

**AN EVALUATION OF
CAVE MINING IN WEAK HANGINGWALL CONDITIONS
AT HARTEBEESTFONTEIN GOLD MINING COMPANY LIMITED.**

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**A project report submitted to the Faculty of Engineering,
University of the Witwatersrand, Johannesburg, in fulfilment of
the requirements to convert a Graduate Diploma in Engineering
to the Degree of Master of Science in Engineering.**

Stilfontein, 1993.

DECLARATION.

I declare that this project report is my own unaided work. It is being submitted for the conversion of a Graduate Diploma in Engineering to the Degree of Master of Science in Engineering in the University of the Witwatersrand, Johannesburg. It has not been submitted before for any examination or degree in any other university.



David Anthony Arnold

Signed this

Twenty Fifth day of October 1993

ABSTRACT.

A mining area at Hartebeestfontein Gold Mine, designated as the 5A North Block, is traversed by a geological alteration zone known locally as the "pseudo-breccia". The rockmass in this zone has been rendered friable, weak and prone to collapse as a result of this phenomenon. The area is also marginally payable, making stoping width control vitally important. Any failure of the hangingwall which dilutes the ore can render the mining area uneconomical.

The stoping method employed in the 5A North Block, and extensively elsewhere on the mine, is called "cave mining" or "caving". The method, which has been in use on the mine since the early 1960's, allows the back area to collapse behind a zone of face support which consists of either timber sticks or steel hydraulic props.

Various stope support strategies were investigated to deal with severe strata control problems arising from the prevailing poor hangingwall conditions. Serious consideration was given to backfilling as an alternative to caving, but the option was finally rejected for reasons outlined in this report.

The development of improved prop-based face support strategies and an evaluation of their effectiveness in combination with cave mining formed the main thrust of the project work. Some adaptations had to be made to suit the caving method, and these aspects are described in some detail in this report.

The report thus concentrates on the development and implementation of stope support strategies designed to cater for weak ground conditions, rather than dealing with the cave mining method per se. Nevertheless, a comprehensive review of the history and theory of cave mining is presented together with a fairly detailed evaluation of the backfill alternative.

Some quantitative assessments of strata behaviour under caving conditions have been made from time to time in the past. It was therefore deemed worthwhile to measure, quantify and interpret hangingwall rock movement at a number of stope sites within the pseudo-breccia alteration zone, for comparative purposes. Attempts were also made to measure the actual stresses acting in the immediate hangingwall of these stopes, and in the caved material.

Simplified representations of the observed structure, stress and deformation in the discontinuous rock surrounding stoping excavations was modelled using a two-dimensional, non-linear computer program. The influence of support and strata behaviour in determining caving mechanisms was investigated, highlighting the important role of lateral displacement along weak shear planes. A possible explanation, based on these investigations and on the observations of previous researchers, of the suitability or otherwise of the cave mining methodology under certain environmental conditions is advanced.

Relatively little modelling of the inelastic behaviour of rock, and its relationship with installed support in cave mining operations, has been possible to date because developments in the field of non-linear modelling technology which make this possible are comparatively recent. The modelling work described in this report was superficial, and pointed to the need for more thorough investigation and the necessity to calibrate inelastic models carefully to enable more meaningful interpretation of phenomena observed underground.

It was observed that the stability of the stoping excavations monitored during the course of this work was overwhelmingly dependant upon three factors: the local geological conditions (particularly with regard to the immediate hangingwall), the effectiveness of installed support (more particularly the extent of unsupported span between the support units themselves, and between the support units and the face), and, blasting practice (specifically the containment of hangingwall blast damage).

The achievement and nature of the cave was of significant, but of lesser importance. The classic model used to describe the behaviour of the face, hangingwall, footwall and internal support in stopes where caving is practised in other areas of the mine, seemed to have somewhat less application in weak ground conditions. Evidence of increased stress damage at, and ahead of the stope faces resulting from "hang-up" of the cave in weak ground where the observations were made, was not as overwhelming, or was masked, in comparison with areas on the mine where the strata are more competent.

Although it was recognised that the actual achievement of a cave behind the supported face area contributes significantly to the effectiveness and efficiency of the cave mining method, it was concluded that the contribution of face support and face advance may be even more meaningful. Overall stability and safety at the stope face was greatly enhanced by reducing face to support distances, and by distributing the support load more uniform over the hangingwall surface, particularly in conditions of weak and friable ground. Ensuring a continuous and rapid rate of face advance was found to be of equal importance. The containment of blasting damage was also found to play a significant role in contributing to hangingwall stability at the face.

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- Appendix 5B - Stress Interpretation Summary.**
- Appendix 5C - Stress Measurement Method And Problems.**

APPENDIX 6. CAVE CHARACTERISTICS.

- Appendix 6A - Cave Pressure Load Cell Calibration.
Device No.1 - 78A-42W N6 Panel.**
- Appendix 6B(i) - Cave Pressure Load Cell Readings.
Site No.1 - 78A-42W N6 Panel.**
- Appendix 6B(ii) - Cave Pressure Load Cell Readings.
Site No.2 - 78A-42W S4 Panel.**
- Appendix 6B(iii) - Cave Pressure Load Cell Readings.
Site No.3 - 78A-42W S4 Panel.**

APPENDIX 7. COMPUTER MODELLING.

- Appendix 7A(i) - Intuitive Caving Model Input Parameters.
Intact Hangingwall Model.**
- Appendix 7A(ii) - Intuitive Caving Model Input Parameters.
Rigid Hangingwall Model.**
- Appendix 7A(iii) - Intuitive Caving Model Input Parameters.
Pseudo-Plastic Model.**
- Appendix 7B - Caving Simulation Model Input Parameters.
Dynamic Model.**

AN EVALUATION OF CAVE MINING IN WEAK HANGINGWALL CONDITIONS AT HARTEBEESTFONTEIN GOLD MINING COMPANY LIMITED.

1. INTRODUCTION.

Hartebeestfontein is a deep gold mine currently exploiting the Vaal Reef horizon in the Klerksdorp district of the Western Transvaal. Figure 1 shows the locality of the mine.

The region is geologically complex and the reef is heavily faulted, resulting in scattered mining techniques being employed. Four major fault zones, which trend approximately northeast/southwest, divide the property into four mining limbs. Figure 2 shows current mining on the Vaal Reef at Hartebeestfontein Gold Mine.

The Vaal reef, a gold and uranium bearing conglomerate, is tabular, varies between a few centimetres and a few metres thick, and has a variable, but generally shallow ($\pm 10^\circ$) dip. In the deeper north-western portion of the mine (from 1,8km down to 2,3km below surface) the reef lies in a graben block where the rocks are folded into a rough syncline with a shallow plunge to the south-west.

A mining area known as the 5A North Block lies in the northern portion of the most westerly limb, between Hartebeestfontein No.5 Shaft and the boundary with the now closed Stilfontein Gold Mine. The block is traversed by a wide east/west trending geological alteration structure, known as the pseudo-breccia zone, which, in addition to its heavily faulted nature, has rendered the rock friable, weak and prone to collapse. The area is also marginally payable, making stoping with control vitally important as any failure of the hangingwall which dilutes the ore will render the area uneconomical.

The stope support strategies described in this report were developed to deal with strata control problems arising from these poor hangingwall conditions. Some relatively minor modifications of the support layout were required to enable these strategies to be integrated into the stoping method employed extensively elsewhere on the mine which allows the back area to collapse, and which is known locally as "cave mining". As an alternative to caving, serious consideration was also given to backfilling, but this option was discarded following detailed evaluation and comparison of the two methods.

Cave mining with steel props was pioneered on the neighbouring Stilfontein Gold Mine in 1961, and was first tried on Hartebeestfontein Gold Mine shortly thereafter in 1962. A modification, using timber sticks in place of steel props, was introduced on the then adjoining Zandpan Gold Mine between 1964 and 1965. Following the merger of Zandpan and Hartebeestfontein in 1972, caving has been successfully employed as the principle stoping method on Hartebeestfontein Gold Mine, using both sticks and props.

Although quantitative assessments of in-situ strata behaviour under caving conditions have been made from time to time, relatively little modelling of discontinuous behaviour of the rockmass surrounding stoping excavations and its relationship with installed support units has been carried out to date. Developments in the field of non-linear modelling technology, which make this possible, are comparatively recent.

Monitoring and measurement stations were established at a number of test sites in stope panels situated within the pseudo-breccia zone to provide accurate information against which to calibrate numerical models. The structure of the hangingwall was mapped, measurements of closure, ride and stress were undertaken and the extent, nature and effect of the cave was monitored. These observations were interpreted and related to the various stope support geometries in place at each site at the time, using non-linear computer modelling techniques.

In this way, an attempt was made to assess the effectiveness of the support strategies which were developed during the course of this work. The influence that the cave mining method had on the success of these strategies was also assessed.

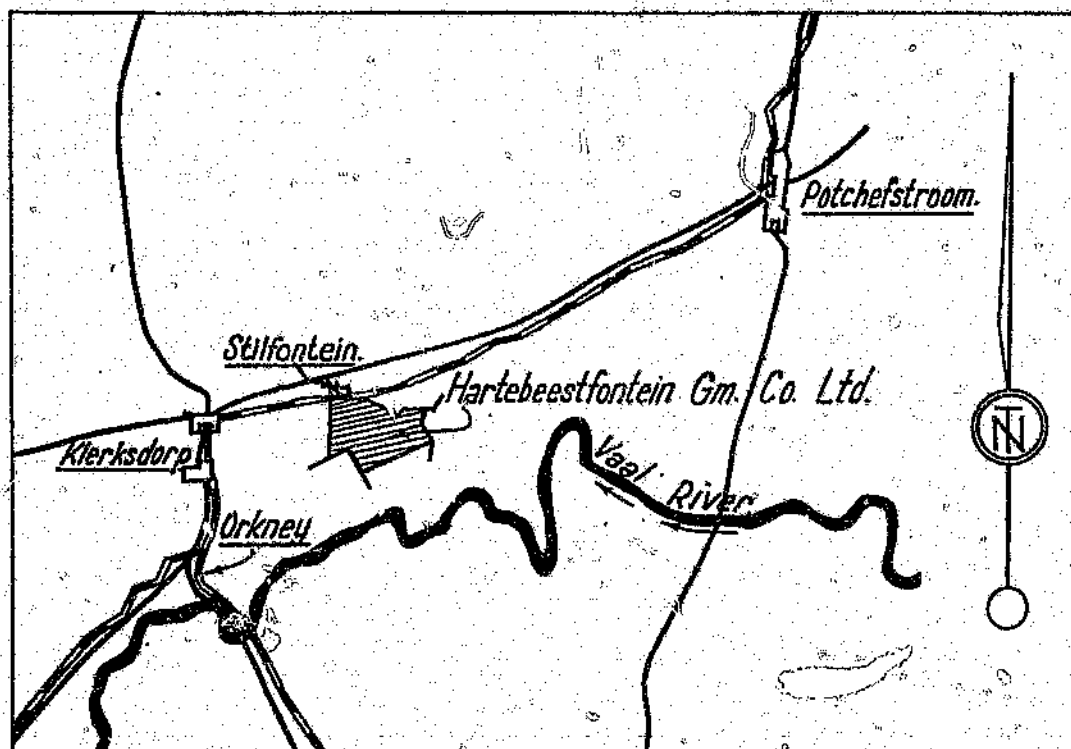


Figure 1 - Plan showing the Locality of Hartebeestfontein Gold Mine.

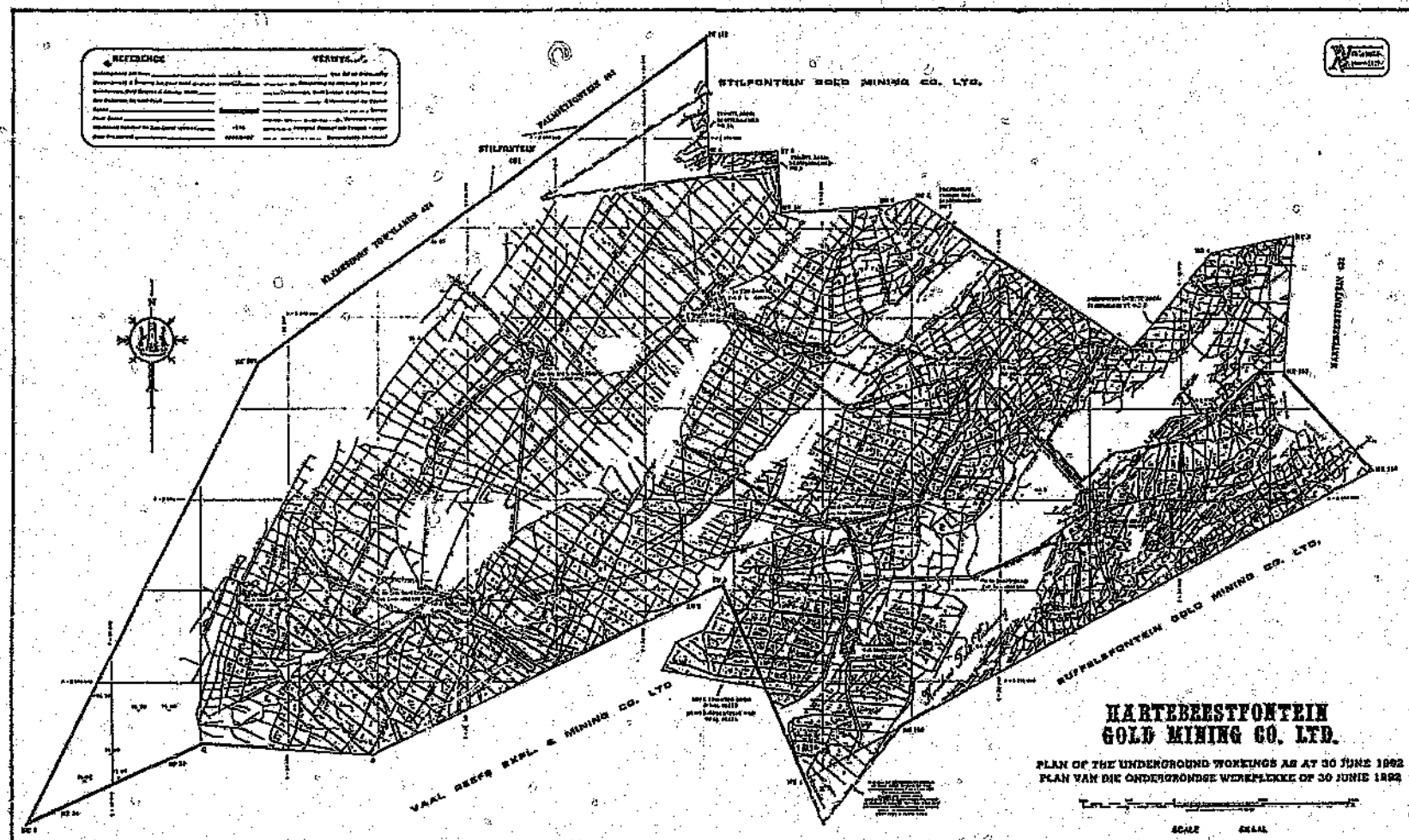


Figure 2 - Plan showing Mining on the Vaal Reef Horizon at Hartebeestfontein Gold Mine.

2. GEOLOGY OF THE KLERKSDORP GOLDFIELD.

2.1 STRATIGRAPHY.

The gold-bearing reefs of the Klerksdorp goldfield occur in the Dominion Group (the Dominion Reefs), the Witwatersrand Supergroup (many reefs, including the Vaal Reef), the Ventersdorp Supergroup (the Ventersdorp Contact Reef), and the Transvaal Sequence (the Black Reef). An abbreviated stratigraphic column showing this sequence is presented in Figure 3.

The Dominion Group, which rests on the archaean granite basement, contains two conglomerate reefs within the Renosterspruit Quartzites, both of which were previously mined. The succession is more than 900m thick west of Klerksdorp, thinning down northwards to a narrow strip of porphyritic lavas northwest of the small town of Hartbeesfontein.

Unconformably overlying the Dominion Group is the Witwatersrand Supergroup, which is more than seven kilometres thick. Gold occurs in varying concentrations in a number of relatively thin conglomerate reefs within this ancient sedimentary succession. The supergroup is divided into a lower group, the West Rand Group, and an upper group, the Central Rand Group. The lower group is dominantly argillaceous with some arenaceous horizons, while the upper group is composed almost entirely of quartzites and conglomerates.

Some reefs of the lower, West Rand Group have been profitably mined in the Klerksdorp Goldfield. The most important include the Elandsplaagte or Government Reefs of the Government Subgroup and the Afrikaner Reefs of the Jeppestown Subgroup.

The largest proportion of economic reefs occur in the upper, Central Rand Group, with the Vaal Reef being by far the most important in modern times, and the subject of this report. Other reefs within this group include the Ada May, the Commonage, the Livingstone, the Gold Estates and the Elsburg Reefs. Figure 4 shows the relative positions of reefs and markers within the Witwatersrand Supergroup.

The lavas of the Ventersdorp Supergroup, with the Ventersdorp Contact Reef at their base, lie unconformably on the Witwatersrand Supergroup. The Ventersdorp Contact Reef conglomerate is derived from erosion products of Witwatersrand sediments. The lavas decrease in thickness from more than 1500m in the west where they outcrop on surface, to only tens of metres in the east.

The Black Reef, which rests directly on the Ventersdorp Lava, is the basal member in the Black Reef Formation of the Transvaal Sequence. Gold in this conglomerate was also derived from reworked Witwatersrand reefs. This formation is overlain by the Chuniespool Group dolomites of the Transvaal Sequence which outcrop on surface over almost the whole of the lease areas occupied by current mining companies (Stilfontein, Hartebeestfontein, Buffelsfontein and Vaal Reefs Gold Mines). They increase in thickness from zero in the west to approaching 1000m in the east. The upper groups of the Transvaal Sequence are represented to the east of the goldfield.

Finally, an outlier of the Karoo Sequence, consisting of Dwyka Tillite, Eccca Shale and Eccca Sandstone, occurs on surface across the north of the Hartebeestfontein Gold Mine property.

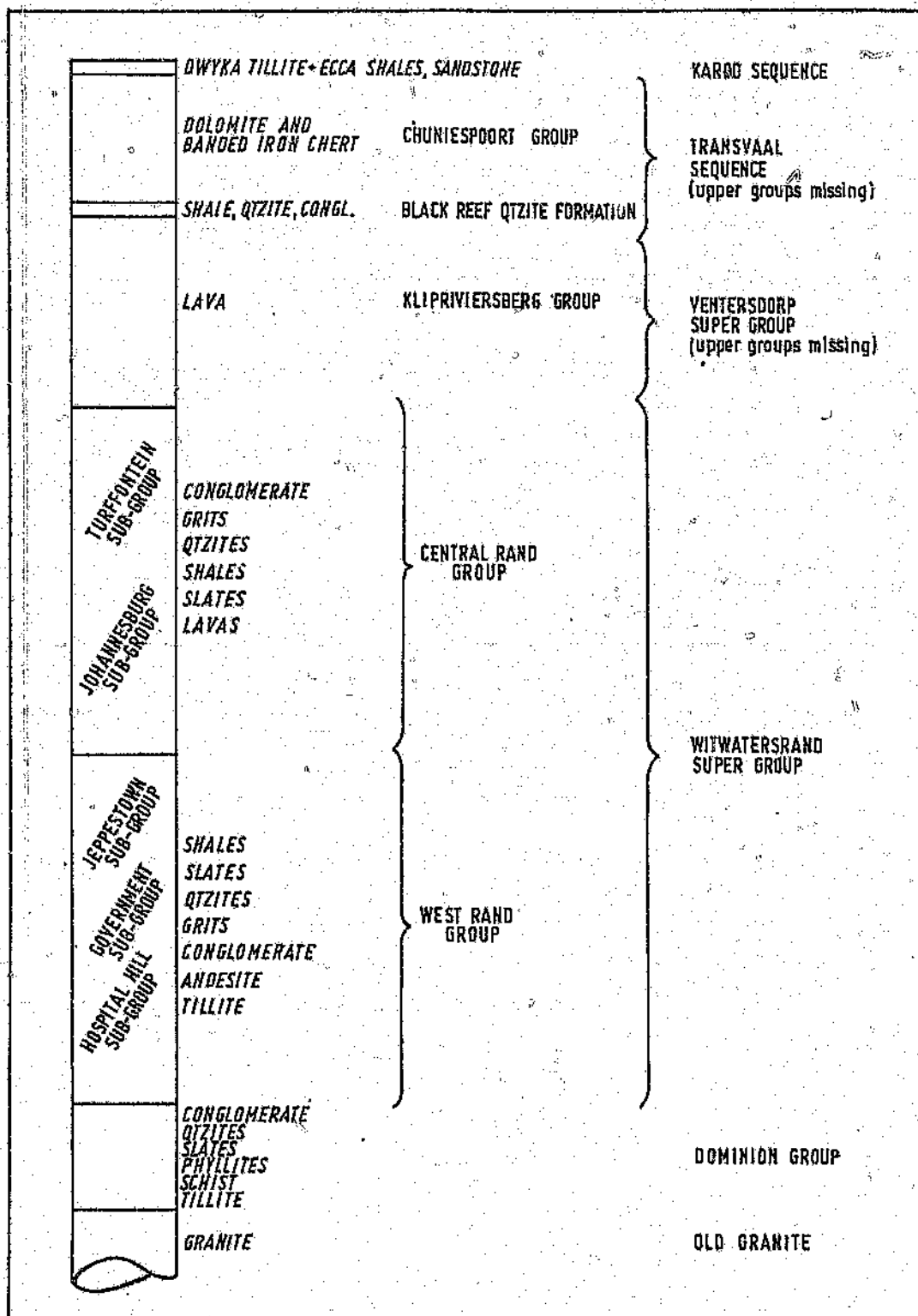


Figure 3 - Abbreviated Stratigraphic Column of the Klerksdorp Goldfield.

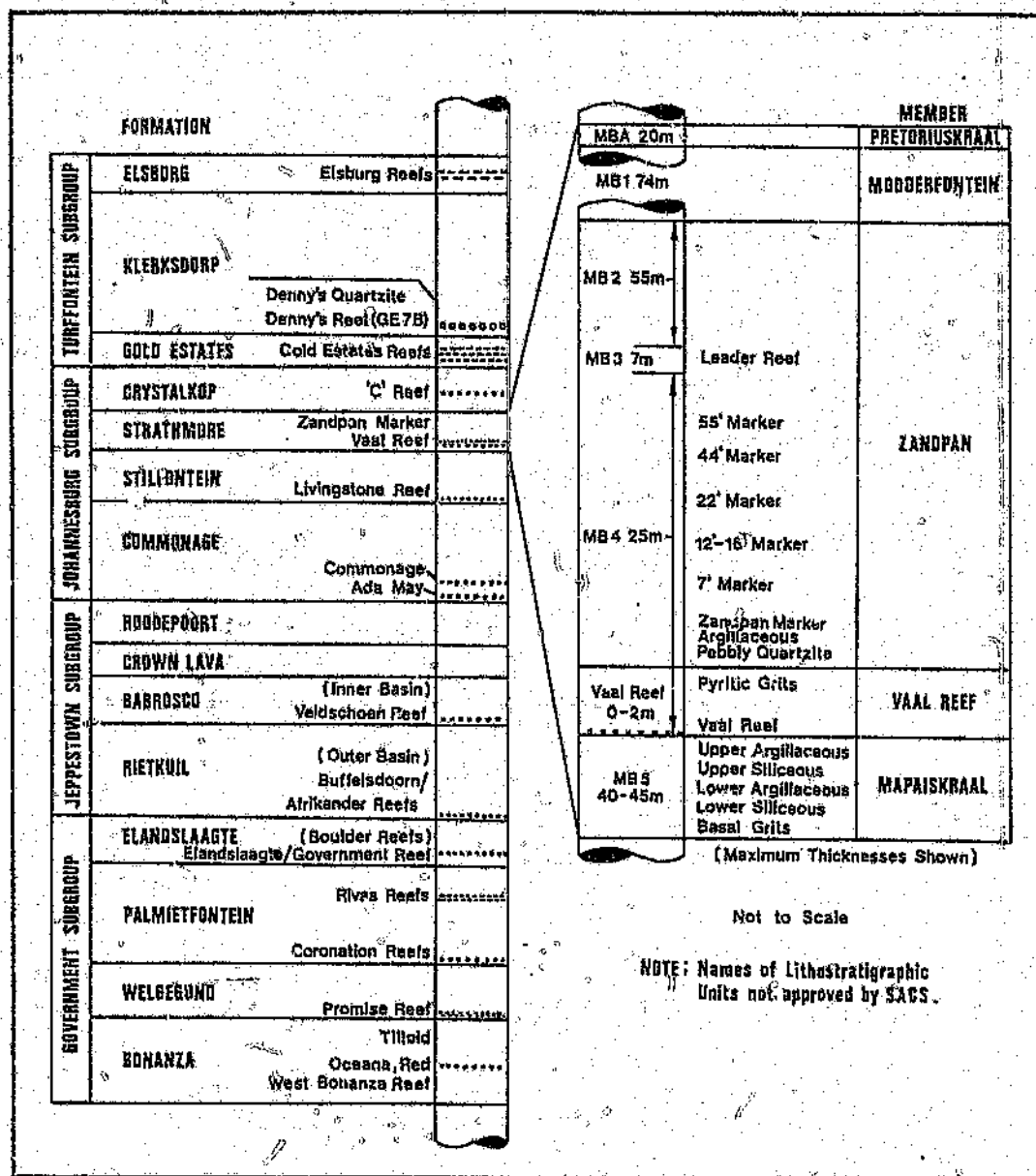


Figure 4 - Relative positions of Reefs and Markers within the Witwatersrand Supergroup.

This elongated depositional feature, which runs from west to east and rests on the dolomites, is approximately 2,5km wide and 70m thick. Thus, major portions of the South African geological record spanning more than three billion years exist at Hartebeestfontein Gold Mine. Figure 5 is a plan of the present day goldfield showing the surface outcrops of the major geological units occurring in the area.

2.2 STRUCTURE AND FAULTING.

The Klerksdorp goldfield is situated on the northwestern rim of the main Witwatersrand depositional basin which forms a 350km x 150km arc occupying the southern Transvaal and northern Orange Free State. Although a number of other reefs, such as the VCR, Eisburg and "C" reefs are mined around Klerksdorp, the Vaal Reef is today the most important economic member of this field. The Vaal Reef basin occurs due east of the town of Klerksdorp, extending 30km along its long axis (northeast-southwest) and is 15km wide. Apart from a small upthrown section which outcrops on surface to the northwest, the Vaal Reef extends from approximately 500m below surface to depths exceeding 3000m. Figure 6 is a locality plan showing the lease areas of the mines currently working the Vaal Reef, giving some indication of the extent of the basin.

The basin is structurally complex, being dissected and intersected by numerous intrusive features. The major fault systems have a consistent north/northeast-south/southwest trend, giving rise to a series of horst and graben structures which have rendered the Vaal Reef accessible to mining. Displacements of many hundreds of metres are common.

On Hartebeestfontein Gold Mine the major fault zones are the Buffelsdoorn Fault (1000m downthrow to the southeast), the No.5 Shaft Fault (450m downthrow to the northwest), the HB 18 Fault (200m downthrow to the southwest), and the seismically active Kromdraai Fault complex (1000m downthrow to the northwest). These faults divide the mine into four northeast-southwest orientated mining blocks. Figures 7 and 8 show the geological structure of the Vaal Reef horizon on Hartebeestfontein Gold Mine.

The Vaal reef in the western blocks (called the "deep blocks" to distinguish them from the shallower blocks to the east), lies in a syncline with limbs dipping at less than 10° and a shallow plunge to the southwest. The 5A North area, containing the pseudo-breccia zone, and the subject of this report, lies at the northern extremity of the most westerly block. Figure 9 is a section looking east showing the structure of the Vaal Reef horizon in this area.

In addition to these major faults, the Vaal Reef basin is broken into numerous smaller blocks by lesser faults of varying age, displacement and orientation. Most of the faulting is of post-Witwatersrand Ventersdorp and Transvaal age, with many faults being active throughout these epochs. Some of the more minor faulting continued into the Karoo period. The faulted nature of the reef necessitates extensive mapping and diamond drilling to resolve structure for mine planning purposes.

Intrusives, mostly dykes, are also common. They are often related to faults of varying displacements, thereby adding to the structural complexity. Due to the differences in their composition, strength and orientation relative to the layered Witwatersrand sediments, problems are commonly encountered where mining has to be carried out adjacent to, or in them. Their width varies from a few centimetres to tens of metres, and most are continuous over long distances.

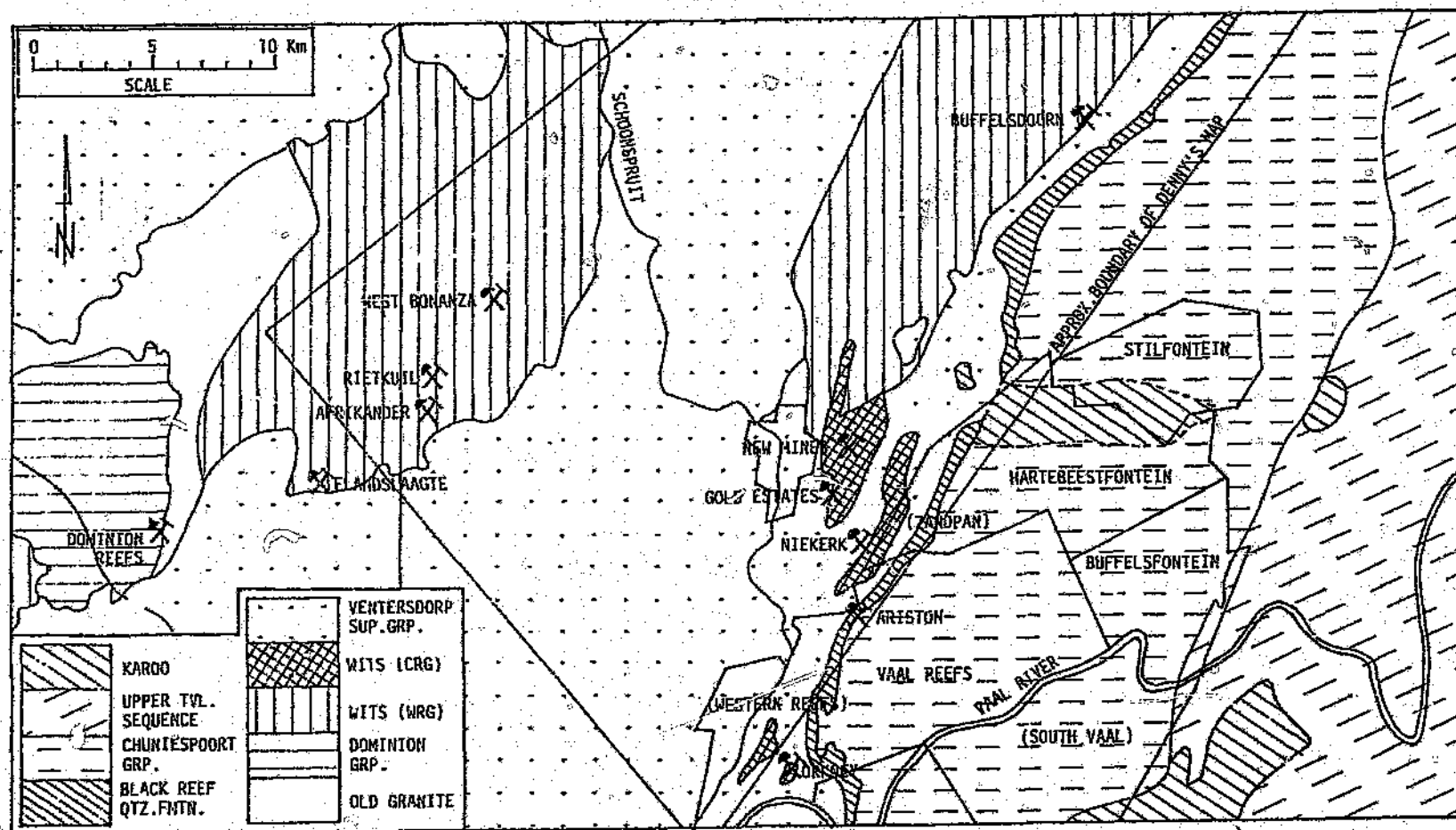


Figure 5 - Plan of the present day Goldfield Showing the Surface Outcrops of the Major Geological Units.

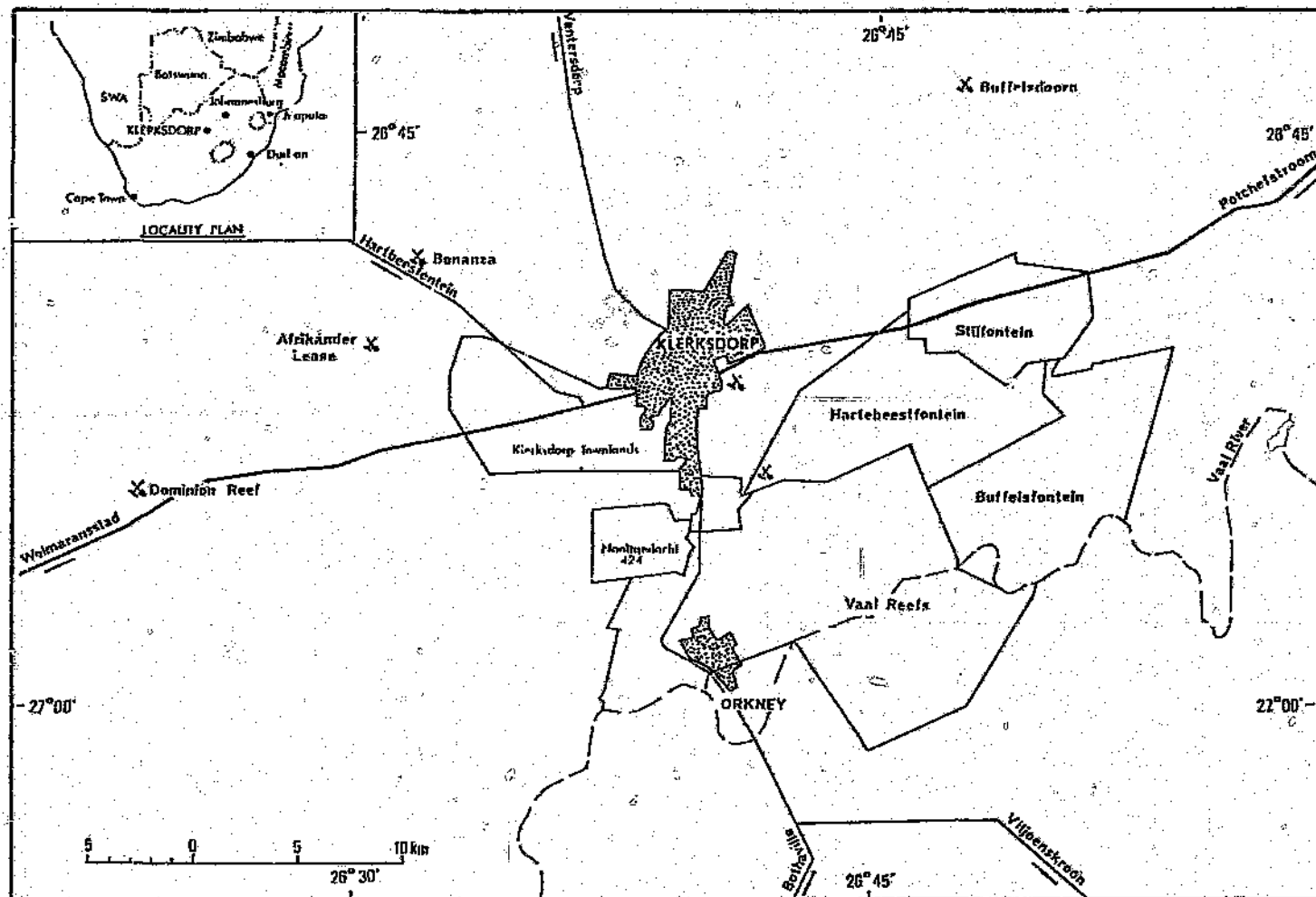


Figure 6 - A Locality Plan showing the Lease Areas of the mines currently working the Vaal Reef.

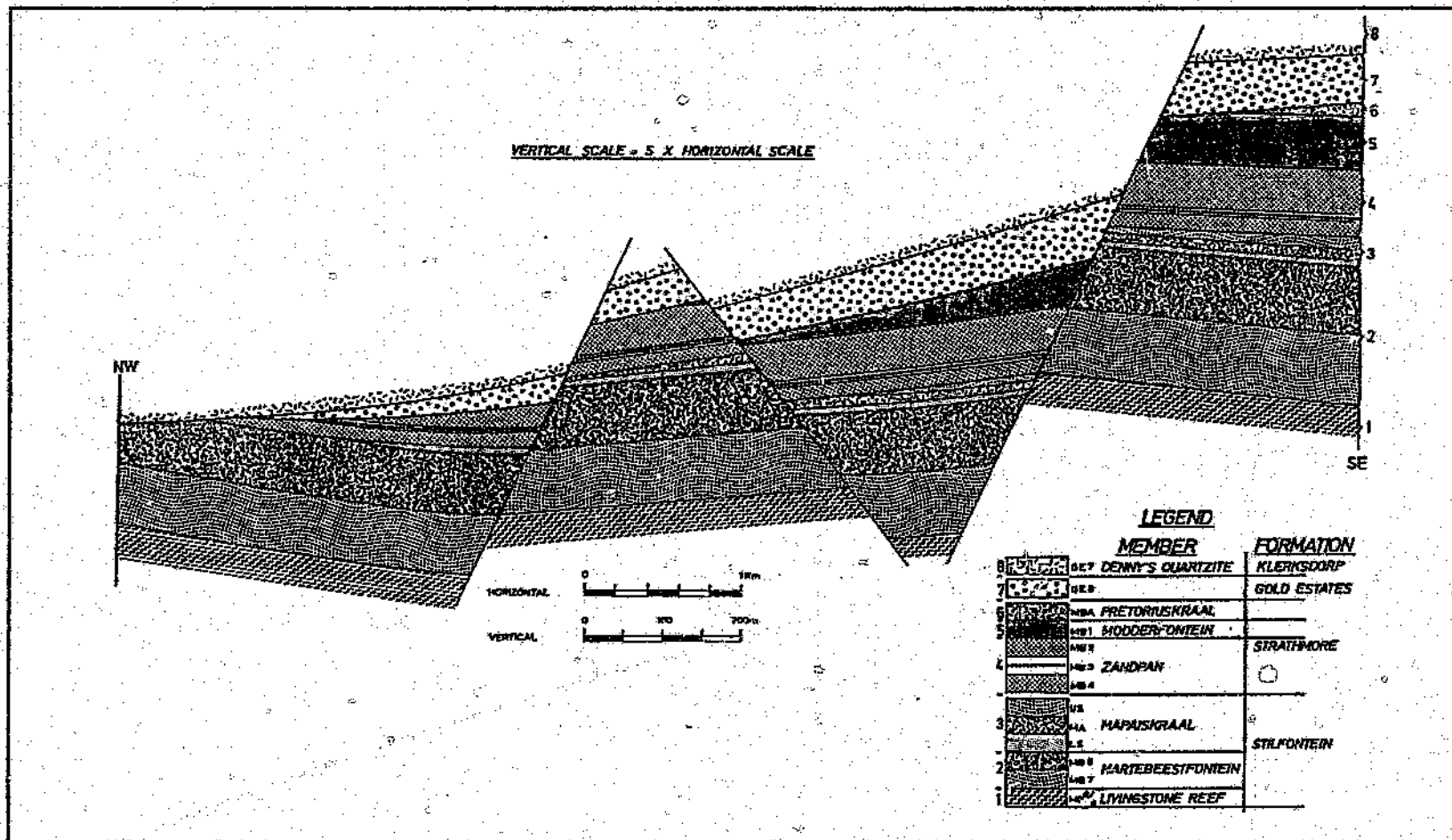


Figure 8 - Sectional view looking north showing the Geological Structure of the Vaal Reef Horizon on Hartebeestfontein Gold Mine.

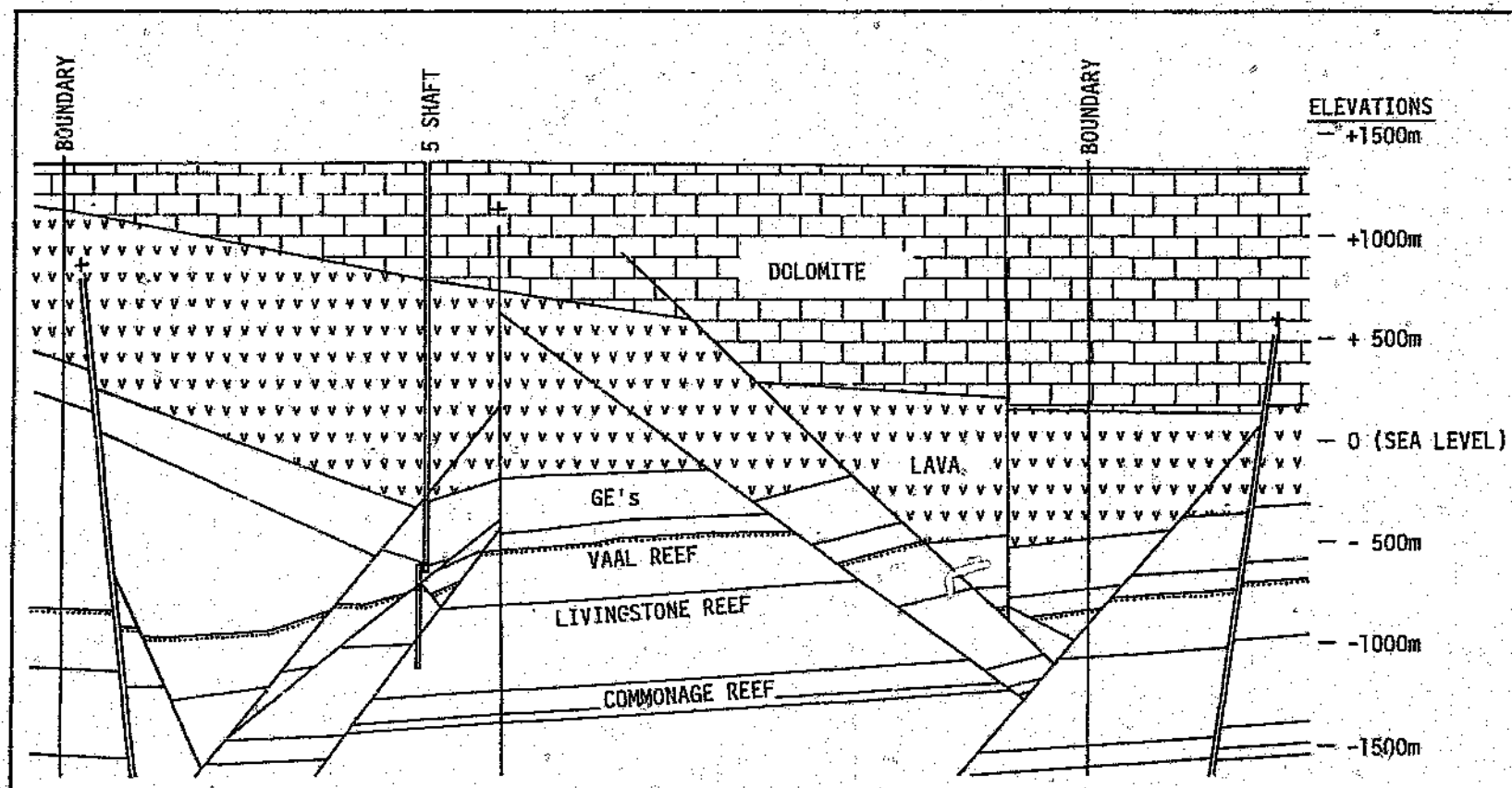


Figure 9 - Sectional view looking east showing the Geological Structure of the Vaal Reef Horizon in the 5A North Area.

Six intrusive types of differing age and composition have been recognised. They are, from oldest to youngest:-

- Quartz Diabase - Ventersdorp age.
- Ilmenite Diabase - Post Ventersdorp age.
- Carbonatite - Post Transvaal age.
- Pilanesberg Dolerite - Post Waterberg age.
- Karoo Dolerite - Post Karoo age.
- Kimberlite (Olivine Lamprophyre) - Post Karoo age.

Whereas the pre-Waterberg intrusives are usually associated with major faulting, the post-Waterberg intrusives are not.

2.3 THE VAAL REEF.

By far the greatest proportion of economically minable gold in the Klerksdorp goldfield occurs in the Vaal Reef, a narrow conglomerate band seldom exceeding 50cm in thickness, but more often than not only 10cm thick. The reef consists of mainly quartzose pebbles, no more than 22mm in diameter, in a siliceous matrix.

Due to structural complexity, the average strike and dip of the reef varies greatly over the Hartebeestfontein lease area, but is fairly constant in localised areas with only gradual changes. Dips in the deep block are generally flat, of the order of 10° or less. However, the base of the syncline is irregular with shallow domes and depressions causing rapid changes in strike.

The Vaal Reef lies unconformably on, and truncates the entire underlying Mapaiskraal (MB5) footwall succession which comprises Upper Argillaceous, Upper Siliceous, Middle Argillaceous, Lower Siliceous and Basal Grit zones. To the northwest, in the vicinity of the Vaal Reef sub-outcrop against the Gold Estates Quartzites, the MB5s are totally absent and the Vaal Reef rests unconformably upon MB6 Quartzites. Figure 10 is a plan showing the sub-outcrop contours of the footwall units against the Vaal Reef unconformity.

Lower Siliceous and Basal Grit zones comprise the reef footwall north of No.5 Shaft. The Lower Siliceous quartzites are uniformly fine grained and trough cross-bedded. They are 4m-5m thick and have a strength range of between 200MPa and 300MPa. The Basal Grit is a small-pebble, oligomictic, quartzose conglomerate or pebbly quartzite, varying in thickness from 15cm to 1,5m which grades into the Lower Siliceous quartzites. This marker bed unconformably overlies the Hartebeestfontein (MB6) quartzites which accommodate most of the footwall access development in the area. The Mapaiskraal unconformity marks this transition.

The immediate reef footwall of the stopes in the 5A North area consists of well bedded Lower Siliceous quartzites. Individual beds making up the sequence rarely exceed 1m in thickness, typically being between 20cm and 70cm thick. The intervening bedding planes are predominantly smooth, planar and extensive, and invariably contain some degree of argillite filling material which lubricates them and makes them prone to differential lateral movement under the influence of induced horizontal stoping stresses. This often results in the progressive closure of shallow footwall excavations such as boxholes and gullies, particularly centre gullies, as mining proceeds.

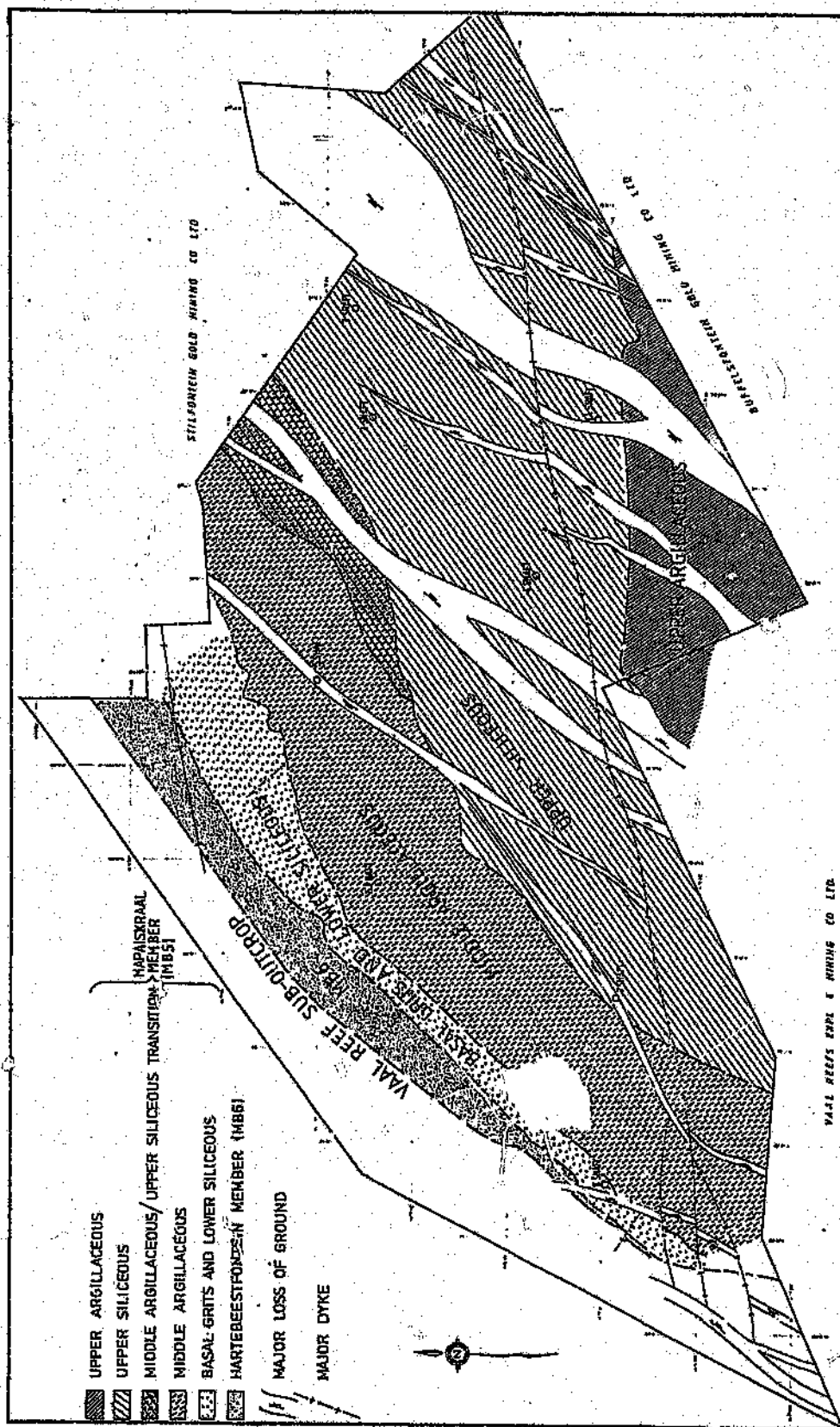


Figure 10 - Plan showing the Sub-Outcrop Contours of the Footwall Units against the Vaal Reef.

The Vaal Reef is overlain by a siliceous, often trough cross-bedded quartzite, which frequently contains the Pyritic Quartzose Grits. An erosion surface (the Zandpan unconformity) which marks the base of the overlying Zandpan member (MB4, MB3 & MB2 sequences) may erode the Vaal Reef member locally. Consequently the Vaal Reef has a thickness varying from nil to a maximum of about 1,5m. In the 5A North area the Vaal Reef conglomerate and quartzite horizon is about 40cm thick.

Figure 11 is a generalised section showing the Mapaiskraal, Vaal Reef and Zandpan unconformities, and their effect on the horizons comprising the immediate hangingwall and footwall of the Vaal Reef.

The MB4 sequence of the Zandpan member forms the main hangingwall of the Vaal Reef. It is this unit which is involved in caving operations at Hartebeestfontein Gold Mine, and which has been the subject of measurement and observation at No.5 Shaft.

At its base the Zandpan member consists of a coarse-grained, generally argillaceous, sometimes pyritic, pebbly quartzite. Over most of the property this grades into an overlying polymictic conglomerate containing shale, quartz, chert, quartzite and acid lava pebbles - the so-called "Zandpan Marker". In the east the Zandpan Marker lies directly on the Vaal Reef. However, it becomes less prominent westwards where it is separated from the reef by an increasing thickness of quartzites and grits, and is in fact not developed at all in the 5A North area.

In the 5A North area, the immediate hangingwall quartzites of the Vaal Reef area are coarse to medium grained, with some interbedded shaly phases. They are generally friable and weak, and are characterised by numerous well-defined parting planes which lack cohesion, allowing the hangingwall to collapse over relatively short spans.

Above this follows a sequence of well-bedded argillaceous, slightly argillaceous and arenaceous medium-grained to coarse-grained quartzites with some conglomeratic, gritty layers, these latter forming marker beds (7ft, 12ft, 22ft, 44ft and 55ft markers) with fairly distinct characteristics and relatively constant stratigraphic positions. They are extensively used for structural interpretation, the most prominent lying at about 16m above the Vaal Reef (the 55 foot marker). Figure 12 is a section showing detail of the typical Vaal Reef hangingwall sequence in the 5A North area.

2.4 PSEUDO-BRECCIA.

The 5A North Block is traversed by zones of lithologic alteration, known as pseudo-breccia. This has caused the affected rock to become friable, weak and prone to collapse, resulting in serious hangingwall control problems.

Pseudo-breccia is the term used locally to describe zones in the rock mass displaying alteration of the composition and appearance of the rock material associated with various intensities of fracturing. Areas displaying these features are sometimes also referred to as "concrete fissure zones", "sugary quartzite zones", "bleached zones", "leached zones" or "brecciated zones".

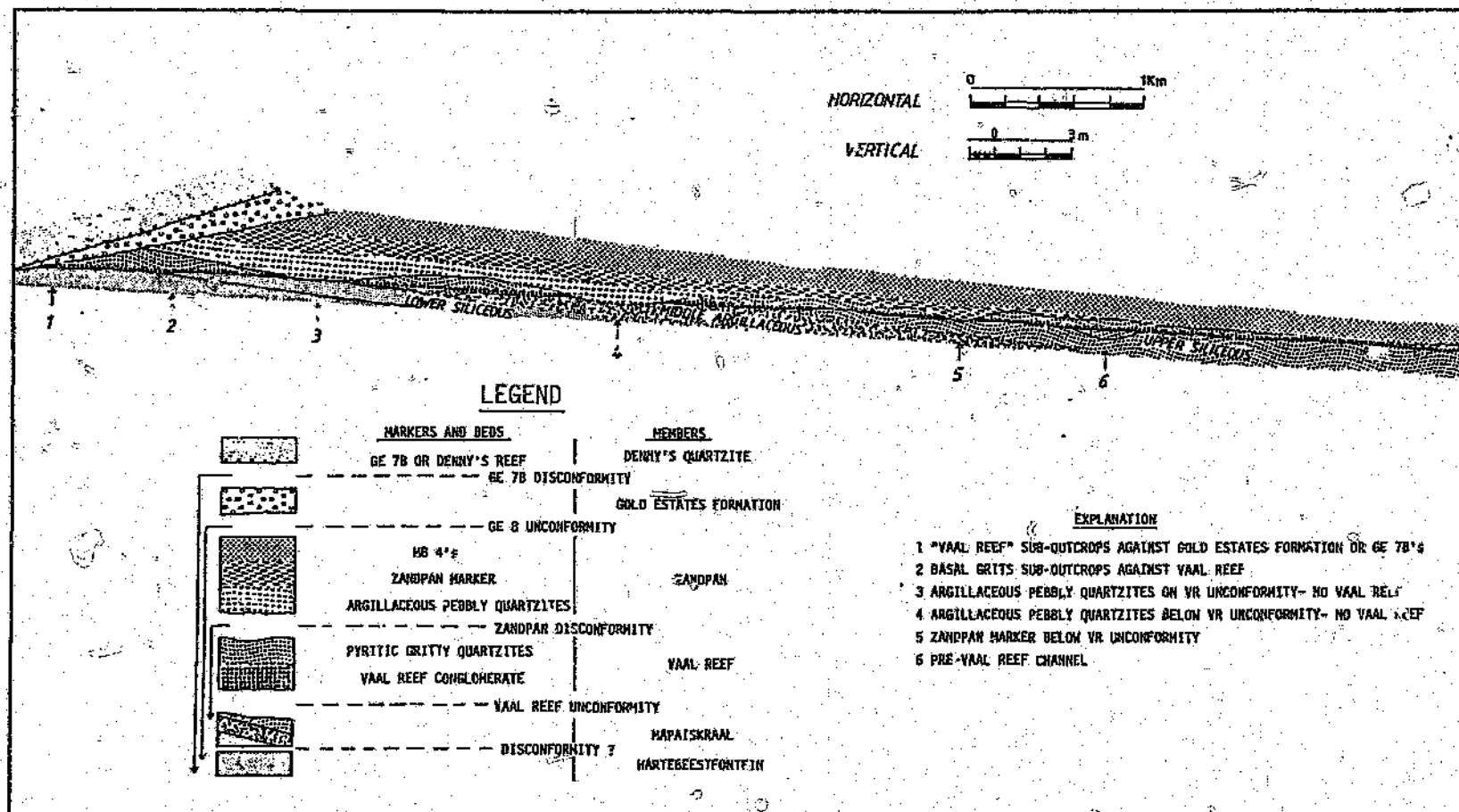


Figure 11 -

Generalised section showing the Basal Grits, Vaal Reef and Zandpan unconformities, and their effect on the Horizons comprising the immediate hangingwall and footwall of the Vaal Reef.

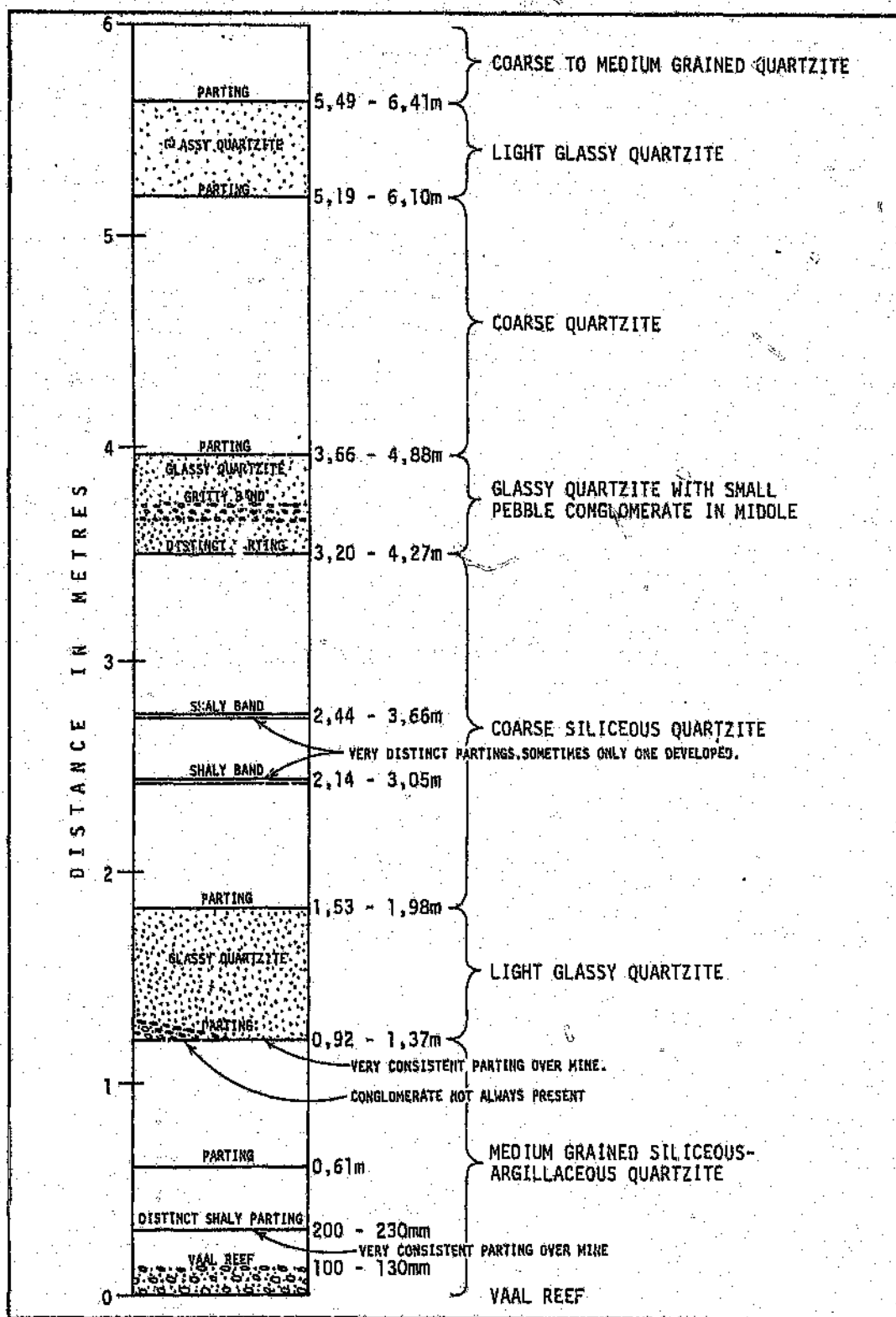


Figure 12 - Typical Vaal Reef hangingwall sequence in the 5A North Area.

Pseudo-breccia fracture zones may or may not be associated with displacement, and may or may not be confined to boundary fractures. Alteration is witnessed as a bleached discolouration relative to the colour of the surrounding rock, and a marked reduction in the strength and integrity of the rock material. Pseudo-breccia is often soft and crumbly to the touch.

Alteration generally propagates along the fractures. Depending on the fracturing and alteration, individual manifestations may be inter-connected or totally separated. The two requisites for a fracture zone to be classified as a pseudo-breccia are that it should display, firstly, fracturing and, secondly, alteration.

These features occur throughout the footwall, reef and hangingwall successions of the Vaal Reef. From drilling information it would, however, appear that the footwall is more vulnerable to alteration. Nevertheless, it is the marked detrimental effect these phenomena have on the strength of the immediate reef hangingwall which is of over-riding importance when considering the support and stability of stoping excavations situated within pseudo-breccia zones.

The distribution of pseudo-breccia in the immediate vicinity of the Vaal Reef in the 5A North Area has been predicted by Theart⁽³⁾ using an influence projection of all recorded observations of these features. Figure 13A shows this predicted distribution in terms of low, moderate, high and very high probability of occurrence. The distribution could, however, be regarded as biased, because sampling may have been influenced by the non-random and uneven distribution of exposures in underground workings and borehole intersections shown in Figure 13B. A strong correlation exists between the derived distribution, which was not prejudiced by any theory on the origin of the phenomena, and the independently mapped structural trends shown in Figure 13C.

Three zones, or prominent controls on the distribution of the pseudo-breccia phenomena in the 5A North area, were identified:-

- a) A zone with a high probability of containing pseudo-breccia is related to a prominent east-west direction of fracturing and faulting. The southern margin of this zone is sharply demarcated by a fault with a 20m throw, with the northern margin more diffuse. The extension of this feature correlates well with known troublesome exposures of pseudo-breccia in stoping operations north of No.5 and No.6 Shafts.
- b) A second probable pseudo-breccia zone is related to the Scott Shaft Dyke, and trends slightly more east-northeast-west-southwest, possibly converging with the first zone further to the west of the study area. The Scott Shaft Dyke, which postdates the Ventersdorp volcanics and the Chuniespoort dolomites, is probably of Hekpoort age.
- c) A third feature which apparently influences the distribution of pseudo-breccia phenomena, is the axial trend of an open syncline with a northeasterly strike. It is possible that an incipient axial plane cleavage was dilated between the first and second zones which aided the formation of the pseudo-breccias. Only the part of this axial planar trend north of the first zone is affected.

The detrimental effect of pseudo-breccia on mining, and in particular on stoping operations, is well known from experience in other areas of the mine; notably the 34, 35 and 36 lines northwest of No.5 Shaft mentioned previously. Even a perfunctory appraisal of the stoping

plans gives an insight into the strata control problems experienced in these areas. The re-development and changes in stoping rate and stoping direction caused by the presence of pseudo-breccia can be clearly seen demarcating the zone in Figure 14.

The more serious consequences of hangingwall weakness experienced in this area were an inability to prevent falls of ground at the face, and the pre-disposition of the cave to over-run the back line of support. This, understandably, led to the installation of timber packs at the face and between the rows of props. In some instances packs were installed right up against the face and spaced within less than 1m of each other in an effort to maintain stability. Back-area caving was effectively prevented by the pack support, and the long delays, which were incurred while pack support was being installed, adversely impacted upon the attainment of regular and adequate rates of face advance. The upshot was a complete breakdown of cave mining routine, and a cycle of self-perpetuating hangingwall instability.

Clearly, there was a need to address the real problem of providing adequate areal support coverage close to the face to deal with these weak hangingwall conditions, while at the same time retaining the substantial advantages of back-area caving. How this was successfully achieved will be discussed later in this report.

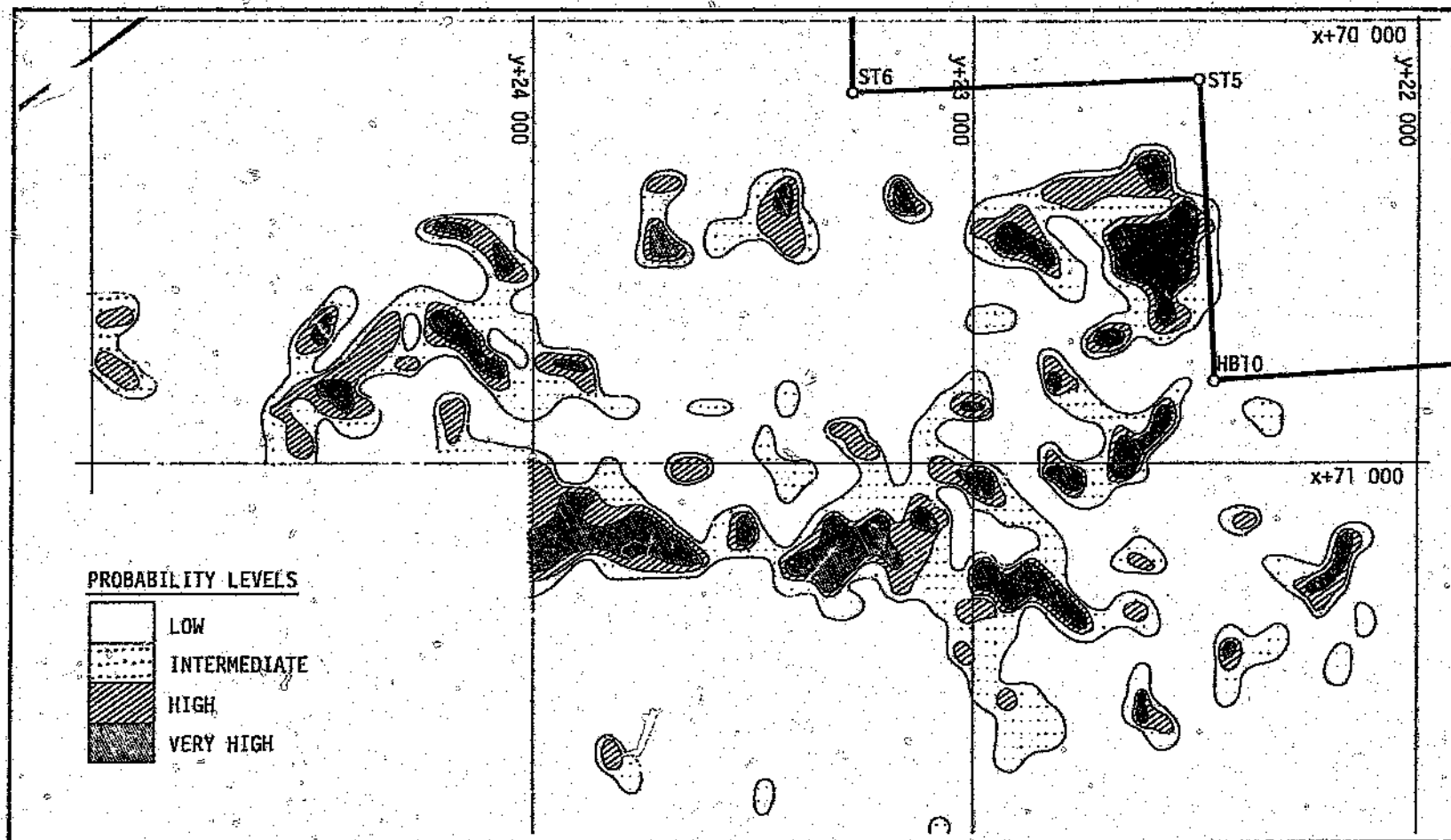


Figure 13A - Predicted distribution of Pseudo-Breccia Phenomenon in the vicinity of the Vaal Reef in the 5A North Area.

