The development of rock-burst control strategies for South African gold mines

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SYNOPSIS

The hazard of rock bursts and rock falls as experienced in South African gold mines is examined in the light of recent research findings. The problem areas include high fatality rates, losses in production and equipment, and a detrimental influence on the recruitment of labour. The nature of the hazard is analysed, and it is shown that rock bursts are strongly related to the spatial rate of energy release by mining but that rock falls are not. The role of blasting and geological discontinuities is discussed. Practical rock-burst control strategies are compared, including the use of stabilizing pillars, back-filling and, in particular, techniques for reducing the frequency and severity of accidents resulting from large rock bursts. The latest stope-support experiments and their potential implications to mining are considered. It is concluded that a better understanding of the rock-burst and rock-fall hazard has resulted from the industry's research efforts. The findings can be applied towards obtaining an improved underground environment for men and machines.

SAMEVATTING

Die rotsbarsting- en rotsstortingsgevaar in Suid-Afrikaanse goudmyne word geanaliseer aan die hand van onlangse navorsingsbevindinge. Probleemgebiede sluit in hoë sterftesyfers, produksieverliese, beskadiging van toerusting en 'n nadelige invloed op arbeidswerwing. Die aard van die probleem word ondersoek en daar word aangetoon dat rotsbarstings sterk verband hou met die ruimtelike tempo van energievrystelling as gevolg van mynbou, maar rotsstortings nie. Die rol wat skietwerk en geologiese diskontinuïteite speel ten opsigte van die rotsbarstings-probleem, word beskou. Verskeie praktiese beheermaatreëls vir die bekamping van die probleem word met mekaar vergelyk, insluitende die gebruik van strekkingspilare en terugvulling van die gemynde gebied, en in die besonder, tegnieke vir die vermindering van die voorkoms en skade as gevolg van ongelukke veroorsaak deur groot rotsbarstings. Die mees onlangse afboubestutting eksperimente en die potensiële uitwerking daarvan op toekomstige mynboupraktyk word bespreek. Die slotsom is dat die bedryf se navorsingspoging tot op hede gelei het tot 'n beter begrip van die rotsbarsting- en rotsstortingsprobleem en dat die bevindinge aangewend kan word ten einde 'n verbeterde werksomgewing vir ondergrondse werkers en masjinerie te verkry.

Introduction

The Rock Burst Research Project was instigated in 1973 by the Anglo American Corporation in conjunction with the Mining Operations Laboratory of the Chamber of Mines of South Africa. Its aim is a study of all aspects of the rock-burst problem as encountered in deep-level South African gold mines, and the recommendation of solutions to the problem. The research forms part of an intensified industry-wide programme on the improvement of productivity in the gold-mining industry.

The practical significance of the research completed so far is proving to be considerable. Not only is it already assisting in providing clearer guidelines for the industry but, perhaps even more important, it will also allow future deep-level mining to be planned with greater confidence.

The ideas advanced in this paper are essentially of a strategic nature; while they are generally applicable, the greatest benefits will be derived from their utilization in the planning stages of mines.

For the sake of clarity, the terms *rock burst* and *rock fall* as used here are defined as follows:

rock burst — a seismic event radiating sufficiently intense shock waves to cause visible damage to an excavation

rock fall — a fall of rock fragment(s) from the roof or sides of an excavation

Mine Accidents

The improvement in safety standards in South African gold mines is reflected in Fig. 1, which shows the rates of fatal accidents since 1906.

Despite the rapid increase in the average depth of mining over these years, the average number of fatal accidents due to 'fall of ground' (i.e., combined rock burst and rock fall) has been gradually reduced and maintained at a more or less constant rate of 0,7 per thousand employees per annum. However, the combined contribution of rock-burst and rock-fall accidents to the overall accident rate has risen from about 20 per cent to more than 50 per cent over the same period, making it now the dominant cause of fatal accidents in the industry. In deep-level mines, the contribution is typically 70 per cent.

A more detailed analysis, distinguishing between rock-burst and rock-fall accidents, revealed that the relative contribution of the rock-burst component also gradually rose as the average depth of mining increased.

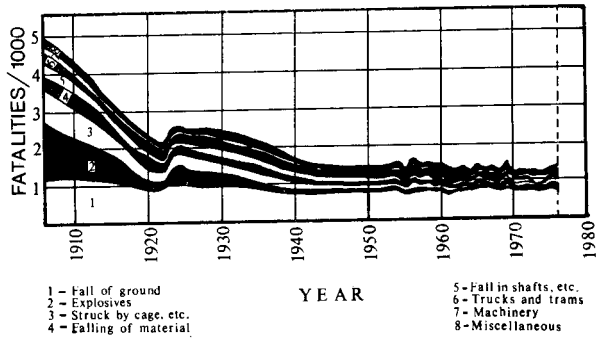

Fig. 1—Accident fatality rate in gold mines per thousand persons in service per annum, allocated under causes[1]

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Fig. 2 shows a breakdown into causes of fatal accidents at Western Deep Levels, a typical deep-level mine, over a recent three year period. The combined fatality rate due to rock bursts and rock falls was 1,75 per thousand employees per annum, or 80 per cent of the total for the mine. The rock-burst contribution equalled 1,0 per thousand per annum, or 46 per cent of the total. Compared with the corresponding estimates of 0,73 per thousand per annum or 55 per cent of the total, and 0,19 per thousand per annum or 15 per cent of the total respectively for five typical shallow and medium-depth gold mines, it is clear that, as the average depth of mining increases in the future, rock bursts and rock falls will tend to dominate increasingly as the major cause of fatal accidents, the relative importance shifting from rock-fall to rock-burst accidents.

When the above fatality rates are compared with the corresponding national average mortality rate of around 0,8 per thousand per annum for males aged 30 to 40, it is clear that, in deep-level mines, the additional risk posed by rock bursts and rock falls is indeed significant.

Apart from average fatality rates, the *temporal* distribution of accidents of a given size, expressed in terms of number of fatalities per accident, was also considered. Based on an analysis of accident data pertaining to Western Deep Levels, the mean time $T(n)$ between accidents resulting in approximately n fatalities among E employees was found to be represented by

$$T(n)=\frac{24\,000}{E}n^{1,5} \text{ months.} \qquad (1)$$

For example, on a mine employing 16 000 people, an accident involving approximately 10 fatalities is expected to occur once every 47,4 months or approximately 4 years.

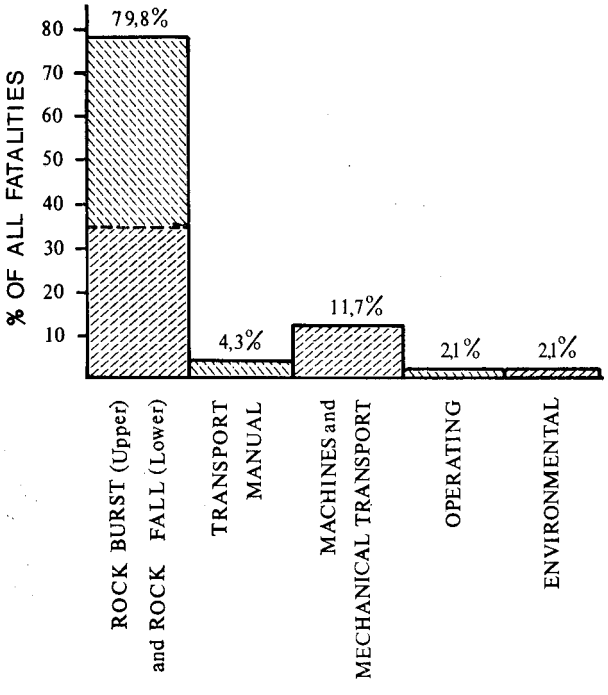

Fig. 2—The relative contributions to the accident fatality rate on a typical deep-level mine, allocated under various causes and taken over a recent three-year period

