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Stress-displacement behaviour of the fractured rock around a deep tabular stope of limited span

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SYNOPSIS

The effects of the mining of deep-level tabular reefs are usually evaluated by use of continuum, elastic mining-simulation analyses. This paper gives the results obtained from a series of analyses in which stopes of limited span are considered to be excavated in a non-linearly non-elastic, jointed and fractured rockmass. In particular, the theoretical influence of soft backfill in these conditions is evaluated. It is shown that the stress distribution and behaviour of the rockmass, alternately taken as an intact-elastic and as a fractured non-elastic medium, are completely different. The relevance of the energy-release rate commonly calculated in mining-simulation analyses, and the efficacy of its correlation with seismicity and the associated rockbursts, are reviewed. In deep-level mining operations, it is not practically feasible to obviate releases of seismic energy. The fact of their occurrence should therefore be accepted and, rather than considering the reduction of energy release as the primary objective, efforts should be concentrated on minimizing the effects of such release. Analytical results are given to demonstrate that this approach is not only correct, but also the only feasible solution to the rockburst-seismicity problem in deep mines.

SAMEVATTING

Die uitwerking van die ontginning van diep tafelvormige riwwe word gewoonlik met gebruik van kontinuum- en elastiese mynbousimulasieontledings geëvalueer. Hierdie referaat gee die resultate wat verkry is deur 'n reeks ontledings waarin afbouplekke met 'n beperkte spanwydte geag word uitgegrawe te wees in 'n nie-lineêr nie-elastiese gebreekte rotsmassa met nate. Veral die teoretiese invloed van sagte terugvulsel in hierdie omstandighede is geëvalueer. Daar word getoon dat die spanningsverdeling en gedrag van die rotsmassa om die beurt geneem as 'n onversteurd-elastiese en as 'n gebreekte nie-elastiese medium, heeltemal verskillend is. Die toepaslikheid van die energie-vrystellingstempo wat algemeen in mynbousimulasieontledings bereken word, en die doeltreffendheid van die korrelasie daarvan met seismisiteit en die verwante rotsbarstings word in oënskou geneem. Dit is in diep-mynbedrywighede nie praktiese uitvoerbaar om die vrystelling van seismiese energie te vermy nie. Die feit dat dit voorkom moet dus aanvaar word en die werk moet eerder toegespits word op die minimering van die uitwerking van sodanige vrystelling as om die vermindering van die energievrystelling as die primêre doelwit te stel. Daar word analitiese resultate gegee om te toon dat hierdie benadering nie net korrek is nie, maar ook die enigste uitvoerbare oplossing is vir die probleem van rotsbarstingseismisiteit in diep myne is.

INTRODUCTION

Backfill was used as a stope-supporting medium as early as eighty years ago in the South African mining industry. Its benefits were qualitatively assessed in the early years, when it was employed successfully at relatively shallow depth. As the depth of mining increased, subsidence and seismicity became major problems and, owing to a poor appreciation of the mechanics of backfill in a hard-rock environment, the use of backfill was discontinued some fifty years ago. It has recently been reintroduced as a regional supporting medium. The main objectives were to reduce seismicity, to improve the condition of the hangingwall at the face, and to enhance the safety in the working area. Whereas stiff fills are required for the purpose of reducing seismicity, the actual developments in the industry have taken place around soft, hydraulically placed fills. This resulted from the urgent need to improve the safety and stability of the working place, and from comparatively successful applications of deslimed tailings in early trials.

Soft fills had been disregarded as effective stope support because of their apparent inability to develop substantial reactive pressures, and their insignificant effect on the elastic rock stresses around the stope.

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In this paper, an attempt is made to explain the complex way in which joint failure affects the stress-displacement behaviour of the fractured rockmass around a tabular stope, and further to illustrate how the stiffness and compression characteristics of backfill, together with the placement procedures, affect the stability of the rockmass. For these purposes, the rock is assumed to be intersected by two sets of discontinuities on which the degree of 'bridging' or continuity is varied. The one set represents the bedding joints, which are parallel to the stope plane, whereas the orientation of the second set is sub-vertical. The stope is situated at a depth of 2000 m, has a span of 80 m, is 1 m wide, and is assumed to be mined and filled in a single step. Two- and pseudo three-dimensional solution schemes were used to model the stresses and displacements.

DESCRIPTION OF MINING PROBLEM

The behaviour of the rockmass around a stope is affected by the nature and strength of the intact rock and the discontinuities, and by the strength, stiffness, compression characteristics, and installation procedures of the support. The term *specific behaviour* in this paper refers to the stresses, displacements, and fractures in particular localities. The term *global behaviour* refers to the overall equilibrium status and stability of the rockmass and to the prevalent seismicity.

The rocks of the Witwatersrand Supergroup, in which the South African gold mines are situated, consist of sedimentary deposits, which behave as elastic continua if intact, and as non-elastic discontinua if subject to failure on the bedding joints or on other naturally present joints, or if subject to failure of the rockmass itself. The different kinds of behaviour that can result when any of the various types of failure develop depend upon the extent of the failures and on the variability in strength of the discontinuities and the host rock. If the failures are of limited extent, some re-adjustment of excessive stresses will occur, and the rockmass will stabilize as failure terminates. The process of re-adjustment, that is failure, is associated with the release of seismic energy, the magnitude of which depends on the extent of failure. If the failures are of a non-ceasing nature, the ensuing unstable conditions can be arrested only by stope support. This situation is also associated with a continual release of seismic energy in the form of micro-seismicity. Owing to non-homogeneities in the strength of the rockmass or of the discontinuities, failures that have terminated at any one stage may be reactivated subsequently by continued mining. Such circumstances are manifest as instabilities of regional extent, and the release of the associated stored energy is identified with individual seismic events of major proportions.

Stope support can either be continuous as in the case of backfill, or can be applied at particular points as in the case of conventional systems. The backfill can, in turn, be either soft or stiff as associated with hydraulically placed deslimed tailings and aggregate-added mixtures respectively. At small strains, soft and stiff fills are equally highly compressible. A conventional support system provides the equivalent of a constant globally, uniformly distributed support pressure.

Sixteen typical categories of overall rockmass behaviour can be identified, as shown in Table I, in terms of various combinations of the rock strength and stope-support conditions outlined above. The associated equilibrium status and seismicity are also shown.

ROCKMASS BEHAVIOUR AROUND DEEP TABULAR EXCAVATIONS

Some axiomatic aspects of the stress regimes surrounding a tabular stope at depth are referred to in the paper. These are reviewed briefly as follows.

Stress Regimes around an Excavation in an Elastic Continuum

A tabular excavation is surrounded symmetrically by zones of biaxial compression, biaxial tension, and tension–compression as shown in Fig. 1 on the assumption that the rockmass comprises an intact-elastic continuum and on the assumption that the natural ground surface is remote. The sub-horizontal components of tension represent the bending action of the overlying strata that are induced by mining. In broad terms, the sub-vertical compressive stresses in the zones of combined tension–compression are smallest in magnitude at midspan, and increase in principle away from the stope upwards and downwards and towards the abutments over the stope faces. The effect on the stresses around the stope of the natural ground surface in fact not being remote, is not significant in the context of the analvses presented in this paper.

