Solid-waste packing as a support medium at depth

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SYNOPSIS

The effects of extensive solid-waste packing as a support medium are described. It is shown that the waste packing resulted in excellent hangingwall conditions with only minor deleterious effects.

SAMEVATTING

Die uitwerking van omvattende vasteafvalpakking as 'n steunmiddel word beskryf. Daar word getoon dat die afvalpakking tot uitstekende daktoestande met slegs geringe nadelige gevolge gelei het.

Introduction

There is considerable interest in the filling of worked-out areas with waste material as a means of regional support for deep gold mines. Many different approaches to the filling of worked-out areas are being investigated, and one such approach being followed by the Chamber of Mines has progressed to the stage where a large area has been completely waste-packed.

In this approach, rock is broken from the face by machines. The waste rock is sorted from the broken rock and packed manually into the back area. Most of this type of mining has been done at the Doornfontein Gold Mining Co. Ltd, where more than 45 000 m² have been mined. It is felt that some of the results of this investigation could be of interest to those concerned with filling as a means of regional support.

Experiment at Doornfontein

The mechanized rockbreaking experiment at Doornfontein is being conducted on part of a longwall face at a depth of 2400 m and where the energy release rate varies from 10 MJ/m² to 30 MJ/m². Parts of the face have been advanced more than 200 m in the strike direction over a five-year period. The face was divided into panels by strike gullies nominally 20 m apart and was mined as straight as possible without any leads between panels and without any headings (Fig. 1).

Mining was done entirely without explosives, a stoping width of 1 m being sought. The actual stoping width was determined mainly by the disposition of the parting planes, which yielded widths of less than 0,8 m in some parts and more than 1,2 m in others. The size of the fragments produced in the rockbreaking process was determined mainly by the prevailing conditions of energy-release rate. The fragments were slab-shaped, typically having dimensions similar to those of books. A very small proportion of fine fragments was produced, which resulted in the large fragments being clean and readily distinguishable for the sorting of the waste rock from the reef.

The hangingwall and the footwall of the stope were severely broken owing to mining-induced fractures and geological discontinuities. Several patterns of mining-induced fractures could be identified, vertical fractures parallel to the face being by far the most common. Fractures dipping at 20 to 40° towards the face occurred relatively infrequently. The intersection of these fractures with the other fractures resulted in unstable wedges and constituted the most significant day-to-day hangingwall control problem.

The major geological structure cutting through the stope was a 5 m wide diabase dyke that intersected the face at an angle of 60 to 70°. A set of joints and faults, the largest having a displacement of 1,5 m, occurred parallel to the dyke. The dyke and associated joint system posed no unusual hangingwall control problem. Nearly all the serious strata control problems were associated with a second system of joints and minor faults that intersected the face at an angle of 140°. A seismic recording network established that nearly all the seismic activity was also associated with this system of joints. Rockbursts, the largest of which was 2,8 on the Richter scale, were invariably located about 10 m ahead of the face on the joints.

The method of support used in the mechanized rockbreaking experiments consisted of hydraulic props along the face, timber packs along the gullies, and solid-waste packing. The hydraulic props were arranged in two rows spaced about 0,5 m apart. Within a row, the props were spaced 2 m apart, and were offset from the props in the other row. The rockbreaking machine advanced the face by about 0,5 m in a pass. Thus, by advancement of the rear row of props by 1 m after the passing of the machine, the props were always maintained within 2 m of the face.

The waste rock was packed about 1 m behind the rear row of props so that the waste pack was maintained within 3,5 to 4 m of the face. The size and shape of the waste rock fragments were well suited for packing. The dimensions of the fragments packed ranged from 20 by 150 by 200 mm to 150 by 250 by 500 mm. The shape of the fragments allowed them to be arranged compactly, and the top layer was forced against the hangingwall (Fig. 2). In principle, it is possible to achieve a packing density of more than 60 per cent of that of solid rock. However, in practice, it was found that the packing density was generally about 50 per cent. The pack was constructed from the down-dip end of the panel, where stulls were attached to the up-dip side of the gully packs. The waste pack was advanced by about 0,5 m, corresponding with the advance of the face after the passing of the rockbreaking machine. Sticks were installed along