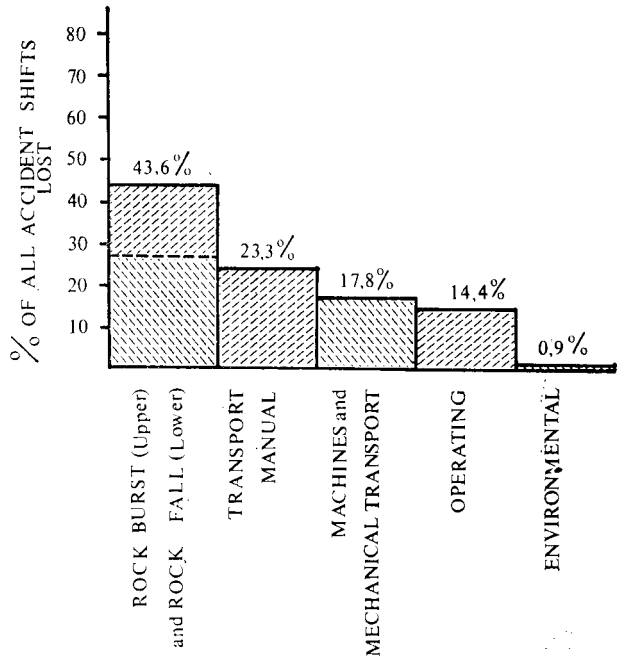

Fig. 3—The relative contributions to the rate of loss in working shifts as a result of accidents on a typical deep-level mine, allocated under various causes and taken over a recent three-year period

A recent study[2] by the South African Chamber of Mines of the *geographical* distribution of 350 fatal rock-burst and rock-fall accidents in gold mines on the West Rand indicated that 60 per cent of fatal accidents occur either in strike gullies within 6 m of stope faces or in the stope face itself. It was also found that approximately 74 per cent of all fatal accidents occur near the stope face. A similar detailed investigation for Western Deep Levels confirmed that about 73 per cent of all rock-burst and rock-fall fatalities occur in or near the stope face.

An assessment was made of the significance of *non-fatal* rock-burst and rock-fall accidents from an analysis of man-shifts lost due to accidents at Western Deep Levels over a three-year period. The results are summarized in Fig. 3. A comparison of these non-fatal losses with the fatality losses depicted in Fig. 2 shows that rock-burst and rock-fall accidents are generally more severe than other accidents, resulting in 80 per cent of the fatalities but only about 44 per cent (2,42 per thousand shifts worked) of the accident shifts lost (5,5 per thousand shifts worked).

Worker Response

Information concerning the attitude of in-service and potential mine labour towards the risk of rock bursts and rock falls is relatively scanty. So that at least some idea could be gained of worker responses, interviews were conducted with first-line personnel officers to establish the attitude of workers to the rock-burst hazard. In combination with results derived elsewhere in the industry[3], a somewhat unexpected picture emerged. It appears that the rock-burst hazard does not affect labour availability and morale as much as is commonly assumed. An analysis of absenteeism and

TABLE I

ANALYSIS OF LABOUR TURNOVER TRENDS IN AN AFFECTED SECTION AFTER A FATAL ROCK-BURST OR ROCK-FALL ACCIDENT

Total no. of fatalities per accident	No. of accidents analysed	Monthly turnover expressed as a percentage of average monthly turnover in the affected section of the mine				
		Months after occurrence of accident				
		−1	0	1	2	3
2	10	97	105	97	81	121
3 to 5	7	88	132	103	88	88
10 to 40	4	103	72	92	137	91

turnover shows that only a small proportion of the total labour force is noticeably affected by rock-burst and rock-fall accidents or incidents. A consideration, for example, of the worst case situations (namely, the turnover of White labour in affected mine overseers' sections on four different mines before and after the occurrence of large accidents) showed no significant trends. Table I shows the responses; it is clear that the fluctuations in labour turnover in the affected sections were well within normal limits.

The absence of significant fluctuations in turnover when and where the impact of major accidents should be at a maximum, makes the existence of significant longer-term effects unlikely.

A similar comparison of absenteeism in the affected sections following particularly severe rock-burst accidents at Western Deep Levels, resulting in the loss of at least 42 panel shifts, is shown in Table II.

It is clear that, in the month after the accident, absenteeism rose to about 15 per cent, compared with the usual average level of 4 per cent, but soon returned to normal. Only 3 of these catastrophic events occurred on the mine during 5 years of mining, contributing only 2 to 3 per cent of the total number of man-shifts lost due to absenteeism. Similar comparisons for smaller accidents, in which the number of panel shifts lost per accident was less than 10 and which constitute approximately 95 per cent of all rock-burst accidents, fail to show any consistent rise in absenteeism in the affected sections after such accidents.

Although similar information pertaining to Black labour is not available, there are indications that the effects of the rock-burst hazard on morale are even less. For example, a 1978 survey[3] conducted by the Chamber of Mines on Black novice labour entering the mining industry failed to show safety as a major factor affecting the attitude of Black labourers once employed on the mines. Risk was regarded as being the *least* important unfavourable factor associated with their new jobs.

TABLE II

ANALYSIS OF WHITE LABOUR ABSENTEEISM TRENDS IN AN AFFECTED SECTION AFTER A CATASTROPHIC ROCK-BURST OR ROCK-FALL ACCIDENT

No. of panel shifts lost per accident	Percentage of absenteeism in affected section				
	Months after occurrence of accident				
	−1	0	1	2	3
42	2	1	13	2	3
≈50	4	5	17	15	3
76	3	1	13	0	2

However, before entering the mine labour market, risk to safety was regarded as being the *most* important unfavourable factor.

Losses in Production

Production losses attributable to rock bursts and rock falls over a four-year period on Western Deep Levels are shown in Figs. 4 and 5. Losses directly attributable to rock bursts and rock falls are expressed as percentages of total production. The combined losses due to rock bursts and rock falls averaged 11,5 per cent, as can be seen from the two diagrams. Although the average of the rock-burst losses is considerably higher than the 2 to 3 per cent claimed for the industry, a much more disturbing figure is the short average period between damaging rock bursts or rock falls in a particular panel, which constitutes a face length of approximately 35 m. An analysis of rock-burst data pertaining to 250 panels over a three-year period showed the

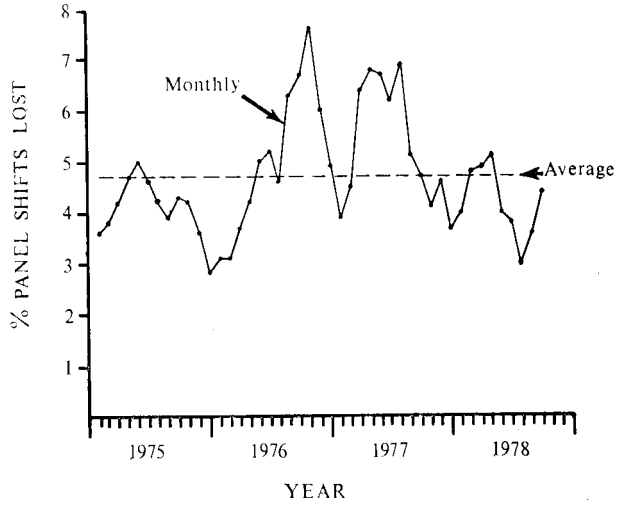

Fig. 4—Production losses due to rock bursts for a typical deep-level mine, expressed as a percentage of actual production over a period of four years

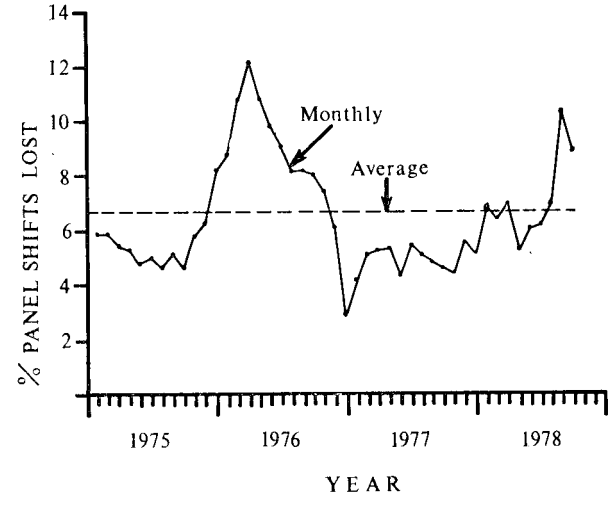

Fig. 5—Production losses due to rock falls for a typical deep-level mine, expressed as a percentage of actual production over a period of four years

