.49	644.79	754.07	498.84

Another method of modifying the structural arrangement of soil particles can be accomplished through the use of flocculants. A flocculated structure is disorganized and haphazard; consequently, a reduced strength material would exist. The loose arrangement of the individual particles would facilitate entrapment of excess water and create a soil that would be highly compressible and lightweight. Although flocculant structures are not desirable in soil foundations, unconfined compressive strength tests are being determined on backfill specimens modified with cement and flocculants. It is thought that the coarser backfill materials may react differently as compared with soils, which may contain a high percentage of minus 200-mesh particles.

Originally, the use of flocculants was considered because it was thought that the flocculant would decrease the quantity of fines being carried in stope decant water.

Laboratory tests have shown the flocculation of sandfill will retain a substantial portion of the minus 20-micrometer material that would escape with sandfill decant water. Percolation rates (4, 36) through sandfill varied,

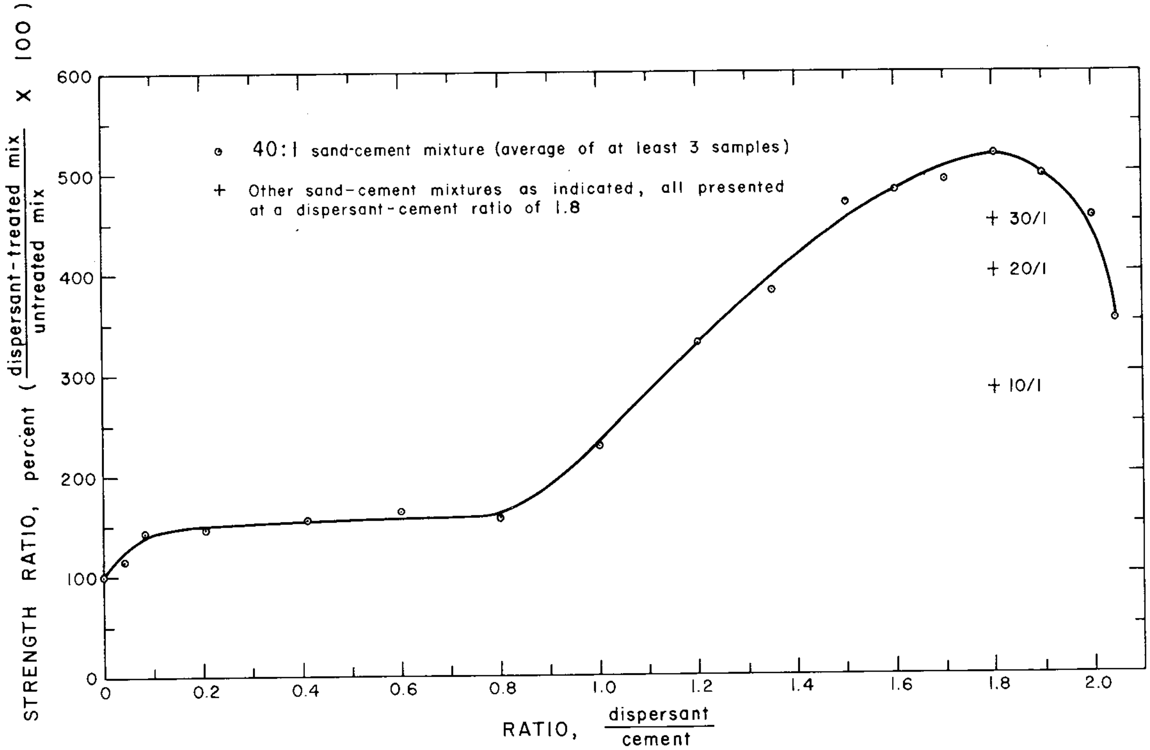

FIGURE 16. - Effect of dispersant addition on strength of 40:1 sand-cement samples.

depending upon the type of flocculant added (table 6). Results of percolation tests on samples with no flocculant and 0.01 pound of polyacrylamide B[10] per ton of tailings at a 20:1 sand-cement ratio were 14.74 and 23.62 inches per hour, respectively (6). Laboratory observation during sample preparation indicated that polyacrylamide C retained practically all water mixed with sample, but water escaped from those samples prepared with ferric sulfate (20).

TABLE 6. - Effect of flocculant on percolation rate, inches per hour

Flocculant	Conc., lb/ton	No cement added		20:1 sand-cement mixture (37, p. 996)	
		No flocculant	Flocculant	No flocculant	Flocculant
Synthetic, acrilamide polymer..................	0.05	2.0	10.5	-	-
Polyacrylamide A..........	.25	2.0	15.5	-	-
Galactomannon gum-Guaran..	.10	2.0	9.6	-	-
Polacrylamide B...........	-	-	-	14.74	23.62

[10] The basic component of three of the flocculants tested was polyacrylamide. Therefore, they have been distinguished by letters A, B, and C.

Unconfined compressive strengths of cemented sandfill modified with different types of flocculants are presented in table 7. In all tests, samples containing flocculants experienced an increase in strength with curing time. Strength values on test samples containing ferric sulfate were higher than samples containing no flocculant. After a 28-day curing period, strength of samples containing polacrylamide C would exceed the values obtained on samples having no flocculant modifier. This strength increase was attributed to drying of the specimen with curing time.

TABLE 7. - Average unconfined compressive strengths of cemented sandfill modified with flocculants, pounds per square inch[1]

Sand-cement ratio	No flocculant	Ferric sulfate 10 lb/ton	Ferric sulfate 20 lb/ton	Polyacrylamide C 10 lb/ton	Polyacrylamide C 20 lb/ton	Polyacrylamide B (37, p. 995) 0.01 lb/ton	Polyacrylamide B (37, p. 995) 0.03 lb/ton
30:1	53.56	113.27	122.67	113.27	201.01	21	24
20:1	75.0	-	-	-	-	41	42
15:1	138.66	160.22	138.99	160.22	323.44	-	-
10:1	107.0	-	-	-	-	126	126
7:1	412.91	527.3	264.09	431.88	443.44	-	-

[1]Curing period--28 days.

Because of successful application of fly ash with concrete, the increasing supply of fly ash due to air pollution legislation, and its low cost, fly ash appeared as a likely sandfill modifier. The mineralogical composition varies, but silica, alumina, and iron generally comprised over 80 percent of the total constituents. Fly ash is a pozzolan that is not an adhesive itself, but reacts with lime in cement to form a compound that exhibits cementitious characteristics. A limited series of laboratory unconfined compressive and triaxial tests have been performed to evaluate the effect of mixing fly ash with cemented fill. Weaver and Luka (36) found that similar sample strengths (1:8 cement-sand ratio) were achieved with up to 20 percent fly ash content. At leaner ratios, a comparison of sample strengths is much less favorable. Compressive strength values are listed in table 8. Corson (9) made a similar investigation substituting fly ash for part of the cement component. The research was expanded to identify the influence of fly ash particle size. Using classified mine tailings from mine B (fig. 5), 1 part fly ash was mixed with 2 parts cement in a 20:1 sand-cement sample batch. Compressive strength of 65 and 121.4 psi were obtained using coarse and fine fly ash, respectively. A strength of 72.9 psi was recorded on a straight sand-cement mixture of 20:1. Unconfined compressive strengths for 2 parts fly ash to 1 part cement were lower, but the values were again higher for those samples mixed with a finer sized fly ash.

Triaxial test results supported the aforementioned strength values using fill from mine B. Fine size fly ash yielded significantly higher strength than the coarse size material. The mixture composed of one-third fine fly ash had higher strength than samples containing two-thirds coarse fly ash.

TABLE 8. - Strengths of cemented sandfill with addition of fly ash,[1] pounds per square inch

Sand-cement ratio	Fly ash-to-cement ratio, percent						
	Intermixed					Interground	
	0	10	20	33	66	10	20
20:1	-	-	-	[2]121.4	[2]40.0	-	-
				[3]65.0	[3]37.9	-	-
10:1	-	-	-	-	18.1	-	-
[4]8:1	-	[2]141	[2]117	-	-	[2]144	[2]121

[1]Curing period--7 days.
[2]Fine fly ash.
[3]Coarse fly ash.
[4]Taken from page 993 of reference 36.

Testing with other additives to replace or to enhance stabilization with portland cement met with varied success (9). Results of limited testing of sodium silicate, emulsified asphalt, and gypsum as stabilization additives to sandfill were not encouraging. An evaluation of expansive cement indicated that the effectiveness of this type of additive is hampered by the excess water present in a hydraulically emplaced sandfill. Strengths of samples incorporating expansive components were generally less than those containing a comparable amount of portland cement, and expansive stresses developed during a 7-day restrained cure were minimal.

Triaxial tests of a relatively fine (unclassified) and a well-graded fill reflected the influence of grain-size distribution on cemented strength. The well-graded material exhibited a higher confined strength and larger angle of internal friction, whereas the finer fill appeared to have the higher value for cohesion. Likewise, triaxial testing indicated that partial substitution of fly ash for the cement in a 20:1 mix resulted in a lower strength and higher cohesion. If cohesion is indicative of the ability of a material to withstand abrasion, incorporation of fly ash may be desirable when pouring a slusher floor (9, 36).

Attempted simulation of stope sandfilling using a plexiglass model demonstrated, at least qualitatively, the effect of some factors on the dispersion of cement within a fill mass. The cement tended to segregate from the fill and collect where the excess water ponded at the ends of the model. This condition was reduced to some extent by adding a dispersant and by using a manifold, which permitted simultaneous discharge at several points along the length of the model. Pouring through a manifold also produced a relatively level surface compared with pours made through a single outlet held at one location in the model. This type of filling technique would be desirable in achieving level scraping floors, particularly when a cement capping is used. A small volume of cemented fill would be required to produce an adequate thickness of capping if the underlying fill surface were level (9).

Electroosmosis

The application of electroosmosis to dewater soils having low permeability has been used by soil engineers just as other techniques have been used to strengthen soils as already discussed. Until recent research (31) at the Federal Bureau of Mines Spokane Mining Research Center in Washington, raw or unclassified mill tailings had never been successfully dewatered using electroosmosis or any other method. The general theory of electroosmosis is the placing of negative electrodes (cathodes) in such a pattern near a proposed water discharge point and applying a direct current. This current will generally encourage the water to move toward the cathodes.

Bureau engineers designed a series of laboratory experiments to evaluate the effectiveness of electroosmosis in dewatering unclassified mill tailings. The success of this research was directed toward the possible placement of unclassified tailings in underground stopes instead of classified tailings. The data presented herein represent laboratory experiments conducted in small, clear, 0.3 cubic foot plexiglass models which would contain approximately 37 pounds of slurry. At the time of the writing, larger scale model experiments were being completed and a full-scale in-mine demonstration was ready to be undertaken. Although many variables can influence the degree of dewatering achieved for any material, this report will outline only the major procedural changes made during the investigation. The various procedures tested to dewater and densify the unclassified tailings were current density, periodic current reversal, intermittent current, agitation of tailings, variation of anode size and shape, cathodic current densities, vertical versus horizontal electrode position, variation in electrode materials, modifying sand-to-water placement ratios, and placement of tailings in layers.

The results of these preliminary tests indicated that unclassified tailings can be dewatered and densified by electroosmosis. The results of this series of tests include the following:

1. Tailings with low initial water contents (slurry having a lower volume of transport water) will achieve a lower water content in the final product.

