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Ground-Water Problems in Open-Pit and Underground Mines

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ABSTRACT

Ground water constitutes a significant operational problem and potential hazard in open-pit and underground mining. The *pressure* of ground water in geologic discontinuities adversely affects the safety of height and slope angles of open-pit mines, justifying measures to reduce ground-water pressures in their vicinity. The principal hazard in underground mining and tunneling arises from *massive* inflows of ground water when the mine unexpectedly intersects large, water-bearing geologic discontinuities; *normal* inflows are on the order of 2.5 tons of water per ton of rock mined. In tunneling, pilot drilling ahead of the advancing face is used to detect such discontinuities, which can then be sealed by ring drilling and cover grouting. Extensive deformations resulting from large areas of unsupported roof make such grouting ineffective in many mine excavations.

Surprisingly few data are available with which to calculate the permeability of rock masses at depth. Two sets of data quoted yielded permeabilities of 0.8×10^{-9} m/s for tunnels, and 1.6×10^{-6} m/s for mined excavations, at a depth of about 1.5 km below surface and before the permeability of the rock mass is enhanced by deformations induced by mining. The difference between these values may be attributed to local reductions in the permeability of the rock mass by the grouting of major discontinuities in tunneling, which is not done ordinarily in mining.

Where precautions such as regional dewatering, installation of watertight bulkheads, and adequate spare pumping capacity are not sufficient to cope with potential inflows in mining, the extent of the unsupported roof in mines must be broken by regularly spaced pillars, so that pilot drilling and cover grouting can be done.

INTRODUCTION

Ground water may constitute a significant operational difficulty and potential hazard in open-pit and underground mining and in tunneling. Problems related to ground water range from preventing flooding in shafts sunk through water-bearing rocks, to environmental pollution after mining. Shaft sinking, on the one hand, is a highly specialized activity in mining and tunneling, and methods for coping with ground water by cement grouting, freezing, and other techniques have been documented well (Atherton and Garrett, 1959; Marsh, 1959; Weehuisen, 1959). On the other hand, environmental pollution of ground water by mining activities has been the subject of recent debate and legislation (Federal Water Pollution Control Act, 1972). Accordingly, neither of these two topics is discussed in this paper. Rather, the purpose of this paper is to examine the nature and magnitude of operational and hazardous aspects of ground water in open-pit and underground mining and tunneling. Of the few published sources of data sufficient to enable calculation of the permeability of a rock mass at depths of more than 1 km, some are presented. These data are used to estimate values of permeability when the major water-bearing discontinuities are sealed by grouting and when they are not. The effects of deformation, caused by mining, on the permeability of the rock mass are shown by reference to the rate of water inflow.

OPEN-PIT MINES

Although the presence of ground water does cause significant operational problems in open-pit