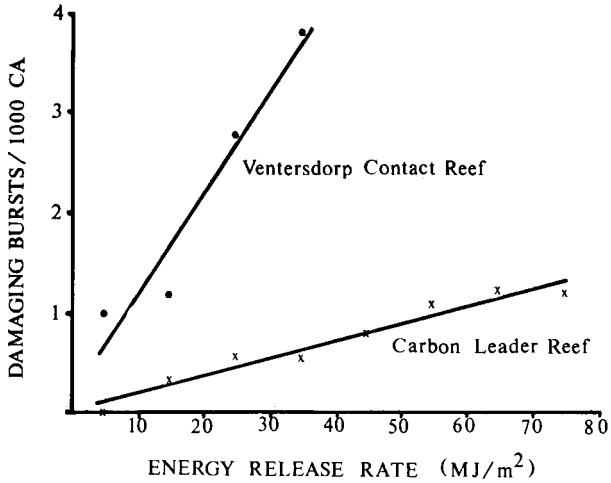

Fig. 6—The relationship between production losses due to rock bursts and the energy release rate for two different reefs. The average number of panel shifts lost per damaging rock burst equalled 4,5 independent of energy release rate

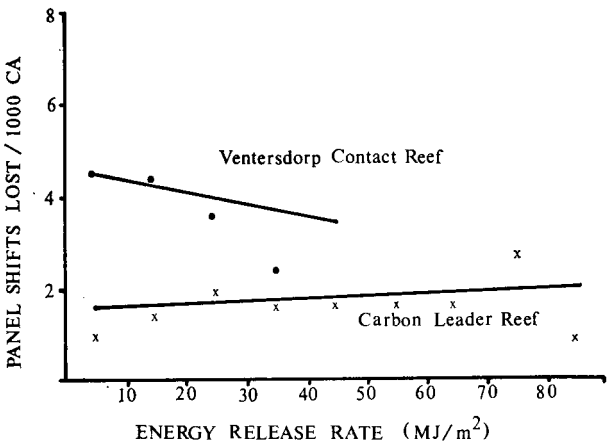

Fig. 7—The relationship between production losses due to rock falls and the energy release rate for two different reefs

interval between damaging rock bursts in any particular panel to average approximately 100 days. This almost certainly would become an important factor for consideration when capital-intensive equipment is introduced in the stopes of deep-level mines.

Nature of the Hazards

The findings of the Rock Burst Research Project highlighted a basic difference between the nature of the rock-burst and the rock-fall hazards[4, 5].

It was found that the most suitable measure for describing the relative difficulty of mining on a particular reef or in a particular area is the spatial rate of energy release (ERR) due to mining. This single scalar quantity expresses the combined effect of the mining geometry, depth of mining, stoping width, and elastic properties of the rock. Methods for the calculation of ERR have been reported extensively[6–8]. Figs. 6 and 7 summarize the results, based on two years of production data, for the Ventersdorp Contact and Carbon Leader Reefs. Production losses are expressed in terms of panel shifts, i.e. the number of lost blasts on a 35 m panel. One panel shift produces approximately 70 t of ore.

It is clear from these diagrams that a sympathetic relationship exists between production losses through rock bursts and ERR, and that rock-fall losses were highly independent of ERR. It is also clear that the actual levels for a given ERR differed markedly for the two reefs, probably owing to different country rock. Scanty data at high ERR on the Ventersdorp Contact Reef reduces the significance of the slight negative slope of the curve for rock-fall loss versus ERR. Little error is introduced if behaviour similar to that shown by the well-established curve for the Carbon Leader is assumed, but at a higher average level of 4,5 panel shifts lost per thousand centares (ca). The relationship between accident man-shifts lost versus ERR was found to be less predictable, probably also because of insufficient data.

Similar, supporting results were obtained through independent investigations conducted by the Chamber of Mines Research Laboratories at Harmony Gold Mine[9] and at the East Rand Proprietary Mines[10]. Although the results obtained in these three instances need not all be directly applicable to other mines, it is nevertheless felt that the key concepts can generally be applied to the control of rock-burst and rock-fall hazards, or to a comparison of the potential effectiveness of various options available to a mine in adopting a rock-burst and rock-fall control strategy.

From the above it appears that the rock-burst hazard has an almost linear relationship to ERR in a particular area, whilst the overall rock-fall hazard is apparently not strongly related to ERR and is more likely to be related to a combination of quality of support, stoping width, mining method employed, and geological structures in the vicinity of the reef excavations. The strategic control of rock bursts can therefore be effected only through a reduction of the average ERR, while rock falls, given a particular reef and mining method, can be reduced only by an improvement in the quality of support.

Another important factor emerging from the analysis was that the *total* number of panel shifts lost per thousand centares, i.e. due to rock burst, rock fall, and other causes, showed only a slight dependence on ERR. In the Carbon Leader Reef, for instance, the following relationship was found to closely describe the actual total loss rate L_t:

$$L_t = 45 + 0{,}333 \times \mathrm{ERR}(\mathrm{MJ/m^2}) \text{ panel shifts per thousand centares} \quad (2)$$

From Figs. 6 and 7, the combined rock-burst and rock-fall component, L_r, for the same reef is given approximately by

$$L_r = 2 + 0{,}018 \times \mathrm{ERR}(\mathrm{MJ/m^2}) \text{ panel shifts per thousand centares} \quad (3)$$

If the average ERR of the mine is reduced fourfold, from say 40 MJ/m² to 10 MJ/m², the rock-burst loss rate would be reduced to 25 per cent, the combined rock-burst and rock-fall loss rate to 80 per cent, and the overall loss rate to 83 per cent of the original levels. A very significant reduction in average ERR would therefore lead to a significant reduction in the rock-burst hazard, but only a relatively small improvement in production.

The above findings are of a general nature and are not intended for use in the analysis of specific isolated areas prone to rock bursts over a short period of time, but should prove valuable in the making of strategic decisions concerning the evaluation of mining layouts.

Geological Discontinuities

It is generally known that most mining-induced seismic events are associated with geological discontinuities in the surrounding rock mass, including dykes, faults, joints, and bedding planes. The discontinuities represent planes of weakness and are therefore likely to fail before mining-induced stresses reach the high levels required for the fracturing of intact rock. Recently considerable effort in the modelling of the behaviour of discontinuities has led to a better understanding of the problem[11, 12].

A practical example of the influence of a regional predominant joint system on seismicity along longwalls is shown in Fig. 8. The left side of Fig. 8 shows the cross-sectional distribution of seismic events taken over a two-year period around an east advancing longwall at Western Deep Levels, while the right side shows the distribution for a nearby west advancing longwall. The directions of the longwall and predominant joints are respectively approximately perpendicular and parallel in the two cases. The difference in seismic behaviour may have a bearing on the rock-burst and rock-fall hazard[13], and is currently under investigation.

Recent findings of the Chamber of Mines Seismic Research Project in the Klerksdorp area included the interesting observation that the maximum magnitude of seismic events originating on faults is strongly related to the displacement on the fault. The following expression was quoted[14]:

$$M_{max}=0{,}32+1{,}88 \log D, \quad \ldots \ldots \ldots \quad (4)$$

where M_{max}=maximum Richter magnitude, and
D=fault displacement.

From the above it appears that, in large displacement faulting, very large rock bursts are possible.