2. Material should be permitted to drain naturally prior to application of direct current.

3. Using periodic current reversal, best test results were achieved with current reversals in the ratio of 8.33/15 and 17/30 minutes.

4. Improved dewatering was achieved with agitation.

5. Intermittent current tests gave improved densification results with reduced power costs.

6. Lead cathodes, although more costly, were more effective than iron cathodes.

Results of tests indicate that water content of treated tailings decreased from 42- to 63-volume-percent water to 18 to 36 volume-percent. At the same time, densities increased from values ranging from 62 to 78 pounds per cubic foot to 81 to 120 pounds per cubic foot after electroosmotic treatment and agitation.

FIELD EVALUATION OF SANDFILL EFFECTIVENESS

Among the research efforts to evaluate and enhance the support capability of sandfill, several studies have been conducted to monitor fill pressures and vein-wall deformation. These studies have provided vital data that not only enable the understanding of in situ mine conditions, but also offer the opportunity to theoretically model these conditions and to determine what sandfill competency will be required to alleviate heavy ground conditions.

Instrumentation

Pressure Readings

1. Hydraulic pressure cells (HPC) were used to monitor sandfill pressure changes. The cell, connecting tubing, gage block, and readout gage were filled with glycerin, and fill pressure on the HPC was reflected by a corresponding reading on the gage (24, p. 11). In order to interpret the hydraulic pressure cell readings in terms of fill pressure a series of laboratory tests were conducted using identical HPC's, as well as smaller cells of the same design, at various depths in backfill of minimum and maximum density. Results indicated that the observed pressure is sufficently close to the actual pressure, particularly in the range of 400 to 600 pounds per square inch, to be used without any further correction. Major problems with these gages are that (1) instruments cannot be repaired once stopes has been sandfilled, and (2) HPC may be reoriented with stope wall closure and shifting within the sandfill.

Bourdon gages of 500- and 1,000-pound-per-square-inch capacity, depending upon the orientation of the associated HPC, were used to indicate pressures within the backfilled stope. The stainless steel gage block was fitted to accept a pump to faciliate filling the connecting 1/4-inch-OD steel tubing during installation of the cells. A valve between the gage block and the gage provided control of fluid pressure and allowed the replacement of gages, should they malfunction or their capacity be exceeded. Gages were housed in protective cabinets within the manway.

Borehole pressure cells (BPC) were installed in the rib rock to monitor rock pressure changes (24). The system consists essentially of the same components as the HPC's except that steel tubing, which replaces the flexible connecting tube, is used to withstand the greater pressures (up to 2,500 pounds per square inch). The major difficulties encountered with this instrumentation were (1) the BPC must be set in a cement-mortared borehole and care must be taken to assure that air pockets do not exist; (2) it is impossible to repair or place the equipment after the stope is filled; and (3) corrosion and rupture of steel tubing, again not repairable.

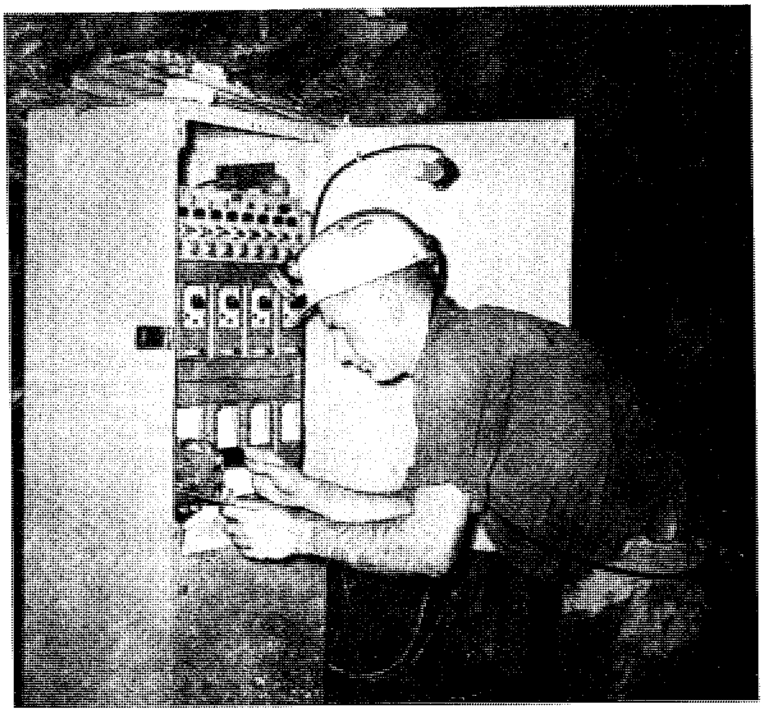

FIGURE 17. - Replacing tape on first-generation continuous recorder.

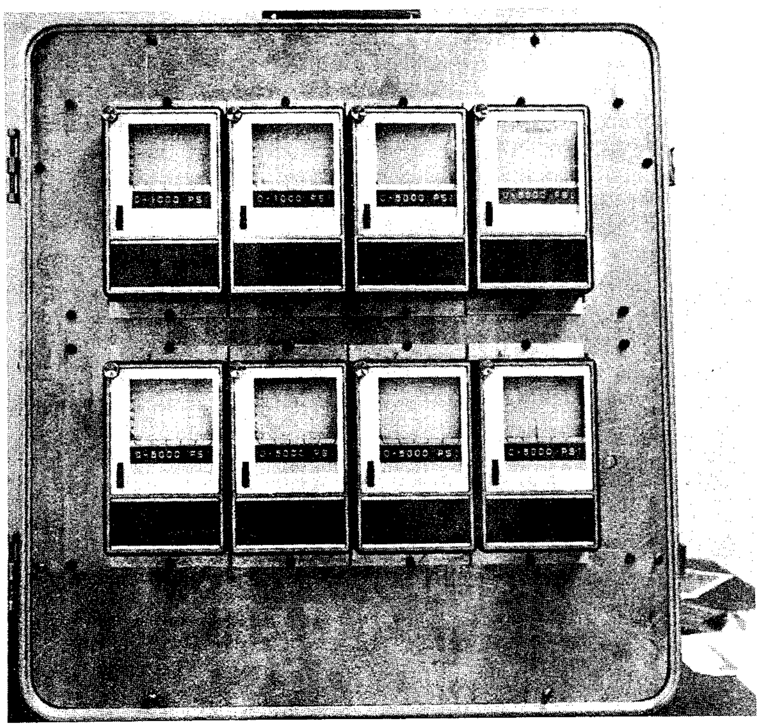

FIGURE 18. - Second-generation continuous recorder.

Later, an automatic recording system (fig. 17), which was successfully utilized to monitor pressures, consisted of full-bridge strain gage-type pressure transducers, a transducer power supply and balance unit, signal conditioning amplifiers, and strip chart recorders. The transducers replaced pressure gages in existing installations in the manway and were connected by shielded strain gage cables to a metal cabinet located in the crosscut that housed the other components.

A second-generation continuous recorder was designed and fabricated; this recorder was more compact (fig. 18). The basic modification included replacing the full-bridge, strain gage-type pressure transducers, balance unit and signal conditioning amplifiers with a potentiometer-type, high-output transducer with signal conditioning.

2. Vein-wall closure was monitored in the raise area by measuring between rock bolts placed diametrically opposite each other in the walls. The ends of the bolts were allowed to extend into the manway crib and any excess protrusion was cut off (fig. 19). Measurement were taken with a dial extensometer having a 4-inch travel. A combination of different length extensions and points allowed measurement of the varied distance between bolt ends.

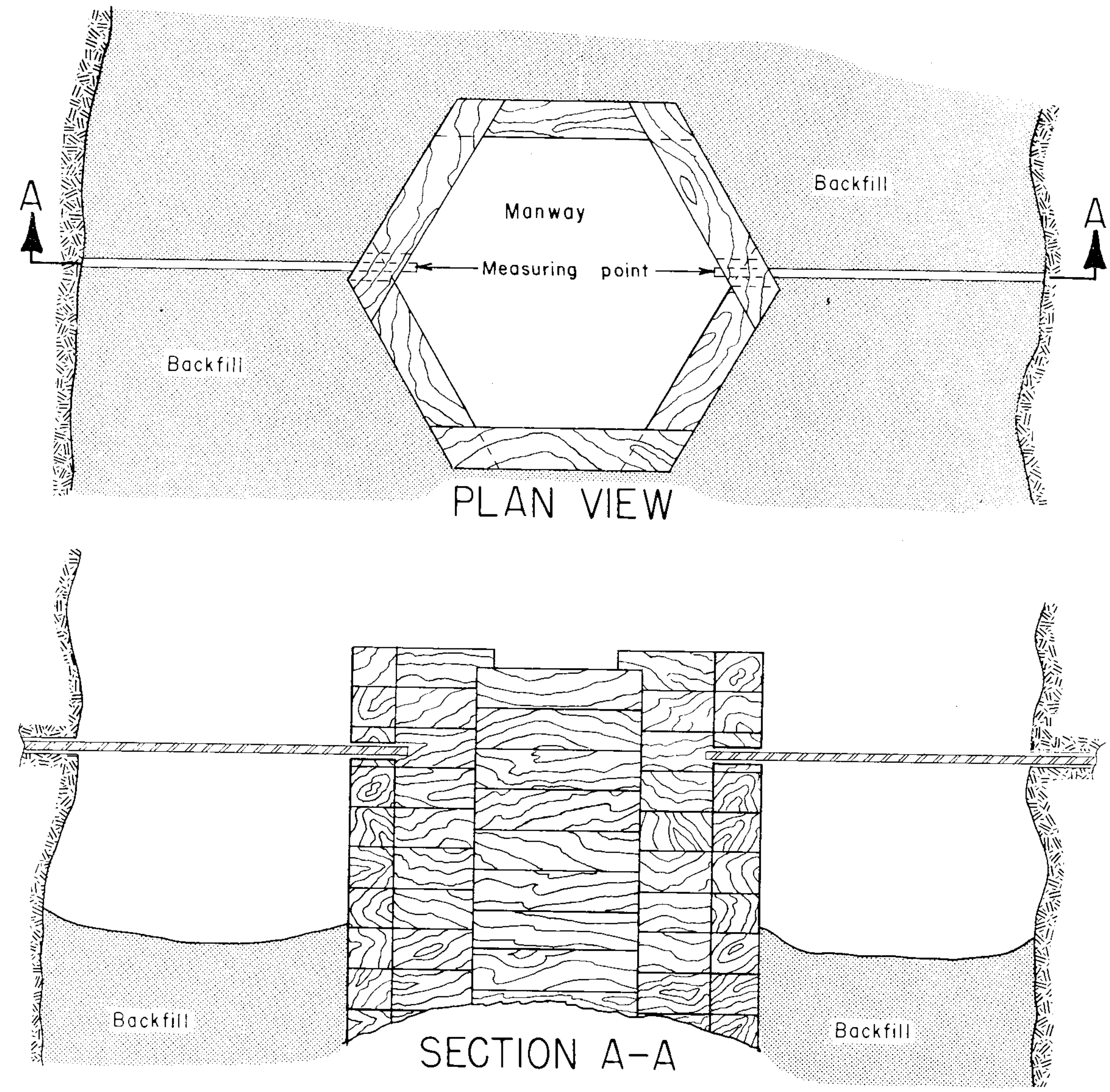

FIGURE 19. - Raise closure-measuring installation.