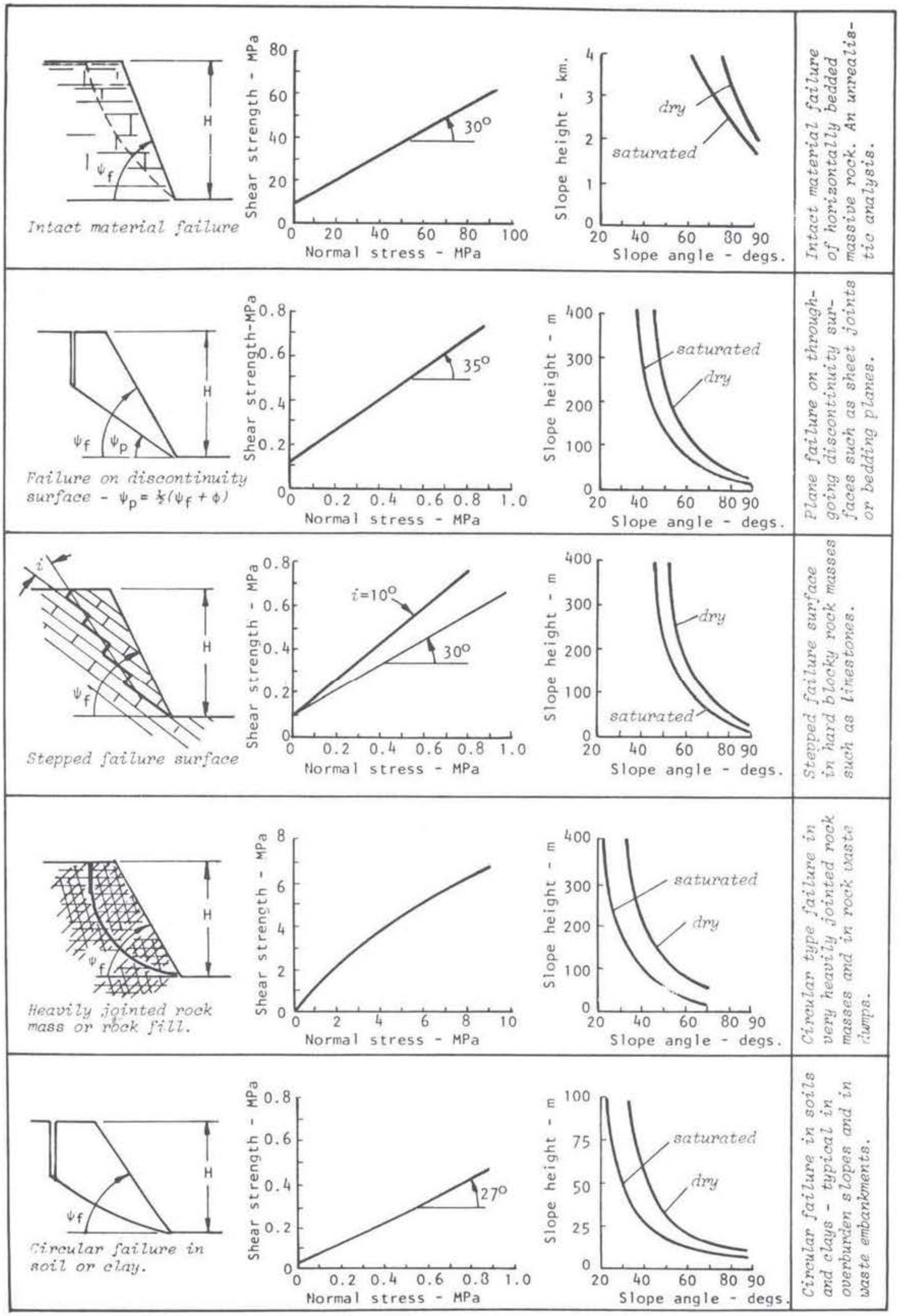

Figure 1. Diagrams illustrating five modes of potential slope failure for materials with different properties showing the adverse effects of ground water (saturation) on slope angle and slope height. (After Hoek and Bray, 1977.)

mines (such as the need to use waterproof blasting agents and accessories in wet blast holes), the major concern is with the effects of ground-water pressures rather than quantities on the stability of pit slopes. These effects have been studied extensively, and a comprehensive appraisal of them has been made by Hoek and Bray (1977).

The stability of open-pit slopes is maintained by the cohesion of the slope material and its frictional resistance to sliding. Several forms of slope failure can occur, depending upon the nature of the slope material and its geologic structure. Hydraulic pressure from ground water tends to promote all the different forms of slope failure. This happens by two separate mechanisms. First, the pressure of ground water reduces the value of the effective stress in the slope material, thereby reducing the frictional resistance to sliding. Second, ground-water pressure in back of a slope may generate a significant force toward the open pit, tending to promote sliding. It is important to understand that both mechanisms are results of ground-water *pressure* rather than *quantity*. Ground-water pressure can have a devastating effect on the stability of any open-pit slope.

One of the most meaningful ways to demonstrate stability of open-pit slopes is by showing the graphic relationship between slope height and slope angle. Such relationships are sketched in Figure 1 for five principal forms of slope failure, for both dry and saturated conditions. These forms of failure fall into two categories: plane failure and circular failure. Both have been analyzed in detail by Hoek and Bray (1977). Circular failure is the classic form of failure in soil mechanics (Terzaghi, 1943; Taylor, 1948; Bishop, 1955). Wedge failures can be regarded as a more complicated, three-dimensional plane failure (Londe, 1965; Goodman and Taylor, 1967). A different mode of slope failure, *toppling*, has been recognized recently (Muller, 1968; Goodman and Bray, 1976, Hoek and Bray, 1977). However, toppling is not yet understood as well as other forms of slope failure, and the effect of ground water on toppling has not been taken into account.

The effects of ground-water pressure on the stability of open-pit slopes have been excellently illustrated by Hoek and Bray (1917) and are reproduced here in Figure 1. For each of the five principal forms of slope failure, the maximum height of the slope for any given slope angle decreases drastically from the dry to the saturated condition. Alternatively, for any slope height a significant reduction in slope angle is necessary to maintain stability under saturated conditions compared with dry conditions. There are great economic advantages to maintaining the steepest slope angle possible within the limits of safety, as this reduces the amount of waste rock that must be mined to extract the ore. These advantages justify substantial efforts to achieve and maintain dry rather than saturated conditions in the mine (Moffit and others, 1971; Stewart and Kennedy, 1970). This involves careful hydrogeological studies and engineered remedies. These remedies range from simple procedures, such as installing surface drainage to divert precipitation from the proximity of the open pit, to drilling wells and developing drifts adjacent to the slopes in order to reduce ground-water pressures in their vicinity (Hoek and Bray, 1977; Durston, 1979; Lopaschuk, 1979; Pentz, 1979). The measurement and analysis of ground-water flow in the vicinity of open-pit slopes, the evaluation of the effects of ground water on slope stability, and the drainage of slopes are described fully in the Pit Slope Manual (Canada Center for Mineral and Energy Technology, 1977).

TUNNELS AND UNDERGROUND MINES

Although some underground mines and tunnels are excavated in rock strata with relatively homogeneous porosity and permeability, many are not. The continuity of most rock is interrupted in some degree by bedding planes, joints, fractures, and faults. In massive and stratiform rocks with low hydraulic permeabilities, these discontinuities provide the principal conduits for the movement of ground water.