Large displacement faulting was not considered in the studies at Western Deep Levels since the maximum throw encountered on faults in the vicinity of mining

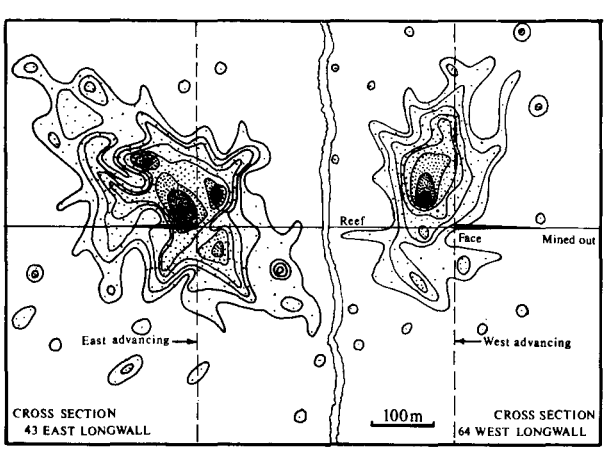

Fig. 8—The distribution patterns of seismic events around two longwalls advancing in opposite directions, taken over two years

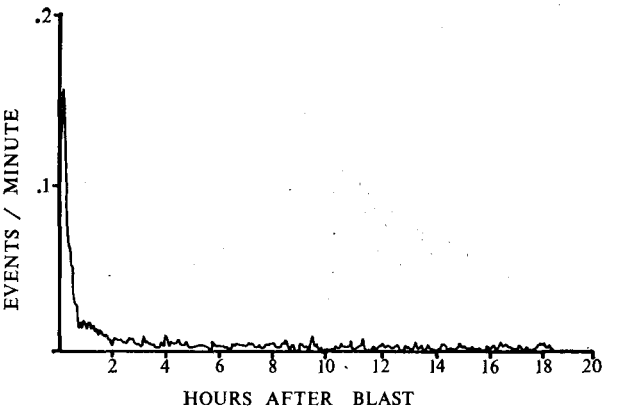

Fig. 9—The diurnal distribution of seismic events on the Carbon Leader Reef, normalized to the daily blasting time

operations rarely exceeds[4] 40 m. Instead, an attempt was made to quantify the strategic importance of the dykes and small faults abounding in the area in terms of their contribution to the overall rock-burst and rock-fall hazard. It was found that the effect of any dyke or fault intersecting, or within about one panel length (35 to 40 m) of, a panel, could be accounted for by the addition of 30 MJ/m² to the calculated ERR for that panel. The proportional correction (ERR+30 MJ/m²)/ERR, if multiplied by the production losses predicted in terms of the theoretical ERR as in Figs. 6 and 7, would then be a good estimate of the loss rates due to rock bursts and rock falls in an affected panel.

A situation often encountered, in which the longwall shape is not maintained in the immediate vicinity of dykes striking perpendicular to the direction of longwall advance, could be explained quite effectively by the inclusion of the 30 MJ/m² term. A rule of thumb was found; if dyke panels were planned to achieve $2\frac{1}{2}$ additional blasts per month, the longwall shape would be more or less maintained over a period of time.

Since Western Deep Levels has approximately 240 panels with an average ERR of 40 MJ/m², approximately 45 of which intersect dykes or faults at any one time, the contribution of the geologically affected areas to the overall rock-burst and rock-fall hazard should be $45\times70/(195\times40+45\times70)$, i.e. approximately 30 per cent. This is considerably less than the estimated average figure[2] for the industry of 60 per cent, illustrating that the influence of geological discontinuity becomes less as the average ERR increases.

A further important observation concerning dyke–dyke, dyke–fault, and fault–fault intersections[13, 15] was made at Western Deep Levels. The intersections act as stress concentrators, generating clusters of small seismic events in their immediate vicinity. The positions of the intersections can be determined from a study of the location of these seismic-event clusters. The implication is that this phenomenon, which should manifest itself most strongly during the very early stages of mining when the original stress field is perturbed for the first time, could possibly be used to delineate dykes and faults long before mining approaches them. The approach of longwalls to major dykes and faults could

then be planned timeously to minimize the risk of rock burst. Placement of critical excavations such as replacement haulages or inclined shafts could also be planned to avoid major dykes and faults.

Seismicity and Blasting

A detailed study was made of the relationship between blasting and seismicity on the Ventersdorp Contact and Carbon Leader Reefs at Western Deep Levels[16, 17]. A large number of diverse conclusions resulted from the analysis, the most important being the following:

(1) A significant portion of the total number of seismic events on a mine occurs contemporaneously with enlargement of the excavation. The size of this portion may vary from reef to reef, e.g. 25 per cent on the Carbon Leader Reef versus 8 per cent on the Ventersdorp Contact Reef, as can be seen from Figs. 9 to 11.

(2) If it is assumed that most of the mining-induced seismicity is closely coupled to the actual enlargement of the stoping excavations, then continuous mining will distribute seismicity more uniformly in time, resluting in a likely on-shift seismicity increase of about 200 per cent.

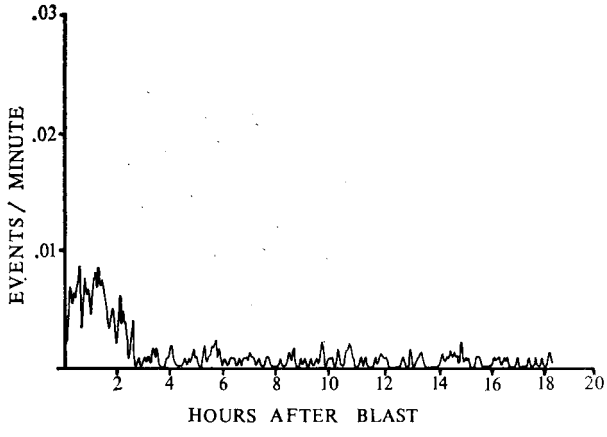

Fig. 10—The diurnal distribution of seismic events in the hangingwall (lava) of the Ventersdorp Contact Reef, normalized to the daily blasting time

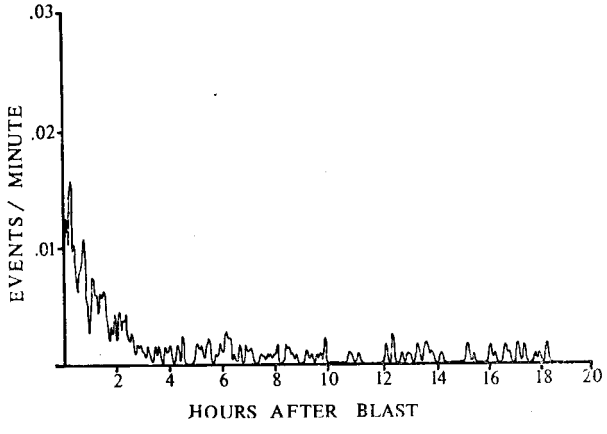

Fig. 11—The diurnal distribution of seismic events in the footwall (quartzite) of the Ventersdorp Contact Reef, normalized to the daily blasting time

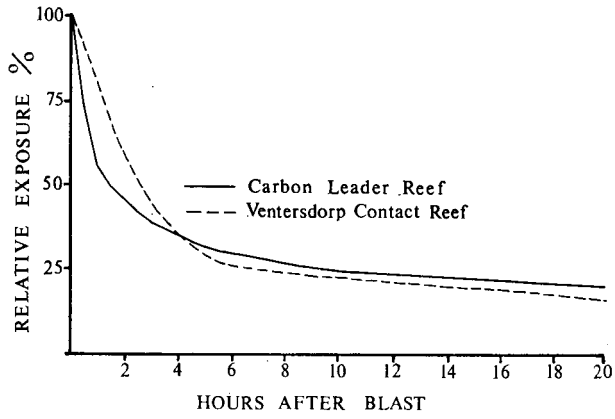

Fig. 12—The relative rock-burst exposure versus re-entry period following blasting, for the Carbon Leader and Ventersdorp Contact Reefs

(3) Blasting in one area of a mine was found to increase the probability of rock bursts elsewhere by up to 500 per cent, depending on the distance and amount of blasting. This is of importance where adjacent mines blast at different times, and would complicate the use of scattered blasting methods for reducing ore-surge capacity requirements in deep-level mines.