3. A hydraulic cylinder (fig. 20) was fabricated to measure total stope wall closure after the sandfill had been placed. The cylinder was attached to two rock bolts diametrically opposite each other in the stope. As wall closure occurred, hydraulic fluid was transferred from the cylinder through a plastic tube to a manometer-type readout unit. Closure of the vein walls displaced oil within the cylinder; fluid displaced is measurable in the readout unit. This unit has two columns for measurement of fluid flow: a larger tube having the same inside diameter as the hydraulic cylinder and a second smaller tube. Oil from the cylinder is introduced into the smaller tube, whose small diameter produced approximately a 10 to 1 magnification of the amount of wall movement to be measure. A 1-inch movement of the piston within the hydraulic cylinder yields a 10-inch rise in height of the fluid in this smaller readout tube, as determined by laboratory calibration prior to

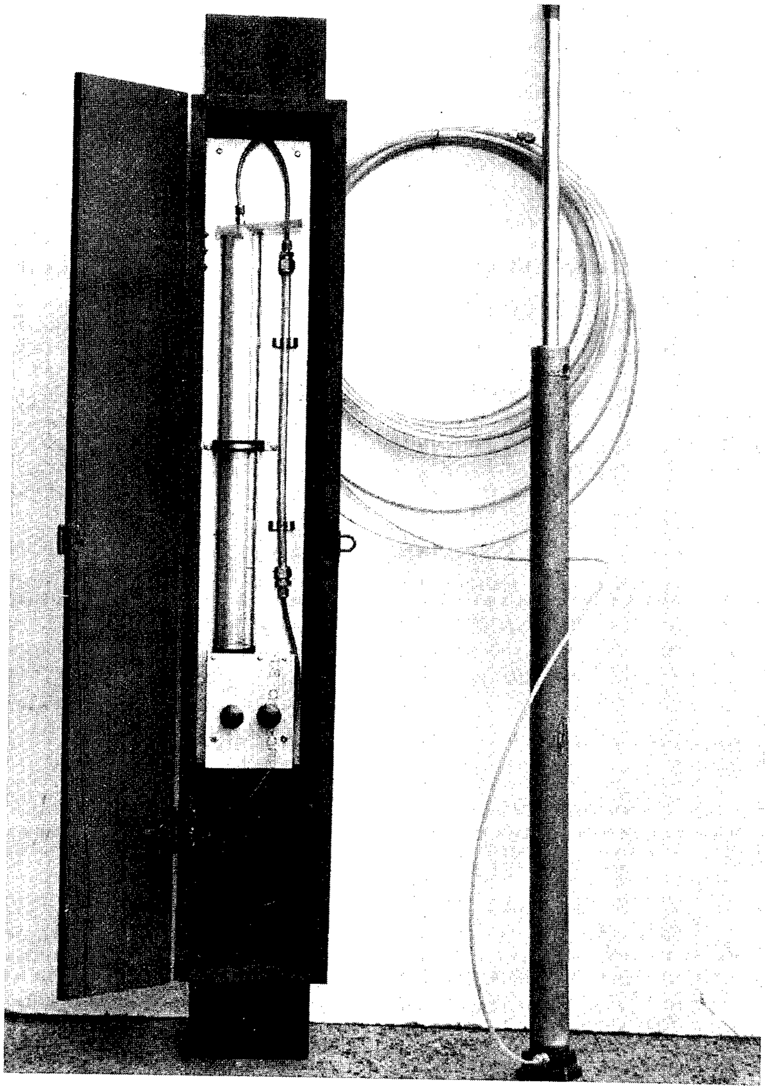

FIGURE 20. - Readout unit and hydraulic device for measuring stope wall closure.

installation. Provision was made to permit emptying the smaller tube into the larger without disconnecting any of the fittings; any overflow from the smaller tube would automatically be transferred to the larger container. Thus, the larger tube provides a gross (1 to 1) measurement of wall movement, whereas the smaller tube (10 to 1 magnification) permits measurement of small increments of closure (10).[11]

This device was uncomplicated. The only drawback was the readout device. This particular unit was large for use in a mine, and the readout tubes were prone to damage.

One new innovation was a mechanical device to monitor wall closure in the backfilled stope, as shown in figure 21. Rock bolts were placed opposite each other in the stop walls, and extensions from these bolts overlap approximately 2 feet in the middle of the stope. Stainless steel wires, affixed to the ends of the extensions, are led through 1/8-inch plastic tubing to the raise. The interior ends of the tubing are fastened to a T-shaped member called a tee) located in the center of the stope and adjacent to the overlap of the rock bolt extensions. The exterior ends (in the raise) of the tubing are fastened to a curved, metal bracket; the wires are allowed to extend below the ends of the tubing. Metal clamps are affixed to the wires at some set distance from the ends of the tubing, and lead weights are suspended from the ends of the wires.

[11]The numbers indicate wall movement. In the large tube, the level of hydraulic fluid would rise 1 inch for each inch that the stope walls would close; whereas, the fluid in the smaller tube would rise 10 inches for each inch the stope wall would close. This process enables the engineer to measure wall movements of hundredths or thousandths of an inch.

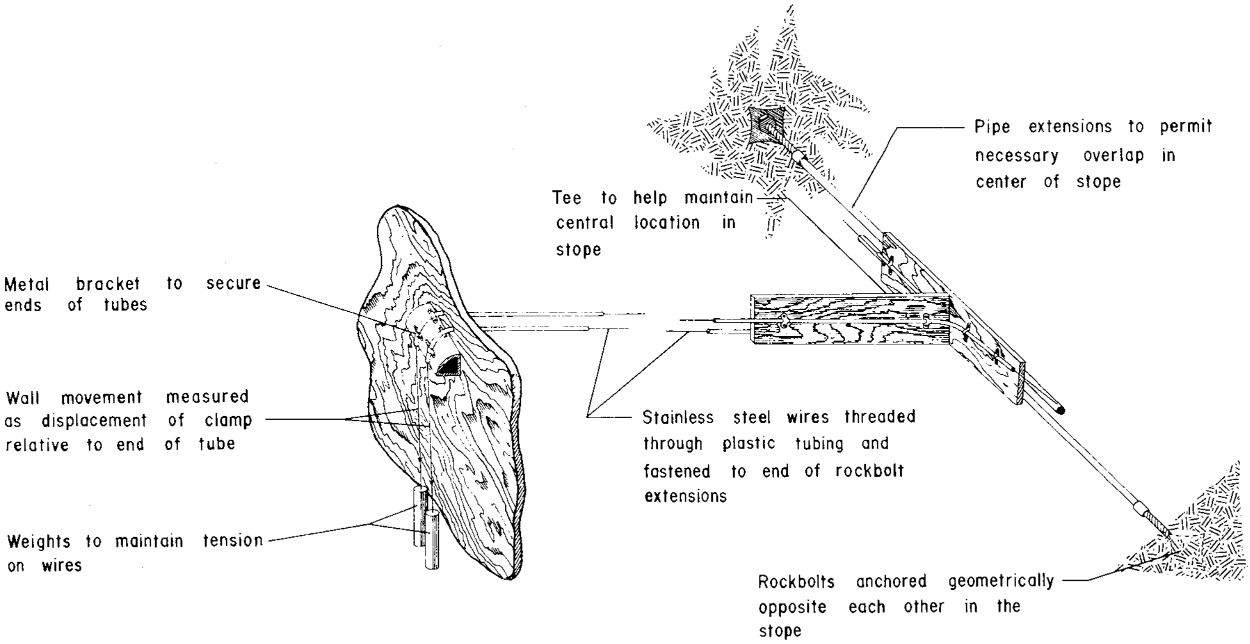

FIGURE 21. - Schematic diagram of mechanical closure device.

As the stope walls close, movement of the rock bolts draws the wires through the plastic tubes. The rock bolts and the T-shaped member are assumed to remain stationary because of their central position, relatively static within the backfill. Individual vein-wall movement is measured by the amount of displacement of the clamp on its respective wire with relation to the end of the corresponding plastic tube. To inhibit access of water, both ends of the plastic tubes are filled with grease.

4. Explosive borehole anchors, called down-the-hole anchors (DTH), were also installed to measure rock movement on each side of the vein in advance of mining (fig. 22). Diamond drill holes were placed through which the DTH anchors could be set at a predetermined depth, and with the detonation of a mild end primer and low-energy detonating cord, the outer tube would swell against the borehole rock. The indicated movement of the expanded anchors is monitored by measuring the distance between a stationary point and a button attached to wire leading to the anchor. Detailed information can be found in a report prepared by Parsons and Osen (26). Additional information on field application of the instrument was discussed by Waddell (34).

5. Commercially available probes made by the Nuclear Chicago Corp. were used to determine depth density and moisture of placed sandfill. Both the model P20 depth density gage and the model P19 depth moisture gage are capable of monitoring soils to a depth of 200 feet below the surface. Both probes were used in conjunction with a Nuclear Chicago model 2800 portable scaler (fig. 23). Count rates indicated by the scaler were converted to wet density in pounds per cubic foot and pounds of water per cubic foot (moisture content) by using the calibration graphs supplied with each probe. Thus, monitoring of the same point in the tubing with both probes yielded the dry density of

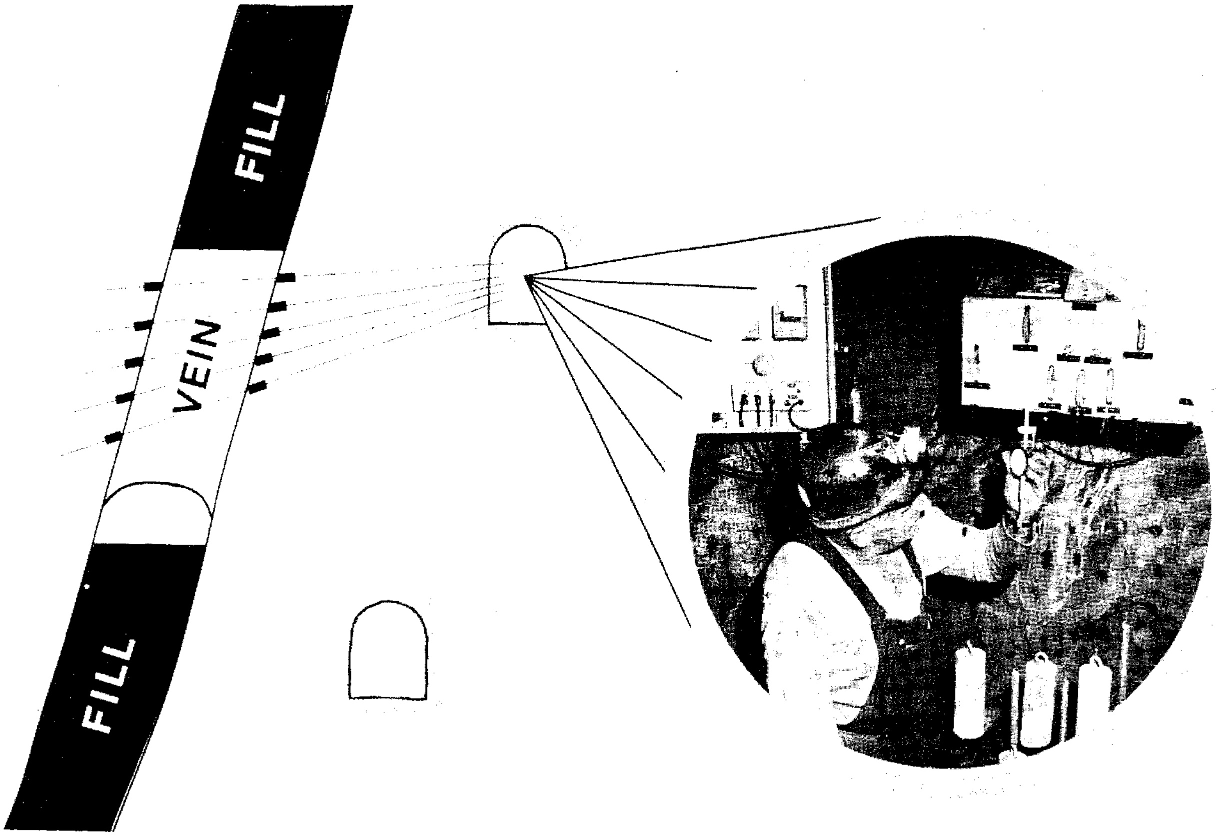

FIGURE 22. - Monitoring movement of explosive borehole anchor.

the backfill at that point. The probes functioned well when calibrated in the laboratory; however, this equipment is highly susceptible to distortion and erroneous readings when exposed to varying degrees of temperature and humidity. Consequently, with the fluctuating atmosphere in the deep vein mines, the reliability of the instrumentation was questioned.