In rock with homogeneous but not necessarily isotropic hydraulic properties, the effects of ground water on tunneling and mining can be predicted with comparative ease and certainty. Unfortunately, this situation is the exception rather than the rule; most mines and tunnels are excavated in rocks where the dominant permeability arises from discontinuities. For example, Trexler (1979) reported the primary source of ground water in an old, deep mine as being natural fractures intersected by drillholes and drifts; of about 2,340 holes and drifts, only 115 holes and 2 drifts had significant inflows (26 to 33 l/s). Significant inflows of ground water into the workings of mines and tunnels are common. In general, it is necessary to install pumps with capacities large enough to remove this water. For example, in 1958, Charbonnage de France pumped 2.4 tons of water for each ton of coal mined; in England and Wales in 1962, 2.44 tons of water were pumped for each ton of coal mined (Orchard, 1969). Gold mines of the Witwatersrand System pump about 1 ton of water per ton of rock mined at a weighted average depth of 1.5 km (Whillier, 1978). Unfortunately, other measurements, which would be needed to infer information such as the permeability of the rock mass, are not complete, so that the above data are not sufficient for such calculations.

There is even less information about catastrophic inflows than there is about normal inflow, so that few, if any, meaningful calculations concerning this hazard can be made. Major disasters or potential disasters from sudden, unexpected inflows or inrushes of water are fortunately rare, but have occurred throughout the history of mining. In July 1847, the sea broke into Workington Colliery in Great Britain, causing the loss of 27 lives. This coal mine was being worked with headings from 3.7 m (12 ft) wide to 4.6 m (15 ft) wide and pillars 13.7 m (45 ft) by 9.1 m (30 ft) wide at depths from 27.4 m (90 ft) to 45.7 m (150 ft) beneath the sea floor (Orchard, 1969). In October 1968, the world's richest gold mine, West Driefortein, on the Witwatersrand System, South Africa, experienced an inrush of 3,700 l/s that took 26 days to bring under control, and which put the mine out of full production until August 1969 (Tress, 1974). The information pertaining to the source of this inflow is a hurriedly written note which reads as follows: "Stope 46B on 6 level west of 4 shaft has broken open and water is running 6 inches deep into the shaft" (Cartwright, 1970). Cummins and Given (1973) reference eight cases of disastrous mine flooding from sudden, unexpected inflows of water.

New gold mines in the Orange Free State portion of the Witwatersrand System were opened during the 1950s. Because of the significant costs of pumping large quantities of water from underground mines, the new mines were used to make measurements of the rates of ground-water inflow as a function of primary mine development by tunneling and subsequent mining of the stratiform gold reef (Venter, 1969). These data are virtually unique in that sufficient information concerning the rate of inflow, the length and size of tunnels developed, the area mined, and the height of the water table was obtained to enable simple calculations to be done on the permeability of the stratiform rock mass within which these excavations were being made.

The flow of ground water through geologic discontinuities has been compared with the laminar flow of fluid between closely spaced parallel plates (Huitt, 1956; Snow, 1968). The volume rate of flow through such a discontinuity can readily be shown as

$$q = \frac{g(2b)^3 i}{12\eta}, \qquad (1)$$

where q = the volume rate of flow per unit length of the discontinuity in a direction normal to that of the flow ($m^3/m/s$);
$2b$ = the aperture of the discontinuity (m);
g = the gravitational acceleration (9.81 m/s^2);
i = the hydraulic gradient (m/m); and
η = the kinematic viscosity of water (m^2/s).

A similar expression for the volume rate of flow through such a discontinuity into a circular hole from the outside of a concentric cylinder (Narasimhan and Witherspoon, 1976) is:

$$Q = \frac{2\Pi(2b)^3 g}{12\eta(\ln\ r/r_h)}\ (H-H_h), \qquad (2)$$

where Q = the volume rate of flow through the discontinuity into the hole (m^3/s);
r = the radius to the cylinder (m);
r_h = the radius of the hole (m);
H = the hydraulic head outside the cylinder (m);
H_h = the hydraulic head in the hole at the discontinuity (m); and the other symbols are as defined above.

The volume rate of flow into a horizontal, subsurface tunnel can be found using the theory of images (Ferris and others, 1962) from an equation similar to equation 2, namely:

$$Q = \frac{2\Pi(2b)^3 g}{12\eta(\ln\ 2r/r_t)}\ H, \qquad (3)$$

where 2θ = twice the distance to the water table, that is, the separation between the tunnel and its image (n);
r_t = the radius of the tunnel (m);
H = the hydraulic head of the water table (m); and the other symbols are as defined above.

These equations demonstrate the overriding effect of the aperture of the discontinuity on the rate of ground-water flow. The volume rates of flow through discontinuities with various spacings are shown as a function of aperture in Figure 2.