TABLE I

INFLUENCE OF STOPE-SUPPORT REACTION AND DISCONTINUITY CONDITION ON EQUILIBRIUM AND SEISMIC WAVES

<table>
<tr><td rowspan="4">Type of material</td><td rowspan="4">Condition of discontinuity bridges</td><td colspan="4">Type of stope support</td></tr>
<tr><td>No support</td><td>Conventional matpacks and pipesticks</td><td>Deslimed-tailings backfill</td><td>Aggregate-added backfill</td></tr>
<tr><td colspan="4">Stope-support reaction</td></tr>
<tr><td colspan="3">Decreasing</td><td>Increasing</td></tr>
<tr><td>Elastic continuum</td><td>Intact</td><td colspan="4">Stable equilibrium; no seismicity</td></tr>
<tr><td rowspan="5">Non-elastic discontinuum</td><td>Failed partly on first loading</td><td colspan="4">Stable equilibrium; seismic waves belatedly follow face advance at relatively high frequency and low amplitude in homogeneous ground; one-off (low-frequency) energy releases (seismic waves) of major amplitude occur in non-homogeneous ground</td></tr>
<tr><td rowspan="2">Failing continually (intermediate condition unlikely in practice)</td><td>Neutral or unstable equilibrium</td><td colspan="3">Stable equilibrium</td></tr>
<tr><td colspan="4">Seismicity as either above or below</td></tr>
<tr><td rowspan="2">Failed completely on first loading</td><td>Unstable equilibrium</td><td colspan="3">Stable equilibrium</td></tr>
<tr><td colspan="4">Seismic waves immediately follow face advance at relatively low frequency and low amplitude in homogeneous ground; one-off (low frequency) energy releases (seismic waves) of major amplitude occur in non-homogeneous ground</td></tr>
</table>

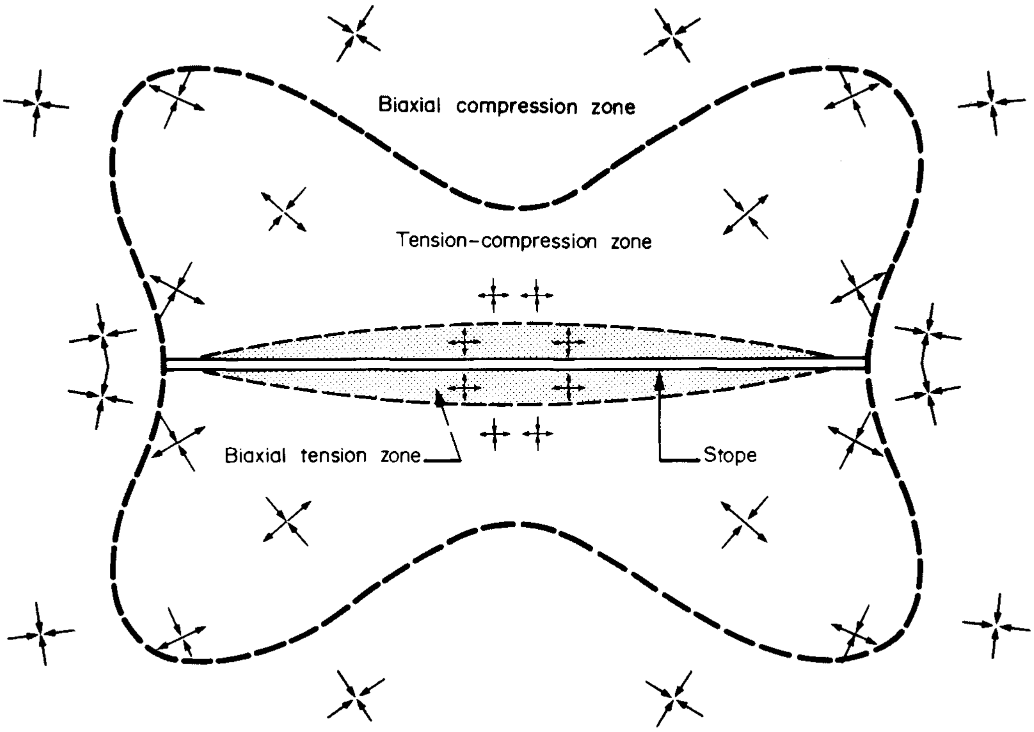

Fig. 1—Extent of tensile-stress regimes around a stope in intact-elastic rock

Stress Regimes around an Excavation in a Non-elastic Discontinuum

The rocks of the Witwatersrand Supergroup are discontinuous by virtue of the bedding planes, cross-joints, and major geological features such as faults and dykes, which occur naturally. The extent to which the stresses around the excavation deviate from that described for an intact-elastic continuum depends on the extent to which shear and tensile failures occur on the natural discontinuities and on the extent to which the intact rock itself fails. Such intact rock failures usually include conjugate shear and extension strain fractures as shown diagrammatically in Fig. 2, and crushing of the reef horizon in the immediate area of the face.

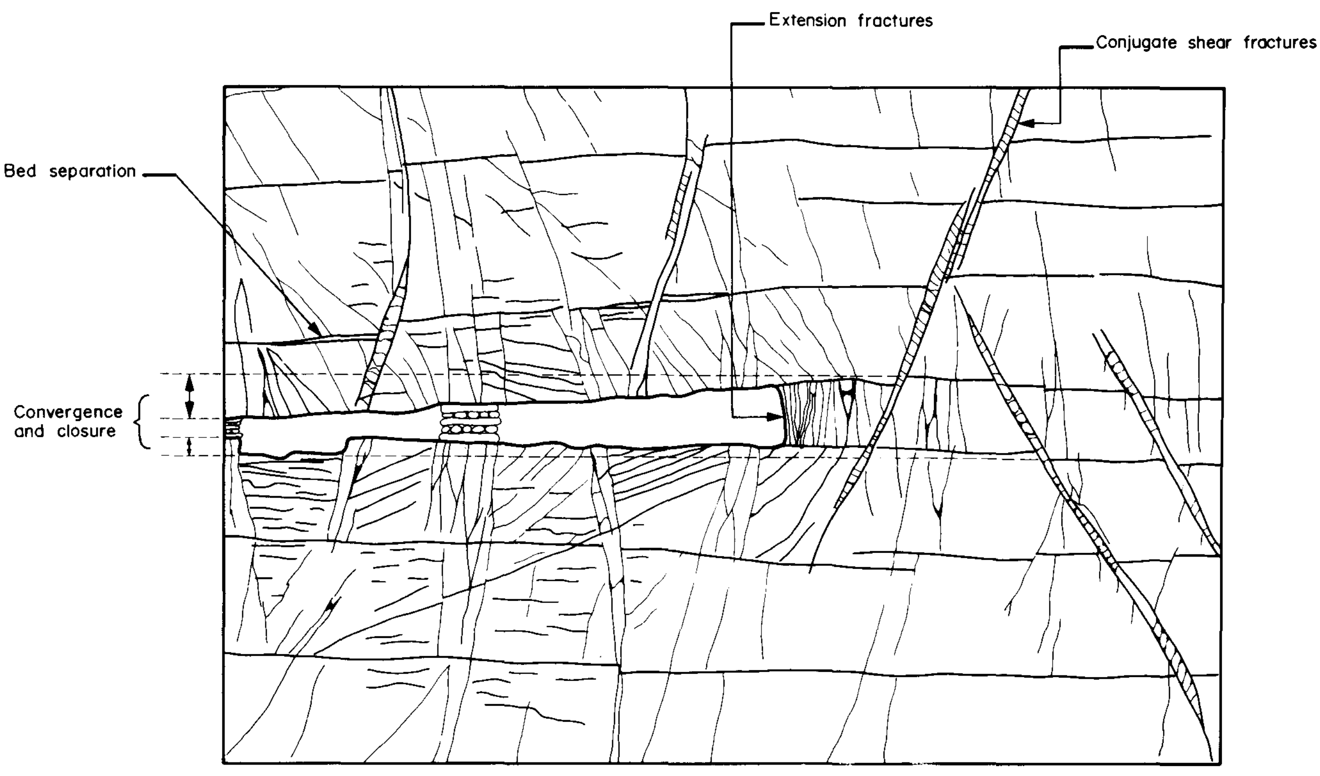

Fig. 2—Fracturing around a stope (adapted after Roberts and Brummer[5])