Field Data

Field projects have been conducted by the Federal Bureau of Mines at mines operated by Hecla Mining Co. in the Coeur d'Alene mining district of Idaho, and by Department of Energy, Mines and Resources, Ontario, Canada, at the Geco mine, Noranda Mines Ltd. and the Garson mine, International Nickel Co.

Star Mine Installation

The Star mine (11) workings extend vertically from an elevation of 5,130 feet above sea level to 1,947 feet below sea level, or some 7,900 feet below the surface. Stope filling with classified mill tailings was instituted in 1958, and in general, the mine used untimbered cut-and-fill stoping above the 4,000 level and a timbered cut-and-fill method in stopes below this level. The ground in this lower country where the instrumentation was located is considered very incompetent, with a history of movement and hydrothermal alteration.

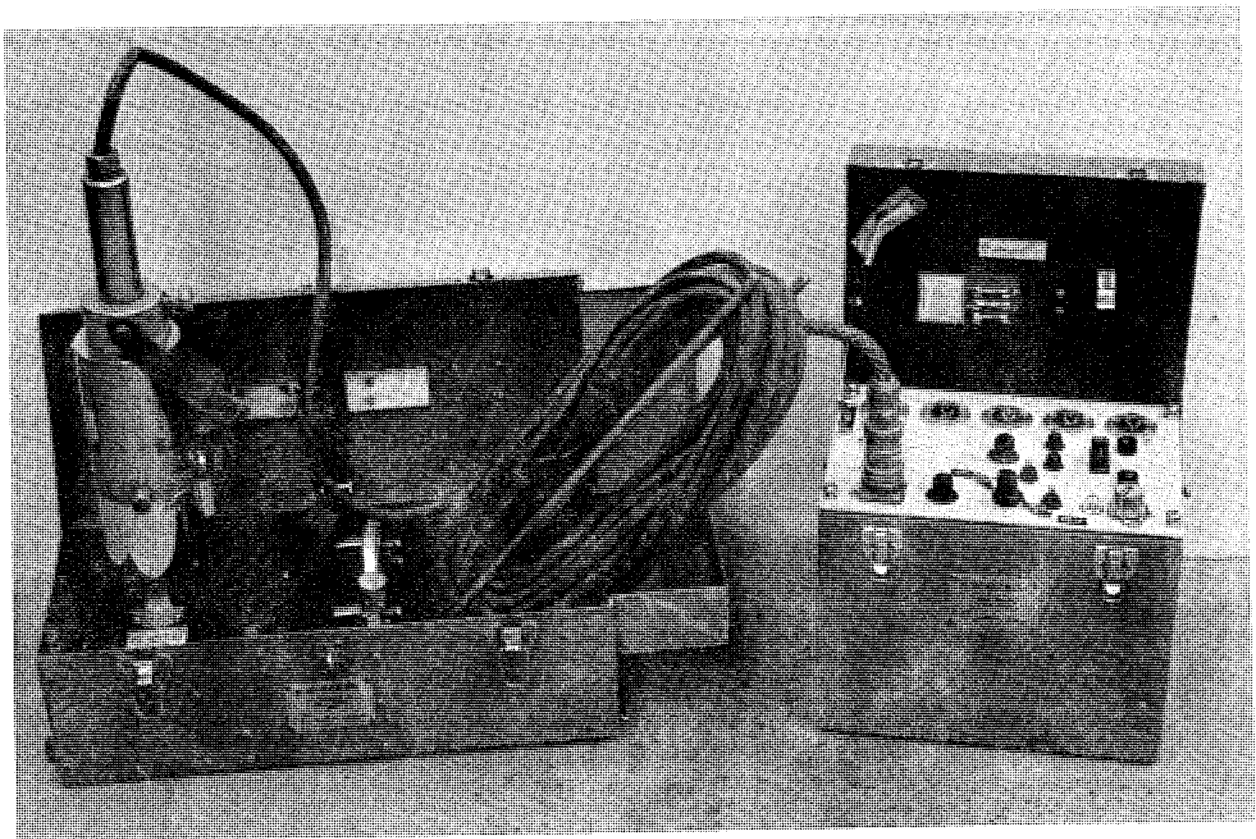

FIGURE 23. - Radioactive density probe and scaler.

A "blind stoping" method of mining, utilizing prefabricated, hexagonal cribs for ore chutes, manway, and timber slide, was used on the instrumented level. The vein is essentially straight and nearly vertical with an average width of approximately 10 feet. Stopes are mined both directions from the cribs, taking a 10-foot-high cut along the vein. Raising for the next floor is done before mining is completed on the work floor. Stopes are timbered with heavy 12- to 14-inch-diameter caps on 6-foot centers supported with 6- to 8-inch posts. A combination of solid and staggered planking is placed above each set of timber. Solid planking is used up to 21 inches, and any distance over this to the back is filled with a latticework of planks. Rock bolts are installed in both walls across the raise area.

Stopes are sandfilled as soon as possible after cleanout. A fill retention wall is built at the end of the cleaned out stope, and a burlapped sandwall is erected next to the raise area prior to filling. The stoping areas on either side of a raise are normally filled before raising the cribs and filling the raise area.

The field installation encompassed stopes on each side of No. 5 raise, 6,700 level of the Star mine through a vertical distance of 200 feet and a distance of 120 feet along the strike of the vein. A total of 22 pressure cells were placed in the fill at various orientations to the vein, and 6 veinwall closure measuring points were installed at selected elevations in the raise.

Pressure cells oriented both parallel and transverse to the vein on the 7th and 14th floors indicated that initial pressures developed were essentially hydrostatic. Pressure registered by the parallel cells increased at a higher rate than the pressure indicated by the transverse cells, as was anticipated. However, after a reading of approximately 100 psi was exceeded, the pressure longitudinal to the stope fell off to zero. A possible explanation for this is that the cell attempts to orient its large surface perpendicular to the direction of principle stress, and this movement of the cell ruptures the connecting tubing.

Vein-wall closure on the various floors was monitored as long as the measure points remained accessible. Closures are shown in relation to their time of installation in figure 24.

Maximum vein-wall closure and recorded pressures within the backfill are summarized in table 9.

TABLE 9. - Maximum pressures and closures by floor (Star mine)

Floor	Maximum transverse pressure, psi[1]	Months[2]	Maximum longitudinal pressure, psi[2]	Months[2]	Maximum vertical pressure, psi[1]	Months[2]	Maximum closure, inches[3]	Months[2]
4	464	30	-	-	-	-	22.0	36
7	340	10	116	6	-	-	20.9	29
9	-	-	-	-	-	-	20.7	25
10	530	10	-	-	-	-	-	-
11	-	-	-	-	-	-	19.5	22
14	500	9	112	4	-	-	16.5	18
16	450	15	-	-	115	9	-	-
17	-	-	-	-	-	-	8.7	7

[1]Maximum observed pressure before cell malfunctioned.
[2]Approximate time after installation.
[3]Maximum closure before points became inaccessible.

Lucky Friday Mine Installation

The Lucky Friday mine (10) vein, which has little surface expression, has been developed to a depth of 4,050 feet and is one of the major lead-silver producers in the district. The vein is tabular and sinuous, and it ranges in width from a few inches to several feet. Mining employs the conventional cut-and-fill system, wherein the ore is removed in a series of successive horizontal slices, and the resulting void is sandfilled with mill tailings as each slice is removed. A cemented capping on the sandfill serves as a floor during mining of the next slice. Raises between levels may be timbered or formed, using either hexagonal, wooden cribs, or circular steel sections contained within the sandfilled area and extended upward as the mining progresses.

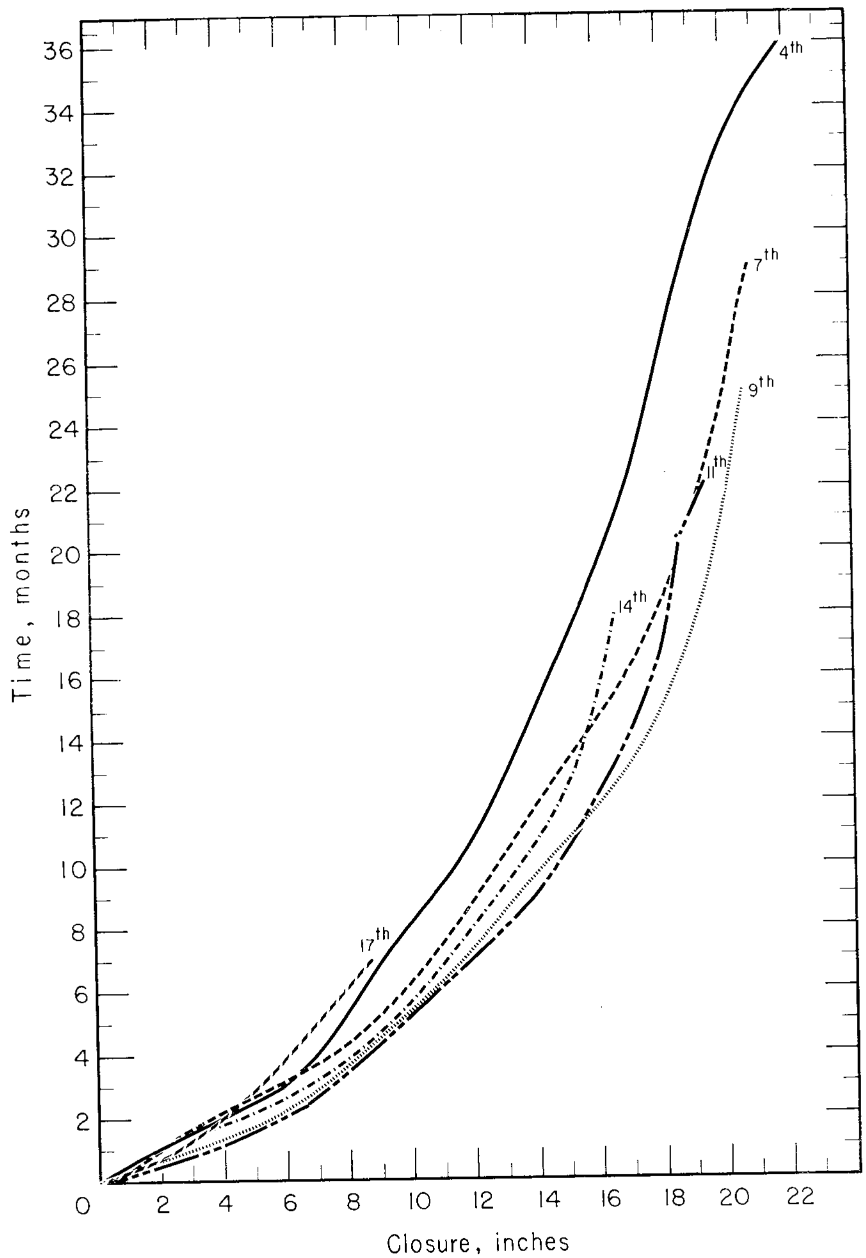

FIGURE 24. - Vein-wall closure versus time (Star mine).