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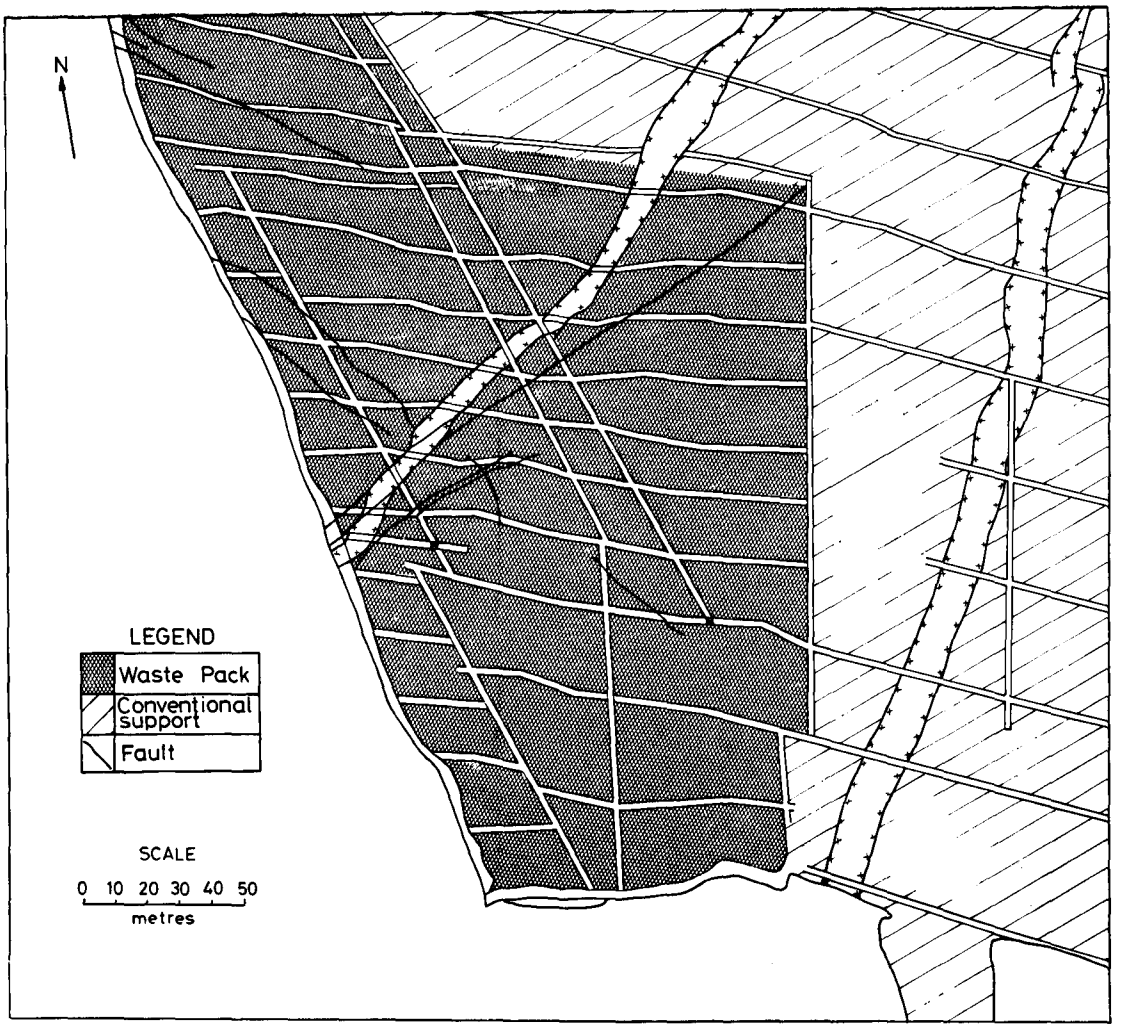

Fig. 1—A plan showing the area mined and packed with waste rock during the course of the mechanized rockbreaking experiments at Doornfontein

the front of the waste pack at about 1 m intervals on strike and 3 m intervals on dip. These sticks were eventually left in the waste pack and were intended to provide support until the waste pack came under compression. The waste filled the entire worked-out area except at the gullies, so that about 80 per cent of the total area mined was supported by solid-waste packing.

Sticks and, infrequently, packs were used on the face where the local hangingwall conditions were considered to be poor, and where the installation and removal of hydraulic props would be undesirable.

The support along the strike gullies consisted of 0,6 by 0,6 m mat packs installed on the sides of the gullies at 3 m intervals. These packs were installed because it was feared that, when the waste pack came under pressure, it would cause cracks to develop along the gully and result in difficult hangingwall problems over the gullies. Initially, the support along the dip gullies consisted only of 0,6 by 1,2 m mat packs.

Effects of Solid-waste Packing

On the commencement of the mechanized rockbreaking experiments with waste packing, the most striking effect was that, after a distance of only 5 to 10 m had been advanced, the hangingwall appeared to be more coherent and the face more intensely fractured. Similar observations have been made in all the other experiments in which mechanized rockbreaking with waste packing has been practised. The total face length over which these effects have been observed exceeds 800 m. Although these effects could be attributed partially to mining without explosives, it is felt that they were predominantly due to the waste pack. It is believed that the solid-waste packing bound the fractured hangingwall and footwall rock together, and reduced the tendency of this fractured rock to be squeezed away from the face. Thus, it is thought that the waste packing promotes the crushing and squeezing out of the rock at the face while keeping the fractured hangingwall rock keyed together.