Effects of Mining as a Multi-step Process

The stresses that develop around an excavation in an elastic continuum are not dependent upon the mining process. The fact that the excavation is made in steps is of no consequence in an ideal material. However, in the real situation in which failure occurs on the discontinuities, the stresses around the excavation are affected by the fact that the excavation is created as a multi-step process. This is the case, not only because of the failures that are induced and because of the irreversibility of the associated displacements, but also because of the variability in strength of the discontinuities. Instead of failing progressively as mining proceeds, some of the discontinuities may remain stable for a period of time and fail only at particular stages when the stress accumulations become excessive.

Manifestation of Discontinuity Failure as Seismic Activity

Progressive failures on the discontinuities are manifest as continual seismicity. One-off failures of potentially disruptive proportions tend to be associated with readily noticeable single seismic events, the intensity of which depends upon the extent of the failures and on the magnitudes of the released stresses.

Correlation between Equilibrium State and Stability Condition

States of stable and neutral equilibrium of the rockmass are associated with stable underground conditions. States of unstable equilibrium are associated with excavation instability. Intact-elastic continua are associated with stable equilibrium, hence stable conditions. Non-elastic media in which the discontinuities are in a terminating state of failure are also associated with stable equilibrium and stable excavation conditions. Non-elastic media in which the displacements around the excavation associated with failure of the discontinuities are unstable are associated with stable or neutral equilibrium if the excavation is actively supported. If not, such media are associated with unstable conditions. Non-elastic media in which the discontinuities fail continually are associated with stable equilibrium if the excavation is actively supported. If not, such media are associated with unstable equilibrium and unstable conditions.

NUMERICAL SOLUTION SCHEMES

Parameter studies for the intact-elastic cases in Table I were carried out by means of two- and pseudo three-dimensional solution schemes, which are based on displacement-discontinuity element formulations. The scheme described by Crouch[1], MINAP, is suitable for two-dimensional problems and is particularly useful in simulating multi-step mining situations. It has also been used to validate, for particular cases, the finite-element scheme that has been developed for non-elastic jointed rock. The scheme used to simulate the pseudo three-dimensional effects of actual mining layouts, MINSIM-D, has been described by Ryder and Napier[2].

Various methods have been developed for dealing with jointed rock, ranging from anisotropic and orthotropic elastic formulations to detailed modelling of individual joints. The first approach is unsatisfactory since it does not allow for non-elastic behaviour, whereas the second is impracticable because of the large numbers of joints usually involved in real applications. The finite-element solution scheme used in this investigation overcame these problems.

It involves a continuum approach as described by Stacey[3], which was developed further by Diering[4]. The rock mass in the scheme adopted is assumed to be a continuum in which threshold values for shearing and tensile stresses are defined for specific discontinuity sets. When the calculated stresses exceed these limits, the excess shear and tensile components are redistributed to unfailed areas elsewhere in the mass. The corresponding displacements are irreversible and are accumulated to represent non-elastic behaviour. This process allows only failed or unfailed states to be simulated. It does not permit the simulation of the actual progression of failure as an intermediate state. The either-failed-or-unfailed-condition-of-discontinuities in the continuum approach adopted is fundamental to the various cases of rockmass behaviour illustrated in Table I.

Although the continuum stress–strain approach inherently accounts for small strains, the accumulation of irreversible deformations that accompany the redistribution of excessive stresses enables large deformations of fractured rock to be analysed provided the iterative procedure is not curtailed prematurely. A modified Newton–Raphson over-relaxation routine is employed in the scheme to accelerate convergence of the iteration process.

Only two-dimensional stress states can be simulated in the scheme, which makes use of linear quadrilateral and triangular elements. The modulus of elasticity, Poisson's ratio, and the density of the intact material in the elements are specified. The orientations, dilatational potentials, and shear and tensile strengths for a maximum of two-joint sets per element, which are assumed to be ubiquitous in the element, are also specified. The shear and tensile strengths are derived from the broken parts of the joints and from the intact rock 'bridges' along the unbroken parts. The shearing strength of the broken parts comprises both cohesive and frictional components, while the tensile strength is taken to be zero. The shearing strength of the rock bridges also consists of cohesive and frictional components. Their tensile strength is taken to be a fraction of the corresponding unconfined compressive strength. The net strength of the rock bridge so defined amounts to a fraction of that of the host rock. Both peak and residual values are specified according to a straight-line Mohr–Coulomb criterion for the shearing-strength components.

Depending upon the magnitudes and orientations of the stresses in any particular locality, any of the two sets of joints can fail either in tension or in shear. Failure of the rock bridges is allowed to progress only along the plane of the discontinuity along which it is located. It is not allowed to step at an angle from the plane of one discontinuity to that of another either parallel to it or crossing it. It should also be observed that, although a discontinuity can fail in tension, the development of actual bed separation, 'opening', under such conditions is not simulated.

MODELS ANALYSED

Geometry of MINSIM-D Model

Six different mining geometries were taken into account in the MINSIM-D analyses. The relevant detail with regard to depth, stoping width and inclination, and support characteristics are given in Table II.

Geometry of MINAP and Finite-element Models

The MINAP and finite-element solution schemes are two-dimensional simulation systems for which purposes the section shown in Fig. 3 was adopted. The depth and stope span for the MINAP analyses were varied together with other parameters as indicated in Table II.