The 108 raise was timbered and open from foot to hanging wall, whereas the 106 raise consisted of hexagonal, wooden cribs "floating" in the fill throughout most of the 200-foot elevation. When it became difficult to maintain the raise, circular steel chutes and manway sections were substituted in the upper portion of the stope.

In the initial stages of the study, monitoring of pressure and closure instrumentation was performed on a weekly basis. This schedule was later reduced to biweekly as the range of change in pressure and wall movement slowed down. At some intervals, readings were even more infrequent because of lack of access to one or the other raise. For the most part, data are presented graphically to facilitate comparison.

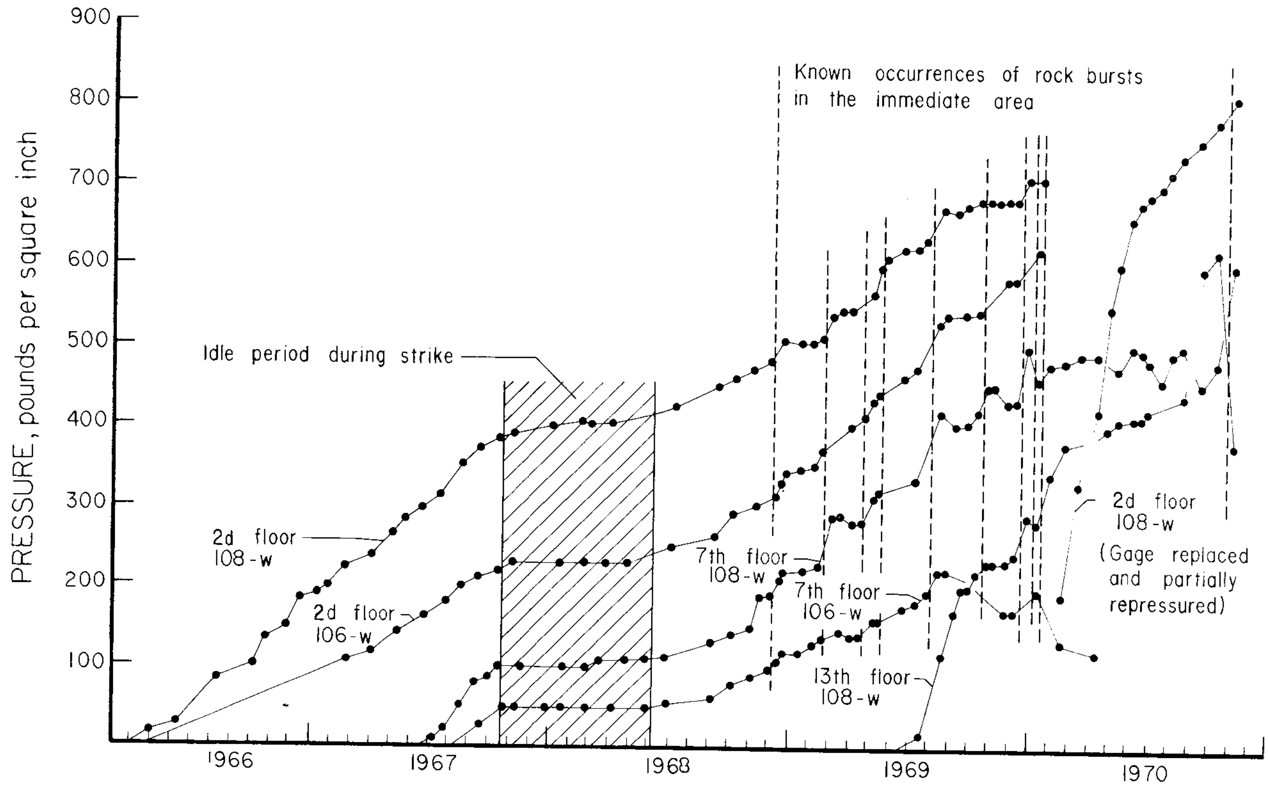

FIGURE 25. - Pressure across stopes in the Lucky Friday mine.

Figure 25 shows a comparison of transverse pressures in the five instrumented areas. The period of inactivity while the miners were on strike is reflected by the minimal increase in pressure. When mining was resumed, several rock bursts of varying magnitude occurred in the general vicinity of the instrumented stopes. As indicated on the graph, the rock bursts were frequently accompanied by an appreciable increase in pressure within the fill. This increment implies that a greater portion of the wall load was transferred to the sandfill during the failure producing the rock burst. The fact that this event is not shown at all instrumented areas for every burst may be due to its location in relation to the instrumentation. For example, a rock burst occurring within the sill pillar over an instrumented stope would probably affect pressure readings at all elevations within the stope, whereas a rock burst within the stope wall would have a more localized effect.

Comparison of closure rates in the two stopes is hampered by their variation in width, strike, etc. The rate of strain (inches of closure per inch width of stope) for the seventh cut in both stopes is shown in figure 26. As noted with the pressures monitored, strain during the period of the strike was negligible. Likewise, occurrences of rock bursts in the general area were accompanied, in most instances, by appreciable increases in strain rate. Despite the relatively small increment achieved in fill density in 108-W, the compacted fill did exhibit somewhat improved strain characteristics as compared with the natural fill in 106-W.

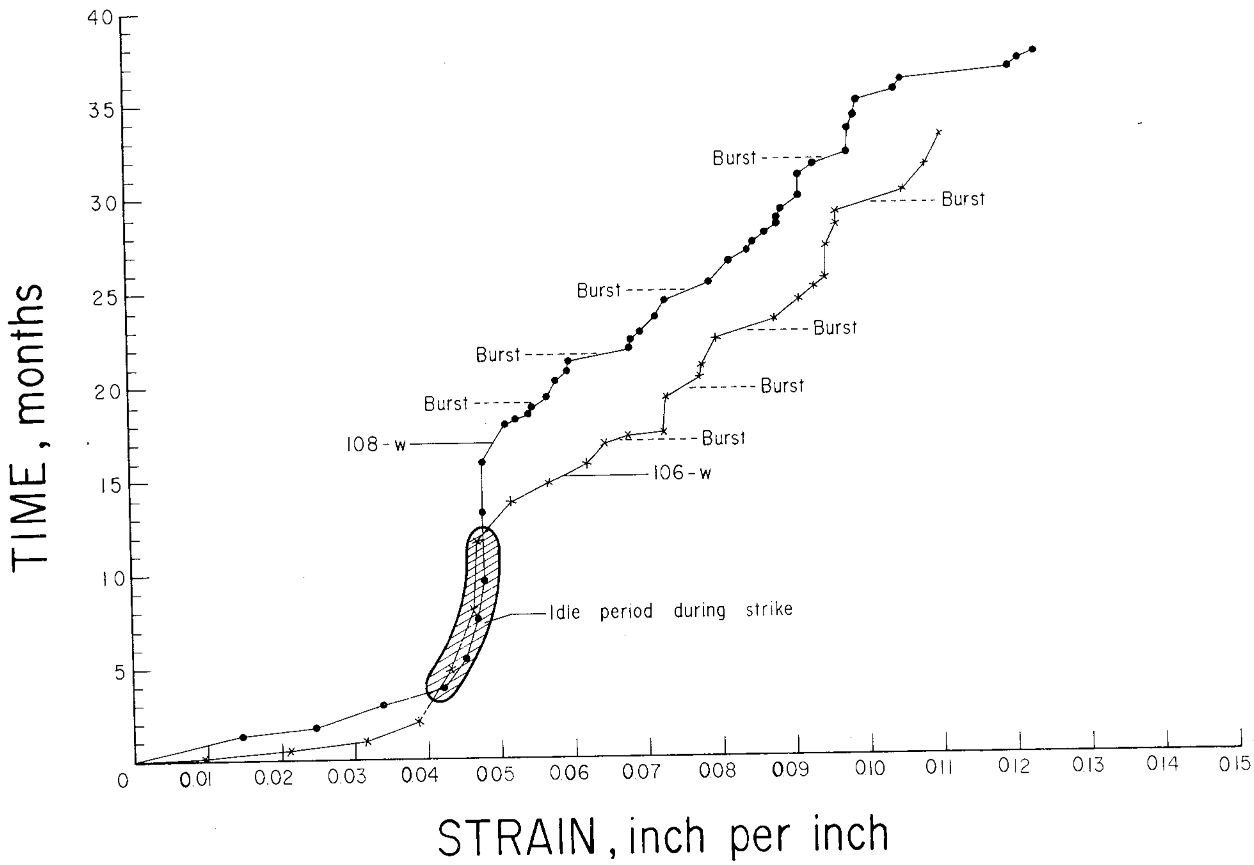

FIGURE 26. - Comparison of strain on seventh floor (Lucky Friday).

Geco Mine Installation

The Geco mine (37) has workings extending to a depth of 1,850 feet. Conventional cut-and-fill stopes are filled with classified mill tailings, whereas vertical blast-hole stopes are filled with quarry rock and stabilized with a cemented sandfill.

The investigation was undertaken to determine vein-wall deformation during the extraction of a vertical ore pillar and relate this data to the support offered by sandfilled stopes. Horizontal extensometers were installed from an access drift on the 1,050 level toward the vein. The four extensometers per hole were located to monitor rock deformation at the pillar to be mined and on both sides of the pillar. A fifth extensometer installation was placed on the opposite side of the access drift. Convergence was also measured in the access drift.

The resultant deformations following mining of the pillar were as follows:

1. After the initial blast, the access drift expanded in excess of 0.5 inch near the contact between current mining and the adjacent rock and sandfilled stope, but contraction occurred at the solid pillar area. Following a

2-year observation, drift expansion at the filled stope-mining contact approached 1.5 inches. The drift section near the pillar contracted over 0.5 inch. Practically no noticeable amount of deformation occurred in the drift opposite the rock-sandfilled stope as the distance from the mine site increased.

2. Stope-wall deformation measured with borehole extensometers was similar to the access drift measurements. Expansion was approximately 1.5 inches near the mined pillar and little or no deformation occurred away from the mined area whether or not the measurement was near a solid pillar or the rock-sandfilled stope.

