Rock Mechanics in the Design of Mine Layouts

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SYNOPSIS
Most structures are designed so that the possibility of failure is eliminated. However, some degree of failure in the rock around deep mines excavations is inevitable if mining operations are continued. The presence of failed rock constitutes a hazard the magnitude of which can be predicted by computer models from calculations of the energy change per unit area mined in stopes, or the field stress around service excavations. Empirical relationships between these calculable quantities and the incidence of rock failures and delays associated with them have been established. Experiments on the computer using different layouts enable the mining engineer to select the least hazardous of otherwise acceptable alternatives, and to predict the incidence of damage ahead of actual mining, so that appropriate measures to maintain an adequate degree of safety and productivity can be planned in advance.

INTRODUCTION
In engineering practice the criterion customarily applied to ensure that a structure is stable requires that the stresses in every part of the structure should always be less than the stress of that part. The stresses in the structure are generally calculated from some well-established theory concerning the mechanical behaviour of the structure, using values of the loads to which it is subjected when in use. The values for the strengths are usually empirical, being determined from experiment and experience.

This criterion is intuitively satisfying, but carries the implication that the strengths should have been determined under conditions of applied load similar to those obtaining in the structure under consideration. To a large extent this is taken into account by using the most appropriate of the different strengths, such as the ultimate tensile or shear strengths, the yield point, the impact strength or the fatigue strength. Uncertainties about the accuracy of the values of the calculated stresses and the assumed strengths are expressed in terms of safety factors. The safety factor is the ratio between the value chosen for the strength and that calculated for the stress. In terms of the above criterion, the value of the safety factor must always be equal to, or greater than, unity. In civil and mechanical engineering practice, values are usually chosen to three, approaching unity only when a high degree of confidence in the methods of calculating the values of stress and in choosing those for strength has been established.

In the case of those structures comprising the underground excavations of deep mines, calculations of the value of the stress in the rock around them and measurements of the rock strength show that very often the values of the stress exceed those of the stress. This implies that the safety factor in some of the rock around the excavations is less than unity and that this rock must fail. Such predictions are confirmed by underground observations. Prior to mining, most of the rock is remarkably solid, its mechanical behaviour being interrupted only infrequently by significant geological disturbances such as faults and dykes. Nevertheless, most deep excavations are surrounded by rock which is broken to a much greater extent than this or to that which can be attributed to the effects of blasting during excavation.

Structures such as this would ordinarily be regarded as unstable and unsafe. Fortunately, experience proves that instability is the exception rather than the rule; manifestations of it such as rockbursts and rockfalls being rare but, nevertheless, hazardous phenomena.

The design of mines according to the accepted engineering criterion for structural stability would certainly eliminate these hazards, but it would preclude underground mining from most situations where it is already practised with safety, and would render virtually all of it uneconomic. This criterion must, therefore, be regarded as both unnecessarily and unacceptably restrictive.

Over the past decade, research into the behaviour of rock around underground excavations and into the processes of rock failure has resulted in the development of computational methods and the evolution of procedures for the design of underground excavations. The purpose of this paper is to examine the essential features of this work and to define the method now used to design most of the excavations in South African gold mines.

REVIEW
In general, the Witwatersrand reefs are extensive tabular deposits of thin gold-bearing conglomerate, most of which dip at less than thirty degrees. Measurements to determine the virgin state of stress in the rock undisturbed by mining show that the vertical component is due to the weight of the overburden and the horizontal components have an average value of about half this, (Pullister et al., 1970).

The principal excavations are the stopes, formed in the plane of the reef when mining by drilling and blasting at a nominal average width of one metre. These form extensive, flat voids measuring several square kilometres in area down to depths now approaching 4 km below surface. Each year these voids are enlarged by a total of some 30 km² as a result of mining.

Besides these stopes, the rock outside the plane of the reef is traversed by service excavations mostly in the form of shafts and tunnels, which are usually of a near-circular or square cross-section of from 10 to 100 m² in area. At any one time, some 10 000 km of tunnel are in use and these are extended at a total rate of about 1 000 km per year.