(4) The exact sequence of blasting between various sections using a centralized blasting system does not appear to affect later on-shift seismicity significantly. For instance, synchronized blasting is neither expected to aggravate nor alleviate the later on-shift rock-burst safety hazard. The important factor is to minimize the exposure of people in stopes while blasting takes place in the vicinity.

(5) Optimum re-entry periods vary from reef to reef. Fig. 12 shows the rock-burst exposure at Western Deep Levels if the re-entry period after blasting is varied on the Ventersdorp Contact and Carbon Leader Reefs. The rock-burst exposure is normalized to the exposure associated with re-entry immediately after the blast, on the assumption of an 8-hour underground shift.

Control of Rock Bursts

Two important objectives in the development of an effective strategy for rock-burst control are, firstly, a reduction in the average rock-burst fatality rate per thousand employees per annum of a mine if this exceeds the average for the industry and, secondly, a limitation of the incidence and effects of single, very large rock-burst accidents.

As indicated earlier, it appears from the findings of the Rock Burst Research Project and other studies that a statistical reduction of the rock-burst hazard can be achieved through a reduction in the energy released by mining, which in turn can be achieved by the practice of partial extraction, by the back-filling of mined-out areas with waste rock, slimes, sand, etc., or by a reduction in the stoping width if total closure occurs in back areas. The installation of timber or prop support does little to reduce the incidence of rock bursts, as already stated.

Irrespective of the method used, the effectiveness of reducing the energy released by mining depends strictly on the *prevention of elastic closure and ride* in the mined-out area.

Design criteria, and the advantages and disadvantages of various mining methods using partial extraction have been dealt with in considerable detail[6–8,10]. In deep-level mines practising longwall mining, the introduction of on-reef strike pillars, spaced regularly along the entire length of the longwalls, appears at this stage to be the most attractive method for reducing the energy released by mining, for the following reasons.

(a) Maximum protection is afforded at the face *where* and *while* people are exposed. The protection remains very constant throughout the life of the longwall, which is not the case for dip pillars.

(b) Strike-stabilizing pillars do not normally seriously interfere with the development of footwall drives, and only minor modifications to existing layouts would be required.

(c) The life of overstoped or understoped access excavations should increase considerably since they would remain destressed through the life of the mine. Replacement development should therefore be reduced.

(d) Segmenting of longwalls allows for greater flexibility of mining when major unplanned stoppages occur, e.g. due to fires or rock damage in stopes and haulages.

It was found[18] that properly designed pillars would in practice typically result in a sixfold to eightfold reduction of the energy released by mining and, hence, an expected proportional reduction of the rock-burst hazard. The combined rock-burst and rock-fall fatality rate would then drop to 50 per cent of its previous level in the case of Western Deep Levels, since the rock-fall component is not expected to be significantly influenced by the introduction of pillars. Although approximately 15 to 18 per cent of the reef would be permanently locked up in a well-designed strike-pillar system, the final overall extraction ratio at the end of the life of a mine should not be much reduced because the use of stability pillars will reduce the size of the final remnants, which would otherwise not be extracted. From equation (2) relating total production losses to ERR, it is also clear that some reduction can be expected in production losses if the average ERR is reduced by a factor of 6 to 8. With Western Deep Levels as an example, it is expected that the introduction of a strike-pillar system on the Carbon Leader Reef would result in a 7 to 9 per cent improvement in production per metre of available face.

The problems associated with the introduction of strike pillars on existing longwalls arise mainly from the development of ventilation raises or cross-cuts through the pillars for single-district ventilation of the longwall. Since the development is always close to or on the reef plane, the pillar has to be regularly slotted on-dip in order to destress the intersecting raises or cross-cuts. In the past many problems have been encountered with the slotting operation.

If the position of the face immediately down-dip of the pillar is allowed to lead that of the face immediately up-dip of the pillar by a distance equal to the pillar width, the stress environment for the on-reef slotting operation will be lower than the average for the longwall, provided the slotting operation is completed before the face has advanced a further pillar width. Fig. 13 shows how the maximum ERR increases if the operation is delayed unduly. Although the ERR does not exceed 50 MJ/m² in the typical example shown, the pillar would normally be highly cross-fractured by a combination of a strike-fracture system due to the pillar and possibly a dip-fracture system parallel to the longwall and face directions. The necessity for an *early completion* of the slotting operation, while stress levels and ERR remain relatively low, has been borne out by experience at Western Deep Levels.

The effectiveness of back-filling in reducing ERR, and hence the rock-burst hazard, was modelled for various typical mining situations, using a MINSIM-type computer programme modified to model non-linear back-fill behaviour. The results were compared with those obtained from the modelling of strike pillars[19, 20].

The model involved the mining of a hypothetical 3 km by 3 km area with a 1 km by 1 km shaft pillar at the centre through the life of the area. Mining was simulated as breast longwalling with advance on strike. The reef was given an average depth of 3200 m, a dip of 21 degrees, and a stoping width of 1 m. The following cases were compared:

(i) Complete extraction.

(ii) 20 m wide on-strike stabilizing pillars spaced 140 m apart, i.e. one pillar per three 40 m panels, giving 85 per cent extraction.

(iii) 40 m wide on-strike waste pillars, spaced 120 m apart, i.e. 33 per cent of the area, or every one in three 40 m panels, stowed. Ultimate compactions to 50, 60, and 70 per cent of initial stoping width were assumed.

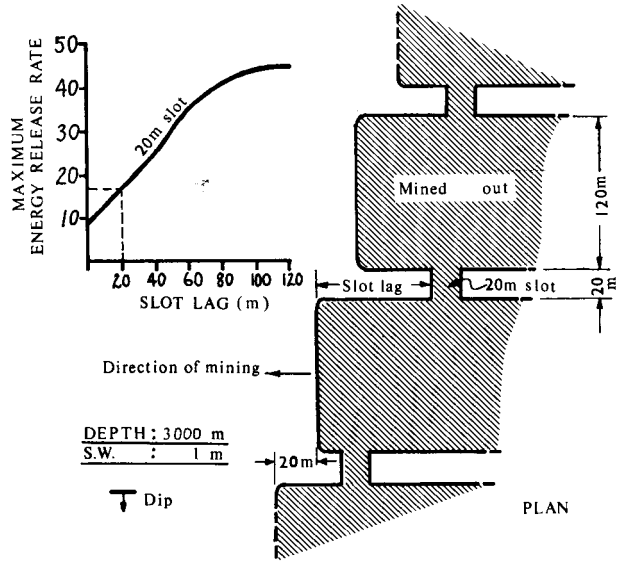

Fig. 13—The maximum energy-release rate that is encountered with the on-reef destressing of strike pillars increases rapidly if this lags behind the longwall

TABLE III

COMPARISON OF AVERAGE ENERGY RELEASE RATE (MJ/m^2) FOR STABILIZING PILLARS VERSUS BACK-FILLING FOR THE HYPOTHETICAL EXAMPLE

Back-filling				Complete extraction	Stabilizing pillars
Ultimate compaction	50%	60%	70%	—	—
Area stowed					
33%	43	36	28		
50%	40	33	25	68	11
100%	37	29	22		

(iv) Same as (iii) but 50 per cent of the area, or one in every two panels, stowed.

(v) Same as (iii) but 100 per cent of area, i.e. all panels, fully stowed.

The results are summarized in Table III.

The amount of energy absorbed by typical practical fill materials was calculated to be between 1 and 4 MJ/m^2 in the above mining situation. As can be seen from Table III, the fill effectiveness is determined largely by the ultimate compaction, rather than by its ability to absorb energy.

It appears that strike pillars are generally much more effective than back-filling in reducing the ERR. This applies particularly to situations in which the closure without support would be small compared with the stoping width.