It was found that the hangingwall condition was so improved that the need to install sticks or other temporary support between the first line of support and the face was greatly reduced. Also, the incidence of accidents due to simple falls of ground was relatively low.

The strike gullies were found to be in excellent condition (Fig. 3), even after a period of 5 years and almost 200 m away from the face. While undoubtedly the good state of the hangingwall over the gullies can be attributed

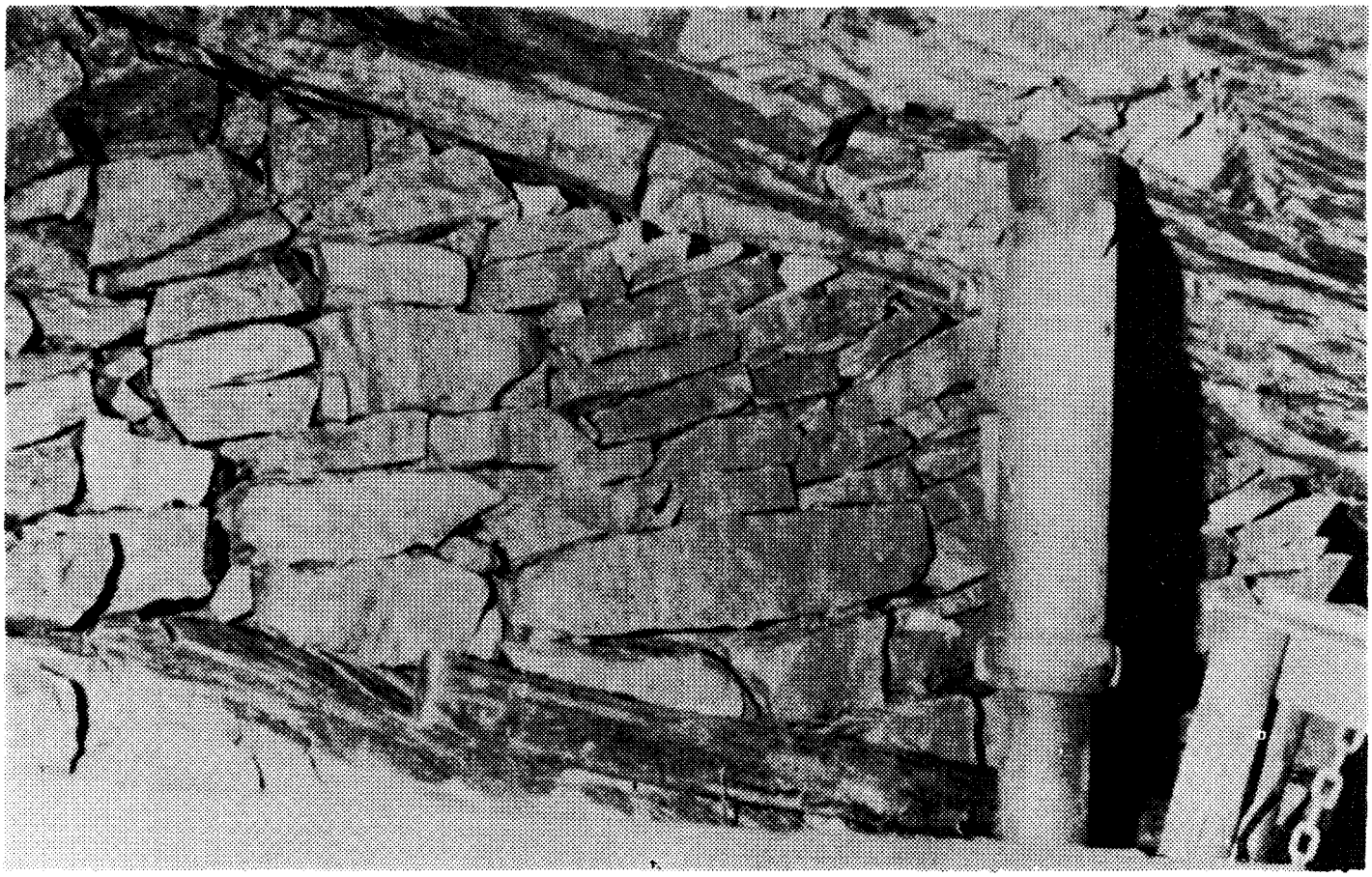

Fig. 2—The waste pack 3,5 m from the face

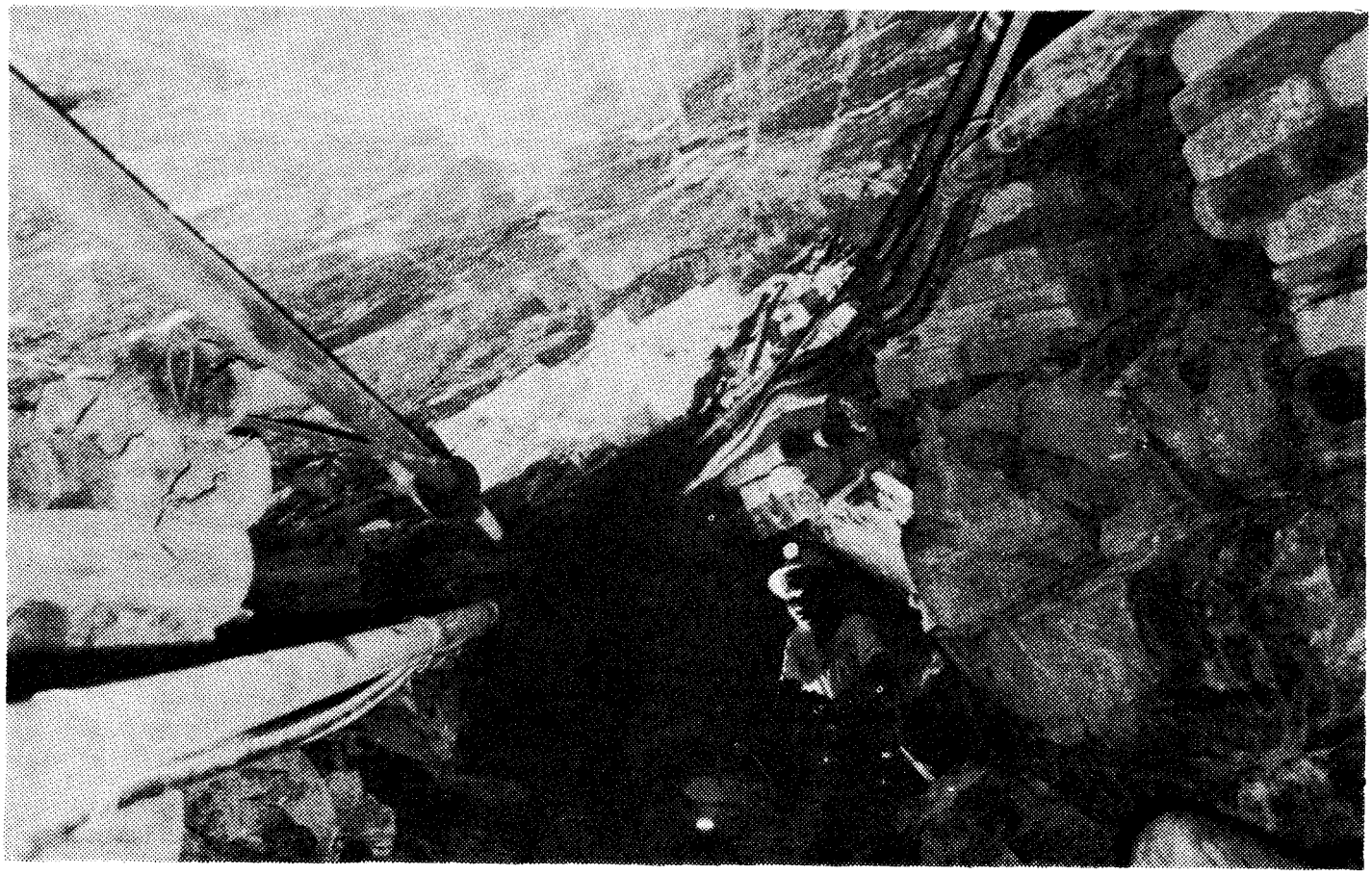

Fig. 3—The hangingwall over a strike gulley 70 m from the face

to the effects of the waste pack keying the fractured hangingwall rock together, it is believed that the maintenance of a straight face and the avoidance of headings and leads between panels also contributed to the good conditions. In conventional mining, where headings or leads betweeen panels are used, fractures parallel to the gully are induced in the rock, which makes the hangingwall above the gullies more difficult to support. There was no evidence that the waste pack induced new fractures along the hangingwall of the gully.

Considerable difficulty was experienced in the dip gullies. Since the dip gullies formed small angles with the face, the mining-induced fractures tended to run along the length of the dip gullies and so gave rise to weak hangingwall and sidewall conditions in the first instance. As the face was advanced and the waste pack adjacent to the gullies came under load, the gullies deteriorated rapidly. Some of the dip gullies had been cut in the hangingwall, and these proved very much more troublesome. Consequently, all the new dip gullies have been cut in the footwall, and rockbolts together with steel mesh have been used successfully in controlling the hangingwall. However, scaling of the sidewalls is still a problem, and it may be necessary to use rockbolts to stabilize them.