Figure 3 illustrates the cumulative distribution of exposed fractures in granite as a function of aperture at Pikes Peak, Colorado (Snow, 1970). This shows that the hydraulically dominant fractures with large apertures are much less frequent than those with small apertures. Accordingly, the flow of large quantities of ground water into underground excavations is likely to be a capricious phenomenon, resulting from the infrequent intersection of discontinuities with large apertures. Ground-water inflows during tunneling have been studied with the use of physical and numerical models by Goodman and others (1965), but these studies assume a knowledge of hydraulic permeability or transmissivity along the tunnel axis. Even the most careful site investigations before excavation commonly fail to detect the presence of some of the most hazardous hydrogeologic features and certainly do not quantify their properties (Olivier, 1970). Snow (1968) has shown

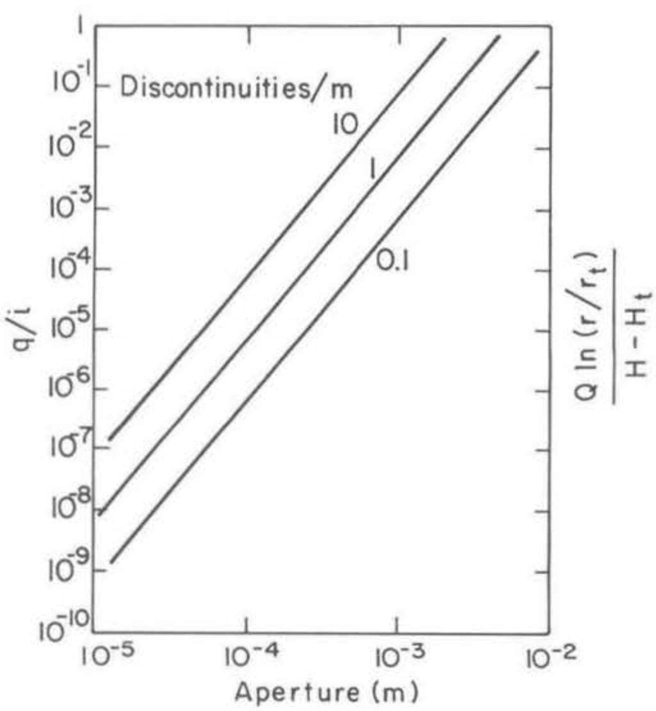

Figure 2. Volume rates of linear and radial flow through discontinuities as a function of their aperture for three different discontinuity spacings.

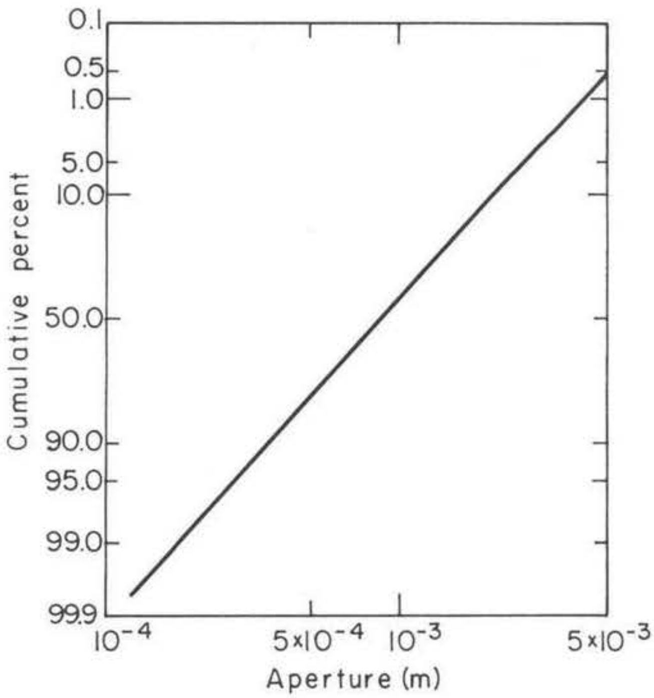

Figure 3. Cumulative distribution of exposed fractures with various apertures in granite from Pikes Peak, Colorado. (After Snow, 1970.)

that the aperture of fractures decreases with increasing depth (compressive stress), as shown in Figure 4. The extent to which the excavation of underground openings reduces the compressive stress across discontinuities is therefore also likely to affect their aperture and hydraulic transmissivity.

Horizontal and vertical tensile stresses are induced in the roofs and floors of underground openings as a result of changes in the distribution of stresses in the rock adjacent to them. These changes can be calculated by analytical and numerical methods (Obert and Duvall, 1967; Jaeger and Cook, 1976). Figure 5 shows the height above a tubular underground opening at which the total vertical stress reverts from tension to compression, as a function of the ratio between the depth and minimum horizontal dimensions of the unsupported roof. The extent of total horizontal tension is generally less than that of total vertical tension.