Extensive measurements in tunnels and boreholes of the displacements induced in the rock by mining have established that the bulk of the rock mass responds elastically to changes in the size of the stopes of the order of tens of metres over periods of years. (Cook, et al., 1966).

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The tabular form of the stope excavations and the linear behaviour of the bulk of the rock mass allow of certain simplifications which enable the displacements and stresses around stopes of virtually any outline to be calculated with the aid of analog (Salamon, et al., 1964) and digital (Plewean, et al., 1969) computers.

The predominant component of the virgin rock stress across the plane of the stopes is normal, as are the predominant stress changes induced by mining. For deep, tabular excavations this plane may be regarded as one of symmetry. Thus, the problem of calculating the stresses across the intact reef and the displacements between the hanging-wall and footwall in the stopes reduces to solving Laplace's equation in a semi-infinite body with prescribed values of a Newtonian potential and its first derivative with respect to distance into the body on the surface (Jaeger and Cook, 1971). Stresses and displacements outside the reef plane can be found by an integration technique making use of the calculated displacements at the reef plane.

The most important question is, how does one make practical use of these facilities? Where the calculated stresses are less than the strength of the rock, failure should not occur, and the elastic displacements are usually so small as to be unimportant. Elsewhere, the stresses exceed the strength of the rock, failure must ensue and the calculated values of the stresses do not apply. Fortunately, calculation and observation show that the region of failed rock is limited to within some tens of metres of the surfaces of the excavation. Thus, using St. Venant's Principle (Timoshenko and Gootier, 1951) the results are still valid away from this failed region. However, it is just the failed region surrounding the excavations which is of the greatest practical concern.

Two different approaches to this dilemma have evolved; one for stopes and the other for service excavations.

In the case of stopes, the faces are advanced into unmined rock. If the rock were infinitely strong the stresses at a face would vary from values slightly above those of the virgin stresses, for a new stope starting from an isolated raise (reef tunnel), to values many times greater than the actual strength of the rock when two adjacent stopes approach one another, forming a permanent reef between them. In fact, the stresses at a face are always less than the strength of the rock, because the rock fails when the stress attains this value. The amount of failed rock and the extent to which it is damaged by failure are the quantities of practical concern.

Recently it has been demonstrated that, for most rocks, energy in excess of the strain energy at failure is necessary to cause fracture (Jaeger and Cook, 1971; Hudson, et al., 1971). The energy released by mining generates the region of failed rock around stopes. This can be estimated from the average force on a small element of reef prior to mining it and the displacement across it after mining it, as determined from one of the computer models. This energy release per unit area mined should provide a measure of the degree and extent of rock failure produced by mining.

Taking two different gold mines, Hodgson and Joughin (1967) showed that the incidence of rock failures, measured seismically, and the damage in stopes, from recorded interruptions to mining, increased as the spatial rate of energy release increased, Fig. 1. Using the same data, Cook (1970) showed that the proportion of large to small rock failures (seismic events) increased at high rates of energy release. Subsequently, these findings have been corroborated by experience of the use of energy release in planning the layouts for gold mines (Wilson and Moro-O'Terrall, 1970).

In the case of service excavations, failure of the rock around them usually arises from an increase in the values of the field stresses in the rock through which they pass. The field stresses are those stresses which would exist in the rock in the absence of the service excavation, that is, they are independent of the stress concentrations and rock failures around these excavations. The values of the field stress increase as a result of the pervasive stress concentrations resulting from stoping, and are readily calculated. The degree of rock failure and associated damage to service excavations can be related to calculated values of the field stress by underground observations (Wilson and Moro-O'Terrall, 1970) as was done in the case of stope damage and the rate of energy release.

![Graph](https://via.placeholder.com/150)

*Fig. 1.*

(a) The incidence of rock failures, measured as the total radiated seismic energy, as a function of the spatial rate of energy release in various parts of two mines. The solid line is corrected for the detection characteristics of the two seismic networks.

(b) The average incidence of stope damage, measured in terms of the days for which stoping was interrupted, as a function of the spatial rate of energy release in various parts of two mines.
Essentially these two procedures are similar in that, as the geometry of service excavations does not change, the energy released by them to cause rock failure is a direct function of the field stress.