Some severe rockbursts have occurred in the stope. In most instances, the damage was restricted to the face and to the hangingwall and footwall between the face and the first line of hydraulic props. In every instance the waste pack was effective in preventing the damage from extending into the worked-out area. In all instances the strike gullies remained open, and it was possible to gain almost immediate access to the face after the burst. In the case of one severe burst, rescue work was impeded because there was insufficient space between the face and the waste pack into which the rock from the burst could be lashed.

At one stage, a fear was expressed that the waste pack might alter the fracture pattern of the rock in such a way that it would increase the incidence of rockbursts. An investigation was carried out in which the occurrence of rockbursts on the rest of Doornfontein mine was compared with that at the mechanized rockbreaking site. It was concluded that, when the prevailing conditions of energy-release rate and the geological structures were taken into account, there was no reason to believe that the incidence of rockbursts was any different from that on the rest of the mine. It will soon be possible to confirm this when the new seismic network is brought into operation on the mine.

It is of interest to note that there was no observable seismic activity that could be related to the compression of the waste pack.

Measurements of the convergence between the hangingwall and the footwall were made in the part of the face in which the energy-release rate was calculated[1] to be 28 MJ/m^2. In the calculation of the energy-release rate and the convergence profile, it was assumed that all

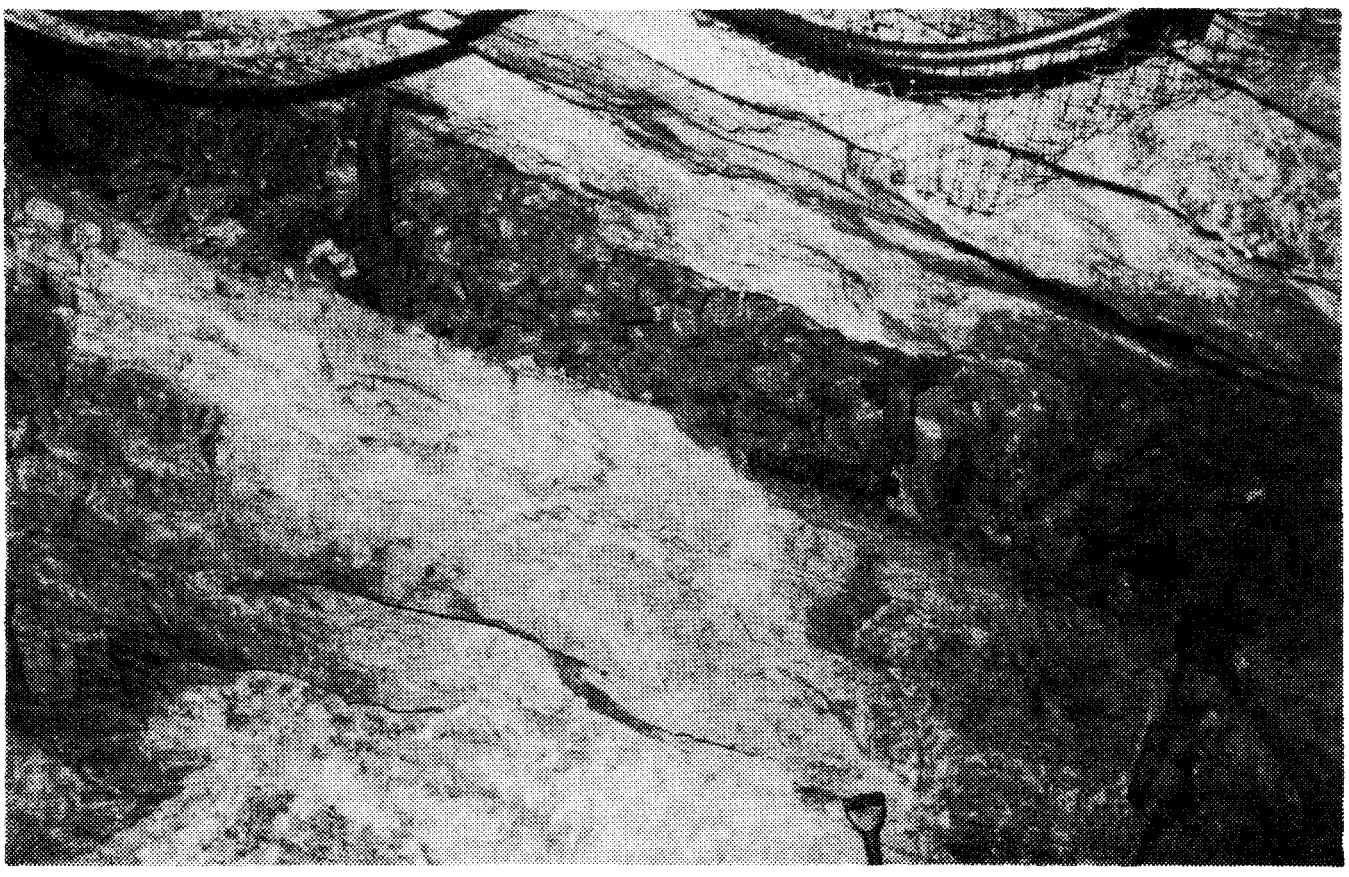

Fig. 4—The partially compressed waste pack 35 m from the face

Fig. 5—The fully compressed waste pack 150 m from the face (the original footwall of the stope is just below the rule)

mining had taken place at a stope width of 1 m and that the waste pack had limited the convergence to 0,5 m. The direct observations of convergence corresponded very well with the calculated values. It was found that the hangingwall and footwall had converged by 120 mm at the point where the waste pack was to be built, and by 330 mm at 27 m from the face. This was the furthest back that measurements were made.

The condition of the waste pack far back from the face was examined directly by mining into it. Fig. 4 shows that, at a distance of 35 m from the face, the waste pack was heavily compressed but there were still some voids in it. Fig. 5 shows the waste pack at a distance of 150 m from the face close to where mining with waste packing was first started. The waste pack was fully compressed and resembled a solid breccia. It is estimated that it has been compressed by 45 to 50 per cent, which corresponds with the initial packing density. At this particular place, the waste pack has been subjected to pressures approaching the virgin-rock pressure, which would eventually be re-established if the entire longwall stope had been mined with waste packing.