Figure 6 shows the rate of pumping from a new gold mine in the Witwatersrand System as a function of the cumulative length of the tunnels and cumulative area mined (Venter, 1969). Note that the rate of pumping, or water inflow, increased initially as a linear function of the cumulative length of the tunnels, as would be expected from equation 3. Soon after mining began on the stratiform gold reef, the rate of inflow accelerated rapidly. If the rate of inflow is examined as a function of the cumulative area mined, we see that initially the rate of inflow increased linearly with the area mined and then at a much greater rate. A similar pattern for an adjacent mine is shown in Figure 7. The ratio of the depth to the minimum horizontal dimension of the area mined at which the

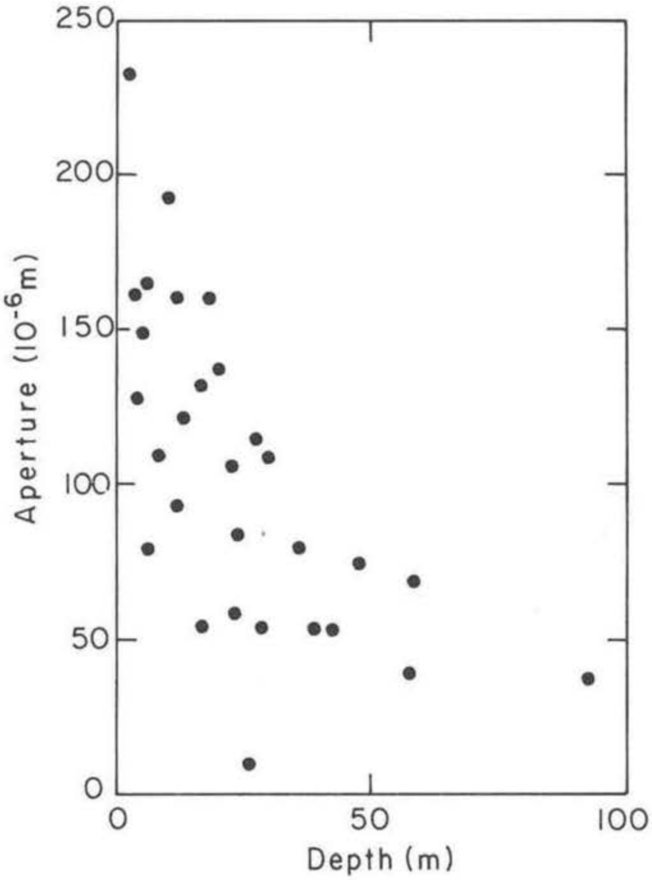

Figure 4. Changes in the apertures of fractures as a function of depth. (After Snow, 1968.)

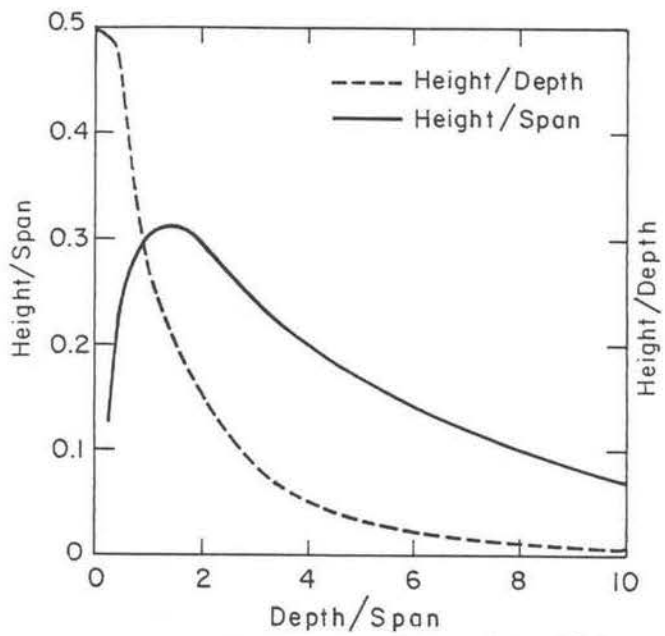

Figure 5. Height above a mine roof to which the zone of vertical tension extends as a function of the ratio between the depth of the roof below surface and its unsupported span or minimum horizontal dimension.

inflow accelerated, D/S, is about 8 in the first example and about 5 in the second example.

Figure 5 shows that at values similar to these, the height at which vertical tension is induced begins to increase rapidly. Presumably, the sudden increase in inflow occurs when the induced vertical tension extends to the height of water-bearing discontinuities in rocks with little permeability other than that caused by discontinuities. Both mines are located in the Orange Free State gold field of the Witwatersrand System. This system comprises alternating groups of argillaceous and arenaceous sediments with an age of about 2.5 b.y. In the Orange Free State these sediments are overlain by lava of the Ventesdorp System and by impermeable sandstones, mudstones, and shales of the Karoo System (Haughton, 1969).

Using equations 1 and 3 and the linear portions of the data plotted in Figure 6, the permeability can be estimated. This gives a value of 0.8 x 10^{-9} m/s from the inflow to the tunnels and 1.6 x 10^{-6} m/s for the initial inflow into the mined area. This disparity may be accounted for by the practice of pilot drilling and cover grouting during excavation of shafts and tunnels. For safety, pilot boreholes are drilled ahead of the advancing end of the tunnel to detect discontinuities that carry large volumes of water. If necessary, such discontinuities are sealed by the injection of cement grout through rings of cover holes before tunneling is continued. Injection grouting has been found to have little effect in the case of mining, because the extensive changes in stress and deformation associated with large mine excavations destroy the effect of grouting. Thus, the largest discontinuities that account for the greatest inflow of water are sealed as tunneling proceeds but not as mining proceeds. Where it is necessary to prevent large inflows through the disturbance of discontinuities as a result of mining, the extent of the exposed mine roof must be