The frequency and effects of large single rock-burst accidents can be reduced as follows.

(1) A reduction of the overall energy released by mining should also reduce the incidence of large rock bursts.

(2) Segmentation of longwalls by strike pillars should usually confine rock-burst damage to the faces between two adjacent pillars. From this point of view, slender pillars would be sufficient, e.g. 20 m pillars spaced 140 m apart with a 1 m stoping width, to yield 85 per cent extraction.

(3) Segmentation of longwalls by allowing, for example, one half of the longwall to advance ahead of the other half by a distance of 100 m or more. MINSIM-type simulations show that the two parts will behave almost independently, and experience at Western Deep Levels has also shown that they are unlikely to be affected simultaneously by a single large rock burst.

(4) Segmentation of longwalls into short panels of 20 to 40 m, each leading its upper or lower neighbour by about 10 m, results in a similar effect. Apart from being a desirable layout for cleaning purposes, where dip and strike scrapers are used, and allowing for some production 'slackness' in the system to cope with unplanned stoppages, the short leads tend to confine rock-burst damage to a very limited number of panels, each lead acting as a damage 'barrier'.

The segmenting of longwalls by the introduction of very long leads has been practised at Western Deep Levels for the past ten years, but it was found that this method, though effective in reducing the extent of the damage due to large events, introduced other undesirable features. The trailing panels immediately adjacent to the long lead will always be subject to extremely high energy-release rates and intensive cross-fracturing patterns, caused by the acute intersection between the long lead and the face directions. It was also found that longwall segmentation into panels, though effective in controlling damage due to small and medium rock bursts, was not effective in controlling damage resulting from very large events. Over the same ten-year period, there were many instances in which more than five contiguous panels were simultaneously affected by a single event. Also, the presence of a 10 m lead between panels is sufficient to result in cross-fracturing of the hangingwall and, since this zone usually coincides with the position of the scraper strike gully, it is probably the main cause of the high accident rates associated with these gullies. The argument is supported by the generally lower accident rates exhibited by the strike or diagonal gullies of breast-mined longwalls.

The strain energy stored in the rock surrounding a longwall supported by stabilizing pillars was shown to be much lower than that without pillars[21]. Evidence to date also indicates that strike pillars do not increase the rock-burst hazard in any way, provided their width is sufficient to inhibit pillar failure and provided the average pillar stress is sufficiently low not to cause foundation failure.

It was concluded that strike pillars appear to present the best solution to confining rock-burst damage to a relatively small face length, in addition to reducing the energy released by mining. If at all possible, longwalls between two pillars should not be further segmented by the introduction of large inter-panel leads since this would tend to aggravate gully conditions.

Control of Rock Falls

The findings of the Rock Burst Research Project indicate that the hazard of rock falls is not strongly

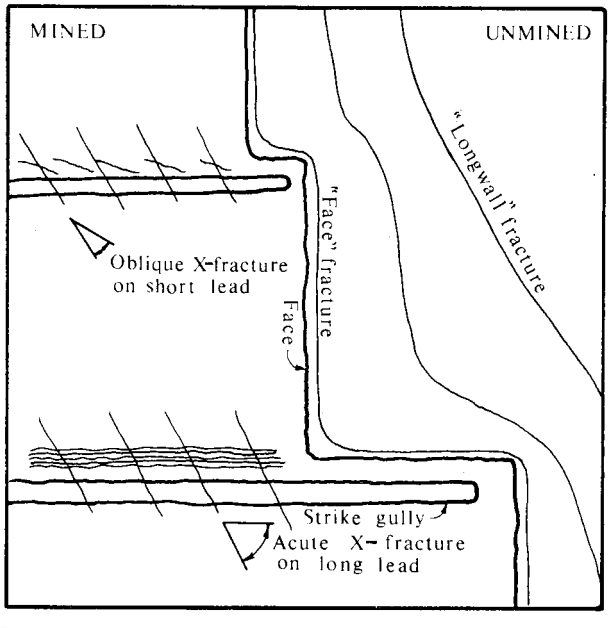

Fig. 14—Plan view of a typical long and short lead panel, showing the interaction of the longwall fractures with face fractures and the resulting hangingwall conditions in the strike gully for the two cases

related to the energy released by mining, but rather to the mining method and quality of installed support for any given reef. The average rock-fall hazard certainly varies widely from one reef to another, as illustrated in Fig. 7 for the Ventersdorp Contact and Carbon Leader Reefs on Western Deep Levels.

The importance of the interaction of various mining induced fracture systems with one another, or with pre-existing geological joint systems, has been highlighted by recent research findings[13, 15]. Although their implications to the rock-fall hazard appear complex and are not well understood at this stage, certain general but useful observations have been made.

The most undesirable situation from the point of view of rock falls appears to be the *presence of a cross-fracture pattern*, i.e. two fracture systems intersecting at an acute angle, in the hangingwall where people are regularly exposed. Such a condition was often found to exist near or in strike barrier and boundary pillars, and also in the strike gullies of stope panels having excessively long lags or leads, as is often found in overhand or underhand longwalls. Fig. 14 shows how longwall and face fracture systems are formed, and how they tend to form cross-fracture patterns in the vicinity of strike gullies as the lead between stope panels is increased. Two distinct fracture systems, longwall and face fractures, were found to exist almost without exception in about a hundred panels that were mapped intensively. Fig. 15 shows a typical distribution of fracture orientations in one of the measured panels. It appeared that leads of even 5 m were sufficient to result in a prominent cross-fracture pattern, adversely affecting strike-gully conditions in the vicinity of the face.

The Rock Burst Research Project attempted to assist in defining the requirements of good or 'optimum' installed stope support for deep-level mines having stoping widths of not more than 1,5 m, and to evaluate the *in situ* performance of existing support types. Firstly, it was found that support should be provided as *close to the face* as possible; secondly, that only *act've* support was capable of providing meaningful support load close to the face[22].

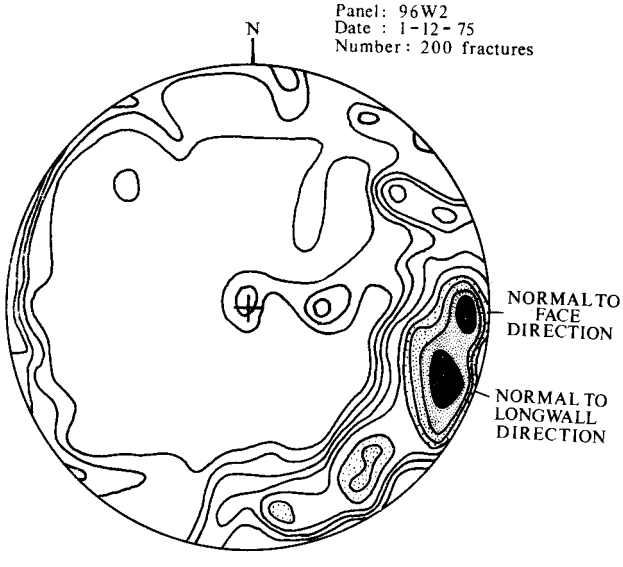

Fig. 15—Lower-hemisphere projection of the distribution of normals of 200 hangingwall fracture orientations in a typical breast-mined panel within an overhanded longwall configuration. For comparison, the normals to the actual face and longwall directions are also shown

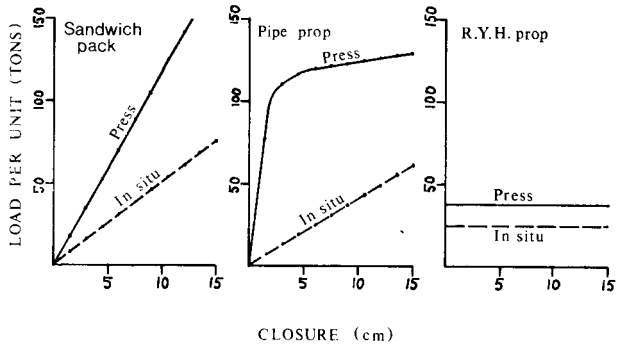

Fig. 16—Comparison between the press-tested versus *in situ* load-compression performances of a standard 60 cm by 60 cm sandwich pack, a 20 cm diameter resinated pipe-stick telescopic prop, and a 400 kN rapid-yielding hydraulic prop (initial stoping width 1 m)