From this it would appear that solid-waste packing has the potential to reduce the convergence to about half of that which would occur without the waste packing. This implies that the use of solid-waste packing could halve the energy-release rate at the face and bring about a commensurate reduction in the strata-control problems that are associated with depth.

A very important benefit derived from filling the mined-out area was that the ventilation was vastly improved. In fact, it was found that the quantity of air supplied to the stope had to be reduced simply because the air speeds were so high that blinding dust was being produced.

Conclusion

It is apparent that the extensive solid-waste packing at Doornfontein has resulted in more competent and stable hangingwall conditions at the face and along strike gullies. The support provided by the waste pack allowed good access to the face after rockbursts. A deleterious effect of the waste pack was that it aggravated the poor conditions in the dip gullies.

Waste packing reduces the convergence between the hangingwall and the footwall, with the result that, when extensive areas are waste packed, the energy-release rate would be limited to about half of that with conventional support. There would be a corresponding reduction in the problems associated with deep-level mining.

Acknowledgement

This paper describes some of the work that is being carried out by the Mining Technology Laboratory of the Chamber of Mines of South Africa. The assistance of the Doornfontein Gold Mining Co. Ltd in providing the mining site is gratefully acknowledged.

Reference

1. Van Proctor, R. J. Thesis submitted to the University of the Witwatersrand, 1978.

Discussions of the previous paper

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In the study of underground situations by use of the electrical resistance analogue or MINSIM-type programmes, it is common practice to model the effects of stow or fill material by the assumption of appropriate local reductions in effective stoping width. This is equivalent to the assumption of a sharp 'knee' in the fill reaction characteristic: no support is offered until a critical amount of stope convergence (strain in the fill material) has occurred, at which point the fill reaction becomes unlimited. Fig. 1 (*page 17*) illustrates a typical realistic fill characteristic (unconfined fine slime material), showing critical strain at about 50 per cent and small but non-zero reactions at lower strains.

A simple computer programme was written to model this kind of material in a 2D environment, approximating the curved fill characteristic by a set of straight-line segments. Energy-release rates, ERR, for an isolated longwall stope at a depth of 3000 m and a stoping width of 1 m are plotted in the diagram as a function of half-span in metres.

Curve 1 shows asymptoting of ERR to over 80 MJ/ca for a stope with no fill.

Curve 2 (broken line) shows a halving of ERR if the stoping width is reduced to 0,5 m.

Curve 3 shows the effect of 100 per cent areal stowing with the fill material illustrated. The discrepancy between curves 2 and 3 shows the nature of the MINSIM-type approximation: an over-statement of ERR of about 20 per cent in this particular case.

Curve 4 shows the importance of establishing fill as near the face as possible: with a 45 m lag in filling, about half of the potential benefits are lost.