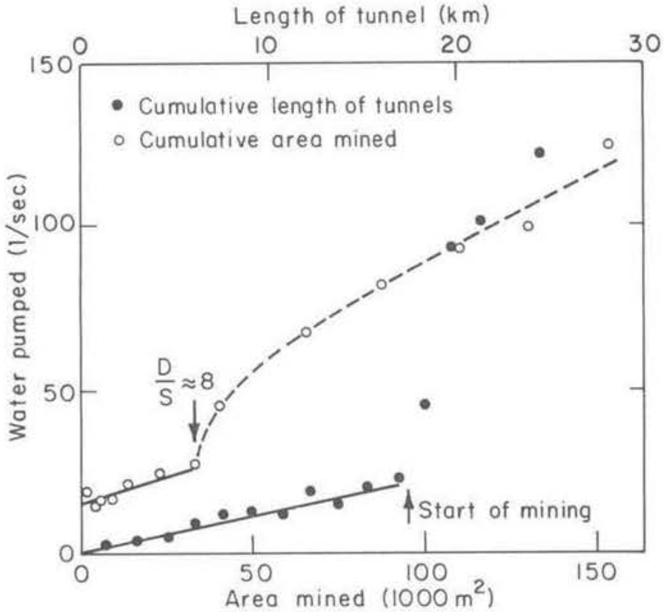

Figure 6. The rate of pumping from a new mine as a function of the cumulative length of tunnels developed and the cumulative area mined. (After Venter, 1969)

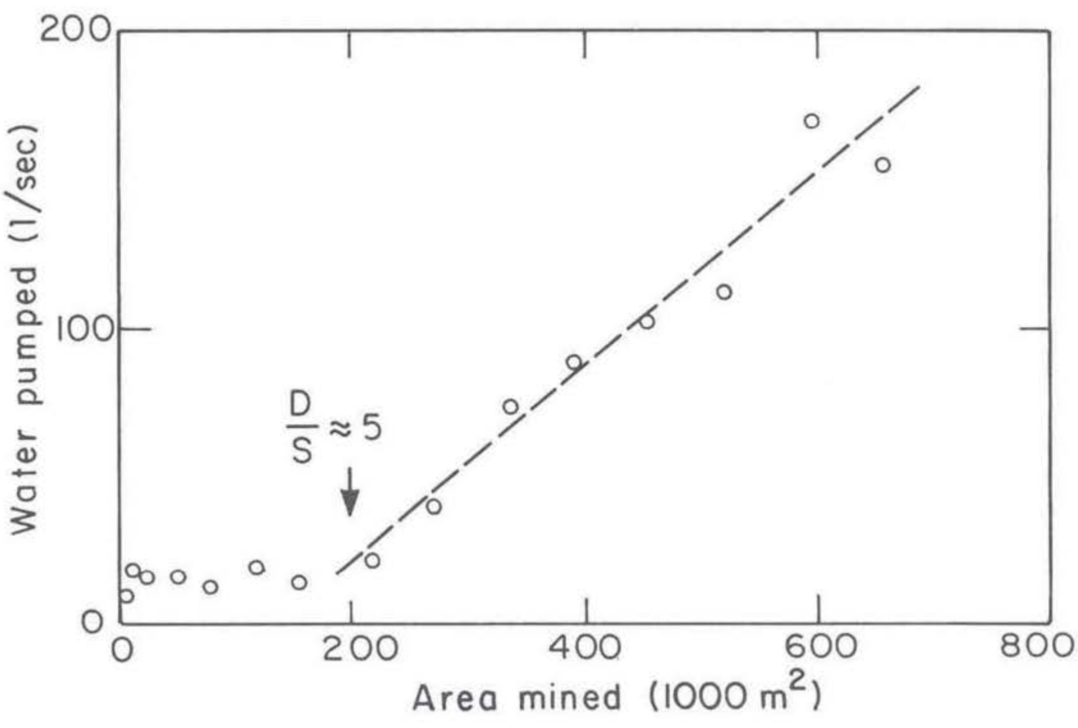

Figure 7. Rate of pumping from a new mine as a function of the cumulative area mined. (After Venter, 1969.)

limited by leaving regular and adequate pillars, as is done in room and pillar coal mining. Such procedures, though they sterilize the ore in the pillars, may be necessary when mining beneath large volumes of ground water or beneath oceans and lakes, where dewatering is impractical. Also, pilot drilling and cover grouting can then be used to reveal and seal major discontinuities.

MODELING

The effects of ground water in open-pit and underground mines and tunnels are twofold. First, there are those resulting from ground-water pressure on slope stability, particularly open-pit slopes. Second, there are those effects related to the quantity of ground-water inflow, which may be manageable or hazardous.