3. The effect of mining on the filled stope was estimated from the deformation data. Measurements indicated that the stope wall moved 1.8 inches. The investigator estimated that the 60-foot filled stope compressed 0.25 percent and, with all effects taken into consideration, a total fill compression of 1 percent was established.

Garson Mine Installation

In a field study conducted by the Canadian Mines Branch at the Garson mine, fill pressure and wall convergence were monitored in a horizontal cut-and-fill operation that used a natural sand as fill (38). A comparison of stress-strain data in areas of cemented and uncemented sandfill indicated that the benefit to increased bearing capacity may be limited to the initial loading period. After exceeding the theoretical strength of the 10:1 sand-cement mixture, the cemented fill deformed about the same rate as the natural material.

Considering three categories of stope wall convergence, the fill was believed to have an important influence on convergence in the sandfilled area, limited influence in the working area of the stope, and no influence on convergence at and beyond the stope back.

THEORETICAL ANALYSIS OF HYDRAULIC SANDFILL

The preceding discussion in this report has described the results of laboratory and field studies to determine the effectiveness of hydraulic sandfill. Despite the information gained from this and other related research, there was needed a method of determining the quality of sandfill required for a particular ground support situation. As a result, preliminary numerical analyses of the support performance of various qualities of hydraulic sandfill were undertaken.

A two-dimensional model performing elastic-plastic, finite-element analyses was modified to investigate and evaluate the support capability of hydraulically placed sandfill. The latest model (14, 25) was developed for two vertically adjacent stopes separated by a sill pillar (fig. 27). The computer program was designed to predict stope convergence, stress in the sandfill, and stresses in the vein wall and sill pillar.

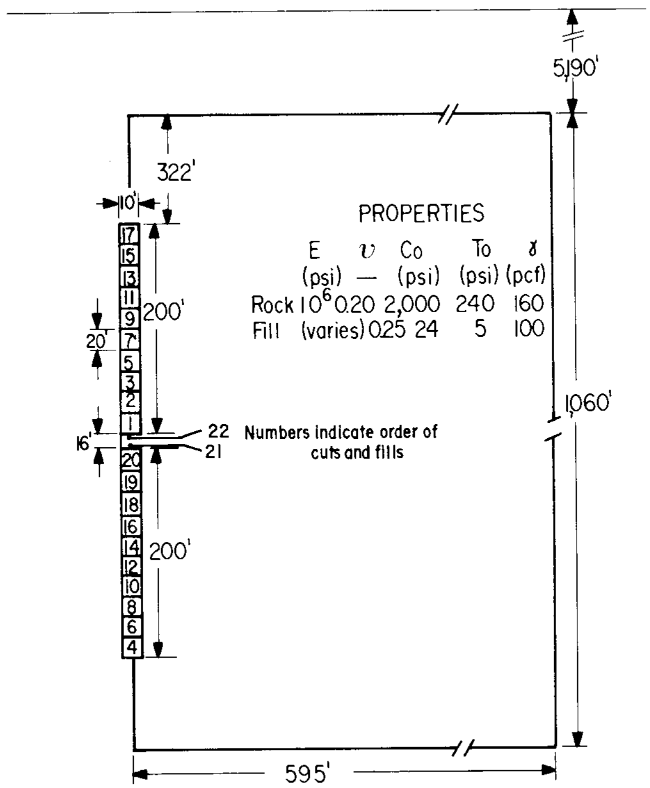

FIGURE 27. - Idealized stope, rock, and fill properties, and prescribed mining sequence.

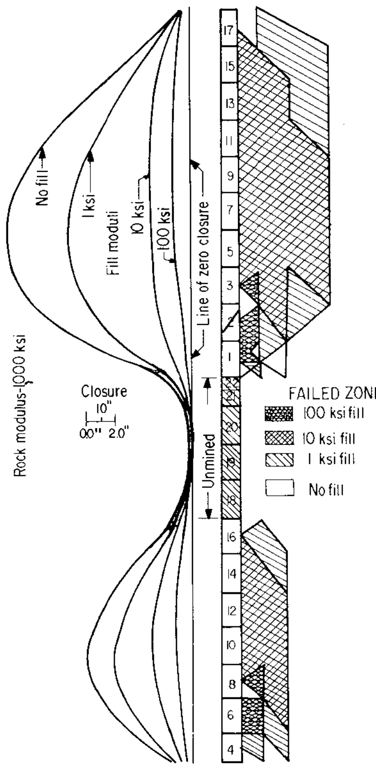

FIGURE 28. - Predicted stope closure as a function of fill modulus (25).

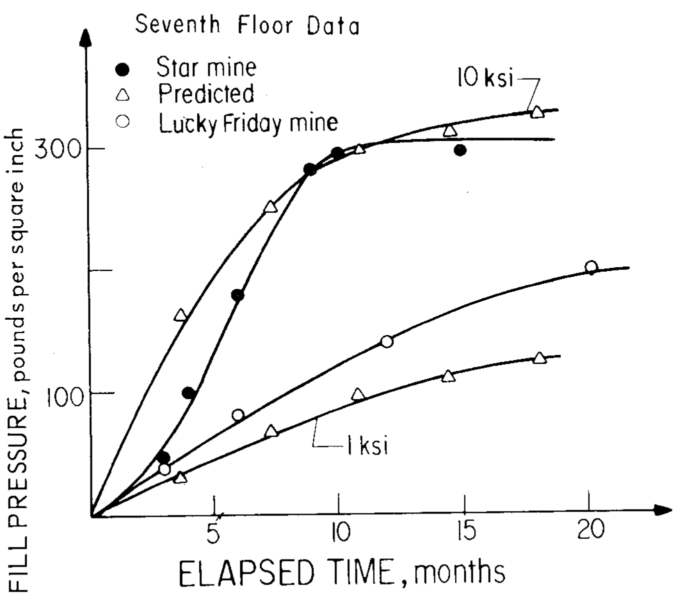

FIGURE 29. - Measured and predicted fill pressure versus time.

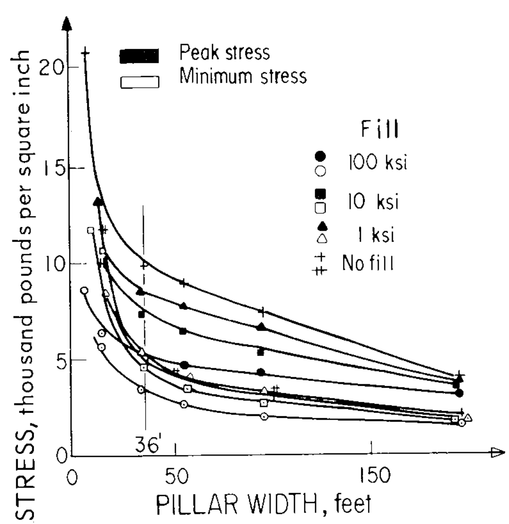

FIGURE 30. - Pillar stress normal to the vein at the face and in the core as mining of the pillar occurs.

The support performance of several sandfills of differing qualities were completed. The predicted stope closure as a function of fill modulus is shown in figure 28. A threefold decrease in stope closure was realized with a fill modulus increase from 1 to 10 thousand pounds per square inch (ksi), which is well within the realm of possibility. Theoretical sandfill pressures using a fill modulus of 1 and 10 ksi agree reasonably well with actual fill pressures measured at the Lucky Friday and Star mines, respectively (fig. 29).

Stresses in the vein-wall and, more importantly, the sill pillar were analyzed to compare the effectiveness of various sandfill in reducing increased sill pillar stress as mining approaches the pillar area. Initial stresses prior to mining were determined using normal gravitational or hydrostatic loading. Pillar stress as a function of sandfill is shown in figure 30. The stresses normal to the pillar are shown to be greater on smaller pillar width than on greater pillar width. Similar relationship exists in a stoping area. As stope mining advances upward toward the upper level, the ore pillar shrinks, and the incidence of heavy ground or rock bursts magnifies.

The results of these analyses indicate that, in addition to other benefits such as improving mine ventilation, controlling mine fires, and providing work platforms, hydraulic sandfill can be effectively used for ground control. The results also indicate that better quality sandfill alone would permit greater quantities of ore to be mined from the pillar before the rock burst situation became critical.

DISCUSSION

Placement of hydraulic sandfill in deep vein mines (cut-and-fill) has become recognized as an efficient, time-saving method of mining over previously used systems. Sandfill placement has obviously reduced the costly placement of timber, improved mine ventilation, increased ore production, and to some degree provided support to surrounding rock masses. This latter criteria is evident in those mines not experiencing heavy ground pressure conditions or in those circumstances where loosely bedded roof and rib rock capable of immediate spawl must be supported. The major problem, of course, is to develop and place a sandfill possessing sufficient strength that will minimize vein-wall closure, rock stress orientation upon vein pillars and resultant rock burst situations. As mining depths increase, ground conditions will become increasingly aggravated.

Classified tailings, used as a mine sandfill, are a cohesionless material whose physical properties and support characteristics are well documented. Because the material normally placed in mines is limited to classified tailings, which are transported underground hydraulically, adequate support will probably never be achieved, especially if heavy ground conditions are encountered. Maximum relative density and accompanying material strengths potentially capable with sandfill cannot be attained until excess water is removed from the stope and the individual grains are at their maximum relative density. In other words, the void ratio of the particular sandfill must approach its densest condition whereby shear or movement of the grains will only occur with an increase in volume.

Even with treatment (physical or chemical), 100-percent maximum relative density will seldom be achieved. In many instance, 90- to 95 percent relative density will satisfy normal construction requirements. Field and laboratory experiments have shown that hydraulic sandfill is placed at a minimum relative density ranging from 65 to 90 pounds per cubic foot and water content exceeding 40 percent. After natural drainage, but prior to mining, densities will not normally exceed 30- to 40-percent relative density and the water content is yet well above 20 percent. Such sandfills will provide little more than a walking platform in this condition. With resumption of mining, additional water will escape from the sandfill and the density will increase. This sandfill alteration is predominately attributed to vein-wall closure. However, once closure begins, the concept of using sandfill as a competent artificial support becomes questionable.

A more effective sandfill can be placed through minor operational changes and proper maintenance. Most components of the hydraulic system adequately perform their designed function, but over a period of time, a few components become less reliable. Cyclone maintenance often occurs only when a breakdown forces such attention. Because of the lack of periodic maintenance, many cyclones are allowing a greater percentage of slimes (<200 mesh) to pass into the sandfill system. Sand lines are not properly installed; often, they are laid with minimal engineering. Sand lines should be rigidly alined and periodically inspected. In mines that experience heavy ground conditions, single-point discharge should be replaced with multispigoted discharge to prevent grain segregation and formation of impervious slime barriers. Adequate drainage systems should be installed to dewater the sandfill as efficiently as possible.

Improved sandfill operation and maintenance will upgrade the quality of the material placed in the stope. However, many investigators argue that even the best-placed sandfill will not control vein-wall closure and rock pressure buildup in deep vein mines. This position is not only realistic, but has been proven through conduct of field investigations. Sandfill placed at 50-percent relative density will not support vein walls in deep metal mines at depths exceeding 2,000 to 3,000 feet.