Finally, it has now been shown that the use of improved support can reduce the hazard associated with any degree of rock failure to a significant extent. For example, the incidence of damage for all rates of energy release over a whole mine was reduced by the use of rapid-bearing, timber and concrete packs to a third of what it had been when using conventional timber packs, Fig. 2. (Houghin, 1967). Even under the most arduous rockburst conditions, the use of rapid-yielding props virtually eliminated damage in the stopes and reduced the delays per rockburst from an average of 5,4 days to 2,8 days (Hodgson, et al, 1972).

**Fig. 2.** Statistically-fitted lines showing the average incidence of stope damage, measured in terms of the days for which stopping was interrupted, as a function of the spatial rate of energy release for different parts of a whole mine before, I, and after, II, introducing rapid-bearing timber and concrete packs for stope support.

**METHOD**

Analog and digital computer facilities have been developed which enable the energy release per unit area mined in stopes and the field stresses around service excavations to be calculated for any stope layout, assuming that the rock behaves in a linear elastic manner.

The rock around most deep excavations fails as a result of the disturbances caused by mining. Failed rock constitutes a hazard the magnitude of which increases with the extent and degree of failure. By analyzing field observations it has been possible to establish a relationship between the spatial rate of energy release in stopes and the incidence of rock failures and damage. A similar relationship has been established between field stresses and damage in service excavations. Also, it is well known that the presence of geological disturbances, particularly faults and dykes, increases the probability of rock failure and the severity of damage.

The method of designing stope layouts consists of three steps. First, projected stope layouts and the sequences of mining them are modelled with these computer facilities to find the spatial rates of energy release and the field stresses so as to select the least hazardous of the otherwise acceptable alternatives.

Second, the empirical relationships are then used to predict the incidence of rock failures and damage ahead of mining, so that appropriate measures to maintain an adequate degree of safety and productivity can be planned in advance. In both these steps, allowance must be made for the aggravated hazard in the vicinity of geological disturbances. Wherever possible the layout should be planned so as to stope through faults and dykes at as low a rate of energy release as possible. If service excavations must pass through such disturbances, the field stress around them should be kept as low as possible in the vicinity of the intersections.

Third, if the conditions are such that the best layout would ordinarily be unacceptably hazardous, improved support, such as rapid-yielding props, must be adopted so as to reduce the amount of danger and damage to a tolerable level.

**CONCLUSION**

The methods traditionally used by engineers to design various structures is directed towards eliminating any possibility of failure. In practice, this is done by maintaining safety factors, that is, the ratios between empirically-chosen values of strength and the calculated values of stress in the structure greater than unity.

This criterion is certainly sufficient to ensure the safety of structures, including those in the form of underground excavations. However, in the case of deep mine excavations it would preclude mining from most situations where it is currently practised. In mining, the criterion is, therefore, neither necessary nor acceptable. Some degree of failure in the rock around such excavations must be accepted as inevitable if mining is to be continued.

Analog and digital computer facilities have been developed for calculating the stresses, displacements, and spatial rates of energy release in the rock around tabular excavations of any outline, assuming that the rock behaves in a linear, elastic manner. The spatial rate of energy release during stopping and the field stresses around service excavations have been related empirically to the incidence of rock failures and damage.

Failed rock constitutes a hazard, the magnitude of which depends upon the degree and extent of failure. The existence of a small but finite hazard calls for a method of design different from that directed at eliminating the hazard altogether. The method aims first to minimize the hazard by examining alternative stope and excavation layouts on the computer models and choosing that with the lowest values of energy release or field stress. The empirical relationships between these quantities and the incidence of rock failure and damage are then used to predict the conditions in advance of mining so that appropriate measures to maintain an adequate degree of safety and productivity can be planned. Finally, if these conditions are unacceptably hazardous, special precautions to reduce the damage and danger to a tolerable level must be adopted, such as the use of rapid-yielding props in stopes where a high incidence of rockbursts cannot be avoided.

**ACKNOWLEDGEMENT**

This paper is a summary of the practical results achieved in strata control for gold mines by the Rock Mechanics Division of the Mining Research Laboratory of the Chamber of Mines of South Africa over the years 1965 through 1971.
REFERENCES