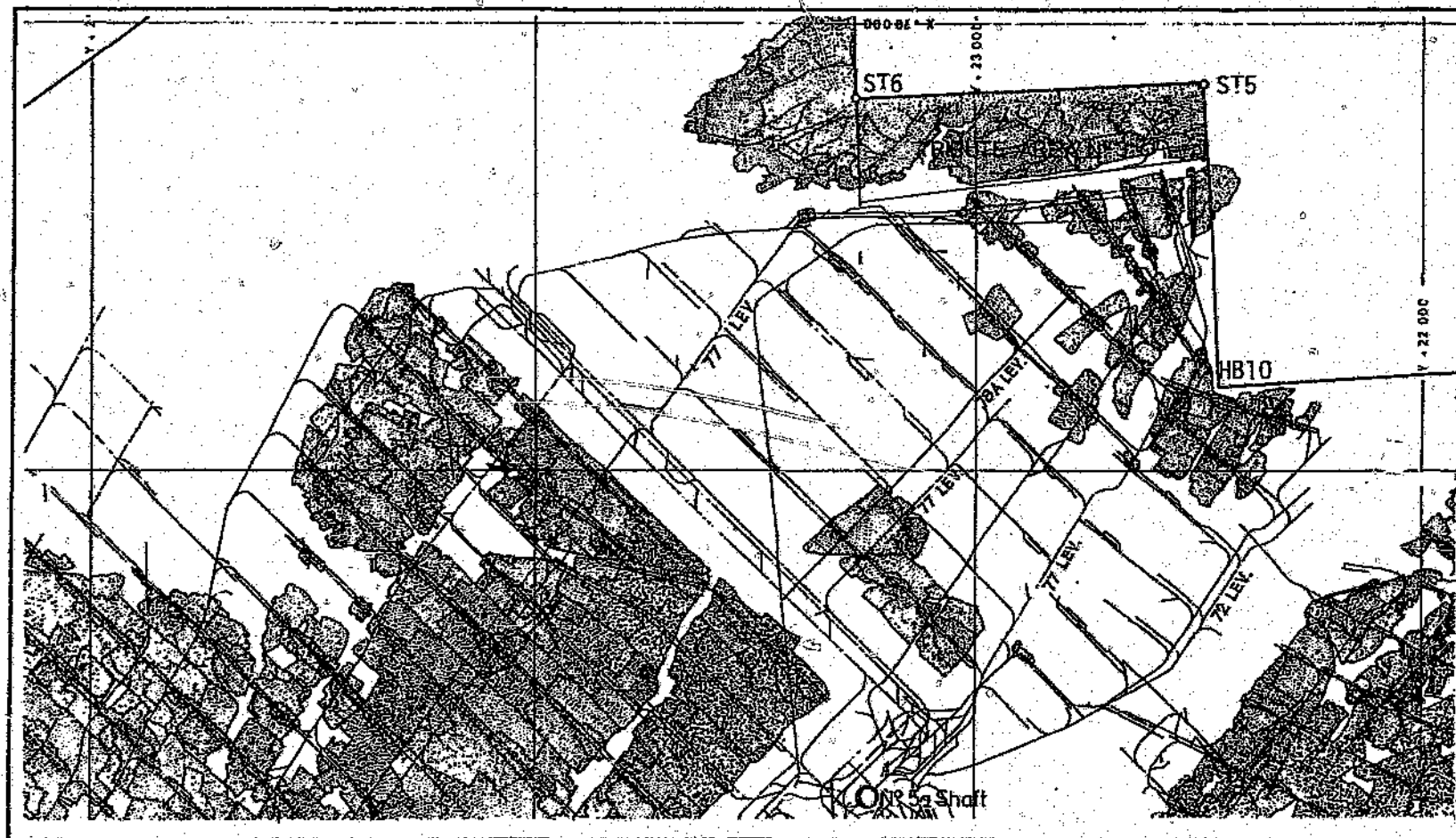


Figure 13B - Plan of Stope and Development workings in the 5A North Area at the time the Pseudo-Breccia survey was undertaken.

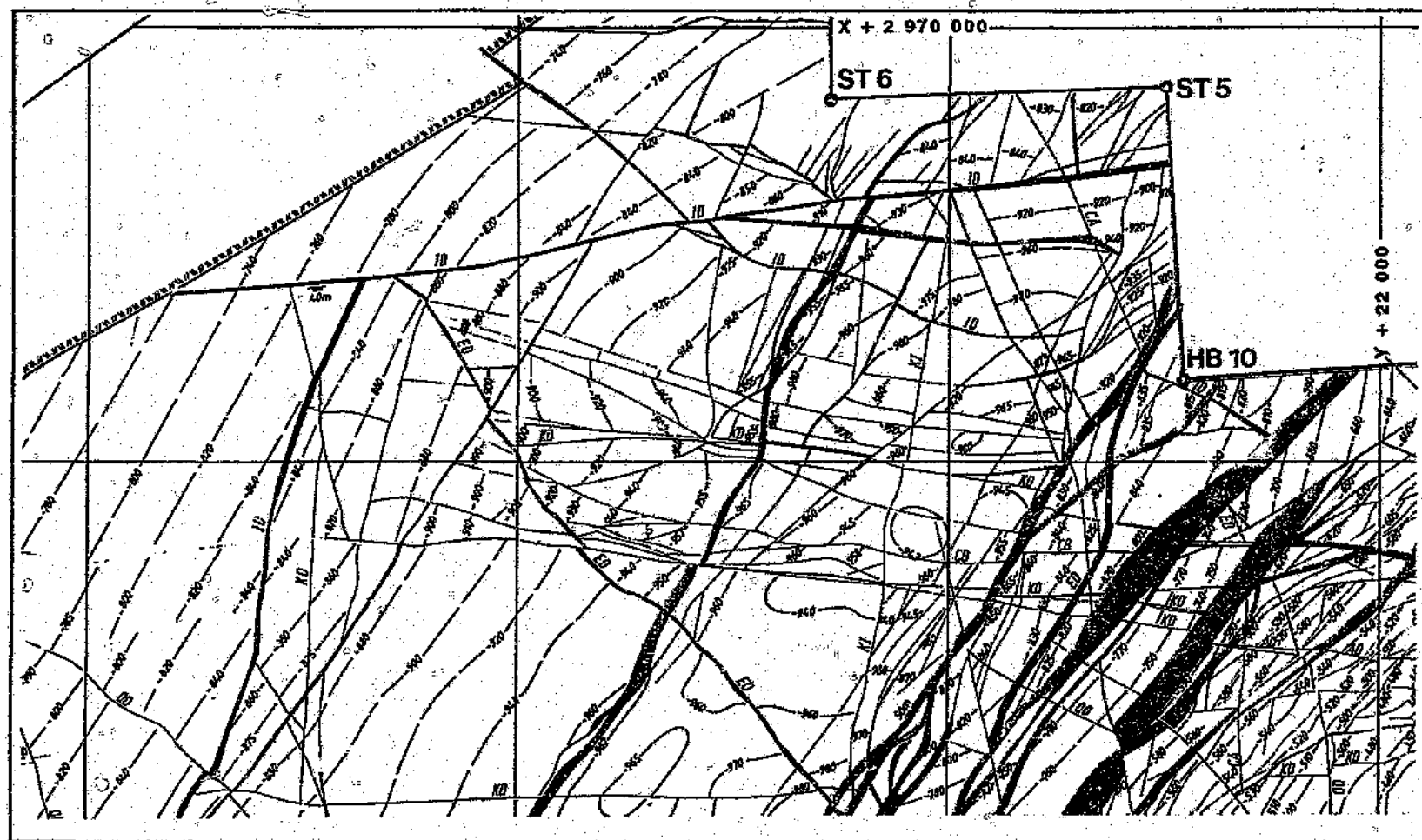


Figure 13C - Plan of independently mapped Structural Trends in the 5A North Area.

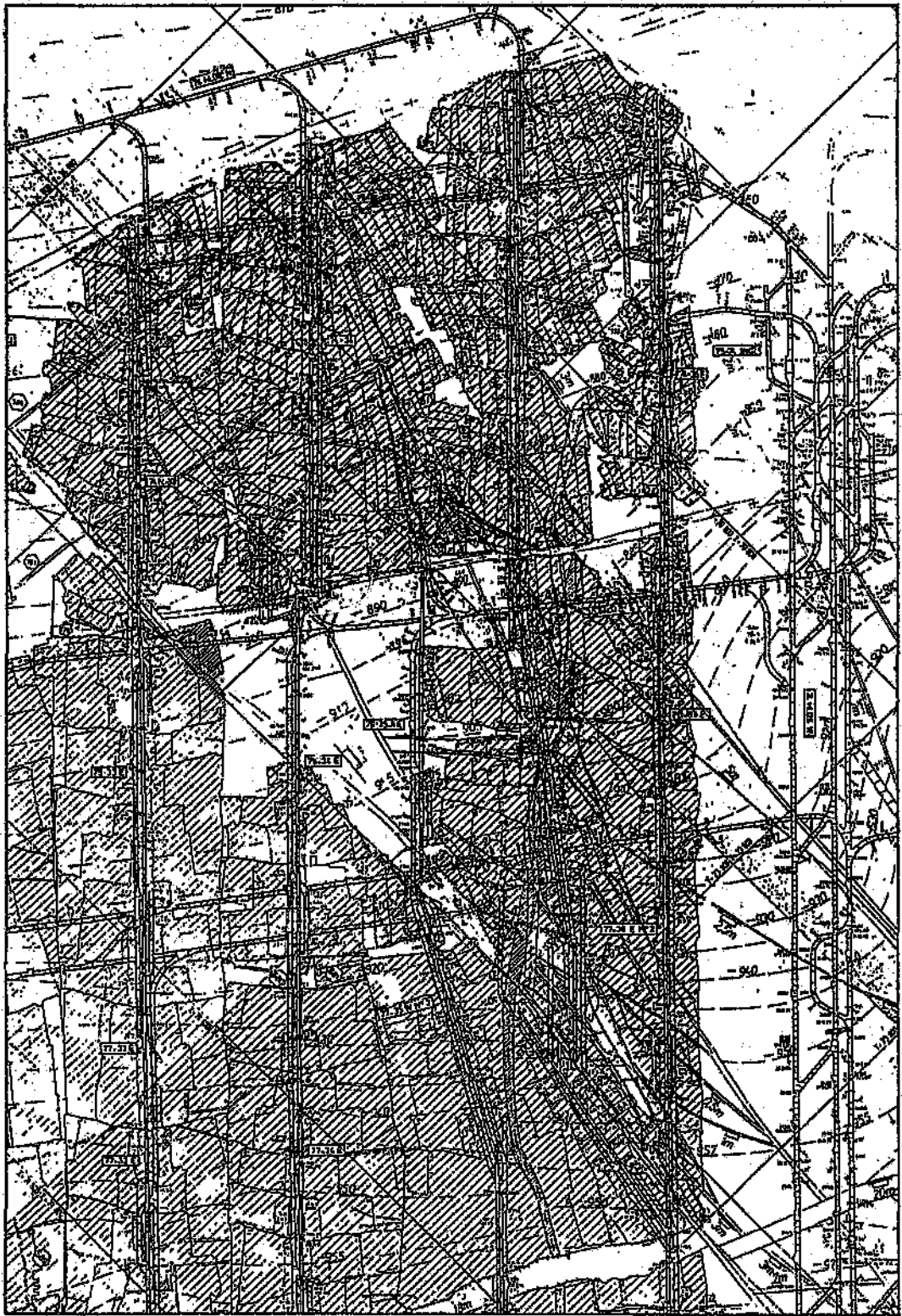


Figure 14 - Plan of Stopping Area Northwest of No.5 Shaft showing the effect of Pseudo-Breccia.

3. REVIEW OF CAVE MINING.

3.1 CAVING ON THE WITWATERSRAND.

Consideration of stick prop support as an alternative to compressible mat pack support for intermediate and deep level stoping operations began in the early 1940's at Randfontein Estates Gold Mine, as reported by Beringer⁽⁶⁾.

The change-over from pack to stick support in this instance was based upon the observation that, during re-opening of shallow workings, it was found to be a relatively easy matter to access old panels in which rigid stick support, installed a generation previously, prevailed. Panels supported at a later date with compressible packs were all rendered inaccessible.

In particular, the action of rigid support in preventing the potentially unstable immediate (0m-3m) hangingwall zone from becoming loose was recognised as being of paramount importance. It was suggested that the increased frictional resistance between strata layers, maintained by the rigidity of the face abutment and the stick (and pillar) supports, was responsible for locking hangingwall blocks in place and preventing collapse. Non-rigid support types such as matpacks, pigsties, stonewalls and, peculiarly, sticks with headboards and sticks with wedges, permitted the hangingwall to move down along planes of weakness, resulting in the loss of cohesion between layers and the destruction of interlocking mechanisms in the hangingwall strata. In addition, it was recognised that increased rates of face advance, brought about by concentrated mining methods, decreased the time-dependent loosening or unravelling of the hangingwall blocks and improved hangingwall stability.

In a contribution to Beringer's paper, Rice⁽⁷⁾ reported that stick support without pillars was used with beneficial results on the South Reef at the Witwatersrand Gold Mine down to a depth of 1200m. In particular, it was noticed that "spitting" of the faces stopped following the change-over to stick support. At depth, sticks would start breaking against the hangingwall or footwall some 12m back from the face. These foreshortened sticks would continue to carry reduced load and would finally collapse about 24m back from the face. This would bring the hangingwall down hard on the shrinkage which had been left as a precautionary measure.

Rice expressed the view that, at greater depth where closure would cause sticks to fail more rapidly, stick support alone might not keep the hangingwall in-situ, nor, because of bursting problems, would the cutting of pillars provide the necessary extra rigidity.

Hill⁽⁸⁾ reported upon an experiment carried out at East Rand Proprietary Mines to reduce the incidence of rockbursts by inhibiting bending, bulging and sag of strata into the stoped-out area using "incompressible" support. From a practical and economic point of view, stick support seemed most nearly to meet the requirement of an incompressible support. Accordingly, it was decided to support a stope at a depth of 2100m with 300mm sticks; the object being to prevent bending of the strata about the face, and to totally inhibit sag, or stope closure, for a distance of approximately 8m back from the face.

In practice the sticks broke sooner than expected. Fig 15 shows the percentage of broken sticks plotted against distance from the face. About half had broken by the time they were 10m back from the face and all sticks had broken 25m from the face. Nevertheless, stick support at depth was felt to be safe and more economical than other methods of support used.

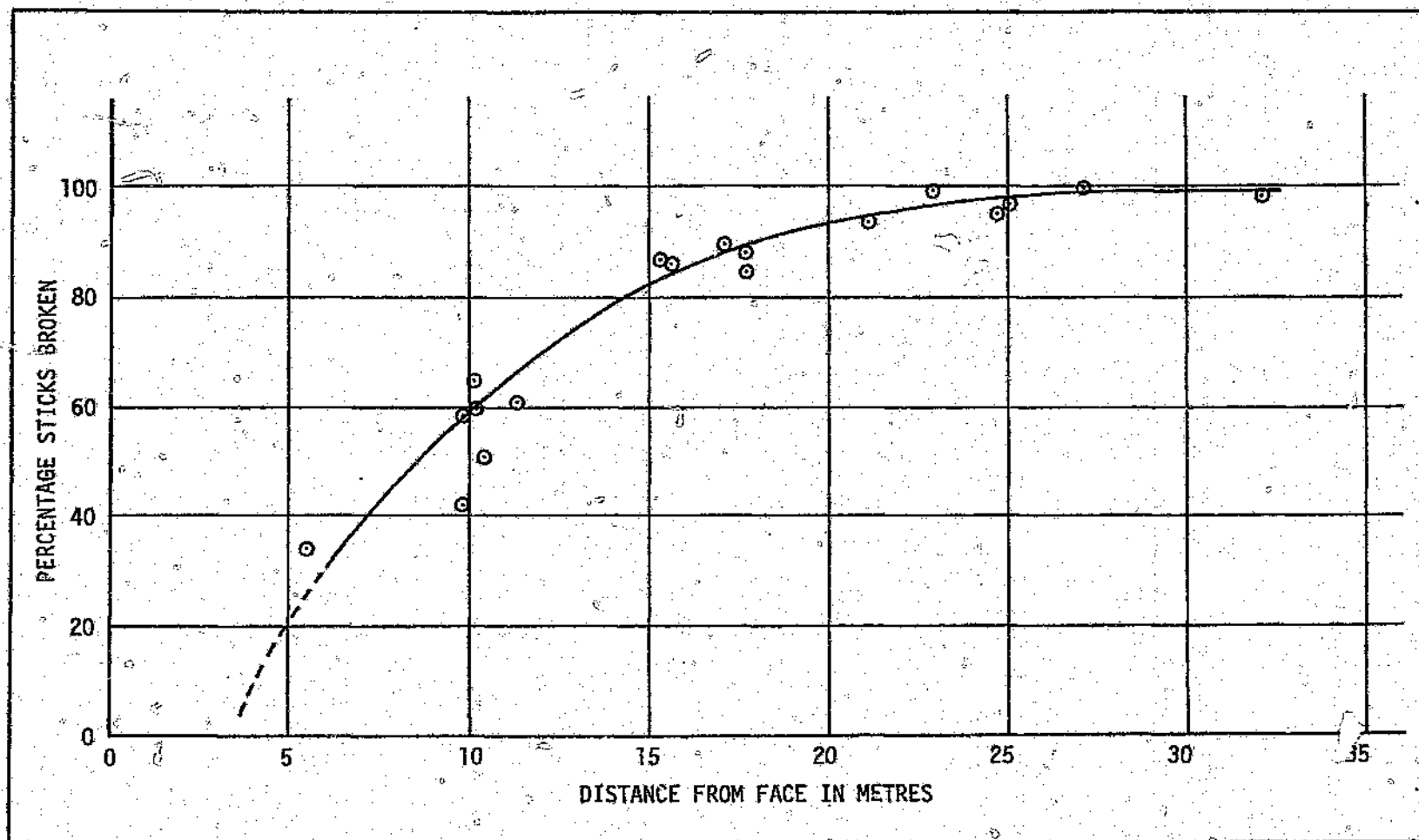


Figure 15 - Graph showing the percentage of Broken Sticks plotted against Distance from the face at ERPM.

Other important observations made by Hill related to the monitoring of sag, or closure, in adjacent pack supported and stick supported panels. He concluded that closure was not influenced by the type of support installed (similar rates were recorded for both panels), nor the distance the face was advanced (rates of closure actually decreased slightly as the span increased over an eight month period). However, he noted that closure increased dramatically as the intensity of stress in the intact ground ahead of the advancing face increased (ie, as the remnant stage was approached).

In the first recorded instance of a deliberate attempt at stope caving in deep tabular South African deposits, Batty and Bell⁽⁹⁾ reported on caving at a depth of more than 1500m below surface at the Sub Nigel Gold Mine during 1940/41. The attempt was motivated by the need to remove pack timber from back areas for re-use at the working faces to decrease support costs associated with the increasing channel widths which were being encountered at the deeper levels.

Removal of packs from the back area commenced at the third row from the centre gully at a time when the face was approximately 30m from this position, with the face continuing to be advanced during pack removal operations. Extraction of the initial row of packs proved easy, but this operation became increasingly more difficult as subsequent rows, working towards the face, were removed, blasting was necessary as increasing pressure squeezed mats into a compact mass. The remaining packs also started canting towards the face indicating differential movement of the footwall towards the back area with respect to the hangingwall. In the now unsupported area, the footwall showed signs of bulging. Attempts at inducing caving by blasting resulted in only minor falls about 1m thick. The stope face started showing signs of pressure bursting during this time. The hangingwall near the face was reported as smooth and solid, but badly fractured away from the face.

Following extraction of the sixth row of packs, with the unsupported span now approximately 25m, a large cave occurred accompanied by an air-blast which blew out all lamps in the stope. Conditions at the face were observed to improve following the cave and the canting of packs towards the face ceased. The hangingwall above the cave, which was estimated to be between two and three metres thick, assumed a dome-shaped appearance.

Conditions remained satisfactory, and pack rows, which became now increasingly more easy to remove, were recovered to a distance of 15m from the face. Inspection showed that the thickness of caved hangingwall had increased at this stage to an estimated three to four metres. The hangingwall continued to cave as the face was advanced and pack rows were removed. The hanging at the advancing face remained solid and in good condition. Examination of boreholes drilled into the hangingwall strata revealed that bed separation had occurred to a maximum height of 15m above the stope, with the intervening strata showing two well defined cavities. The orientation of fractures in the hangingwall exhibited a gradual but dramatic change in direction and intensity from near vertical and closely spaced before caving operations began, to more widely spaced, low angle, almost horizontal, features dipping towards the face (Figure 16).

In this experiment, caving was applied in the first instance with the objective of saving timber costs, but, as work proceeded, it became apparent that caving released pressure on the face and the adjacent hangingwall, resulting in improved hangingwall and stoping width control. Consequently, stope caving was extended in subsequent years to workings in the lower levels (down to 1800m below surface), with extremely satisfactory results in all cases. After two years the method represented a quarter of total production from the mine, eventually

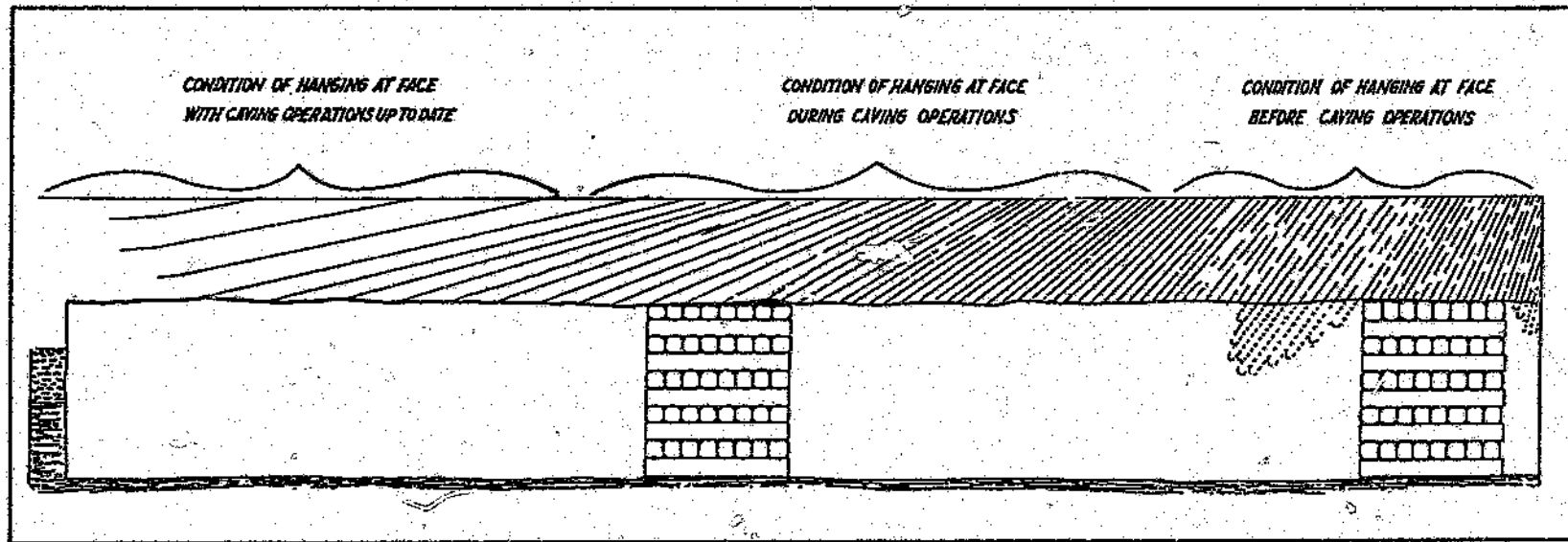


Figure 16 - Section through a Caving Stope Panel at Sub Nigel showing the changing orientation of Hangingwall Fractures.

becoming a standard practice in those areas which warranted the application of the system.

Following these initial experiments, no further reference to the cave mining method is documented until Briggs⁽¹⁰⁾, in a contribution to a paper by Morgan and Theron in June 1963, reported that caving continued to be practised with complete success at Sub Nigel until the early 1950's. The method was discontinued when the mine's wide pay-shoots became depleted, leaving only the narrower and more erratic shoots. Stopping blocks then became too narrow on strike to effectively cave (a distance of some 18m from the centre gully was required). In addition, the supply of labour became scarce in the off-seasons, creating a further problem, as pack removal operations associated with caving demanded extra labour in comparison with other stopping methods.

Briggs commented more than ten years after the cessation of caving at Sub Nigel that caving normally started to take place after the removal of three or four rows of packs, but in many cases it occurred much later, and in a few cases caving never actually took place at all. Pressure on the face was removed almost as soon as the first two or three rows of packs were blasted out, and did not depend upon actual caving taking place. Briggs attributed this to the cantilever effect lifting weight off the face. Face conditions improved and it immediately became easy to cut the panel to the desired width. Initially, attempts were made to induce caving in stopes which failed to cave naturally, by drilling and blasting the hangingwall, but in not one single instance were these attempts successful.

3.2 CAVING ON STILFONTEIN GOLD MINE.

Cave mining with steel friction and hydraulic props was pioneered on Stilfontein Gold Mine in 1961 as part of a successful, large scale experiment to improve breaking, clearing and strata control on that mine. Morgan and Theron⁽¹¹⁾ reported that, from the commencement of stopping operations, great difficulty had been experienced with falls of ground adversely affecting mining efficiencies. The hangingwall, an extremely friable argillaceous quartzite completely lacking cohesion, collapsed readily. Taking advantage of this fact, it was felt that caving would release pressure at the working face, resulting in improved hangingwall conditions. A trial, involving five-stopping connections at a depth of 1200m, was initiated.

The main requirements of face support would be to maintain the hangingwall beam adjacent to the face in as close to its original state as possible, to control bending and inhibit any tendency to sag. The support units would also have to be easy to install, quick and safe to remove, and have a high initial setting load and yield capability. A further requirement was that they should not be easily displaced by blasting. The obvious choice was steel props.

From experience it was established that the following factors were important for the provision of safe working conditions in prop caving operations:-

a) Back-Area Caving.

To promote an early cave, hangingwall slots were generally cut at the ledging stage. Any tendency to hang-up had to be rectified by inducing a cave, either by blasting the hangingwall or by increasing the strength of the breaker row of props. Even a superficial cave involving less than 1m of hangingwall gave immediate relief. Failure to cave at an early stage of stopping operations soon led to excessive closure, with resultant damage to winches, pipes, cables and

accessway support. In one instance, subsidence of 1m occurred over a weekend in a stope which had failed to cave, causing serious damage to installations and loss of equipment.

b) **Prop Density.**

The provision of an adequate density of props in the support area, to inhibit hangingwall sag, was critical to the success of the method. When the prop support density was correct, the hangingwall caved behind the back row of props, the face was relieved of pressure, and the dip of fractures in the hangingwall was towards the worked-out area (ie. the fractures were inclined over the face). Conditions in the working area remained stable and safe. Typically 400kN props were spaced one metre apart on strike and dip and two metres from the face before the blast, providing a calculated support resistance of about 300kN/m². On the other hand, if the prop density was insufficient, caving would not take place, pressure would build up at the face and over the props, and fractures would be inclined away from the face. General hangingwall conditions at the face, and over the props, would deteriorate dramatically. The task of prop extraction became extremely difficult and dangerous, and flushing of the hangingwall at the face could occur. Figure 17 illustrates the prop density situations described by Morgan and Theron.

c) **Support Distance.**

The strike distance from face to backline of props over which the hangingwall was supported by props had to be kept to an absolute minimum. If this cantilever was too long, pressure would build up over the props causing them to yield. This would rapidly be followed by disintegration of the hangingwall and unsafe working conditions would result. As the prime objective of prop support was to provide as rigid a support as possible between the cave and the face, thereby reducing sag to an absolute minimum, the shorter the length of hangingwall supported by props, the easier it would be to achieve this objective. Consequently, the number of rows of props used in the experiment was reduced from four to three, resulting in a reduction in the face to cave distance from 5.8m to 4.6m (before the blast), with satisfactory results. The situation of an excessive strike distance supported on props is illustrated in Figure 18.

As no suitable steel props were available in the country at that time, pit props, developed and in use in overseas coal mining operations, were imported. Three types of prop were used during the course of the experiment:-

- Friction props.
- Hand-operated hydraulic props.
- Pump-operated hydraulic props.

Of the 11000 props on the mine, 10500 were hand-operated hydraulic props. Initially these props had a design setting load of 150kN and a yield load of 200kN. About 20% of props installed with this setting capability were loosened by the blast. Consequently, setting and yield loads were increased to 250kN and 300kN respectively, and the blast-out rate decreased

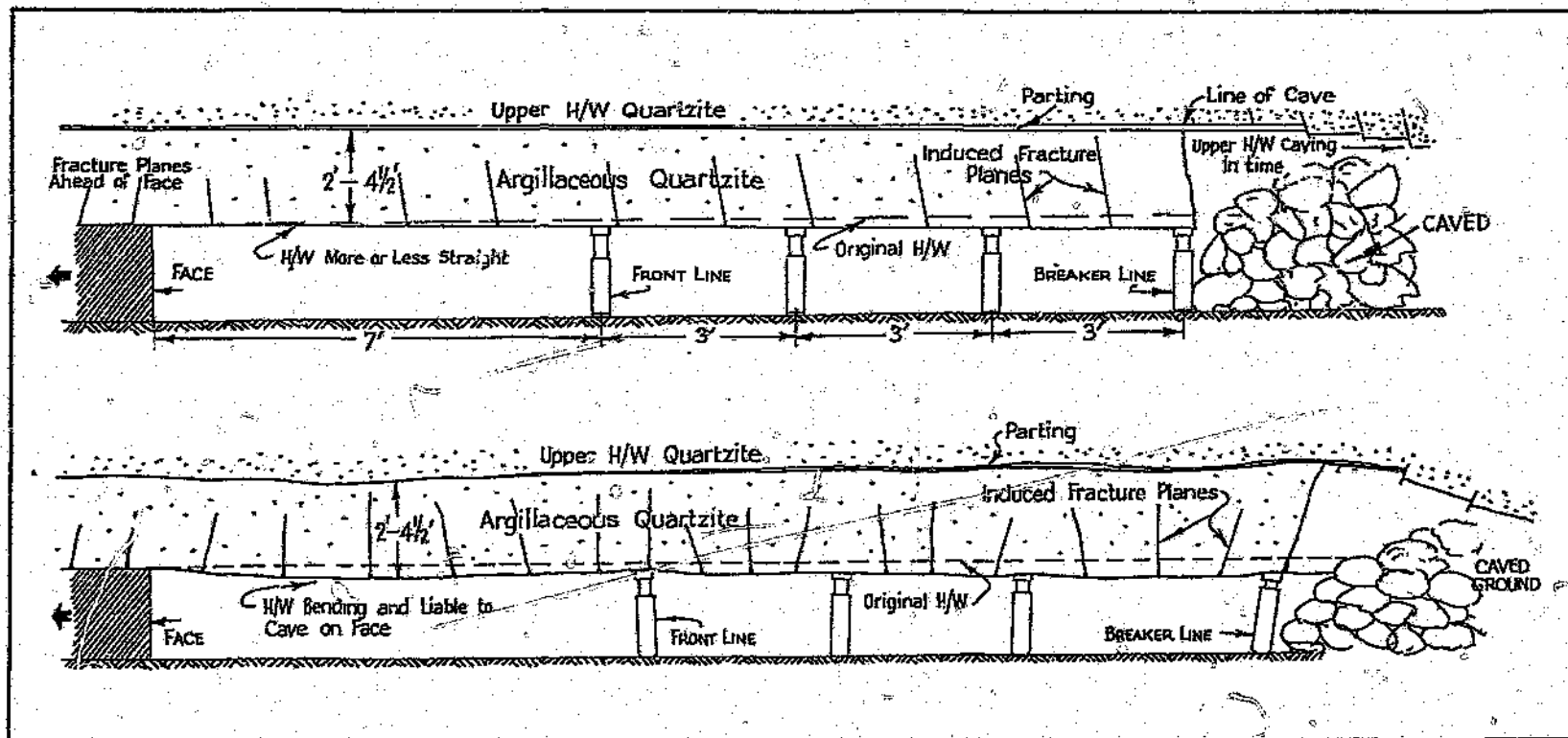


Figure 17 - Illustration of Correct and Incorrect Prop Densities at Stilfontein Gold Mine.

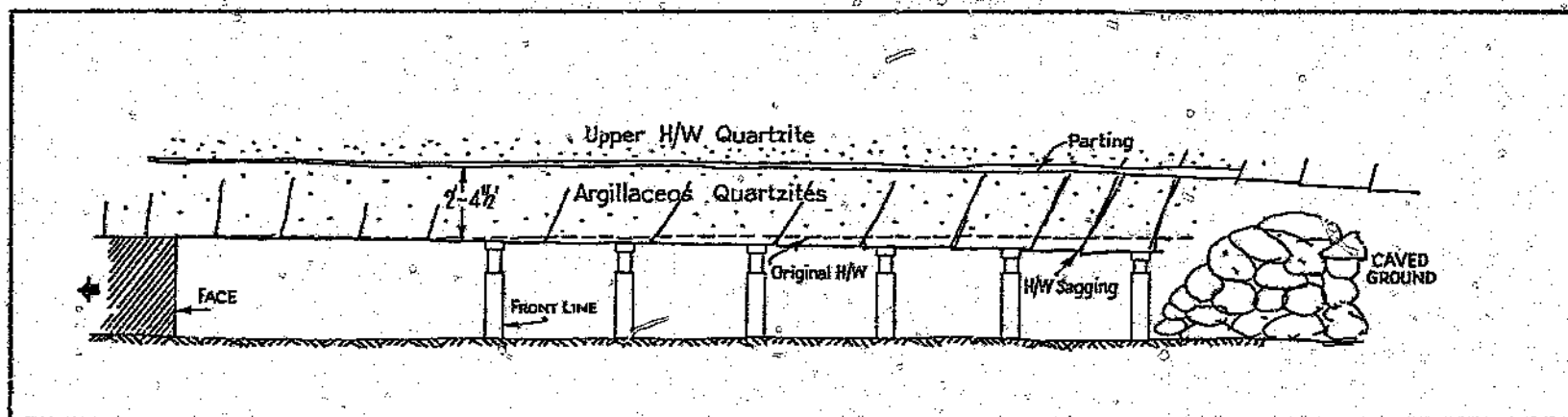


Figure 18 - Illustration of Excessive Distance on strike supported by Props at Stilfontein.

to less than 5%. It was subsequently concluded that a more suitable prop for Stilfontein conditions would have a setting load of between 300kN and 350kN, and a yield load of between 350kN and 400kN.

Morgan and Theron expressed the hope that eventually a prop designed specifically for the South African gold mining industry would become available. In this regard, they put forward a proposal for a prop specification which covered the following design aspects:-

- a) The provision of extension pieces to accommodate variations in stoping width while maintaining a standard prop length with a limited (30cm) working stroke.
- b) The internal and external protection of cylinders, rams and seals from damage caused by handling, blasting, abrasion and corrosion.
- c) Ensuring the ready collapse of props when released, particularly during remote release operations carried out prior to caving.
- d) Articulated and spherically seated head and foot pieces fitted with hobnail studs to improve grip and blast-out resistance.
- e) The development of some form of air-pocket or spring-loaded appliance to increase the resilience of props.
- f) The development of a portable and lightweight air-operated pump which could be accommodated at the face in narrow stoping widths, and which would ensure that props were set to uniform loads.

In addition to the obvious strata control benefits, Morgan and Theron pointed to increased drilling and blasting efficiencies, better face advance and stoping width control, reduced accidents, improved supervision and face utilisation, and more effective ventilation control brought about by the cave mining method.

Prior to the introduction of steel props and caving at Stilfontein Mine, Morgan⁽¹²⁾ reported in 1955 on an unsuccessful attempt to introduce rescue stoping as a method of reducing the milling width. From the outset it was apparent to him that the narrow Vaal Reef channel called for rigid control of the stope width. However, the extremely unsound condition of both the immediate hangingwall and footwall of the reef made this task very difficult. Despite close supervision over a period of three months, the rescue method failed, mainly because the hangingwall invariably collapsed as soon as the unsupported span exceeded two or three metres. On the subject of safety, he reported that the mine had an unenviable record, and that the majority of serious accidents underground were due to falls of ground.

3.3 UNSUCCESSFUL CAVING EXPERIMENTS.

3.3.1 Caving In The Orange Free State.

At about the same time as caving experiments were taking place at Stilfontein Mine (early 1960's), attempts were also being made in the Orange Free State to apply the caving method, using steel friction props, to the problem of thick Khaki Shale hangingwall above the Basal

Reef. While there was broad agreement with the general principles of caving outlined by Morgan and Theron, some aspects peculiar to the OFS situation, notably the extreme weakness of the shale and the strength the massive overlying Leader Quartzite, led to the eventual abandonment of the method.

Jooste⁽¹³⁾, in describing experimental caving work started in April, 1960 at Free State Geduld Mine, doubted that it would ever be possible to carry directly a soft shale hangingwall with props, as was being attempted at the time on President Steyn Gold Mine. He reported, however, success with the method at FSG No.2 Shaft, where a basal quartzite middling, varying in thickness from 30cm to 100cm, was carried beneath the shale layer, which was up to 4m in thickness. Caving occurred spontaneously behind four rows of props when the middling quartzite was less than 60cm thick. If the middling was too thin, the cave would tend to run forward over the props, but where the quartzite beam exceeded 60cm in thickness, the cave tended to lag, and had to be induced by blasting.

Serious hangingwall stability problems, which threatened the future of the project during the first few months of experimentation, were traced to props being knocked out on night shift by the scraper operating between prop lines. This was overcome by confining scraping operations to the face area using a blast barricade attached to the face line of props. Other minor difficulties were overcome and the experiment was extended to the whole shaft during 1961, with the importation from Germany of an additional 4000 friction props.

Jooste agreed fully with the need to adhere to the caving practices propounded by Morgan and Theron. In particular, he noted that the early achievement of a cave, the inducement of caving when necessary, the maintenance of an adequate prop density, and the reduction of the distance between the face and the cave, were important factors for successful caving. He conceded that the principle of rigid support apparently did not always apply, as partings did develop and were present in the hangingwall between both the lower and upper shale interfaces with the adjacent quartzites, without serious ill-effects.

To determine whether caving extended into the strong quartzite overlying the shale, a raise was developed for a distance of 25m, in the hangingwall 10m above a caved area, and 50m behind the face. Figure 19 illustrates the extent of bed separation, and the orientation of other open fractures observed in this excavation. It was concluded that caving did indeed extend into the overlying quartzites, and that there was no fear of sudden collapse of these strata.

Limited experimentation with the installation of skeleton packs in the back area led to the conclusion that this was not good practice. The packs tended to keep the hangingwall intact, throwing excessive weight onto the face and abutments, and over-loading the back rows of props. Cracks appeared in the hangingwall, and the situation was described as being similar to, but less severe than, the case where caving was delayed.

Despite the apparent success of the project, Jooste must intuitively have known that caving was alien to conventional mining wisdom of the time, and would ultimately not find favour in the Orange Free State. Although he was pessimistic regarding the future widespread use of full caving, he correctly predicted that the use of reclaimable steel props would become general practice throughout those fields.

An unsuccessful attempt to use four rows of steel props, link-bars and caving at President Steyn Gold Mine to undercut shale, which would otherwise have been mined by rescue

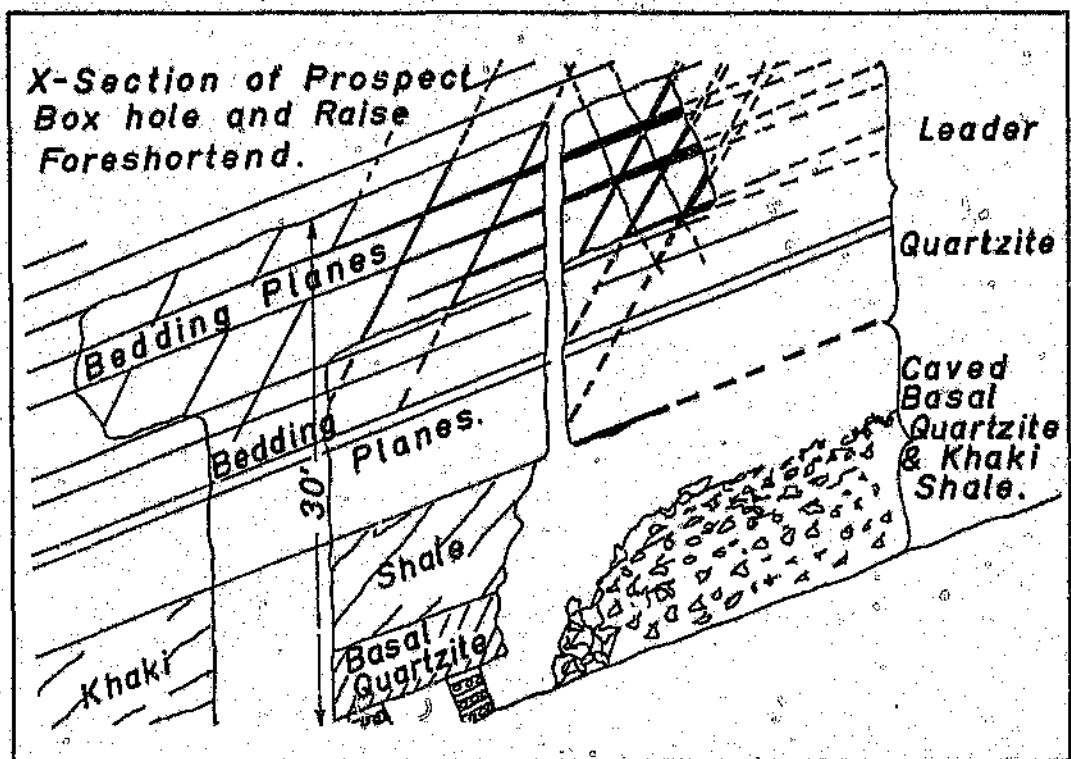


Figure 19 - Illustration of the extent of Bed Separation and the orientation of other Open Fractures observed in Leader Quartzite overlying a Cave Stope at FSG.

methods, was described by Pearson^(14,15). The attempt involved carrying shale, which lay less than 60cm above the Basal Reef, as the stope hangingwall. The experiment started early in 1962, and was carried out in the approaching panels of two adjacent raise connections. After much perseverance under difficult and sometimes dangerous conditions, and despite some early success, the attempt was finally abandoned for the following reasons:-

- a) The span over which the shale hangingwall was self-supporting was extremely limited. This rendered initial undercutting operations, which were essential to the success of the method, extremely difficult.
- b) Similarly, the amount by which the face could be safely advanced once undercutting had been achieved was severely restricted. In practice, this turned out to be less than 60cm per blast.
- c) The shale hangingwall was prone to collapse in the vicinity of even small strike faults. Several attempts were made to overcome the problem using various spiling techniques, but a satisfactory solution was not found.
- d) Unless a solid down-dip abutment was provided, the shale would exhibit a tendency to ride over the props and collapse towards any free face provided at the bottom of a panel.

No difficulty was experienced with caving of the shale. The cave line formed regularly on the last line of props and, contrary to expectations, there was no tendency to flush or over-run into the supported area. Any tendency of the shale not to cave was immediately cured by wetting it down thoroughly.

Although the shale caved well, the Leader Quartzite above did not. Inspection showed that it remained almost intact, and that a substantial space existed between the caved shale and the overlying quartzite over most of the back area. This situation persisted for the duration of the experiment. Attempts to induce the quartzite to cave by blasting only dislodged the limited amount of rock directly involved in the blast.

This was obviously cause for concern, and may have contributed to the decision to abandon the experiment. It was known from experience in nearby rescue stopes that the quartzite was strong and massive, with few bedding planes, and that it was prone to fail in large blocks, often breaking off down the line of the face and crushing timber support.

Caving was also briefly experimented with on the Leader Reef horizon at Harmony Gold Mine during the 1970's. A small number of panels were converted to cave mining over a period of approximately one year. Three rows of hydraulic props were used, with the back row being doubled up to form a breaker line.

Although the trial was reasonably successful, and the prop lines always remained intact, caving conditions were not ideal. The back area took a long time to collapse, and the height to which caving occurred was generally quite modest. Nevertheless, the classic problems associated with failure of the hangingwall to cave, such as drilling difficulty, high closure rates and prop freezing, disappeared immediately a cave occurred.

The experiment never really found favour with production staff, mainly because of their unfamiliarity with the method, and because of the dust, discomfort and unease associated with

on-shift caving. The experiment was abandoned shortly after a change of management pronounced the method to be unsafe under the prevailing conditions.

3.3.2 Caving On Doornfontein.

In 1963, Hinds⁽¹⁶⁾ reported upon an experiment to extract a remnant caved at a depth of 1100m below surface at Doornfontein Gold Mine using mechanical props and caving. Mechanical props had been introduced on the mine on an experimental basis as early as 1958 to reduce timber support costs, without success. Problems which at that time had led to the failure of the experiment included the unavailability of a suitable blast barricade resulting in excessive blast damage to the props. Experimentation resumed in 1962 following the development of an effective rubber screen barricade. Four rows of props were used, 1.2m apart on strike and 1m apart on dip. Because removal of props using long-handled hammers to knock out the two prop wedges was considered unsafe, a small compressed air percussion machine was adapted for the purpose. As an further safety measure, an additional line of props was installed 1m apart midway between the back two lines of props.

Stoping was carried out on the Carbon Leader Reef horizon which had a 1m thick strong glassy quartzite hangingwall overlain by a 2m thickness of shaly quartzite (the "Green bar"). From the outset the hangingwall did not cave naturally. All caving was induced by blasting each time the back row of props was removed. It was never ascertained whether more than 0.5m of hangingwall brought down by the blast caved in the back area because the blasted rock disguised any natural caving that might have occurred. It was nevertheless observed that failure to induce the cave resulted in increased load on the props making prop removal difficult and dangerous. This occurred when a 25m unsupported span was mined without blasting the hangingwall. Despite this patently superficial induced cave, there was a marked improvement in hangingwall conditions and mining efficiencies in the experimental panels. A change in the nature and orientation of stress fracturing at the face was also noted. The experiment was apparently terminated as a result of the high cost of having to induce the cave and the high capital cost of the props.

3.4 CAVING ON HARTEBEESTFONTEIN GOLD MINE.

3.4.1 Historical Review.

Cave mining with steel hydraulic props was first tried on Hartbeestfontein Gold Mine in 1962. This was in response to a marked deterioration in hangingwall and footwall conditions, first noticed in 1959 as the depth of stope operations began to exceed 1000m below surface. In a contribution to Morgan and Theron's Stilfontein paper, Deacon⁽¹⁷⁾ described experiments in which steel props were introduced as face support on Hartbeestfontein. He noted that observation of changing stope fracture patterns gave clues to the behaviour of the rock surrounding stope excavations.

Footwall heave was first noticed in reef drives situated about 1000m below surface during 1959. The first visible evidence of large scale lateral movement of the hangingwall and footwall in stopes occurred when rockbolts were found to be failing in shear at depths below 1500m, initiating serious research into the problems of ground movement. As stoping

progressed yet deeper, considerable footwall movement was observed in haulages and crosscuts.

It was also found that stope fracturing became more intense at these depths. Hangingwall falls became the rule rather than the exception, and stoping widths became increasingly more difficult to control, particularly as panels advanced beyond 30m from the centre gully. It became apparent that lateral deformation of the hangingwall and footwall was being inhibited by waste walls, which were used as a form of support at that time. It was felt that this caused increased stress ahead of the face, resulting in the observed deterioration of conditions. This led to experimentation with steel prop support units, which could be withdrawn and would not inhibit lateral deformation.

The first trial commenced in July 1962 in a remnant measuring approximately 30m on strike by 60m on dip situated at a depth of 1750m below surface. The general configuration is shown in Figure 20. Four rows of steel hydraulic props, spaced 90cm apart on strike and dip, were used during the experiment. It was not found necessary to create a breaker line by reinforcing or installing additional props in the back row.

Closely spaced fractures, running parallel to the previously mined faces, and extending 5m into the solid, were encountered at the edges of the pillar. Fractures ahead of the newly created up-dip panel were superimposed on these, resulting in broken hangingwall conditions and frequent falls of ground at the edges of the remnant.

Six days after the start of the experiment, having advanced the face 3m, and with an unsupported span behind the last row of props of 10m, a spontaneous cave, estimated to be 0,5m thick, occurred. All attempts to induce the cave by blasting prior to this, had been unsuccessful. Immediate stress relief was noticed at the face, and there was a decrease in the amount of dressing and barring required.

When the face had advanced a further 6m, another spontaneous cave occurred, and the hangingwall fractures became vertical, having previously been inclined at 85° towards the back area. With continued advance, the face parallel fractures swung further, until they were inclined at 85° towards the face. The spacing of fractures also increased from less than 5cm, to more than half a metre apart. With this change in the dip and frequency of the fractures, the nature of the cave changed. The hangingwall now slid down in a series of stepped blocks, without the occurrence of violent falls, and the footwall began to heave into the unsupported area, mimicking the hangingwall pattern. The hanging at the side of the remnant also became more stable. Figure 21 is an idealised section through the remnant illustrating the observed change in fracture orientation and intensity. The entire remnant was extracted without major incident using the prop caving method.

Following this success, the caving experiment was extended to other stopes. The number of prop rows was reduced from four to three, and the spacing increased to 1,2m on strike by 1,0m on dip, with no adverse effects. Conditions in the caving panels were excellent in comparison with adjacent stone wall supported areas. Caving was spontaneous and complete, with stress relief afforded at the face. The general appearance was that of stopes at 600m below surface, instead of their actual depth of 1800m. It was a relatively easy matter to carry stoping widths of 80cm, while in the surrounding stone wall supported panels stoping width control was difficult in the extreme.

An instrumentation program was embarked upon to monitor strata behaviour and to determine

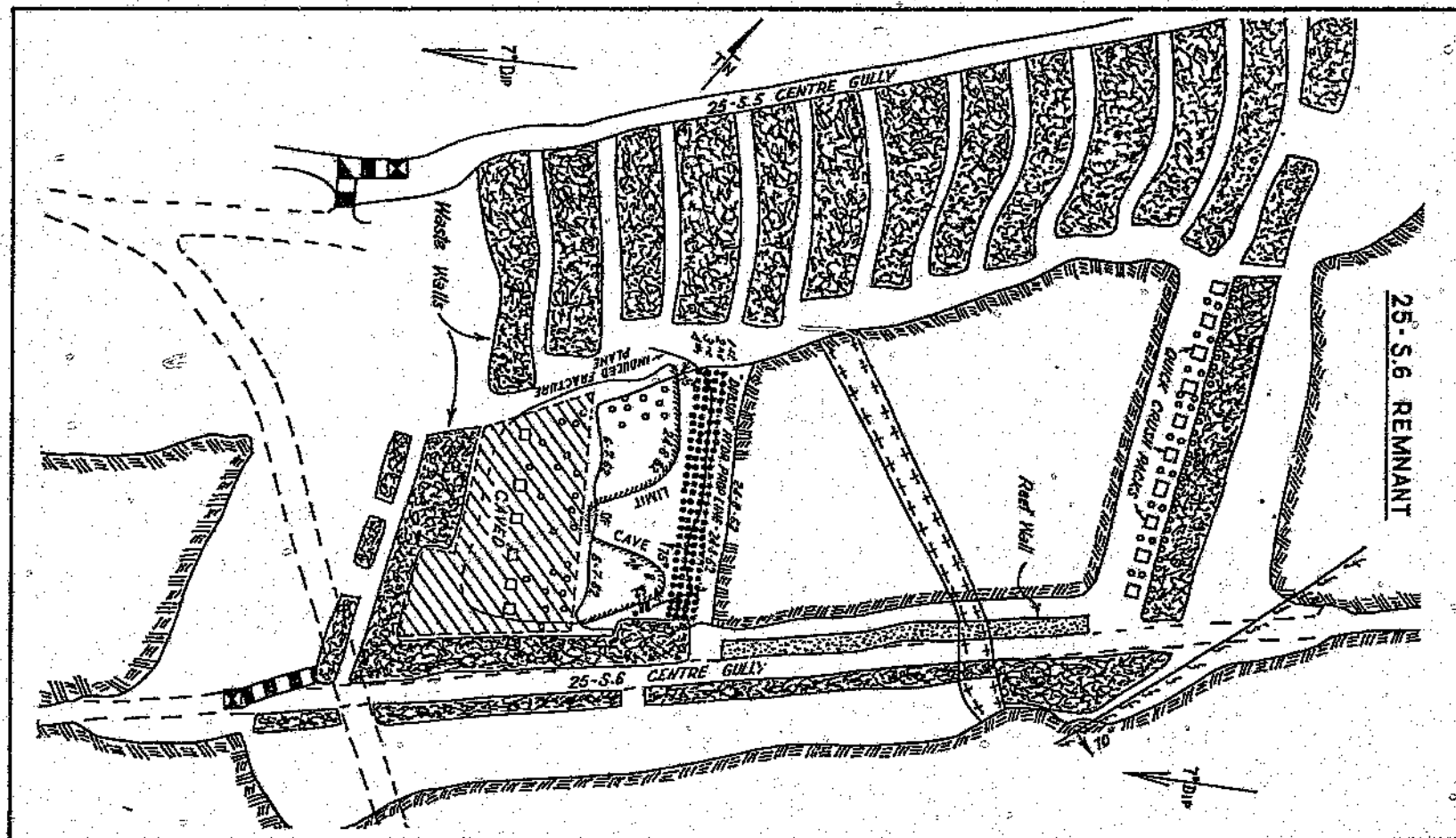


Figure 20 - Plan showing the site of the First Caving Experiment on Hartebeestfontein Gold Mine in 1962.

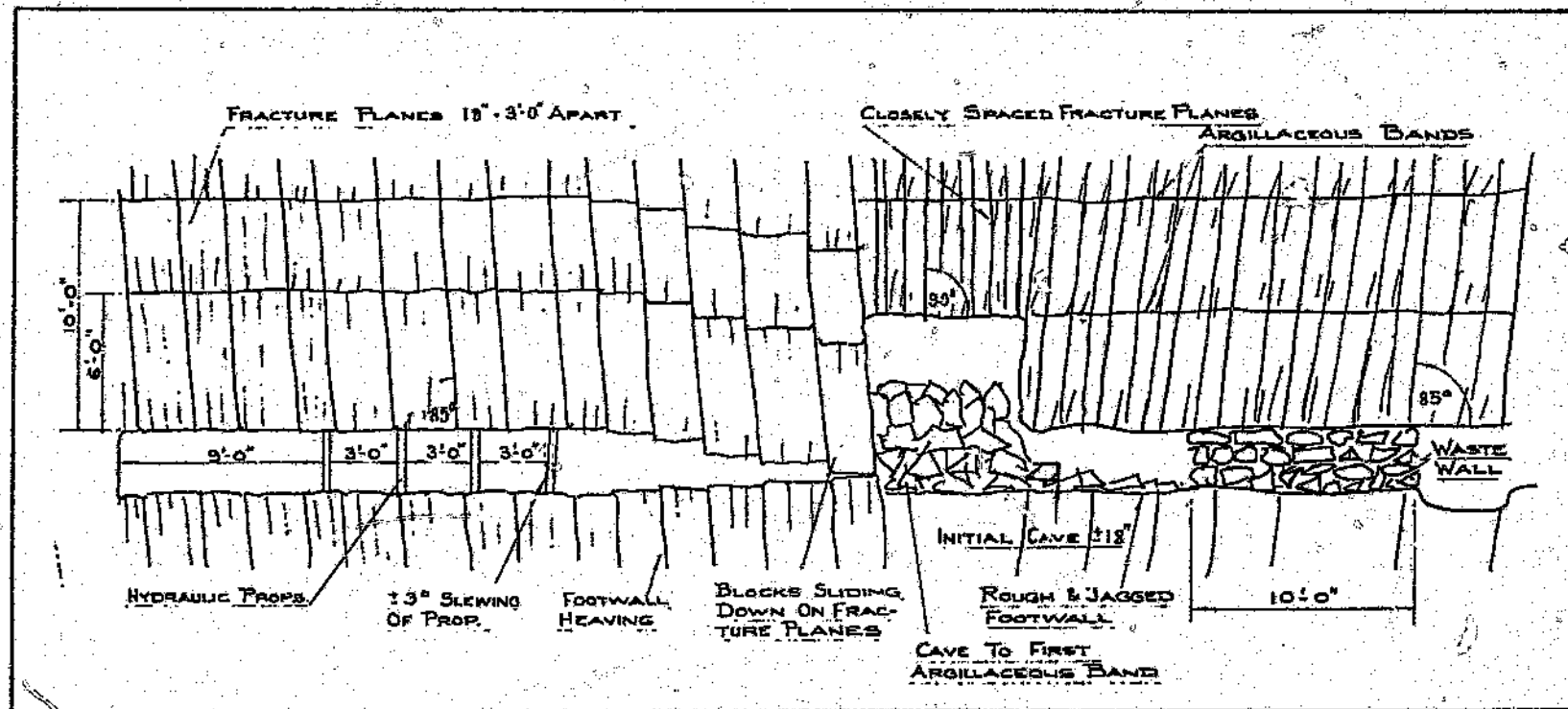


Figure 21 - Idealised section illustrating the observed Change In Fracture Orientation and Fracture Intensity Following Caving.

the effects of caving on the rockmass around one of these stopes. This involved measuring the load and deformation induced in gully packs, measuring the loads on panel props before and after caving took place, measuring rock stresses ahead of the faces, measuring the displacement of rock in boreholes drilled to within 15m of the stope from twin haulages situated 160m in the hangingwall, and measuring rock deformation in a number of tunnels situated 30m in the footwall of the stope.

It was found that stress levels in the rock up to 4m ahead of the face was immediately reduced following a cave, while stresses beyond this zone were increased. At the same time, loads generated in the back area gully packs increased, and those in panel props decreased. Lateral movement of the hangingwall of up to 60mm per day was measured. While there was evidence to suggest that the effects of mining operations on the surrounding rock diminished in magnitude away from the stope, rockmass movements nevertheless manifest themselves deep into the hangingwall and footwall.

Deacon suggested that his observations supported the concept of macro-plastic flow, first propounded in 1961 by Black and Brown⁽¹⁹⁾, as a possible contributory mechanism describing strata movement at depth. The presence of numerous weak argillaceous bands above and below the Vaal Reef was found to have a considerable effect on the behaviour of the hangingwall and footwall. Inhibiting lateral or shear movement across these bands caused bowing and tensile failure of the immediate hangingwall, and heave in the footwall. Props would then yield beyond their limit and become damaged. It was found that dip slots, cut into the hangingwall and footwall adjacent to the centre gully, would promote an early cave, and negate the effects of lateral compression.

A short while later, when he became manager of the adjoining Zandpan Gold Mining Company which had just started stoping operations, Deacon introduced, and modified, the prop support and cave mining techniques pioneered on Stilfontein and Hartebeestfontein Gold Mines. In a paper describing stoping at Zandpan⁽²⁰⁾ in 1965, he explained how large diameter timber sticks were used as an alternative to steel props, because relatively few props of the required specifications were available at the time.

A maximum of five rows of 150-200mm diameter sticks were required in caving panels, with the fifth row being blasted out concurrently with the face blast. This was achieved by drilling a short 25mm diameter hole in each of the sticks in the back row and charging these with a primer cartridge. If this failed to produce a spontaneous cave of the back area, and the unsupported span behind the sticks started to exceed 10m, short 1m holes were drilled into the hangingwall and blasted to induce caving. Sticks that were not removed by blasting tended to retard caving, but would normally break after a short period.

Adverse stoping conditions resulted when caving did not occur, or when more than five rows of sticks were left intact. Stress increase at the face caused the hangingwall to break up between supports, and footwall heave became apparent in the back area, often forming mushroom-like domes. This implied that too much restraint was being applied to the immediate hangingwall and footwall strata, impeding lateral movement. The situation was immediately rectified by inducing a cave.

The provision of hangingwall and footwall dip slots to initiate hangingwall caving, and to limit lateral closure of centre gullies, became standard practice. These were cut to a depth of 1m and sited 6m on either side of the original connection (see Figures 22A and 22B). Dip slots were also cut when faces approached within about 20m of each other during the remnant

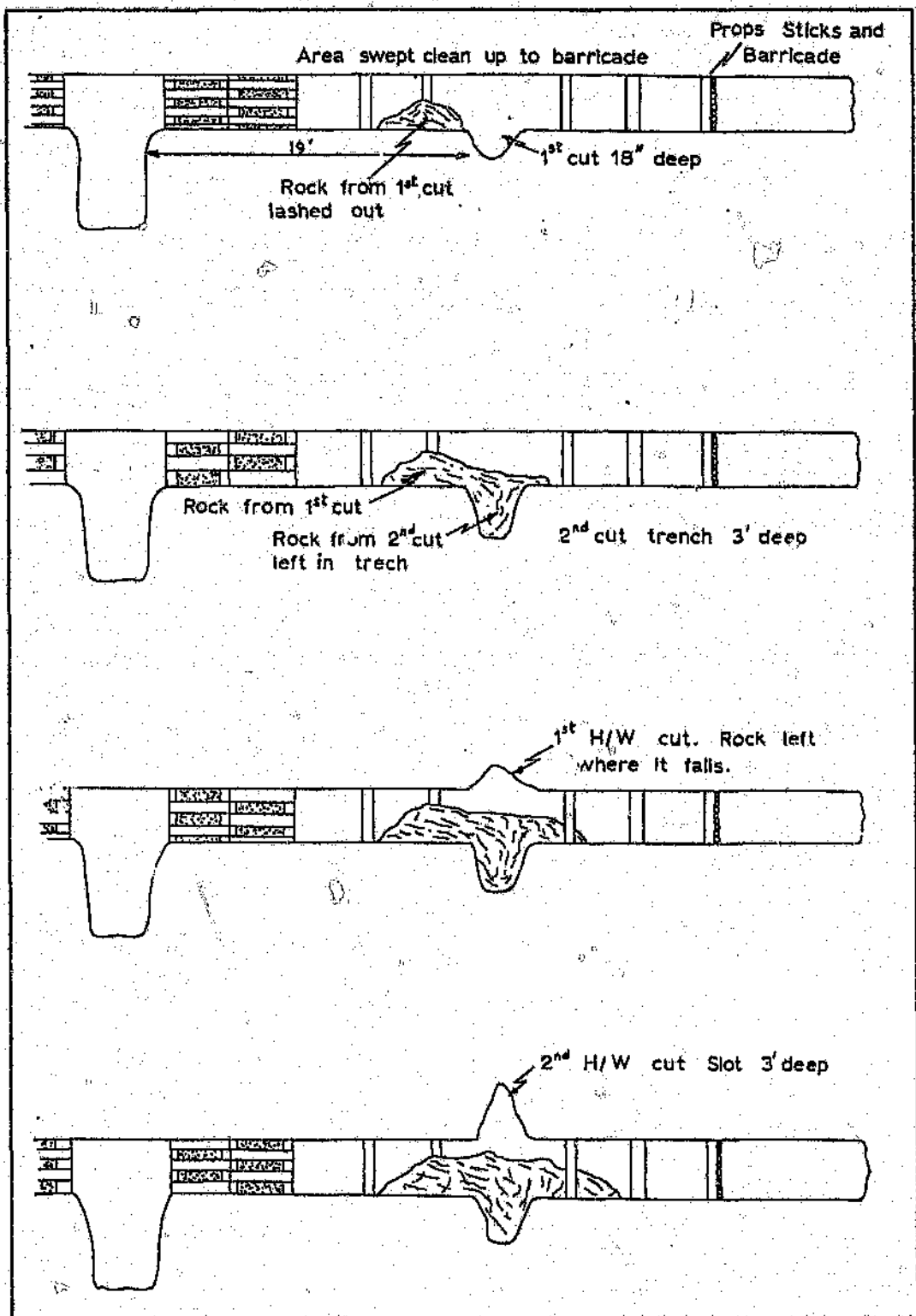


Figure 22A - Section showing the Sequence of Cutting Footwall and Hangingwall Slots adjacent to a Centre Gully.

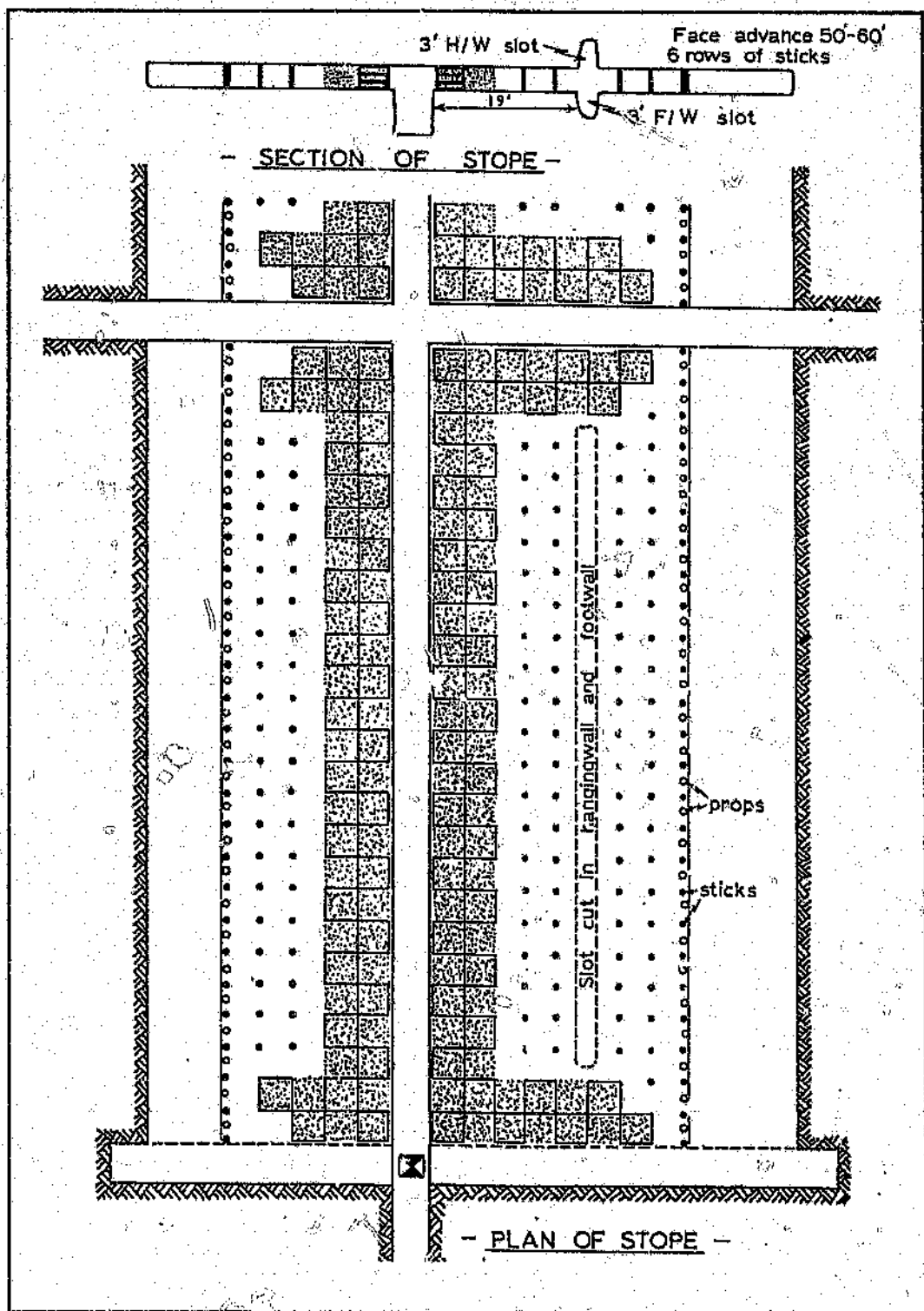


Figure 22B - Plan showing Footwall and Hangingwall Slots adjacent to a Centre Gully.

stage. This successfully overcame the manifestations of high stress and accelerated rock deformation associated with these situations.