The next obvious approach was to develop and test methods of altering traditionally placed sandfill to improve its support characteristics. The majority of methods tested to modify sandfill and the results have been discussed. Vibratory compaction produced a denser sandfill and effectively dewatered the fill predominately through densification and decanting the water. The addition of cement increased both the unconfined compression and confined strength of sandfill. The richer the sand-cement ratio, the greater the strengths achieved. For example, at sand cement ratios of 40:1 and 5:1, the unconfined strength was 25 and 440 psi, respectively. Using the same test mixtures, the angle of internal friction as determined by the triaxial tests increased from approximately 36.7 to 46.8, thus increasing the overall shear strength of the richer cemented sandfill. Similar internal friction angles were achieved on compacted, uncemented sandfill of similar densities. One questions whether cemented sandfill is more effective than compacted, uncemented material. The addition of cement to sandfill alters the sand from a cohesionless material to a substance possessing cohesion. The cohesive property increase the effective shear strength of cemented sandfill and, until failure occurs, cemented sandfill will support greater stress or load. The following questions remain to be answered: Will mine or fill water percolating through the failure zone in the failed cemented sandfill rebond the material? What bonding strength can be anticipated?

Addition of modifiers such as dispersants, flocculants, fly ash, gypsum, and other substances influenced the unconfined compressive strength of cemented sandfill. Research results have shown the improved strength can be obtained at specific mixtures. The addition of increasing quantities of these modifiers will not necessarily improve the strength characteristics of the mixture. Investigations have shown that each modifier must be examined through a series of tests to arrive at the optimum mixture ratios and maximum strength. Increased strengths at shorter curing periods are possible using certain modifiers. This characteristic is an important asset where quick reentry into working areas such as sublevel or underhand stoping is desirable.

These laboratory experiments have documented the strength increases that can be expected through treatment or modification of sandfill with vibratory compaction, cement and/or additives. Field investigations have provided vein-wall closure and sandfill pressure data. Using all this information, theoretical analysis of deep-vein mines through use of finite element models was undertaken. The insertion of actual field data was incorporated into the analysis and served as a reference base. Additional analyses and comparison were made in which various sandfill properties were placed in the program. Improved sandfill properties will reduce vein-wall closure and rock and pillar stresses. Consequently, hazardous and costly rock falls and bursts should be decreased using the proper sandfill material.

An actual large-scale demonstration to prove that the proper sandfill can be placed and can be expected to function as a competent artificial support has not been undertaken. Verification or rejection of the potential capability of sandfill is yet unknown. Efforts are underway at the time of this writing to gather the necessary data and to develop a detailed research effort so that a demonstration program can be undertaken.

REFERENCES

1. Agarwal, V. M. The French Hydraulic Stowing Practice and Its Applicability in India. J. Min., Metal and Fuels, v. 11, No. 1, January 1963, pp. 1-7.

2. American Society for Testing and Materials. Standard Method of Test for Soil Properties. 1973 Annual Book of ASTM Standards: Part 2, Bituminous Materials; Soil and Rock; Peats, Mosses, and Humus; Skid Resistance. Philadephia, Pa., 1973, 1080 pp.

3. Ash, S. H., and J. Westfield. Backfilling Problems in the Anthracite Region as it Related to Conservation of Anthracite and Prevention of Subsidence. BuMines IC 7342, 1946, 18 pp.

4. Bardill, J. D., and D. L. Cenis. Studies of Several Flocculants To Improve Hydraulic Backfill Characteristics. BuMines RI 5597, 1960, 19 pp.

5. Bates, R. C., and W. R. Wayment. Laboratory Study of Factors Influencing Water Flow in Mine Backfill: Classified Mill Tailings. BuMines RI 7034, 1967, 45 pp.

6. Cenis, D. L. Some Aspects of the Design of Hydraulic Backfill Systems for Metal Mining. M.S. Thesis, Mont. School of Mines, Butte, Mont., May 1957, 182 pp.

7. Coates, D. F., and Y. S. Yu. Analysis of Grading Effects on Hydraulic and Consolidated Fill. Can Min. and Met. Bull., v. 62, No. 683, pp. 279-284.

8. Corson, D. R. Laboratory Study of Effect of Cement and Dispersant Additives on a Hydraulic Backfill. BuMines RI 6831, 1966, 15 pp.

9. _____. Stabilization of Hydraulic Backfill With Portland Cement: Including Results With Other Additives. BuMines RI 7327, 1970, 33 pp.

10. _____. Field Evaluation of Hydraulic Backfill Compaction at the Lucky Friday Mine, Mullan, Idaho. BuMines RI 7546, 1971, 24 pp.

11. Corson, D. R., and W. R. Wayment. Load Displacement Measurement in a Backfilled Stope of a Deep Vein Mine. BuMines RI 7038, 1967, 51 pp.

12. Espley, G. H., H. F. Beattie, and A. R. Pasieka. Cemented Hydraulic Backfill Within the Falconbridge Group of Companies. CIM Bull., v. 63, No. 701, September 1970, pp. 1002-1010.

13. Ghose, A. K., V. S. Vembar, and S. Bagchi. Hydraulic Stowing in Mines--Some Theoretical Design Consideration. J. Mines, Metals and Fuels, v. 9, No. 3, March 1961, pp. 1-6, 10.

14. Hill, J. R. M., M. M. McDonald, and L. M. McNay. Support Performance of Hydraulic Backfill: A Preliminary Analysis. BuMines RI 7850, 1974.

15. Jerabek, F. A., and H. L. Hartman. Investigations of Segregation and Compressibility in Discharged Fill Slurry. Trans. Soc. Min Eng., AIME, v. 232, No. 1, March 1965, pp. 18-24.

16. Kealy, C. D., and R. A. Busch. Determining Seepage Characteristics of Mill-Tailings Dams by the Finite-Element Method. BuMines RI 7477, 1971, 113 pp.

17. Kealy, C. D., and R. L. Soderberg. Design of Dams for Mill Tailings. BuMines IC 8410, 1969, 49 pp.

18. Lambe, T. W., and R. V. Whitman. Soil Mechanics. John Wiley & Sons, Inc., New York, 1969, 553 pp.

19. McCreedy, J., and R. J. Hall. Cemented Sandfill at INCO. Can. Inst. Min. and Met., Trans. Min. Soc., Nova Scotia, Canada, v. 69, 1966, pp. 281-285.

20. McNay, L. M., and J. R. M. Hill. Preliminary Laboratory Test Results; available upon request from L. M. McNay, Bureau of Mines, Spokane, Wash.

21. Nicholson, D. E., and R. A. Busch. Earth Pressure at Rest and One-Dimensional Compression in Mine Hydraulic Backfill. BuMines RI 7198, 1968, 40 pp.

22. Nicholson, D. E., and W. R. Wayment. Properties of Hydraulic Backfills and Preliminary Vibratory Compaction Tests. BuMines RI 6477, 1964, 31 pp.

23. _____. Vibratory Compaction of Mine Hydraulic Backfill. BuMines RI 6922, 1967, 52 pp.

24. Panek, L. A., and J. A. Stock. Development of a Rock Stress Monitoring Station Based on the Flat Slot Method of Measuring Existing Rock Stress. BuMines RI 6537, 1964, 61 pp.

25. Pariseau, W. G., M. M. McDonald, and J. R. M. Hill. Support Performance Prediction for Hydraulic Fills. Proc. of Symp. on Mine Filling, Mount Isa, Australia, August 1973, pp. 213-219; available for inspection at Bureau of Mines library, Spokane, Wash.

26. Parsons, E. W., and L. Osen. Explosive-Expansion Center-Hole Anchor. BuMines RI 6704, 1966, 14 pp.

27. Peele, R. Filled Stopes. Sec. 10 in Mining Engineers Handbook. John Wiley & Sons, Inc., New York, v. 1, 3d ed., 1952, pp. 237-274.

28. Rausch, D. O., and R. C. Stitzer. Underground Mining Systems and Equipment. Sec. 12 in SME Mining Engineering Handbook. Pt. 16--Filled Stopes and Combination Methods. American Institute of Mining, Metallurgical, and Petroleum Engineers, New York, 1973, pp. 233-253.

29. Sloane, H. N., and L. L. Sloane. A Pictorial History of American Mining. Crown Publishers, Inc., New York, 1970, 342 pp.

30. Sowers, G. B., and G. F. Sowers. Introductory Soil Mechanics and Foundations. MacMillan Co., New York, 3d ed., 1970, 556 pp.

31. Sprute, R. H., and D. J. Kelsh. Laboratory Experiments in Electrokinetic Densification of Mill Tailings. Part 2, Application to Various Types and Classifications of Tailings. BuMines RI 7900, 1974, 43 pp.

32. Stewart, R. M. Hydraulic Fill Slurries. Trans Can. Inst. Min. and Met., v. 53, No. 581, 1960, pp. 687-702; available for inspection at Bureau of Mines library, Spokane, Wash.

33. Thomas, E. G. The Important Properties of Hydraulic Fill, With Particular Reference to Mechanized Cut and Fill Mining Operations. Proc. Australasian Inst. Min. and Met., No. 220, December 1966, pp. 1-20; available upon request from AIMM, Clunies Ross House, 191 Royal Parade, Parkville, Victoria, Australia, 3052.

34. Waddell, G. G. In Situ Measurement of Rock Deformation in a Vein-Type Deep Mine. BuMines RI 6747, 1966, 47 pp.

35. Wayment, W. R., and D. E. Nicholson. A Proposed Modified Percolation Rate Test for Use in Physical Property Testing of Mine Backfill. BuMines RI 6562, 1964, 24 pp.

36. Weaver, W. S., and R. Luka. Laboratory Studies of Cement-Stabilized Mine Tailings. CIM Bull., v. 63, No. 701, September 1970, pp. 988-1001.

37. Zahary, G. Effect of Backfill on a Stope Wall. Pres. at 6th Can. Rock Mech. Symp., Ecole Polytechnique, Montreal, Canada, May 1970, pp. 119-129; available upon request from G. Zahary, Can. Dept. Energy, Mines and Res., Elliot Lake, Ontario, Canada, and from the Bureau of Mines library, Spokane, Wash.

38. Zahary, G., Zorychta, H., and Zaidi, S. Results--Fill Pressure Measurements in a Cut-and-Fill Stope. Can. Dept. Energy, Mine and Res.--Mines Branch unpublished report 72/119, 1972, 13 pp; available upon request from G. Zahary, Can. Dept. Energy, Mines and Res., Elliot Lake, Ontario, Canada.

SELECTED BIBLIOGRAPHY

1. Australasian Institute of Mining and Metallurgy. Proc. Jubilee Symposium on Mine Filling, Aug. 19-22, 1973, Mount Isa, Queensland, Australia, 282 pp.; available from the Secretary, North West Queensland Branch, The Australasian Mining and Metallurgy, c/o Mount Isa Mines, Ltd., Mount Isa, Queensland 4825, Australia.

2. Bardill, J. D., D. R. Corson, and W. R. Wayment. Factors Influencing the Design of Hydraulic Backfill Systems: 2, Friction-Head Losses of Barite and Limestone Slurries During Pipeline Transport. BuMines RI 6066, 1961, 33 pp.

3. Blake, W. Rock Burst Mechanics. Colo. School Mines Quart., v. 67, No. 1, January 1972, 64 pp.

4. Brock, A. F., and W. J. Taylor. Square-Set Stoping at the Frood Mine of the International Nickel Co. of Canada, Ltd. Ch. 3 in Mining in Canada: Pt. 7, Square-Set Stoping. 6th Commonwealth Min. and Met. Cong. Northern Miner Press Ltd., Tronto, Canada, 1957, pp. 181-186.