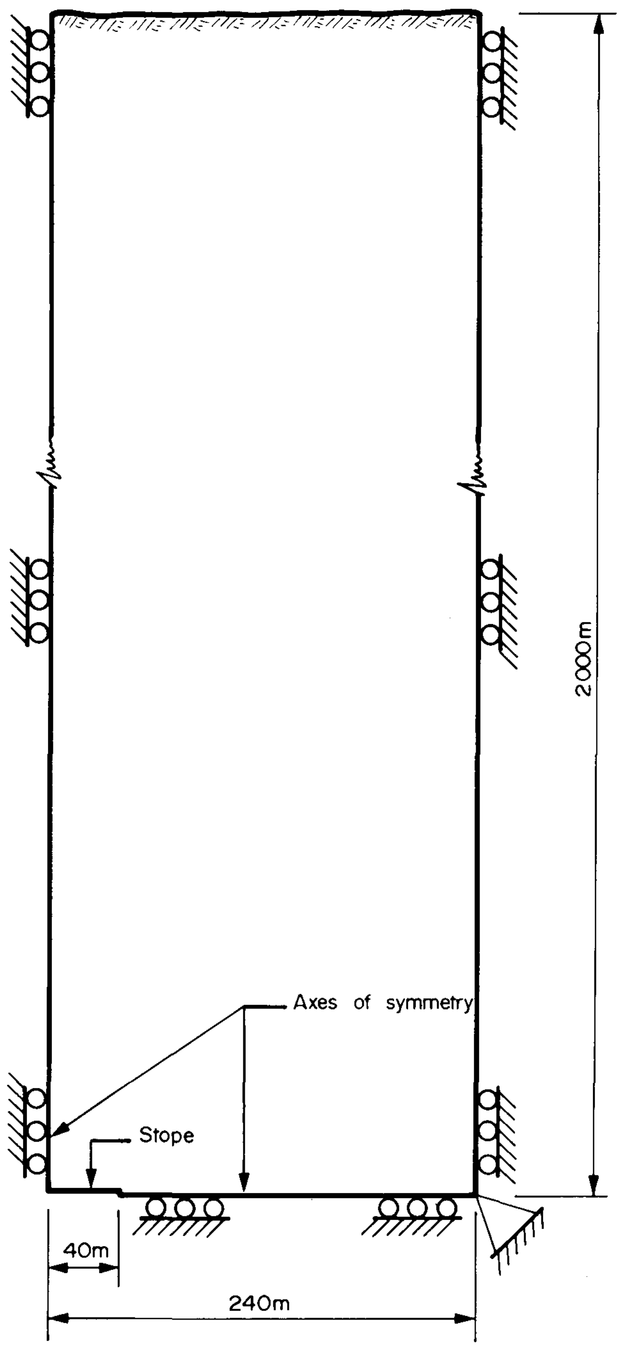

Fig. 3—Two-dimensional model simulated in MINAP and finite-element solutions

Coarse and fine divisions of the finite elements were considered as noted in Table III. The fine mesh, which contained 1340 nodes and 1311 elements, was justified in some analyses in which the accuracy of the solution was affected by the element size. The coarse mesh was found to be adequate in the series of analyses in which the sensitivity of the solution to joint orientation was studied. It contained 997 nodes and 932 elements.

The depth below surface and the span of the stope were taken as 2000 m and 80 m respectively, which corresponds to a deep-level stope in a scattered mining environment. The excavation was considered to be mined in a single step. In those instances in which backfill was included, it was considered to be placed in one step up to 4 m from the face.

Basic Rockmass Properties

The intact rock was assumed to have a cohesion of 32 MPa, a friction angle of 50 degrees, an unconfined tensile strength of 20 MPa, a modulus of elasticity of 70 GPa, a Poisson's ratio of 0,2, and a specific weight of 26 kN/m^3. Horizontal to vertical field-stress ratios of 0,5 and 0,25 were used in the displacement-discontinuity (MINSIM-D and MINAP), and finite-element analyses respectively).

In the finite-element analyses, the rock was assumed to be intersected by horizontal bedding joints and subvertical cross-joints of variable orientation. The continuity of the cross-joints was taken as 50 per cent, while that of the bedding joints was varied. The bedding joints were assumed to be spaced at 0,5 m and the cross-joints at 0,2 m. The friction angle and tensile strength of both joint sets were assumed to be 35 degrees and zero respectively. The peak cohesion of the cross-joints was taken as 200 kPa and that of the bedding joints to be variable. The residual cohesion of both sets of joints was taken as zero. The tensile strength of the rock bridges on both sets of joints was taken to be a variable fraction of that of the host rock.

Backfill Properties

The backfill was considered to be subject to uniaxial strain in all displacement-discontinuity (MINSIM-D and MINAP) analyses, and to a constant proportional loading path in the finite-element analyses. The stress–strain behaviour of the fill, measured in laboratory tests, was modelled by appropriately fitted exponential and power laws as follows:

MINSIM-D Analyses

$$s_1 = e_1/(a + be_1).$$

The symbols s and e denote normal stress and strain respectively, and the subscript 1 the maximum principal components. The values for constants a and b in the various analyses are given in Table II.

MINAP Analyses

$$s = ce^m$$
$$t = dg^n.$$

The symbols t and g denote shearing stress and strain respectively. The values for the constants c, d, m, and n for the various analyses are given in Table II.

TABLE II

SUMMARY OF PARAMETERS DISTINGUISHING MINSIM-D AND MINAP ANALYSES

Run number	A2	A3	A4	P5	A6	A7	A8	A9	A10	A11	B1	B2	B3
Depth (m)	2 252	2 000	2 000	2 000	2 000	2 000	2 000	2 000	2 000	2 000	2 252	2 252	3 252
Stope width (m)	1,0	1,0	1,0	1,0	1,0	1,0	1,0	1,0	1,0	1,0	1,0	1,0	1,0
Dip angle (degree)	7	10	10	10	10	10	10	10	10	10	7	7	7
Strike span (m)	–	30	210	120	180	240	300	360	420	480	–	–	–
Dip span (m)	–	90+	240	320	480	640	800	960	1 120	1 280	–	–	–
Pillar size (m × m)	–	30 × 90	–	–	–	–	–	–	–	–	–	–	–
Tailings content in backfill (%)	No fill	No fill	No fill	No fill	No fill	No fill	No fill	No fill	No fill	No fill	100	100	100
Aggregate content in fill (%)	–	–	–	–	–	–	–	–	–	–	0,0	0,0	0,0
Backfill parameters:													
a (m^2/MN)	–	–	–	–	–	–	–	–	–	–	4 760	10 000	12 000
b (m^2/MN)	–	–	–	–	–	–	–	–	–	–	347	220	450
c (MPa)	–	–	–	–	–	–	–	–	–	–	–	–	–
d (MPa)	–	–	–	–	–	–	–	–	–	–	–	–	–
m	–	–	–	–	–	–	–	–	–	–	–	–	–
n	–	–	–	–	–	–	–	–	–	–	–	–	–
Fill height (m)	–	–	–	–	–	–	–	–	–	–	0,95	0,95	0,95
Mining/filling step (m)	1,0	–	–	–	–	–	–	–	–	–	1,0	1,0	1,0
Fill face-lag (m)	–	–	–	–	–	–	–	–	–	–	–	–	–
Coarse grid size (m)	16,7	5,0	5,0	5,0	7,5	10,0	12,5	15,0	17,5	20,0	16,7	16,7	16,7

Run number	B10	B11	B12	B13	B18	B19	B20	B21	C1	C2	C3	C4	C5	C6
Depth (m)	2 000	2 000	2 000	2 000	2 000	2 000	2 000	2 000	2 252	3 252	2 000	2 000	2 000	2 000
Stope width (m)	1,0	1,0	1,0	1,0	1,0	1,0	1,0	1,0	1,0	1,0	1,0	1,0	1,0	1,0
Dip angle (degree)	10	10	10	10	0,0	0,0	0,0	0,0	7	7	0,0	0,0	0,0	0,0
Strike span (m)	300	360	420	480	40	75	120	180	–	–	40	25	120	180
Dip span (m)	800	960	1 120	1 280	Large	Large	Large	Large	–	–	Large	Large	Large	Large
Pillar size (m × m)	–	–	–	–	–	–	–	–	–	–	–	–	–	–
Tailings content in backfill (%)	100	100	100	100	100	100	100	100	25	25	25	25	25	25
Aggregate content in fill (%)	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	75	75	75	75	75	75
Backfill parameters:														
a (m^2/MN)	12 000	12 000	12 000	12 000	–	–	–	–	9 000	9 000	–	–	–	–
b (m^2/MN)	450	450	450	450	–	–	–	–	120	120	–	–	–	–
c (MPa)	–	–	–	–	1 000	1 000	1 000	1 000	–	–	5 000	5 000	5 000	5 000
d (MPa)	–	–	–	–	160	160	160	160	–	–	800	800	800	800
m	–	–	–	–	3	3	3	3	–	–	3	3	3	3
n	–	–	–	–	3	3	3	3	–	–	3	3	3	3
Fill height (m)	0,85	0,80	0,80	0,77	1,0	1,0	1,0	1,0	0,95	0,95	1,0	1,0	1,0	1,0
Mining/filling step (m)	–	–	–	–	1,0	1,0	3,0	3,0	1,0	1,0	1,0	1,0	3,0	3,0
Fill face-lag (m)	–	–	–	–	3,0	3,0	3,0	3,0	–	–	–	–	–	–
Coarse grid size (m)	12,5	15,00	17,5	20,0	–	–	–	–	16,7	16,7	–	–	–	–