Curves 5 and 6 show remarkable insensitivity to the use of only partial filling (in this case in dip rib form). Only relatively slight increases in ERR result when the area filled is reduced to 50 per cent, and even to 25 per cent, provided such fill as is used is again inserted as close to the face as possible. If 50 per cent fill is used but is established at a minimum lag of only 11 m, *curve 7* shows again a significant loss of efficiency of the fill. These results depend to some extent on the relative widths of the dip fill ribs (11 m for the curves plotted). For a given percentage fill, wider ribs would worsen the ERR somewhat.

Other studies (not shown in the diagram) have been carried out for a more rigid slimes fill material showing critical strain at only 20 per cent. The results were qualitatively similar to the above, except that, of course, the ERR levels with this fill are very much lower.

The following conclusions can be drawn.

(1) The use of even relatively uncompacted fill material can drastically reduce ERR levels at depth. The MINSIM-type modelling approximation is pessimistic to the extent of about 20 per cent.

(2) Fill at depth must be introduced close to the face (in practice, before stope convergences reach about 20 per cent); otherwise, much of the benefits of fill are vitiated. Dense fill established 45 m behind the face is considerably less effective than even 25 per cent fill density established close to the working face.

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The Colloquium† on Practical Support Methods in Hard-rock Mines has highlighted the significant progress that has been made over the past decade in the understanding of the behaviour of the various kinds of stope support used in the industry. It is gratifying to see that the importance of a high initial support stiffness for the control of the hangingwall strata is generally accepted, and that many of the papers presented at this colloquium describe how this can be achieved in practice. It is equally gratifying to note that the effects of time and rate of loading on the behaviour of stope support utilizing timber in one way or another is acknowledged, and that efforts are being made to keep this influence as small as possible.

Two important points emerge from a critical examination of the various papers. Firstly, most of the papers deal with improvements of individual support elements rather than with an evaluation of the performance of overall support systems. Secondly, there is a marked lack of information about design criteria for the selection of stope-support systems.

It is generally acknowledged that the function of stope support in deep-level hard-rock mining is not to prevent the virtually irresistible elastic convergence between the hangingwalls and footwalls of the stope, but to control the zone of fractured rock surrounding most of the stopes. This can be achieved if the fracture surfaces are kept in intimate contact with one another. In deep-level hard-rock mines, two characteristic sets of discontinuities are encountered in general. The first set of discontinuities are the parting planes, which are of geological origin and generally parallel to the stope hangingwall. The second set of discontinuities are near-vertical stress-induced fractures parallel to the face. The density of these fractures depends on the stress conditions at the stope face, being greatest in the immediate hangingwall but decreasing in discrete steps with distance from the stope hangingwall.

Two important support requirements follow from the above description of the fracture systems in the stope hangingwall. First, the support resistance, expressed in mega-newtons per square metre, of the support system must be sufficient to prevent separation of the immediate hangingwall strata. Experience indicates that, in the

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case of relatively closely spaced parting planes, (i.e.,a fairly laminated hangingwall), it is usually sufficient to support the mass of the first 7 m of hangingwall strata. In the case of fairly massive hangingwall strata, the design criterion for stope support is that the full thickness of rock up to any troublesome geological weakness within the first 10 m of hangingwall strata must be supported. It follows from the above that strata separation can, in general, be prevented with a support system that generates a support resistance of 0,2 to 0,25 MN/m^2. The second support requirement is that intimate contact between the near-vertical face-parallel fractures should be maintained to prevent relative movement between the near-vertical slabs. To achieve this, it is important that the hangingwall strata should be supported at regular intervals. The thickness of the immediate hangingwall beam as defined by the first well-developed parting plane determines the distance between individual supports. The thicker the immediate hangingwall beam, the further apart the support elements can be spaced. In general, highly laminated hangingwall strata require a closer spacing of individual support elements. Since the support resistance is expressed in terms of a support force per square metre of hangingwall supported, the individual load-bearing capacity of the support elements can be lower in the case of the more densely spaced support. On the other hand, support elements of high individual load-bearing capability can be used in the case of massive hangingwall strata, provided the immediate hanging and footwall strata are not being damaged as a result of high contact stresses.

A critical examination of the stope-support systems used in the industry shows that the requirement of a minimum support resistance of 0,2 to 0,25 MN/m^2 is satisfied generally some 10 to 20 m from the stope face but not in the immediate face area. Apart from the time-dependent phenomena, which were highlighted by several of the speakers, one of the most important reasons for the low support resistance in the actual working area is the need for a fairly large span from the face to the first line of support to accommodate the blasted rock. Unsupported spans after the blast of up to, and sometimes in excess of, 4 m are not uncommon.

Since most of the activities in deep-level stopes are confined to the first 5 or 6 m from the stope face, the problem of adequate support *close* to the face is of utmost importance and deserves our full attention. The introduction of hydraulic gold-mining props that are designed to withstand the vigour of the blast have helped to ease this problem to some extent.