5. Burton, J. F. Cut-and-Fill Stoping at Eldorado, Beaverlodge. Ch. 3 in Mining in Canada: Pt. 6, Cut-and-Fill Mining. 6th Commonwealth Min. and Met. Cong., Northern Miner Press Ltd., Toronto, Canada, 1957, pp. 174-180.

6. Canada. Department of Energy, Mines and Resources--Mines Branch. Tentative Specifications--Test for Percolation Rate, or Coefficient of Permeability, of Fill. Tech. Bull. 101, April 1968, 9 pp.

7. Cenis, D. L. Some Aspects of the Design of Hydraulic Backfill Systems for Metal Mining. M. S. Thesis, Mont. School Mines, Butte, Mont., 1957, 182 pp.

8. Cleland, R. S., and K. H. Singh. Development of "Post" Pillar Mining at Falconbridge Nickel Mines Ltd. Bull. Can. Min. and Met., v. 66, No. 732, April 1973, pp. 57-64.

9. Coates, D. F. Probability of Failure in Pillar Design. Proc. 1973 Pacific Northwest Metals and Minerals Conf., Coeur d'Alene, Idaho, April 1973, 16 pp.; available for inspection at the Bureau of Mines library, Spokane, Wash.

10. Corson, D. R., and L. M. McNay. Field Evaluation of Backfill Behavior in Deep Vein Mines. Proc. Golden Jubilee Symp. Mine Fillings, Mount Isa, Australia, August 1973, pp. 205-212; available for inspection at the Bureau of Mines library, Spokane, Wash.

11. Crayson, E. G., L. H. Van Loon, and A. F. Heather. Sand Filling at Madsen Red Lake. Ch. 3 in Mining in Canada: Pt. 10, Stope Filling. 6th Commonwealth Min. and Met. Cong. Northern Miner Press Ltd., Toronto, Canada, 1957, pp. 250-271.

12. Crouch, S. L. Analysis of Rockbursts in Cut-and-Fill Stopes. Proc. 1973 Pacific Northwest Metals and Minerals Conf., Coeur d'Alene, Idaho, April 1973, 21 pp.; available for inspection at the Bureau of Mines library, Spokane, Wash.

13. Dempsey, R. W., and M. D. Kennedy. Cut-and-Fill Mining at Malartic Gold Fields, Limited. Ch. 3 in Mining in Canada: Pt. 6, Cut-and-Fill Mining. 6th Commonwealth Min. and Met. Cong. Northern Miner Press Ltd., Toronto, Canada, 1957, pp. 164-173.

14. Fowkes, R. S., and G. A. Wancheck. Materials Handling Research: Hydraulic Transportation of Coarse Solids. BuMines RI 7283, 1969, 36 pp.

15. Hall, R. J., and A. R. Clarke. Sand Filling at the Frood Mine of the International Nickel Co. of Canada, Ltd. Ch. 3 in Mining in Canada: Pt. 10, Stope Filling. Commonwealth Min. and Met. Cong. Printed by Northern Miner Press Ltd., Toronto, Canada, 1957, pp. 243-249.

16. Johnson, H. E. Square-Set Stoping at Normetal. Ch. 3 in Mining in Canada: Pt. 7, 6th Commonwealth Min. and Met. Cong. Northern Miner Press Ltd., Toronto, Canada, 1957, pp. 187-197.

17. Karelin, No. T. Some Problems of Hydraulic Transportation of Solid Materials Through Pipes. J. Mines, Metals and Fuels, v. 13, No. 6, June 1965, pp. 187-193.

18. Kis-Tamis, L. Comparative Study of Solid Stowing Methods. Colliery Guardian, v. 207, No. 5350, October 1963, pp. 547-552.

19. Krebs, R., and J. C. O'Donnell. Sand Slime Stope Filling Proves Satisfactory. Eng. Min. J., v. 150, No. 1, January 1949, pp. 54-60.

20. Lewis, R. S. Elements of Mining. John Wiley & Sons, Inc., New York, 2d ed., 1958, 579 pp.

21. Love, L. R., and S. F. Whirl. Duquesne Light Buries Its Fly-Ash Problem. Coal Age, v. 74, March 1969, pp. 90-94.

22. Martson, W. F. Sand Fill From Mill Tailings. Trans. Can. Inst. Min. and Met., v. 54, 1951, pp. 198-202.

23. Mathews, K. E. Excavation Design In Hard and Fractured Rock at the Mount Isa Mine, Australia. Pres. at 8th Can. Symp. Rock Mech., Nov. 30-Dec. 1, 1972, 20 pp.

24. McCreedy, J., and R. J. Hall. Cemented Sand Fill at the International Nickel Company. Pres. at Ann. Meeting of Can. Inst. of Min. and Met., Quebec City, Canada, Apr. 25-27, 1966, Trans., v. 69, pp. 281-285.

25. McCreedy, J., and W. J. Taylor. The Use of Hydraulic Fill Underground at the Mines of the International Nickel Co. of Canada, Ltd. Can. Min. J., v. 81, No. 9, September 1960, pp. 95-103.

26. McNickle, L. S., Jr. Shortcoming in Hydraulic Designs Derail Good Maintenance Practices. Coal Age, v. 77, No. 11, November 1972, pp. 84-87.

27. Metal Mining and Processing. Sand-Fill Spurs Production at the Christmas Mine. V. 2, No. 1, January 1965, pp. 23-25.

28. Michle, D. G., and H. L. Hartman. Permeability and Compressibility Tests Aid in Selecting Suitable Hydraulic Fill Materials. J. Min. Eng., v. 14, No. 11, October 1961, pp. 1246-1248.

29. Mining World. Pumping Hydraulic at the Lucky Friday Mine. V. 23, May 1961, pp. 22-23.

30. Muller, H. B., H. J. Rich, and A. B. Grant. The Preparation, Storage, and Placing of Hydraulic Fill, C. S. A. Mine, Cobar. Proc. Australasian Inst. Min. and Met., No. 229, March 1969, pp. 27-37.

31. Murphy, E. M., M. O. Magnuson, P. Suder, Jr., and J. Nagy. Use of Fly Ash for Remote Filling of Underground Cavities and Passageways. BuMines RI 7214, 1968, 27 pp.

32. O'Brien, M. P., and R. G. Folsum. The Transportation of Sand in Pipelines. Eng., v. 3, No. 7, 1937, p.

33. Pariseau, W. G., and C. D. Kealy. Support Potential of Hydraulic Backfill. New Horizons in Rock Mechanics, ed. by H. R. Hardy, Jr., and R. Stefanko. American Society of Civil Engineers, New York, 1972, pp. 501-526.

34. Park, M. A. Automated Sand Fill System. Min. Cong. J., v. 48, No. 10, October 1962, pp. 24-26.

35. Parry, K. F. Hydraulic Stope Filling on the Golden Mile. Proc. Australasian Inst. Min. and Met., v. 211, September 1964, pp. 193-215; available for inspection at the Bureau of Mines library in Spokane, Wash.

36. Patton, F. E. Backfilling at Noranda. Ch. 3 in Mining in Canada: Pt. 10, Stope Filling. 6th Commonwealth Min. and Met. Cong. Northern Miner Press Ltd., Toronto, Canada, 1957, pp. 229-236.

37. Rice, G. R. Hydraulic Sand Filling of Stopes in the Star Mine. Pres. at CIM British Columbia Section Meeting, Trail, British Columbia, Canada, October 1959, 11 pp.; available for inspection at the Bureau of Mines library, Spokane, Wash.

38. Rice, H. R. Introduction to Mining Methods In Canadian Mines. Ch. 3 in Mining in Canada: Mining Methods. 6th Commonwealth Min. and Met. Cong. Northern Miner Press Ltd., Toronto, Canada, 1957, pp. 43-46.

39. Romaniuk, A. S. Backfill for Underground Mining Operations. Part 1, Backfill Techniques Used Outside Canada. Can. Dept. Energy, Mines and Res., Div. Rept. FMP 67/2-MRL, 1967.

40. Rosser, R. P., and R. R. Khanna. Hydraulic Stowing in India. Trans. Min. Geol. and Met. Inst. of India, v. 59, No. 1, April 1962, pp 21-66; available for inspection at the Bureau of Mines library in Spokane, Wash., and the Central Library, U. S. Department of the Interior, Washington, D. C.

41. Sinha, K. N. Hydraulic Stowing in India--Scientific Tests, Investigations and Observations. Trans Min. Geol. and Met. Inst. of India, v. 56, September 1959, pp. 77-124; available for inspection at the Bureau of Mines library in Spokane, Wash., and the Central Library, U. S. Department of the Interior, Washington, D. C.

42. Sixth Commonwealth Mining and Metallurgical Congress. Horizontal Cut-and-Fill Stoping at Hollinger Consolidated Gold Mines, Ltd. Ch. 3 in Mining in Canada: Part 6, Cut-and-Fill Mining. Northern Miner Press, Ltd., Toronto, Canada, 1957, pp. 157-163.

43. Sixth Commonwealth Mining and Metallurgical Congress. Stope Filling at the Sullivan Mine of the Consolidated Mining and Smelting Co. of Canada, Ltd. Ch. 3 in Mining in Canada: Part 10, Stope Filling. Northern Miner Press Ltd., Toronto, Canada, 1957, pp. 237-242.

44. Sprague. J. W. A Method of Measuring the Costs and Benefits of Applied Research. BuMines IC 8414, 1969, 42 pp.

45. Sprute, R. H., and D. J. Kelsh. Laboratory Experiments in Electrokinetic Densification of Mill Tailings. Part 1, Development of Equipment and Procedures. BuMines RI 7892, 72 pp.

46. Suttie, J. Application of Hydraulic Fill. Min. Cong. J., v. 46, No. 4, April 1960, pp. 48-50.

47. Turnbull, W. J., and C. I. Mansur. Compaction of Hydraulically Placed Fills. J. Soil Mech. and Foundations Div. Proc. Am. Soc. Civil Eng., v. 99, No. SM 11, November 1973, pp. 939-955.

48. Twidale, M. A. Backfill Methods in Canadian Mines. Dept. Mines and Tech. Surveys, Mines Branch, IC 141, Fuels and Min. Proc. Div., Ottawa, Canada, August 1962, 21 pp.

49. U. S. Bureau of Reclamation. Design of Small Dams. 1965, 611 pp.

50. _____. Earth Manual. 1968, 783 pp.

51. Wayment, W. R., and D. E. Nicholson. Improving Effectiveness of Backfill. Min. Cong. J., v. 51, No. 8, August 1965, pp. 28-32.

52. Wayment, W. R., G. L. Wilhelm, and J. D. Bardill. Factors Influencing the Design of Hydraulic Backfill Systems. 1. Friction-Head Losses of Sand Slurries During Pipeline Transport. BuMines RI 6065, 1961, 43 pp.

53. Wilhelm, G. L. Hecla's Mayflower Mine Used Hydraulic Sandfill in Stoping Operations. J. Min. Eng., v. 20, No. 5, May 1968, pp. 80-82.

U.S. GOVERNMENT PRINTING OFFICE : 1975 O—565-423

B254
1