Finite-element Analyses

The stress–strain relationships for deslimed tailings and aggregate-added fills were considered to be linear for strains, D, in excess of 20 and 17 per cent respectively with corresponding moduli of elasticity, E, of 470 and 738 kPa, and Poisson's ratios, F, of 0,3 and 0,3. The constitutive equation for strains less than D was assumed to be given by the power law:

$$s_1 = A(e_1 - B)^c.$$

It was further assumed in the finite analyses that 7 per cent closure of the stope would occur before the fill would begin to take load. This allows for the initial closure of the hangingwall, which occurs immediately after mining before the fill is placed or, alternatively, for the shrinkage of the fill prior to the onset of loading. The values for constants A, B, C, D, E, and F for the various analyses are given in Table III.

GENERAL FINDINGS

The findings that were drawn from the various analyses can be presented as follows in terms of the global and specific behaviour of the rock mass as defined earlier.

Global Behaviour

From the two-dimensional and pseudo three-dimensional displacement-discontinuity analyses, it was found that the effectiveness of the fill in reducing the conver-

B4	B5	B6	B7	B8	B9
2 000	2 000	2 000	2 000	2 000	2 000
1,0	1,0	1,0	1,0	1,0	1,0
10	10	10	10	10	10
30	30	210	120	180	240
90+	90+	240	320	480	640
30 × 90	30 × 90	–	–	–	–
100	100	100	100	100	100
0,0	0,0	0,0	0,0	0,0	0,0
12 000	12 000	12 000	12 000	12 000	12 000
450	450	450	450	450	450
–	–	–	–	–	–
–	–	–	–	–	–
–	–	–	–	–	–
–	–	–	–	–	–
0,96	0,96	0,90	0,91	0,86	0,88
–	–	–	–	–	–
–	–	–	–	–	–
5,0	5,0	5,0	5,0	7,5	10,0

gence and energy-release rate is sensitive to its stiffness and the depth of mining, depending upon the span of the stope. For spans less than 180 m, the fill is shielded, and the associated elastic convergence for any depth is insufficient to consolidate the fill significantly. Under such circumstances, uncemented backfilling will not have an effect on convergence and will not reduce the energy-release rate. Both soft and stiff uncemented hydraulically placed fills will be equally ineffective. For spans in excess of 180 m, a stiff fill, such as an aggregate-added fill, will be more effective than a fill of soft deslimed tailings by approximately 15 per cent with regard to reducing the convergence and the energy-release rate. The effectiveness of stiff fills compared with soft fills will, under these conditions, be greater as the span of the stope becomes greater. Also, in a scattered mining layout, fill will provide a significant reduction in energy-release rate only if placed close to the face and provided that it has an early load-generation characteristic.

The behaviour of the rockmass as derived from the analyses in which it is assumed respectively to represent intact-elastic and non-elastic-fractured media can be summarized as follows from a global point of view in terms of select hangingwall closures and rock-fill contact pressure as given in Table IV.

(i) The stress regimes overlying a stope in an intact-elastic medium are completely different from those in a non-elastic discontinuum in which joint failure is allowed. The respective stress regimes are illustrated in Figs. 4 and 5. The high abutment stresses and zones of tensile stress in the intact-elastic medium, Fig. 4, are completely changed in the medium in which shear and tensile failure of joints occur, Fig. 5. The nether hangingwall in the non-elastic discontinuum is subjected in addition to substantial horizontal clamping stresses, compared with the biaxial tension that is derived from the elastic analysis. The beneficial effect of horizontal clamping stresses in the fractured nether hangingwall is self evident.

Energy-release rates that are determined from solution schemes in which the rock is assumed to be intact-elastic can therefore not be expected to correlate in general with actual seismicity, nor with the hazardous effects associated with corresponding rockbursts.

(ii) Failure of the joints occurs in characteristic zones and locations around the stope, depending upon their shearing and tensile strengths, the depth and span of mining, and the lateral to vertical field-stress ratio.

(iii) Above certain values for the shearing and tensile strengths of the bedding joints, shear failure and bed separation spread to a limited extent only. No backfill or conventional support is required to arrest the redistribution of excessive stresses on the joints. The iterative solution process in the corresponding numerical analysis converges normally for this condition. However, below certain values for their shearing and tensile strength, failure of the discontinuities spreads extensively in the hangingwall. Unless supported by backfill or substantial conventional support, unstable conditions result in the stope. The corresponding numerical analyses are associated with non-convergence of the iterative solution process.

(iv) The discontinuity 'bridges' attract load in preference to the stope support and afford the rockmass a self-supporting capacity. The extent to which this self-supporting characteristic shields the backfill from load or allows the stiffness of the backfill to affect closure depends upon the amount of initial closure of the stope or shrinkage of the backfill. The predilection of the discontinuity bridges to be loaded in preference to the support renders seismicity and the support reaction independent of each other in the early stages of compression. The degree of this independence is increased by the amount of initial closure of the hangingwall or shrinkage of the fill close to the face. Seismicity will generally tend to occur irrespective of the type and status of the support close to the face, and will in principle precede actual unstable behaviour and closure of the stope in the working area because of the relatively small support reactions developed there.

(v) Under conditions of initial closure or fill shrinkage and stable joint failure, stope closure does not depend significantly on support stiffness, but is very much affected by its compression characteristics. Whereas the closure profile is not much affected by the type of fill, closure in the back area is reactively contained by backfilling without significantly affecting the occurrence of seismicity. Conventional support will not reduce closure nor suppress seismicity irrespective of its stiffness.