Sticks were installed 1,2m apart on dip and strike, and support was kept a maximum of four metres from the face. The general layout of stick caving stopes on Zandpan is shown in Figure 23.

Because of the narrow nature of the Vaal Reef channel, it was then, as now, deemed important, from a mill grade and strata control point of view, to carry stoping widths as low as possible. The importance of drilling discipline was recognised in this regard. Light-weight rockdrills were introduced, and more than passing attention was given to reducing burden, improving drilling accuracy, and controlling hole direction and depth. In this way, average stoping widths of well below 1m were maintained.

Monthly face advances of 18m were generally achieved in caving panels. However, it was found that if the rate of face advance dropped below 6m per month, stoping width control problems were encountered, pointing to the time-dependant nature of macro-plastic deformation. This was further reinforced by the observation that great difficulty was experienced in restarting panels which had stood dormant for any length of time.

Stick caving was applied at depths exceeding 2000m, where measured stope convergence sometimes exceeded 75mm per day. This rate of convergence was considered too great for even 400kN steel props. In addition, differential lateral displacement of the hangingwall and footwall, which normally commenced when the stope span exceeded 18m, led to canting of large diameter sticks. It was felt that this phenomenon would result in instability of steel props, which were more slender.

During the early stages of mining on Zandpan, on-reef slushers were pre-developed well ahead of stoping panels, allowing rock conditions ahead of advancing panels to be monitored. Stopping induced fractures were observed to extend to about 5m ahead of stope faces before the cave, but extended up to twice this distance once caving had occurred. The practice of pre-developing slushers was phased out with increasing depth, for obvious reasons.

The photographs accompanying Deacon's Zandpan paper belie the often quoted statement that stress fractures in the hangingwall of cave panels ideally dip away from the face. His photographs clearly show the majority of stress fractures inclined in the opposite direction, towards the back area.

3.4.2 Current Caving Practice.

Zandpan and Hartebeestfontein merged in 1972 to form the present day Hartebeestfontein Gold Mining Company and cave mining has continued to be successfully employed as the principle stoping method. Stick caving was adopted throughout the mine, and is still practised in essentially its original form. The only significant change has been the reduction of the maximum face to support distance from 4,0m to 3,5m.

There has in recent years, however, been a swing away from stick caving towards prop caving in the deeper areas of the mine, as reported by de Witt, Greeff and Kersten⁽²⁰⁾ in 1989. This was motivated by the increased rates of convergence and increased levels of seismicity associated with mining at depths exceeding 2000m below surface. Despite Deacons earlier

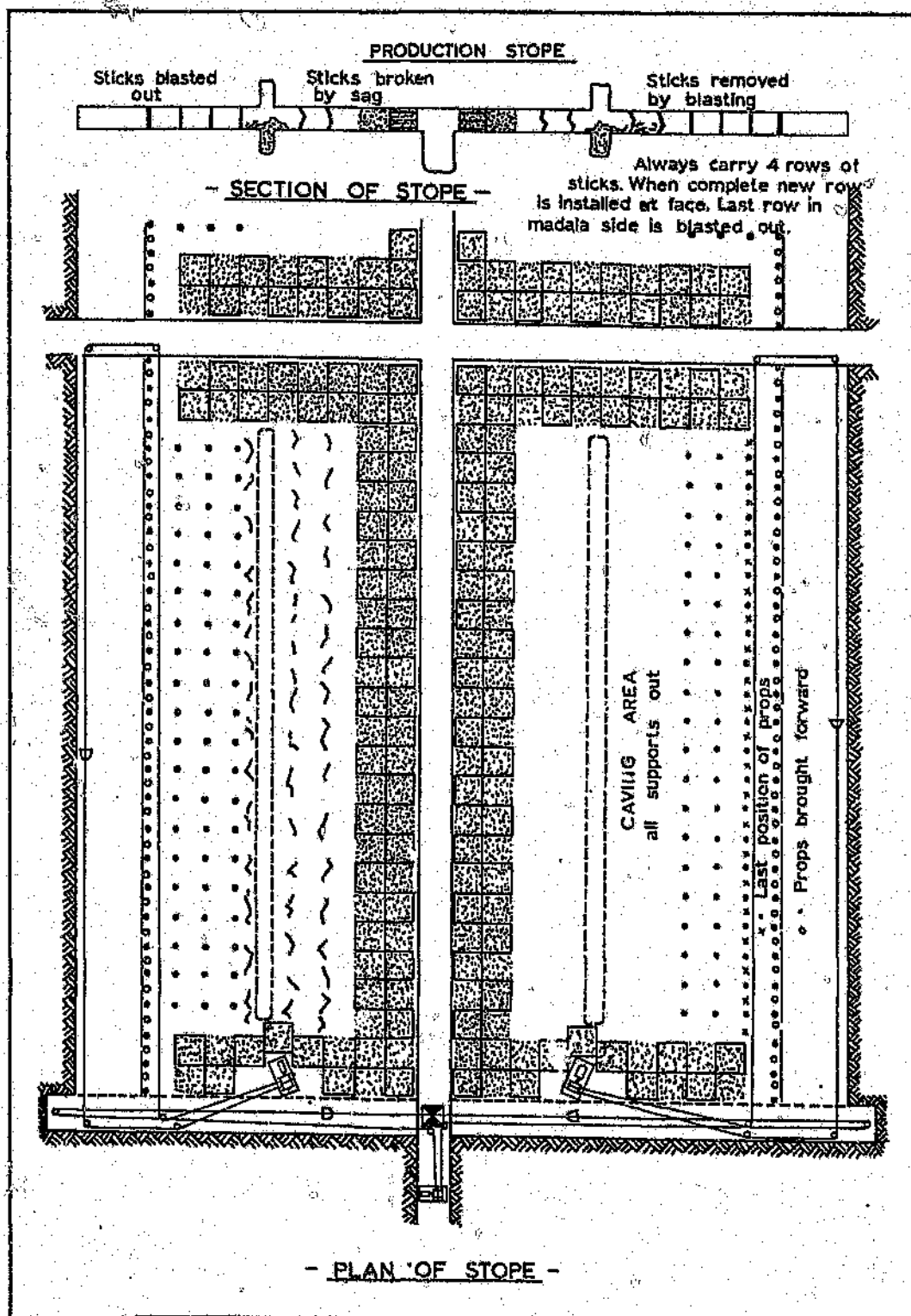


Figure 23 - Plan illustrating the Stick Caving Method applied on Zandpan Gold Mine.

reservations concerning the suitability of steel props, cave mining on Hartebeestfontein is today predominantly practised using three rows of rapid yielding hydraulic props.

In contributed remarks to a paper by Tupholme⁽²⁰⁾ in 1972, Wheeler and Cloete⁽²⁵⁾ reiterate the concept of macro-plastic behaviour, and highlight the role of the thinly bedded quartzites and inter-layered argillaceous partings associated with the immediate hangingwall and footwall of the Vaal Reef. They maintain that the success of the stick caving method, perfected on No.7 Shaft at depths between 2000m and 2200m, lies in the ability to regulate relative movement between quartzite layers and to inhibit bending and tensional failure of the 15cm-55cm thick quartzite beams. This is achieved by controlling frictional and lateral resistance to movement between layers, using the twin techniques of manipulating the rigidity of face support, and ensuring adequate caving of the back area.

The achievement of this delicate balance between inhibiting bedding plane separation, maintaining lateral compression of the hangingwall, and allowing sufficient lateral movement of strata to reduce stress levels ahead of the face, is a relatively straightforward matter. It involves the monitoring of the following well defined "malfunction indicators", and taking the necessary action to correct the imbalance:-

a) Fracture Inclination.

The inclination of face parallel, stress induced fractures, particularly in the middle of cave panels, should be vertical, or, better still, should be steeply inclined towards the face. Fractures that are steeply or shallowly inclined over the back area result from excessive stress build up on the face, indicating that an adequate cave has not taken place. Caving should be induced, or other stress relief measures, such as slotting, should be taken.

b) Footwall Heave.

The occurrence of heaving or doming of the footwall behind and between the first or second rows of support (which differs noticeably from punching of support units into the footwall), is also caused by increased stress at the face, indicating that lateral movement is being restricted in either the footwall, the hangingwall, or both. If caving is taking place normally, the provision of a footwall slot will allow lateral movement and reduce stress at the face.

c) Excessive Barring.

One of the first indications of excessive stress build up at the face is an unusual increase in the amount of barring or dressing required. This indicates that an abnormal situation has developed, which can be quickly rectified by applying the appropriate stress relief measure.

d) Jumpers Sticking.

Percussion holes drilled into the stope face for blasting purposes are prone to rapid scaling under high stress conditions, and take on a flat elliptical shape. In addition, closely spaced, face parallel fractures create thin slivers of rock which tend to displace relative to one another, forming off-set steps in drill holes. This leads to difficulty in extracting drill steel from these

holes after drilling. Once again, an abnormal situation is indicated, and an appropriate stress relief measure will remedy the problem overnight.

e) **Excessive Closure.**

Abnormally rapid closure is an indication of high stress at the face, and soon causes premature failure of support units. Sticks start to break in the second or third row, instead of in the fourth row. Props soon run out of travel, and start to freeze. If left unattended, they may punch into the hangingwall or footwall, and may need to be blasted out.

f) **Hangingwall Buckling.**

Buckling of the hangingwall between the front row of support and the face is an indication that excessive support resistance is preventing shear movement between hangingwall layers. This can lead to instability and falls of ground at the face. The condition is fairly rare, as other factors mentioned above clearly indicate that stress relief measures are required long before this stage is reached.

In short, mining personnel engaged in caving operations are able to monitor a number of simple visual indicators on a daily basis, which, if correctly interpreted, give a clear picture of prevailing stress and strata conditions, and an early indication of any abnormal situation developing. These include keeping a check on barring and drilling difficulties, fracture orientation, hangingwall and footwall buckling, and support behaviour. They then have at their disposal a number of remedial measures which are usually very effective in restoring normal mining conditions. These include: inducing a cave, cutting a slot, and/or increasing the rate of face advance.

3.4.2.1 **Stick Caving.**

The current stick caving standard calls for four rows of 150mm minimum diameter timber sticks to be installed on a square pattern 1,2m apart on dip and strike, and a maximum of 3,5m from the face after the blast. Temporary support units, in the form of sprags or mechanical props with headboards, are required a maximum of 1,5m from the face spaced 2m apart on dip (except during scraper cleaning operations).

A blast barricade is suspended from the face row of sticks, and a fines barricade on the second row. A no-entry barricade is suspended from the back row of sticks.

After the installation of a new row of sticks at the face, the fifth, or back row of sticks must be blasted out concurrently with the face blast. To assist caving, short holes must be drilled into the hangingwall between sticks in the first or face row, these to be blasted if necessary when they become situated in the back row.

Double-pack support is installed on gully edges to a checker-board pattern, and must be complete up to the second row of sticks from the face. A number of pack types are used for this purpose: solid mats, brick composites, end-grain hercules, waste-filled skeletons, and others. The size of pack unit must be matched to the prevailing stopping width.

Hangingwall and footwall slots are required to be cut adjacent to the centre gully ledge packs. These must be to a depth and width determined by the rock mechanics official.

3.4.2.2 Prop Caving.

The current prop caving standard calls for three rows of 300kN rapid-yielding hydraulic props to be installed on a square pattern 1,0m apart on dip and 1,5m apart on strike, also at a maximum distance of 3,5m from the face after the blast.

The removal of props from the back row to facilitate caving must be carried out remotely from a position of safety between the first and second rows of props. Prop removal must commence in the centre of the panel, or at the weakest point in the panel, and must then proceed in both directions away from the starting point.

To assist caving, short holes must be drilled into the hangingwall between props in the first or face row. These are to be blasted if necessary behind the back row, after the removal of props and the installation of temporary support.

The required standards for temporary support, blast barricades, no-entry barricades, gully packs, and for cutting hangingwall and footwall slots, are precisely the same as described for caving with sticks.

Figures 24A and 24B show the mine standards for caving with sticks and with hydraulic props respectively.

A minor variation of this prop caving standard, which is in limited use in various parts of the mine, involves the incorporation of additional sticks or props into the back row to form a breaker, or barrier line. This is used to overcome localised cave over-run problems.

Other, more radical, modifications to the prop caving standard, which were developed and introduced to deal with weak hangingwall conditions, and which form the major part of the project work described in this report, will be discussed and dealt with in detail later.

3.4.2.3 Caving Problems.

Cave mining is applied on approximately 75% of all working panels on Hartebeestfontein Mine, with about 40% on stick caving and 60% on prop caving. The more important advantages of the caving method are as follows:-

- a) Caving is economically more attractive than other methods which employ back area support.
- b) Provided caving is correctly implemented, statistics show the method to be comparatively safe.
- c) Changes in strata conditions can be monitored and accommodated by observing well defined, physical indicators.
- d) Cave mining fosters a disciplined and methodical approach to the stoping operation.

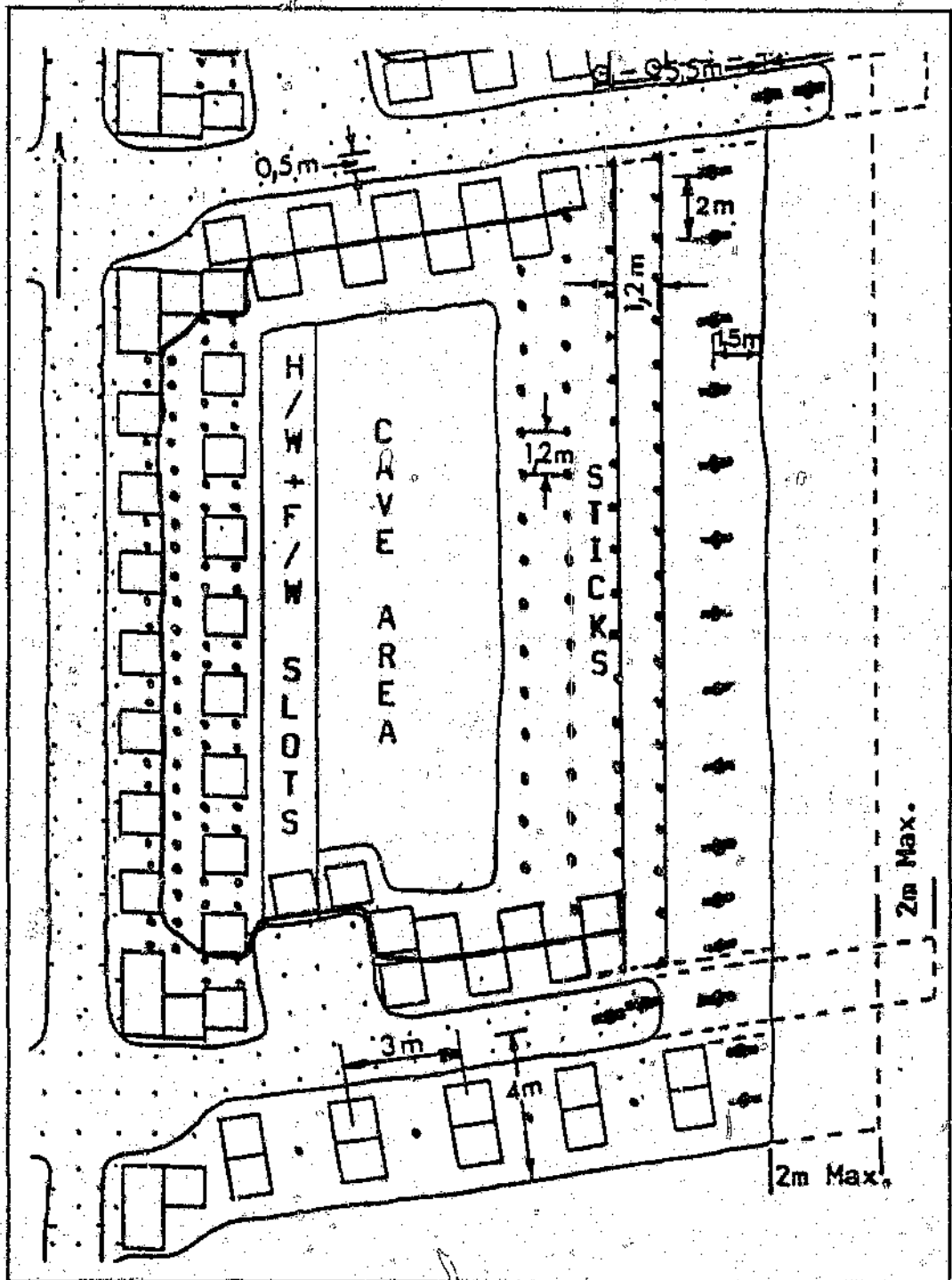


Figure 24A - Hartebeestfontein Mine Standard for Caving with Sticks.

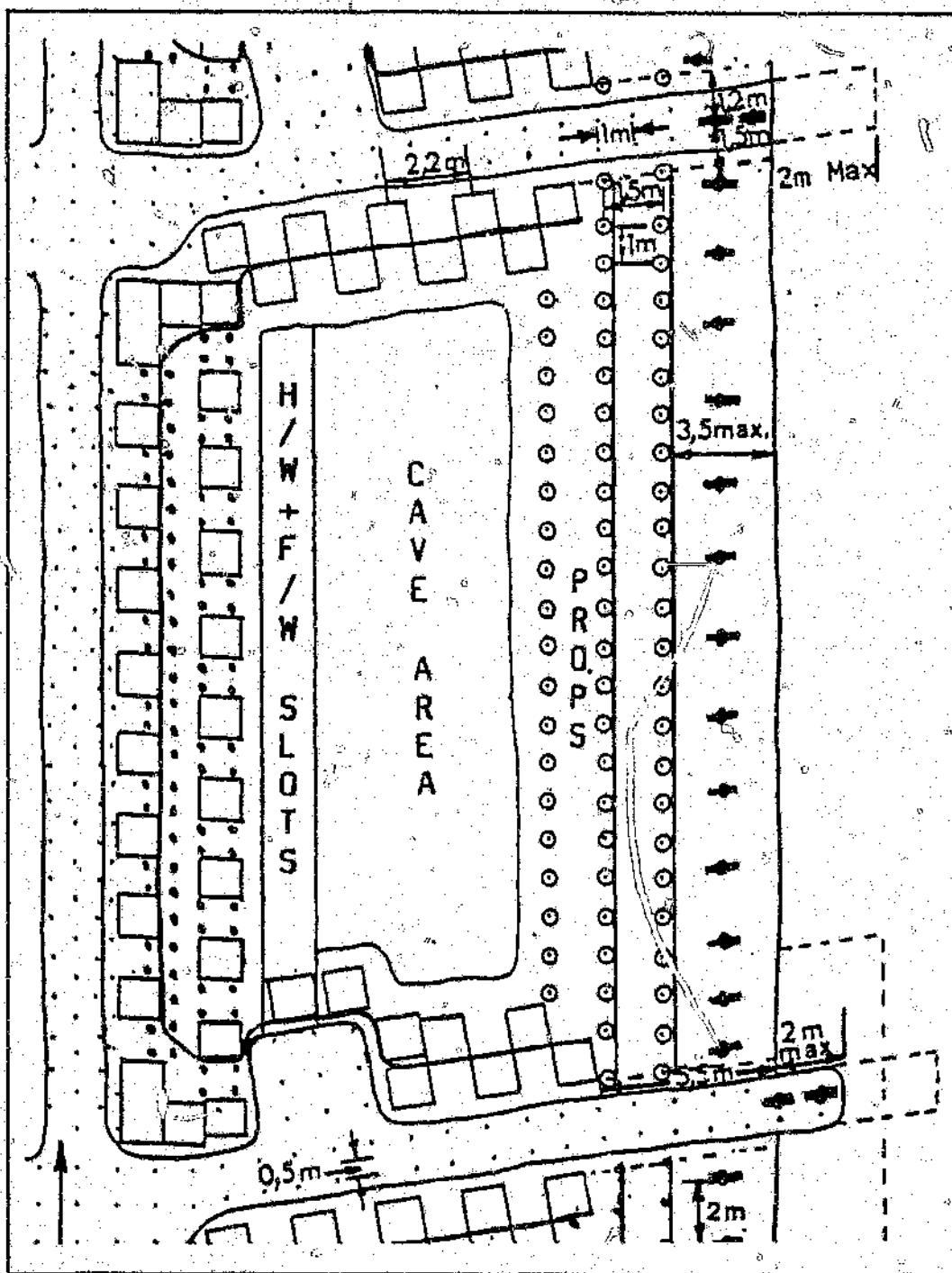


Figure 24B - Hartebeestfontein Mine Standard for Caving with Hydraulic Props.

Caving is, however, not without its problems. The following hangingwall control difficulties are sometimes associated with the method:-

- a) The failure to achieve back-area caving because of the failure to provide hangingwall slots and/or to initiate the cave by blasting or other means.
- b) The non-standard, but intentional practice of inhibiting the cave by installing and leaving sticks and/or packs in the back-area when conditions deteriorate.

Plates 1 and 2 illustrate situations in which hangingwall caving has been discouraged or inhibited. It is found that cave-inhibiting is generally resorted to in the following circumstances:-

- a) When conditions in stick and prop cave panels deteriorate or become dangerous, rows of packs, which later inhibit caving, are installed at the face to provide areal support and restore control.
- b) In such circumstances, it is also not uncommon, in stick caving panels, for stick support not to be blasted out, as is required by the standards, and to leave them standing in the back area where they inhibit the cave before breaking.
- c) In blocky or friable ground conditions, where there is a danger of the cave over-running the back row of props in prop cave panels, sticks and/or skeleton packs are installed to inhibit the cave and to prevent over-runs.
- d) Similarly, where prop removal operations are deemed to be dangerous, and/or where there is the likelihood of the loss of props because of poor hangingwall conditions, the cave is inhibited.
- e) Prop removal in the caving environment is a hostile operation. When the cave occurs on-shift, as is normal with prop caving, it is dusty and unnerving. Inhibiting the cave avoids this unpleasantness.

Loss of control of the hangingwall and cave over-runs occur largely as a result of the following factors:-

- a) Failure to achieve or induce a cave.
- b) Low or insufficient support resistance at the face.
- c) Inherently weak ground conditions.
- d) Inadequate rate of face advance.
- e) A combination of two or more of the above factors.

This is borne out by the observation that the phenomenon has become more prevalent since the change-over from stick caving to prop caving, and occurs more commonly in areas of weak ground such as the pseudo-breccia zone. The stress imposed on the hangingwall and footwall by a small diameter stick or prop foot-piece is in the order of 25MPa, which is considerably in excess of the stress required to punch into weak rock. When this occurs,



Plate 1 - Caving Discouraged By Failure To Blast Down Or To Provide Slots.

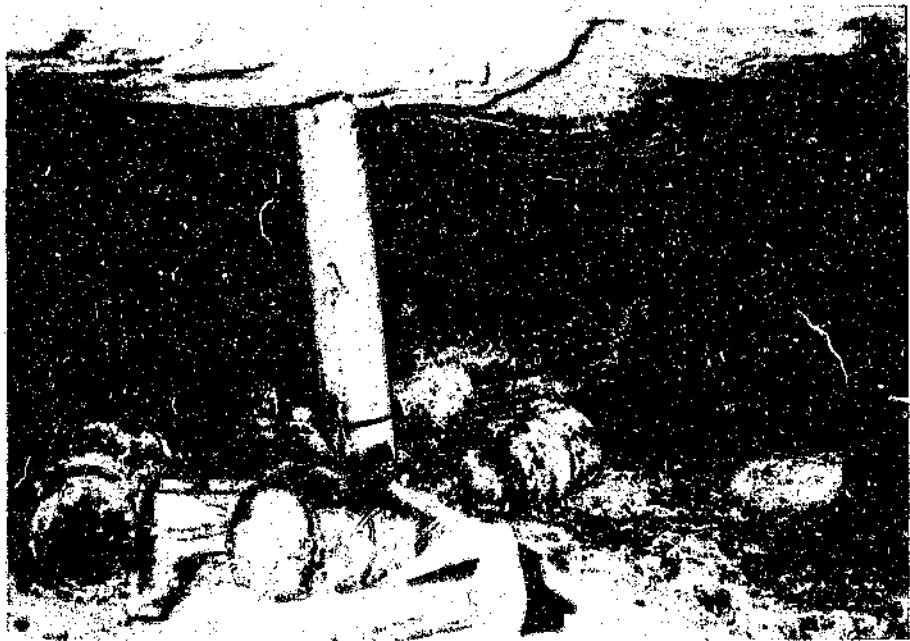


Plate 2 - Caving Inhibited By Leaving Sticks In The Back Area.

support resistance is dramatically reduced, resulting in the observed phenomena: loss of horizontal restraint, falls of ground at the face, increased stoping width, and cave over-runs. A number of solutions present themselves for consideration:-

- a) Stricter control to ensure effective caving.
- b) Increasing the support resistance.
- c) Doubling-up the back row of props to form a barrier line.
- d) Introducing temporary blast-out sticks.
- e) Authorising the practice of inhibiting the cave under certain controlled conditions.

3.4.3 Theory of Caving.

Nearly all discussion of the mechanisms or mechanics of caving in tabular deposits at depth involve some reference to macro-plastic flow or pseudo-plastic deformation of the rockmass surrounding the stope excavation. The ability of researchers or commentators to accurately and reliably model this type of non-linear behaviour has been, and remains, limited. Such models are by their nature complex, requiring cognisance to be taken of the time-dependant nature of this type of deformation. While non-linear computer modelling facilities are becoming more readily available, they are, relatively speaking, still in their infancy.

In trying to explain the phenomenon, a comparison is usually made between areas (or mines) which do cave, and those which do not. The fact that mining men involved on non-cave mines regard caving as "hazardous", cannot be ascribed simply to a reluctance on their part to try something new, or to take a chance on, what is in reality, a well tried and proven mining method. The financial benefits of caving are too great, and the fall of ground safety record of caving stopes too impressive to allow this deduction. One is intuitively drawn to the conclusion that geology, or more precisely, the presence of highly laminated or stratified strata, plays a major role in caving. A further conclusion is that it is the friction between these beds, and not necessarily the strength of the rock within individual layers, which is the important feature determining whether caving is a viable mining option or not.

By way of illustration, consider the following three simplistic and idealised visualisations of a stoping excavation at a depth of, say 2000m below surface. Lamination is limited to two bedding planes, one forming the stope hangingwall, the other, say one metre above, creating a 1m thick hangingwall layer. Imagine that, although there is no cohesion across the bedding planes in each case, the resistance to shear, or friction between the layers, can be very different. Also, the rockmass in each instance is fairly homogeneous, and the rock material itself (not necessarily the rockmass) behaves elastically rather than plastically. Support is similar in each case and is to a typical cave mining standard, consisting of a 3m wide zone of rigid support installed, say, 4m behind the face, with no support in the back area.

In the first case, imagine that no stress fractures develop in the hangingwall beam. The rockmass will rebound into the stoping excavation in the usual elastic fashion. The hangingwall beam will, however, fail at some critical span by shearing off at the face abutments, crushing the face support. Clearly, the full mass of the beam needs to be supported, implying that back area support is required.

The second situation is similar to the first, with the exception that, now, the continuity of the hangingwall beam has been destroyed by near vertical stress fractures induced ahead of the face. The dilation associated with this process sets up horizontal stress in the beam. Consider also in this case, that friction between the beds is high. Failure will take place in the back area when the mass of the beam overcomes the horizontal clamping stress. The support will hold up the portion of the beam immediately above it, but, because there is no lateral sliding, horizontal clamping stress in the beam between the support and the face is reduced as a result of the cave, causing, or increasing the risk of, failure in this area. Once again, back area support is required to maintain the integrity of the entire hangingwall beam, including the face area.

The third and final scenario is precisely the same as the second, except that friction between the beds is now very low. In fact, imagine that a small thickness of plastic material such as clay, shale or argillite is contained between the beds. Dilation ahead of the face is now translated into fairly substantial horizontal movement towards the stoped-out area. The behaviour of the rockmass as a whole takes on a pseudo-plastic aspect. Failure of the beam in the back area need not necessarily lead to failure at the face in this case, provided the support zone offers some limited resistance to lateral movement, thereby maintaining horizontal clamping stress in the portion of the beam between the support and the face.

These elementary examples, which attempt to explain why back area caving seems to work only in some deep level mining areas and not others, are illustrated in Figure 25. Some fairly basic UDEC²⁰ modelling of these simple scenarios was also attempted. The results, which are summarised in Figure 26, tend to confirm the intuitive premise that a large degree of plasticity is required to maintain stability in caving operations. In the initial case, where the beam is unfractured, zones of high shear stress occur adjacent to the face abutments, indicating that a natural beam failure mechanism would pertain. In the second case, where the beam is fractured and has a large resistance to horizontal sliding, total collapse occurs in the back area and in the unsupported face areas. In the final case, where the beam is fractured and has a low resistance to sliding, caving only occurs in the back area, and not between the face and the support, provided the distance between face and support is sufficiently small, and the support resistance is sufficiently large. The input parameters for these models are presented in Appendix 2, and the modelling results themselves are discussed in more detail later.

The theory or philosophy of cave mining on Hartebeestfontein, as propounded firstly by Deacon and latterly by Cloete, is firmly founded on the concept of plastic, or macro-plastic, deformation of a well bedded rockmass in which the interfaces between individual beds are well lubricated. The concept, outlined by Wheeler and Cloete in their 1972 contribution to Tupholme's paper, envisages individual layers, and the rockmass as a whole, flowing into the stoping excavation.

In tabular excavations, the load previously carried by a stoped out area is transferred to the abutments. If the rockmass is truly elastic, the vertical stress distribution would be as shown by curve "A" in Figure 27 - infinitely high right at the face, tailing off to primitive or virgin stress some distance away. In reality, however, the strength of rock is finite, and failure takes place for a short distance into the face. The transition from failed to intact rock will occur at the point where the strength of the rock exactly balances the triaxial loading condition. This point will coincide with the stress peak, which will be finite and lower than the elastic abutment peak. Because the additional load carried by the abutment is the same as for the truly elastic case, the tail of the curve will also be shifted such that the area under

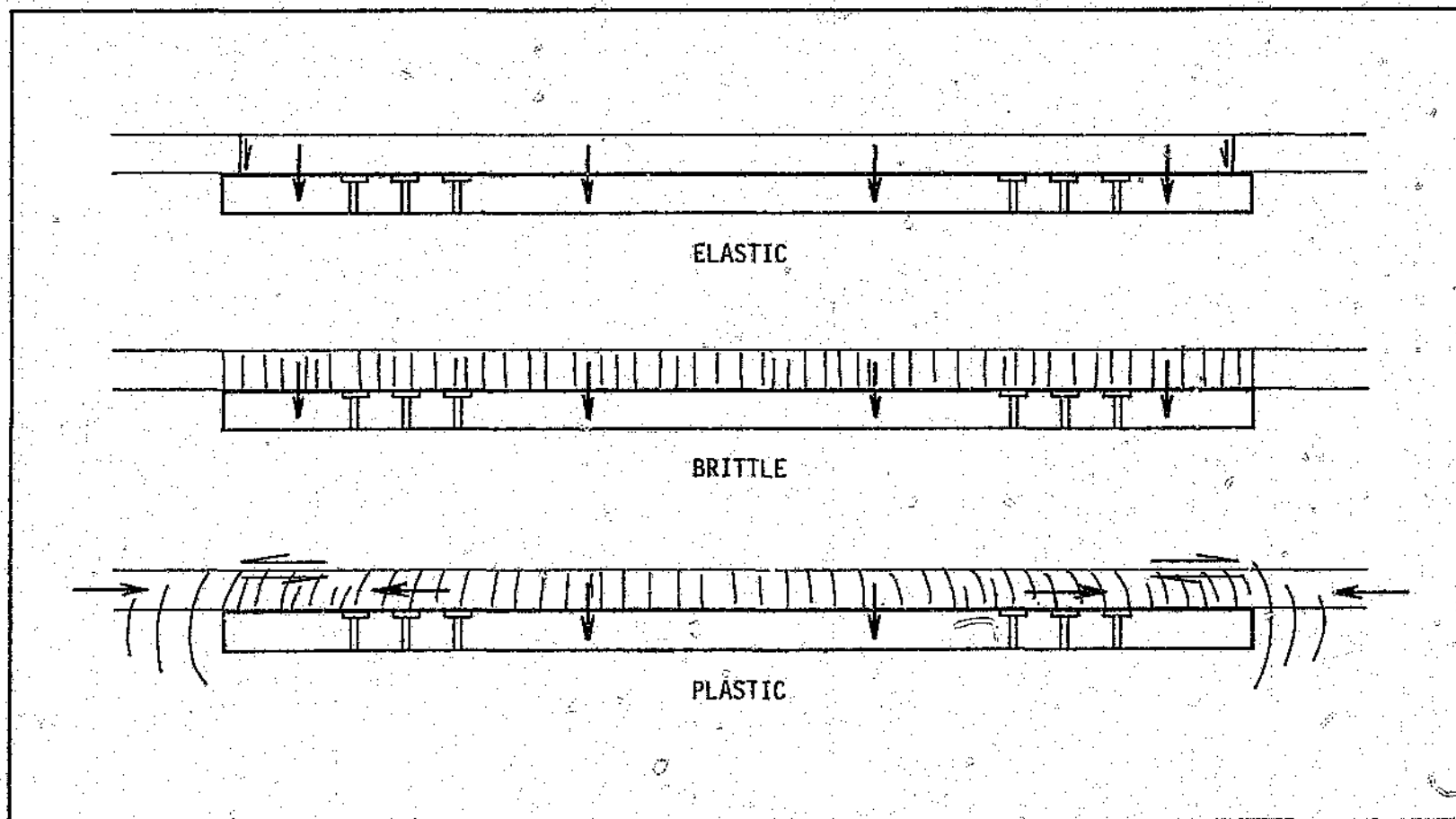


Figure 25 - Simplified and Idealised Comparison of Caving Behaviour in Elastic, Brittle and Plastic environments.

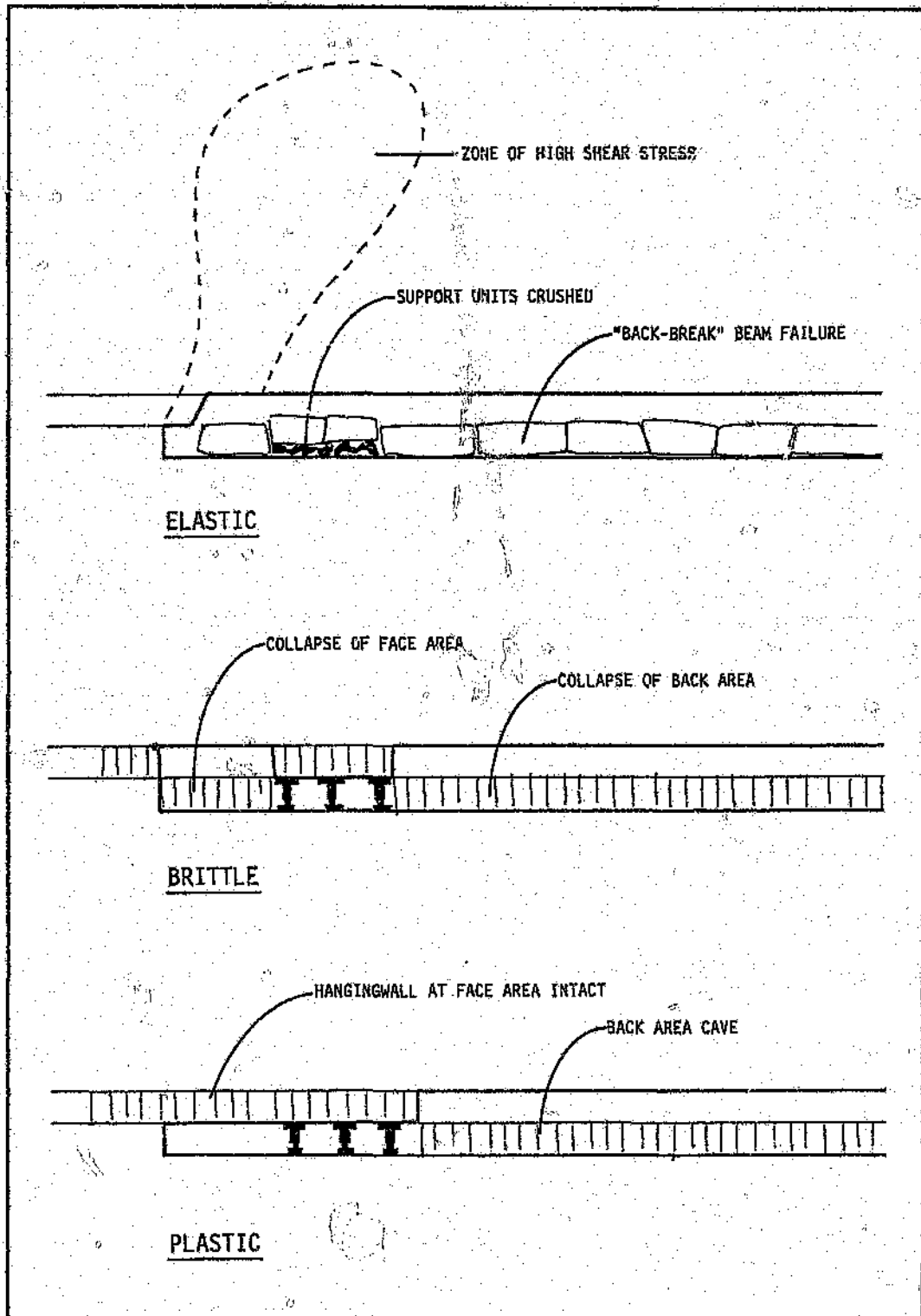


Figure 26 - Results of Computer Modelling of Some Fundamental Cave Mining Scenarios.

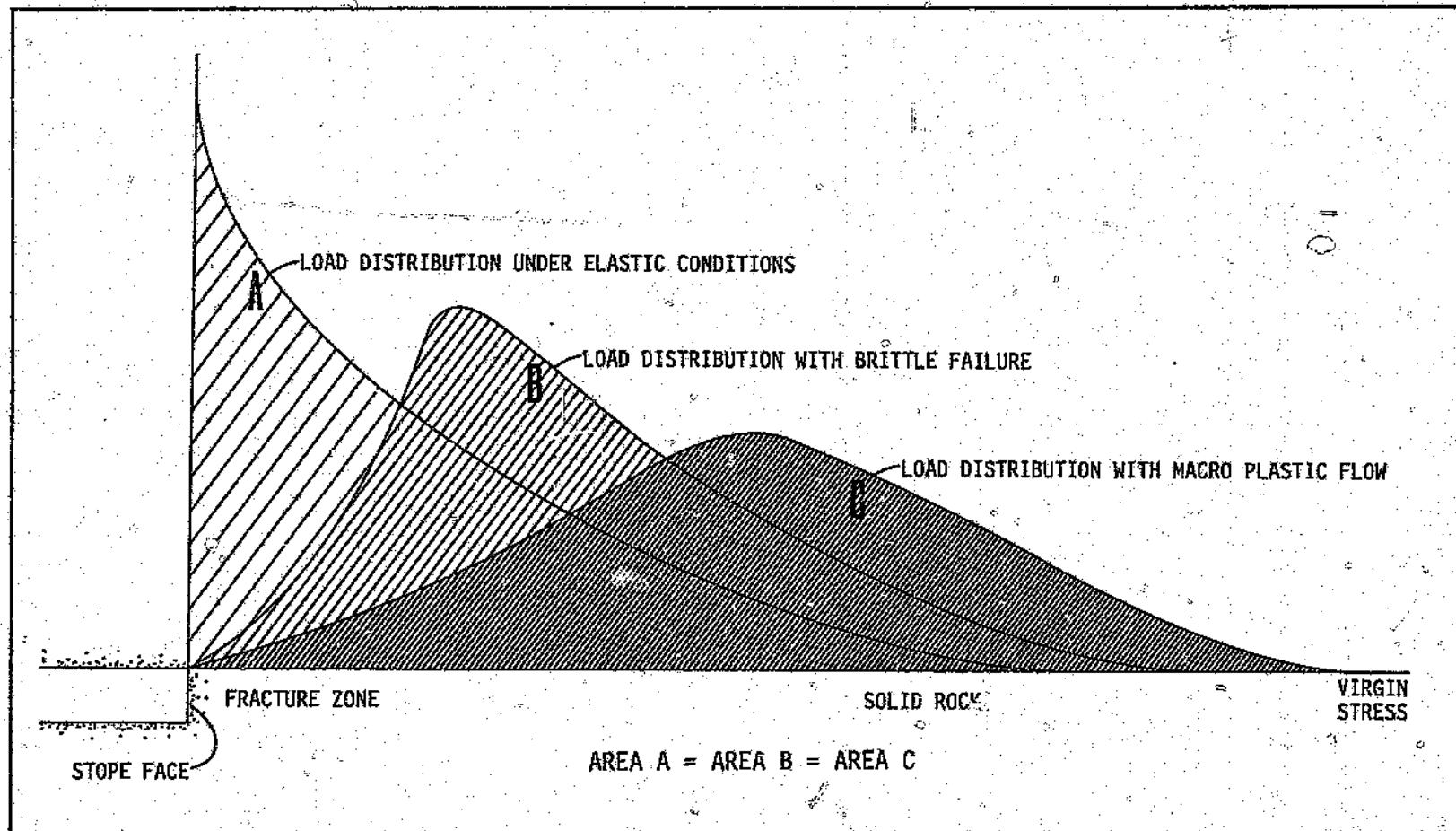


Figure 27 - Illustration of Theoretical Vertical Stress Distribution ahead of a Slope Face for Elastic, Brittle and Plastic Conditions.

this curve, shown as "B" in the figure, is equal to the area under curve "A". For macro-plastic flow conditions, further lateral restraint is lost. The peak is even lower, and is shifted yet further away from the abutment, as shown by curve "C", the area under the curve remaining substantially constant.

Thus, while the overall induced load resulting from stoping operations remains constant regardless of rock behaviour, under macro-plastic flow conditions, it is distributed over a larger area, further ahead of the face. Consequently, induced stress levels are relatively low, and stress induced fracturing less intense. If, however, macro-plastic flow is opposed or restricted, by uncaved hangingwall for example, the stress distribution ahead of the face tends to revert back towards the brittle condition, resulting in higher stress levels and more intense fracturing at the face.

In addition to the intensity of fracturing, caving is also purported to modify the geometry of the curved fracture front. The inflection plane, which in non-cave stopes lies close to the axis of symmetry of the excavation, is shifted up into the hangingwall in caved stopes, to coincide more closely with an axis of symmetry which includes the cave region. The curvature of the fractures is also less sharp. Hangingwall fractures at the face are thus more steeply orientated, and may even incline towards the face, as illustrated in Figure 23. These modifications are thought to have a beneficial influence on the stability of the hangingwall.

Rock failure, fracture generation and, particularly, macro-plastic flow processes which occur ahead of the face, and which are responsible for inducing horizontal stresses in the hangingwall and footwall, are time-dependant phenomena. They rely on continuous face advance to regenerate and maintain clamping stress in the hangingwall. It follows that, if the rate of face advance reduces, support resistance must increase to compensate for the gradual loss of horizontal clamping stress which will follow. Similarly, if the rate of face advance increases, support resistance must be reduced so as not to restrict the macro-plastic flow mechanism.

The strategy of effectively trading off or shedding a proportion of horizontal hangingwall clamping stress for the purpose of reducing and transferring stress from the immediate face area, is highly dependant upon the continual regeneration of horizontal stress in the hangingwall by the macro-plastic flow mechanism and the attainment of a regular and adequate rate of face advance. If regeneration does not take place, back-area caving will result in the unclamping of the vertically sliced hangingwall strata at the face. The method is, for this reason, an anathema for most deep level mines, where high horizontal clamping stresses are vital for the stability of the stope hangingwall, particularly during dynamic or seismic loading.

The role of face to support distance in the maintenance of stable hangingwall conditions at the working face is as vital in caving stopes as it is in non-caving stopes. The spacing and installation of support as a function of face advance is important in this regard. This is particularly so where the immediate hangingwall layer is intrinsically weak. This aspect will be discussed in more detail later.

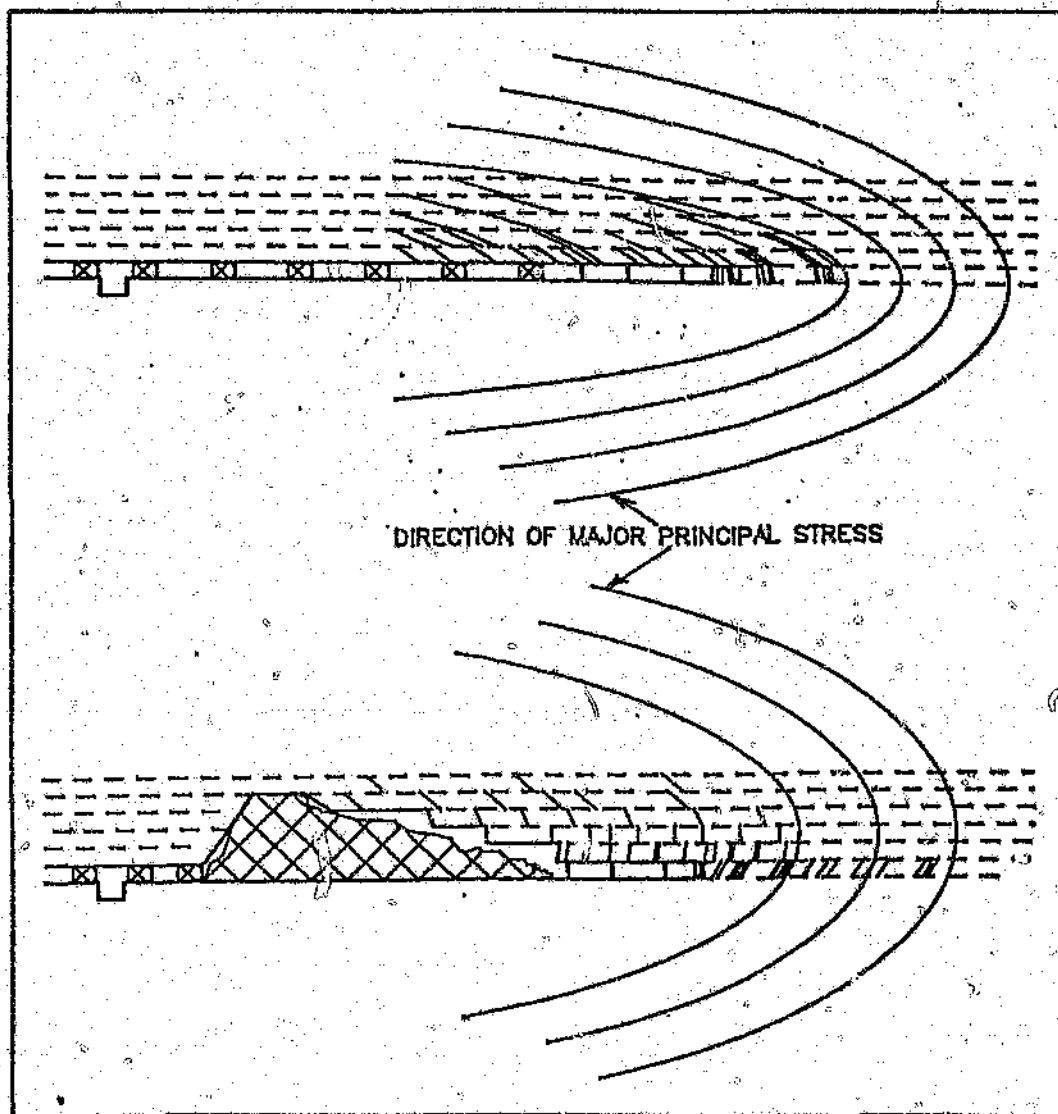


Figure 28 - Idealised section showing Fracture Development around a Non-Caving Stope and around a Caving Stope.

3.4.4 COMRO Research.

In 1986, the Chamber of Mines Research Organisation investigated cave mining operations in two stope panels situated at a depth of 2300m below surface at Hartebeestfontein No.6 Shaft. The results of this field trial were documented in a COMRO project report⁽²⁷⁾, and formed the basis of a Master of Science dissertation by Herrmann⁽²⁸⁾.

The trial set out to investigate rock deformation in two adjacent cave panels with markedly different face support. One of the panels was equipped with three rows of 200kN hydraulic props, giving a support resistance of 95kN/m², while the other was equipped with four rows of 400kN hydraulic props with additional 1600kN barrier props in the back row, giving a support resistance of 277kN/m². Figure 29 shows the general support configuration applied in each of the two panels. Stope closure, strata deformation and fracture orientation were monitored in these panels over a twelve month period, during which time the faces were advanced a distance of 120m to holing.

The findings of the COMRO investigation may be summarised as follows:-

a) Extent of Back-Area Caving.

The height to which the initial cave occurred was typically only 40cm, the distance to the first stratigraphic parting plane. This was followed by later caving which occurred 2m to 3m further back. The maximum height to which the cave extended was estimated from visual observation to be approximately 3m.

The forward extension of the cave normally took place simultaneously with, or a short time after, the removal of temporary remote release mechanical props, which were used as an additional safety device during the removal of the hydraulic props. However, the cave did on occasion hang-up for a distance of 10m or more on strike. Conversely, on a number of occasions the cave did also over-run the last row of props. The dip distance over which this occurred never exceeded 5m, and the collapse never extended beyond the next row of props.

The cave area was closest to the props in the mid-panel region during most of the experiment. At the top and bottom of the panels the cave front tended to lag, and extended back obliquely towards the gully-edge packs, with one exception at the top of the panel adjacent to a previously worked out area. Here, because of the presence of siding parallel fractures from the earlier mining, the cave extended up to the edge of the gully packs right against the back prop line.

b) Fracture Frequency and Orientation.

In the experimental panels, neither the fracture orientation, nor the fracture spacing was altered significantly by the occurrence of caving. The plane around which the fracture inclination inflected from forward-dipping to backward-dipping was situated in the footwall of the stope both before and after caving. Hangingwall fractures were thus predominantly inclined towards the back area. The spacing of hangingwall fractures before and after

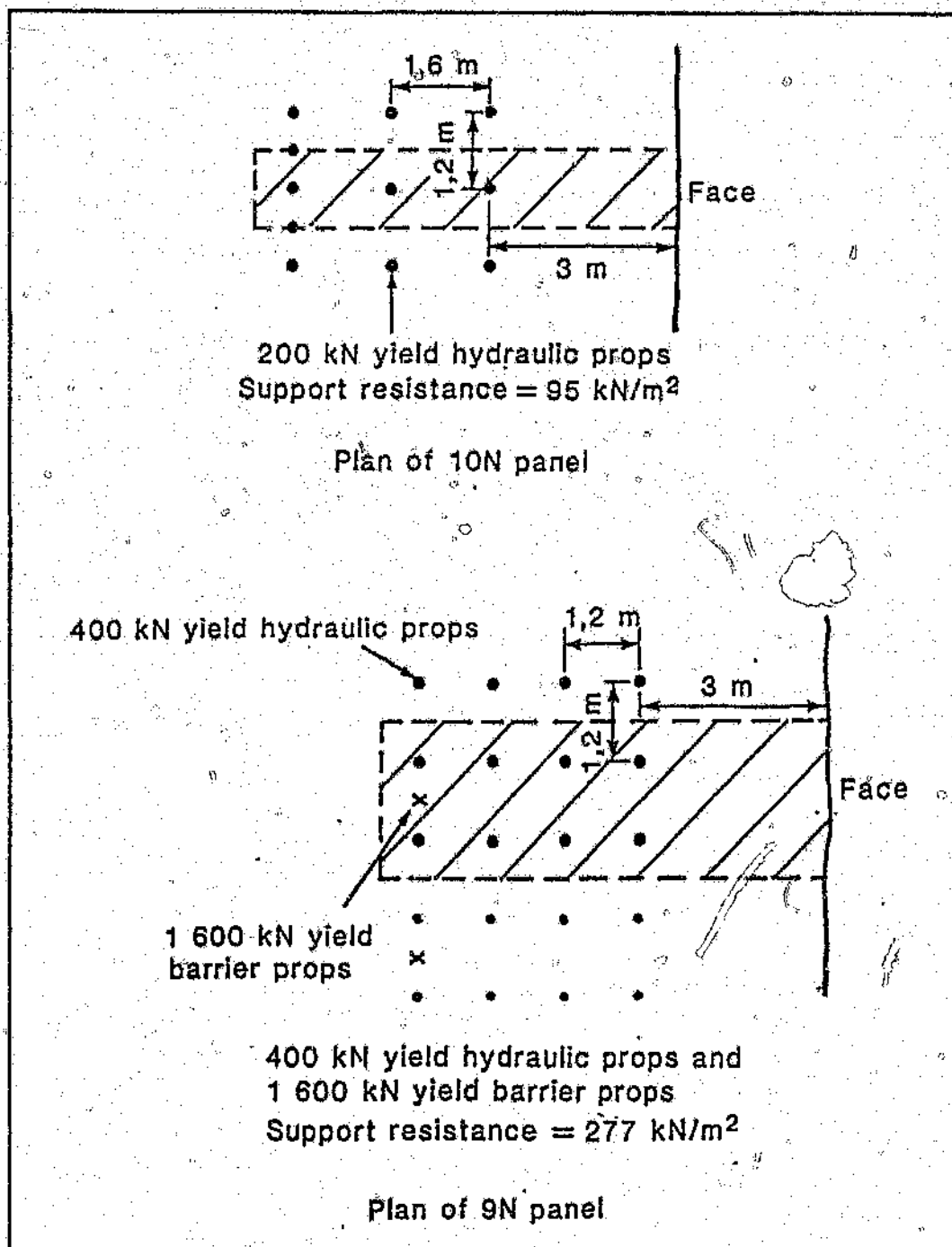


Figure 29 - Sketch of the General Support Configuration applied in each of the two COMRO Cave Mining Research Panels.

caving showed a slight increase from an average value of 208mm to 232mm apart.

This is atypical of experience elsewhere on Hartebeestfontein, where the plane of inflection moves up, the inclination becomes steeper, and the intensity of fracturing is normally reduced following caving; and can possibly be ascribed to the erratic face advance and long delays incurred during the installation of instrumentation, and the difference in support systems used at the experimental site.

c) Stope Closure and Inelastic Dilation.

As expected, inelastic deformation and separation of hangingwall and footwall strata resulted in actual stope closure rates far exceeding those predicted by elastic theory. A comparison of hangingwall dilation and stope closure revealed that, up to a distance of 55m from the face, attributable closure exceeded recorded dilation, indicating that dilation must have extended higher into the hangingwall than the 16m above the stoping horizon recorded by the extensometers. Total closure occurred on the ledges of the strike gullies 55m behind the face, effectively preventing further dilation of the strata.

Vertical dilation of strata was found not to be entirely restricted to the stoped out area, and was noted to initiate in the fracture zone ahead of the face. (It is, in fact, part and parcel of this fracture process). A large portion of dilation occurred between 8m and 10m above the stope, indicating the presence of a major, weak parting plane at this level. A substantial component of inelastic closure was found to be due to slip, caused by high horizontal stresses generated by fracturing ahead of the face, on hangingwall-parallel discontinuities such as these.

Closure measurements taken in the face working areas throughout the duration of the investigation showed no significant difference between the two panels, despite the very significant differences in applied support resistance. Closure in deep working areas is thus largely unaffected by conventional support types, which must be selected or designed in terms of their resistance to large scale deformation rather than just their strength. Although conditions in the two test panels were not significantly different, this is not to say that hangingwall stability is not affected by the degree of support resistance applied, especially under rockburst loading conditions. However, it is the distribution of support resistance which remains the important feature.

Contrary to what would be expected from an elastic analysis, and echoing Hill's earlier observations on ERPM, it was also noted that the rate of closure dropped as the mining span increased. As only a small fraction of the closure recorded can be attributed to elastic rebound of the rock in these circumstances, it is the intensity of fracturing and shearing, and the associated dilation of the fracture zone ahead of the stope face, which contributes to the amount of inelastic closure in the stope face area. Possible, but not very convincing, reasons for a reduction in these phenomena are that the remnant ahead of the test panels became progressively more stress relieved, with a

consequent decreased ability to fracture, or that bulked cave material in the back area reconsolidated, with a consequent reduction in the rate of stress increase in the face abutment.

d) **Horizontal Stress, Deformation and Ride.**

Horizontal stresses measured in the stope hangingwall during the course of the experiment indicated substantial and increasing compression of strata as mining proceeded. Stress relaxation occurred when slots were cut, but only up to the bedding layer in which the slot terminated. Bedding planes above this boundary were not affected by slot cutting. Also, relaxation in the lower layers was quickly followed by re-compression, particularly in the panel with the stiffer support system.

Large, horizontal shear displacements were measured across weak bedding planes in the stope hangingwall, and were observed to correlate well with measured vertical dilation, and thus the large component of inelastic closure in the stoping excavation. Clearly, shearing across non-planar and undulating horizontal features results in irreversible vertical opening of bedding planes.

Differential movement, or ride, between the hangingwall and footwall was strongly influenced by the presence of gullies and slots. Some down-dip movement of the footwall towards strike gullies was observed. Strike ride of the footwall towards the centre gully was also noted in the initial stages, but this soon changed to hangingwall ride once the hangingwall slots had been cut.

e) **Time-Dependant Deformation.**

Deformation of strata surrounding the stoping excavations seemed to take place on a continual basis, and did not appear to be strongly related to the erratic incremental nature of the blasting operation. Stope closure in particular was found to be markedly time-dependant, and persisted at a substantial rate (more than 5mm per day) for more than a month after stoping operations had ceased. While time-dependant behaviour is commonly observed and acknowledged, it is nevertheless not well understood.

The observations made during the course of this investigation led to the development of a conceptual model describing the inelastic behaviour of rock surrounding stoping excavations at depth. Figures 30A and 30B are diagrams showing the essential features of the model. Briefly, the development of stress induced fracturing and consequent horizontal dilation ahead of the stope face gives rise to high horizontal stresses and bedding plane shear in the hangingwall and footwall. Shear along uneven and undulating horizontal discontinuities and bedding planes in turn leads to irrecoverable vertical dilation and buckling of hangingwall and footwall into the stoping excavation. Far from being mere gravity induced separation on bedding planes as previously envisaged, the large inelastic component of closure is now recognised as being driven by substantial and, from the point of view of conventional support types, irresistible forces. The model also explains why stresses measured in backfill invariably exceed elastic predictions.

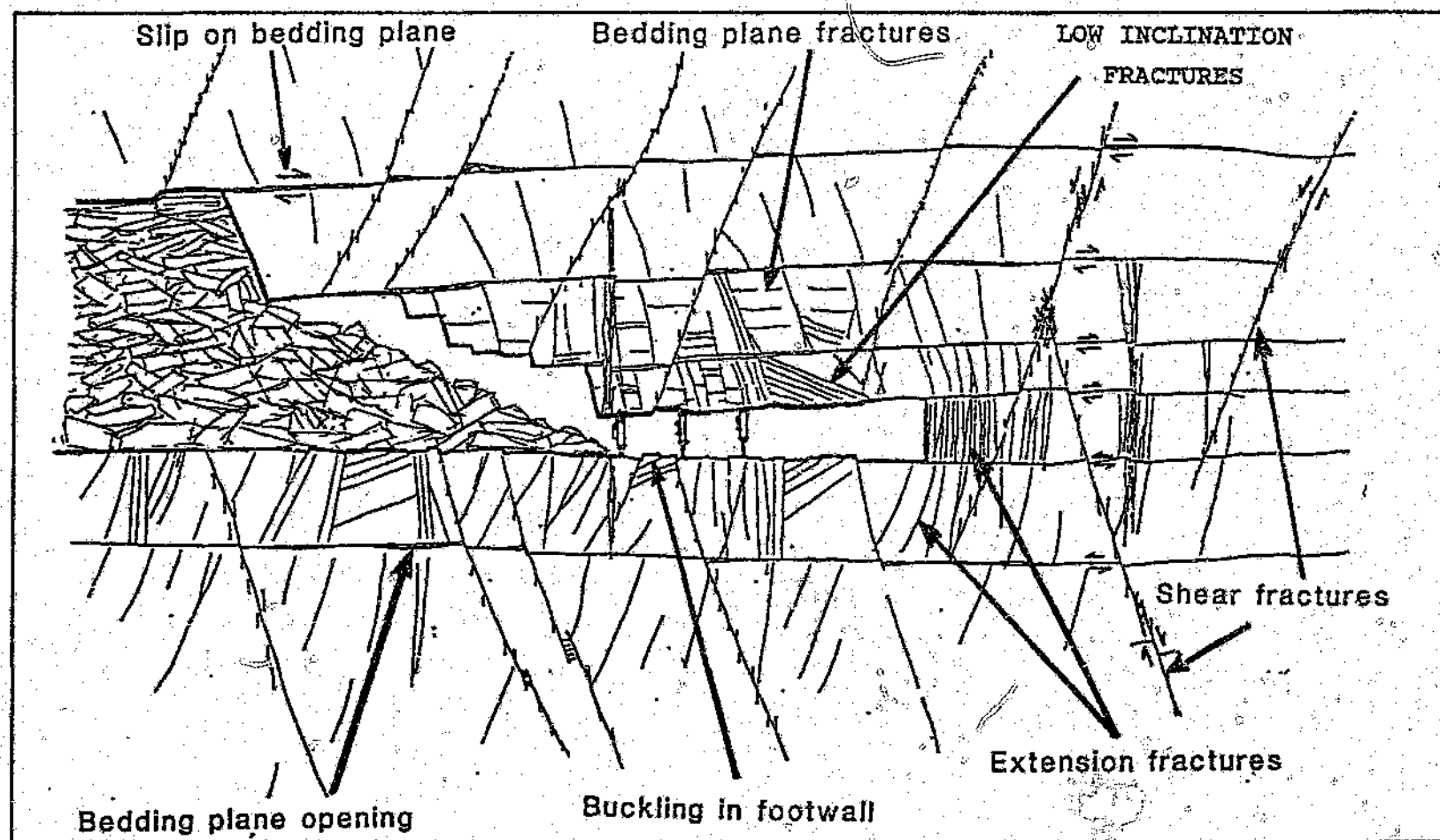


Figure 30A - Conceptual Model of Fracturing and Deformation Around a Deep Stopping Excavation.

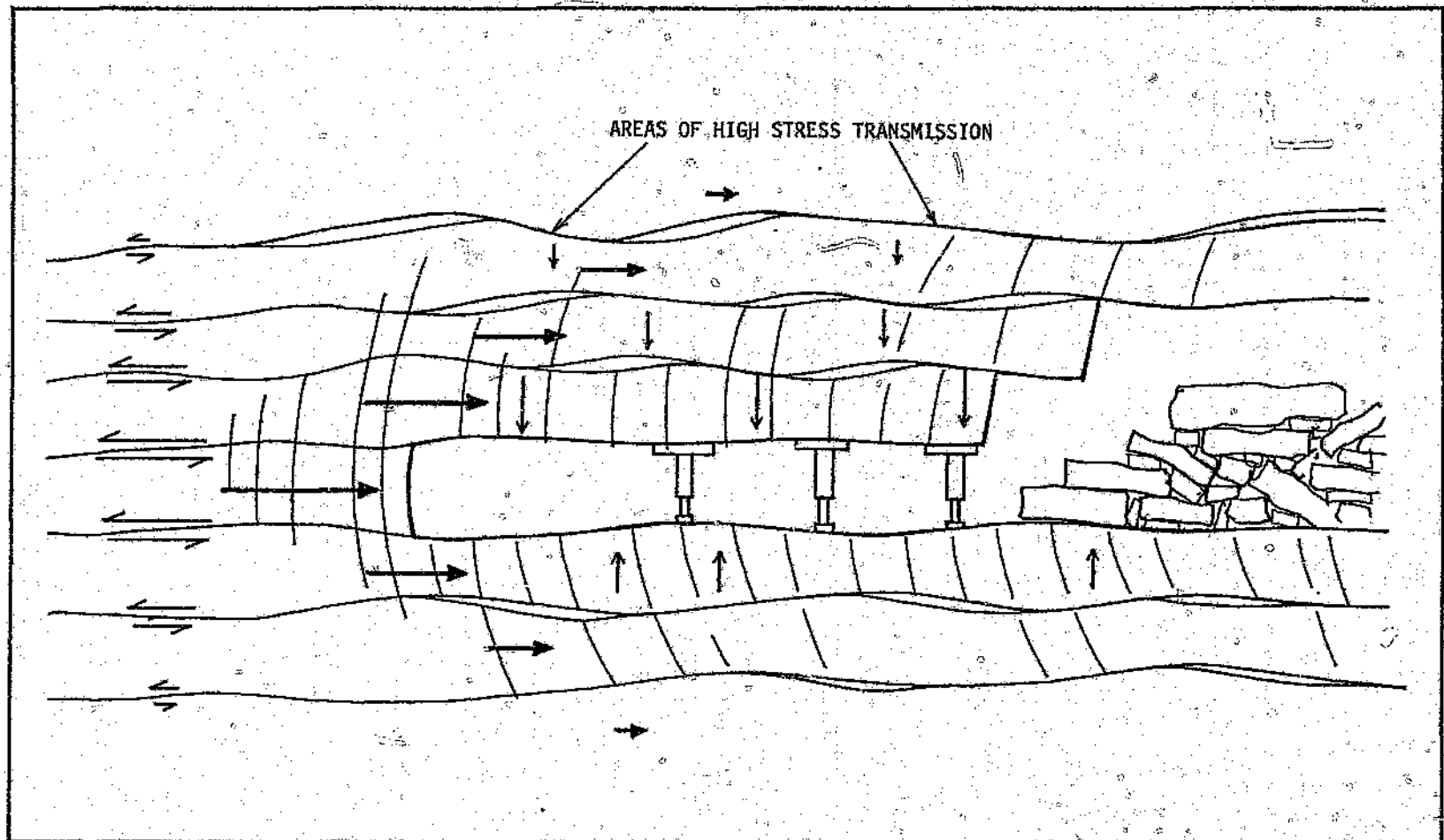


Figure 30B - The Effect of the Fracture Process and Shear Displacement on Bed Separation and Stope Closure.

4. HANGING WALL STABILITY.

4.1 STOPING WIDTH CONTROL.

Stoping width control is vitally important in determining the profitability of narrow (say less than 850mm) reef-channel operations. Stoping width essentially dictates the grade at which a narrow-reef panel is worked. Consider by way of simple example the case of a 1000cm.g proposition being worked at an average break-even grade of say 8g/t. At a stoping width of 100cm, the grade will be 10g/t. If the stoping width were reduced to 90cm, the grade would be slightly in excess of 11g/t; a 10% stoping width reduction giving an 11% improvement in grade. Of more importance, however, is the improvement to the profit line represented by the 11% increase in grade: a 5% increase in profit can be achieved provided constant tonnage is produced, i.e. provided the face advance is increased commensurately to compensate for the reduction in tonnage caused by the reduced stoping width. A moderating factor in this regard, is that increased face advance is more readily achieved at lower, rather than higher, stoping widths. On Hartebeestfontein Gold Mine it is estimated that a 10cm reduction in the current average stoping width would result in direct benefits to the profit of the company in the order of eight million rand per month.

Naturally, the converse also holds true. Any increase in stoping width, and/or dilution of the ore resulting from falls of ground, can seriously impact upon stoping cost-benefit relationships. In extreme cases, profit margins may even be completely eroded. Stoping width and face advance are thus critically important economic aspects associated with narrow channel, tabular mining operations.

Besides the direct grade improvement benefits discussed above, a reduction in average stoping width has the potential to impact on literally every aspect of tabular, narrow-channel mining operations. Even assuming that there is no increase in areal extraction rate, constant gold will be achieved from fewer tons, creating the potential to reduce both working and overhead costs almost in proportion to the tonnage decrease. The following are examples of areas in which savings can be achieved:-

a) Rock Breaking, Handling and Processing.