A number of numerical models exist with which to simulate the flow of ground water into mines, although they have not been used widely for this purpose. In general, there are too few field data available to make meaningful use of these models. For example, Brown (1979) stated that simple hydrological analyses are usually most appropriate for mining problems, because the information available is limited and the details of complex analyses are difficult to evaluate in terms of mining practice. Ignoring the unexpected consequences of an undetected, large, water-bearing discontinuity and the effects of major deformations of unsupported mine roofs, these models can be used to evaluate ground-water inflows to underground workings and ground-water pressure distributions in the vicinity of open-pit slopes. For example, in deep mines under saturated conditions, the finite-difference models of Pinder and Bredehoeft (1968) and Prickett and Lonnquist (1971) or the finite-element models of Gupta and Tanji (1966) and Javandel and Witherspoon (1968) can be used. For shallow or open-pit mines, free-surface models such as those of Jeppson (1968) and Neuman and Witherspoon (1970) can be used. If information is available concerning variable saturation, models such as those developed by Freeze (1971), Neuman (1973), and Narasimhan and Witherspoon (1977) may be more applicable.

CONCLUSION

Ground water constitutes one of the principal operational difficulties and potential hazards in mining and tunneling. Several numerical models have been developed to simulate ground-water flow and pressures. These models can be applied to the study of ground-water problems in open-pit and underground mines to a limited extent only. In general, meaningful simulation of practical problems with the use of these models requires more field data than usually exist. In particular, the hazard of major water-bearing discontinuities arises because they are difficult to detect, and the effects on the permeability of major deformations of unsupported mine roofs are not understood.

In open-pit mines, the pressure of ground water in discontinuities has a major deleterious effect on the stability of slopes. Measures to depressurize this ground water to maintain slopes at greater angles than would otherwise be safe are often economically very rewarding.

In tunnels and underground mines unexpected inflows constitute the major hazard. Few mines and tunnels are completely dry. The ventilation air removes about 10 g of water per cubic metre of air; rates of airflow in tunnels are typically 10 m^3/s and in mines, 500 m^3/s. Most underground mines have pumps to handle the ordinary inflow of ground water, which is normally in the range of 100 to 2500 l/ton of ore mined. Flooding commonly occurs when an unanticipated major water-bearing discontinuity is exposed. In tunneling and in mine-tunnel development, pilot drilling ahead of the advancing tunnel face, with appropriate precautions, is used to guard against flooding. If a major discontinuity is located it can be sealed by cover drilling and cement grouting. Such discontinuities have proved very difficult to locate by means other than pilot drilling. A single discontinuity can result in an inflow of about 1,000 l/s. Grouting has not proved to be effective in sealing such discontinuities in the vicinity of excavations with extensive unsupported roofs made in mining the ore; thus the hazard of flooding is present wherever mining is done below the ground-water table. Precautions against such flooding include pilot drilling, regional dewatering, the provision of water-tight bulkheads that can be used to seal portions of a mine, and adequate spare pumping capacity. Where such precautions are inadequate, the extent of the unsupported mine roof must be limited by regularly spaced pillars, so that pilot drilling and cover grouting can be used to detect and seal major discontinuities, as in tunneling.

REFERENCES CITED

Atherton, F. G., and Garrett, W. S., 1959, The history of cementation in shaft sinking, in Symposium on Shaft Sinking and Tunnelling: London, The Institution of Mining Engineers.

Bishop, A. W., 1955, The use of the slip circle in the stability analysis of earth slopes: Geotechnique, v. 5, p. 7-17.

Brown, A., 1979, Drainage excavation and control for underground oil shale projects, in Proceedings, First International Mine Drainage Symposium: San Francisco, Miller-Freeman Publishing Co.

Canada Centre for Mineral and Energy Technology, 1977, Pit slope manual: Ottawa, Printing and Publishing, Supply and Services.

Cartwright, A. P., 1970, West Driefontein--Ordeal by water, in Goldfields of South Africa Limited: Johannesburg, J. G. Ince & Son (Pty.).

Cummins, A. B., and Given, I. A., 1973, SME mining engineering handbook: New York, American Institute of Mining, Metallurgical, and Petroleum Engineers.

Durston, K. J., 1979, Open pit dewatering at the Pine Point operations of Cominco Limited, in Proceedings, First International Mine Drainage Symposium: San Francisco, Miller-Freeman Publishing Co.

Ferris, J. G., and others, 1962, Theory of aquifer tests, ground water hydraulics: U.S. Geological Survey Water-Supply Paper 1536-E.

Freeze, R. A., 1971, Three dimensional transient saturated-unsaturated flow in a groundwater basin: Water Resources Research, v. 7, p. 347.

Goodman, R. E., and Bray, J. W., 1976, Toppling of rock slopes, in Proceedings, Specialty Conference on Rock Engineering for Foundations and Slopes: New York, American Society of Civil Engineers, p. 201-234.

Goodman, R. E., and Taylor, R. L., 1967, Methods of analysis of rock slopes and abutments: A review of recent developments, in Fairhurst, C., ed., Failure of breakage of rocks: American Institute of Mining Engineers Transactions, p. 303-320.

Goodman, R. E., and others, 1965, Ground water inflows during tunnel driving, Engineering Geology, v. 2, p. 39-56.

Gupta, S. K., and Tanji, K. K., 1976, A three dimensional Galerkins finite element solution flow through multi aquifers in Sutter Basin, California: Water Resources Research, v. 7, p. 347.