(vi) In the absence of initial closure or fill shrinkage and for any condition of joint failure, stope closure does not depend significantly on support stiffness, but it is very much affected by its compression

TABLE III

SUMMARY OF PARAMETERS DISTINGUISHING FINITE-ELEMENT ANALYSIS

Run number	A1	F1	G1	G2	H1	H2	J1	P1	P2	P3	P4	P5	P6
Depth (m)	2 000	2 000	2 000	2 000	2 000	2 000	2 000	2 000	2 000	2 000	2 000	2 000	2 000
Stress ratio	0,25	0,25	0,25	0,25	0,25	0,25	0,25	0,25	0,25	0,25	0,25	0,25	0,25
Span (m)	80	80	80	80	80	80	80	80	80	80	80	80	80
Mining/filling step (m)	40	40	40	40	40	40	40	40	40	40	40	40	40
Mesh type	Coarse	Fine	Fine	Fine	Fine	Fine	Fine	Fine	Fine	Fine	Fine	Coarse	Fine
Bedding joints: continuity (%)	100	75	75	75	75	75	75	90	75	75	75	100	100
Peak cohesion (kPa)	100	200	200	200	200	200	200	100	200	200	200	100	100
Cross joints: inclination (degree)	90	90	90	90	90	90	90	90	90	90	90	90	80
Tensile strength rock bridges: Host rock	–	0,16	0,16	0,16	0,16	0,16	0,16	0,01	0,01	0,01	0,04	0,01	0,01
Tailings content in fill (%)	No fill	No fill	100	100	0,0	0,0	No fill	No fill	No fill	No fill	No fill	No fill	No fill
Aggregate content in fill (%)	–	–	0,0	0,0	100	100	–	–	–	–	–	–	–
Backfill parameters:													
A (MPa)	–	–	3 916	3 916	6 150	6 150	–	–	–	–	–	–	–
B	–	–	0,08	0,01	0,08	0,01	–	–	–	–	–	–	–
C	–	–	3,0	3,0	3,0	3,0	–	–	–	–	–	–	–
D (%)	–	–	20	20	17	17	–	–	–	–	–	–	–
E	–	–	489	489	1 136	1 136	–	–	–	–	–	–	–
F	–	–	0,3	0,3	0,3	0,3	–	–	–	–	–	–	–
Constant support pressure (kPa)	–	–	–	–	–	–	300	–	–	–	–	–	–

Run number	P11	P12	P13	Q1	Q2	Q3	Q4	Q5	Q6	Q7	Q8	Q9	R1
Depth (m)	2 000	2 000	2 000	2 000	2 000	2 000	2 000	2 000	2 000	2 000	2 000	2 000	2 000
Stress ratio	0,25	0,25	0,25	0,25	0,25	0,25	0,25	0,25	0,25	0,25	0,25	0,25	0,25
Span (m)	80	80	80	80	80	80	80	80	80	80	80	80	80
Mining/filling step (m)	40	40	40	40	40	40	40	40	40	40	40	40	40
Mesh type	Coarse	Coarse	Coarse	Coarse	Fine	Fine	Fine	Coarse	Coarse	Fine	Fine	Fine	Fine
Bedding joints: continuity (%)	100	100	100	100	100	100	100	100	100	100	100	100	100
Peak cohesion (kPa)	100	100	100	100	100	100	100	100	100	100	100	100	100
Cross joints: inclination (degree)	85	90	95	90	80	90	100	80	90	70	110	90	90
Tensile strength rock bridges: Host rock	0,01	0,01	0,01	0,01	0,01	0,01	0,01	0,01	0,01	0,01	0,01	0,01	0,01
Tailings content in fill (%)	No fill	No fill	No fill	100	100	100	100	100	100	100	100	100	0,0
Aggregate content in fill (%)	–	–	–	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	100
Backfill parameters:													
A (MPa)	–	–	–	3 916	3 916	3 916	3 916	3 916	3 916	3 916	3 916	3 916	6 150
B	–	–	–	0,08	0,08	0,08	0,08	0,08	0,08	0,08	0,08	0,01	0,08
C	–	–	–	3,0	3,0	3,0	3,0	3,0	3,0	3,0	3,0	3,0	3,0
D (%)	–	–	–	20	20	20	20	20	20	20	20	20	17
E	–	–	–	489	489	489	489	489	489	489	489	489	1 136
F	–	–	–	0,3	0,3	0,3	0,3	0,3	0,3	0,3	0,3	0,3	0,3
Constant support pressure (kPa)	–	–	–	–	–	–	–	300	–	–	–	–	–

characteristics. Soft and stiff fills will generally produce similar closure profiles because of their similarity in consolidation rate at low load. Under these conditions, backfill will reactively limit closure in the back area and will, over and above its small effect on the occurrence of seismicity, limit the associated adverse effects irrespective of its stiffness. Conventional support will not reduce closure nor contribute to suppressing seismicity, because of its limited load-carrying capacity.

(vii) Irrespective of its effect on closure and potential seismicity, backfill will ensure a condition of stable equilibrium; hence a condition of excavation stability in non-elastic rock masses for any con-

P7	P8	P9	P10
2 000	2 000	2 000	2 000
0,25	0,25	0,25	0,25
80	80	80	80
40	40	40	40
Fine	Coarse	Coarse	Coarse
100	100	100	100
100	100	100	100
100	70	75	80
0,01	0,01	0,01	0,01
No fill	No fill	No fill	No fill
–	–	–	–
–	–	–	–
–	–	–	–
–	–	–	–
–	–	–	–
–	–	–	–
–	–	–	–
–	–	–	–

R2	S1	T1
2 000	3 252	2 000
0,25	0,25	0,25
80	80	80
40	40	40
Fine	Fine	Fine
100	100	100
100	100	100
90	90	90
0,01	0,01	0,01
0,0	No fill	No fill
100	–	–
6 150	–	–
0,01	–	–
3,0	–	–
17	–	–
1 136	–	–
0,3	–	–
–	3,0	300

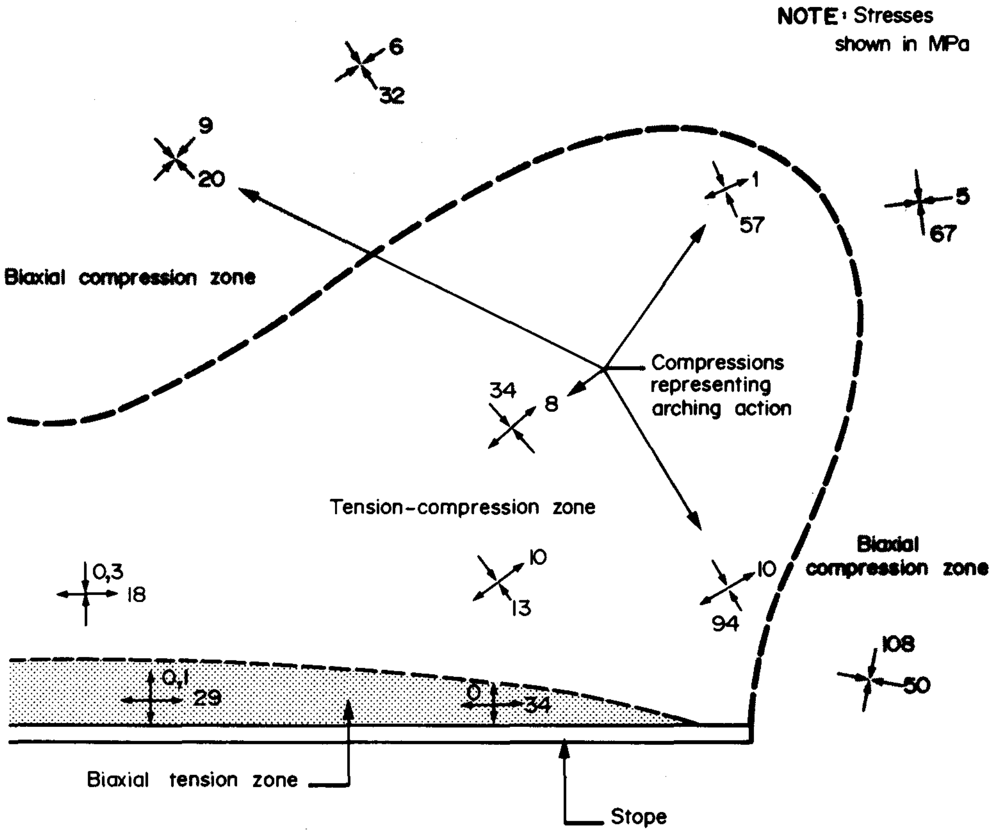

Fig. 4—Stress regimes surrounding a stope in intact-elastic rock, illustrating arching action