A major proportion of underground and surface mining activity is directly, or closely, related to the transport and breaking of rock. If fewer tons are being generated by the system, direct savings should accrue from reduced scraping, tramming, hoisting, conveying and milling requirements.

b) Air, Water and Electricity Services.

The power demand for cleaning, tramming and hoisting, and for compressed air, pumping, ventilation and refrigeration services is closely related to volume of mining and the tonnage produced. A reduction in tonnage will impact favourably on these costs.

c) Stores, Consumables and Maintenance.

The consumption of materials and stores such as timber, rolling stock and reduction reagents, and the maintenance of equipment such as winches, locos,

crushers and conveyor belts, is almost directly related to stoping width and production tonnages. A reduction in either or both of these parameters will precipitate a reduction in costs.

d) **Infra-structure and Services.**

The provision and maintenance of the infra-structural systems associated with deep mining operations, such as engineering works, reduction plants, office buildings and housing, and the servicing of these installations, may be significantly reduced by a reduction in tonnage through-put.

e) **Labour and Staff.**

Up to 50% of the operating costs of an average mine are those associated directly or indirectly with labour. A reduction in stoping width and production tonnage will allow a reduction in front-line labour requirements. The domino effect will ensure that reductions can be made in all facets of the operation, and at every level of the organisation.

The question of the safety of persons is always of paramount importance in any mining operation. The incidence of most accident types can generally be related to the number of people employed, and hence to the tonnage produced. In deep mines, falls of ground usually account for the majority of the more serious accidents. These rock-related accidents are particularly closely related to stoping width and to the rate of face advance.

For a number of reasons to do with the effectiveness of support and the release of energy into the mining system, the incidence of both rockfalls and rockbursts, and the effects of seismicity, are substantially reduced in narrow stoping width environments. In addition, a rapid rate of face advance invariably results in improved hangingwall conditions. This is particularly the case with cave mining. Once again, stoping width and face advance are identified as critical aspects of the stoping operation.

The major issues to be addressed in respect of reducing stoping width and increasing face advance are, firstly, blasting and, secondly, support. More specifically, the issues are:-

- i) reducing current high levels of blast damage which are taken for granted,
- ii) increasing the efficiency of blasting operations to improve advance per blast,
- iii) reducing face to support and inter-support spans to increase areal coverage.

The reduction and containment of blast damage is best achieved by attention to drilling, particularly reducing the spacing or burden between adjacent blast holes. However, although blasting aspects must be an important and integral part of any strategy to reduce stoping width and improve hangingwall control, these aspects fall outside the scope of this project report, which will concentrate on the work done to develop support techniques to reduce stoping widths in one geologically disturbed section of the mine.

4.2 STOPE SUPPORT.

4.2.1 Purpose Of Stope Support.

Conventional stope support cannot control the rate of elastic and inelastic closure. The primary function of support installed at the stope face is thus to stabilise the rock making up the immediate hangingwall, with the objective of preventing falls of ground, thereby promoting safe and productive working conditions and avoiding unnecessary dilution of ore.

The nature of the hangingwall in a deep gold mine may vary from coherent and essentially self-supporting, to blocky and potentially unstable, and in extreme cases to very broken and in imminent danger of collapsing. Support strategies must be sufficiently flexible to maintain the integrity of the hangingwall under this wide range of conditions.

Many complex factors govern stope support requirements, but the design of any support system depends upon knowledge of three fundamental aspects: the nature and extent of the rock to be supported, the deformations to which the support will be subjected (normally defined by the closure profile), and the generated reaction load characteristics of the support elements making up the system. Because these aspects cannot in practice be assessed accurately enough to allow the strict engineering design of each installation, reliance is placed on qualitative understanding of the rock and support behaviour based on past experience. Stope support designs can be improved by combining past experience with an assessment of the underlying engineering principles.

4.2.1.1 Rock To Be Supported.

The extent above a stoping excavation of the tensile stress zone in which one or more of the principle stresses are tensile is a conservative way of characterising the region of rock for which support may be required. The height of this zone, according to elastic theory, depends upon the ratio of depth to span of the excavation, and decreases with increasing depth. Inelastic movements generate horizontal stresses which are not recognised in conventional elastic numerical models. At great depth the tensile zone is eliminated because abutment stresses are sufficient to fracture rock at and ahead of the stope face, with the resulting dilation inducing high compressive horizontal stresses in the immediate hangingwall and footwall of stope excavations. This allows large spans of broken hangingwall rock to be held in place and become essentially self-supporting.

Blocks are formed in the hangingwall by the presence of geological discontinuities such as faults, joints, bedding and cross bedding; by stress induced fractures such as extension and shear fractures; and by blast fractures. Unfavourably orientated discontinuities in the rock mass, whether large or small, natural or induced, can lead to the formation of unstable keyblocks if clamping forces are insufficient. The primary function of support at depth then becomes that of stabilising keyblocks and preventing unravelling and deterioration of hangingwall integrity. Hangingwall failures at depth are invariably bounded by combinations of pre-existing natural and induced discontinuity features.

The span over which the hangingwall is self supporting depends upon many factors such as the inherent strength of the rock, the thicknesses of bedded layers making up the hangingwall beam, the nature of the bedding planes, the intensity and nature of fracturing and geological

and other discontinuities, and the degree of horizontal clamping stress.

An important (perhaps the most important) aspect of deep-level stope support design then becomes that of economically minimising the areal extent of unsupported hangingwall between support elements and particularly between support elements and the stope face; especially in regions where weak, friable or blocky ground conditions prevail. This of necessity involves reducing the maximum face to support distance, reducing inter-support spacing, and the use of areal cover methods such as umbrellas or load spreaders.

4.2.1.2 Rockmass Deformation.

Underground excavations deform in response to both elastic and inelastic relaxation of the surrounding rockmass. The degree and rate of deformation is dependant upon excavation characteristics such as depth, span and geometry. Elastic deformation, which is essentially instantaneous, is dependent upon the elastic properties of the rockmass. Inelastic deformation on the other hand is highly time-dependant, and is strongly influenced by the strength of the rock material and the nature, frequency and orientation of discontinuities in the rockmass surrounding the excavation.

The horizontal dilation of rock towards an excavation in response to the fracture process occurring ahead of an advancing stope face can, for example, cause differential movement along undulating bedding planes resulting in vertical expansion of the hangingwall and footwall into the excavation. This is one of the mechanisms giving rise to inelastic closure in stopes. With cave mining this phenomenon is thought to be exacerbated as a result of low resistance to shear displacement between bedding planes, and the freedom afforded the hangingwall to displace horizontally into the caved area.

Stope closure at depth is largely due to inelastic deformation which, unlike elastic deformation, cannot be readily calculated. In addition, the proportion of closure due to inelastic rock deformation increases with increasing depth, and can be up to an order of magnitude larger than the elastic component.

The forces driving closure are large and, for the most part, cannot be resisted by conventional support. Support units nevertheless generate reactive loads in response to closure, thereby supporting the hangingwall blocks or beams and assisting in the maintenance of horizontal clamping stresses.

Differential lateral movement of the hangingwall and footwall manifests itself as tilting and ride of support units, and generally has a deleterious effect on their performance. The extent of this ride is affected by factors such as reef dip, stope geometry, varying hangingwall and footwall rock quality, and the movement of hangingwall and footwall strata into gullies, slots and caved areas.

4.2.1.3 Support Reaction.

Deformation, imposed on support units by the action of stope closure, develops reaction forces within them which act to support the hangingwall beams and blocks. Load-deformation characteristics of typical examples of the variety of support types used in stopes can be determined in laboratory tests, and by underground measurement. The force-

deformation behaviour of timber support underground has been found to be lower in comparison to its laboratory performance due to the effects of, primarily, loading rate sensitivity and, to a lesser extent, bearing surface roughness.

The important stages and characteristics of a typical force-displacement curve for a stope support unit are shown in Figure 31. Firstly, and most importantly, the initial stiffness determines the rapidity with which forces are generated in the unit during the early stages of compression. These must be high and must be generated quickly to maintain the integrity of the hangingwall between adjacent support units and between support units and the face. Obviously in this regard, the closer support is installed to the face, the better. Active support, such as an hydraulic prop, which applies a force immediately on installation, maximises this effect. Secondly, the yieldability, or useful deformation range of a support unit determines the amount of deformation or closure the unit can accommodate while still performing a useful support function, giving an indication of the distance support is still operational in the back area. Finally, load shedding or strain softening behaviour marks the onset of complete failure of some support units.

4.2.2 Stope Support Design.

4.2.2.1 Support Resistance.

A fundamental aspect of stope support design is the "height" of hangingwall being supported or requiring support. This may be translated as the "support resistance" requirement of the design. In essence support resistance is a measure of the force applied to the stope hangingwall by the support system. It is thus also a measure of the ability of the support system to resist downward, inelastic movement of the hangingwall strata and to prevent unravelling and to promote horizontal clamping stresses in the hangingwall. Falls of ground reduce horizontal stress in the hangingwall, which can then lead to further falls. Support resistance, expressed as force per unit area, is also a measure of a support systems ability to contain rockburst damage. However, under dynamic conditions the energy absorption ability of support units is a more important parameter, this being a function of both force and displacement.

Support resistance is defined as the product of the force exerted by individual support elements (in-situ load generated) and the number of elements per square meter (installed support density); and is expressed in terms of support force per square meter of hangingwall with units of kilo Newtons per square metre (kN/m^2).

The load generated by a support element is variable and is characterised by the unique load/deformation curve of the unit. The load generated by a support element at any particular stage in the mining cycle will thus be dependant upon the state of compression of the element and its in-situ performance characteristic. There is a great deal of variability associated with both these parameters. Factors such as the rate of elastic and time-dependant closure, the distance of the support element from the face, the quality of installation, the pre-stress applied, the stoping height at installation, and the sparsity of in-situ performance information, make the exact calculation of load generation difficult. For this reason, average laboratory derived characteristics, which have been de-rated to take into account the slower underground loading conditions and timber creep characteristics, are generally used in support resistance calculations.

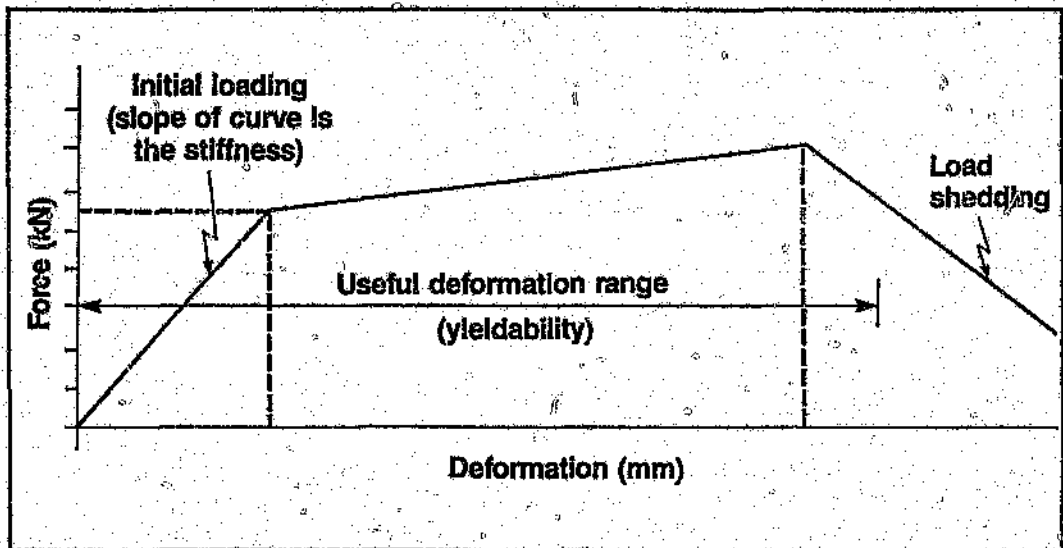


Figure 31 - Idealised Graph showing Salient Slope Support Force-Displacement Characteristics.

The following conventions enable the support resistance criteria to be used for comparing stope support standards:

- a) The area used in the calculation of support resistance is restricted to the whole area between the stope face and the no-entry barricade demarcating the commencement of the back area. The effect of back area support, if any, is ignored.
- b) Gully-edge support is also ignored in the calculation of panel support resistance. The area used in the calculation extends to the boundaries of the gully support, and excludes the supported areas adjacent to gullies.
- c) The support resistances generated by all the support types installed in any one panel (including temporary support) are summed to obtain the total support resistance experienced by the hangingwall. The support effect of the face itself is ignored.
- d) The support resistance on any particular panel fluctuates with the mining cycle. When the permanent (or rolling) support to face distance is the maximum allowable, the support resistance is usually a minimum. The calculation for this situation is conservatively used for comparative purposes.
- e) Despite the fact that support element reaction load changes with the degree of deformation and with time, a fixed value, representative of the elements estimated average performance, is allocated to each support element type for the purposes of calculation. This is a gross simplification of the calculation and can introduce fairly large errors because it ignores all of the variable factors which influence the performance of support elements, including stoping height and even failure of the unit.

In the case of packs and rapid-yielding hydraulic props, separate support reaction load values are used for the calculation of support resistance under slow (gravity, or fall-of-ground) and rapid (seismic, or rockburst) loading conditions respectively. For packs the values are derived from the derated and non-derated laboratory performance, and for hydraulic props they are derived from the slow yield and rapid yield valve settings. Other support types are deemed to have the same load characteristic under both slow and rapid loading conditions. Once again, these assumptions are a gross simplification of the practical situation. For example: sticks can fail completely when dynamically loaded, while matpacks are capable of resisting severe rockburst closure.

Some of the commonly accepted values of support element load carrying capacity used in the calculation of support resistance are listed in Table 1 overleaf. No cognisance is taken of the deformation needed to generate the loads.

Two scientifically derived support resistance figures are recommended by the Industry Guide. Firstly, a figure of 50kN/m^2 is adequate for the prevention of most falls of ground. This is equivalent to supporting the dead-weight of a 2m thickness of hangingwall strata. (This figure has been determined by collating data on many falls of ground, from which it was found that falls of ground with a thickness of greater than 2m are comparatively rare). Secondly, a support resistance of 200kN/m^2 is suggested for rockburst conditions. This is capable of arresting the dynamic displacement of a 3m layer of hangingwall rock ejected downwards at a velocity of 3m/s within a displacement distance of less than 0.3m. It is thus important to ensure that sufficient travel is available on all props in seismic risk areas.

TABLE 1.

Commonly Accepted Values Of Support Element Load Carrying Capacity
Used In The Calculation Of Support Resistance.

Timber Elongates:

Face Sprag	80-100mm diam.	50kN
Stick	100-120mm diam.	80kN
Mine Pole	> 150mm diam.	300kN
Yielding-type Elongate		300kN

Mechanical Props:

Light Duty Prop		1kN
Medium Duty Prop		100kN
Heavy Duty Prop		100kN

Hydraulic Props:

8 Ton		80kN
20 Ton		200kN
20/40 Ton	200kN (R/F)	400kN (R/B)
De-rated 40 Ton	300kN (R/F)	300kN (R/B)

Packs:

55 Solid	500kN (R/F)	1000kN (R/B)
75 Solid	750kN (R/F)	1500kN (R/B)
75 Hercules	1000kN (R/F)	2000kN (R/B)
110 Composite	1000kN (R/F)	2000kN (R/B)
110 Solid	1250kN (R/F)	2500kN (R/B)
110 Hercules	1500kN (R/F)	3000kN (R/B)

It is possible to justify a reduction of the rockfall and rockburst support resistance requirements to below 50kN/m^2 and 200kN/m^2 respectively based on local hangingwall beam thickness estimates of less than 2m and 3m, and peak ground velocity predictions of less than 3m/s. These support resistance values were, however, accepted as the minimum requirement for Hartbeestfontein design purposes because it was argued that they could be achieved without incurring significant cost penalties and would be capable of offering greater protection from falls of ground and rockburst damage than would lower values.

While the concept of support resistance, expressed in Newtons per square metre, provides a useful guide for static and dynamic support design purposes, it would be more accurate, and more meaningful, to quantify the performance of support units in terms of their ability to do work against potential (rockfall) and kinetic (rockburst) energy. An index of the energy absorption capabilities of a support system, expressed in Joules per square metre, would be more appropriate, particularly in the case of dynamic (rockburst) loading conditions. This would give an indication of the ability of the support system to arrest the dynamic displacement, or velocity, of a volume of wall-rock within a given displacement distance. Typically 60kJ/m^2 is required. As a further refinement, it may be useful to compare the performance of different support systems in terms of their power, expressed in Watts per square metre, to arrest the dynamic displacement of a volume of wall-rock being subjected to seismic impulse (rapid), or impact (shock) loading, within a given displacement distance and within a given (very short) time span.

4.2.2.2 Lateral Forces.

A further purpose of stope support is to maintain lateral restraint in the hangingwall. High horizontal stresses are generated in the hangingwall (and footwall) by dilation caused by the process of vertical fracture development in the rock ahead of the face. (Horizontal stress values of about 5MPa were measured in the immediate stope hangingwall during the course of this project). This forces the rock laterally into the stope, resulting in additional closure of the excavation due to movement along undulating bedding planes. It is also the primary cause of constriction and damage experienced in stope centre gullies and boxholes on Hartbeestfontein Mine. The higher the stress level in front of the working face and the weaker the rock, the more it will fracture, and the greater will be the horizontal stress generated.

In some circumstances, high horizontal stress can lead to downward buckling and even failure of the hangingwall at the stope face. Upward buckling of the stope footwall is also commonly observed under such conditions. Failure of the hangingwall rock takes the form of low angle fracturing, which aids the process of buckling and which can lead to hangingwall instability and falls of ground. To inhibit this, support must be kept very close to the face, and space (such as the caved area or a slot) must be provided for the rock to move into. Vertical clamping stresses which inhibit horizontal shear movement of the hangingwall and footwall must also be relaxed by reducing the support resistance generated by the support system. There is obviously a fine balance between reducing the support resistance to avoid low angle fracturing and buckling, and providing sufficient support resistance to maintain hangingwall stability.

4.2.2.3 Support Requirements.

The determination of the optimum stope support resistance requirement to cater for any particular situation is difficult, and dependant upon a large number of variables:-

a) **Rockmass Parameters:**

The degree to which the hangingwall is self-supporting.
 The volume or extent of hangingwall rock which requires support.
 The inherent strength, condition and integrity of the rock.
 The intensity and nature of strata lamination and bedding, and the cohesion associated with these features.

b) **Rock Damage Parameters:**

The magnitude, orientation and rate of change of the ambient stress regime.
 The degree and nature of fracturing; its distribution, frequency, orientation, and complexity.
 The extent and influence of blast damage.

c) **Discontinuities:**

The distribution and orientation of natural and mining induced discontinuities.
 The degree, nature and extent of blockiness.
 The influence of local geology.

d) **Seismic Risk:**

The potential for rockburst damage.
 The magnitude, location, proximity and mechanism of seismic events.
 The mechanism, extent and nature of seismic damage.

e) **Mining Parameters:**

The mining depth, span and stoping geometry.
 The rate of face advance.
 The stope width.
 The degree and rate of closure.
 The frequency and duration of standing periods.
 Whether approaching holing or remnant situation.

These parameters change from mining region to mining region, from shaft to shaft, from working place to working place, from panel to panel, and even from day to day. Heavy reliance must therefore be placed on a qualitative understanding of rock and support behaviour based on the experience of underground staff, and on their observation of changing conditions on a daily basis.

It is nevertheless incumbent upon management to specify minimum stope support standards to cater for the range of conditions encountered on the mine. It then becomes possible for underground production staff to choose the appropriate standard which meets the observed conditions in any particular area, and to monitor its performance.

While this approach is suitable for selecting an appropriate support resistance for rockfall conditions, special care must be exercised in seismically active areas. A support resistance which may be observed to be quite adequate to control the hangingwall under static loading conditions may be hopelessly inadequate during a rockburst. For this reason a support resistance of 200kN/m^2 must be enforced in areas deemed to be at seismic risk, regardless of whether this is in excess of the support resistance requirement for normal circumstances.

If sweeping generalisations are made, it is possible to identify separate regions on a mine which have broadly similar rock conditions and thus, broadly similar support requirements. On Hartebeestfontein Gold Mine, for example, some stoping areas adjacent to the Kromdraai Fault at No.4 Shaft are highly faulted, have massive, blocky hangingwall, and are at considerable seismic risk. Support must thus be active, stiff, capable of generating high reactive loads, and be rockburst resistance. On the other hand, the peninsular remnant area between No.5 Shaft and No.6 Shaft is highly stressed, experiences plastic rock flow, has moderate to intensely fractured hangingwall, and is subjected to a constant but fairly low level of seismic activity. Here, support must accommodate high closure rates, provide large areal coverage, and prevent unravelling of the hangingwall.

It is slightly more difficult specifying exact support resistance requirements for each mining situation. Once again, minimum requirements can be stipulated, but it is up to experienced and well trained underground production staff to "read" hangingwall conditions and apply appropriate measures to increase the support resistance where necessary.

An analysis of falls of ground at the stope face, carried out during 1990 by the Chamber of Mines Research Organisation^{25,36} on many South African gold mines, indicated that a support resistance of 50kN/m^2 in the stope face area is sufficient to prevent 95% of such falls, provided that the support load is applied evenly over the whole face area. Unless this support resistance is present in the stope face area as well as the permanent or rolling support area, falls of ground will still occur in unsupported areas; usually the face. But, if this level of support resistance is in place, particularly at the immediate stope face, the possibility of rockfalls will be greatly reduced, even if conditions change during mining. It is, however, vital that the support resistance is spread as evenly as possible over the hanging by employing support units that are capable of providing a high degree of areal coverage close to the face; such as props and headboards.

The support resistance on a panel may be manipulated and improved in a number of ways:-

- a) The face to support distance can be decreased.
- b) Permanent support can be installed or moved forward after each blast.
- c) The density of temporary and/or permanent face support can be increased.
- d) The spacing between support elements can be decreased.
- e) Stronger support elements can be introduced.
- f) Support units can be pre-stressed.
- g) Load-spreaders (head and/or foot) can be introduced to reduce punching and loss of load in elongate elements.

With the exception of reducing the face to support distance, all of the above options have cost, labour and production implications. It does, however, sometimes make good economic sense to incur additional support costs in order to maintain low stoping widths and to avoid costly fall-of-ground delays and accidents. Once again, a considered balance between support cost and support benefit is called for.

The most important parameter in stope support remains the "unsupported span", and, more particularly, the span between the face and permanent support. In the studies referred to above, it was found that the average distance at which all face-area fall-of-ground fatalities occurred in the Kleiaksdorp region was 1,6m from the face. The average distance of the permanent support from the face was 4,6m. A similar trend was shown in the West Rand and Orange Free State goldfields. The study clearly indicated the need to concentrate support improvement efforts on the area between face and permanent support where most activity takes place, and where people are most at risk.

Reducing the face to permanent support distance, installing permanent or rolling support after each blast, and, to a lesser extent, increasing the amount of temporary support, calls for discipline, commitment and enthusiasm, and in the extreme, a revision of mining methodology to accommodate intensified face support measures. Hartebeestfontein Mine has, over the past number of years, had some considerable success in its efforts to reduce the face to permanent support distance, which currently stands at a maximum of 3,5m. Major positive advances have also been made in the area of temporary support requirements. No person is now allowed to be more than one or two metres, depending on circumstances, from support at any time. Figures 32 and 33 show the reduction in fall-of-ground accidents and in average stoping width achieved by the mine in the recent past by the application of these measures.

Although installing or moving support after each blast is time consuming and labour intensive, it remains the only way to keep the unsupported face span to an absolute minimum. Ideally, support should be installed a fixed distance from the face after each blast, and the strike spacing of support units should thus be determined by the actual advance obtained per blast.

The concept of support resistance was originally conceived to pertain only to the area where it mattered most; the working area immediately adjacent to the advancing face. Applying this strict approach, effective support resistance in the area where people are actually working, ie. in the area between the face and permanent support, will, in most cases, be extremely low. It is, however, equally true that the higher the support resistance in this critical area, the lower will be the overall support resistance requirement. In other words, the closer support is to the face, the less support resistance is needed; a philosophy borne out to some extent by stick and prop cave mining practice, and by the success achieved during experimental work with 80kN blast-on props at Hartebeestfontein No.6 Shaft, which was carried out and reported upon by the Chamber of Mines Research Organisation⁽⁴³⁾, and which will be elaborated upon later.

The further development of the blast-on prop / diagonal barricade / water jetting system, in joint collaboration between the Chamber of Mines Research Organisation and Hartebeestfontein Gold Mine, was specifically aimed at dramatically reducing face to support distances. Because the system was ideally suited to overcoming the weak ground conditions being experienced in the 5A North pseudo-breccia area, experimental sites were established there. The success of these experiments led to the later wide-spread introduction of 200/400kN blast-on props on the mine.

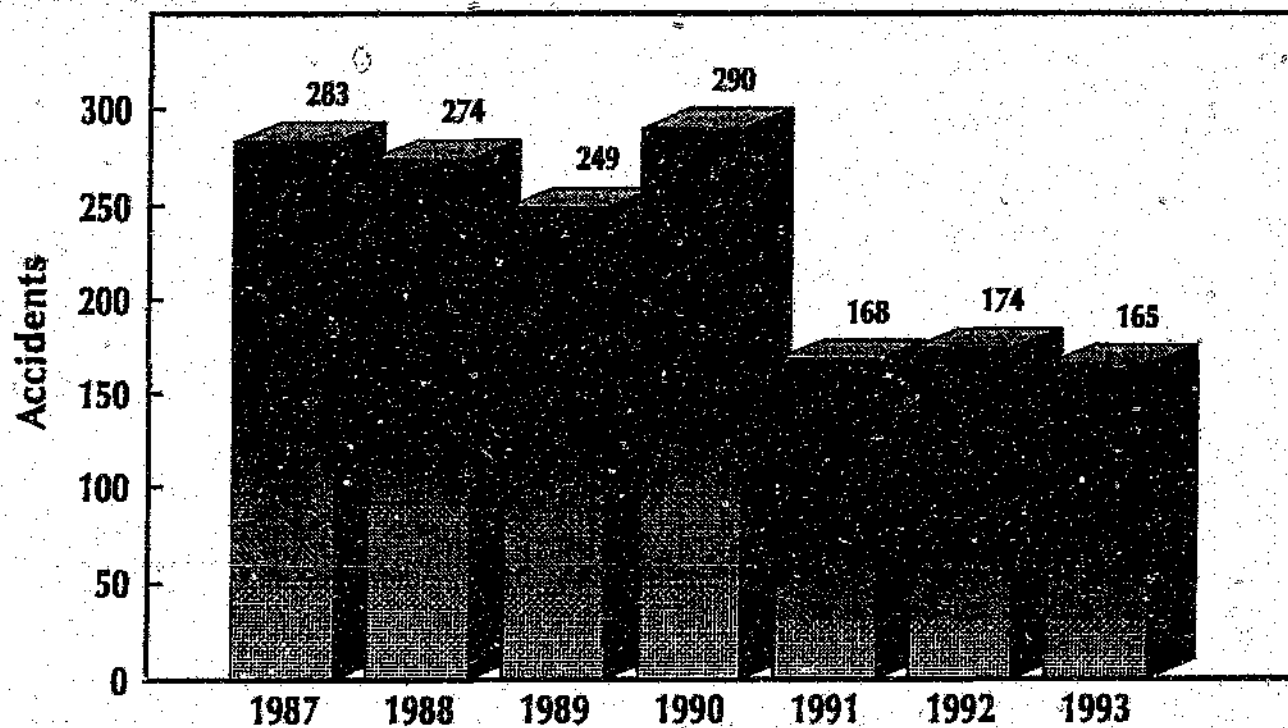


Figure 32 - Fall of Ground Accidents on Hartbeestfontein Gold Mine for the period 1987-1993.

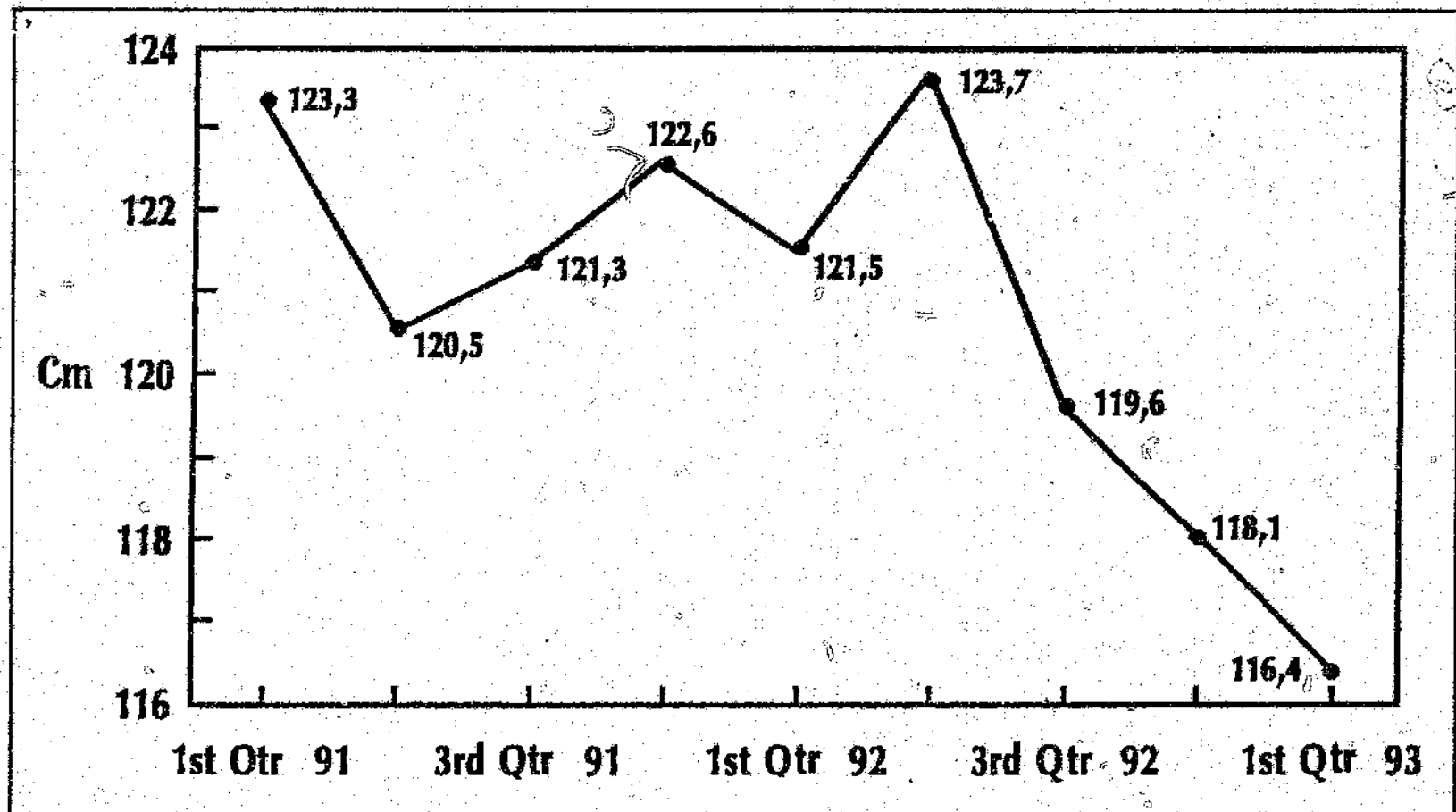


Figure 33 - Average Stoping Width on Hartebeestfontein Gold Mine for the period January, 1991 - March, 1993.

This project report is based upon the work done in developing these stainless steel props, and in adapting the blast-on technology to cave mining conditions. These aspects will thus be discussed in more detail later.

The effect of time-dependant rock condition deterioration is sometimes not fully appreciated. No mention is normally made of the phenomenon in mine standards, other than to specify limit support in the case of stopped panels. Time dependant "unravelling" of the hangingwall on panels which are standing or advancing only slowly can, and does, result in serious strata control problems which can only be solved by increasing support resistance in the affected areas.

The standard support layouts currently employed on Hartebeestfontein Gold Mine are illustrated in Figures 34A to 34G. The support resistances associated with these layouts are presented in Table 2 overleaf. The maximum figure refers to the support resistance when the support to face distance is the maximum allowed by the applicable standard, normal 3,5m. The minimum is calculated for the situation when the face line of support is 2m from the face.

Bearing in mind that face to permanent support distance plays an important role in determining the effectiveness of support measures, minimum face support resistance requirements have been suggested as a rough guideline for the various hangingwall conditions and mining environments encountered on Hartebeestfontein Gold Mine. These are presented in Table 3 overleaf.

TABLE 2.

Calculated Support Resistance Associated With Standard Support Layouts
Currently Employed On Hartebeestfontein Gold Mine.

SUPPORT STANDARD	SUPPORT RESISTANCE (kN/m ²)			
	MINIMUM		MAXIMUM	
	R/F	R/B	R/F	R/B
Lagging and Pack Mining	192	224	238	278
Stick and Pack Mining	158	196	205	255
Prop and Pack Mining	119	158	155	205
Stick Cave Mining	144	144	183	183
Prop Cave Mining	142	142	185	185
Blast-On Prop Caving without Waterjetting	96	188	125	245
Blast-On Prop Caving with Waterjetting	97	194	130	267

TABLE 3.

Guideline Of Minimum Face Support Resistance Requirements For Various Hangingwall
Conditions And Mining Environments Encountered On Hartebeestfontein Gold Mine.

HANGINGWALL CONDITION	MINIMUM SUPPORT RESISTANCE (kN/m ²)			
	MAX. FACE-SUPP. DIST.	<2m	2m-3m	> 3m
Massive, Blocky, Non-Caving.		100	150	200
Moderately Fractured, Low Closure.		50	100	200
Highly Fractured, Rapid Closure.		100	150	200
Seismically Active, Rockburst Risk.		200	200	200

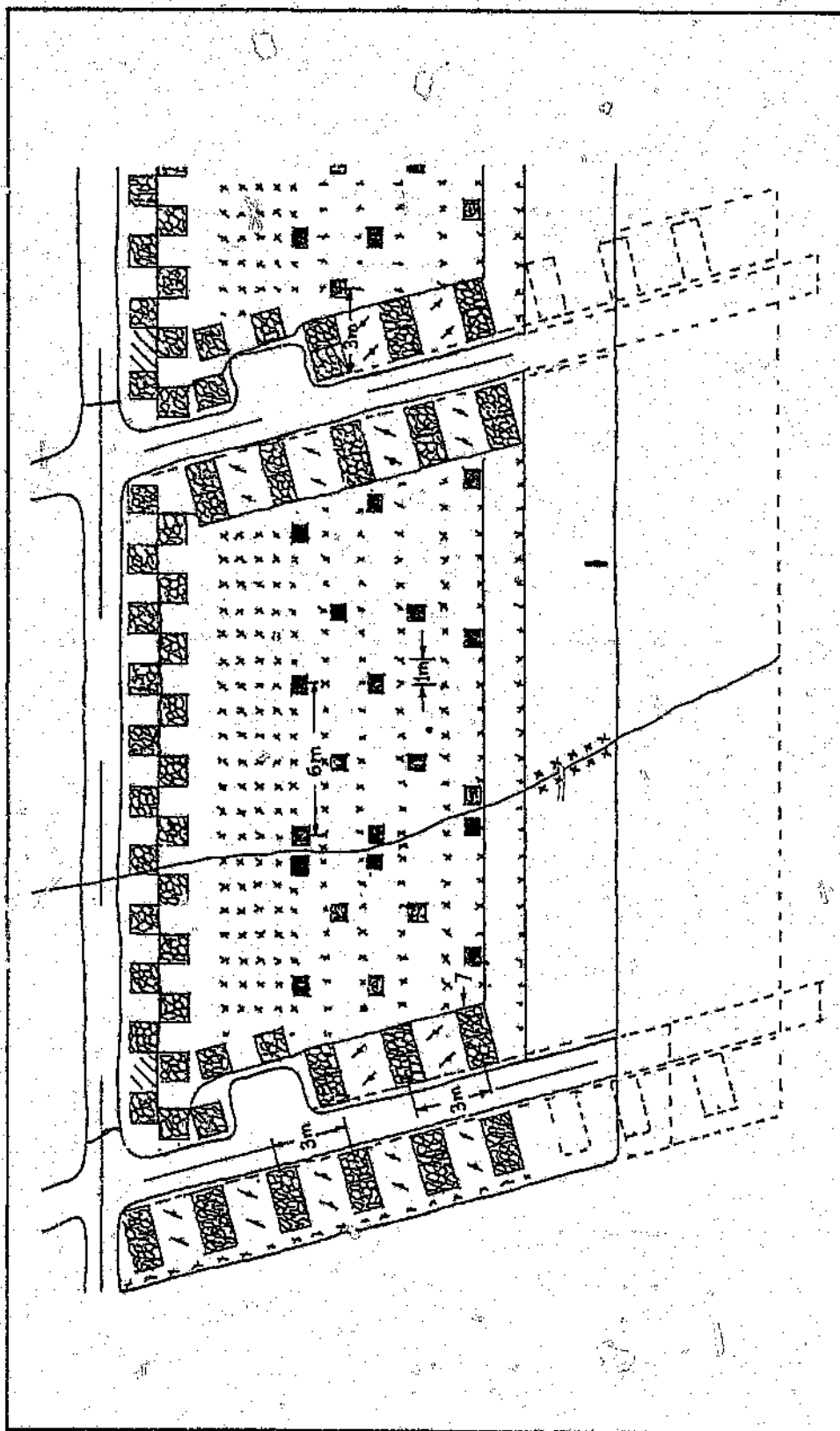


Figure 34A - Stope Support Standard for Lagging and Pack Mining.

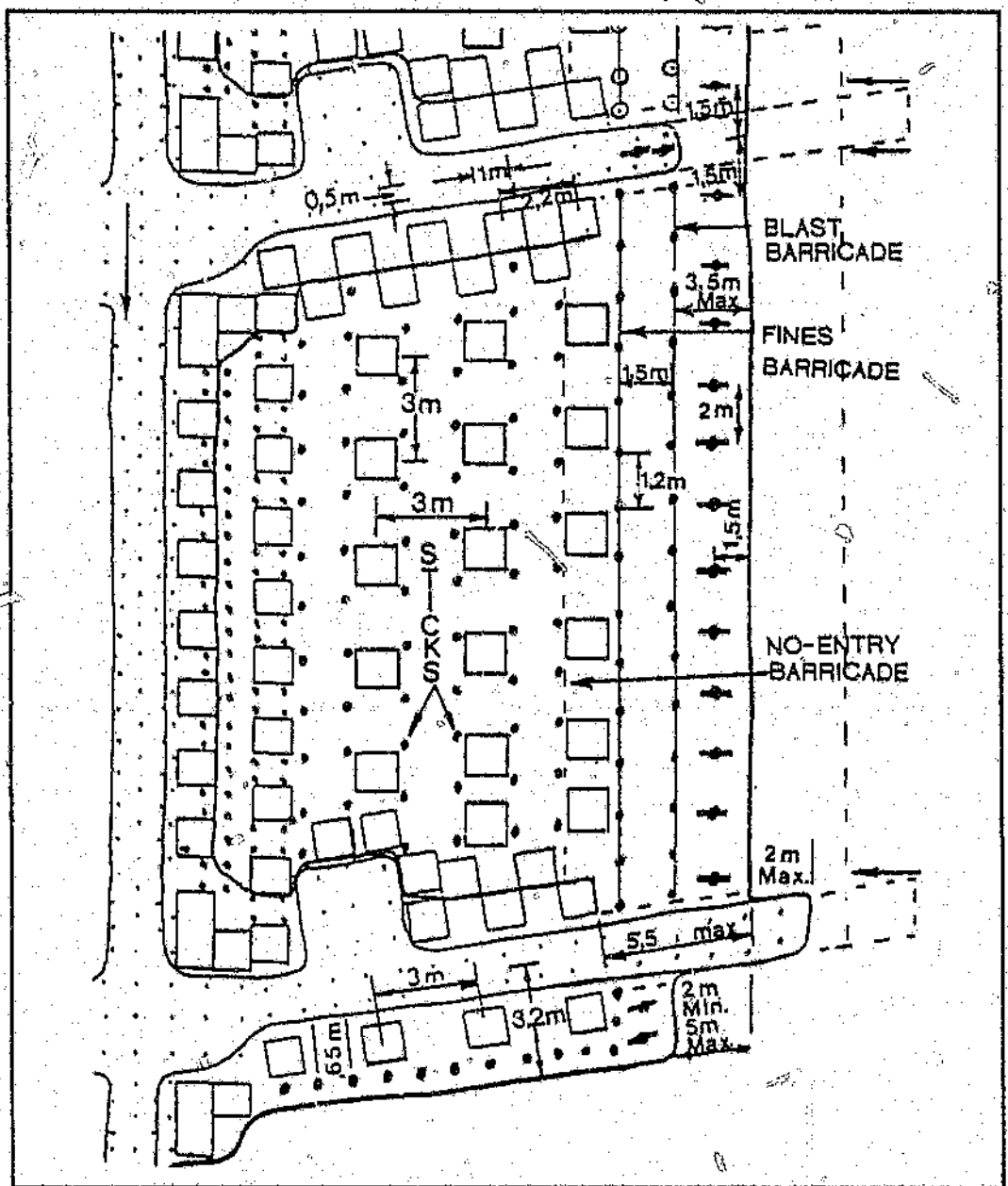


Figure 34B - Stope Support Standard for Stick and Pack Mining.

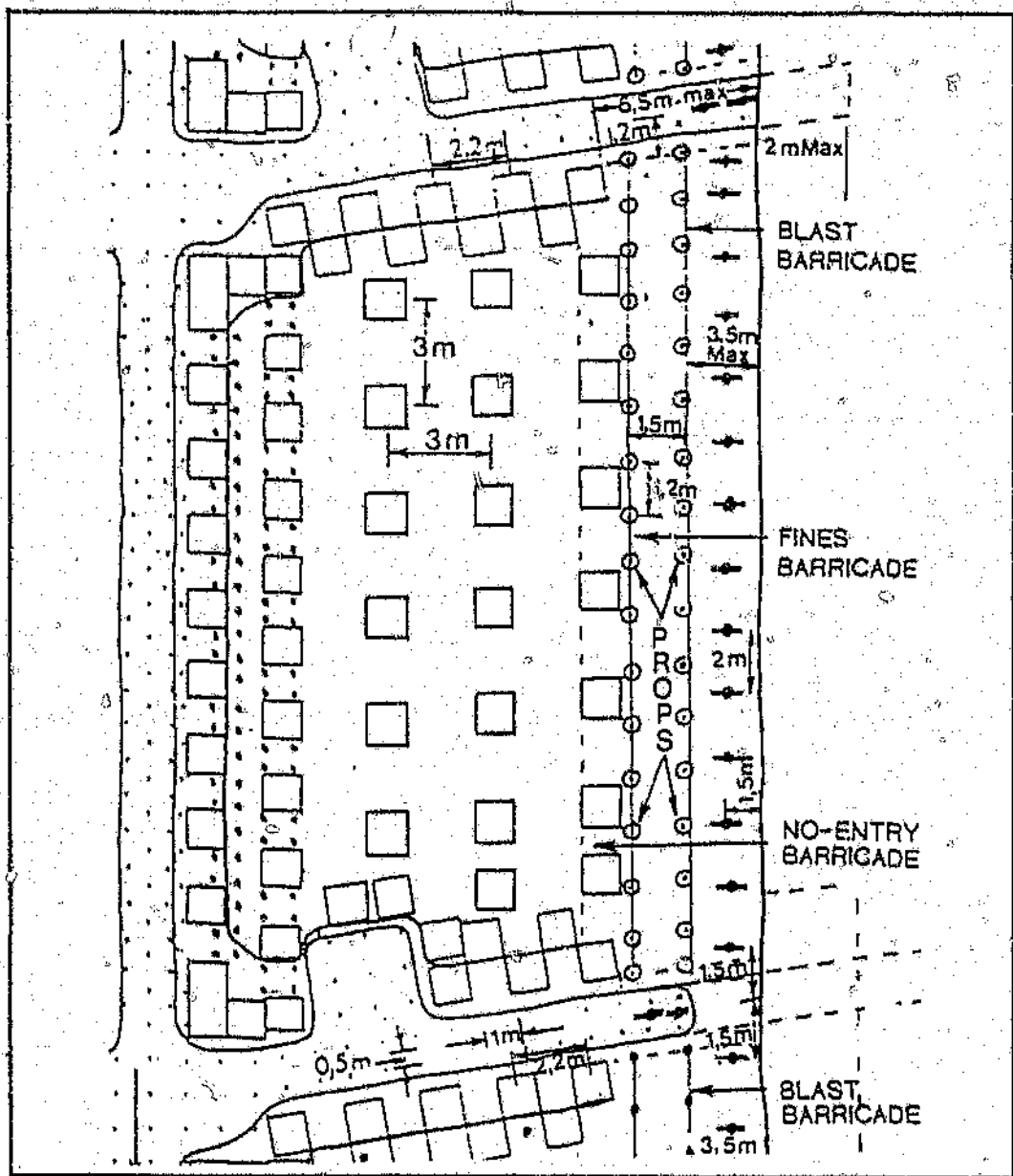


Figure 34C - Stope Support Standard for Prop and Pack Mining.

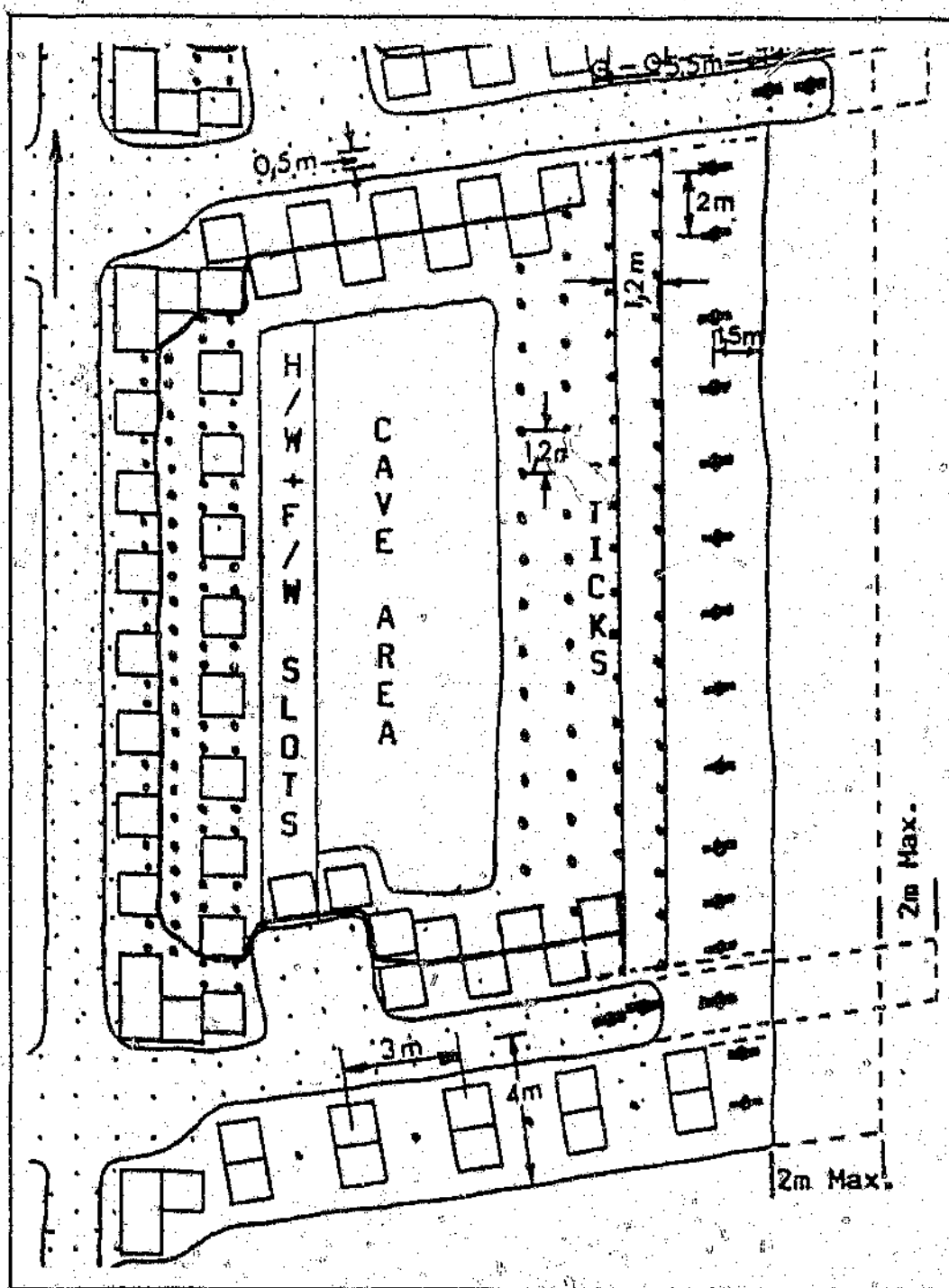


Figure 34D - Stope Support Standard for Stick Cave Mining.

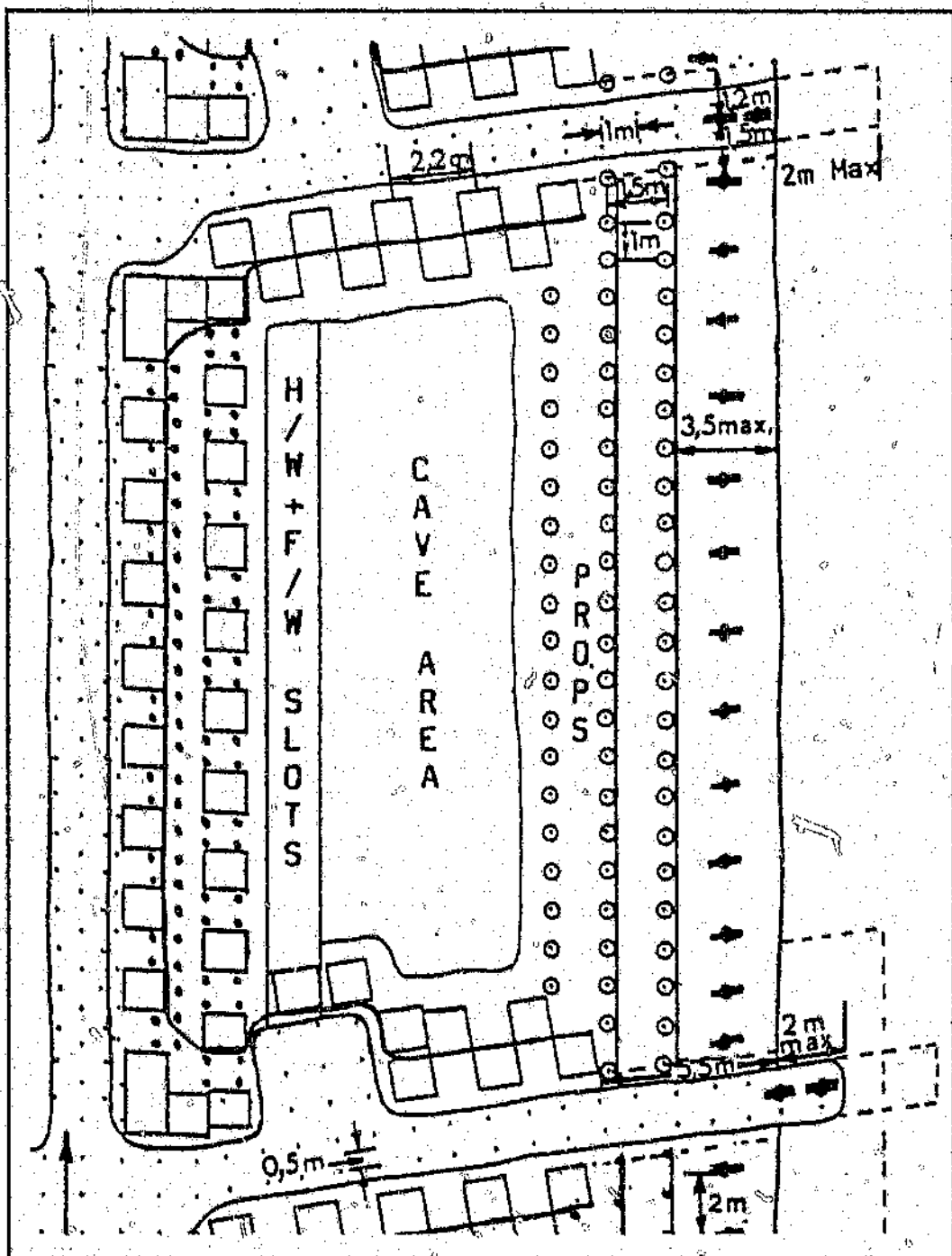


Figure 34E - Stope Support Standard for Prop Cave Mining.

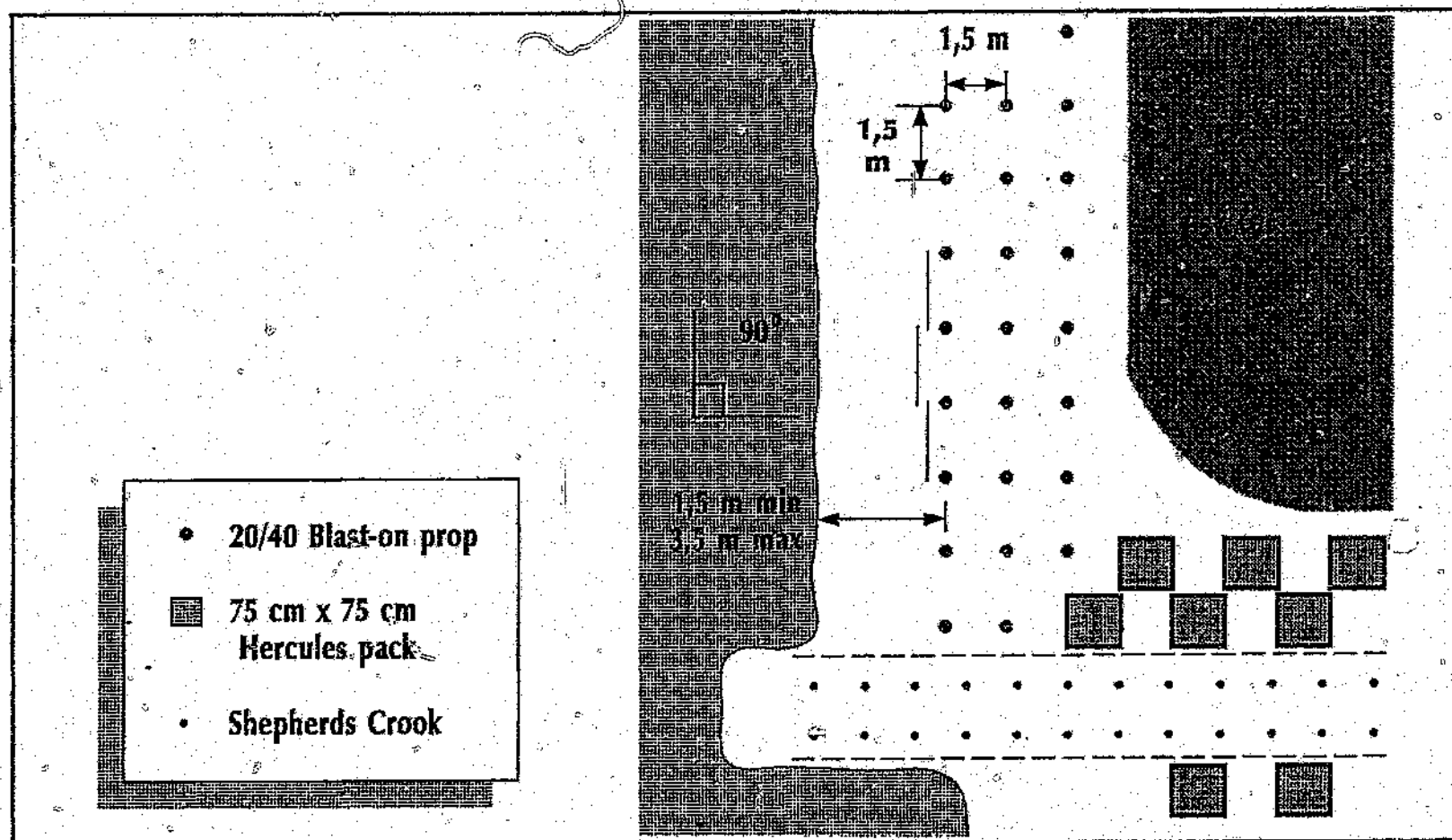


Figure 34F - Support Layout for Blast-on Prop Cave Mining.

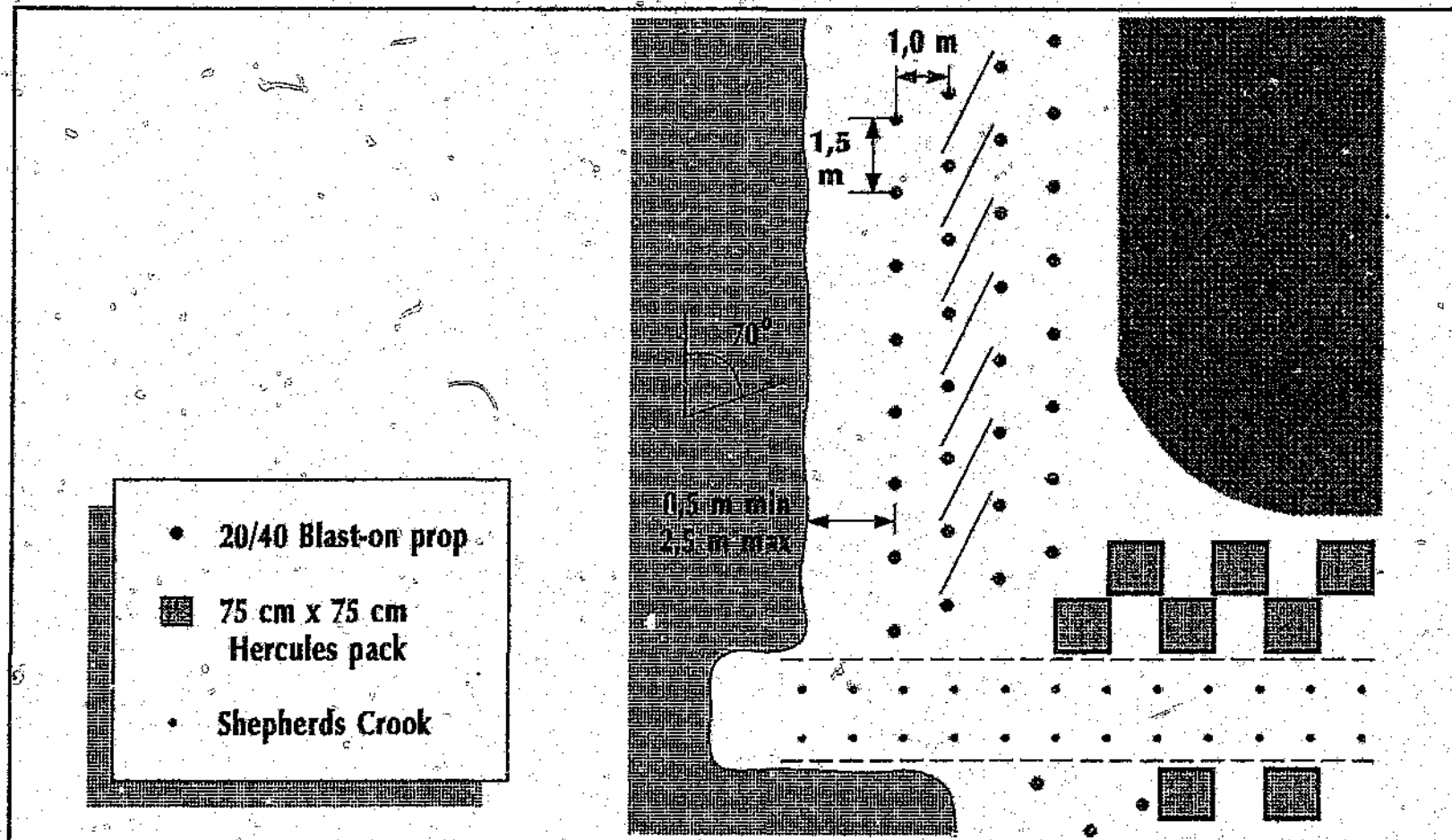


Figure 34G -

Support Layout for Blast-On Prop Cave Mining
with Waterjetting.

5. ROCKFALL CONTROL STRATEGIES.

5.1 BLAST-ON PROP SYSTEM.

Having dealt with the historical development of stope support strategies for cave mining applications on Hartebeestfontein Gold Mine in the preceding sections, this section will address the implementation of more recent rockfall control measures, in particular those aimed at maintaining the stability of the immediate stope face area in weak hangingwall conditions such as are experienced in the pseudo-breccia zone at No.5^A Shaft. Central to the implementation of these measures was the development of a light-weight blast-on hydraulic prop stope support system by the Chamber of Mines Research Organisation, in collaboration with Hartebeestfontein Gold Mine⁽⁴⁹⁾.

An investigation⁽⁴⁹⁾ by that organisation of the various factors relating to the occurrence of rockfall accidents in gold mines of the Witwatersrand basin indicated that a fairly moderate support resistance of 50kN/m² was sufficient to prevent more than 90% of fall of ground accidents in the stope face area, provided that this was combined with a high degree of areal coverage. This observation led to the development of a number of support options comprising mechanical and/or hydraulic props of various types equipped with suitable load-spreaders. As part of the initial underground experimental work conducted in a test stope at Hartebeestfontein No.6 Shaft during 1990, 80kN blast-on hydraulic props were installed approximately 1m from the stope face, in combination with diagonal barricades and waterjet cleaning⁽³⁷⁾. Considerable development and modification to the prop units was then undertaken in order to ensure that they were mineworthy. The ultimate success of this experiment in turn led to the further development and adaptation of the system to suit both weak ground and rockburst conditions encountered in deep gold mines.

Towards the latter half of 1990, the Chamber of Mines Research Organisation drew up detailed design and performance specifications⁽⁴⁹⁾ for a significantly lighter rapid yielding hydraulic prop than had hitherto been in service. Prop manufacturers were invited to design and manufacture props which would be capable of coping with ground velocities associated with large ($M_L > 3$) rockbursts, during which stope closure rates sometimes reach three metres per second or more, while at the same time providing suitable support for normal strata control requirements. The main departure points of the new specification in comparison with the old specification for 400kN rapid yielding hydraulic props are as follows:-

a) Normal Operating Requirements.

An important feature of the new specification is the recognition that a support resistance of 50kN/m² is sufficient to provide adequate control of rockfalls in the stope face area. Consequently, during most of their working life props need only be subjected to the relatively modest operating loads required to generate a maximum slow yield force of 200kN. Thus, new generation props operate at less than half of their rockburst design capability during most, if not all, of their working life, thereby reducing wear and tear on prop units and minimising damage caused to hangingwall by high prop forces.

b) Transient Rockburst Loading.

A detrimental feature of old hydraulic prop types is that excessive transient

forces, which can be as high as 700kN, occur during rapid yield. This requires more robust prop and load-spreader design, and can cause hangingwall damage at the onset of a rockburst. The new rapid yield specification, which remains at 400kN, is based on a rockburst model in which a 3m thickness of rockwall is accelerated to a velocity of $\leq 3\text{m/s}$. The support system is required to arrest the motion of this rock within 200mm of movement without generating destructive transient forces, ie. without developing transient forces exceeding 500kN. As each prop is designed to be effective over an area of $1,5\text{m}^2$ of hangingwall, this equates to an energy absorbing capability of $\pm 60\text{kJ/m}^2$.

c) **Handling and Ease of Operation.**

Because normal operating forces and transient peak forces have been limited to 200kN and 500kN respectively, props and accessories are designed to less stringent structural strength requirements, while at the same time providing a factor of safety in excess of two for normal operating conditions. This results in the production of much lighter props and accessories, with a commensurate reduction in the amount of effort required to operate, manhandle and install the support system. In the event of a rockburst the equipment may be stressed to near its limit, but this is acceptable for the limited number of occasions that it is subjected to severe events.

d) **Unsupported Span.**

Perhaps the most important features incorporated into the design of the new support system are those which provide the potential to reduce unsupported spans at the stope face. Blast protection sleeves allow props to be installed to within less than a metre of the face with little adverse effect. Specially designed three-point load-spreaders reinforce the ability of props to resist being blasted out, while at the same time provide for a high degree of areal coverage and reduce inter-support and support to face spans. The additional incorporation of waterjet cleaning allows the support system to be utilised to its full potential.

Three prop supply companies responded to the invitation and involved themselves in the design, manufacture and development of a range of 200/400kN rapid-yielding, lightweight, blast-on hydraulic props and accessories. Underground tests were conducted on the three product ranges under conventional and cave mining conditions at the Hartebeestfontein No.5 Shaft. This included extensive trials in a number of stopes situated within the pseudo-breccia zone, and which formed part of the work described in this project report.

The new 200/400kN props and accessories conform to Chamber of Mines Research Organisation design specifications which are listed in Table 4 overleaf. The force specifications are graphically illustrated in Figure 35 which shows the cycle of setting force, force decay and recovery, slow yield, rapid yield, and return to slow yield experienced by props at the stope face. Other quantitative and qualitative specifications laid down by the Chamber of Mines Research Organisation cover aspects such as maximum ram travel, minimum piston restraint, eccentric loading stability, remote release performance, impact resistance, working fluid type, corrosion resistance, valve protection, system integration, ergonomics, material flammability and toxicity, and acceptance testing.

TABLE 4.

**Chamber of Mines Research Organisation Design Specifications
For The New 200/400kN Hydraulic Props And Accessories,**

Setting Force:	120kN - 160kN
Slow Yield Force:	170kN - 200kN
Rapid Yield Force:	400kN - 500kN
Maximum Dynamic Transient:	500kN
Residual Force:	80kN - 200kN
Slow Yield Rate:	up to 0,5m/s
Rapid Yield Rate:	0,5m/s - 3m/s
Minimum Resilience:	0,05mm/Kn
Minimum Yield Travel:	300mm
Maximum Prop Mass:	30kg/550mm
Minimum Load Spreader Span:	800mm
Acceptable Load Spreader Mass:	±15kg

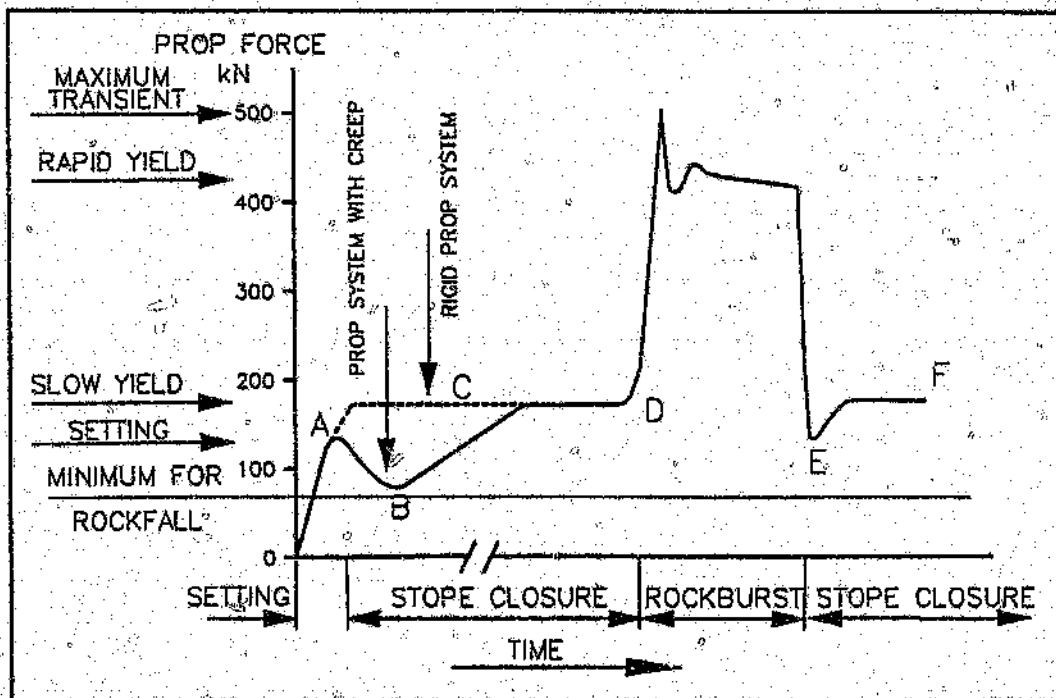


Figure 35 - Graphical Illustration of Prop Force Cycle.