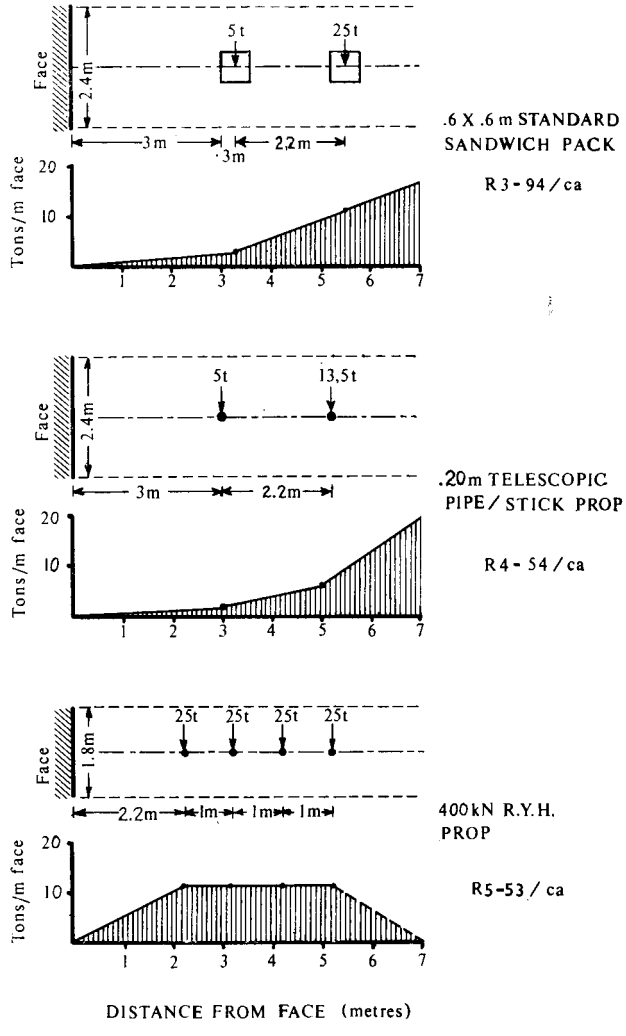

Fig. 17—An *in situ* load performance versus cost comparison of three stope-support systems. The distances from the face to the first support line are averages obtained from actual measurements

Fig. 16 shows the measured laboratory and *in situ* load–compression curves for three popular types of stope support. The measured *in situ* performance of the same support types in the vicinity of the face are shown in Fig. 17, together with 1979 cost estimates for each support type. It is clear that rapid-yielding hydraulic props are superior in providing good support close to the face, and at reasonable cost. It should be noted that neither the telescopic pipe–timber props nor the concrete–timber sandwich packs, both of which are supposed to be stiff support elements, afford good protection until considerable compression has occurred, at which time the face normally would already have advanced by about 12 m.

After the low initial stiffness characteristics of passive (installed with minimum prestressing) stope-support elements had been determined, and after a preliminary experiment by the Chamber of Mines had been completed at the Blyvooruitzicht Gold Mining Company, the concept of concentrated active (installed with significant prestressing) stope support was introduced at Western Deep Levels[13, 22]. The closely supervised experiment was conducted initially on two stope panels over a period of more than one year, and has since been considerably extended. Three rows of 400 kN rapid-yielding hydraulic props were installed at a density of 1,5 props per metre of face, with the first row approximately 2 m from the face. A fourth row, consisting of 1600 kN rapid-yielding hydraulic props called 'barrier props', was installed 1 m behind the third row of props at a density of 0,4 props per metre of face. Fig. 18 depicts a typical layout. No permanent stope support was installed other than the usual solid timber gully support for the local protection of strike and dip gullies. The initial aim of the experiment was *to prove that stope support in the back area of deep-level mines, i.e. where the ratio of depth below surface to maximum open span exceeds 5, did little to improve conditions at the face,* where most of the workers are exposed. The observed stability of the method is attributed to the narrowness of the hangingwall vertical tensile zone in the vicinity of the face of deep-level stopes. Theory predicts that the tensile zone extends typically to not more than about 1,5 m into the hangingwall at distances up to 20 m behind the face, so that massive caving is not expected to occur immediately behind the last row of props. A number of fairly large seismic events occurred within 100 m of the experimental panels, yet little rock-fall damage was observed following the events, indicating a stable support system. All in all, the experiment appeared to be highly successful from the point of view of strata control, and should, after further refinement,

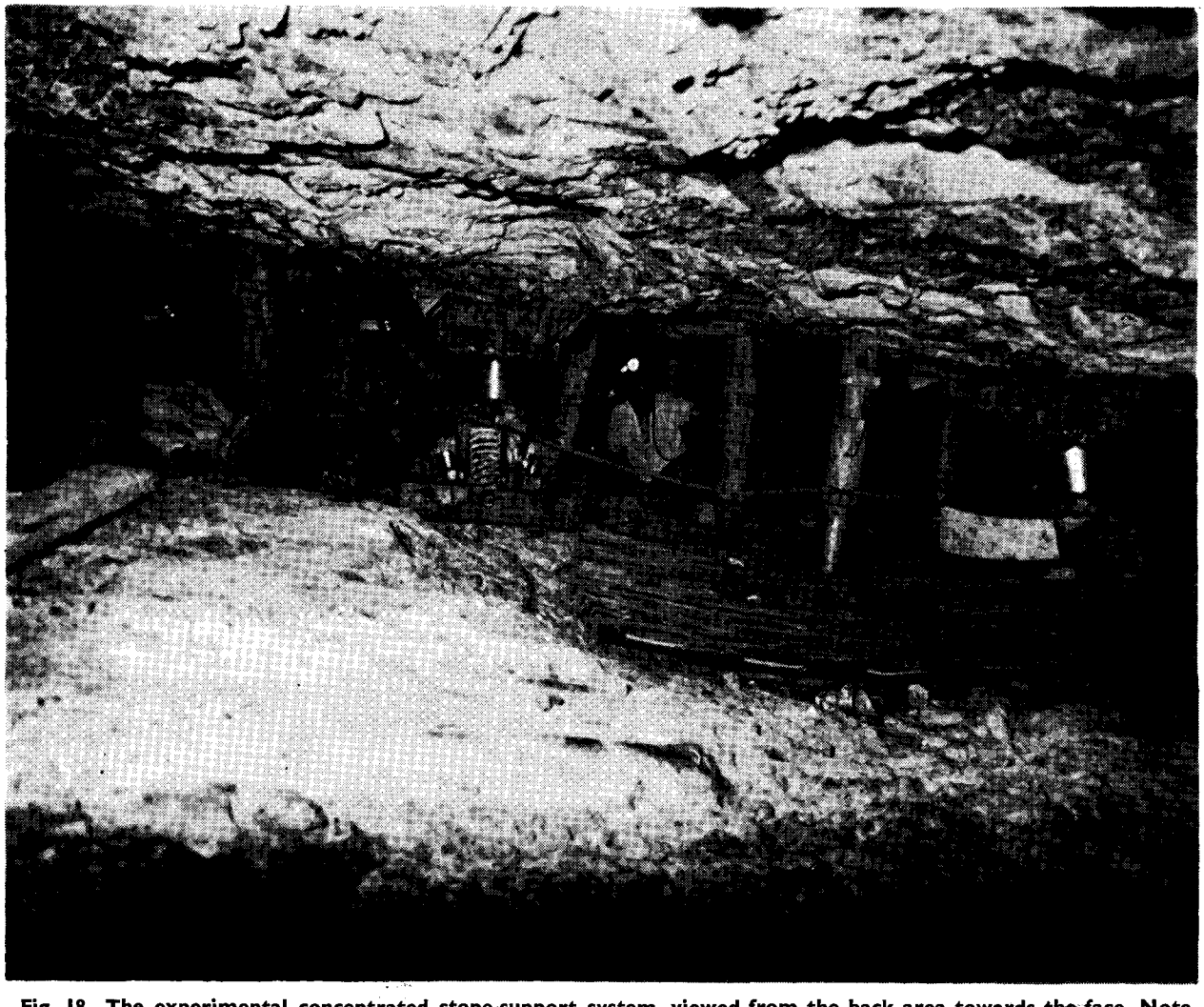

Fig. 18—The experimental concentrated stope-support system, viewed from the back area towards the face. Note the condition of the unsupported hangingwall in the foreground behind the last support line

result in an improved environment for people working in stopes.