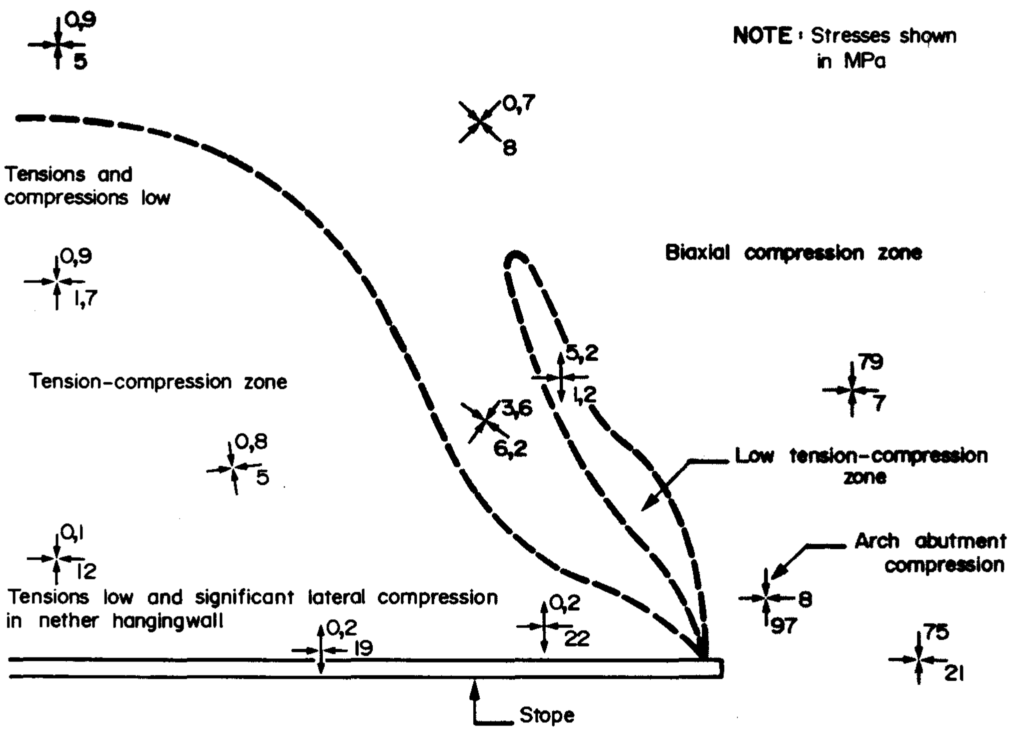

Fig. 5—Stress regimes surrounding a stope in fractured rock, illustrating lateral clamping action in the nether hangingwall

TABLE IV

HANGINGWALL CLOSURES AND ROCK-FILL CONTACT PRESSURES AT 5 m FROM THE FACE AND IN THE BACK AREA AS DETERMINED FROM NUMERICAL ANALYSIS

Type of material	Status of joint failure	Type of backfill: None	Type of backfill: Deslimed tailings	Type of backfill: Aggregate added	Relative magnitude of constant stope-support pressure: Small 3 kPa	Relative magnitude of constant stope-support pressure: Large 300 kPa
Intact elastic continuum	–	*A*[1] 31,8 and 62,7[2]	*B* 31,8 and 62,7[3]	*C* 31,8 and 62,7[3]	*D* 31,8 and 62,7[3]	*E* 31,8 and 62,7[4]
Non-elastic discontinuum	Terminating	*F* 73,2 and 109,5[4] 0,0 and 0,0[8]	*G* 69,2 and 94,1[4] 59,1 and 76,3[4] 30,0 and 190,8[7] 190,8 and 436,7[9]	*H* 70,3 and 90,3[4] 57,8 and 70,0[4] 52,6 and 240,8[7] 278,6 and 519,7[9]	*I* 73,2 and 109,5[5] 0,1 and 0,1[8]	*J* 71,9 and 107,4[4] 11,5 and 11,5[8]
	Neutral	*K*	*L*	*M*	*N*	*O*
	Running	*P* 77,3 and 130,5[6] 0,0 and 0,0[8]	*Q* 71,4 and 99,2[4] 63,1 and 78,8[4] 37,3 and 250,0[7] 236,3 and 484,3[9]	*R* 72,7 and 94,1[4] 57,6 and 71,4[4] 66,2 and 299,6[7] 275,4 and 554,0[9]	*S* 0,1 and 0,1[8]	*T* 77,3 and 130,5[4] 11,5 and 11,5[8]

Notes

1 Categories of behaviour
2 Hangingwall closures (mm) from MINSIM-D and MINAP analyses respectively 5 m from face and in back area
3 Assumed to be similar to closures in category A
4 Hangingwall closures (mm) from FEA respectively 5 m from face and in back area
5 Assumed to be similar to closures in category F
6 Assumed to be larger than closures in category T
7 Rock-fill contact pressures (m head of rock) corresponding to FEA closures
8 Support pressures (m head of rock) specified
9 Rock-fill contact pressures (m head of rock) corresponding to closures for zero initial closure/shrinkage

dition of joint failure. It will therefore contribute substantially towards reducing the effects of large releases of seismic energy.

(viii) It follows from the observations above, viz that closure is dependent upon initial closure of the hangingwall, shrinkage of the fill, and its compression characteristics rather than on its stiffness, that deslimed tailings is an optimum support medium. Aggregate-added tailings will not perform significantly differently from deslimed tailings with regard to closure, support, or seismicity. The most significant attribute of any backfill, compressible as it may be initially, is that its support capacity develops very rapidly under compression. This is the case for both soft and stiff fills. It also follows that the most important aspect of the backfilling procedure is that the fill should be placed as close behind the face as possible. Every effort should, in addition, be made to reduce the initial displacement of the hangingwall and the shrinkage of the fill. Conventional face support would still be required in the working area to minimize initial closure, to ensure stability of potentially loose blocks, and generally enhance the integrity of the hangingwall.

(ix) The fill–hangingwall contact pressures can generally be increased fivefold by obviating initial closure and shrinkage of the fill. If this is not achieved, closures will generally be larger by 30 per cent, contact pressures will be much reduced, stability in the working area will not be assured, and the adverse effects of seismicity will not be maximally limited. In practice, it will not be possible to completely obviate closure. Some closure will inevitably occur before it is possible to place the fill.

(x) Conventional support cannot ensure stable equilibrium during or after a seismic event, because of its discontinuous disposition and because of its inability to generate increasing reactive loads under progressive compression. The potentially unstable conditions associated with conventional support are, as a result, accompanied by rapid and uncontrolled increases in closure.

The above findings are based on analyses in which the rockmass properties are homogeneous. No variations in joint strength across the medium have been considered, nor have the effects of single features such as dykes or faults been taken into account. The differences in properties between the two sets of joints simulated have also not been varied. The ratio of lateral to vertical field stress, the depth of mining, and the span of the stope have also not been varied. All the analyses referred to have been based on a single mining and filling step as indicated earlier.