5.2 IMPLEMENTATION ON HARTEBEESTFONTEIN.

5.2.1 Initial Underground Trials.

Underground testing of the first 200/400kN props commenced early in 1992 in the S4 and S5 Panels of 77-42W Stope at No.5 Shaft. The location of this, and other experimental stopes, all situated within the pseudo-breccia zone, is shown in Figure 36. Two prop manufacturers were involved in the development of 200/400kN props at this stage; Elbroc Strata Control and Klockner-Becorit Corporation. The in-situ performance of the new props was closely monitored during these initial tests. As a result, minor modifications were made to some valve and seal components to suit local conditions and improve overall performance of the props, but other than this, the props and associated equipment conformed to, or exceeded, desired performance requirements.

During this period a very significant head design change was introduced by Elbroc. In place of the conventional square head, props were manufactured with a tapered or conical head. The taper was matched to extensions of various lengths which were similarly equipped with conical male and female end pieces designed to fit snugly one into the other, allowing little freedom for rotational or bending movement. This led to a vast improvement in the stability of prop and extension combinations, and has enabled the use of very short (550mm closed length) props to be used in stoping widths up to 2m without compromising support effectiveness. Plate 3 illustrates the stability of the combination of short (550mm) conical head props fitted with a number of 150mm and 300mm conical extensions accommodated in a stoping width exceeding 2m.

This innovation has naturally had a major impact on mine prop population requirements as well as on prop utilisation statistics. The multiplicity of prop lengths required to accommodate the range of stoping widths encountered underground is now much reduced, as is the number of mis-matched idle props lying around underground. In addition, major advantages accrue when undercutting, which is a fundamental operational requirement of reducing stoping width, and which in turn was one of the primary objectives of introducing the 200/400kN prop system. Their natural stability and flexibility allows the same short length of conical-head prop to be used during the entire undercutting process, from the high stoping width stage through to the attainment and maintenance of low stoping widths. As further testimony to the stability of the system, Y-piece extensions were developed and used to provide lateral support to undercut brows during undercutting. Plate 4 illustrates the use of a Y-piece to provide footing for a diagonally mounted prop imposing lateral restraining force on an undercut brow. The lateral or horizontal force imposed by the diagonal prop on the vertical prop/extension combination is in excess of 80kN. This side-loading is borne without adverse effect, even during direct blasting of the undercut face.

The only major criterion now determining the length of prop required for any particular stope is that of closure, or more correctly, the amount of ram travel required to accommodate the anticipated rate of closure. This in fact represents a limitation in that stopes which experience rates of closure greater than can be accommodated by the shorter length props (300mm travel in the case of 550mm props) must perforce be carried at stoping widths capable of accommodating longer prop lengths if freezing of props in the back row (or rows) is to be avoided. In this regard, the importance of minimising the amount of closure which must be accommodated, by maintaining rapid rates of face advance and by carrying the prop support zone as close to the face as possible, is emphasised.

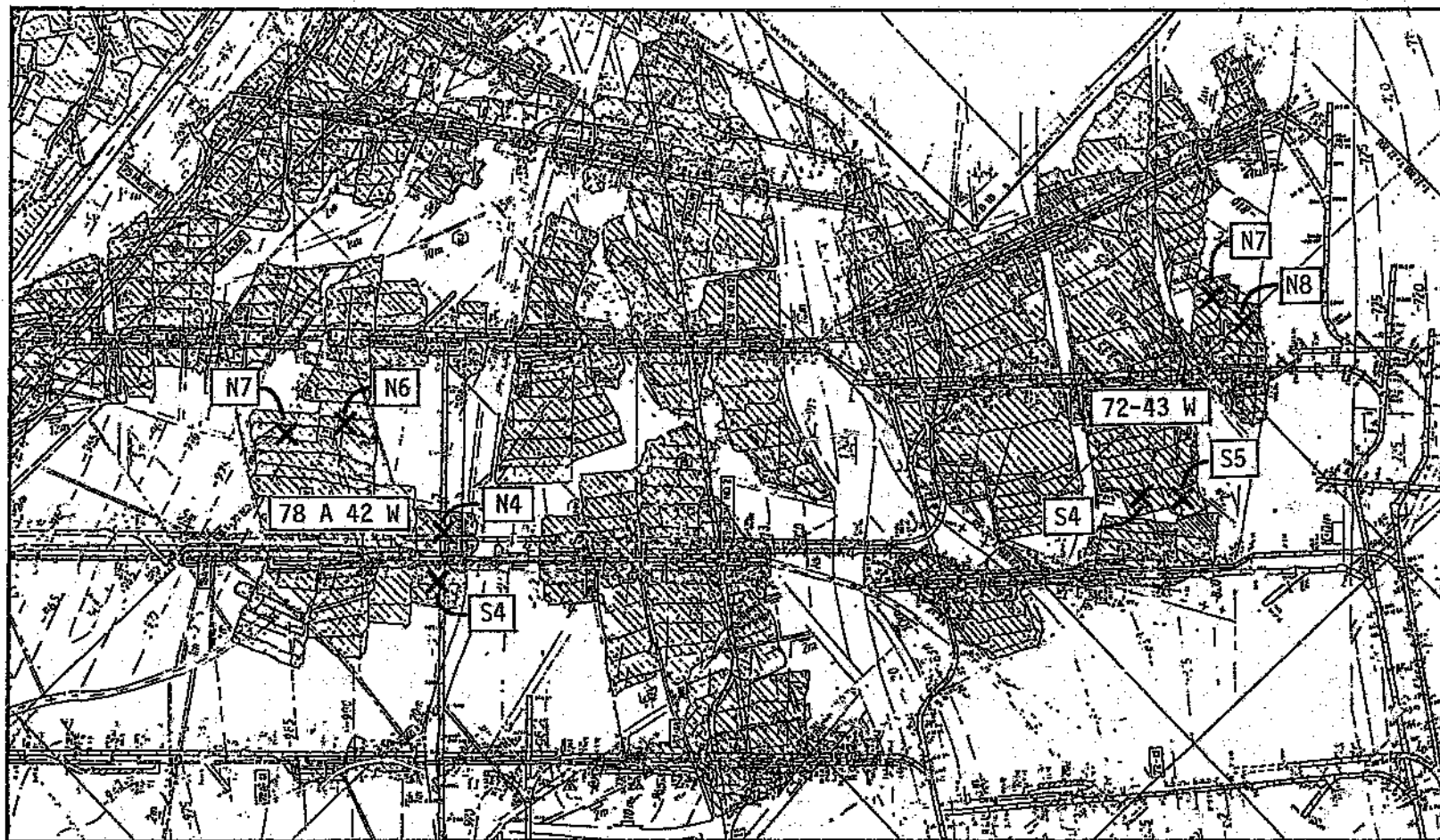


Figure 36 - Plan showing the Location of Experimental Sites in the Pseudo-Breccia Zone at No.5 Shaft.



Plate 3 - Stability Of Conical Head Props And Conical Extensions.



Plate 4 - Use Of A Y-Piece On An Undercut Brow.

Early success in reducing stopping widths in the experimental panels was found to be restricted by the length of the props employed. Original stopping widths of between 1150mm and 1200mm were quickly reduced to 1050mm following the introduction of 650mm closed length blast-on props. Further reductions below about 1050mm were not possible or practical because allowance had to be made for a minimum of 300mm ram travel and 50mm for the load-spreader. Stopping widths thus plateaued at this 1000mm-1100mm level. However, when the original 650mm props were replaced by 550mm length props a further reduction of average stopping width to less than 950mm quickly followed. The graph shown in Figure 37 illustrates the stopping width reduction achieved following the introduction of 650mm length blast-on props on the initial test panels and the subsequent step-down following conversion from 650mm props to 550mm props.

The conical head produced a further unexpected spin-off benefit over its square head predecessor. It was recognised fairly early on that much of the success achieved with blast-on props under supervised experimental conditions was due to the disciplined use of effective 800mm long load-spreaders, and that much of this benefit would be lost if the use of load-spreaders was not carried through to the full production phase. However, because of the much reduced bearing area at the top of the taper, conical-head props cannot be jacked directly against friable or weak hangingwall without punching into the rock and causing some damage. The use of load-spreaders is therefore absolutely essential to avoid this concomitant, and thus the conical head almost guarantees that load-spreaders will always be an integral part of the 200/400kN prop system.

The design of an innovative load-spreader, or so-called tri-spreader, was undertaken by the Chamber of Mines Research Organisation some time ago to improve upon and replace the older generation headboards which suffered the disadvantage of not supplying sufficient areal coverage and, on occasion, of applying point loads to the hangingwall. Many attempts have been made, and continue to be made, to try and spread the prop load more evenly across the length of these types of headboard. The most successful of these include the introduction of deformable rubber or plastic inserts, sometimes encasing a steel spring, designed to mould the headboard to the contours of the hangingwall. The major disadvantage of these designs is that of weight. The Chamber of Mines Research Organisation design on the other hand incorporates a methodology in which the force generated by the prop is not directly transferred to the hangingwall immediately above the prop head, but is applied at the centre of an essentially elongated triangular load-spreader, which in turn transfers and applies the load to the hangingwall along its base and at its apex. This off-centre arrangement has the disadvantage of not allowing the load-spreader to balance particularly well on top of the prop during installation. The difficulty has, however, been largely overcome by providing load-spreaders with a permanent plastic insert which moulds to the shape of the conical head of the prop and greatly improves the stability of the prop/extension/loadspreader combination during the installation process. This, however, remains a two-man operation, one person is required to hold the prop upright, the other to balance the load-spreader. Resistance by production staff to this "labour intensive" operation stems from the previous use of props without headboards, or with relatively short, "balanced" headboards, which allow props to be installed by only one operator, but which are far less effective than props installed with the new 800mm tri-spreaders.

In addition to scrutinising the performance of the props and accessories themselves, other aspects of project which were closely monitored included: i) the effectiveness of waterjet cleaning, ii) the overall efficiency of stopping operations, iii) the improvement of hangingwall

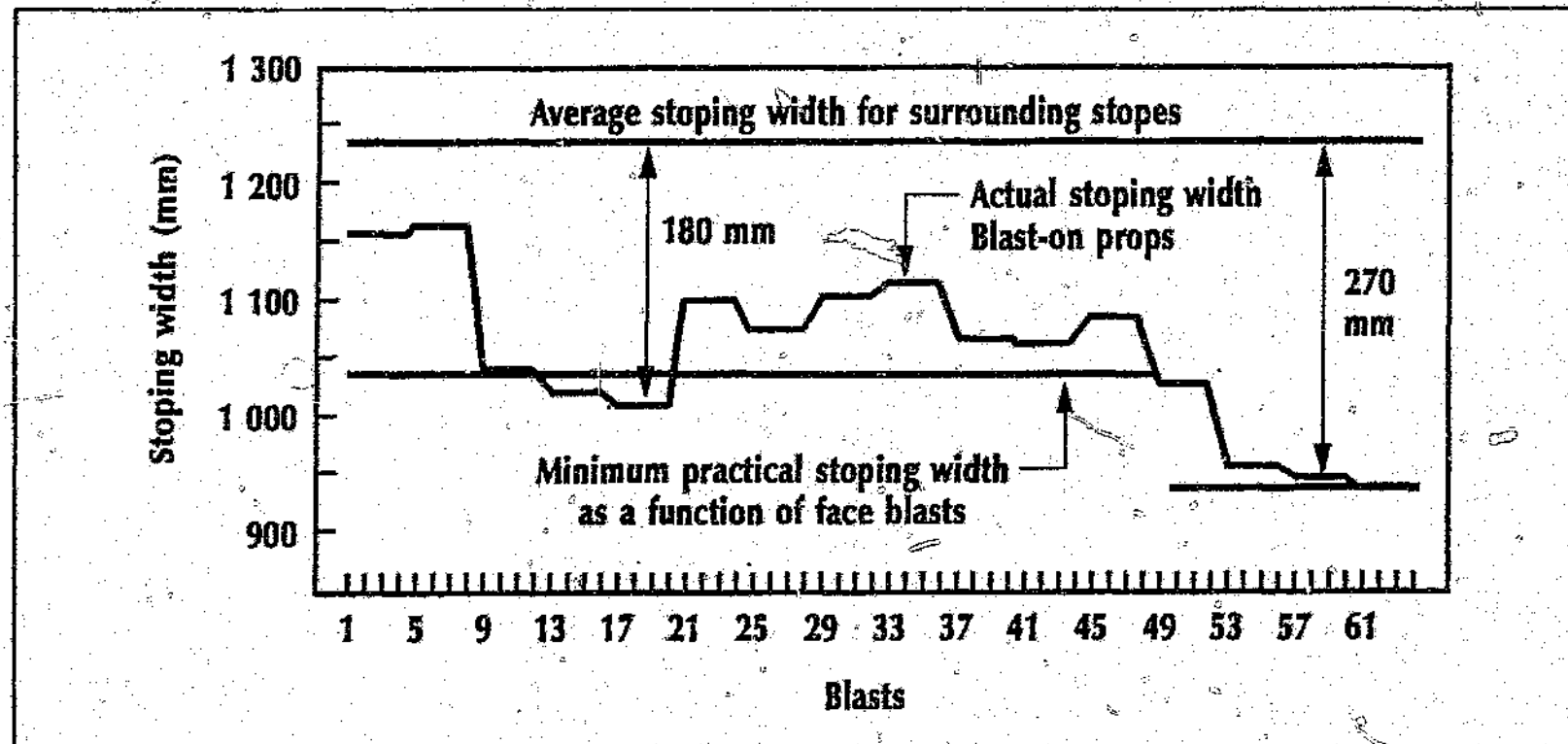


Figure 37 - Graph illustrating Stopping Width Reduction Achieved following the introduction of Blast-On Props.

stability, iv) the increase in the rate of face advance and, most importantly, v) the reduction of stoping widths. Significant improvements were recorded in all these areas during the initial trials. As an illustration, Figure 38 demonstrates the effectiveness of the 200/400kN prop system in reducing stoping widths. The figure shows stoping width contours measured in 72-43W S5 Panel during the course of the trial. A reduction from stoping widths in excess of 140cm to predominantly less than 120cm is clearly demonstrated. Of particular significance is the improvement recorded in the area immediately adjacent to the scraper gully, where stoping width is notoriously difficult to control, and which is generally considered to be an area at above average risk to falls of ground.

Following the initial success achieved in 72-43W S4 and S5 Panels with the 200/400kN blast-on prop and waterjetting system, the application was extended to the N7 and N8 panels in the same connection, and to the N6 and N7 Panels in 78A-42W Stope. Instrumentation was installed in these latter panels to quantify and record rock deformation, rock stress and related parameters. The results of this instrumentation program are presented later in this report.

At about this time 200/400kN conical head props were received for underground testing from the third and final manufacturer; Unique Engineering. The three prop types (Elbroc, Klockner-Becorit and Unique) were installed for comparison purposes in three adjacent panels (N12, N13 & N14) in the 29-N10 Inter Stope, and were closely monitored. Differences between these prop types relate to design detail and appearance rather than specification and performance, although there are significant differences in the bore of the props which give rise to a differences in the setting loads achieved by the various props at constant setting pump pressure. This has the effect of substantially reducing setting pump pressure tolerances if damage to yield valves is to be avoided when props are mixed in the same stope. In retrospect, a standard internal dimension should have been included in the prop specification. This aspect is elaborated upon in Appendix 1.

One other notable difference between the prop types relates to the location, arrangement and design of the setting, slow-yield and rapid-yield valves. This varies from a single externally mounted "omni-valve" which caters for all these functions in the case of Elbroc, to three separate valves mounted externally in the handle and on the head, and internally on the ram in the case of Unique. The KBC prop has an externally mounted setting and slow-yield combination valve and an internally mounted rapid-yield valve. (In the case of Elbroc and Klockner-Becorit the external valve casing doubles as a handle for the prop). The rationale behind these differing designs encompasses many aspects relating to manufacturing, performance and maintenance efficiencies and costs, a discussion of which falls outside the scope of this document.

5.2.2 Production Applications.

It has already been noted that prior to the development of the light-weight, blast-on prop and waterjetting system, Hartebeestfontein Gold Mine had achieved some considerable success with a policy of reducing support-to-face distances at the stope face to an allowable maximum of 3.5m and of not allowing persons to operate beyond a distance of more than 1m from support except in the case of face cleaning operations, in which case this "golden rule" is relaxed to a distance not exceeding 2m. Figure 32 shows the fall of ground accidents for the total mine averaging 274 per year for the period 1987 to 1990 prior to the implementation of these measures. This is followed by a dramatic and sustained 38% reduction to an average of 169 per year immediately following their introduction at the end of 1990. The 1993

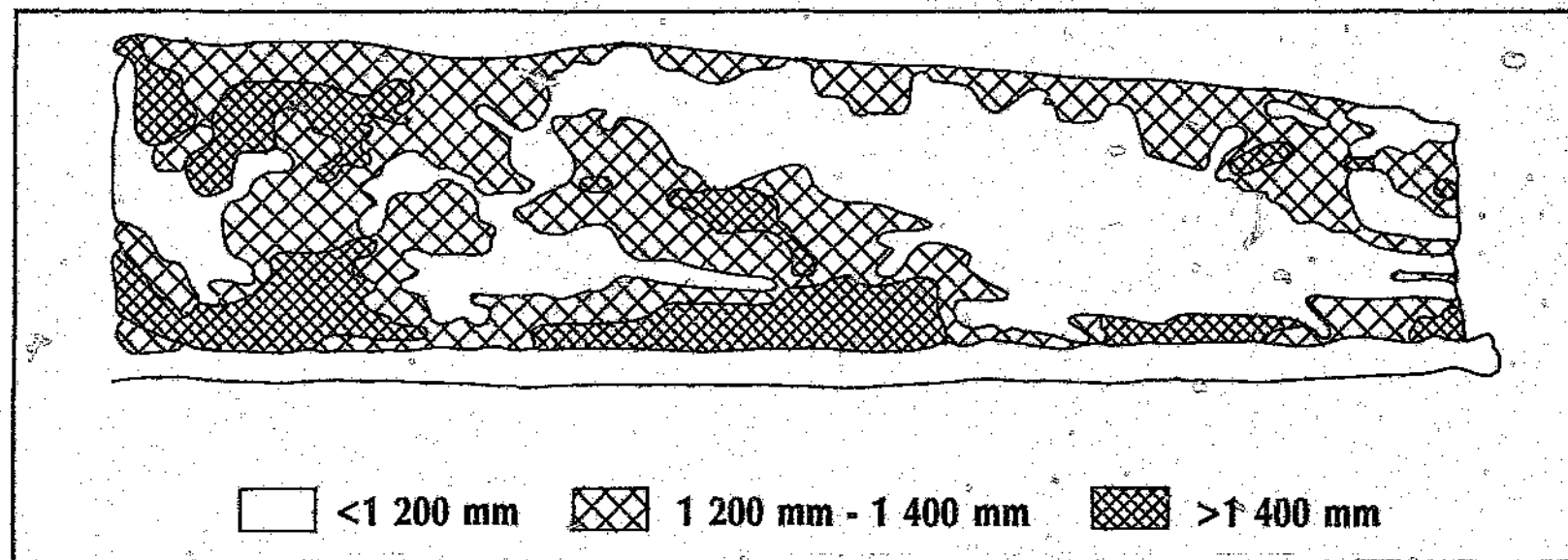


Figure 38 -

Stoping Width Contours in 72-43W N6 Panel using
200/400kN Props.

statistic is extrapolated from accident figures recorded for the first quarter of 1993. Latest second quarter figures indicate that this reduction in fall of ground casualties is being maintained, almost certainly due in part to the continuing expansion of the blast-on prop and waterjetting stoping system into other areas of the mine during this year.

Given this background, and the obvious success, albeit on a small scale, of the experimental work being carried out with 200/400kN blast-on props and waterjetting, management at Hartbeestfontein was quick to approve substantial sums of money for the purchase of props and ancillary equipment for large scale conversion to this stoping method on Nos. 4, 5 and 6 Shafts. The first orders were placed in April 1992, and deliveries commenced that June. To date in excess of 10 000 lightweight, blast-on props have been purchased at a cost of R20m, and more than 150 panels have been equipped with the system.

5.2.2.1 Mine Specifications.

Before any prop purchases were made, some fairly basic policy decisions were made by the mine regarding the adherence to the specification of 200/400kN blast-on props and prop components to specifically suit caving and other conditions pertaining on Hartbeestfontein Mine. Some of these requirements are already built into the system, others were suggested by experience during the developmental stage, and yet others are aimed at ensuring the long-term cost competitiveness of the system:

a) Performance Capabilities.

i) Blast-on Capability.

All props, extensions and load-spreaders are required to continuously withstand, without adverse consequences, the effects of being installed and left 1m from the stope face during blasting operations.

ii) Rockburst Capability.

All load-spreaders and extensions are required to form an integral part of the support system. In particular, they must have rockburst loading capabilities similar to that of the rapid-yielding props they support.

b) Prop Accessories.

i) Load-Spreaders.

Load-spreaders must provide adequate areal coverage, must reduce unsupported spans, and must minimise the blast-out rate of props installed close to the face. They must also be robust, light and easy to use.

ii) Extension Pieces.

Extensions must be of the conical variety imparting stability and flexibility to the system and allowing the use of the minimum number of prop sizes to accommodate a wide variety of stoping widths.

iii) Foot-Pieces.

Swivel-type foot-piece attachments must be available for use in weak footwall situations where footwall punching may present a problem.

c) 200/400kN Hydraulic Props.

i) Operation.

Props must have a low mass to facilitate ease of installation and removal, and to allow for temporary removal and replacement in the event of their interfering with drilling operations.

ii) Caving Conditions.

It is essential that props are equipped with an efficient means of remote release, and that this operation occurs flawlessly every time, particularly under caving conditions. To this end, the moving surfaces (ram and cylinder) of all props are to be fabricated from high quality stainless steel to avoid jamming caused by corrosion.

iii) Maintenance.

The use of expensive stainless steel fabrication materials is seen as contributing towards lowering long term maintenance costs and increasing the life-span of props.

iv) Stability.

All props must have conical heads to conform with the requirements of stability and flexibility discussed above.

d) Equipment Specifications.

i) Compatibility.

A high degree of compatibility and interchangeability must exist between the various prop and accessory components, the most important being intensifier pumps, setting nozzles, release tools and head extensions. This does not mean that equipment must be identical, but it does mean that it should be completely transparent to the operator underground that he is using, for example, an Elbroc pump and setting pistol to set a Klockner-Becorit prop fitted with a Unique extension and load-spreader.

ii) Quality Testing.

All props and accessories must be checked through the manufacturers quality control system, and representative sample batches must be fully evaluated by Anglovaal quality control engineers and the Chamber of Mines Research Organisation for compliance with the appropriate specifications.

iii) Competitive Pricing.

A policy of encouraging healthy competition between the various suppliers is actively pursued with a view to optimising cost benefits. Care is, however, exercised to avoid a proliferation of prop varieties which could lead to high spares and maintenance costs.

A cut-away diagram of a typical 200/400kN rapid-yielding, light-weight, blast-on, hydraulic prop developed for use at Hartebeestfontein Gold Mine, complete with blast protection sleeve and conical head, is shown in Figure 39. Plates 5 and 6 are views of these props installed in 78A-42W N7 Panel situated 2370m below surface at No.5 Shaft.

5.2.2.2 Blast-on Prop Layouts.

In much the same way as load-spreaders, conical extensions and blast protection sleeves are envisaged as not being just useful accessories, but rather as an integral part of the total 200/400kN prop design, so too should blast-on prop layouts be designed to take full advantage of the systems potential to create the safest possible working environment at the stope face. This can only be achieved by bringing the support as close to the face as possible, taking full advantage of the blast resistant qualities of these props. This approach naturally does not mesh well with the practice of confining blast rock ahead of the face row of support using conventional dip barricades. Hence the development of diagonal barricades and waterjet cleaning.

A support layout which incorporates the uses of blast-on props, diagonal barricades and waterjetting, and combines these with back-area caving, is shown in Figure 34G. The props are installed such that they are never more than 2,5m from the face after the blast. Props are spaced 1,5m apart on dip, and the rows are spaced a nominal 1m apart, or the equivalent of the face advance distance. Four rows of props are required, mainly to provide adequate space and protection for the waterjet operator. Caving occurs behind the fourth row of props. In cases of extreme friability where the possibility of a cave over-run exists, the number of props in the back row may be doubled to create a breaker line.

Props are installed diagonally on strike so that they subtend an angle of about 70° below the direction of strike. Load-spreaders on the other hand are angled 70° up from strike, pointing them into the blast thereby minimising blast-outs, and facilitating areal coverage of the predominantly face parallel fractures. This allows drilling operations to take place under maximum protection and with the minimum of interference from the props themselves. Blast barricades are installed diagonally against the second and third rows of props, presenting a solid barrier preventing fly dirt from being lost in the back area. Because the muck pile is not confined to the immediate face area, lying as it does between the props against the barricades, the face scraper is able to move immediately to the top of face, greatly facilitating the face cleaning operation. Waterjet pressures of the order of 35MPa make waterjet cleaning between props almost effortless. Face scraping operations are rendered more efficient because broken rock is continuously fed into the scraper path from the barricades, a distance of less than 3m. An additional advantage is that cleaning and sweeping now occur simultaneously.

This system allows the all important strike spacing of props to be reduced to an absolute minimum, in this case 1m. The maximum distance of the back row of props from the face

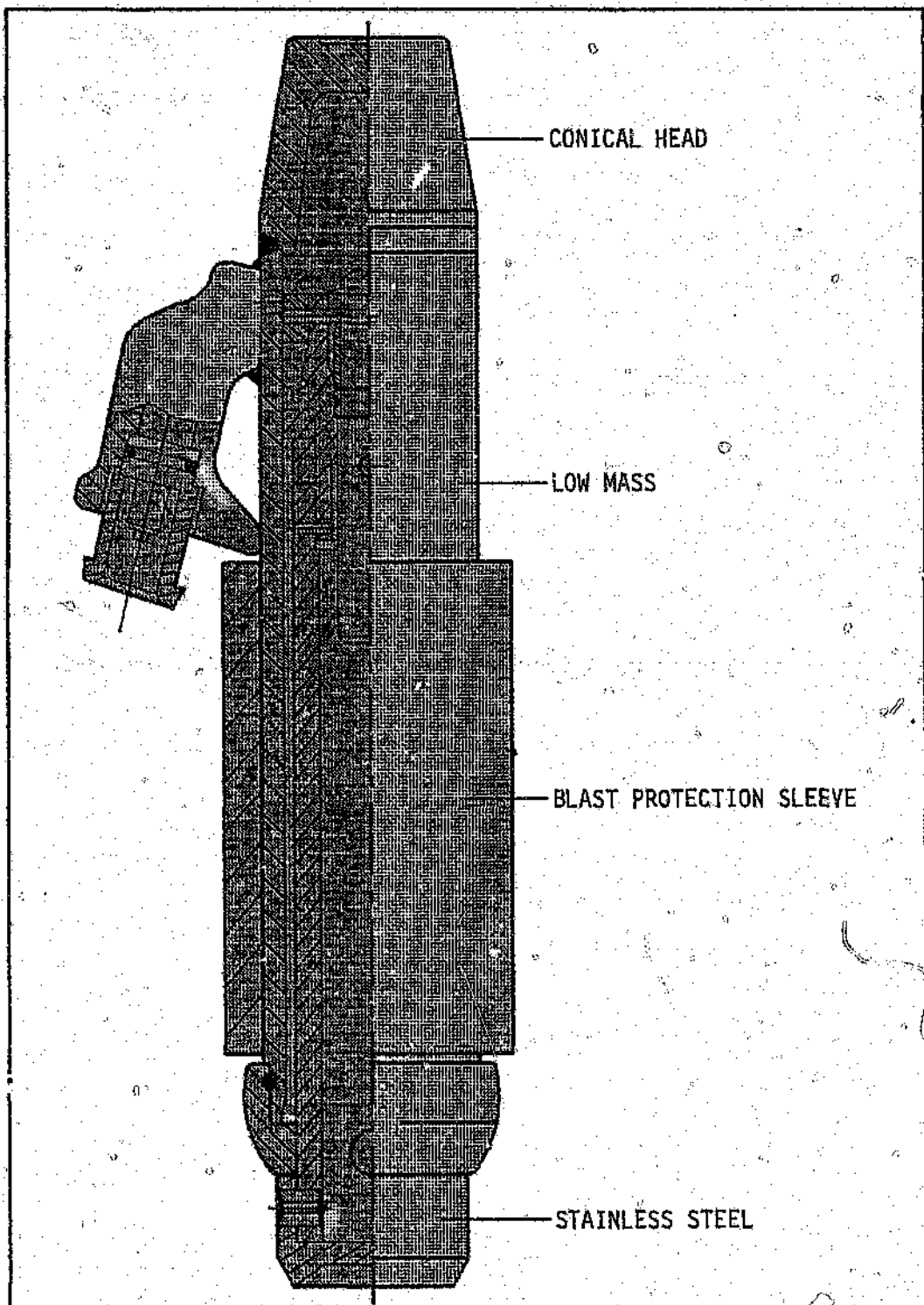


Figure 39 - Cut-Away sketch of 200/400kN Rapid-Yielding, Light-Weight, Blast-On, Hydraulic Prop.



Plate 5 - 200/400kN Blast-On Props Installed In 78A-42W N7 Panel At No.5 Shaft.



Plate 6 - 200/400kN Blast-On Props Installed In 78A-42W N7 Panel At No.5 Shaft.

is 5.5m. This compares with existing conventional caving standards of 5.5m for three rows of 400kN props and 7.1m for four rows of sticks. Caving thus occurs at about 6m or 7m behind the advancing stope face.

Two fairly minor constraints associated with the blast-on prop / diagonal barricade / waterjetting method are, firstly, drilling interference and, secondly, prop blast-out rate. The first constraint is minimised to some extent by installing props at the same angle to the face as that required for the drill holes. The removal and resetting of props does, however, present legal ramifications as the law requires the setting of temporary support in the place of any props that are removed. As far as the second constraint is concerned, monitoring of blast-out rates of the front line of props has revealed that this is related much more to stoping width than to prop setting load or the distance that props are set from face. Monitoring also revealed that the use of tri-spreaders greatly reduced the incidence of blasted-out props. Fortunately, the props in the second, third and fourth rows, on which the blast barricades are suspended are rarely blasted out.

As an alternative to waterjet cleaning and diagonal barricades, Figure 34F illustrates the layout to be used for 200/400kN props with conventional dip barricades and scraper cleaning. The layout is exactly similar to the current 400kN prop cave support standard.

Plates 7 & 8 demonstrate the effectiveness of the 200/400kN blast-on prop system in providing a practical solution to the problem of supporting cave mining stope faces in the weak hanging wall conditions encountered within the pseudo-breccia zone in the 5A North area at No.5 Shaft. Not only has the support system been successful in maintaining safe working conditions in an environment which might otherwise be regarded as potentially hazardous, but stoping widths in the pseudo-breccia zone have actually reduced since its introduction. This sustained reduction of the average stoping width in the No. 5 Shaft stoping area is shown graphically in Figure 40.



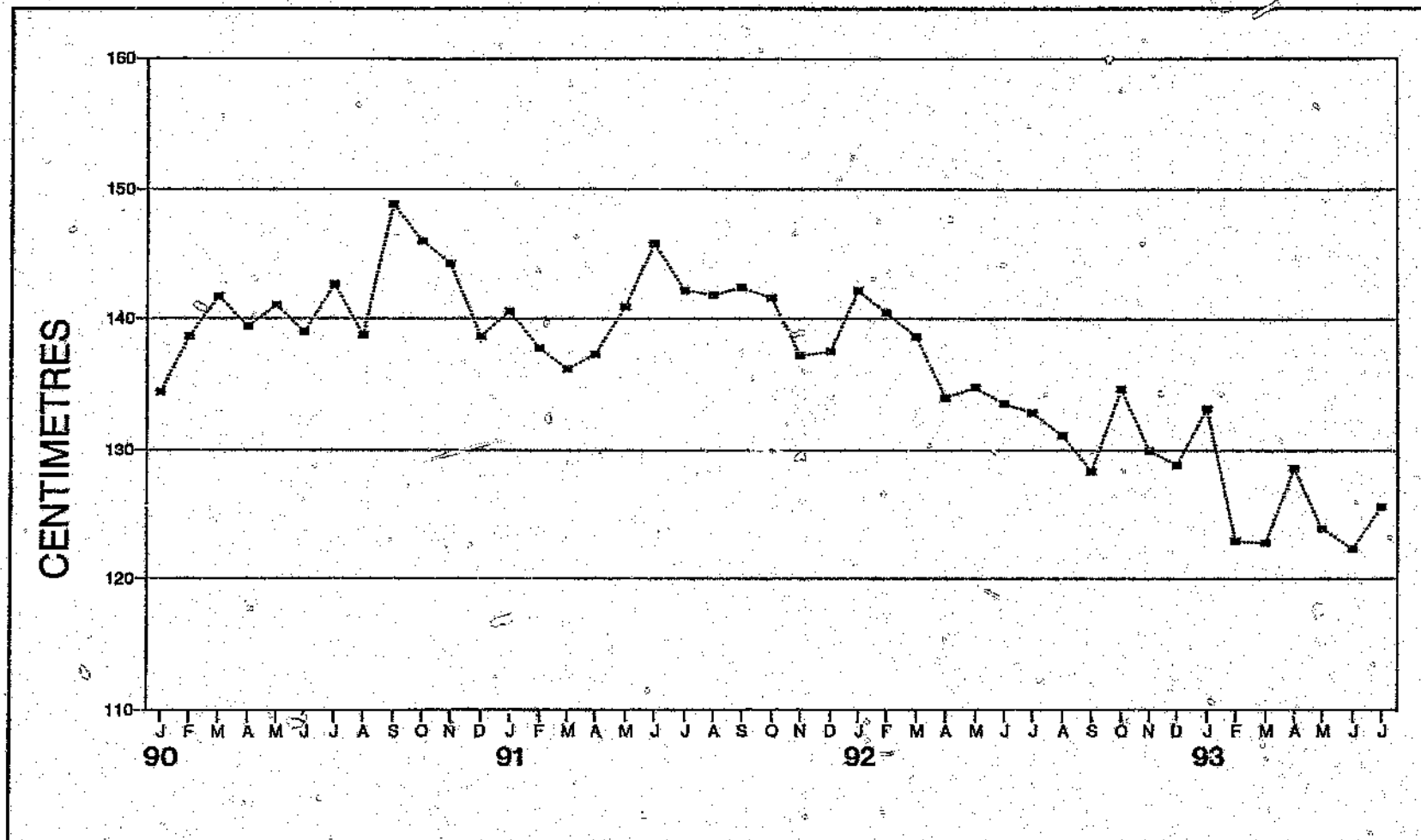


Figure 40 - Graph Showing Sustained Reduction In Average Stopping Width At No.5 Shaft.



Plate 7 - The Effectiveness Of The 200/400kN Blast-On Prop System In Reducing Stopping Width At No.5 Shaft.

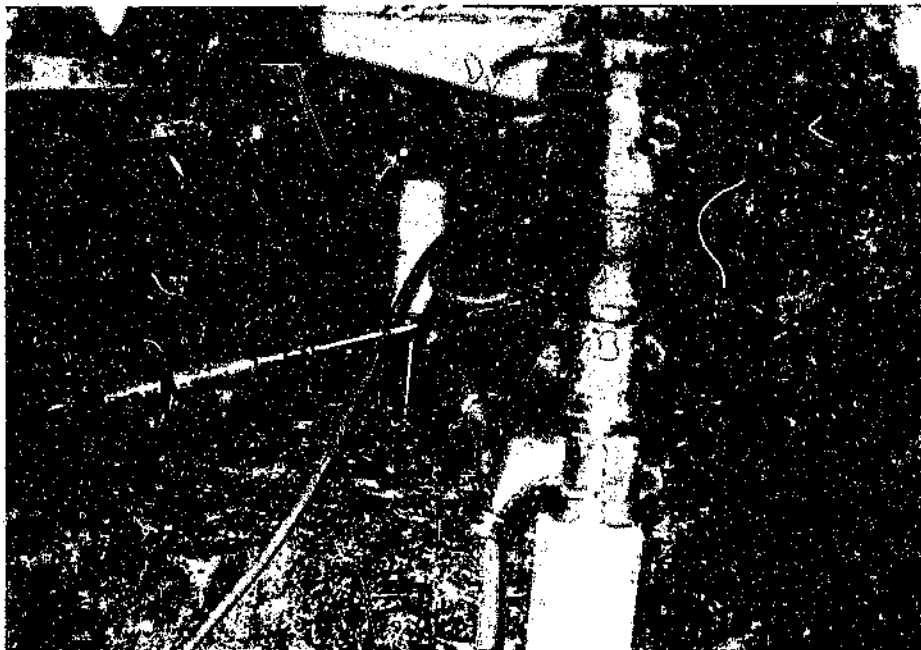


Plate 8 - The Effectiveness Of The 200/400kN Blast-On Prop System In Protecting Workmen At The Stope Face.

6. BACKFILL ALTERNATIVE.

6.1 INITIAL JUSTIFICATION.

The use of backfill as a local and regional support strategy has become increasingly more widespread within the South African gold mining industry during the last decade as technological problems associated with the technique have been addressed and, in most instances, solved or resolved. As Hartebeestfontein Gold Mine experiences a wide range of mining problems for which backfill is a possible solution, it is natural that serious consideration began to be given to filling on the mine during the late 1980's. The major areas of concern for which the role of backfill was investigated were:-

- a) Amelioration of the incidence and effects of rockbursts in areas identified as active seismogenic regions on the mine.
- b) Improvement of strata conditions in situations of weak and friable hangingwall, such as occur in the pseudo-breccia zone.
- c) Reduction of the hazards associated with the extraction of pillars and remnants, thereby increasing the availability of such reserves.
- d) Stabilisation of shafts, accessways and other excavations during shaft and haulage pillar extraction.

Of specific interest in the context of this project report was the consideration of backfill for the 5A North area as a means of overcoming the considerable strata control problems which were anticipated in the pseudo-breccia zone. It is interesting to note that hangingwall conditions were considered so adverse that both backfill and conventional support were contemplated as alternatives to cave mining. These comparisons were made some time before the advent of blast-on props and loadspreaders. A brief historical overview of the main investigative documentation compiled by, or on behalf of Hartebeestfontein is as follows:-

- a) A detailed feasibility study⁽⁴⁹⁾ of mining the 5A North area using cycloned tailings backfill as a means of support was undertaken and completed in February 1988. Specifications for a backfill plant, storage facilities, surface and underground piping, and stoping and placement methods were presented. Capital expenditure of R10 million and running costs of R200 000 per month were estimated to supply all mining in the area (13 500m³/month) with 1200 tons of fill per day. The backfill running cost was calculated to equate to R14,66/m³ mined or R9,59/ton placed. Scheduling priorities were also outlined.
- b) A consolidated assessment⁽⁴⁹⁾ of backfilling at No.5A Shaft was submitted for Anglovaal head office approval early the following year. The proposed deslimed tailings system comprised four elements: processing, storage, transportation and placement. Fill would be pumped from processing to storage, but would be reticulated in the mine by gravity feed. Capital of R14 174 000 was applied for, to be split R10 million in the 1989/90 financial year and the balance in 1990/91. A backfill cost of R17,26/m³, against a conventional support cost of R21,11/m³, was predicted. This, with other potential savings such as reduced refrigeration, stoping width, seismicity, prop losses, on-reef maintenance, and improved safety, realised an

estimated total cost benefit of R14,51/m². For the projected production rate of 158 000m² per year the mean nett working cost saving was calculated to be in the region of R2,3 million per year, which would allow the capital to be recouped in six years.

- c) The special projects division of the Chamber of Mines Research Organisation issued a consultancy report⁽⁴⁹⁾ dealing with theoretical considerations for the backfill system in October 1990. This covered the conceptual design of a classified tailings backfill preparation plant and distribution system for supplying backfill to No.5A Shaft, based on the particle size distribution of residue product produced by the mines gold plant. The design was subsequently confirmed by full scale tests conducted on a bulk sample of tailings at the Chamber of Mines backfill preparation, hydraulic transportation and placement test facility at Western Deep Levels. The conceptual design was developed into detailed process flow, piping and instrument diagrams, providing detailed information on the major equipment, pump, storage tank, jilping and valve requirements. To minimise pipeline failure generally associated with free-fall systems, a full-flow underground distribution system was proposed.
- d) The whole question of the viability of backfill was reassessed as the economic climate continued to deteriorate during 1991. The debate devolved around a number of options:-
 - i) Whether to delay the backfill implementation program indefinitely, or alternatively to press ahead with the project, and if so,
 - ii) Whether to concentrate on using backfill to reduce seismicity and ultimately aid in the extraction of highly stressed shaft and haulage pillars, or alternatively to use backfill to improve strata control in the weak and altered ground at No.5A Shaft, and if so,
 - iii) whether to initially install a 7000t/mth modular mini-plant to supply four panels in the 5A North area with a view to later upgrades, or alternatively to immediately install a 32000t/mth plant to supply the whole 5A North area.

It was decided to consider backfill as only one of the options for the safe extraction of the 5A North area, and to further investigate other possible backfill applications such as its role in the removal of the No.6 Shaft haulage and shaft pillars.

- e) Towards the end of 1991, the Chamber of Mines Research Organisation issued theoretical capital and working cost estimates⁽⁵⁰⁾ for a full-blown system to supply both No.5A Shaft and No.6 Shaft with backfill. The calculations were based upon a system capable producing a maximum of 22 000 dry tons of backfill per month; equivalent to catering for a stoping rate of 14 000m² per month at a stoping width of 130cm and an area filled of 80%, assuming a plant utilisation of 75%. The capital cost was estimated to be just less than R16 million. Working costs were recognised to be dependant upon plant utilisation efficiencies. They varied from R35,86/m² or R10,32/ton mined (R23,22/dry ton of backfill placed) for 75% plant utilisation, to R53,93/m² or R15,52/ton mined (R34,92/dry ton placed) for 35% plant utilisation.
- f) Early the following year, the rock engineering division of the Chamber of Mines Research Organisation issued a final consultancy report⁽⁵¹⁾ (a progress report had been

issued in September 1991) which evaluated backfill for the No.5A and No.6 Shaft areas from a rock engineering point of view. Because it was not possible to model caving, no comparison could be made between the strata control benefits of mining with backfill and caving in the 5A North and the No.6 Shaft areas, but modelling did reveal that strata control and cost benefits resulted from the use of backfill when compared to mining with timber packs.

At this stage a substantial amount of work, covering the justification of fill, the conceptual design of preparation plant and reticulation systems, cost-benefit analyses, and so on, had been documented. Capital had been approved for preliminary design work (R0,5 million) and had been applied for in respect of a facility to supply the 5A North area (R14 million). However, as a result of the declining economic situation, it was decided to re-evaluate the advisability of incurring this expenditure.

6.2 EVALUATION OF BACKFILL.

6.2.1 Industry Literature.

Much has been written on the subject of backfilling in deep South African gold mines, particularly following the escalation of seismicity associated with ultra-deep mines. Because of the extremely fine grind in modern reduction plants and the abrasive quality of the quartzitic rock, the technological requirement of present day backfilling is substantially more stringent than that of yesteryear. Modern backfilling has its origins in the late 1960's and since then a great deal of theoretical and practical experience has been built up and recorded by the mining groups, the Chamber of Mines, consulting firms, universities, the service industry, and other agencies. As it is beyond the scope of this project report to include an exhaustive treatment of this body of literature, only a very brief overview of the more recent material is attempted, concentrating on that which deals with the observation and measurement of the strata control benefits of existing backfill applications.

6.2.1.1 In-situ Behaviour.

Underground instrumentation and monitoring work carried out by the Chamber of Mines Research Organisation at a number of backfill sites in deep gold mines, and reported upon by Gurtunca et.al.⁽⁵⁹⁾ and Adams et.al.⁽⁶⁰⁾, has shown that there is good correlation between the measured in-situ and laboratory behaviour of backfill, and has resulted in the development of a conceptual model which may aid numerical modelling. The main findings of the work are as follows:-

- a) Backfill stresses are low close to the edge of backfill ribs, increase to a peak 1m - 3m from the backfill edge, and then decrease towards the centre of the rib.
- b) Closure, or compression of the fill, is a minimum at the centre of the rib, and increases to a maximum at the edge.
- c) Confinement of the fill decreases outwards from a maximum at the centre of the rib to zero at the rib edge.

- d) Backfill stress at constant strain is a maximum at the centre and decreases outwards to zero at the edge.
- e) Backfilling minimises and eventually stops bedding planes from opening up and slipping relative to each other.
- f) Backfilling reduces closure in stopes.

Extensometer measurements taken in hangingwall holes suggest that small openings do occur along parting planes initially, particularly close to the stope hangingwall and footwall, but that as stresses build up in the backfill the rockmass becomes clamped. This is in contrast to unfilled stopes where there appears to be continued movement along bedding planes.

6.2.1.2 Backfill and Rockfalls.

An extensive analysis of rockfall related accidents was carried out by the Chamber of Mines Research Organisation at several gold mines which use backfill. The analysis was reported upon by Squegli et al.⁽⁶²⁾, and showed that a marked reduction in rockfall accidents was achieved in backfilled stopes compared to unfilled stopes if at least 60% of the mined-out area was filled. Conversely, if the area filled was less than 40% there was an increased occurrence of rockfalls in backfilled stopes. It was concluded that backfill does stabilise the stope hangingwall and has a positive influence on the occurrence of rockfalls provided it covers an adequate proportion of the stope area.

The analysis confirmed that the majority of accidents were associated with falls less than 2m thick. Stress and strain measurements carried out at depths ranging from 0,7m - 2,2m into the hangingwall close to the face indicated that both principle horizontal stresses are compressive in filled stopes while only one is positive in unfilled stopes. The implication is that the biaxial compressive stress regime found in backfilled panels improves the stability of the hangingwall and reduces rockfall accidents.

6.2.1.3 Industry Benefits.

The beneficial influence of backfill on the following aspects of strata control and stoping efficiencies were assessed by the Chamber of Mines Research Organisation and reported upon by Gurtunca⁽⁶³⁾:-

a) Rockburst Damage.

Following an evaluation of about twenty-five rockburst damage observations carried out in backfilled and unfilled stopes over a four year period, it emerged that rockburst damage was minimal in backfilled stopes where fill to face distances were less than six metres and good face area support was used between the face and the backfill.

b) Stope Width Control.

Only marginal differences in stoping widths were observed between filled and unfilled panels over periods of up to two years. It was concluded that the

influence of backfill on stoping width control is insignificant.

c) Centares Mined.

Productivity was also only marginally improved in filled stopes compared to unfilled stopes. However, it was noted that backfill was generally confined to those areas where more extreme conditions existed, making the comparison somewhat biased.

d) Rockfall Accidents.

As noted earlier, the incidence of fall-of-ground accidents is reduced if the area filled exceeds half of the total area mined in backfilled stopes, and increases if the percentage filled is less than half. The likely reasons advanced for the increase in accidents in backfilled stopes are, firstly, the probable lack of conventional support where backfill placement is inadequate, and secondly, probable deterioration of the hangingwall condition caused by low percentage filling.

6.2.1.4 Industry Problems.

The industry has experienced, and largely overcome, a wide range of problems associated with backfill. A short resume, dealing mainly with the Anglo American experience, has been published by Spearing⁽⁷⁾, and is summarised as follows:-

a) Metallurgical Preparation.

Cycloning must be executed correctly and consistently to ensure a free-draining, yet dense product.

Provision must be made to dispose of or deal with the very fine material in cyclone overflow.

The presence of free or complex cyanide must be kept within permissible limits.

b) Overland Transportation.

Pipeline and pump specifications must be correctly matched to avoid unnecessary dilution of an otherwise acceptable product.

Gland water leakage, which can also lead to unnecessary dilution, must be eliminated.

Back-up facilities and stand-by flushing systems must be in place to deal with emergencies.

c) Underground Distribution.

Major pipe repair and replacement costs associated with the enormously high

abrasive qualities of quartzite slurry must be engineered out of the system.

The creation of dust resulting from shaft pipe rupture, which poses a threat to both equipment and personnel, must be avoided.

Fill must be available at the desired rate in sufficient quantity and for an adequate period to ensure that the production cycle is not delayed.

d) Stope Placement.

Fill must be placed close to the face, and be in good contact with the hangingwall to be effective.

Water and solids run-off must be controlled such that fill strength is maximised.

Adequate arrangements must also be made to control the flow of run-off material into gullies, rock-passes, crosscuts and haulages.

The retaining structure must be sufficiently strong to withstand the hydraulic pressures involved.

The breadth of the paddock or bag into which fill is placed must be small; commensurate with keeping fill as close to the face as possible.

e) Face Activities.

Backfill must always be supplemented with high-quality temporary support at the face.

Control and discipline must ensure that sweepings are not lost under backfill.

f) Costs.

Backfill costs must be contained by eliminating wastage, and avoiding unnecessary double support within panels, ie. using packs or elongates and backfill.

6.2.2 Potential Backfill Benefits.

Much has been claimed concerning the effectiveness of backfill in reducing the hazard and cost of deep level mining. In theory backfill has the potential of delivering substantial benefits as a regional support strategy (reducing the energy released by mining activities) and as a local support strategy (improving strata control conditions). In practice these benefits are heavily dependant upon the quality of the fill material delivered underground, the efficiency with which it is placed, the areal extent of filling, and the mining geometry and span.

Provided that the quality and quantity requirements are met, backfill has the following potential benefits:-

a) **Energy Reduction.**

- Reduced seismic frequency.
- Reduced seismic magnitudes.
- Reduced on-reef stress.
- Reduced off-reef stress.

b) **Radiated Seismic Energy.**

- Absorption of seismic wave energy.
- Transmission of seismic wave energy.
- Reduced "free-beam" effect.
- Reduced seismic damage.

c) **Strata Control.**

- Weak ground stabilisation.
- Improved hangingwall conditions.
- Reduced fall-of-ground incidence.
- Reduced stoping width.
- Reduced seismic damage.

d) **Production Benefits.**

- Improved safety and health.
- Improved working conditions.
- Increased mining efficiency.
- Reduced stoping widths.
- Increased ore-reserve availability.
- Increased reef extraction.
- Improved sweepings.
- Reduced gold loss.
- Reduced timber consumption.
- Reduced (timber) transport requirements.
- Reduced access damage.
- Off-reef excavation protection.
- Reduced surface tailings disposal.

e) **Industrial Relations Benefits.**

- Improved safety and health.
- Improved working condition.
- Improved industrial relations.

f) **Environmental Benefits.**

Reduced fire risk.
Increased ventilation efficiency.
Reduced cooling requirements.
Better ventilation controls.
Improved working conditions.

The pros and cons of most of the above points are elaborated upon in the reports and justifications referred to previously. These also emphasise the need to produce high quality material, the need to backfill a significant proportion of the mined area, and the need to place it close to the face in order to realise the full potential of backfill, and to avoid a situation in which intended advantages become disadvantages. Naturally, if forecast benefits are not realised, backfill becomes a financial drain. The important feature is thus, not so much whether backfill has the potential to solve mining and support problems per se, but whether sufficient discipline and control can be exercised to ensure that fill is placed efficiently and effectively to realise its full potential.

6.2.3 Potential Problem Areas.

A fairly substantial body of practical experience has been accumulated concerning the in-situ performance, application, effectiveness and cost of backfilling in the South African gold mining environment. This tends to confirm theoretical predictions that backfill has the potential of making a significant contribution to improving safety and productivity, particularly on deep-level mine, provided the material is of good quality and is correctly placed. The strategy is, however, not without its drawbacks. Some of the more important problem areas may be summarised as follows:-

a) **Shallow Dip.**

Fill effectiveness is markedly influenced by fill material properties and placement factors such as:-

- i) Fill contact with the hangingwall.
- ii) The degree of compaction achieved.
- iii) The amount of fill shrinkage.

A steeply dipping ore-body would tend to counteract or eliminate these effects. Shallow and undulating reef dips, and a high incidence of faulting, on the other hand make effective placement of fill much more difficult and problematical.

b) **Backfill-to-Face Distance.**

Perhaps the most important parameter governing the effectiveness of backfill is the distance at which it is placed behind the advancing face. The placement of fill very close to the face is often hampered by constraints such as paddock building, mining cycle, faulting, and so on. But, unless fill is

effectively placed close to the face, it is a waste of money and effort.

c) **Face Support Requirements.**

Backfill only generates significant load some distance back from the face. Thus, substantial "unsupported" spans can occur at the face in backfilled panels. Conventional face support is therefore also required. This can have the effect of increasing fill support costs or of creating an air of false security if additional face support is not provided. It is not, however, necessary to provide additional conventional permanent support. Temporary, or rolling support, which will not escalate costs excessively, is quite adequate.

d) **Chamber of Mines Findings.**

After thorough, industry-wide investigation, the Chamber of Mines Research Organisation presented the following findings at a report-back to the industry seminar in March 1991:-

- i) Backfill costs are significantly greater than those for conventional support.
- ii) The influence of backfill on stoping width control is insignificant.
- iii) Rockfall accidents are decreased only when the percentage filled exceeds 50%, and in fact increase if the percentage filled is less than 50% of the area mined.
- iv) Production in backfilled stopes is better than in unfilled stopes.
- v) Rockburst damage is reduced only when fill is kept less than 5m from the face and good face area support is used between the backfill and the face.
- vi) Moderate reduction in seismicity occurs only after extensive and prolonged filling.
- vii) Backfill does not reduce seismicity in the presence of geological features.

Although the prognosis was not all negative, backfill was not endorsed unreservedly.

e) **Computer Modelling.**

Elastic modelling is only meaningful in the context of global energy changes, that is, in the assessment of seismic energy reduction, stress changes at points remote from the stope, and so on. It is less meaningful when applied to strata control improvements, which require evaluation of the essentially plastic behaviour of the fractured zone surrounding excavations.

f) **Supervision and Quality Control.**

Most, if not all, of the advantages of backfill will fall away if high quality material is not delivered in sufficient quantity at the face and if it is not placed correctly. A high degree of supervision and control of the operation, from production in the metallurgical plant right through to placement at the face, is absolutely essential to derive benefits from the strategy. This has proved to be a major stumbling block throughout the industry. Wastage of fill product and expensive consumables, such as flocculant and cementitious agents, and failure to recover sweepings before filling, are aspects of the operation which can lead to the rapid eroding of any cost benefits if not effectively controlled.

g) **Planning and Scheduling.**

Fill must be available on the stope face when needed. Production staff cannot afford to hang-up blasts because fill is not available in sufficient quantities as a result of poor planning, delays, breakdowns, and so on. Equally, over-supply of fill material can lead to spillage, and the flooding of gullies, boxholes and crosscuts, with resultant possible loss of production. It is therefore essential that operations are carefully planned and scheduled, and that adequate means of communication and supervision are available to effectively control all stages of the process and to implement speedy corrective action when breakdowns occur.

h) **On-the-ground Perceptions.**

Although the official line is that backfill is serving a useful purpose, the perceptions of ground-level operating staff are sometimes less positive.

i) **High Horizontal Stress.**

There have been reports⁽⁷³⁾ of hangingwall damage due to excessively high horizontal stress resulting from the use of backfill. In some instances this has reportedly led to instability in the hangingwall immediately adjacent to the working face, a phenomenon not uncommonly experienced in "uncaved" caving panels at Hartebeestfontein. As yet these observations are preliminary and have not been fully evaluated or substantiated.

j) **Plant and Equipment.**

The manufacture, transport, distribution and placement of highly abrasive, quartzitic material has presented researchers, engineers and management with many major problems during the years of developing backfilling techniques and equipment. Perhaps the most difficult of these was that of wear, particularly in vertical pipe columns. Happily, solutions have been found for most, if not all of the problems, and this technology is freely available. There are, however, cost and control penalties built into these systems (shaft pipe costs for example escalate two-fold as a result of the implementation of these measures).

6.2.4 Cost Analysis.

The real costs associated with currently operating backfill installations are notoriously difficult to quantify, and are fraught with assumptions and exclusions. The following brief summary of some generally accepted figures and projections is intended as a rough guide and is not an exhaustive analysis of backfill costs.

6.2.4.1 Capital Cost.

Some twenty five mines are currently placing about one million tons of backfill on seventy five kilometres of stope face per month. The capital cost of a backfill installation is high, ranging between 10 and 50 million rand depending upon the size of the operation. Vaal Reefs for example are spending R21 million on extensions to double their No.4 Shaft backfill capacity. The capital cost of introducing backfill on Hartebeestfontein was estimated in October 1991 to be R16 million for a 22 000 ton per month plant (capable of catering for a stoping rate of 14 000 m² per month). Off-setting this against forecast savings in support costs, reduced refrigeration, stoping width, seismicity, prop losses, and improved safety, it was calculated that the capital would be recouped in six years. A debatable argument can be made that, having chosen the backfill route, it becomes necessary to continue along the road, almost regardless of whether the original goals are achieved or not, because of the sheer magnitude of the investment. The decision of whether or not to embark upon a backfill program is thus not lightly taken.

6.2.4.2 Operating Costs.

The operating or working cost of backfill may be calculated in terms of tonnage, but is more commonly quoted in terms of square metres filled or square metres mined. Operating costs depend upon factors such as plant utilisation, rate of filling and, most importantly, stoping width. It has also become conventional to quote backfill operating costs per square metre relative to standard timber support costs, and then to determine the break-even stoping width above which it becomes cheaper to use backfill than conventional timber pack support. For Anglo American⁽⁷⁾ mines the breakdown is as follows:-

Cycloned Backfill breakeven at 125cm.	R40.00/m ²
Dewatered Backfill breakeven at 150cm.	R90.00/m ²
Silicated Backfill breakeven at 150cm.	R90.00/m ²

The approximate total support cost for an average stoping width of 130cm. on Hartebeestfontein has been calculated as follows:-

Stick Caving at 130cm.	R43.28/m ²
Prop Caving at 130cm.	R44.16/m ²
Packs & Props at 130cm.	R77.41/m ²
Backfill & Props at 130cm.	R63.07/m ²

From these figures it is concluded that the break-even stoping width when comparing pack support and backfill on Hartebeestfontein will also be in the order of 125cm. Because stick caving and prop caving are relatively independent of stoping width, it is further concluded that backfilling will always be more expensive than caving.

6.3 BACKFILL CONCLUSIONS.

Although backfill has proven potential, as a regional and as a local support strategy, for stabilising on-reef and off-reef excavations and for reducing both the incidence and damaging effects of rockbursts, and has further spin-off potential relating to ventilation improvements, the following conclusions were reached regarding the application of backfill on Hartebeestfontein:-

- a) Backfill would probably not be effective in reducing seismicity to any large extent.
- b) Future stoping width reductions would be dependant upon good face support, regardless of whether backfill was used or not.
- c) Good quality fill has the potential for reducing, but not eliminating, mining induced stress and deformation damage to shafts and shaft pillar excavations.
- d) High standards of supervision and quality control are essential.
- e) Backfill would be a waste of money and effort if:-
 - i) The quality of product was sub-standard.
 - ii) Fill was not placed close to the face.
 - iii) Fill was not in contact with the hangingwall.
 - iv) Excessive fill shrinkage occurred.
 - v) Substantial areas were not filled.

With regard to its application for the solution of the specific problem of weak hangingwall conditions in 5A North pseudo-breccia area, backfill was rejected as an alternative to caving for the following reasons:-

- a) Equivalent, intense conventional support would be required at the face for both options.
- b) The very shallow dip across the whole area would make effective filling difficult to accomplish.
- c) The high frequency of faulting would mitigate against effective filling.
- d) The level of seismicity was not expected to rise dramatically because of the weak rock conditions.
- e) Backfill would not be a cost effective solution for the 5A North area.
- f) The money required to provide fill could be better spent on improving face support density.
- g) Caving is cheaper than stoping methods which employ backfill or other means of back area support.

Preliminary elastic modelling of the 5A North area, simulated with and without backfill, indicated that relatively little advantage was to be gained from the use of backfill as a regional support strategy. Stresses and strains at the shaft position were affected only marginally.

7. INSTRUMENTATION AND MEASUREMENT.

7.1 TEST SITES.

Although the work described in this project report has tended to focus on the development of a support methodology for pseudo-breccia regions in the 5A North area, an important part of the exercise as originally envisaged related to applying these support developments to cave mining, and evaluating the suitability and effectiveness of both the support and the cave mining method in weak ground conditions. It was therefore necessary to measure, quantify and interpret stress levels, rock structure and strata movement in the hangingwall of stopes situated within the pseudo-breccia alteration zone with a view to comparing and contrasting this information with that previously obtained from caving stopes in less disturbed ground, and in which more conventional support methods were employed. The information was also required to generate input data for non-linear computer modelling discussed later, and for the calibration of these models.

Monitoring stations were established prior to and during the experimental introduction of 200/400kN blast-on props in two stopes situated within the pseudo-breccia zone. The test panels are listed below and are indicated on a plan of the 5A North Area in Figure 36:-

STOPE	PANELS	DEPTH	MONITORING PERIOD
72-43 West	S4 and S5	2200mBS	Dec, '91 - Oct, '92
72-43 West	N7 and N8	2200mBS	Jun, '92 - Feb, '93
78A-42 West	N6 and N7	2370mBS	Dec, '92 - Jul, '93
78A-42 West	S4 and S5	2370mBS	Mar, '93 - Dec, '93

Comparative testing of the new generation Klockner-Becorit, Elbroc and Unique props was also conducted in 78 N10 Inter N12, N13 and N14 Panels, but no measurements of rock deformation were made in this stope.

The instrumentation and monitoring program attempted to address the following features which it is believed need to be quantified in order to gain an understanding of the mechanics involved in the deformation of rock surrounding cave stoping operations:-

- The juxtaposition, intensity and orientation of discontinuities within the hangingwall.
- The surface properties and shear strength of hangingwall discontinuities.
- The strength and failure characteristics of the intact rock making up the hangingwall.
- The degree and rate of stope closure and any associated ride components.
- The extent and nature of strata movements in both relative and absolute terms.
- The magnitude and orientation of rock stress acting in the hangingwall.

- g) The build-up and nature of pressure generated in the caved material.

Fairly standard instrumentation and measuring techniques were used to quantify these parameters. Three borehole sites were established in 72-43W N8 Strike Gully, 78A-42W N7 Strike Gully and 78A-42W S4 Centre Gully, where a number of holes were drilled into the hangingwall, taking care to maximise core recovery for logging and evaluation purposes. The rock structure was surveyed using a borehole camera, following which wire extensometers were installed in the long holes. CSIR doorstopper strain gauges were used to measure stress in shorter holes drilled for this purpose. A large number of simple closure stations consisting of pegs glued to the hangingwall and footwall were installed in panel faces and strike gullies. Four-point closure and ride monitoring stations were also established at a number of positions within the stopes. Fracture mapping was conducted at intervals in the stope faces. Finally, cave pressure was measured using load cells placed in the back areas of two panels. Figures 41 and 42 show the location of the drilling cubby, the closure stations and the cave pressure measuring sites established in 72-43W Stope and 78A-42W Stope respectively. The various aspects of the instrumentation program are described in more detail in the following sections.

7.2 HANGINGWALL STRUCTURE.

In order to generate a structural picture of typical Vaal Reef hangingwall in the 5A North pseudo-breccia area, observations and measurements were made of discontinuities exposed on the hangingwall surface of the test panels and in boreholes drilled up to 25m vertically into the hangingwall of these panels. A borehole camera was used to make observations in the boreholes, and these observations were later correlated with the logged results of the core extracted from these holes.

The data collected from the mapping traverses, from the borehole observations and from the core log are presented in Appendices 2A, 2B and 2C respectively. The data was interpreted using simple statistical and stereographic projection techniques, and a composite, two-dimensional map of the structure of the hangingwall of these two stopes was compiled. The information used in its compilation was by no means exhaustive, but it nevertheless probably gives a fair representation of typical hangingwall structure in the area. A summary of the data used in the analysis of the orientation and the spacing frequency of discontinuities occurring in the stope hangingwall is presented in Tables 5A and 5B respectively, and a pictorial representation of their calculated probability of occurrence is shown in Figure 43. The structure map, which was used as a guide when generating the computer model described later, is shown in Figure 44. This may be contrasted with the conceptual model of fracturing around deep stoping excavations developed by COMRO shown in Figure 30A.

Bearing in mind the sampling bias and the shortcomings associated with the methods of observing fracture and other discontinuity occurrences, the nature of the exposure surfaces, secondary drilling damage in the case of the core log, and the fact that the hangingwall traverses were conducted close to the face while the borehole observations were made some distance into the back area, the following preliminary conclusions were made in regard to this analysis:-

- a) The overwhelming majority of discontinuities (when viewed in strike section) are either near horizontal (bedding planes) or near vertical (stress fractures).
- b) Discontinuities dipping between 0° and $\pm 20^\circ$ make up 46% of all discontinuities, and

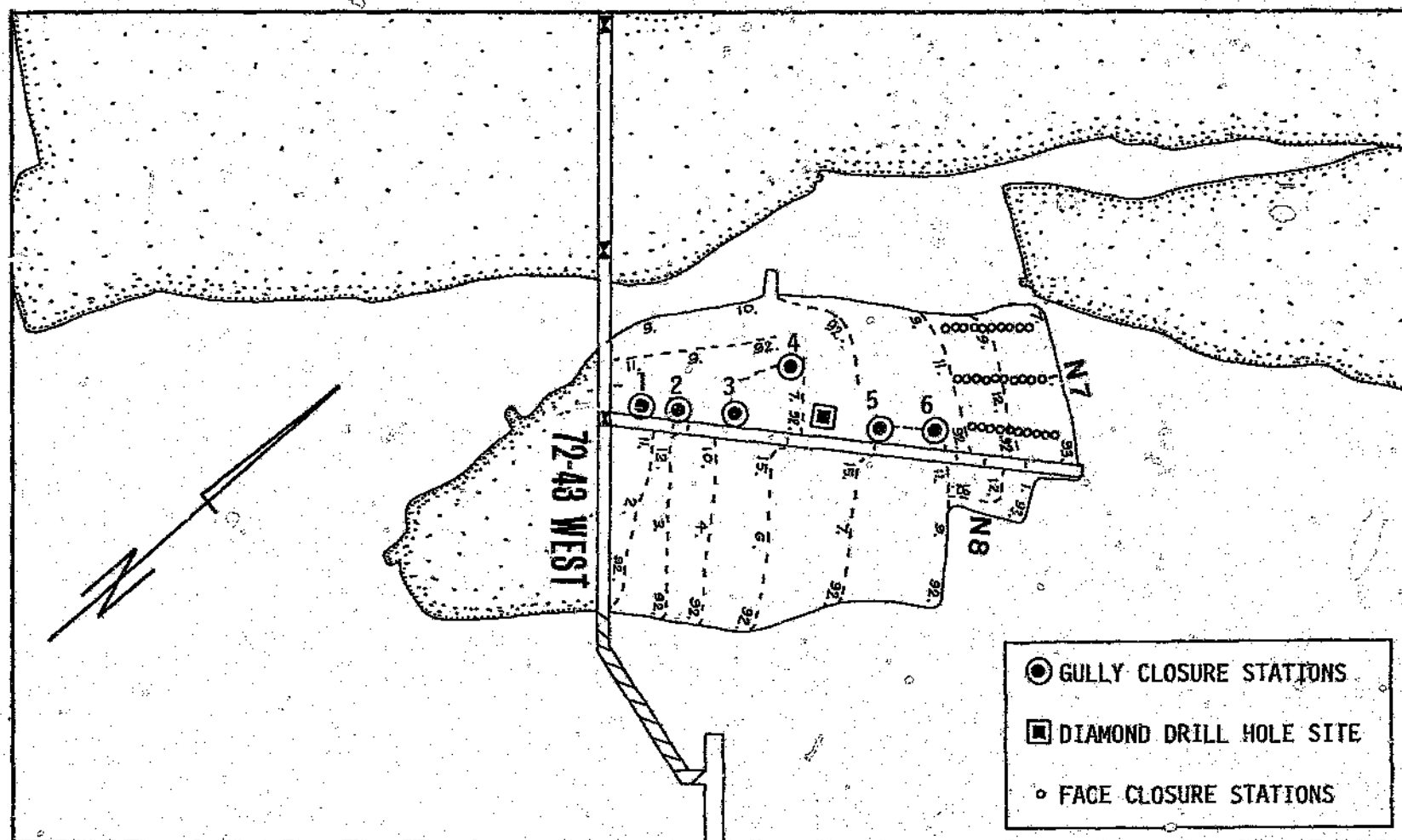


Figure 41 - Plan Showing the Location of Diamond Drill Hole and Instrumentation Sites in 72-43 West Slope.