On the basis of the above findings it was concluded that a reduction of the rock-fall hazard, which remains a complex problem, could best be achieved at this stage by an improvement of the stope support in the immediate vicinity of the face, and by avoidance of cross-fracture patterns in the hangingwall above where people are exposed. The use of concentrated active stope support and the elimination or reduction of leads between stope panels on longwalls appear to offer significant advantages in the control of the strata in narrow stopes.

Progress, which in the foreseeable future should lead to a better understanding of the rock-fall problem, is also being made in the investigation of the stability and behaviour of fractured ground.

Productivity and Rock Bursts

From the above description of some of the recent findings on the nature and extent of rock bursts and rock falls, it is clear that the problem is very much a mining problem in the full sense. It is integrated with many facets of the underground operations such as planning, method of mining, blasting, cleaning, material handling, support, ventilation, mechanization, and human factors. Attempts at improving productivity on a mine should therefore incorporate these rock-burst and rock-fall considerations.

Future Research

The South African gold-mining industry is currently investing an estimated one million rands per annum in rock-burst and rock-fall research alone, indicating its commitment to finding ways of improving the underground environment. It is by nature difficult to predict the directions that future research may take, but good progress can be expected in the development of the following.

(a) The prediction of rock bursts, to allow impending rock-burst sites to be identified and evacuated before the event takes place[23, 24].

(b) An understanding of the interaction between induced and geological discontinuities, and its relationship to the stability of the fracture zone surrounding underground excavations.

(c) Seismic methods for the delineation of dykes and faults ahead of mining excavations, to facilitate long-term planning.

(d) An understanding of the violent failure of large faults and dykes subjected to mining-induced stresses.

Conclusions

The rock-burst and rock-fall hazard is an important safety problem for some gold mines, contributing as much as 80 per cent to the overall accident-fatality rate. The extent of the problem is also increasing on an industry-wide scale, as evidenced by the statistics of the Prevention of Accidents Committee pertaining to South African gold mines.

Accidents resulting in a large number of fatalities receive wide publicity in the media, none of which can be described as beneficial to the industry. Though already-employed labour does not at present seem to be much influenced by such events, potential labour apparently is. Also, in both instances, the situation may be expected to become more fluid in future, as the average standard of living and education rises. Sufficient knowledge exists concerning the problem to allow for the implementation of effective control strategies aimed at reducing the rock-burst and rock-fall hazard.

It would therefore be in the interest of the mining industry to take the lead in adopting a more positive, long-term control strategy aimed, firstly, at reducing the occurrence of large rock-burst and rock-fall accidents and, secondly, at lowering the average accident-fatality rates on the mines worst affected.

Research should continue since the solution of some of the remaining questions should lead not only to a safer underground environment for men and machines in the longer term, but also to significant cost savings through more effective practices of strata control.

Acknowledgements

The author acknowledges the Management of Anglo American Corporation's Gold and Uranium Division and the Mining Operations Research Laboratory of the Chamber of Mines for their support of the rock-burst research effort in general and for permission to publish this paper. Members of the Rock Burst Research Project, who were responsible for many of the findings reported here, are also thanked for their contributions.

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Colloquium: Materiaalseleksie/Materials selection

The first colloquium of the Materials Engineering Specialist Division of the South African Institute of Mining and Metallurgy was held in Pretoria on 28th November, 1979. The subject was the Selection of Materials — Part I: Choice and Specification of Structural and Alloy Steel/Materiaalseleksie — Deel I: Keuse en Spesifisering van Strukturele en Legeringstaal.

Die colloquium is geopen deur die voorsitter van die spesialisteafdeling, dr. J. P. Hugo. Hy rig 'n spesiale woord van verwelkoming aan die sowat sewentig persone teenwoordig by die eerste colloquium van die spesialisteafdeling. Hy deel die colloquium-gangers mee dat daar beplan word om twee tot drie colloquia per jaar te hou en dat daar ook besoeke aan nywerhede in die vooruitsig gestel word.

Professor G. T. van Rooyen van die Universiteit van Pretoria lei die onderwerp in met 'n insiggewende lesing oor die beginsels van materiaalspesifisering. Hy wys daarop dat die verbruikers en vervaardigers die twee sleutelgroepe by materiaalspesifikasie vorm en dat die metallurg tussen die twee groepe staan. Die volgende aspekte is van besondere belang by materiaalspesifikasies:

(i) Die inhoudelike aspek van die spesifikasie
Voldoening aan die spesifikasies ten opsigte van chemiese analise is nie altyd 'n waarborg dat die materiaal aan die verlangde meganiese vereistes sal voldoen nie. Dit kan selfs gebeur dat 'n strenger spesifikasie 'n swakker materiaal vir 'n spesifieke toepassing lewer omdat sekere legeringselemente wat buite spesifikasie lê, dalk 'n voordeliger invloed kon gehad het.

(ii) Kwaliteitsversekering
Dit is uiters belangrik dat voldoende gehalteversekering toegepas word om te verseker dat daar wel aan 'n bepaalde spesifikasie voldoen word.

(iii) Die koste-aspek
Direkte koste wat voortspruit uit die vervaardiging van 'n defektiewe komponent is belangrik maar die gevolgskade wat kan ontstaan uit 'n artikel wat sou faal, moet ook in ag geneem word.

(iv) Die ontwerp van 'n onderdeel
Met 'n goeie ontwerp kan swak materiaal bv. gebruik word, maar aan die ander kant moet 'n strawwe spesifikasie nie neergelê word om vir 'n swak ontwerp te kompenseer nie.

Ten slotte wys professor Van Rooyen daarop dat die oordeelkundige gebruik van spesifikasies van die allergrootste belang is. Daar moet deeglik kennis geneem word van die vereistes wat die toepassing stel en die gebruiker moet goed onderlê wees in die kennis van die toepaslike materiaaleienskappe.

Dr Rocco de Villiers, of Iscor, read a paper on the rationalization of structural steel specifications in South Africa. His paper was aimed at the man in industry who has to select a steel for a specific purpose. This includes the designer, the works engineer, and the fabricator. He emphasized that the supply of steel has to be catered for by the steel merchant and service centre to an ever-increasing extent. Serious difficulties arise in their inventory planning, which could be greatly relieved if agreement could be reached on a basic range of specifications that can meet the bulk of the needs in South Africa. Specialized requirements could then be treated as such by designer as well as manufacturer. Dr de Villiers proceeded to outline the various types of wrought-carbon and low-alloy steels required with respect to a rationalization in their specifications.

The paper that was read by Mr J. R. Campbell, of British Engine Insurance Company, dealt with the problems facing the practising engineer who has to assess the acceptability of known defects in components in service. He discussed specific examples of known defects in structures with reference to code philosophy, and contemplated a fracture-mechanics approach to specifications.

Mr J. G. Price, of TUV Rheinland (SA), discussed management problems associated with the specification of a sophisticated stainless steel that had been used for a torque thickener at a gold mine. In order to initiate a proper quality assurance programme, it had been found necessary to adopt a computerized control mechanism on all aspects of materials supply, identification, and control.

After a lively discussion, the meeting adjourned for a social function in the clubhouse of the university.

(Dr R. J. Dippenaar: Rapporteur)