Haughton, S. H., 1969, Geological history of southern Africa: Johannesburg, Geological Society of South Africa.

Hoek, E., and Bray, J., 1977, Rock slope engineering: London, Institution of Mining and Metallurgy.

Huitt, J. L., 1956, Fluid flow in simulated fractures: American Institution of Chemical Engineering Journal, v. 2, p. 259-264.

Jaeger, J. C., and Cook, N.G.W., 1976, Fundamentals of rock mechanics: New York, John Wiley and Sons.

Javandel, I., and Witherspoon, P. A., 1968, Application of the finite-element method for transient flow in porous media: Society of Petroleum Engineers Journal, v. 8, p. 241.

Jeppson, R. W., 1968, Seepage from channels through layered porous media: Water Resources Research, v. 4, p. 435.

Londe, P., 1965, Une méthode d'analyse à trois dimensions de la stabilité d'une rive rocheuse: Paris, Annales des Ponts et Chaussées, p. 37-60.

Lopaschuk, R., 1979, Experience with drainage control at Cyprus Anvil Mining Company, Yukin, in Proceedings, First International Mine Drainage Symposium: San Francisco, Miller-Freeman Publishing Co..

Marsh, F., 1959, Shaft sinking in Great Britain since 1947, in Symposium on Shaft Sinking and Tunnelling: London Institution of Mining Engineers.

Moffit, R. B., Friese-Greene, T. W., and Lillico, T. M., 1971, Pit slopes--Their influence on the design and economics of open pit mines, in Proceedings, Second Symposium on Stability in Open Pit Mining: New York, American Institute of Mining Engineers.

Muller, L., 1968, New considerations of the Vajont Slide, Felsmechanik und Engenieurgeologie, v. 6, no. 1, p. 1-91.

Narasimhan T. N., and Witherspoon, P. A., 1976, An integrated finite-difference method for analyzing fluid flow in porous media: Water Resources Research, v. 12, p. 57-63.

___1977, Numerical model for saturated-unsaturated flow in deformable porous media, part I, theory: Water Resources Research, v. 13, p. 657-664.

Neuman, S. P., 1973, Saturated-unsaturated seepage by finite elements: American Society of Civil Engineers Proceedings, Hydraulic Division Journal (HY12), p. 99.

Neuman, S. P., and Witherspoon, P. A., 1970, Finite-element method of analyzing steady flow into a free surface: Water Resources Research, v. 6, p. 889.

Obert, L., and Duvall, W. I., 1967, Rock mechanics and the design of structures in rock: New York, John Wiley and Sons.

Olivier, H. J., 1970, Geological-engineering exploration for the Orange-Fish Tunnel, in The Technology and Potential of Tunnelling (Proceedings, The South African Tunnelling Conference): Johannesburg, Associated Scientific and Technical Societies of South Africa.

Orchard, R. J., 1969, The control of ground movements in undersea working: The Mining Engineer, v. 128, p. 259-268.

Pentz, D. L., 1979, Case examples of open pit mine drainage, in Proceedings, First International Mine Drainage Symposium: San Francisco, Miller-Freeman Publishing Co.

Pinder, G. F., and Bredehoeft, J. D., 1968, Application of the digital computer for aquifer evaluation: Water Resources Research, v. 4, p. 1069.

Prickett, T. A., and Lonnquist, C. G., 1971, Selected digital computer techniques for ground water resource evaluation: Illinois State Water Survey Bulletin 55.

Snow, D. T., 1968, Rock fracture spacings, openings and porosities: American Society of Civil Engineers Proceedings, Soil Mechanics and Foundations Division Journal, SM 1, p. 73-91.

___1970, The frequency and apertures of fractures in rock: Journal of Rock Mechanics and Mining Science, v. 7, p. 23-40.

Stewart, R. M., and Kennedy, B. A., 1970, The role of slope stability in the economics, design and operation of open pit mining, in First Symposium on Stability in Open Pit Mining: New York, American Institute of Mining Engineers.

Taylor, D. W.,1948, Fundamentals of soil mechanics: New York, John Wiley.

Terzaghi, K., 1943, Theoretical soil mechanics: New York, John Wiley.

Tress, P. W., 1974, West Driefontein Gold Mining Company Limited, The flooding and the recovery; October, 1968 to January, 1972, in Association of Mine Managers of South Africa, Papers and Discussions 1972-73: Johannesburg, Chamber of Mines of South Africa, p. 617-686.

Trexler, B. D., 1979, Hydrogeology of a lead zinc mine, in Proceedings, First International Mine Drainage Symposium: San Francisco, Miller-Freeman Publishing Co.

Venter, P. P., 1969, The problem of underground water in mines, in Proceedings, Symposium on Groundwater in Southern Africa: Johannesburg, Chamber of Mines of South Africa.

Weehuisen, J. M., 1959, New shafts of the Dutch State Mines, in Proceedings, Symposium on Shaft Sinking and Tunnelling: London, Institution of Mining Engineers.

Whillier, A., 1979, Recent advances in the cooling of deep mines in South Africa, in Proceedings, Eleventh Commonwealth Mining and Metallurgical Congress, Hong Kong, 1978: Institution of Mining and Metallurgy.

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