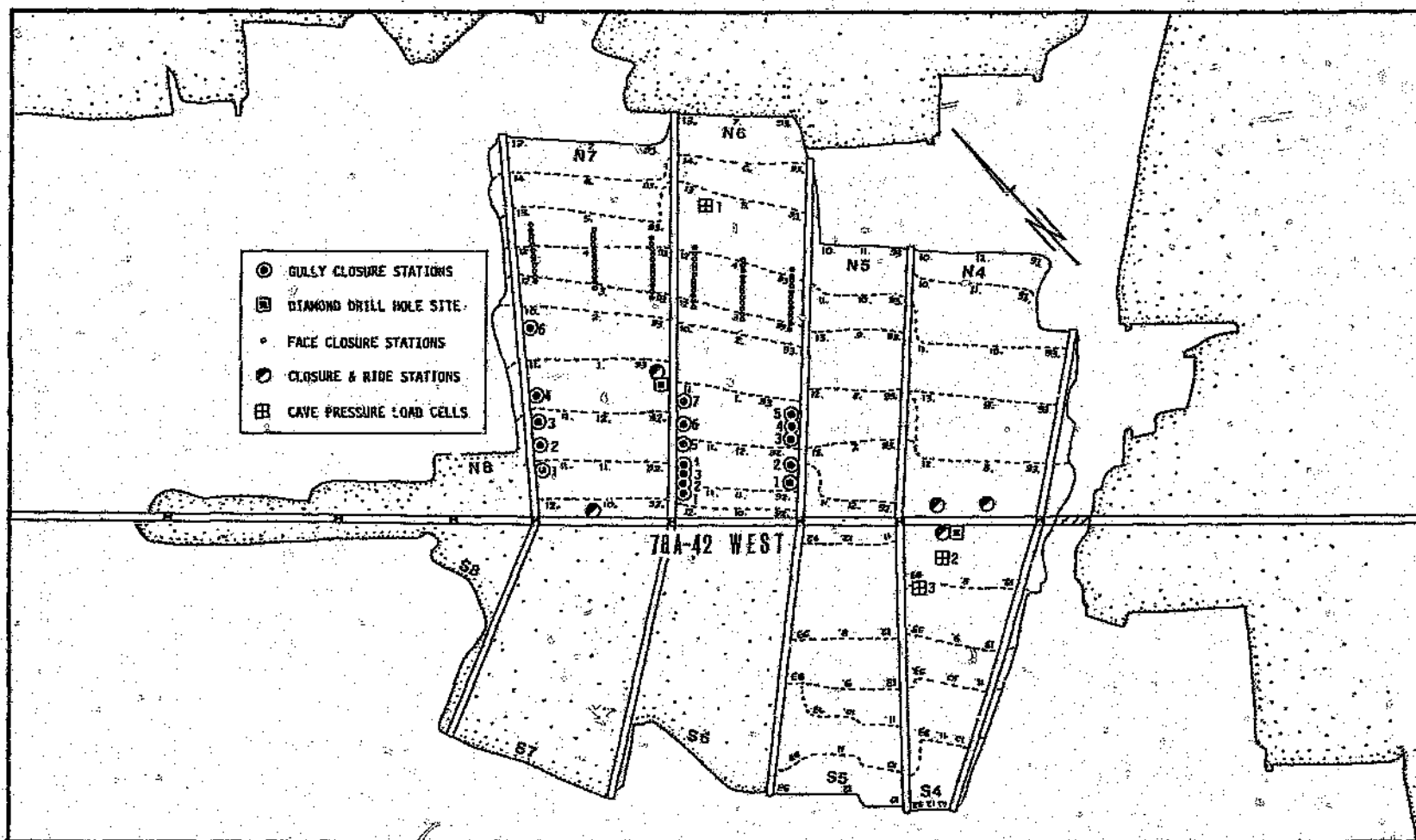


Figure 42 - Plan Showing the Location of Diamond Drill Hole and Instrumentation Sites in 78A-42 West Stope.

TABLE 5A

Analysis of the Orientation of Slope Hangingwall Discontinuities
Based on Strike Traverse, Borehole Camera and Drill Core Observations.

	0° / 20°	20° / 40°	40° / 60°	60° / 80°	80° / 100°	100° / 120°	120° / 140°	140° / 160°	160° / 180°
STRIKE TRAVERSE:									
72-43W N7 Panel	0	0	0	10	6	4	0	0	0
78A-42W N7 Panel	1	0	0	8	6	1	0	0	0
78A-42W N6 Panel	0	1	1	6	9	2	0	0	0
Occurrences	1	1	1	24	21	7	0	0	0
Probability Index	,20	,02	,02	,43	,38	,13	,00	,00	,00
BOREHOLE CAMERA:									
72-43W N8 Gully	40	1	3	11	1	-	-	-	-
78A-42W N7 Gully	32	0	1	5	1	-	-	-	-
78A-42W C/Gully	25	0	0	1	1	-	-	-	-
Occurrences	97	1	4	17	3	-	-	-	-
Probability Index	,80	,01	,03	,14	,02	-	-	-	-
DRILL CORE LOG:									
72-43W N8 Gully	15	1	6	4	0	-	-	-	-
72-43W N8 Gully	34	7	6	10	5	-	-	-	-
78A-42W N7 Gully	31	4	7	6	4	-	-	-	-
Occurrences	80	12	19	20	9	-	-	-	-
Probability Index	,57	,09	,14	,14	,06	-	-	-	-

TABLE 5B

Analysis of the Spacing of Slope Hangingwall Discontinuities
Based on Strike Traverse, Borehole Camera and Drill Core Observations.

	0 /	100 /	200 /	300 /	400 /	500 /	600 /	700 /	800 /	> /
	100 mm	200 mm	300 mm	400 mm	500 mm	600 mm	700 mm	800 mm	900 mm	900 mm
STRIKE TRAVERSE:										
72-43W N7 Panel	5	6	2	1	2	1	0	1	0	1
78A-42W N7 Panel	4	3	4	0	2	0	0	0	0	2
78A-42W N6 Panel	6	3	4	1	1	1	0	0	0	2
Occurrences	15	12	10	2	5	2	0	1	0	5
Probability Index	,29	,22	,19	,04	,10	,04	,00	,00	,00	,10
BOREHOLE CAMERA:										
72-43W N8 Gully	28	10	4	2	2	3	1	1	3	2
78A-42W N7 Gully	12	10	3	4	1	1	0	1	3	4
78A-42W C/Gully	8	7	2	4	2	0	1	0	0	3
Occurrences	48	27	9	10	5	4	2	2	6	9
Probability Index	,40	,22	,07	,08	,04	,03	,02	,02	,05	,07
DRILL CORE LOG:										
72-43W N8 Gully	21	18	1	1	0	0	0	0	0	0
72-43W N8 Gully	41	13	0	0	0	0	0	0	0	0
78A-42W N7 Gully	48	6	2	0	0	0	0	0	0	0
Occurrences	110	37	3	1	0	0	0	0	0	0
Probability Index	,73	,24	,02	,01	,00	,00	,00	,00	,00	,00

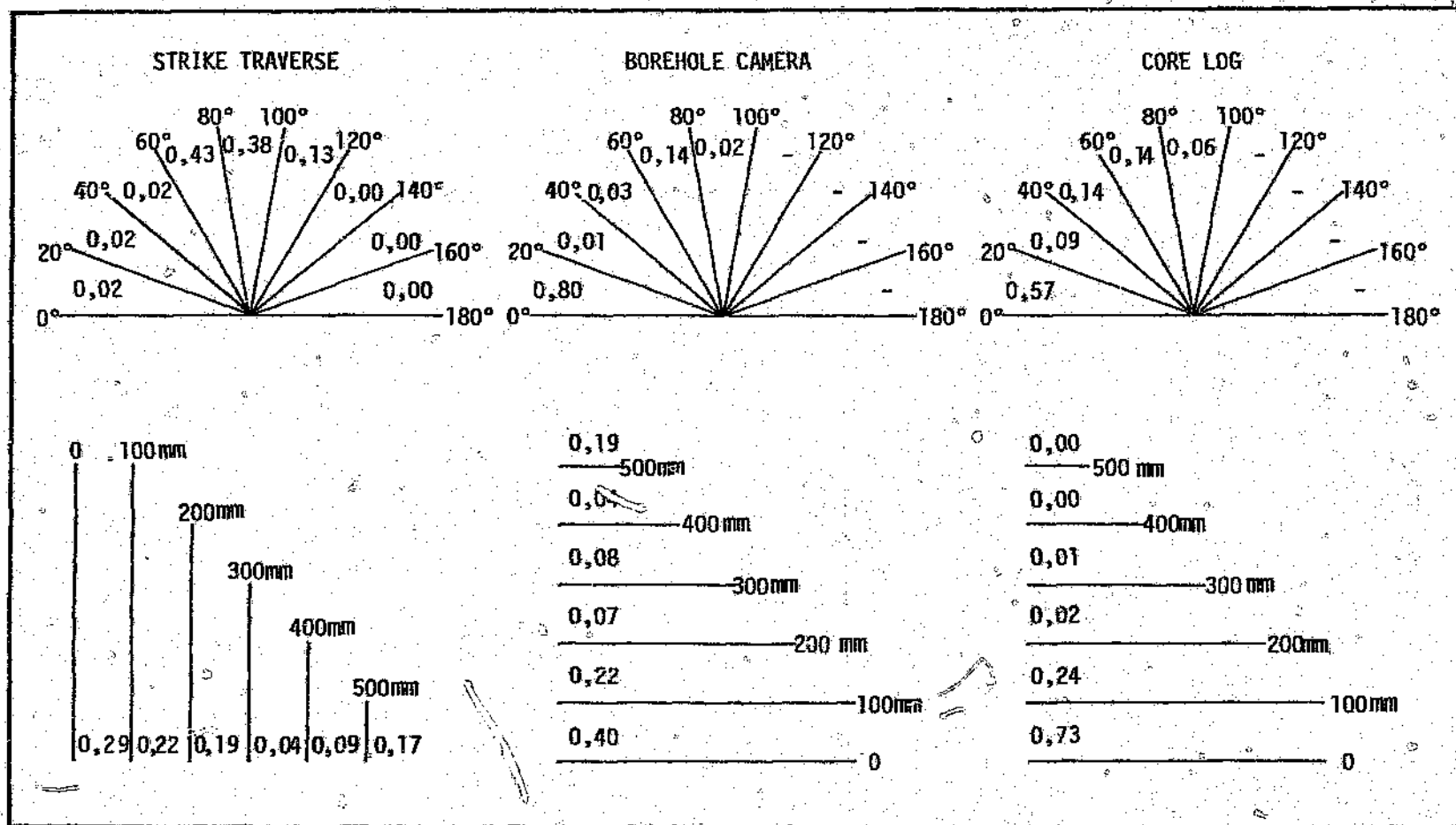


Figure 43 - Pictorial Representation of Discontinuity Spacing Frequency and Orientation Frequency.

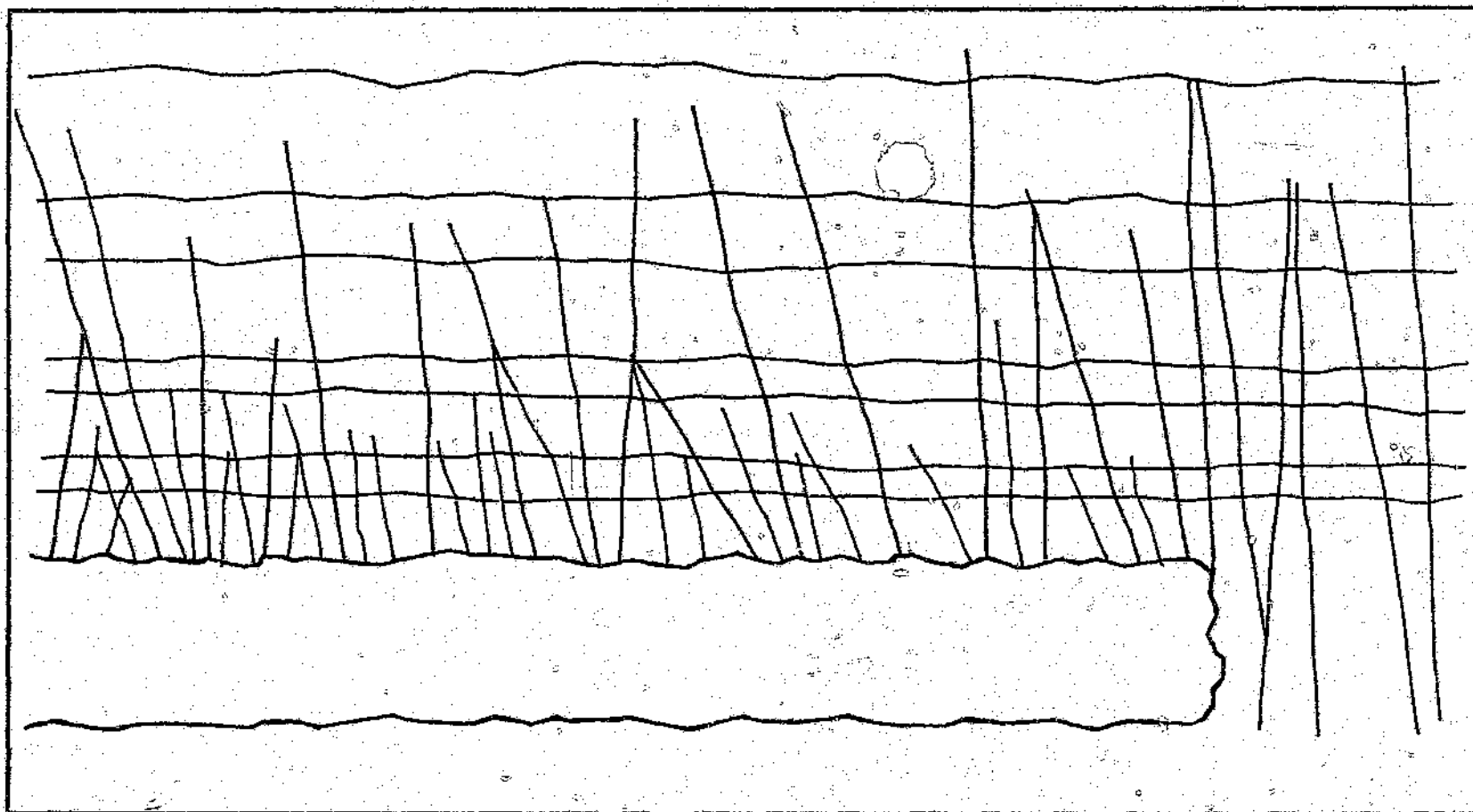


Figure 44 - Typical Structure of Hangingwall in 72-43W and 78A-42W Stopes Compiled from Mapping Results.

discontinuities dipping between 60° and 120° make up 44% - in total accounting for 90% of all discontinuities.

- c) Only 10% of discontinuities dip at between 20° and 60° .
- d) Making up the 44% of discontinuities which may be classified as steeply dipping, 19% dip at between 60° and 80° , 16% dip at between 80° and 100° , and only 9% dip at between 100° and 120° .
- e) Contrary to expectations in a cave mining environment, the majority of hangingwall stress fractures observed in this survey (including those close to the stope excavation) were inclined away from the face rather than towards the face. (Note that planes which are inclined away from the face dip towards the face, and vice versa).
- f) The horizontal strike spacing of discontinuities revealed by hangingwall mapping showed 29% are spaced at less than 100mm apart, 22% are spaced at more than 100mm but less than 200mm apart, 19% at more than 200mm but less than 300mm apart, and 30% are spaced more than 300mm apart. A substantial 17% are spaced more than 500mm apart.
- g) The vertical spacing of discontinuities, reflected with a greater degree of confidence by the results of the borehole camera surveys rather than the borehole core log, showed that 81% are spaced less than 500mm apart and 19% are spaced more than 500mm apart. 62% are spaced less than 200mm apart and 40% are spaced less than 100mm apart.

In this survey the horizontal orientation of discontinuities was measured clockwise where 0° - 180° coincided with the face orientation with zero updip, and the dip of discontinuities was measured clockwise where 0° - 180° coincided with the horizontal plane with zero towards the back area. For the purposes of this evaluation only the first five metres of each borehole core fracture log was considered, while the full extent of each borehole camera survey was included. In the case of the camera survey and the core log it was not possible to determine the horizontal orientation of dipping features with any confidence. Dips were thus categorised in the range zero to one hundred degrees and were conservatively assumed to have an even spread of orientation.

The structure of the hangingwall in the 5A North Area is illustrated in Plate 9 which shows an exposure in the 72-43W S5 Strike Gully. A rare view of back area caving taken elsewhere on the mine is shown in Plate 10. The predominance of horizontal bedding and steeply dipping stress fractures inclined towards the back area is clearly evident. Plate 11 shows a borehole camera being used to conduct a visual examination of the hangingwall structure above 72-43W Stope, while Plate 12 shows a portion of the core recovered from one of the hangingwall drill holes in 78A-42W Stope.

7.3 ROCK QUALITY.

Selected representative samples of the core obtained from boreholes drilled into the hangingwall were subjected to laboratory testing in order to evaluate the quality of the intact hangingwall rock material. The results of the evaluation provided useful guidelines when formulating the input parameters used in the UDEC modelling discussed later.



Plate 9 - Hangingwall Structure In 72-43W S5 Gully.



Plate 10 - A Rare View Of Back Area Caving.

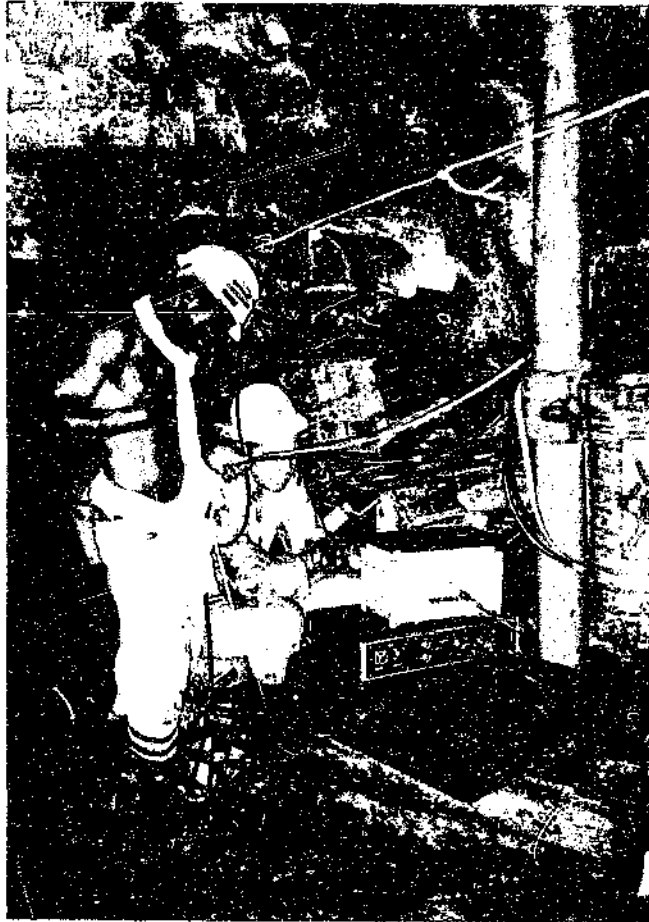


Plate 11 - Examining Hangingwall Structure Using A Borehole Camera.

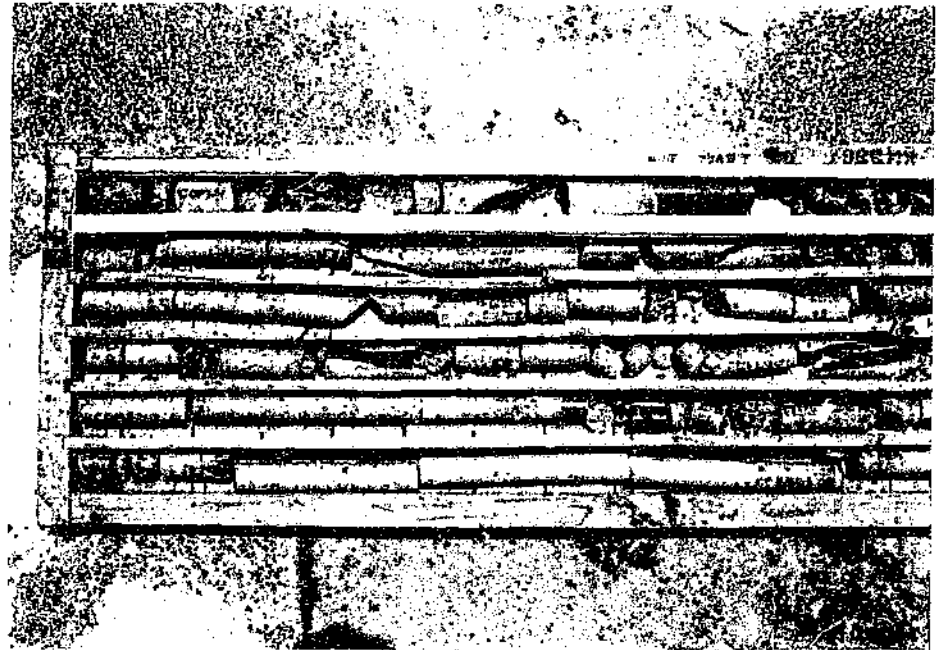


Plate 12 - Portion Of The Core Recovered From One Of The Hangingwall Drill Holes In 78A-42W Stope.

The detailed results of the evaluation carried out on core from a borehole drilled into the hangingwall at the 78A-42W N7 Gully site are presented in Appendix 3. Three sections of core were tested from the lower (1,7m - 4,5m), middle (13,7m - 15,3m) and upper (20,7m - 23,3m) portions of the borehole. The distances are an indication of the location of the sample sections above the Vaal Reef horizon but, because of core loss during drilling, these may not be exact.

Because of time and financial constraints, the core falling between these selected sections has not as yet been tested, but has been stored. The reasoning being that it may be retrieved for testing if the results from the test sections indicate that this is warranted. Visual inspection of the entire core length indicates that it all consists of fairly uniform, pebbly, argillaceous quartzites of the MB4 hangingwall series. The test results do in fact reflect a high degree of uniformity over the 20m distance.

A summary of the test results for the lower, middle and upper groups, which are presented in full in Appendix 3, is given in Table 6. These results indicate that the intact hangingwall rock has a moderate strength range of between 133MPa and 207MPa. The mean uniaxial compressive strength calculated from the triaxial results is approximately 190MPa. The lower group has an average uniaxial compressive test strength of 160MPa (the calculated UCS is 171MPa), the middle group 192MPa (199MPa), and the upper group 162MPa (204MPa). The elastic modulus of the intact rock tested is remarkably consistent, with the averages for all three groups falling between 63GPa and 66GPa. The average Poisson's ratios are 0.17, 0.17 and 0.14, and the calculated tensile strengths are 38MPa, 42MPa and 45MPa respectively.

A shortcoming of the investigation was the failure to quantifiably assess the characteristics of the discontinuity surfaces. A subjective assessment was made however, based upon the drilling record and upon observation of the diamond drill core and other hangingwall exposures. This information was incorporated in the caving models discussed in the next chapter.

7.4 STRATA MOVEMENT.

7.4.1 Closure Stations.

A large number of closure stations were established at the faces and at intervals along the gully edges of all the test panels. The stations consisted of simple epoxy pegs stuck to the stope hangingwall and footwall, between which measurements were regularly recorded using a steel tape. Plate 13 illustrates the measurement of closure in the working area of 78A-42W N8 Panel. Some of these measurements are recorded in Appendices 4A(i) and 4A(ii), and serve to demonstrate that closure in the project stopes considerably exceeded the convergence predicted by elastic theory.

7.4.1.1 Gully Closure Stations.

As an example of the closures recorded in the strike gullies, Appendix 4A(i) shows the results of measurements taken in the 78A-42W N8 Gully (against the top abutment), in the N7 Gully (at approximately centre span), and in the N6 Gully (against the bottom abutment). A total

TABLE 6.

**Summary of the Results of Uniaxial and Triaxial Laboratory Testing
of Hangingwall Borehole Core Recovered from 78A-42W N7 Gully Drilling Site.**

	LOWER GROUP 1,7 - 4,5m	MIDDLE GROUP 13,7 - 15,3m	UPPER GROUP 20,7 - 23,3m
Mean UCS Test Strength (MPa)	160	192	162
Calculated UCS (MPa)	171	199	204
Calculated UTS (MPa)	38	42	45
Mean Elastic Modulus (GPa)	63,9	65,4	65,7
Mean Poisson's Ratio	0,17	0,17	0,14
Mean Angle of Internal Friction	42,0°	43,8°	42,5°
Mean Linear Elastic Limits (MPa)	54 - 164	45 - 173	45 - 183

of 949mm of stope closure was recorded in the centrally located N7 Gully over a period of six months, during which time the adjacent panels were advanced a distance of 66m. Closure in the N8 and N6 Gullies adjacent to the top and bottom (at that time) unmined abutments was considerably less at 388mm and 421mm respectively. As expected, the measurements at all three sites show that by far the largest proportion of closure took place during the early stages of mining while the faces were close by. Thereafter the rate of closure tapered off, but continued even after the faces had reached their limit and were stopped, as a result no doubt of continuing time-dependant settlement and the influence of stoping activities in more remote workings.

7.4.1.2 Face Closure Stations.

Closure stations were established within the working areas of all of the test panels. Because of blasting and scraping operations, it was not possible to monitor closure right at the working face (ahead of the blast barricade) on a continuous basis. Measuring stations were thus established immediately behind the face or barricade row of props (approximately 2m to 3m back from the face) and were monitored until that prop row formed the back row of support (approximately 6m back from the face). Once the row of props had been moved forward it was no longer possible to safely take measurements at that station. Three lines of such face closure monitoring stations were established in each panel; one 5m down from the top of the panel, one approximately in the centre of the panel, and one 5m from the bottom of the panel.

A typical example of the nature and degree of stope closure in the working area is presented in Appendix 4A(ii) which tabulates measurements taken at the top, middle and bottom of 78A-42W N7 Panel. The stoping width measurement taken at the face is provided as a guide only and cannot be used to accurately determine the rate of closure between the face and the first row of props. The progressive measurements taken at the three closure stations are contrasted with the calculated elastic convergence anticipated for a longwall stope at a similar depth, and show that actual closure occurring in the hydraulic prop supported area is at least an order of magnitude greater than that predicted by elastic theory. The measurements also reinforce the argument that hydraulic props should be set with an available minimum of 200mm of travel to avoid damage to both props and hangingwall.

7.4.2 Closure and Ride Stations.

Closure and Ride stations were established at the diamond drill cubby sites and at other selected locations in the centre and strike gullies in 78A-42W Stope. In all instances these stations were established too late to record the magnitude and sense of the large deformations which occur in the rockmass close to the face (as revealed by the S4 Centre Gully extensometer and the N7 Gully closure stations). However, analysis of the data demonstrates that lateral movement does take place continuously throughout the life of the stope.

Data collected over a one month period from one of the closure and ride stations established in the centre gully of 78A-42W Stope is presented in Appendix 4A(iii), and shows the dip ride component associated with closure of some 185mm increasing to a maximum of 22mm and then returning to zero. The strike ride component shows the footwall displace relative to the hangingwall by a maximum of 31mm, returning to settle at a 19mm differential.

These results are clearly not a true reflection of the large dip and strike ride components generally associated with cave mining, manifestations of which are commonly observed as gully closure and support damage. On the other hand, there may be many instances of substantial movement of the hangingwall and footwall in unison, with little or no ride components. More work is required to gain a fuller understanding of the relative movement of hangingwall and footwall at all points in a stoping excavation during its entire life, and to relate this in absolute terms to the deformation taking place in the rockmass as a whole.

7.4.3 Extensometer Stations.

Three wire extensometers were installed into boreholes drilled into the hangingwall at the three diamond drill cubby sites. Histories of the measurements recorded at each site are presented in Appendices 4B(i), 4B(ii) and 4B(iii) respectively. Also included is a tabulation of stope closure recorded at each cubby prior to, during, and, in one case, after the installation of the extensometers, together with pertinent information concerning the time lapse and the relative face positions. Plate 14 shows the extensometer tapes at the 72-43W N8 Gully site being read.

The extensometers installed at the 72-43W N8 and 78A-42W N7 Gully locations were installed long after most of the movement of the immediate hangingwall had taken place. Although substantial stope closure continued to be recorded, extensometer tape measurement readings remained static for the duration of the measuring period, indicating uniform downward movement of at least the first twenty metres of hangingwall at that stage. Other than this observation, no other useful information was forthcoming from these two installations. Plate 15 is a view of the drilling site in 72-43W N8 Gully shortly after the final extensometer readings had been taken. Considerable deformation of the vertical diamond drill mounting bar is apparent. This was not nearly so marked when extensometer measurements commenced five months previously and indicates a considerable degree of stope closure, yet no differential vertical hangingwall movement indicated by the extensometer installed at this site.

The extensometer in 78A-42W S4 Centre Gully on the other hand was installed on April 21, soon after ledging of the No.4 Panels had taken place. After the initial ledging, no stoping activity occurred in these or the immediately adjacent panels for a period of three months. During this time continuous minor differential movement of the hangingwall strata took place as may be seen in Figure 45 which depicts graphically the relative displacements of the five extensometer anchors which were located at intervals from two to eighteen metres into the hangingwall. The first blast on the S4 Panel was taken on July 13 and was accompanied by comparatively large differential strata movements. Prior to this, however, the extensometer registered a downward displacement of the lower ten metres of hangingwall by approximately 200mm. This was in response to the slightly earlier commencement of stoping operations in the adjacent S5 Panel at that time. Unfortunately this sudden activity coincided with a period during which the staff responsible for taking the measurements on a weekly basis were temporarily withdrawn to attend to other, at that time, more pressing responsibilities. As a result, the detail leading up to this emergence of large scale movement was lost.

It is nevertheless apparent that a substantial proportion of the movement of the near-field hangingwall takes place in the immediate vicinity, just ahead of, at, and just behind, the advancing face. Thereafter downward displacement of the hangingwall involves movement of the more remote strata, or is spread over a greater depth of strata.



Plate 13 - Measuring Closure In The Working Area Of 78A-42W N8 Panel.



Plate 14 - Taking Extensometer Measurements In 72-43W N8 Gully.

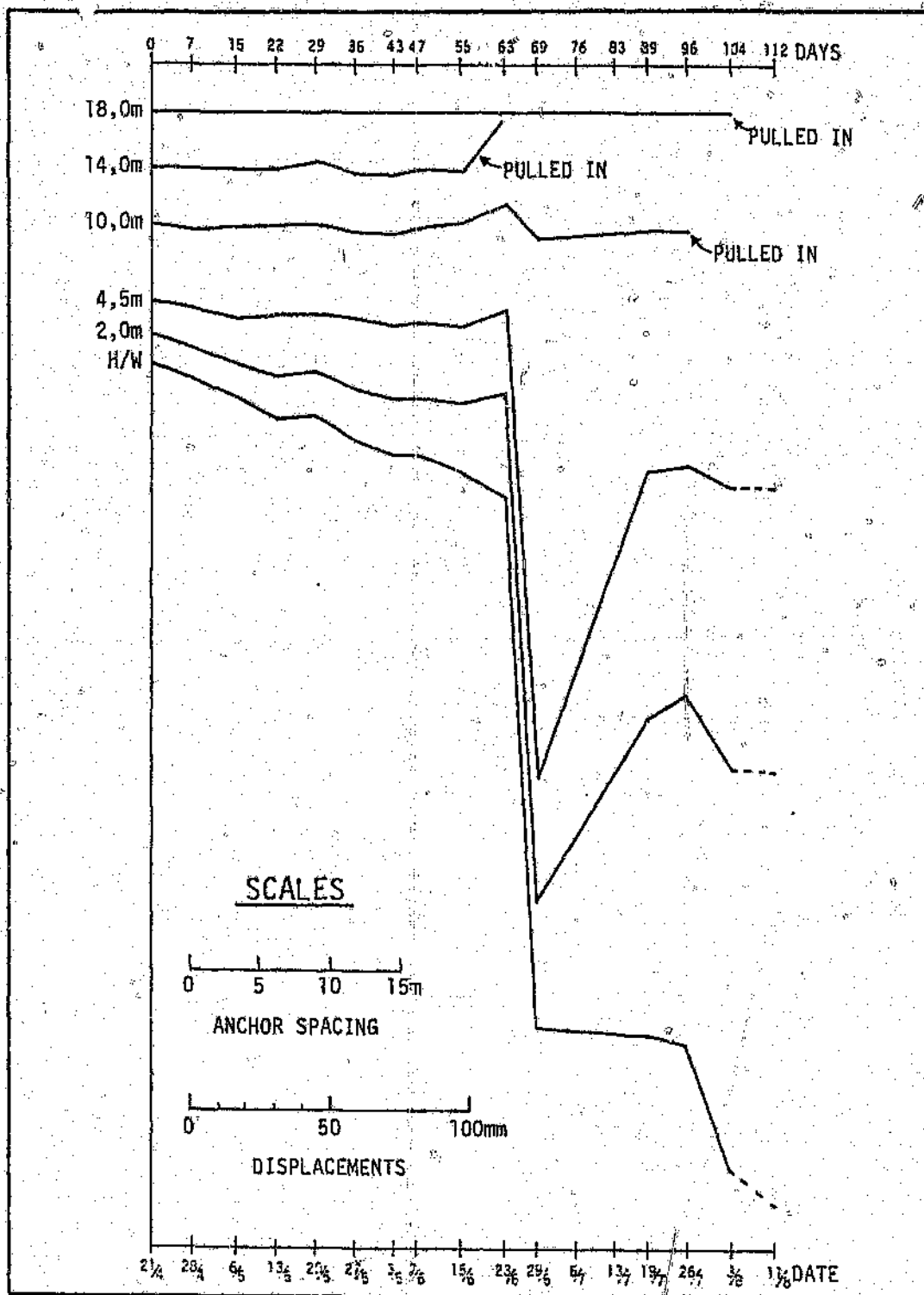


Figure 45 - Graphical Representation of the Relative Vertical Displacement of Extensometer Anchor Points.

Shortly after recording the acceleration in hangingwall displacement, travel on the extensometer tapes started to become exhausted, but, because large, cut-off type dislocations, which would render further extensometer observations meaningless, were noticed in the adjacent stress measurement holes it was decided not to extend the tapes to prolong the life of the extensometer.

Because of the large differential horizontal strata movements associated with stoping operations at Hartebeestfontein Gold Mine, it is difficult to be sure of the interpretation of borehole extensometer data. For example, the conclusion in the case of the two strike gully extensometers that a whole 20m thick mass of hangingwall is moving down as a unit may be wrong. All the tapes could be pinched together a few metres into the hangingwall. Also, the large 200mm downward movement which is thought to have occurred between five and ten metres into the hangingwall of the centre gully site may alternatively be the result of large lateral movement at this level. The observation and interpretation of hangingwall and footwall strata movements would be greatly facilitated by access to a vertical hole of considerably larger diameter than a diamond drill borehole. The development, ahead of the advancing face, of a vertical boxhole is suggested for further evaluation of large magnitude strata deformations associated with, particularly, cave stoping.

7.5 STRESS MEASUREMENT.

The state of horizontal stress, particularly in the immediate stope hangingwall adjacent to the working face, is thought to play a significant role in the stability, and indeed the viability, of cave mining operations. Classical caving theory holds that the resistance offered by the support installed at the face should inhibit lateral strata movement only sufficiently to maintain hangingwall stability at the face without resulting in excessive and damaging horizontal stress. Although this hypothesis has been inferred by interpretation of the physical manifestations of this process, no measurement of actual or absolute stresses acting in rock immediately surrounding stope faces has apparently been successfully achieved in the caving environment.

Considerable effort was expended over a period of about six months during the course of this investigation in trying to determine the state of horizontal stress in the immediate hangingwall of the test stopes. In the region of R50 000 was spent on this part of the project, with only limited success. Of the fifty or so CSIR doorstopper strain cells installed at the three borehole sites, only eight were recovered with core still attached, allowing post-overcoring strain measurements to be performed. The results of these measurements were also not very satisfactory.

As was expected, difficulties were experienced from the start with water loss while drilling the holes, requiring frequent grouting to be carried out. Installing the strain gauges and taking the stress measurements under the prevailing adverse environmental conditions was even more difficult and challenging. The procedure was fraught with pitfalls. Appendix 5C contains a brief description of the method employed, and includes a list of the difficulties encountered during the process. Plate 16 shows core with doorstopper attached being removed from the core barrel after successful overcoring. Plates 17 and 18 show the spoils of some of the successful and unsuccessful stress measurement attempts.

The detailed results of the measurements are presented in Appendix 5A, and are summarised in Appendix 5B and Table 7. The COMRO program "STRAIN" was used to interpret the strain gauge data and to present the results graphically. These illustrations are also included



Plate 15 - Drilling Site After Completion Of The Project Showing Closure.



Plate 16 - Core With Doorstopper Attached Being Removed From Diamond Drill Core Barrel.

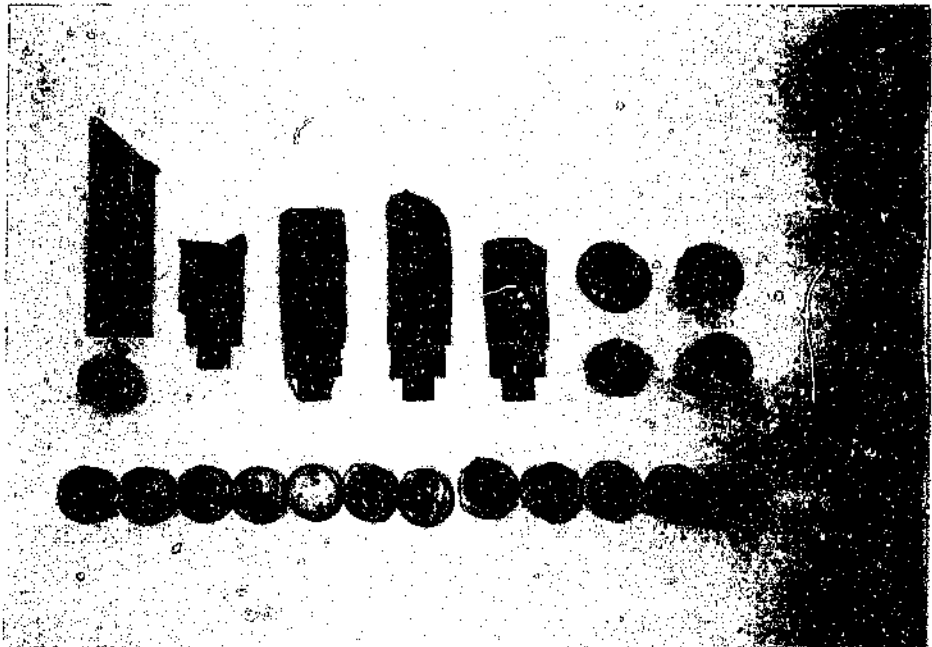


Plate 17 - Doorstoppers And Core From Some Successful And Some Unsuccessful Stress Measurement Attempts.



Plate 18 - Doorstoppers And Core From Some Successful And Some Unsuccessful Stress Measurement Attempts.

TABLE 7.

Summary of the Results of Hangingwall Stress Measurements
in the 5A North Area.

	FACE DIST	HOLE DEPTH	σ_1 (MPa)	σ_3 (MPa)	STRESS ORIEN	REEF STRIKE
72-43W N8 GULLY						
1st	35m	2,30m	+10,3	+ 3,0	085°	
2nd	35m	2,50m	+ 0,4	- 6,8	008°	
Mean	35m	2,40m	+ 5,7	- 1,9	046°	045°
78A-42W N7 GULLY						
1st	20m	4,50m	+36,6	+ 1,0	163°	
2nd	20m	5,00m	- 0,3	- 3,3	078°	
Mean	20m	4,75m	+18,2	- 1,1	120°	045°
78A-42W C/GULLY						
1st	4m	3,80m	+ 3,7	+ 3,0	007°	
2nd	4m	1,55m	+ 6,4	- 1,1	002°	
3rd	4m	2,05m	+ 1,5	- 7,2	104°	
4th	4m	2,30m	+ 1,6	- 0,6	097°	
Mean	4m	2,42m	+ 3,3	- 1,5	052°	045°
Mean	-	-	+ 9,0	- 1,5	070°	045°

NOTES.

Calculation assumptions: Elastic Modulus = 70GPa.
Poisson's Ratio = 0,2.

Maximum stress and reef strike orientations are measured:
Clockwise from North = 000°

in Appendix 5A. The results of the stress measurements reflect such a wide scatter in terms of both magnitude and orientation as to be almost meaningless. For what it may be worth, the average maximum horizontal hangingwall stress is calculated to be 9,0MPa compressive orientated 72° east of north, and the minimum horizontal stress is 1,5MPa tensile. The general strike direction, which is reasonably consistent across the area, is 45° east of north. The orientation of the maximum horizontal stress should, according to theory, correspond with this direction. It may, rather tenuously, be concluded from the results of these measurements that the maximum horizontal stress acting one and a half to five metres into the hangingwall of stopes in the 5A North area has a value of between 1MPa and 6MPa, and the minimum horizontal stress is tensile with a value of between 1MPa and 2MPa. A horizontal stress of 5MPa was applied to the hangingwall in the non-linear caving models.

While the doorstopper method of stress determination may give a good idea of the absolute level of stress at a point in time, future measurements should include vibrating wire strain gauges to record changes in stress as mining activity proceeds.

7.6 CAVE PRESSURE.

Two aspects of the caving process influence stope stability, and thus require investigation. The first is the local effect that caving has on the immediate hangingwall of the adjacent working area. This impacts directly on, not only the behaviour of the hangingwall, but also the magnitude and nature of abutment stresses, the resultant rock failure and fracture intensity, and the stope support requirements. The evaluation of the support requirements has constituted the main thrust of the investigation.

The other aspect which is also not well understood concerns the more global effects of caving. Questions which may be raised in this regard include for example the effect of bulking, and the magnitude and rate of regeneration of stress through the caved material. Fundamental to answering these questions, and to identifying and understanding the global effects of caving, is the determination of the extent and physical nature of the cave, and, perhaps more importantly, the measurement and monitoring of stress generated by the caved material.

As a first step towards discovering whether stress generated by the cave is in fact in excess of the mere dead weight of the collapsed material, a crude attempt was made to simply measure the increase in pressure induced in a 100mm hose filled with water and left in the back-area for the cave to collapse on to. Plate 19 shows a hose fitted with a pressure gauge, and laid on dip behind the back row of props for the full length of the panel, onto which the cave has started to collapse. Unfortunately, despite much effort, it was not possible to seal the hose effectively, and even the "setting" pressure of 500kPa could not be sustained for longer than about 24 hours.

A more technically advanced arrangement of "flat-jack" type load cells was thus designed and fabricated. Each pressure measuring unit consists of five individual load cells shaped and arranged to make up a circular device approximately 1,5m in diameter, with the load cells themselves being protected between two 10mm steel plates. The load cells were calibrated in a laboratory press before being installed underground. An example of the calibration results (in this case for the set installed in 78A-42W N6 Panel) is presented in Appendix 6B. Plate 20 shows this set of instrumentation lying on the No.5 Shaft bank prior to being transported underground, while Plate 21 shows it being installed immediately ahead of the back-area cave

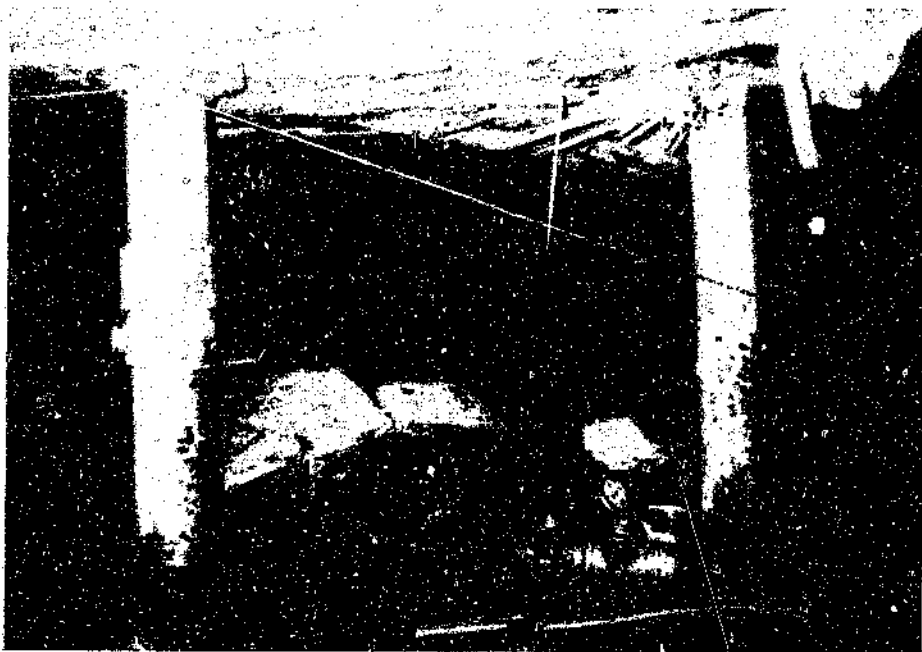


Plate 19 - First Crude Attempt At Measuring Cave Pressure Using A "Boesman" Hose.

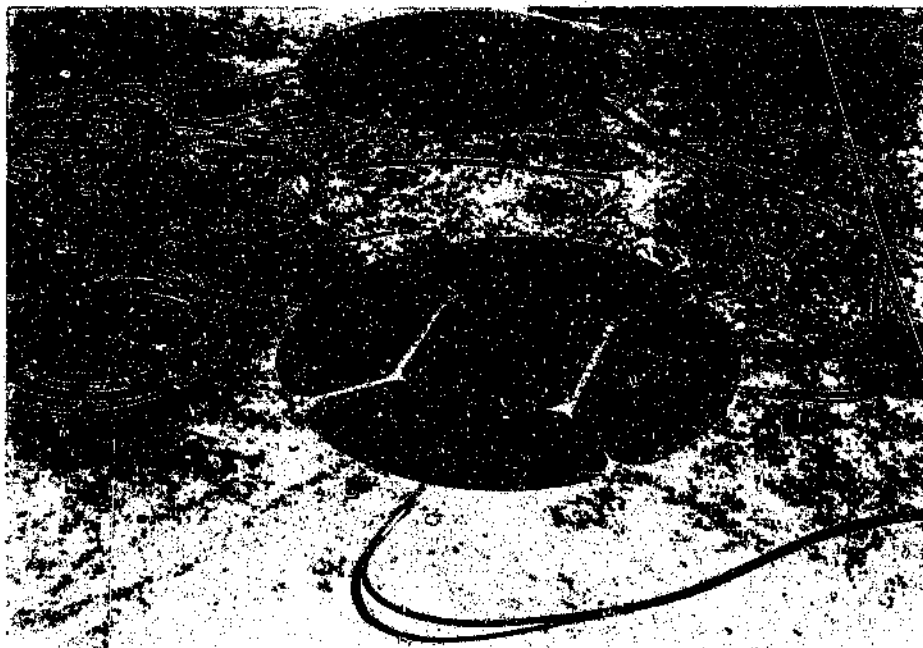


Plate 20 - Cave Pressure Flat-Jack Instrumentation On No.5 Shaft Bank.

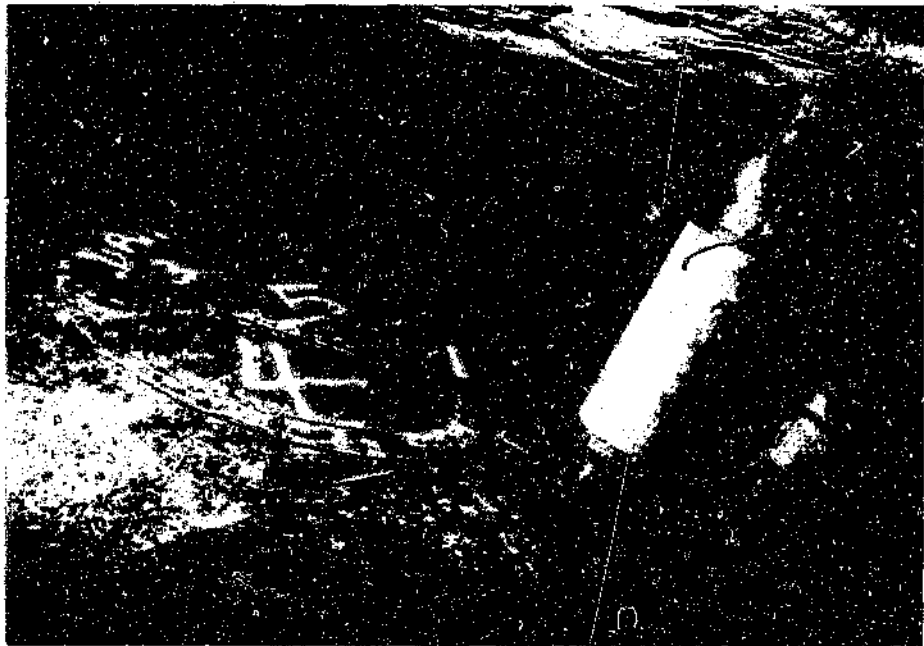


Plate 21 - Flat-Jacks Installed Immediately Ahead Of The Cave In 78A-42W N7 Panel.



Plate 22 - Taking Cave Pressure Readings.

in 78A-42W N6 Panel. Plate 22 shows cave pressure recorded by the load cells being read off gauges located remotely in the adjacent strike gully. Three sets of instrumentation were installed and monitored, one set in 78A-42W N6 Panel and two sets in 78A-42W S4 Panel. The locations of these sites are shown in Figure 42.

A number of the load cells appeared to malfunction and did not register any increase in load during the period of monitoring. Pressures recorded by the remaining load cells did, however, seem to indicate that stress levels generated by the cave are in excess of the mere dead weight of the collapsed material. Within the space of two months after installation, during which time the faces had advanced by some 40m, pressures of 1000kPa and more were being registered. Taking the bearing area of the protection plates in relation to the surface area of the load cells into account, and referring to the load cell calibration graphs, the recorded pressures were interpreted to reflect stress levels in excess of 1MPa generated by the cave, the equivalent of about 40m of dead-weight hangingwall load. The pressures recorded by a set of load cell instrumentation installed in 78A-42W S4 Panel are presented in graphical form in Figure 46. Figure 46A shows the increase in pressure as a function of time, while Figure 46B shows this as a function of face advance. A full record of the measurements is given in Appendices 6B(i), 6B(ii) and 6B(iii).

It would seem reasonable to assume from these preliminary measurements that moderate stresses are regenerated in bulked cave material during the course of mining. If this is in fact the case, it may be possible that caving results in regional stress reductions, similar to, but not to the same extent, as are produced by backfilling. It is, however, too soon to draw firm conclusions in this regard, and more work is planned to, firstly, verify that the pressures being measured are indeed an accurate reflection of the general distribution of stress in caved areas, and, secondly, determine the influence of stress regeneration on local and regional stope stability.

7.7 INSTRUMENTATION BOXHOLE.

Consideration was given during the early stages of formulating instrumentation strategies for this project to the development of a vertical boxhole ahead of an advancing caving stope face in the pseudo-breccia area. The monitoring of stress changes and rock and strata deformation in such an excavation would provide the opportunity of compiling a comprehensive, complete and uninterrupted record of the effects of stoping on the rockmass ahead, at, and behind the stope face. By relating rock movements to some remote base, it would also be possible to estimate with a fair degree of accuracy the deformation of strata surrounding stope excavations in absolute terms.

Such access to strata above and below the reef horizon would be similar to that provided by a shaft or orepass during shaft pillar extraction operations, or by a raise boxhole in conventional mining situations. However, in both these instances the monitoring line would generally be situated on the neutral plane from which stoping would proceed away, and would thus not provide a full picture of rock deformation.

Although this proposal was, for financial and other considerations, not put into practice during the course of this project work, it does nevertheless represent a viable option for future work in the analysis of rock deformation associated with cave stoping operations. Figure 47 is a schematic illustration of this proposal.

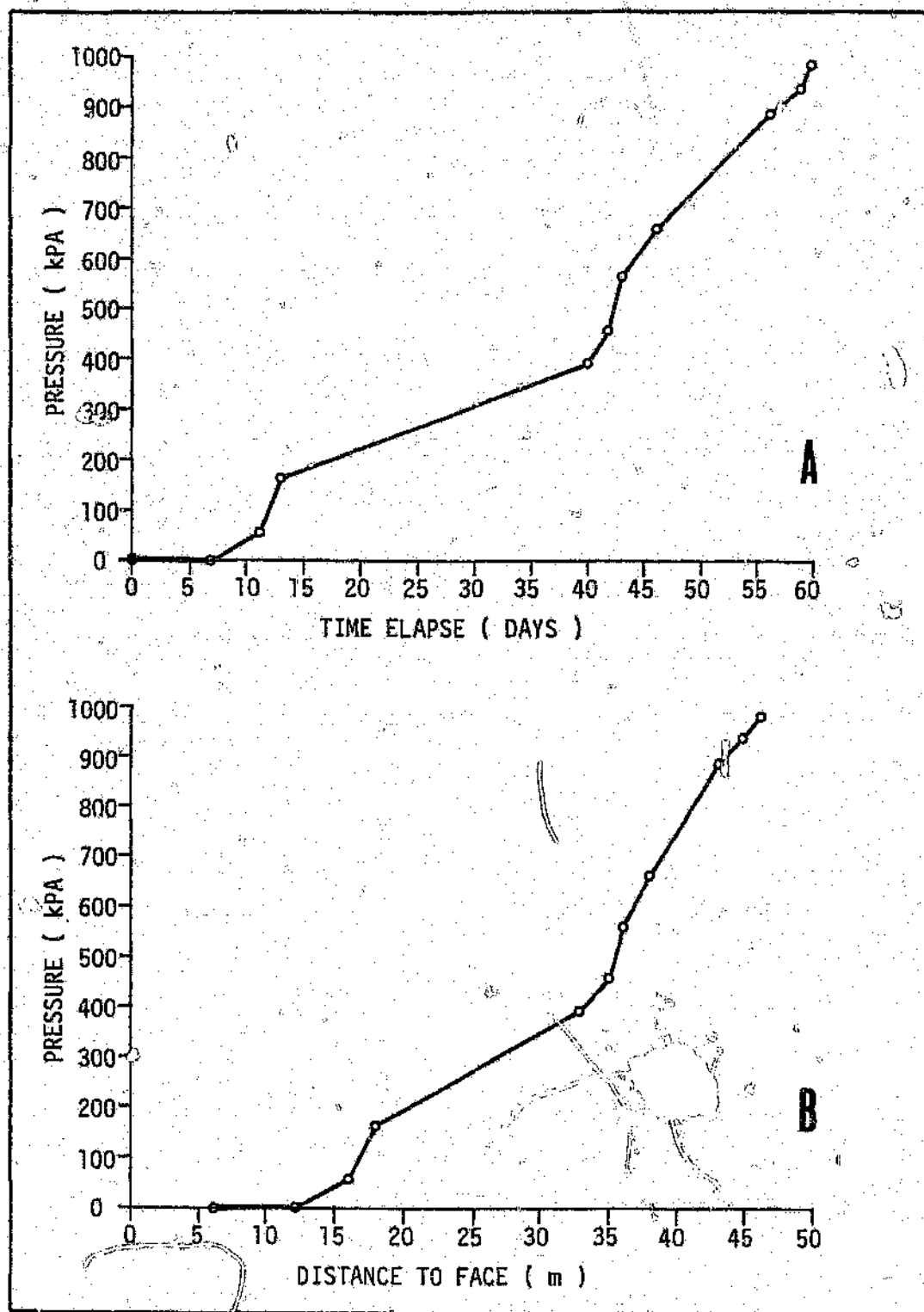


Figure 46 - Graphs Showing The Increase in Cave Pressure in S4 Panel as a Function of Time and Face Advance.

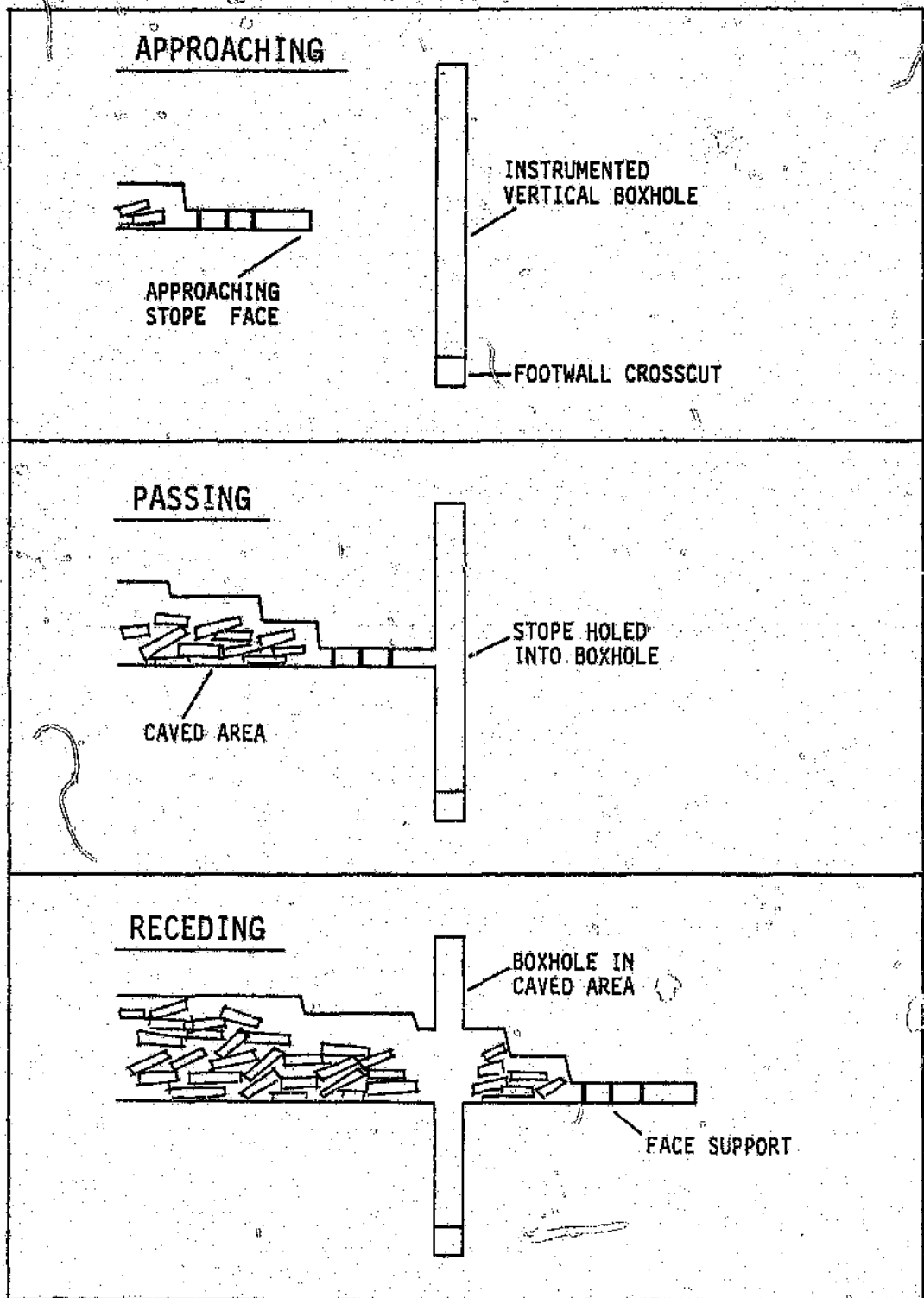


Figure 47 - Schematic of a Proposed Vertical Boxhole to Monitor Rock Deformation Around a Stope.

8. COMPUTER MODELLING.

8.1 INELASTIC BEHAVIOUR.

Quantitative mathematical modelling of the behaviour of rock surrounding deep, tabular mining excavations has, until fairly recently, been conducted almost entirely with the aid of algorithms based on the theory and equations of linear elasticity. This approach has met with considerable success, and, after some thirty years of development, computer programs employing elastic theory still enjoy widespread and almost routine application throughout the industry. Elastic models are naturally subject to the basic assumptions of the theory, i.e. that the material being modelled behaves elastically, and is continuous, homogeneous and isotropic. The behaviour and properties of the far field rockmass surrounding excavations at depth does, by and large, approximate that of a truly elastic material. It has, however, long been recognised that these simplifications impose serious limitations when considering the behaviour of rock in close proximity to underground excavations.

Boundary element codes, which all make use of elastic theory, are unable to accurately predict or model the large displacements and deformations associated with discontinuous and fractured rock immediately surrounding excavations at depth. Although most finite element codes incorporate non-linear, inelastic behaviour, few have discrete element capabilities. And yet it is precisely this type of behaviour with which mining engineers most often have to deal. Clearly, the ramifications and effects of cave mining can also not be realistically modelled on a local scale using elastic theory. For this reason, demonstrations of the mechanics and physics of cave mining have, to date, been restricted to representation by physical or schematic models, rather than by mathematical models.

Developments in the field of non-linear, numerical modelling have, however, led to the comparatively recent commercial availability of a number of computer programs capable of representing the behaviour of materials which do not obey linear stress-strain laws. Two examples of these types of program, FLAC⁽³⁰⁾ and UDEC⁽²⁵⁾, which were developed by the ITASCA Consulting Group in Minneapolis are described as follows:-

- a) FLAC (Fast Lagrangian Analysis of Continua) is a two-dimensional, explicit, finite difference code which simulates the behaviour of materials which may undergo plastic flow when their yield limit is reached. Materials are represented by zones or elements which form a grid that may be adjusted to fit the shape of the material to be modelled. Each element follows a prescribed linear or non-linear stress-strain law in response to applied forces and boundary restraints. The explicit nature of the solution allows any type of non-linear stress-strain law to be followed, and the calculation scheme is thus well suited to the modelling of large distortions.
- b) UDEC (Universal Distinct Element Code) is a two-dimensional, distinct element code which simulates the response of discontinuous media subjected to either static or dynamic loading. The discontinuous medium is represented as an assemblage of discrete, rigid or deformable blocks, with block borders being treated as surfaces upon which boundary conditions act. Large displacements may occur along discontinuities and the rotation of blocks is allowed. The relative motion of blocks is governed by linear or non-linear force-displacement relations for movement of discontinuity surfaces in both the normal and shear directions. Deformable blocks are sub-divided into a mesh of finite difference elements, and each element responds

according to the prescribed linear or non-linear stress-strain law. Several material behaviour models are built into the program, both for the intact blocks and for the discontinuities. The calculation scheme is well suited to the modelling of large movements and deformations of a blocky system.

In order to gain a better understanding of cave mining processes, the UDEC program was used to model some very simple, two-dimensional cave stoping geometries. Input data was drawn and adapted from the results of underground observation and measurement described elsewhere in this report.

The objective was to determine the sensitivity of caving to factors such as rock structure, rock discontinuity characteristics, support density and support spans. It was also hoped to quantify and gain an insight into the effect of caving on abutment stresses, hangingwall stability and support performance, and also to examine aspects such as the height of the cave, back area stress re-generation, and so on. The UDEC program makes it possible to model a large variety of material properties, discontinuity characteristics and structural options. It was, however, necessary to restrict the complexity of models so as to avoid very considerable run times, and to allow manageable sensitivity analyses to be performed.

A further more serious limitation encountered was the inability of the model to simulate the process of fracture development ahead of the face, and thus also the development of horizontal stresses which play such a crucial role in hangingwall stability at depth. It was thus necessary to treat this aspect as yet another variable, and to estimate these stresses and apply them to the model externally.

In essence the models were intended to create a simple, rather than a complex, imitation of the reality being modelled. Information which was deemed unnecessary or unimportant to an understanding of the situation was excluded from the models. On the other hand, the importance of including all aspects which may have had an essential bearing on the results was recognised. The statement by Starfield, Smith and Bleloch⁽⁶⁷⁾ to the effect that "provided the quality and quantity of available data is adequate, and there is sufficient understanding of the problem, models can be built, validated and used with conviction" was adopted as a guideline.

8.2 INTUITIVE CAVING MODELS.

In attempting to identify the underlying principles of cave mining in tabular stopes, and to determine why caving should work in some situations and not in others, three simple "intuitive" caving scenarios were modeled. The reasoning which contributed to the design of these models was discussed in some detail earlier in this report. Briefly, the models visualised a two-dimensional representation of a single 1m thick hangingwall beam situated some 2000m below surface which is supported at each end by the stope face abutments and by a 3m wide zone of rigid support installed 4m from the face, as would be the case in a typical caving stope.

The first scenario attempted to model a rigid, "back-break" situation which would arise when the hangingwall beam is intact and unfractured. This is fairly typical of shallow, competent rock conditions but is rarely seen at depth because of stress fracturing. The second and third scenarios attempted to model the more typical fractured rock conditions pertaining at depth. These models included incipient vertically discontinuities, simulating the effect of stress

fracturing, which were mobilised as stoping progressed towards them. In order to simulate pseudo-plastic rock "flow" and to compare this with "rigid" rock behaviour, friction on the horizontal planes forming the beam was set high in the second scenario and low in the third scenario. Thus the second scenario models the situation in which macro plastic flow does not occur and attempts to demonstrate that support is of necessity required in the back area to maintain horizontal stress and stability in the hangingwall at the stope face, while the third scenario attempts to show that if pseudo-plastic flow occurs, the hangingwall in the face area remains stable despite caving in the back area.

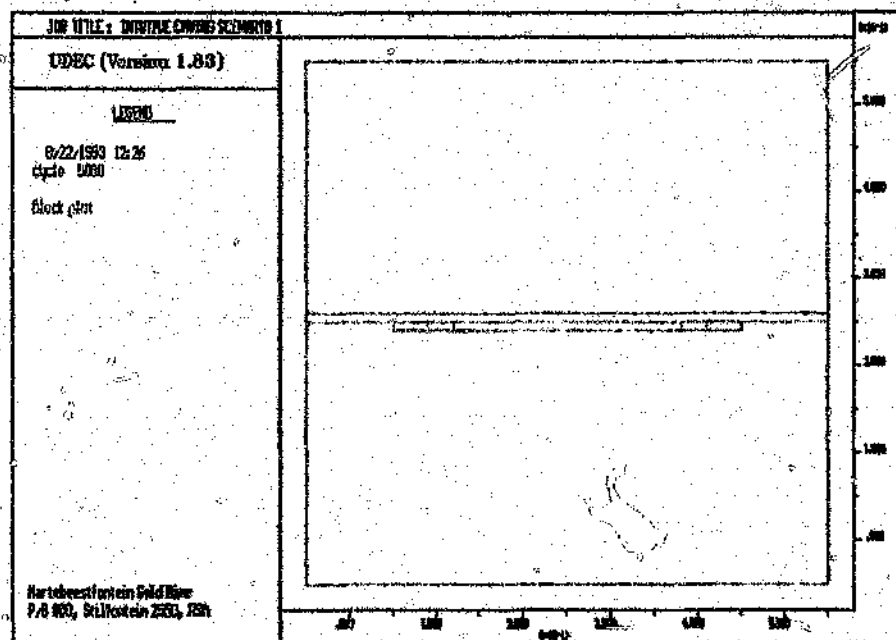
The models also attempted to represent in a simple fashion two of the three aspects which recur in the literature as being pre-requisites for cave mining: a laminated and fractured rock structure and a high degree of lubrication between laminations. Representation of the third requisite, a continual re-generation of horizontal stress in the hangingwall and the associated displacement of strata towards the back area, was attempted by assigning high dilation angles to the vertical fractures. This was found not to be an altogether satisfactory representation because the horizontal stress generated in the model is as a result of differential vertical movement of hangingwall slices falling behind the advancing face rather than as a result of the fracture generation processes occurring ahead of the face as happens in practice. Additional appropriate horizontal stresses were thus artificially applied (as opposed to being naturally generated by the model) at the ends of the beam.