Specific Behaviour

The findings from the finite-element analyses with regard to specific behaviour can be summarized as follows.

Stope Closure

With conventional support, the regional closures in rock in which joint failure occurs range from 1,7 to 2,1 times that in elastic rock. The corresponding range for backfill support is 1,4 to 1,6. In discontinuous rock, the fill is therefore not shielded from regional compression, even for stope spans as small as 80 m.

Deslimed-tailings fill limits the regional closure approximately as much as an aggregate-added fill for either stable or unstable joint-failure conditions, and of the two kinds of fill is therefore the better supporting medium. It adequately prevents the development of unstable equilibrium by ensuring that the hangingwall in the back area does not converge completely onto the footwall, especially for unstable joint-failure conditions.

For the cases analysed, the closure at 5 m from the face amounts on average to between 56 and 78 per cent of that in the back area for elastic and jointed rock respectively. The comparatively high percentage of closure at the face in a non-elastic rockmass is due to the flexibility afforded by joint failure. The support type, nor its stiffness, nor the type of failure condition of the joints in non-elastic rock, has a very significant effect on the closure 5 m from the face. Such closure for a non-elastic discontinuous medium varies between 1,6 and 2,1 times that for an intact elastic continuum for any joint failure. The limited influence of the fill close to the face is due to shrinkage and to its high compressibility at small strain.

The inefficiency of the fill in the working area can be improved by preventing it from shrinking. The use of fills of greater ultimate stiffness will not solve this problem. The greater cost of such fills will therefore generally not be justified.

Vertical Normal Stresses Ahead of the Face

The vertical normal stresses within 2,5 m of the face are from 1,5 to 2,2 times larger in elastic rock than in fractured rock for any support or joint-failure condition. Energy-release rates are primarily related to the vertical normal stresses in the face. Determinations of these rates from elastic analyses therefore cannot, in general, be expected to correlate well with the seismicity in a mine. Near-field and far-field (regional) seismicity should be clearly distinguished in this regard.

Lateral Clamping Stresses in a Detached Stratum

The horizontal stresses for the intact-elastic case are substantially tensile, and those for the jointed rock substantially compressive.

The compressive horizontal stresses in the detached stratum are largest for no stope support. Conventional support reduces the stresses very slightly, deslimed tailings considerably more so, and aggregate-added backfill even further. Backfill is preferred over conventional support because of its continuity.

Within 10 m of the face, the horizontal normal stress in the nether hangingwall increases for increases in orientation of the subvertical joints up to 100 degrees. For orientations exceeding 100 degrees, a rapid reduction in horizontal stress occurs, signifying a low clamping potential in the working area in association with fractures dipping relatively shallowly towards the face.

Effect of Local Rockfalls on Clamping Action

The effect of falls of rock on the integrity of the detached stratum depends upon the extent of the fall parallel to the face. A fall that extends for a significant distance along the face will neutralize the horizontal clamping forces completely. However, the likelihood of an extensive fall under conditions where the fill is kept close to the face is small. It is more likely that falls of rock will be limited naturally in horizontal extent, in which case a widespread loss of clamping action will not occur. The analyses from which this conclusion is derived are not presented in this paper. The conclusion is, nevertheless, important in conjunction with the occurrence of horizontal clamping stresses in the nether hangingwall referred to above.

Effect of Footwall Centre Gullies on Clamping Action

Gullies parallel to the face represent potential extensive interruptions of the horizontal stresses in the footwall. However, analyses, not presented in the paper, have shown that, in the presence of backfill, the destressing effect is reduced and normal clamping conditions pertain within a short distance of the gulley. The influence of the gulley is not likely to be evident when the backfilled stope has progressed more than 10 m from the gulley side.

CONCLUSIONS

For the cases analysed, the stresses and displacements that are calculated around a deep-level stope when the rock mass is assumed to behave as an intact-elastic continuum are completely different from those applying to a real rockmass in which failures on natural joints and induced fractures occur. Consequently, the performance and effectiveness of backfill as stope support cannot be determined reliably with the commonly used mining-simulation analyses in which the rock mass is treated as an elastic medium. Analyses of that type are also totally inapplicable in the evaluation of the stresses and deformations around stopes in discontinuous rock. In particular, energy-release rates, which are determined from analyses in which the rock is assumed to be elastic, cannot be expected to correlate accurately with near-field seismicity nor with the effects of the associated rockbursts in the working area close to the face.

It has been shown that the most important attributes of backfill are the continuity of support that it provides and the exponential increase in stiffness to which it is subjected on compression. The difference between backfill and conventional support in these two respects explains why conventional support cannot provide the stability of which backfill is capable, and why it allows the rapid and uncontrolled closure often observed after a rockburst. The actual stiffness of backfill is not very important. Deslimed and aggregate-added tailings limit closure almost identically. Backfill significantly reduces closure, especially in unstable joint-failure conditions, but will in all likelihood not reduce seismicity significantly.

To optimize the support provided by backfill, the initial closure of the hangingwall and shrinkage of the fill should be minimized. This requires that the fill should be placed as close behind the face as possible and that the settlement consolidational properties of the fill should be minimized by appropriate adjustment of the grain-size distribution. Unless this is achieved, closures will be larger, fill–rock contact pressures will be much reduced, stability in the working place will not be ensured, and the shakeout effects of seismicity will not be minimized.

Despite its relative ineffectiveness in reducing energy-release rate and the concomitant seismicity, backfill is likely to stabilize the stope optimally under dynamic loading conditions. Its stabilizing mechanism is not related primarily to the extent to which it changes the stresses in the rock mass, but to its ability to prevent the loss of potentially unstable blocks, thereby ensuring the integrity of the nether hangingwall. By limiting stope closure, backfill reduces the kinematic freedom of movement of the blocky material in the hangingwall.

Since it is not possible to altogether obviate the release of seismic energy, one of the aims of underground-support practice should be to minimize energy release. The results of the analyses presented in this paper indicate that the optimum use of backfill will contribute substantially towards achieving this aim.

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Engineers—supply and demand*

The Federation of Societies of Professional Engineers (FSPE) has issued its sixth biennial report on the supply of and demand for engineers in South Africa.

Overall, the lack of growth in the economy in recent years has meant a better balance between supply and demand so that, for the first time in the past two decades, there are slightly fewer than two job opportunities for every young graduate. FSPE draws attention to the fact that the demand for engineers is linked directly to economic activity and that, should the economy pick up, its continued growth would be restricted by the lack of engineers as has occurred in the past.

Immigration has fallen to the point at which it is a small factor in the total supply. There is evidence of the influence of emigration of young engineers in the ageing of the population of engineers—over the past five years, the population has become more than two years older on average.

Growing recognition of the contribution to the profession by technologists and technicians has meant better information on the composition of the engineering team than was previously available. The average team comprises two professional engineers, one technologist, and three technicians, but this varies from discipline to discipline, depending largely on the sector of the economy served by any particular discipline.

In the next five years, a tolerable balance between supply and demand seems assured unless the economy improves, but after that there are likely to be growing difficulties as the number of students matriculating with the essential subjects of mathematics and science is likely to drop.

FSPE supports the rationalization of the existing facilities for tertiary education to economize on the cost of educating engineers during those years, and resists proposals to establish yet further facilities.

Copies of the report are available from FSPE, P.O. Box 61019, Marshalltown 2107, tel. 832-2177 ×156.

*Released by FSPE.