The development of pipe-sticks and telescopic pipe-timber props, with their improved initial load-deformation behaviour when compared with conventional timber and with timber and concrete packs and their greater resistance to blast damage, deserves attention. The different characteristics of this type of stope support and the differences in cross-sectional area suggest that they should not be considered merely as replacements for support packs but should be treated on their own merits.

In conclusion, it is my opinion that the future development of stope-support systems for use in deep hard-rock mines must concentrate on providing support to personnel and equipment close to the face. The most important aspect in this respect is to minimize the unsupported span from the face to the first line of support. To achieve this task, it is essential that stope support is not considered in isolation but as part of a mining system whose ultimate aim must be to facilitate the economic extraction of ore at the highest possible level of safety.

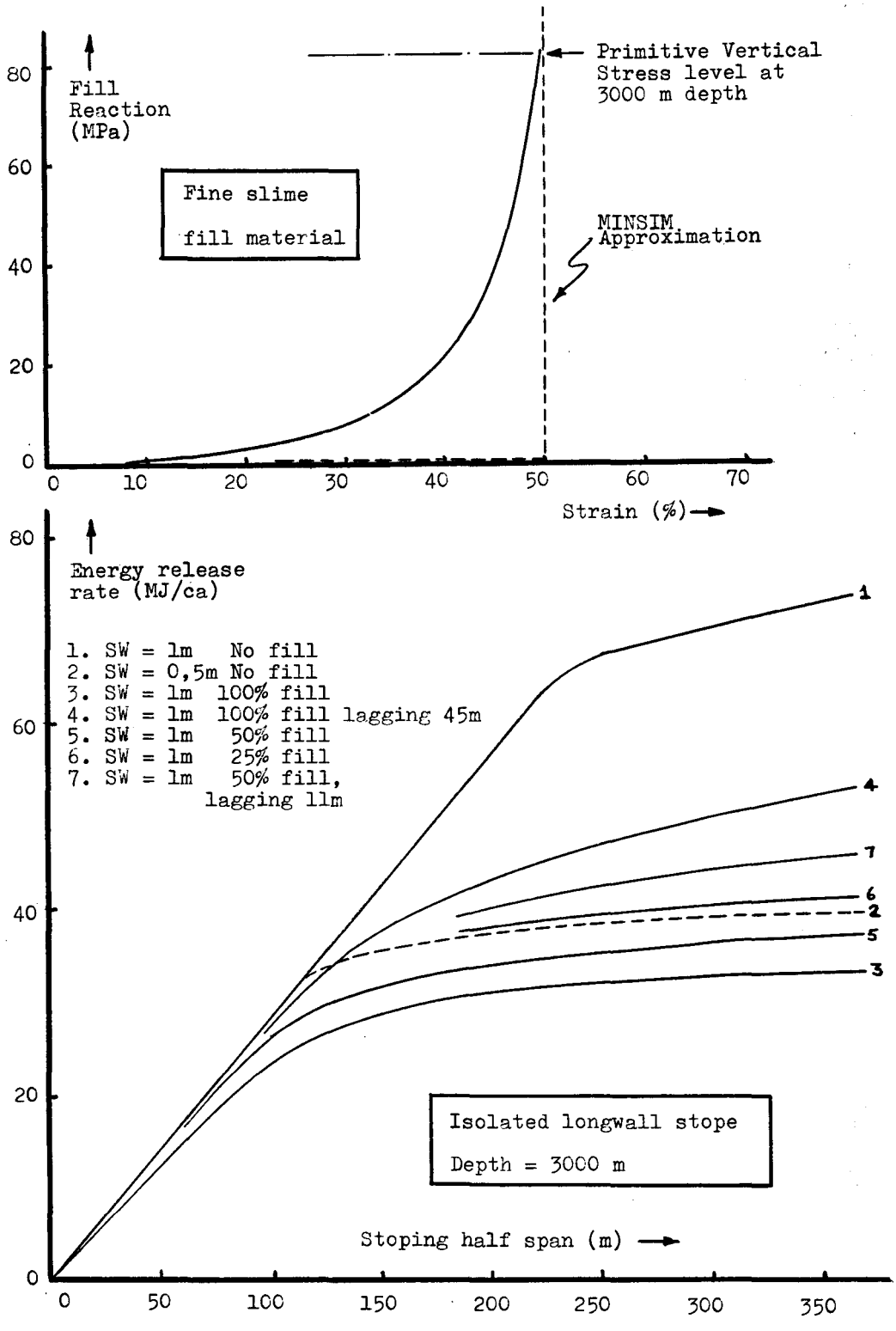

Fig. 1—A typical realistic fill characteristic