Although the non-linear modelling described here represents a very significant advance over previous elastic and other models of the cave mining process, it nevertheless falls far short of being a true and comprehensive reflection of the complexities which exist in the real underground situation. Fairly substantial limitations exist in the ability of the current models to mimic the very important effects of the fracture processes ahead of the face for example. In addition, a great deal more work is required to explore inelastic rock deformation and its relationship with installed support in greater detail. The work described below does, however, give an indication of the enormous potential and power embodied in non-linear modelling programs such as UDEC, and of the possibilities for future modelling.

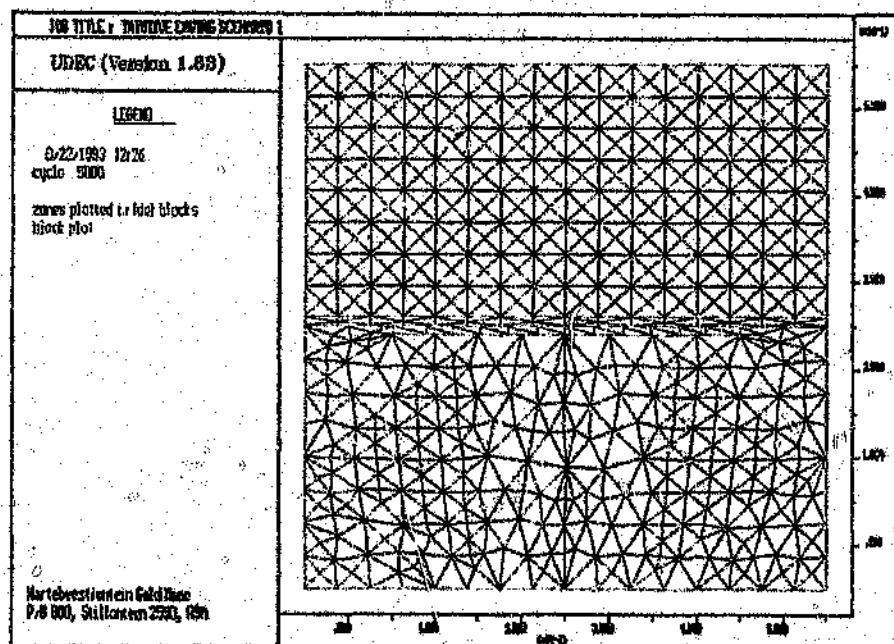
8.2.1 Intact Hangingwall Model.

The input data for the Intact Hangingwall Model (Intuitive Caving: Scenario One), is presented in Appendix 7A(i). A horizontal stoping slot with a span of 40m and a width of 1m was created in the centre of a 60m x 60m UDEC window. A bedding plane was created 1m above the stope and assigned zero cohesion and a 30° angle of friction. The hangingwall beam and the surrounding intact rock were assigned a bulk modulus of 20GPa, a shear modulus of 18GPa and a density of 2700kg/m³. The beam was unfractured. A 3m wide zone of support was modelled 4m behind each face simulating three rows of props spaced 1.5m apart on strike. The back area behind these zones remained unsupported as is typical for caving. The block was made fully deformable so as to model elastic deformation in the continuous rockmass surrounding the stoping excavation. Plot 1A shows the geometry of the intuitive intact hangingwall model, and Plot 1B shows the model blocks sub-divided into finite difference zones. The footwall zones are less regular than those of the hangingwall because of the intrusion of the stope into the footwall block.

A vertical stress of 50MPa and a horizontal stress of 25MPa were applied to the model. Although there is some evidence to indicate that the Poisson's ratio and horizontal stress may be somewhat higher in some areas of the Hartebeestfontein property (Kersten and Greeff⁽⁷⁾),



Plot 1A - The Geometry Of The Intact Hangingwall Model.



Plot 1B - The Model Blocks Sub-Divided Into Finite Difference Zones.

these values are fairly typical of virgin stresses measured in the Klerksdorp goldfield at a depth of 2000m. The model was brought to equilibrium by allowing it to cycle through 5000 steps before the stoping slot was removed.

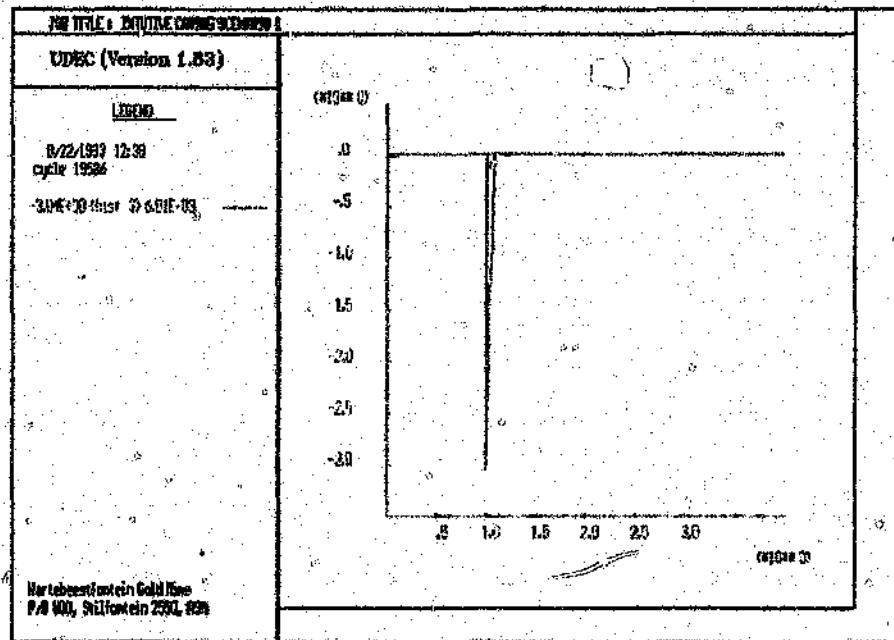
Support zones in the intuitive caving models were represented by blocks of intact material which were assigned appropriate material properties after initial equilibrium. It was necessary to reduce the equilibrium stress condition in the support blocks to prevent elastic rebound when simulated mining was done. In the more complex models described later, where both face advance and support "rolling" were simulated, the three rows of hydraulic props were individually represented by support elements programmed to perform according to specified load-compression characteristics. These elements were applied, removed and reset so as to more realistically reflect the process of releasing and re-installing prop support underground.

Once the model had been set up as described above, it was allowed to cycle through a series of calculation iterations (called time-steps). This calculation process was interrupted at intervals equivalent to half a second of real time in order to save the full state of the model at that particular moment and to thus allow the progress of rock displacement and velocity to be analyzed in some detail. The actual calculation time for each half second of real time was approximately three and a half minutes and required about 2500 calculation time-steps for this particular model. The calculation time taken to model the three seconds required for the model to once more come to equilibrium was thus slightly more than twenty minutes. These iteration versus time relationships vary according to the complexity of the model. Scenario 2 and Scenario 3 which include stress fractures took about eight hours to come to equilibrium, while the more complex "dynamic" caving model which will be described later required one hundred and fifty hours to run.

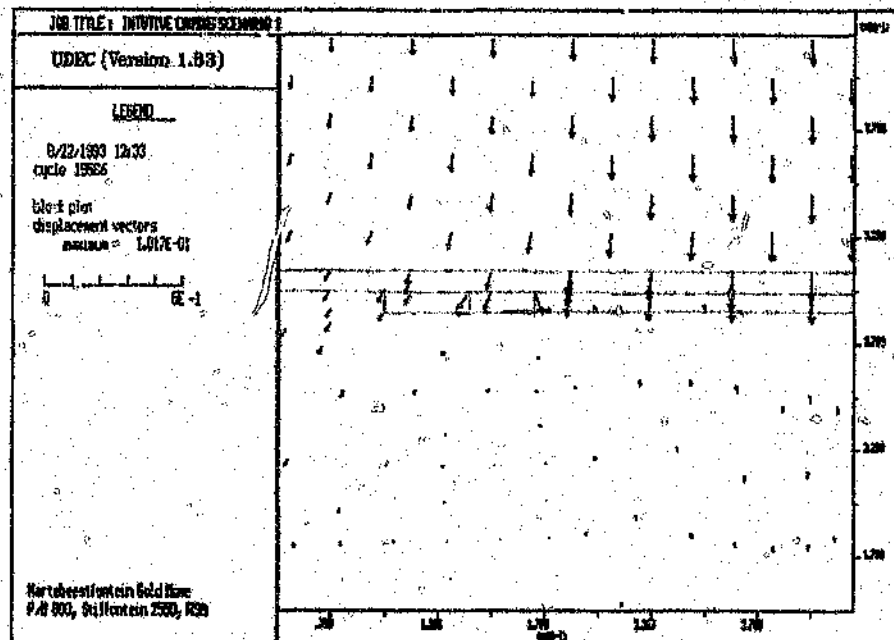
Because only a small number of discontinuities were involved in this unfractured model, deformations were largely elastic and equilibrium was achieved rapidly after mining had taken place. For example, taking a history of the vertical velocity of a point situated in the hangingwall beam at the mid-span showed that downward displacement, followed by equilibrium, took place almost immediately after excavation of the slot. This is illustrated in Plot 2. Once equilibrium was reached only micro velocities in random directions were observed. Contrary to what was expected, hangingwall displacements were not a mirror image of footwall displacements in respect of both magnitude and direction, even though no separation was observed at the bedding plane. While the support influenced the direction of displacements slightly, its effect on the amount of displacement was minimal. Plot 3 illustrates the displacement vectors involved in bringing the model to equilibrium.

In the real world, it would intuitively be expected that, for conditions similar to those imposed in this intact hangingwall model, the hangingwall beam would fail at some critical span by shearing off at the stope face abutments, in the process crushing whatever support was installed at the face. An examination of the contours of shear stress generated by the model, and shown in Plots 4A and 4B, does indeed show lobes of high shear stress over and behind the face abutments and ahead of the support zones. Although the UDEC model is not capable of simulating the failure of intact blocks, the application of a suitable failure criterion would in all probability indicate that the model does predict the expected "back-break" type beam failure commonly experienced in shallow and intermediate stoping environments where the hangingwall is massive and competent.

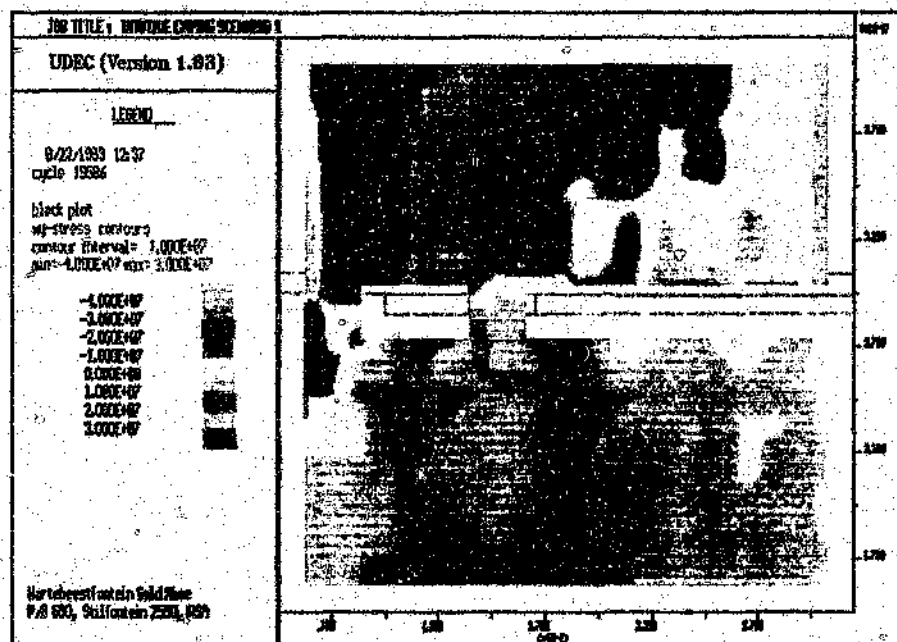
Pearson^(14,15) referred to this mode of failure when describing the behaviour of the Leader Quartzite beam above a Khaki Shale hangingwall at President Steyn Gold Mine. In part he



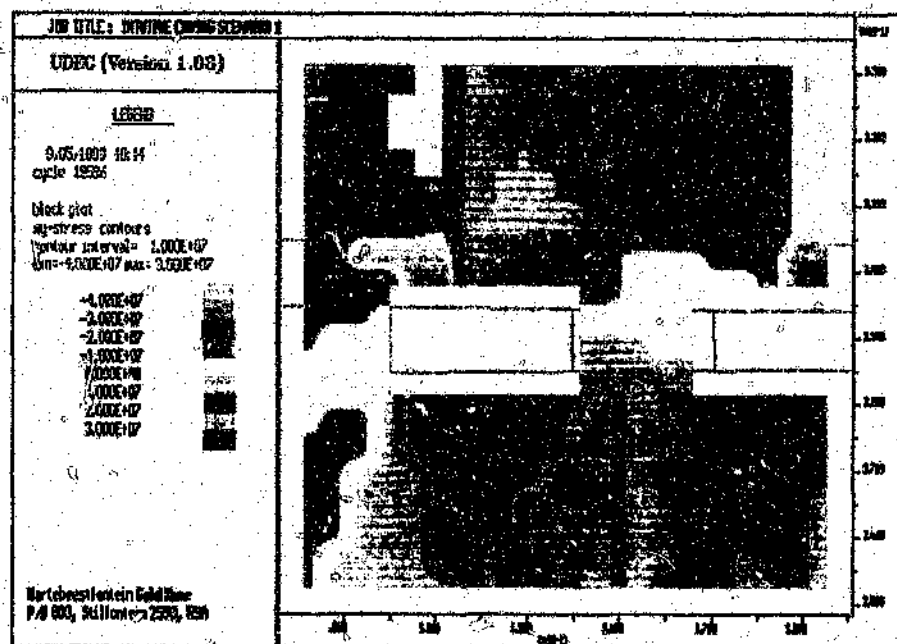
Plot 2 - History Of The Vertical Velocity Of A Point At The Mid-Span Of The Hangingwall Beam.



Plot 3 - Displacement Vectors For The Intact Hangingwall.



Plot 4A - Contours Of Shear Stress Acting In The Intact Hangingwall Model.



Plot 4B - Detail Of Shear Stress At The Face Of The Intact Hangingwall Model.

ascribed the failure of an attempt to introduce stope caving at that mine during 1962 to concern regarding the violent and unpredictable nature hangingwall failure under such conditions. The non-linear model thus reinforces the contention that the nature of the hangingwall rockmass is important in dictating the viability of the cave mining method.

8.2.2 Rigid Hangingwall Model.

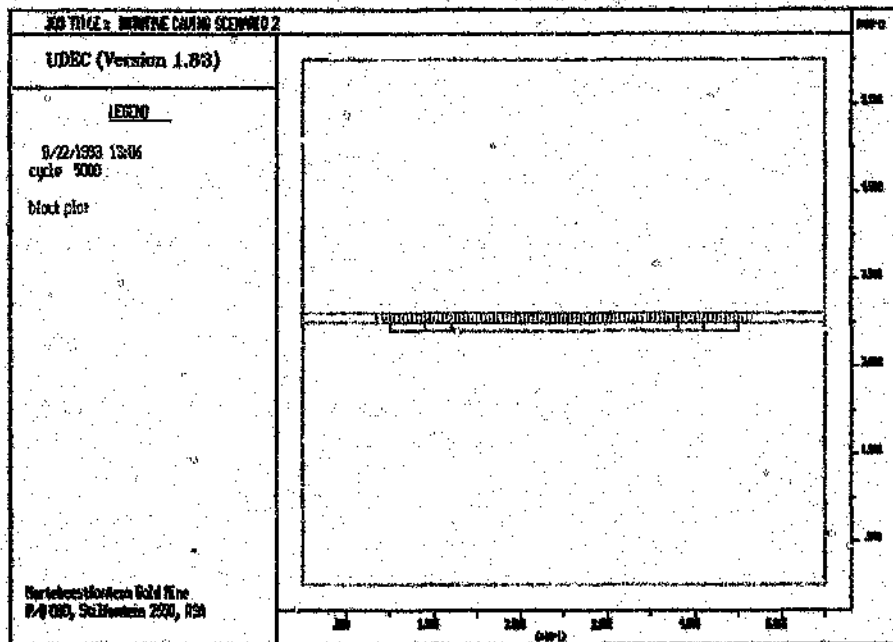
The input data for the Rigid Hangingwall Model (Intuitive Caving; Scenario Two), is presented in Appendix 7A(ii). This second model duplicates the first, with the exception that vertical discontinuities, spaced 0.5m apart and extending 2m ahead of the faces, represent stress induced fractures which break the continuity of the hangingwall beam. Plot 5A shows the geometry of this model.

A further principle of the model is that friction on bedding planes is high, restricting lateral movement of strata towards the stoping excavation. Horizontal forces generated in the hangingwall by dilatatory fracture processes ahead of the faces thus depend upon the integrity of the total hangingwall beam for their continued existence. Relaxation anywhere in the beam will result in loss of restraining force along its entire length. Thus a collapse in the back area will result in loss of horizontal clamping force over the face area, increasing the risk of falls of ground at the face. Plot 5B shows the rigid hangingwall model after failure of the beam in the back area and at the faces.

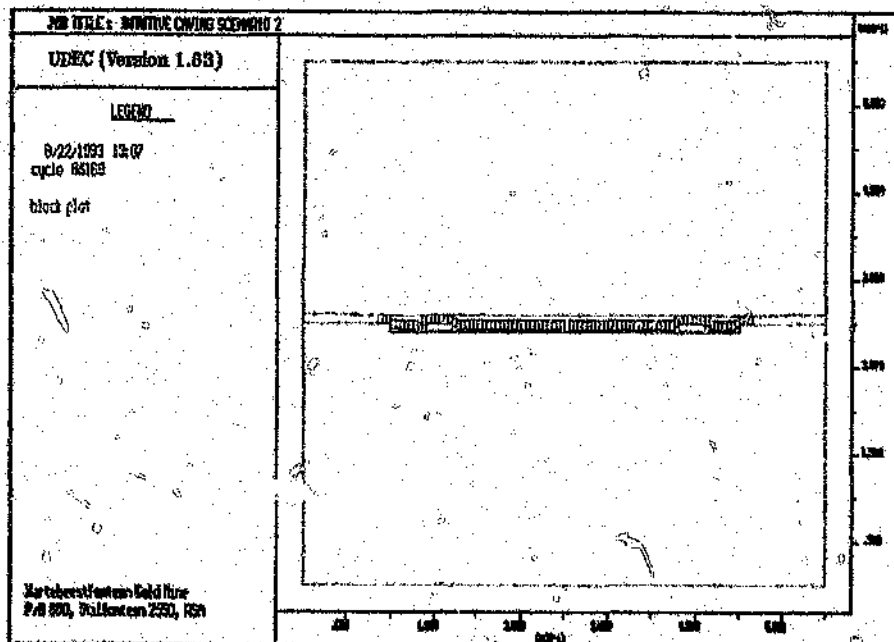
In order to represent this idealised scenario, the friction angle on the bedding planes was set at 90° and the dilation angle on the vertical fractures at zero. A horizontal force of 5MN, similar to that used in the pseudo-plastic model, was applied to the ends of the bedding plane, acting towards the mined-out area. Because the high friction angle restricted horizontal movement, the horizontal force had little effect on horizontal hangingwall displacements as can be seen in Plot 6. Vertical displacements were as expected, with the area between the support zone and the face displaying a more erratic pattern, i.e. the displacements were not as regular as was the case in the back area. This presumably leads to greater stability of the hangingwall in the face area. The large in-elastic displacements representing the falling blocks completely mask the elastic displacements. A plot of shear displacements on the discontinuities (Plot 7) indicates shear movement taking place on the vertical fractures but none on the bedding planes, confirming the absence of horizontal movement.

Horizontal force acting in the beam was quickly lost as hangingwall blocks began to fall, once again confirming the effect that the high friction on bedding planes had in preventing the regeneration of horizontal clamping force. This is illustrated in Plot 9 which shows the horizontal stress during failure of the hangingwall. It may be argued that failure of the hangingwall, particularly that in the face area, was due to the lack of friction on the vertical fractures rather than as a result of the loss of horizontal clamping force when the back area started to fail. This is one of the aspects which requires more detailed study and was not fully explored during the course of this work. Nevertheless, the principle which underlies the use of back area support to maintain horizontal clamping stress at the face in conventional stoping operations is adequately illustrated.

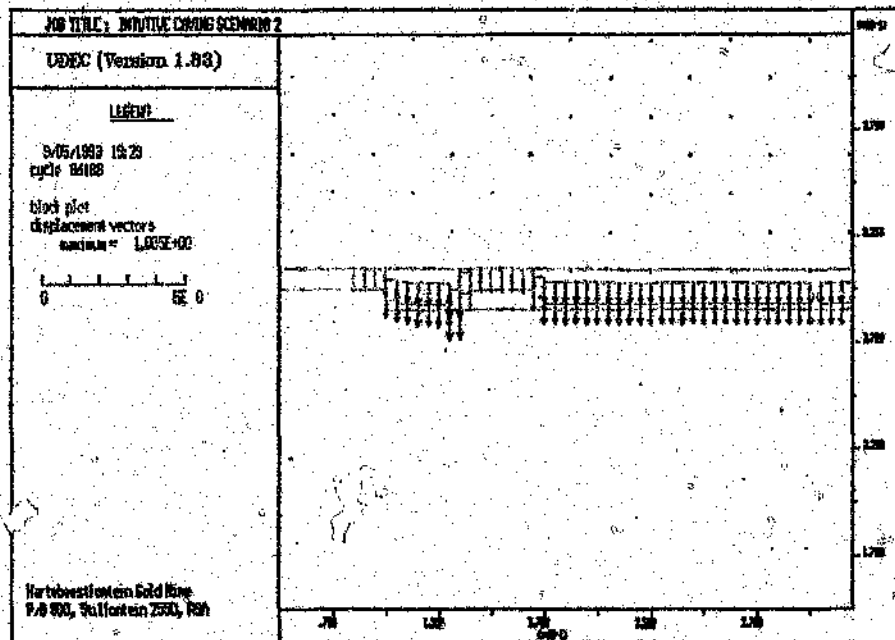
This model is intended to demonstrate that the cave mining method is not suitable for use in situations in which the strata surrounding the stoping excavation, although discontinuous, are essentially locked together and behave as a continuum. Caving requires the strata to "flow" readily and to behave in a pseudo-plastic fashion. If the rockmass surrounding a stoping



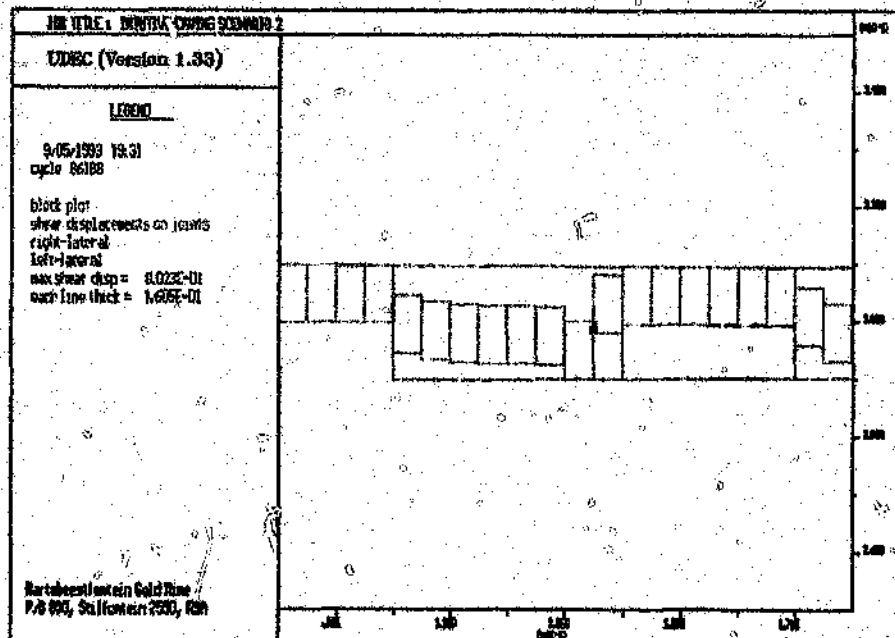
Plot 5A - The Geometry Of The Rigid Hangingwall Model.



Plot 5B - The Rigid Hangingwall Model After Collapse Of The Hangingwall.



Plot 6 - Displacement Vectors For The Rigid Hangingwall Model.



Plot 7 - Shear Displacements On Discontinuities Within The Rigid Hangingwall Model.

excavation does not possess these qualities, caving may be undesirable, or even dangerous.

8.2.3 Pseudo-Plastic Model.

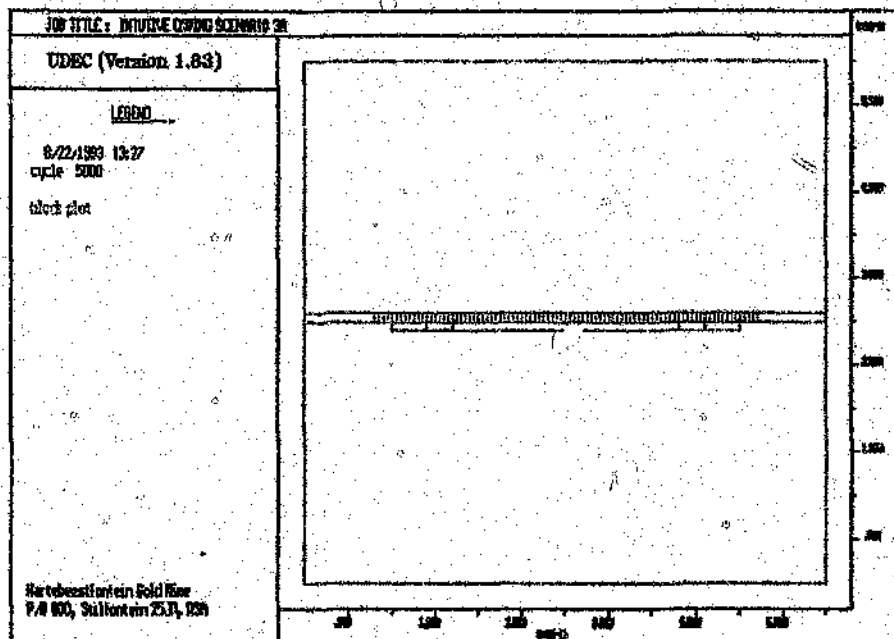
The input data for the Pseudo-Plastic Hangingwall Model (Intuitive Caving: Scenario Three), is presented in Appendix 7A(iii). This third model duplicates the second, with the exception that friction on horizontal bedding planes is drastically reduced to encourage horizontal sliding and to simulate "flow" of strata towards the stoping excavation. The geometry of the model is shown in Plot 8A.

The aim of this model was to include all the elements considered to be essential for a successful caving operation, while at the same time retaining simplicity and parity with the previous models so that the fundamental differences between caving and non-caving environments could be identified. It was discovered that these differences are perhaps more subtle and complex than can be adequately addressed and resolved by the simple approach adopted here. As has already been mentioned, a very serious limitation was that of being unable to accurately model what is undoubtedly one of the most important aspects of a caving operation: the dilation of the rockmass ahead of the face caused by the processes of compressive rock failure. Despite these shortcomings it was, nevertheless, possible to mimic a back area cave while at the same time retaining stability at the face (Plot 8B). The model did, however, reinforce the recognition that caving is essentially a dynamic process requiring an adequate rate of face advance for its success.

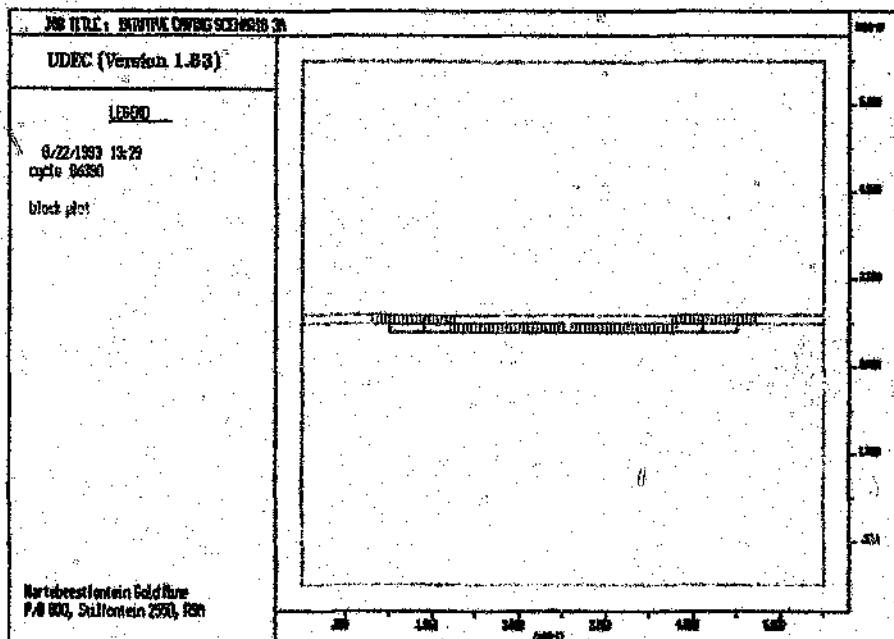
Theoretically, the hangingwall ahead of the support zone should remain stable if the tendency of the rock to dilate towards the back area is resisted to some degree by the clamping action of the face support. In the model this mechanism was created with some degree of success by inducing horizontal movement of the hangingwall beam and by inhibiting this movement across the support zone. Horizontal movement of the beam was achieved by applying a horizontal force of 5MN (the same as was applied in the rigid hangingwall model) to the ends of the beam, by specifying a dilation angle of 10° for the vertical fractures and by eliminating friction on the bedding planes. Resistance to this movement was achieved by specifying a 60° angle of friction at the rock-support interface and by making the support active and relatively stiff. An extremely low friction angle of 1° was specified for the vertical fractures.

Once again it may be argued that it is friction associated with the vertical fractures rather than horizontal force which accounts for the stability or otherwise of the hangingwall at the face area and for the differences between these particular models; and once again the limitations mentioned above apply. Nevertheless, the horizontal clamping stresses acting in the immediate hangingwall of this pseudo-plastic caving model are significantly higher than those acting in the rigid hangingwall model, bearing in mind that the same horizontal force of 5MN was applied to each model. Horizontal stress for this model is shown in Plot 10, which indicates theoretical horizontal stress levels of up to 60Mpa occur in the hangingwall at the face area. This may be contrasted with the much lower, almost tensile, horizontal stresses acting in the rigid hangingwall model (Plot 9).

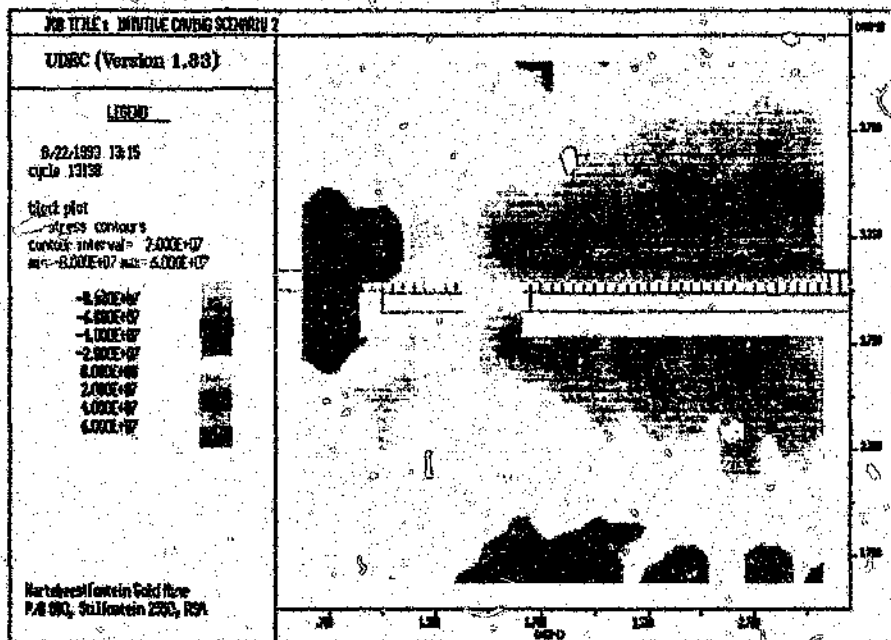
The magnitude and sense of displacements of the blocks and the shear displacements on the discontinuities are presented in Plots 11 and 12 respectively. Differential horizontal movement occurs on the bedding planes, contributing to increased stability of the hangingwall in the immediate face area, where little or no movement occurs on the vertical fractures. The model was, however, finely balanced. Even slight changes to the cohesion and friction



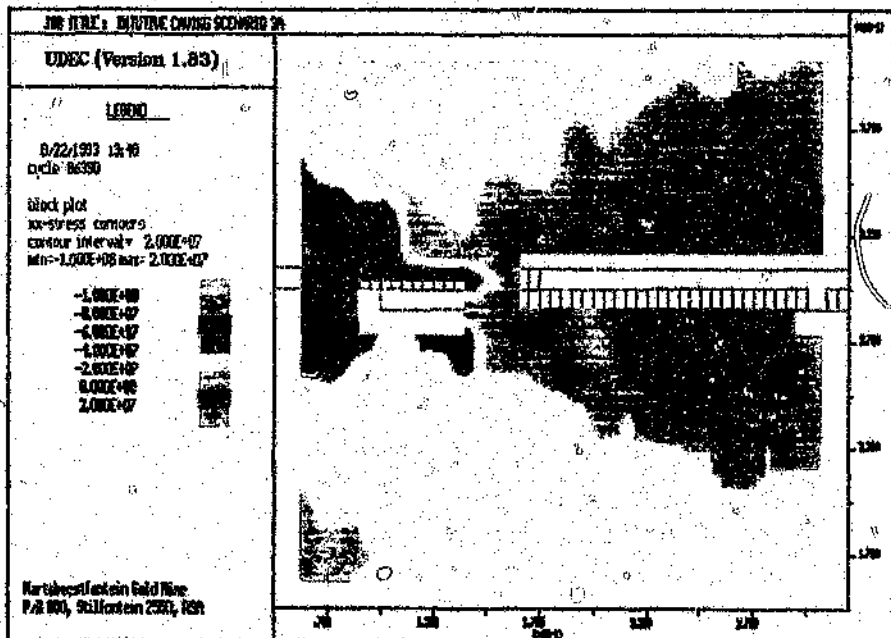
Plot 8A - The Geometry Of The Pseudo-Plastic Hangingwall Model.



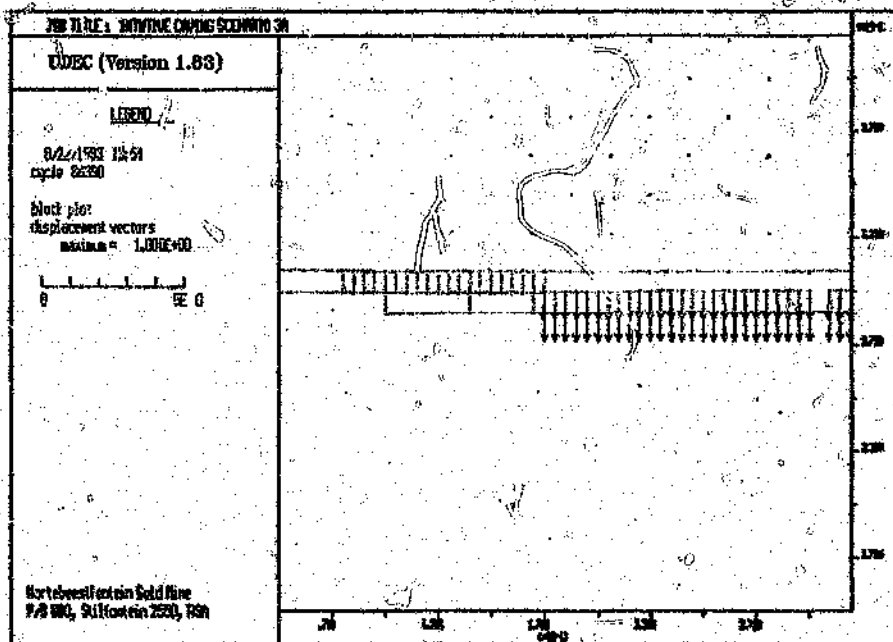
Plot 8B - The Pseudo-Plastic Hangingwall Model After Caving Of The Back Area.



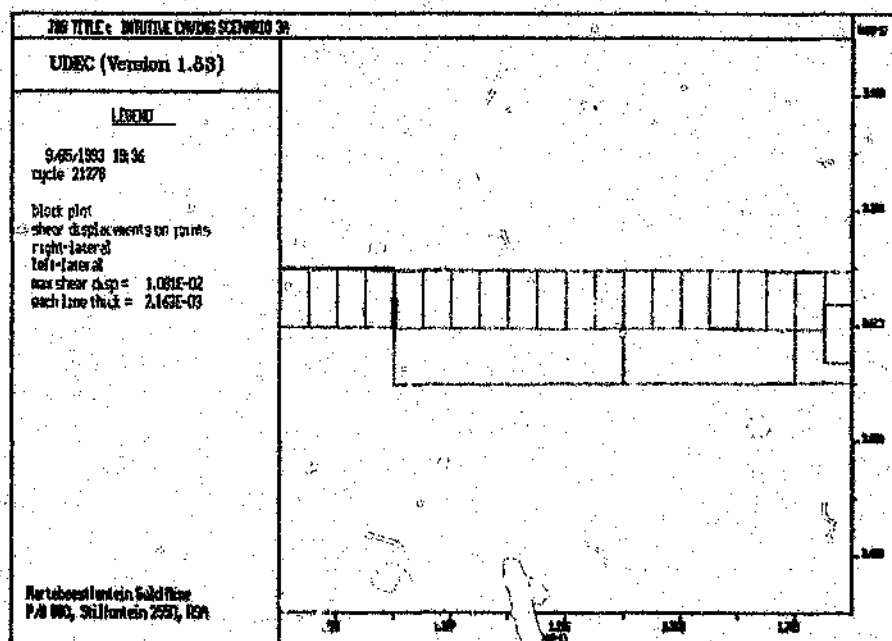
Plot 9 - Horizontal Stress Acting In The Rigid Hangingwall Model During Failure Of The Hangingwall.



Plot 10 - Horizontal Stress Acting In The Pseudo-Plastic Hangingwall Model During Caving Of The Back Area.



Plot 11 - Displacement Vectors For The Pseudo-Plastic Hangingwall Model.



Plot 12 - Shear Displacements On Discontinuities Within The Pseudo-Plastic Hangingwall Model.

parameters resulted in collapse of the fractured beam in the face area. It could perhaps be concluded that the hangingwall in the working area of cave mining panels is in a constant state of unstable equilibrium.

8.2.4 Contrasting the Intuitive Models.

The purpose of the three models described above was to validate "intuitive" perceptions concerning the physical factors which make caving possible in some environments and not in others. The important features were considered to be:

- a) The structure, coherence and tensile strength of the hangingwall which would ensure integrity over relatively small spans yet allow it to cave in a controlled fashion over larger spans.
- b) The continuous development of dilatatory forces ahead of the face and the translation of these forces into horizontal movement of the stope hangingwall strata.
- c) The ability to control this horizontal movement with the action of support such that sufficient stress is generated to maintain the integrity of the hangingwall strata at the face.

The models did demonstrate that fundamental differences in these key factors are instrumental in determining the success or otherwise of achieving a stable back-area cave, thereby confirming the thread which links all literature on deep level caving operations in South Africa. Significantly, the models also exposed the sensitivity of stable face area conditions to changes in the properties of joint surfaces in the hangingwall. It must once again be emphasised that a much more thorough treatment of the subject is required to quantify these sensitivities, bearing in mind the limitations imposed by the current level of program technology and the very simple representation of complex hangingwall structures employed in the model.

Some speculative conclusions (which should be treated with a great deal of circumspection) may be drawn from further comparison of the results of these models. For example, comparing the differences in hangingwall displacement velocity it may be concluded that the pseudo-plastic cave mining scenario may be more prone to time-dependant deterioration and unravelling than the rigid hangingwall model. This conclusion supports the underground observations which suggest that continuous face advance is essential for the stability of caving panels. Experience at No.5 Shaft and elsewhere on the mine when caving stopes are allowed to stand for any length of time is that face conditions deteriorate fairly rapidly. In extreme cases when panels are stopped for an extended period of time, re-raising invariably has to be resorted to as conditions on the face deteriorate to such an extent as to render re-opening unacceptably hazardous. While this phenomenon occurs in most deep mining environments, empirical observation would suggest that it assumes greater proportions in pseudo-plastic ground conditions which lend themselves to cave mining techniques.

Contrary to the well documented and wide-spread observation of the dramatic effects of stress change and stress re-distribution accompanying the achievement of a cave underground, no significant change in either the magnitude or the direction of stress was noted in the model. Once again this may be largely due to limitations in the modelling technique. Perhaps the most significant information obtained from the intuitive models was that they reinforce the

conclusions drawn from underground observations regarding the importance of inhibiting horizontal movement of the hangingwall and of installing support as close to the face as possible to maintain safe and stable working conditions.

8.3 DYNAMIC CAVING MODEL.

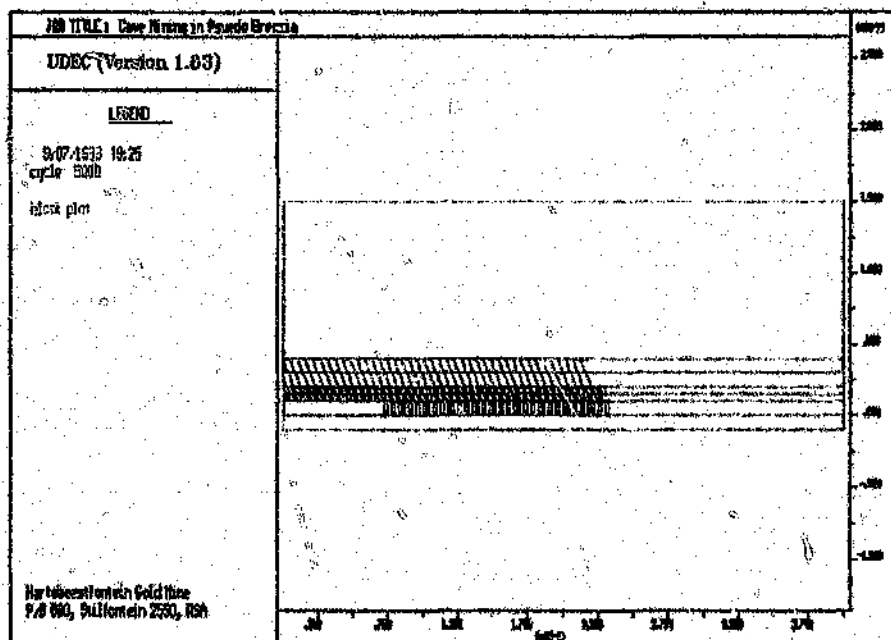
Having established some basic principles with the simple models described above, a more ambitious model was attempted which incorporated a larger number of bedding planes, a more complex arrangement of fractures, and a more accurate representation of the support elements. In addition, the process of advancing the stope face and of leap-frogging the support elements was mimicked (hence the title "dynamic").

8.3.1 Description of the Model.

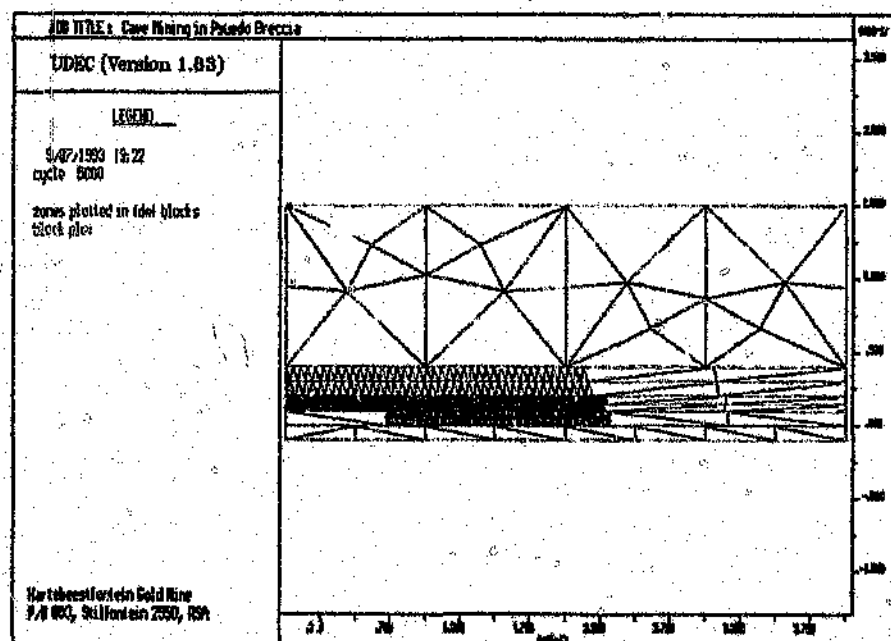
The input data for this Dynamic Caving Model is presented in Appendix 7B. A horizontal stoping slot with a width of 1m was created in the centre of a 80m wide x 30m high UDEC window using two axes of symmetry. The window was originally 60m high, but this was cut to 30m to reduce excessive calculation run times. The results of the model were not affected significantly by this reduction.

Four bedding planes were created 0,5m; 1,0m; 2,0m and 3,0m above the stope and were assigned properties of zero cohesion, 30° friction angle and 2° dilation angle. Vertical fracture planes spaced 0,25m apart were created immediately ahead of the advancing stope face, simulating stress fractures. These fracture planes extended into the stope hangingwall. The immediate hangingwall fractures dipped at 80° towards the face with some terminating against the bedding plane situated 1m above the stope. The remainder, now spaced 0,5m apart and dipping at 70°, persisted for a further 2m into the hangingwall, being truncated by the top-most bedding plane. Although the positions of these fracture planes were defined in advance (because the UDEC program does not allow joints to be created after finite difference zones have been established), they were only mobilised when the face approached within 3,5m of the face fractures, 3,0m of the immediate hangingwall fractures and 2m of the far hangingwall fractures respectively. The fractures could thus be regarded as being continuous, forming 3,5m ahead of the advancing face and reaching their fullest extension 2m ahead of the face. The fractures were assigned initial properties of 10MPa cohesion, 20° angle of friction and 4° dilation angle. The intact hangingwall rock material was assigned a bulk modulus of 28GPa, a shear modulus of 26GPa and a density of 2700kg/m³. A vertical stress of 55MPa and a horizontal stress of 18MPa were applied to the model and it was brought to equilibrium by allowing it to cycled through 5000 steps before the initial ledging slot was removed. The geometry of the dynamic caving model is shown in Plot 13A. The finite difference zone sub-divisions are shown in Plot 13B.

The most recent version of UDEC, capable of simulating support loads with more finesse, was acquired shortly before work started on this "dynamic" model. Using this facility, reactive support loading, equivalent to three rows of 400kN yielding prop units installed 3,5m from the face and spaced 1,5m apart on strike, was applied in the model. Individual support units were required to perform according to specifications typical of the load-compression characteristics of a 40ton hydraulic prop, viz.: 180kN at installation and 400kN following compression. The support loads were removed from the back row and applied 3,5m behind the face and 1,5m ahead of the previous props as the face advanced, mimicking the leap-frog



Plot 13A - The Geometry Of The Dynamic Caving Model Before Mining.



Plot 13B - The Model Blocks Sub-Divided Into Finite Difference Zones.

technique applied underground.

Stoping was specified to commence at, and advance away from, a ledging position 7,25m from the centre gully. Modelling blocks were deleted ahead of the face position to simulate the effect of advancing the stope face in steps of 1,5m. At the same time, the back row of support units were moved forward and the fracture planes were mobilised for a distance of 3,5m ahead of the new face position. It was necessary to allow the model to cycle through sufficient iterations following each mining step to re-establish equilibrium before proceeding to the next mining step. In all, nine mining steps were simulated which advanced the stope face by just short of 20m. The model took just short of one full week (150 hours) to complete the calculations for the nine mining steps, and required in excess of 16 Megabytes of random access memory.

Histories of velocity, displacement and stress were recorded at selected points in the hangingwall ahead, at and behind the advancing stope face allowing the behaviour of the rock at these points to be monitored as the stope face advanced towards and beyond them. These histories are analyzed later in this chapter.

8.3.2 The Mining Steps.

The first mining step simulates the establishment of a 14,5m ledging slot. The half-span of 7,25m is entirely supported with three rows of hydraulic props spaced 1,5m apart which are installed 3,5m from the face. A maximum vertical displacement of 24mm is indicated at the mid-point of the stope over the centre gully. The displacements are largely elastic and no bed separation is observed. This is shown in Plot 14A, which is the first in a series of plots illustrating the behaviour of the model as it progresses through the nine mining steps. The plots represent the last frame of the movie file for each mining step respectively, so the graphic detail of the blocks as they fall during each mining step is not as dramatically illustrated as in the complete movie, but nevertheless give a fair indication of the action.

During the second mining step (shown in Plot 14B) the face is advanced to create a half-span of 9m and an unsupported span of 5m over the centre gully where the vertical displacement has increased from 24mm to 35mm, still with no apparent bed separation. Similarly, the third mining step produces a further 10mm vertical deformation with no bed separation at an unsupported span of 8m (Plot 14C). Bed separation starts to take place at the second bedding plane during the fourth mining step when the unsupported span is 11m (Plot 14D). The total vertical displacement has now increased by 15mm to 50mm.

Bed separation continues above the unsupported back area and by the fifth mining step is clearly visible at the third bedding plane. The vertical displacement measured at the centre span has increased dramatically by more than 100mm to about 170mm. This is shown in Plot 14E. Note that there is a change in the scale of the representation of displacement vectors in comparison with previous plots. At this stage it is apparent that the stiff hydraulic prop support has no effect on elastic displacements, but is able to control bed separation in the working area to some extent. Although inelastic shear displacement is taking place on bedding planes and vertical fractures, the hangingwall between the support and the face appears stable. Downward velocity of the back area hangingwall, although very small, never decreases to zero, and it is probable that if the model was allowed sufficient time to run the hangingwall would start to unravel.

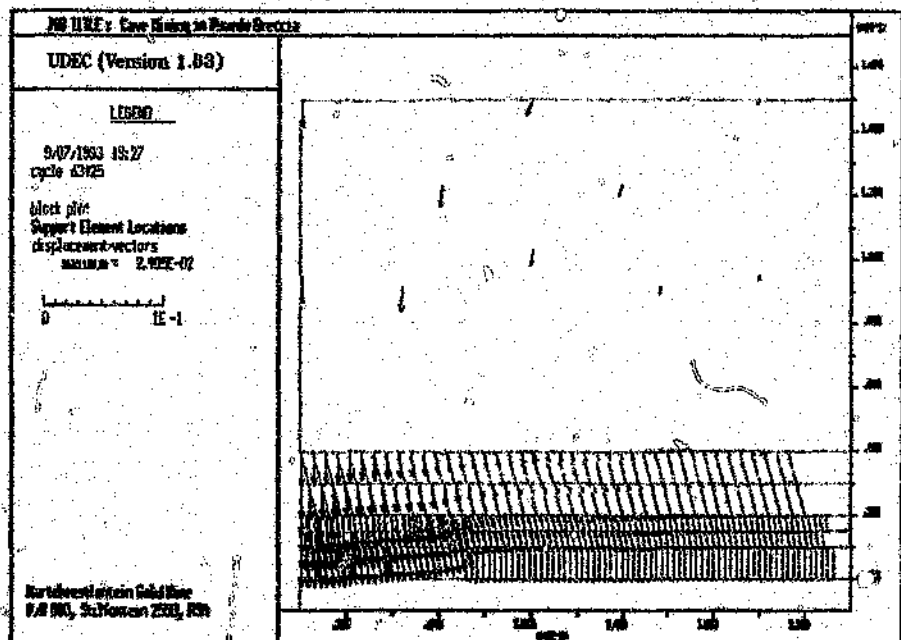
A full back area cave is achieved on the sixth mining step (Plot 14F). At this stage the total span is in the order of 30m, of which 17m is unsupported (bearing in mind that the centre gully was not supported in the model). This corresponds well with the spans at which caving often is reported to occur in practice. This initial cave is 2m thick and extends up to the third bedding plane. During the next mining step (the seventh) the cave extends right up to the fourth bedding plane (Plot 14G). Once caving has been fully initiated, subsequent mining steps eight and nine (Plots 14H, 14I and 14J) result in the forward extension of the caving front. In all case the cave occurs right up to the back row of props.

8.3.3 Discussion of Results.

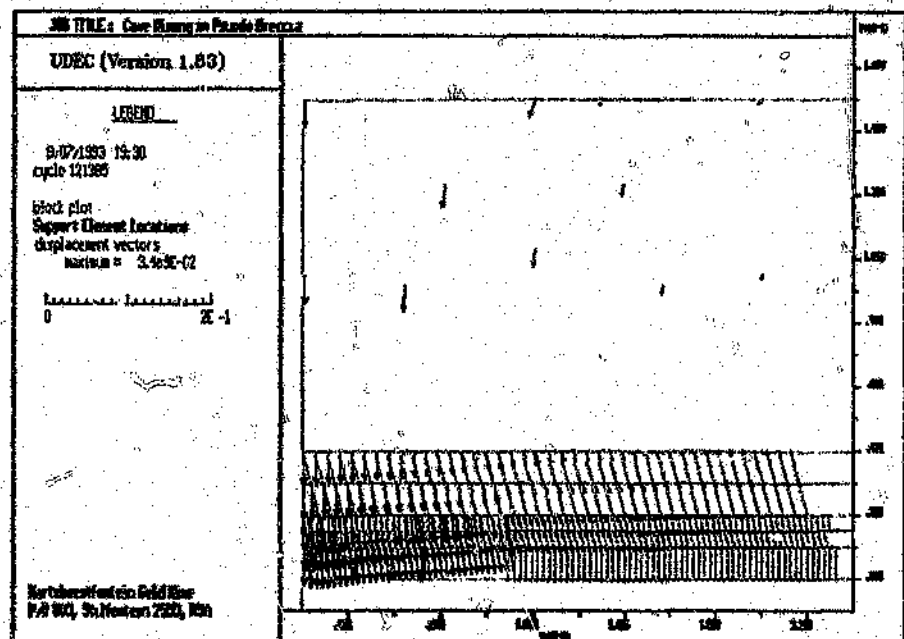
Underground observations and measurements in actual caving stopes strongly indicate a substantial reduction in vertical stress ahead of the face and in horizontal stress in the hangingwall and footwall adjacent to the face immediately following the initial occurrence of a cave. A comparison of vertical and horizontal stress contours generated by the dynamic caving model before the cave (mining step five, Plots 15A and 16A) and after the cave (mining step six, Plots 15B and 16B) do not bear this out. The state of stress in the rockmass surrounding the model stope is largely unaffected by the occurrence of the cave. In fact there is a slight increase in vertical stress at the face following the cave, in all probability as a result of the increased mining span. Although there is a decrease in horizontal stress above the caved region, horizontal stress in the hangingwall adjacent to the stope face is not affected by the cave. The magnitude and orientation of principle stress vectors at and ahead of the face before and after the cave (shown in Plots 17A and 17B respectively) are also not significantly different. The magnitude of the maximum principle stress (which is almost vertical) increases from 283MPa at the face before caving to 317MPa after caving.

Even though the strength properties of the rock ahead of the face have been considerably reduced, model behaviour in the compressive stress zone is elastic, and not representative of the true underground situation. Plot 18 shows the material and joint property designations at mining step five, the detail of which is presented in Appendix 7D. Clearly, in order to model the cave stope environment more realistically it will be necessary to incorporate a failure criterion with some post failure strain softening characteristic. Further, the process and effects of dilation resulting from failure of the rock ahead of the face is not accurately represented and also requires more realistic modelling. An examination of shear displacements along joints in the face area (shown in Plot 19) indicates very little differential horizontal movement on the bedding planes. In fact most of this movement appears on the top-most bedding plane. In practice the maximum shear movement would be expected to occur on the hangingwall bedding plane, decreasing sequentially on bedding planes further removed from the stoping excavation. In the model the entire jointed hangingwall is moving backwards en-mass with very little differential movement between the beds because of the uniformity of the externally applied horizontal loading condition. A more realistic representation of the fracture process ahead of the face and thus also the forces driving this horizontal movement will result in a more realistic approximation of actual hangingwall behaviour.

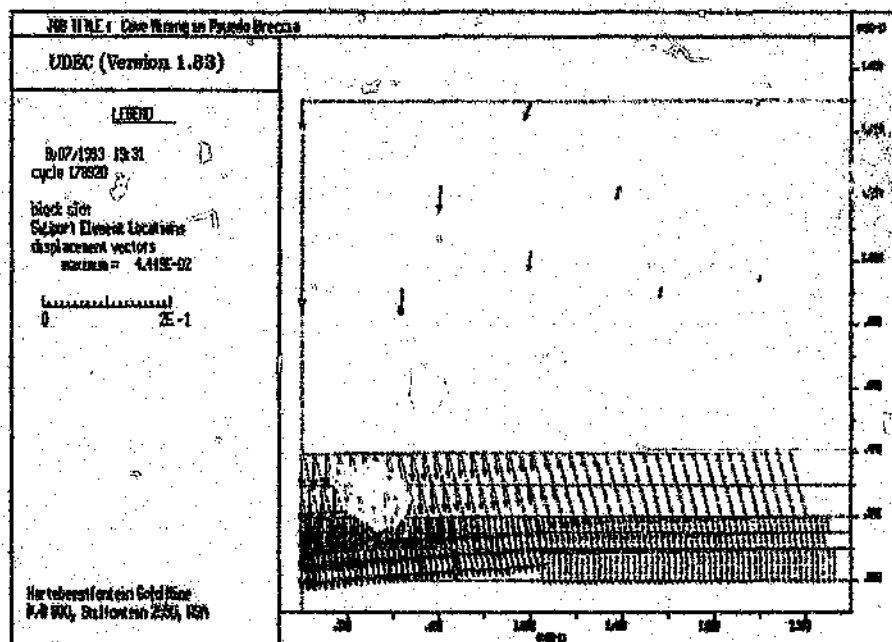
Despite these very significant shortcomings, the model does nevertheless represent a first step towards more comprehensive modelling of the inelastic processes which give rise to successful cave mining, and which will in turn allow more detailed examination of the mechanisms which promote the application of cave mining techniques in some circumstances and not in others. It is suggested that further definitive work may in the future lead to the development



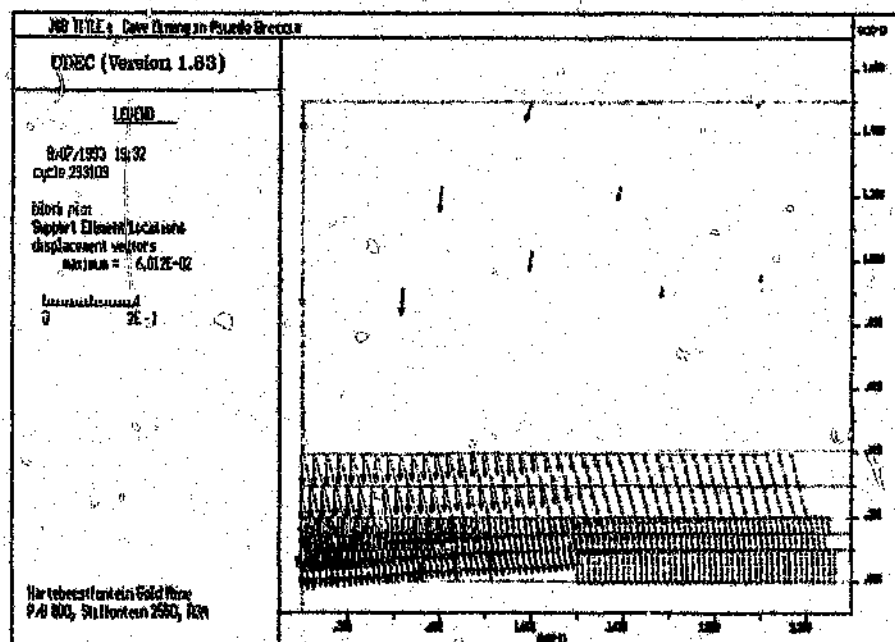
Plot 14A - The Dynamic Caving Model After Initial Ledging (Mining Step 1) Showing Displacement Vectors And Positions Of Support Units.



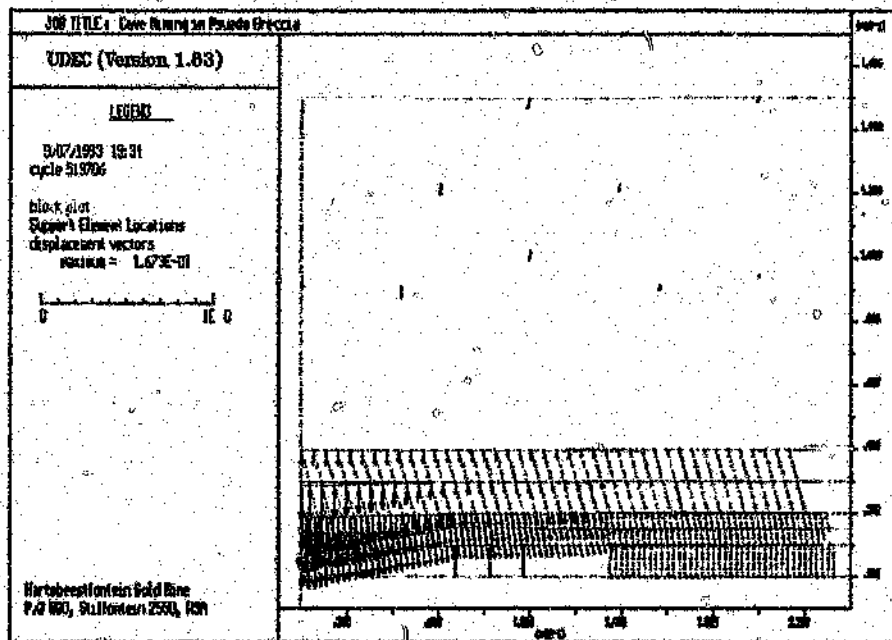
Plot 14B - Final Displacements In The Dynamic Caving Model After Mining Step 2.



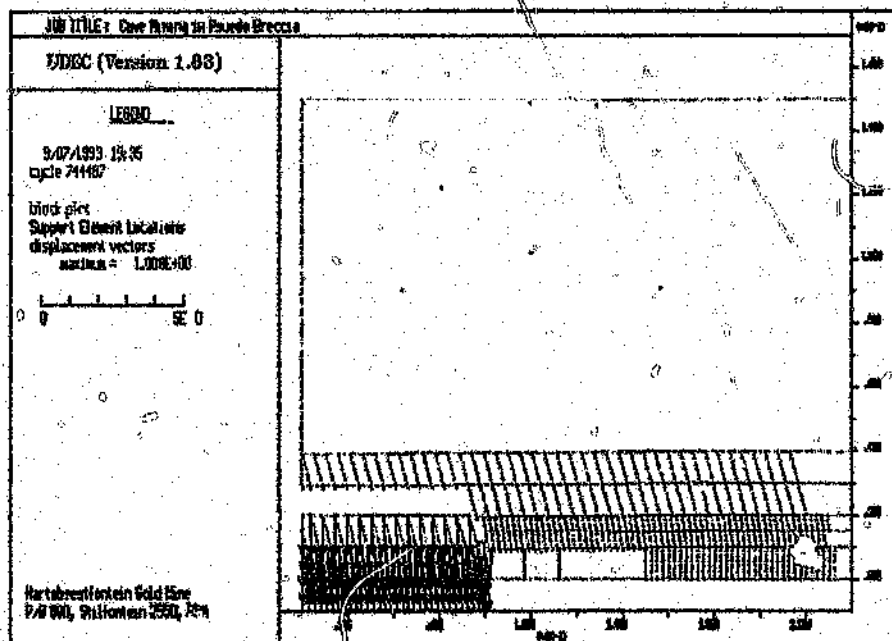
Plot 14C - Final Displacements In The Dynamic Caving Model After Mining Step 3.



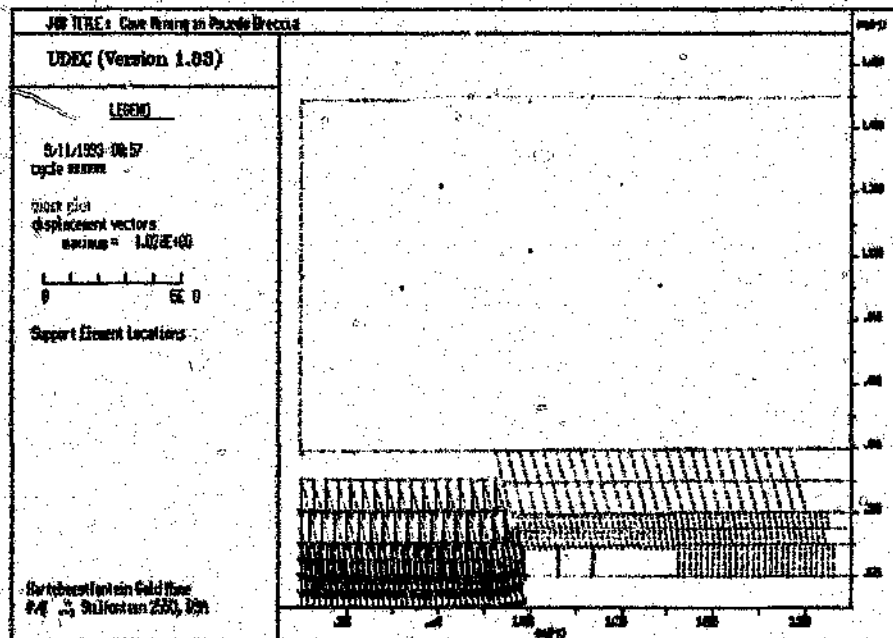
Plot 14D - Final Displacements In The Dynamic Caving Model After Mining Step 4.



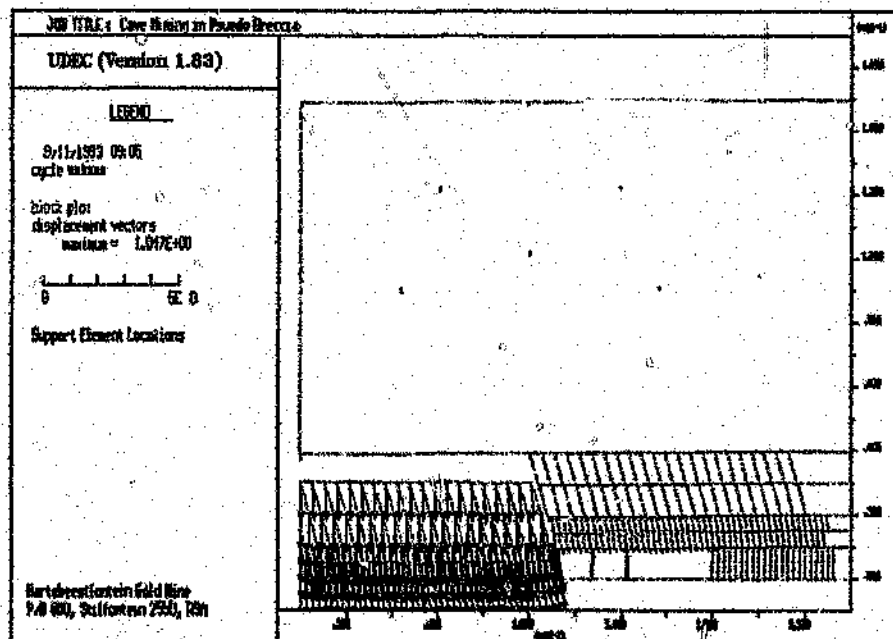
Plot 14E - Final Displacements In The Dynamic Caving Model After Mining Step 5.



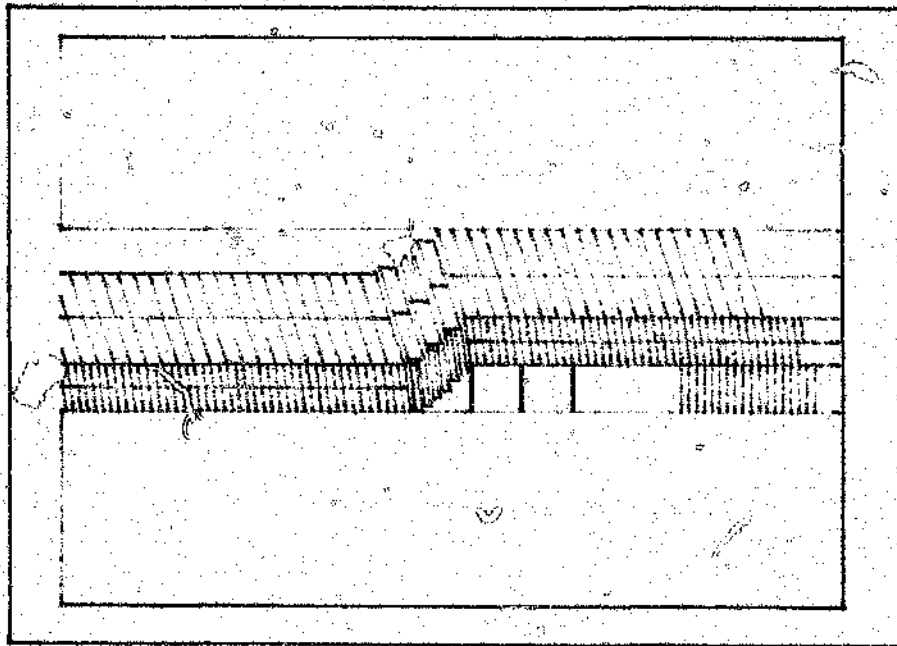
Plot 14F - Final Displacements In The Dynamic Caving Model After Mining Step 6.



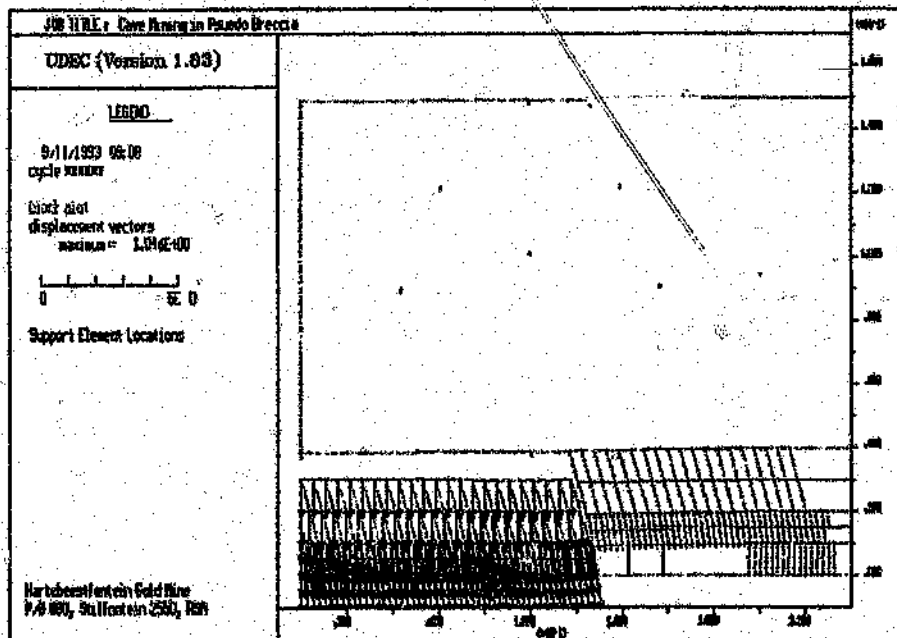
Plot 14G - Final Displacements In The Dynamic Caving Model After Mining Step 7.



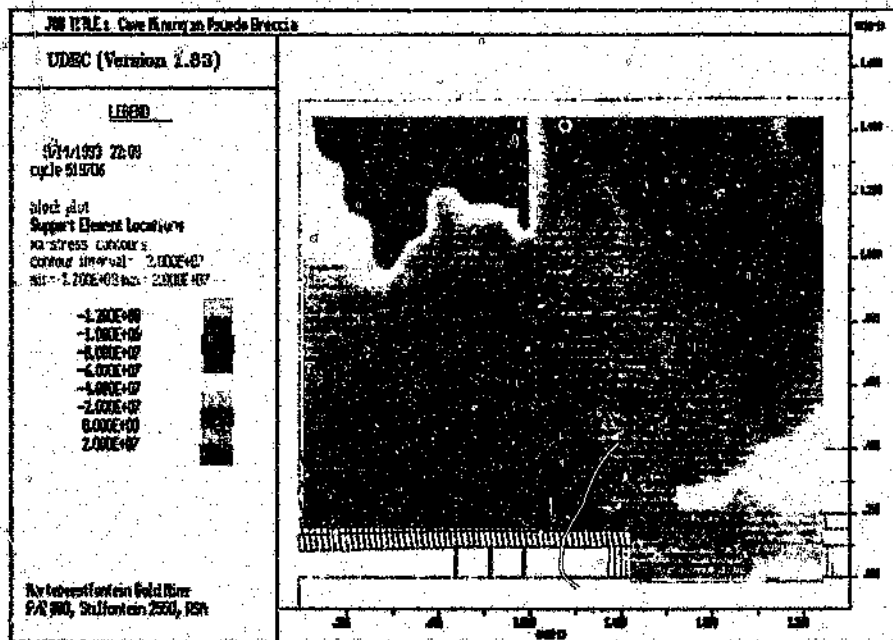
Plot 14H - Final Displacements In The Dynamic Caving Model After Mining Step 8.



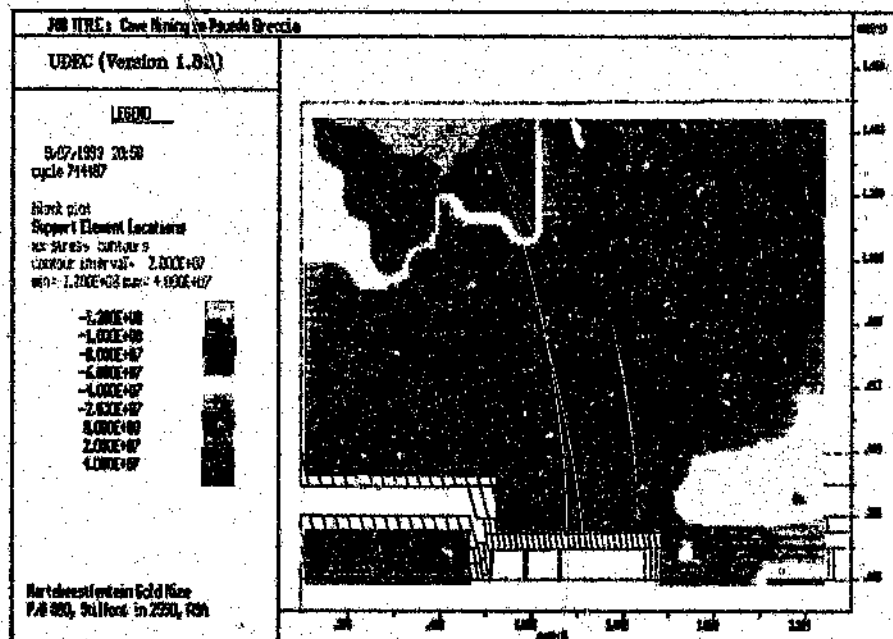
Plot 14I - Displacements In The Dynamic Caving Model During Mining Step 9.



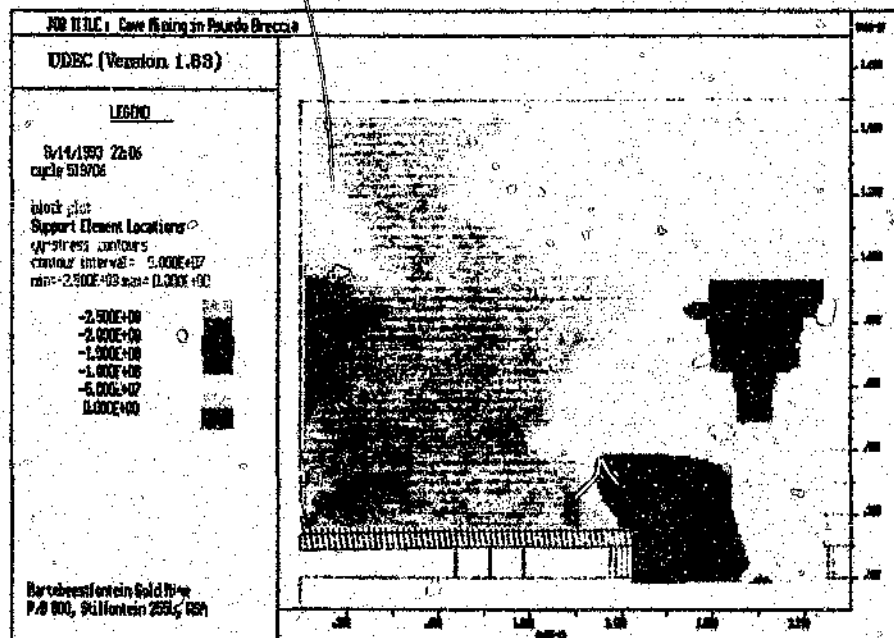
Plot 14J - Final Displacements In The Dynamic Caving Model After Mining Step 9.



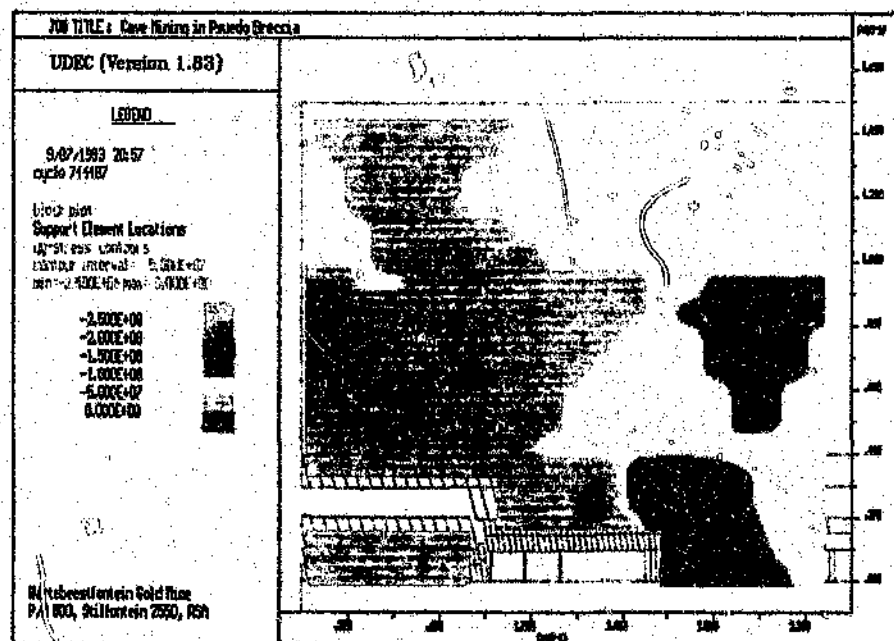
Plot 15A - Vertical Stress Contours At Mining Step 5 Before Caving Occurs.



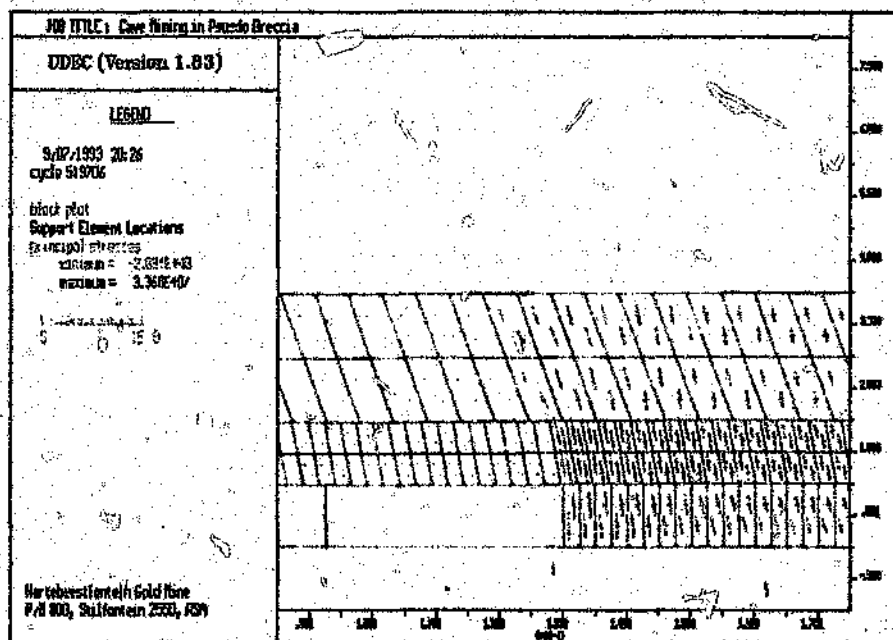
Plot 15B - Vertical Stress Contours At Mining Step 6 After Caving Occurs.



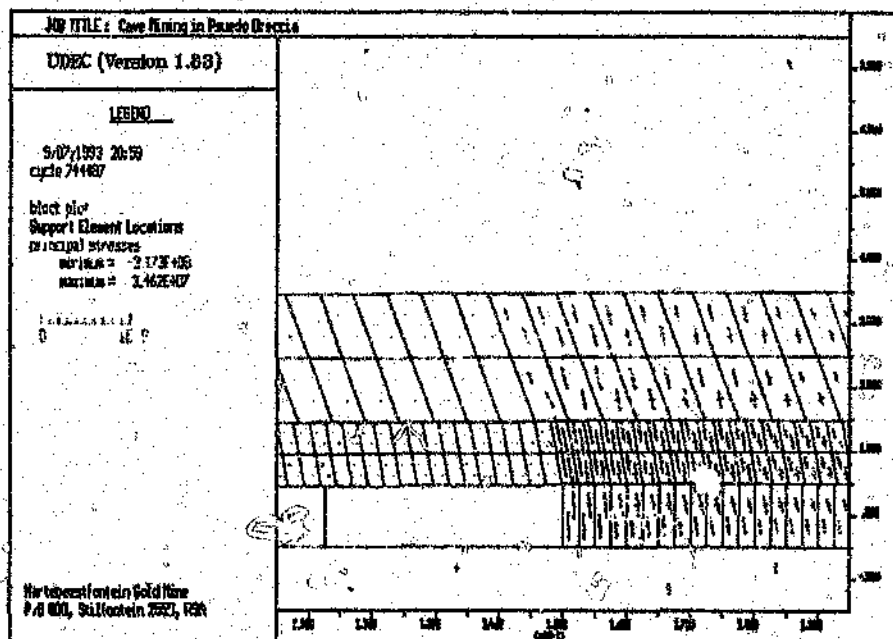
Plot 16A - Horizontal Stress Contours At Mining Step 5 Before Caving Occurs.



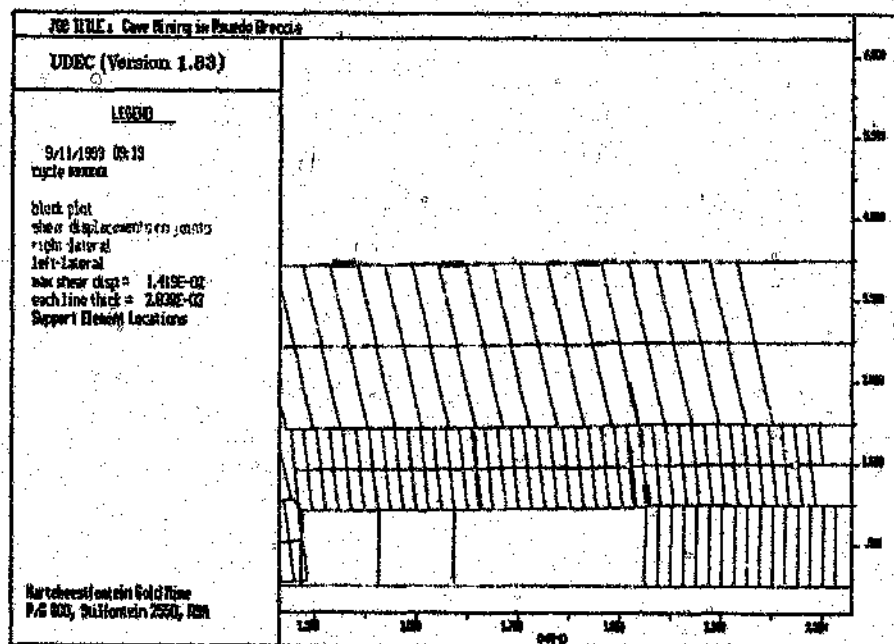
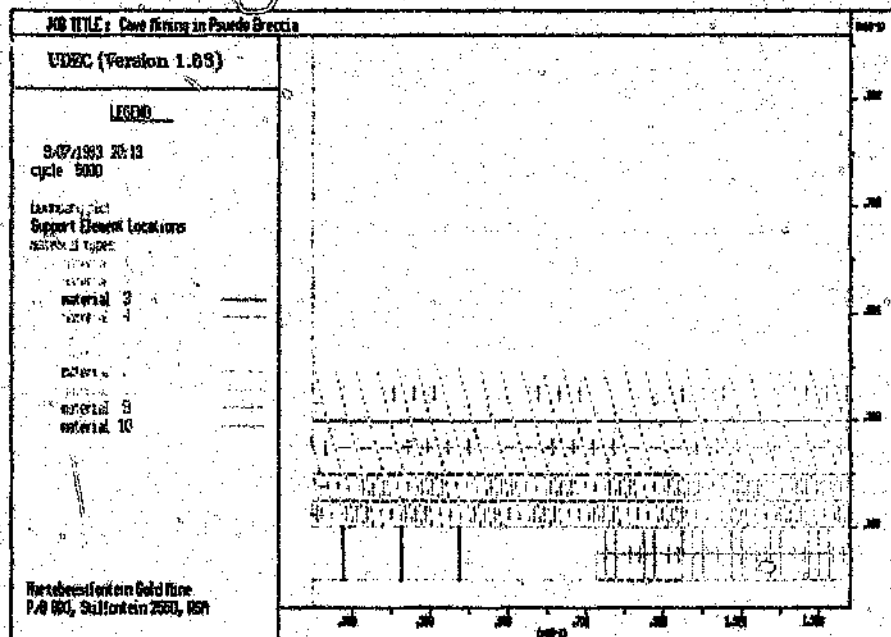
Plot 16B - Horizontal Stress Contours At Mining Step 6 After Caving Occurs.



Plot 17A - Principle Stress Vectors At Mining Step 5 Before Caving Occurs.



Plot 17B - Principle Stress Vectors At Mining Step 6 After Caving Occurs.



of a methodology for determining whether stoping areas are suitable or not for caving by relatively simple characterisation and evaluation of the rock strata surrounding the reef horizon. The development and inclusion in the program of algorithms which describe fracture processes taking place ahead of and around the stope face will undoubtedly facilitate this.

8.3.4 Deformation Histories.

The histories of velocity, displacement and stress were recorded at selected points in the modelled rockmass to determine the effect of simulated mining on the rock ahead and above the stoping excavation. The positions of the points at which histories were requested are described as follows, and are indicated on Figure 48:-

- a) Point No.1 was situated in the immediate hangingwall about 0,3m above the stope. At the start of stoping operations (mining step one) its position was 1,75m ahead of the face. Subsequent mining steps advanced the face beyond this position allowing a continuous history of the deformation of the immediate stope hangingwall to be monitored. Vertical and horizontal velocities and displacements recorded at this point are shown in Plots 20A and 20B respectively.
- b) Point No.2 was also situated 1,75m ahead of the ledging stope face, but was 2,5m above the stope hangingwall within the fourth bedding layer. This allowed deformation of the deep hangingwall to be contrasted with that close to the hangingwall surface. Vertical and horizontal displacements were recorded at this point, and are shown in Plot 21.
- c) Point No.3 was situated 0,3m up into the immediate hangingwall bedding layer and was 4,75m ahead of the face at the start of stoping operations, allowing deformations associated with both the approaching and the receding stope face to be monitored. Vertical and horizontal displacements recorded at this point are shown in Plot 22.
- d) Point No.4 was situated on reef elevation, far ahead of the original face position, allowing stress changes associated with the advancing stoping operations to be monitored. The point was some 12,5m ahead of the face after the completion of ledging (mining step one), and only 0,5m ahead of the face at the end of mining step nine. Vertical and horizontal stresses recorded at this point are shown in Plot 23.

The graph of the vertical and horizontal velocity of displacements recorded at Point No.1 (Plot 20A) shows the sudden increase in velocity at the start of each mining step. The mining steps are easily identified by these spikes. The highest peaks occur, firstly, at mining step two in response to elastic rebound when the face position coincides with the position of the monitoring point, and, secondly, at mining step six when the point is now situated in the back area where it forms part of the collapsing hangingwall. It may be noted that the velocity does not return to zero immediately as before, but oscillates to equilibrium. This is due to the sliding nature of the cave and the effect of damping imposed by the model. The smaller peak at mining step seven is due to the effect of the collapse of the top-most bedding layer at this juncture. Subsequent caves during mining steps eight and nine, which do not enlarge the height of the collapsed region, have little or no effect.

The vertical displacements at Point No.1 (Plot 20B) follow a typical stope closure profile once the point is understoped at mining step two. Initial downward displacements are large in

comparison with subsequent displacements as the face advances away from the monitoring point which is situated in the immediate stope hangingwall. At mining step six the displacement associated with the collapse of the hangingwall is dramatic, and represents a fall through almost the entire 1m stoping width, but with some sliding and slowing down towards the end, the block containing the monitoring point does not quite reach the footwall. The point is situated in a block on the forward edge of the cave. The block is then re-mobilised by the subsequent collapse of the next cave (mining step seven) which includes the top-most bedding layer, finally coming to rest on the stope footwall. Horizontal displacements which take place prior to the occurrence of the cave are slight, and are uncharacteristically directed towards the face. More significant horizontal displacements, also directed towards the face, are associated with the two caving episodes mentioned above.

The vertical displacement at Point No.2 (Plot 21), situated more than 2m higher up in the hangingwall, mimics, and is only slightly less than, that at the stope hangingwall surface. The monitoring point is, however, situated in the top-most bedding layer and collapses later, during mining step seven. The blocks appear to fall almost unimpeded to the full extent allowed by the stoping width. Horizontal displacement is, as expected, absent prior to the initiation of caving of the top-most bedding layer. During the cave, however, horizontal movement is at first, as anticipated, towards the back area. This is soon reversed as the collapsing material strikes and slides forward on the inclined face of the previous cave, coming to rest almost vertically below, but slightly forward of its original position.

The displacement history at Point No.3 (Plot 22), situated in the immediate stope hangingwall layer some 5m ahead of the ledging face position, provides a complete record of the near-field behaviour of strata as the stoping excavation firstly approaches, and then advances beyond the monitoring position. Vertical displacement is as expected. Very little downward movement takes place during the first three mining steps while the monitoring point is situated in the solid ahead of the face. Closure starts to take place at mining step four when the face reaches the monitoring position, and then continues, following a closure profile typical of supported face areas, for the next four mining steps. Displacement associated with a full cave then occurs at mining step eight. Horizontal displacement on the other hand is the opposite of what is expected. Instead of moving towards the back area, the immediate hangingwall moves, albeit very slightly, in the direction towards the face. This emphasises the inadequacy of this model in representing and dealing with real macro-plastic flow phenomena.

Similarly, the history of stress changes at Point No.4 (Plot 23) on the reef plane ahead of the advancing face is representative of a typical elastic analysis, and not of the actual caving environment, indicating once again that the model as formulated is unsuitable for the analysis which was attempted, and that a strain softening model would be more appropriate.

8.3.5 Conclusions.

The following conclusions were made regarding the attempt at modelling two-dimensional representations of a caving stope using the UDEC program:-

- a) Although the models turned out to be limited in their ability to fully represent macro-plastic flow and other pseudo-plastic phenomena associated with the cave mining method, as a first attempt they provided significant insights into what may be regarded as the key elements required to facilitate successful caving. In particular, the dynamic caving model showed promise and warrants a more detailed modelling

effort to more accurately define and quantify caving parameters.

- b) However, foremost amongst the limitations was the inability of the dynamic model, and indeed of UDEC or any other currently available program, to synthesise the actual failure, fracture and dilatation processes which take place ahead of an advancing stope face at depth. These give rise to the continual generation and regeneration of horizontal force in the hangingwall and footwall, and are considered, in combination with a high frequency of well lubricated bedding planes, to be an essential element contributing to the viability of the caving method. Until this aspect can be realistically modelled further progress will be difficult to achieve.

In the absence of these failure processes, typical elastic behaviour was observed ahead of the stope face with, for example, vertical stress peaking at the face and decreasing further ahead. It was apparent that simply down-grading the material properties of the rock for up to 2m ahead of the face was insufficient to simulate failure. The specification of strain-softening material characteristics might be more appropriate, but, as mentioned above, it is the dilation effects of the failure process which is thought to play a vitally important role.

- d) Attention might profitably be given to the specification of boundary conditions. Stress driven boundaries were employed throughout this modelling exercise to reduce the time taken to achieve equilibrium. This may, however, have given rise to some anomalous results which are particularly apparent in the horizontal stress contour plots. A displacement driven boundary condition may prove to be more appropriate in these circumstances.

The industry is currently giving attention to the problem of developing mathematical models capable of simulating fracture processes associated with rock failure. When these have been successfully incorporated into programs such as UDEC, it should be possible to study the mechanisms of caving more realistically than can be done at present. This in turn would lead to a better understanding of the environment required for caving, the ability to assess the suitability of applying caving techniques in differing underground environments, and the design of more appropriate and cost effective support systems to suit different conditions.

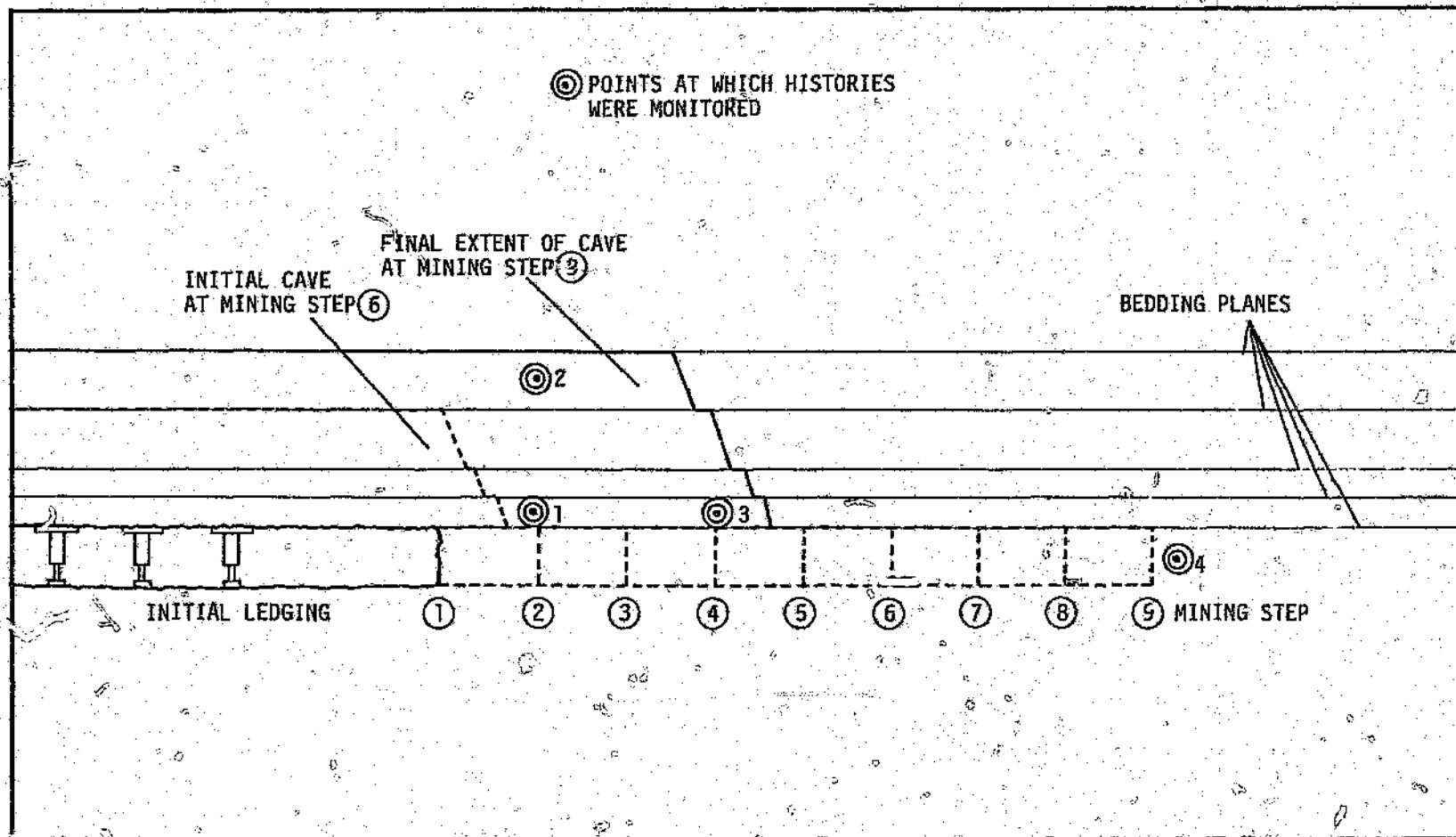
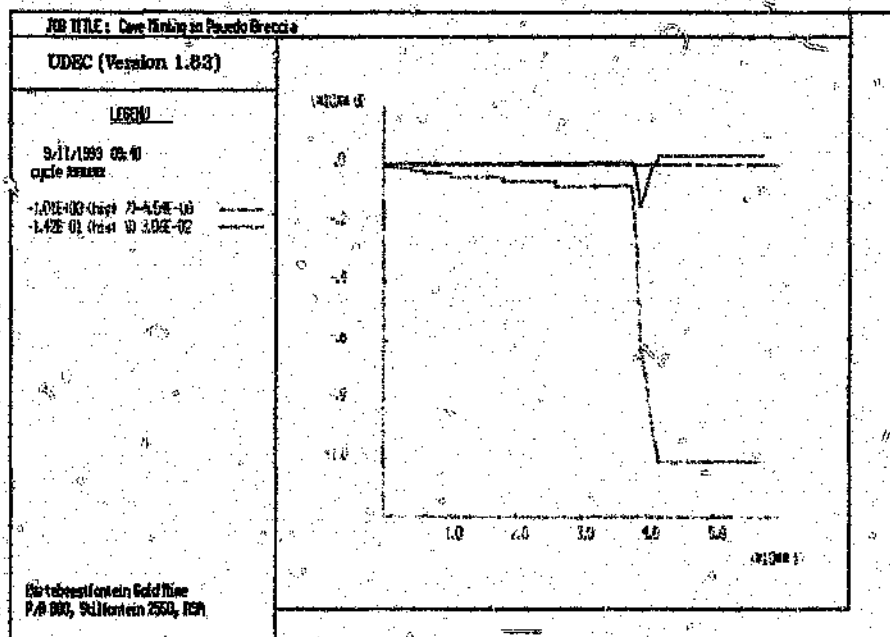
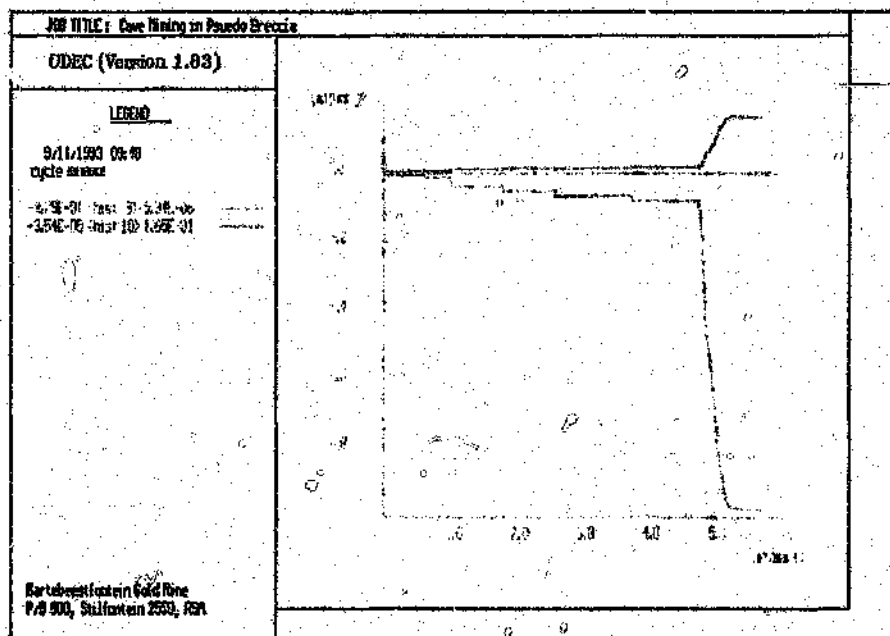


Figure 48

Points at which Deformation Histories were Monitored in the Dynamic Caving Model.



Plot 21 - History Of Vertical And Horizontal Displacements Recorded At Point No.2.



Plot 22 - History Of Vertical And Horizontal Displacements Recorded At Point No.3.

9. DISCUSSION AND CONCLUSIONS.

9.1 THE PROJECT.

The work upon which this project report is based was initiated to seek ways of maintaining the economic viability of a marginal mining area beset by difficulties associated with weak ground at Hartebeestfontein Gold Mine. Failure to find a solution would have rendered the area unprofitable, resulting in inevitable closure of the No.5 Shaft. The application of cave mining, a comparatively inexpensive stoping method employed extensively and very successfully on the mine almost since its inception, was felt to be highly problematical in very weak ground conditions. For this reason alternatives such as backfill and other methods employing more conventional back area support were evaluated.

However, because of economic considerations, it was deemed prudent to also explore alternatives which would retain the cost benefits of caving. This report has thus traced the development of a stope support methodology aimed at maintaining narrow stoping widths and safe working conditions in weak ground, and the application of these innovations in conjunction with back area caving. The report thus evaluates not only the cave mining method but also the support system applied under cave mining conditions in some detail.

There can be no doubt of the success of the project. Not only has the future profitable exploitation of the 5A North area been assured, but the mine has since spent in excess of R25m in extending the implementation of the support and clearing methodology developed during the course of this work throughout the mine. Further expenditure continues to be authorised as the benefits and returns in terms of both profitability and safety achievements continue to be demonstrated.

9.2 GEOLOGY.

The geological environment was considered to be important from two points of view. Firstly, the presence of pseudo-breccia in the No.5 Shaft area was, and remains, an area of concern in terms of its potential to substantially impact upon profitable extraction of gold reserves. Secondly, the nature of the strata immediately surrounding stope excavations is thought to play an over-riding role in determining the viability or otherwise of the cave mining method.

9.2.1 Pseudo-Breccia.

Pseudo-breccia, in combination with associated intense faulting, leads to serious strata control problems in those sections of the mine intersected by the phenomenon. The distribution of pseudo-breccia was investigated and the zones of probable occurrence were identified. The experimental stope panels used during the course of this work were situated in areas predicted to have an intermediate to high probability of containing pseudo-breccia. In practice, stoping and strata stability difficulties were mainly associated with faulting, and these were exacerbated by the presence of pseudo-breccia. Plate 23 shows an exposure of pseudo-breccia in 78A-42W N4 Gully associated with a large fault which defines the limit of the panel, while Plate 24 clearly demonstrates the difficulties encountered when pseudo-breccia was intersected in 72-43W N7 Panel. In most instances, stability of the hangingwall in the affected areas was maintained by the presence of support close to the face. This was greatly enhanced by the



Plate 23 - Pseudo-Breccia Adjacent To A Fault Zone In 78A-42W N4 Gully.

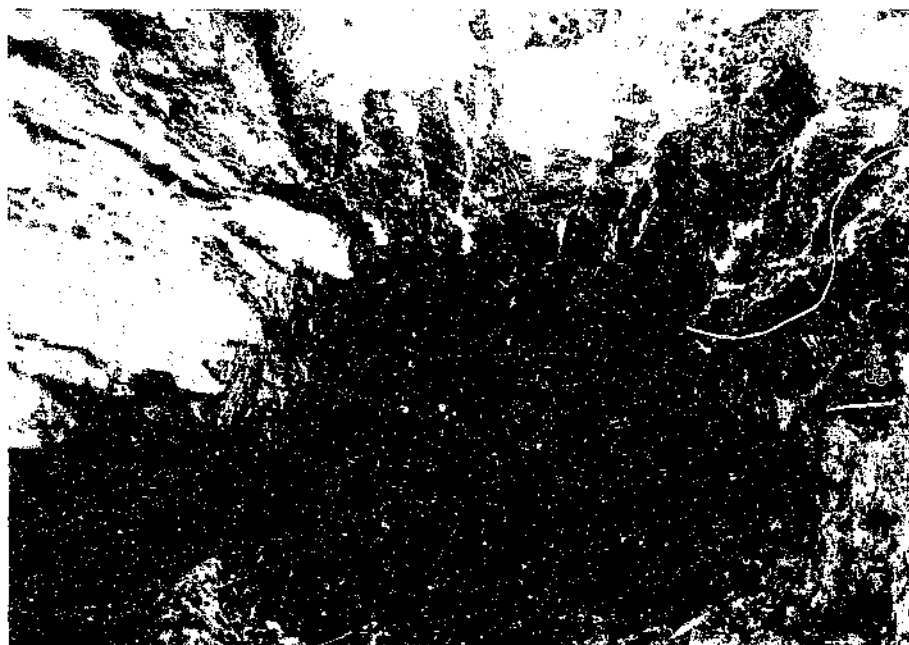


Plate 24 - Pseudo-Breccia Intersected In 72-43W N7 Panel.

flexibility and areal coverage provided by the 200/400kN prop and load-spreader system.

9.2.2 Stratification.

The theory or philosophy of cave mining on Hartebeestfontein Gold Mine is firmly founded on the concept of plastic, or macro-plastic, deformation of a well-bedded rockmass in which the interfaces between individual beds are well lubricated. The concept envisages individual layers, and the rockmass as a whole, "flowing" into the stoping excavation. Strong evidence to support this hypothesis was found in observations of the structure of the rockmass in footwall accessways close to the reef and in holes drilled into the hangingwall, and from interpretation of rock deformation experienced in back areas, centre gullies and raises.

Most lateral closure was found to occur in areas exhibiting a high incidence of bedding with a high degree of lubrication, and vice versa. An effort was made to model this phenomenon, with only limited success as was pointed out earlier. Plate 25 shows the formation of planar rather than curved stress fractures in the sidewalls of 78A-42W Raise Connection prior to ledging taking place. This is indicative of the effect of the argillite which is present on the prominent bedding planes visible in the photograph. Gross lateral displacement of the sidewalls into the raise, estimated to total at least 1m, led to the installation of grouted steel rods in an effort to contain the movement. Plate 26 shows buckling of the hangingwall over a centre gully in response to horizontal movement. This phenomenon is fairly common throughout the mine and necessitates the widespread bolting of most raises and centre gullies. Plate 27 shows pseudo-plastic deformation in a footwall crosscut, the mechanistic explanation of which is essentially similar to that for raises and centre gullies, and which is also very common.

9.3 CAVING HISTORY.

Two developments in gold mining practice on the Witwatersrand during the 1940's had an impact on the later evolution of the cave mining method on mines in the Klerksdorp district. The first of these was the recognition of the strata control benefits of installing stiff, columnar support types, such as timber sticks, at the working face in place of non-rigid or yielding support types, such as packs. The second arose from the need to reclaim timber pack supports from the back area of operating panels to relieve the wartime shortage of timber and to reduce support costs.

9.3.1 Early Observations.

It was observed that the resultant caving of the back area when packs were removed, reduced stress at the face and improved stope width and hangingwall control, which is in keeping with modern observations. It was also observed that the footwall in the unsupported cave area showed signs of bulging. This phenomenon resulted from large scale plastic deformation and displacement of strata towards the stoped out area, and is commonly observed in caving stopes at Hartebeestfontein Gold Mine when adequate footwall slots are not provided.

Observations of the orientation of fractures in the hangingwall were however somewhat at odds with more recent determinations. It was reported that hangingwall fractures exhibited a gradual but dramatic change in direction and intensity from near vertical and closely spaced



Plate 25 - Stratification And Planar Fractures In The Sidewall Of The 78A-42W Raise Connection Prior To Ledging.



Plate 26 - Buckling Of The Hangingwall Over A Centre Gully.



Plate 27 - Pseudo-Plastic Deformation In A Footwall Crosscut.



Plate 28 - Failure To Cave In An Unsupported Back Area.

before caving operations began, to widely spaced, low angle, almost horizontal, features dipping towards the face. This contrasts with current caving experience at Hartebeestfontein Gold Mine where the achievement of good caving conditions is demonstrated by a change in induced hangingwall fracture orientation from shallow angled features dipping towards the face, to fractures dipping steeply away from the face. In addition, shallow angled fractures dipping towards the face are indicative of hangingwall instability caused by high horizontal stresses normally associated with failure to achieve a cave. Early photographs taken in caving stopes at Hartebeestfontein do, however, sometimes show stress fractures dipping towards the face with no apparent adverse effects.

It was recognised that increased rates of face advance, brought about by concentrated mining methods, which were also being introduced at about that time, decreased the time-dependent loosening or unravelling of the hangingwall blocks and improved hangingwall stability. This corresponds closely with present day observations which indicate that hangingwall conditions, and thus support requirements, are favourably influenced by the achievement of a rapid rate of face advance.

In many cases it was noted that caving occurred long after back area support had been removed, and in a few cases caving never actually took place at all. Pressure on the face, however, decreased almost as soon as the first two or three rows of packs were blasted out, and did not appear to depend upon actual caving taking place. Parallels can be drawn between these observations and present day experience on Hartebeestfontein Gold Mine. Often caving is not achieved, and large expanses of the back area remain open and unsupported with little noticeable adverse effect on face condition, fracture orientation and support stability. Plates 28, 29 and 30 are photographs, taken in the No.5A Shaft area, showing a variety of essentially unsupported back areas which have failed to cave, or in which the cave has hung-up or been delayed by minimal back area support. In all cases face conditions did not appear to be adversely affected.

This is, however, not always the case. In many instances failure to cave results in a marked deterioration in operating conditions, heralded by the well defined indicators described earlier in the report. Initially, it was reported, attempts were made to induce caving in stopes at Sub Nigel which failed to cave naturally by drilling and blasting the hangingwall. In not one single instance were these attempts successful. On the other hand, at Stilfontein Gold Mine, where modern prop caving was pioneered, and indeed at Hartebeestfontein Gold Mine, inducement by blasting of a cave which did not occur naturally was generally considered an integral and vital part of the cave mining operation. Any tendency of the cave to hang-up caused unsafe working conditions, which could immediately be rectified by inducing even a superficial cave involving less than 1m of hangingwall.

The reasons for the dichotomies noted above remain unclear, but it seems reasonable to suppose that differences in the precise nature of the surrounding strata, and particularly the hangingwall strata, may have played an important role.

There is almost universal agreement in the literature, however, on the importance of providing an adequate density of support at the face, of keeping this support as close to the face as possible, and of minimising the strike distance back from the face over which the hangingwall is supported. Timber and steel props have proved to be admirably suited to meeting these criteria over many years, and continue to be the primary means of stope face support in deep gold mines.

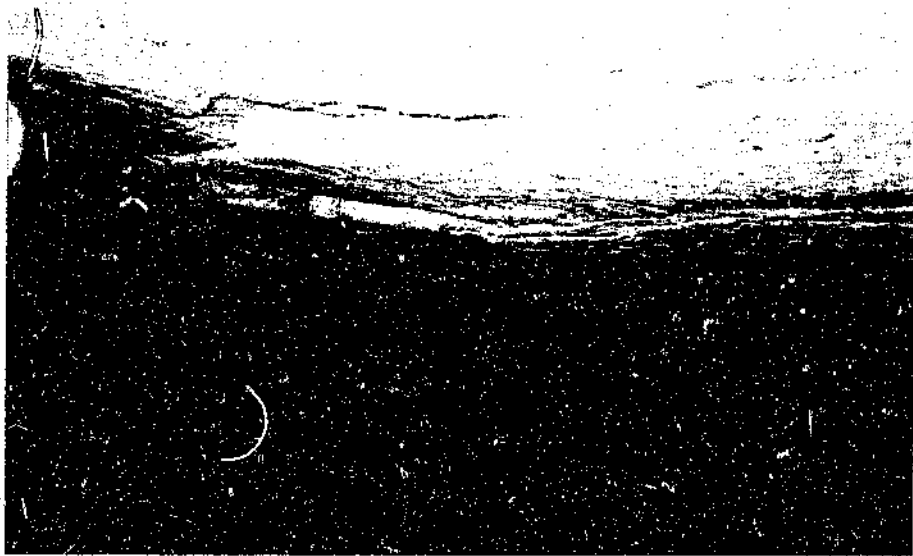


Plate 29 - Caving Hung Up By Minimal Support In The Back Area.



Plate 30 - Failure To Cave In An Essentially Unsupported Back Area.

9.4 STRATA CONTROL.

The development of the "new" support strategy described in this report was undertaken with the objectives mentioned above in mind, and was thus firmly founded on concepts and criteria propounded decades ago. The real development may therefore be described, not so much as the application of new concepts, but rather the advancement of support and cleaning technology which allows the implementation of these old concepts on a much more rigorous level.

9.4.1 The Support System.

Investigations by the Chamber of Mines Research Organisation showed that 90% of fall of ground accidents in the stope face area could be prevented by applying a fairly moderate support resistance of only 50kN/m², provided that this is spread as evenly as possible over the working area, and is applied as close to the face as possible. The adoption of this support philosophy led to the development of a stope support system which incorporates the following components:-

- a) **200/400kN Props.** Because the performance demanded from props is now substantially reduced, units can be made lighter, facilitating ease of transport and installation.
- b) **Blast Protection Sleeves.** The development of an effective blast protection sleeve allows the installation of prop units less than 1m from the face.
- c) **Triangular Load-Spreaders.** The unique design of the load-spreader, which was patented by the Chamber of Mines Research Organisation, ensures an even spread of prop load and a reduction in unsupported spans.
- d) **Head Extensions.** Because of their inherent stability, conical extensions fitted to conical head props allow 550mm props, which are suitable for the maintenance of very low stoping widths, to accommodate a wide and variable range of stoping widths.
- e) **Waterjets.** Cleaning and sweeping operations have been revolutionised by the introduction of waterjetting, which has also facilitated the use of diagonal blast barricades.
- f) **Diagonal Barricades.** In combination with waterjetting, diagonal barricades allow blast-on hydraulic prop support to be installed less than 1m from the face before blasting.

The integration of blast-on props, load-spreaders and waterjetting into the cave mining operation has required only fairly minor adaptations to existing cave mining practice. Four rows of props are required instead of three to provide adequate space and protection for the waterjet operator. This fourth row may be doubled-up in friable hangingwall conditions to reduce the possibility of a cave over-run. The effect of the potentially increased resistance to macro-plastic "flow" of the hangingwall and footwall in the caving environment occasioned by the increased number of rows of hydraulic props is off-set by the lower support resistance offered by the 200kN props in comparison with conventional 300kN props, and the reduction

in the total supported span brought about by installing the support closer to the face.

Two major advantages flow from the application of this system on Hartebeestfontein Gold Mine. Firstly, and perhaps most importantly, the safe extraction of ore reserves, which would otherwise be difficult and even dangerous to extract, is facilitated. Secondly, the reduction in stoping widths achieved with the use of the system has had a substantial impact on the profitability of the mine as a whole, and has opened up reserves which may otherwise not have been considered viable. Plate 31 shows the original change-over from 40ton props (which had all in fact been "de-tuned" from 400kN to 300kN yield load to reduce punching experienced in some weak footwall areas) to the new 200/400kN prop system. The flexibility of the system in accommodating a wide range of stoping widths without forgoing stability is demonstrated in Plate 32. The use of the props in a caving environment is shown in Plates 33 and 34, while Plates 35 and 36 show their use with waterjet cleaning.

9.4.2 Drilling and Blasting.

The major contribution of sound drilling and blasting practice to the attainment of stable hangingwall conditions is invariably overlooked. Some attention was paid to this aspect during the course of the investigation, but little was mentioned in the report because the subject fell outside the scope of the project. It is nevertheless a vitally important issue which could profitably receive more attention. Plate 37 demonstrates the major role of drilling and blasting discipline in contributing to stope stability. Incorrectly drilled and excessively burdened blast holes invariably lead to unnecessary blasting damage and loss of control of the hangingwall, particularly in weak ground conditions.

9.4.3 Backfill Alternative.

Serious consideration was directed at backfill as an alternative to caving in weak ground conditions prior to and during the course of the development of the hydraulic prop and waterjetting support system. After thorough investigation the backfill alternative was shelved for the following reasons:-

- a) Face support requirements were anticipated to be the same for both options in very weak ground conditions.
- b) Prevailing shallow dips were considered to impact negatively on the potential to place backfill effectively.
The high frequency of faulting was also considered to have a potentially negative impact on filling efficiency.
- d) The calculated additional capital and operating cost of backfill was high, especially in comparison with caving related support systems.
- e) It was argued that money required to provide backfill could be better spent on improving face support density.
- f) Seismicity was not expected to be a problem in the weak ground conditions of the 5A North area.



Plate 31 - Change-Over From 300kN Props To 200/400kN Props.



Plate 32 - Flexibility Of 200/400kN Conical Head Props And Extensions In Coping With Variations In Stopping Width.



Plate 33 - Caving With 200/400kN Props, Looking Down-Dip.

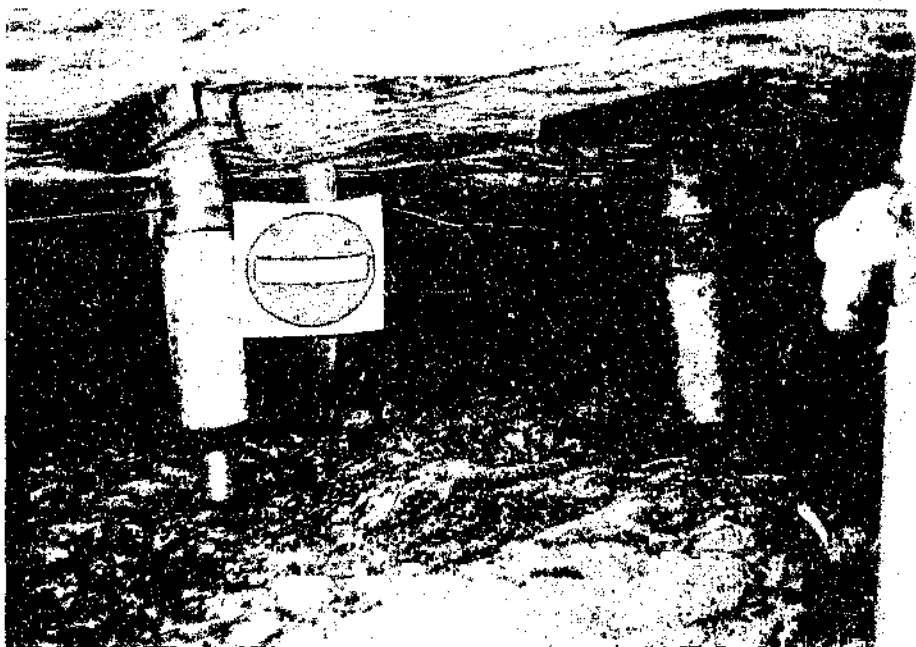


Plate 34 - Caving With 200/400kN Props, Looking Back Along Strike.



Plate 35 - Waterjet Cleaning With The 200/400kN Hydraulic Prop And Diagonal Blast Barricade System.



Plate 36 - Waterjet Cleaning With The 200/400kN Hydraulic Prop And Diagonal Blast Barricade System.

The development of a viable prop-based support system which could be applied with caving finally sounded the death-knell for backfill on Hartebeestfontein Gold Mine.

Very preliminary evaluation of the environmental conditions required for successful backfilling and for successful caving, based on numerous underground observations made on Hartebeestfontein Gold Mine and on the neighbouring Vaa Reefs mine, would seem to indicate that the application of these two stoping techniques are mutually exclusive. The implementation of backfilling in a stoping environment which is conducive to macro-plastic flow, and which would thus lend itself to caving, results in the marked deterioration of hangingwall conditions at the face caused by a build-up of horizontal stress in much the same fashion as occurs in caving panels when the cave hangs up. Conversely, the application of caving in stoping environments which do not exhibit macro-plastic behaviour, and which invariably benefit by the introduction of backfill, could have disastrous consequences. Much more investigation is however required to support this hypothesis.

9.5 CAVING EVALUATION.

9.5.1 Instrumentation.

Although a great deal of money, time and effort was directed at this aspect of the project, the results were somewhat disappointing. For example, many frustrating hours were spent trying to measure stress in the hangingwall of cave stopes in the pseudo-breccia zone. Of the more than forty attempts, only eight doorstoppers were recovered with core still attached. Although these achievements were cause for much celebration underground, no definitive stress estimations could be made from the strain measurements which were recorded. When it is considered that the cost associated with this part of the project conservatively exceeded R50 000, the results were indeed disappointing.

A further example was provided by the decision to delay installation of the extensometer tapes at the 78A-42W N7 Gully site until after stress measurements had been taken. Stress measurements took much longer to complete than originally anticipated, during which time the stope face continued to advance away from the instrumentation site. A major breakdown of a main shaft hoist then resulted in total cessation of production operations for a period of three weeks. Had the extensometer tapes been in place, this would have provided unique and invaluable information regarding the time-dependant deformation of hangingwall strata under conditions of general and prolonged stand-still, to be directly contrasted with similar information obtained during periods of normal face advance. As it was, the extensometer was not installed in time and this opportunity was lost.

Nevertheless, much valuable information was gathered concerning the structure, quality and movement of the hangingwall strata, allowing a simplified model to be constructed. A course of action considered in the early stages of the project, but discarded for want of financial, human and time resources, was that of excavating and instrumenting a vertical box ahead of an advancing panel so as to record the range of deformations undergone by hangingwall and footwall strata as a stope approaches, passes by, and then recedes, including of course the actual effect of the cave. This would have provided information with which to compare and calibrate the dynamic caving model.

Moderately high cave pressures were measured by load cells installed in two of the test

panels. More than 1MPa was registered after a relatively short period of two months, during which time the stop faces had advanced some 40m. While these results are lower than pressures monitored at some backfill sites, they are, nevertheless, equivalent to the dead-weight load of a 40m height of hangingwall rock, more than would be expected to be involved in the collapse of a cave.

The reasons for these relatively high values are not fully understood at present, but it is speculated that the primary mechanism which drives closure, ie. differential horizontal movement of undulating strata, may play a large part in the re-generation of stress through the caved area. On the other hand, the cave pressure instrumentation may be recording point loads associated with the small layer of rock packed over the flat-jack protection plates during installation to lessen the shock impact of the initial cave. The fabrication of additional cave pressure instrumentation is planned and it is intended to pursue the matter further.

9.5.2 Modelling.

Discussion of the mechanisms or mechanics of caving in tabular deposits at depth involves some reference to macro-plastic or pseudo-plastic deformation of the rockmass surrounding stoping excavations. The ability of researchers or commentators to accurately and reliably model this type of non-linear behaviour has been, and remains, limited. Such models are by their nature complex, requiring cognisance to be taken of the time-dependant nature of this type of deformation. While non-linear computer modelling facilities are becoming more readily available, they are, relatively speaking, still in their infancy. Some very useful models were generated using the UDEC program, and many important insights concerning the mechanisms of caving were gained or reinforced. The similarity that the dynamic caving model bore to some, if not all, underground prop caving situations is amply illustrated by Plates 38, 39 and 40.

However, the inelastic modelling programs used do have much greater potential than was realised during the course of this project, and much work remains to be done. In retrospect, more attention should have been paid during the development of, particularly, the dynamic caving model to the question of representing more accurately the fracture processes ahead of the face, and the concomitant generation of horizontal stress and displacement. Current versions of the UDEC program are incapable of simulating the actual fracture process, but are capable of modelling post-failure strain-softening states to a higher level of sophistication. Run times are naturally of a significantly higher order, but it is anticipated that technological developments in the field of computing hardware will make this aspect less onerous in the near future.

Likewise, it was not possible to address the questions of the extent of the cave, the effect of bulking, or the regeneration of stress in caved material in any meaningful way because of the limitations in the modelling technique mentioned previously. Also, the question of stress redistribution at and ahead of stoping abutments concurrently with the occurrence of the initial cave was not satisfactorily dealt with. The model showed little or no change, but this conclusion cannot be regarded as convincing for the reasons referred to above. Once again, these issues will provide a fruitful and useful field for future investigation, but will require substantially more modelling time and a greater degree of model complexity than could be allocated or achieved during the course of this study.



Plate 37 - Blast Damage Caused By Over-Burdened Holes In Weak Hangingwall Conditions.



Plate 38 - View Showing Good Correlation Between An Actual Back Area Cave And The Dynamic Caving Model.



Plate 39 - Fractures In The Hangingwall Above The Cave.

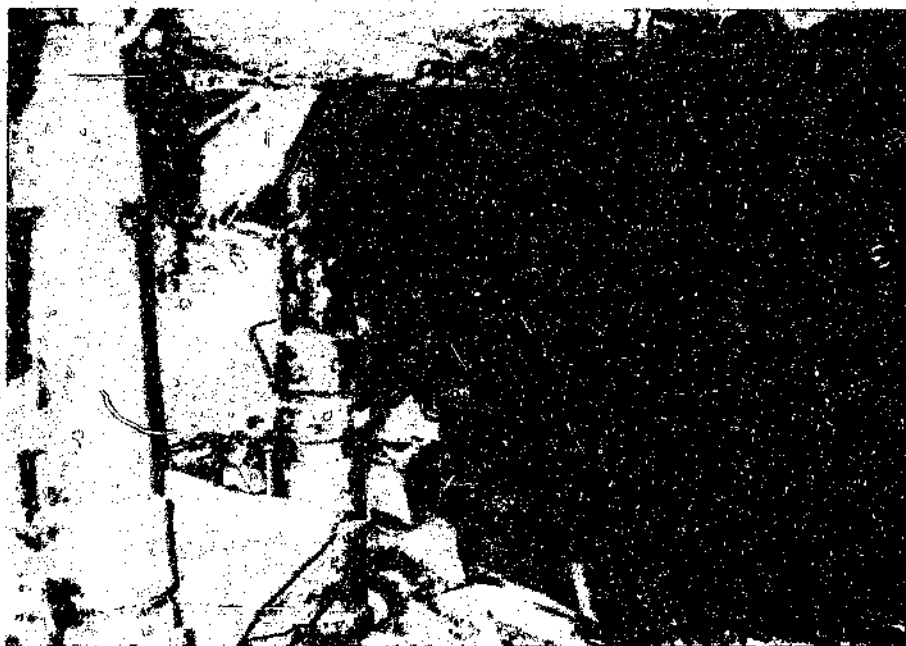


Plate 40 - Caving Right Up To The Back Row Of Props.

9.6 SUMMARY.

The results of the work done to investigate and further the understanding of the mechanisms of back-area caving in general, and to evaluate caving and address the question of stope support in weak hangingwall conditions in particular, may be summarised as follows:-

- a) The suitability of applying the back-area cave mining method in any situation is entirely dependant upon the presence of specific rockmass characteristics. These include, firstly, the presence of moderately laminated strata in which the individual layers are separated by weak filling which allows shear movement to occur readily, and, secondly, a relative weakness of the intact rock material which allows failure in the high stress environment ahead of the advancing stope face to be accompanied by a large degree of dilation. These features must combine to result in plastic, time-dependant behaviour of the rockmass in which the maximum sustainable stress at the face is limited by the ability of the rock to "flow" and dissipate any stress increase above this level, while at the same time continually generating sufficient horizontal clamping stress in the hangingwall strata above the working area to ensure stability without causing hangingwall failure.
- b) The degree to which a laminated rockmass conforms to "true" plastic behaviour will determine, firstly, the applicability of the cave mining method and, secondly, the actual support and caving requirements. For example, a rockmass which is characterised by predominantly elastic behaviour (such as strong, massive, quartzite) will not be suitable for back-area caving because of the inability of the rock to deform plastically. In fractured rock situations, caving will inevitably result in the loss of clamping force at the face which will lead to instability and even collapse of the working area. "Back-break" could occur in shallow, intact-rock environments where back area support is inadequate. On the other hand, a rockmass characterised by predominantly plastic behaviour (such as potash) will flow naturally into any excavation made in it, and will require yielding, areal support only at the working face. A further prerequisite will be that the working area advances at a rapid and constant rate to escape time-dependant deterioration.
- c) Between these two extremes lies the range of pseudo-plastic environments encountered on Hartebeestfontein Gold Mine. Hangingwall and footwall slots, and the achievement of a cave, are required to encourage rock flow in most cases. Yielding support is required at the face to regulate this process and to maintain safe working conditions. A moderate rate of face advance is also required to prevent time-dependant deterioration. The actual support resistance and rate of face advance required for each cave stoping situation will depend upon the nature of that particular rockmass environment and where it slots in on the "scale" of pseudo-plastic behaviour.
- d) It thus follows that, in weak ground conditions, caving can be safely applied provided areal support at the face is intensified and a continuous and rapid rate of face advance is maintained. Areal support coverage was increased in the 5A North area by reducing inter-support and face-to-support distances and by implementing the use of load-spreaders. The lower slow-yield support resistance offered by the individual 200/400kN hydraulic prop units off-set the overall increase in support resistance which would otherwise have accompanied these changes. An overall increase in face advance was also facilitated by the introduction of the 200/400kN hydraulic prop and

waterjet cleaning system. Plate 41 illustrates the consequences of not achieving an adequate rate of face advance.

- e) Taking the salt mine analogy one step further, it may be tentatively concluded that the necessity of achieving actual back-area caving to maintain plastic deformation processes become less important with decreasing rock strength. This could help to explain the contradictory observations made by various commentators to the effect that, on the one hand "the sooner the cave takes place, the better", and on the other that "it does not make much difference whether the back area caves or not". Regular caving was by and large achieved in the experimental panels. On the occasions where this was not the case, no marked deterioration in face conditions was evident.
- f) Having identified the mechanisms and processes which are required to sustain caving, and thus the conditions under which caving can and cannot be considered a viable stoping option, it should become possible to evaluate potential stoping areas to determine their suitability for caving based on definitive characterisation of the surrounding rockmass. This information could be processed using non-linear modelling techniques to verify the mining method and to determine precise support and face advance requirements for individual situations with greater confidence. This approach may also be extended to determine the viability and strata control implications of backfilling. The scope for future work in this field is considerable.
- g) Cave mining is currently more of an art than a science, for, although not quite balanced on a proverbial knife edge, the method does nevertheless require the attentions of a skilled operator for its successful implementation. Caving relies heavily on continual face advance. "Old-timers" testify that stopped faces require support right up to the face, and even so invariably collapse on being restarted, necessitating re-development in the solid. This view is supported by contemporary experience. Once the "engine" ahead of the face supplying horizontal loading to the stope hangingwall is switched off, clamping stress dissipates and stability is lost.

In brief, the conclusions regarding the evaluation of cave mining in weak hangingwall conditions, specifically those encountered in the pseudo-breccia zone prominent in the 5A North area of Hartebeestfontein Gold Mine, are as follows:-

- a) The viability of the cave mining method in tabular deposits is highly dependant upon the presence of specific geological conditions for its success.
- b) The method is thus not suitable for general application throughout the South African gold mining industry, but could be considered where conditions are favourable.
- c) A continuous and moderate rate of face advance is an important requirement for the successful implementation of the cave mining method.
- d) The decision to persevere with caving in the 5A North area, and to concentrate on intensifying areal support coverage at the face while at the same time reducing the support resistance of individual units, was correct.
- e) Substantial spin-offs in the form of reduced stoping widths and improved grades were realised with the introduction of the 200/400kN blast-on prop system. Further benefits may be brought about by improving drilling and blasting techniques.

- f) In retrospect, it is not considered that backfill would have provided an acceptable solution to the problem of weak ground in the pseudo-breccia zone.
- g) It may be possible, through definitive characterisation of the rockmass, combined with non-linear computer modelling, to predict the suitability of caving or of backfilling for specific mining areas. Greater investigation of this topic is required.

As a final conclusion in regard to mining without back area support, it is suggested that while the actual attainment of a cave in weak ground conditions, similar to those pertaining in the 5A North area, is desirable, it is not of overriding importance. Of greater consequence is the provision of an adequate intensity of support, spread evenly over the entire surface of the hangingwall and kept as close to the face as possible, and the attainment of a continuous and sufficiently rapid rate of face advance.



Plate 41 - Reverting To Pack Support To Deal With Deteriorating Conditions Brought About By Inadequate Face Advance.



Plate 42 - Some Of The People To Whom This Project Report Is Dedicated With Affection And Gratitude.

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APPENDIX 1. HYDRAULIC PROPS.**APPENDIX 1.****200/400kN BLAST-ON HYDRAULIC PROPS.****PROP PRESSURE/FORCE RELATIONSHIP.**

Differences in the internal dimensions of the various makes of prop give rise to a variation in the pressure required to set Klockner-Becorit, Elbroc and Unique props respectively. This results in a lowering of allowable setting pump pressure tolerances when Klockner and Elbroc, and Elbroc and Unique props are mixed in the same panel, and results in a total erosion of tolerance when Klockner and Unique props are mixed.

	PROP TYPE		
	KBC	ELB	UNI
Piston Diameter (mm)	85,0	80,0	72,7
Cylinder Area (m ²)	,0057	,0050	,0042
Pressure at 120kN Setting Force (MPa)	21,2	23,9	28,9
Pressure at 160kN Setting Force (Mpa)	28,2	31,8	38,5
Pressure at 170kN Yield Force (Mpa)	30,0	33,8	41,0
Pressure at 200kN Yield Force (Mpa)	35,3	39,8	48,2
Setting Force at 21,2MPa (KBC Min.)	120	107	88
Setting Force at 23,9MPa (Elb Min.)	136	120	99
Setting Force at 28,9MPa (Uni Min.)	164	145	120
Setting Force at 28,2MPa (KBC Max.)		142	117
Setting Force at 31,8MPa (Elb Max.)	181	160	132
Setting Force at 38,5MPa (Uni Max.)	218	194	160

No single pump pressure can be used with all three makes of prop without violating the Chamber of Mines Research Organisation setting load specification of between 120kN and 160kN.

The allowable pump pressure tolerance for use with a mix of Klockner-Becorit and Elbroc props is restricted to between a maximum of 28,2MPa and a minimum of 23,9MPa, and the

allowable pump pressure tolerance for use with a mix of Elbroc and Unique props is restricted to between a maximum of 31,8MPa and a minimum of 28,9MPa.

It will not be possible to set Klockner-Becorit and Unique props using a single setting pressure. The pressure required to set a Unique prop at the minimum allowable setting force of 120kN is 28,9MPa. This pressure will set a Klockner-Becorit prop at a force of 164kN, which is in excess of the maximum called for by the Chamber of Mines Research Organisation specification. More importantly, this is very close to the 30MPa level required to operate Klockner-Becorit yield valves set to yield at the legitimate minimum slow yield force of 170kN. Raising the pump pressure above this value will cause large volumes of pressurising fluid to be by-passed through Klockner-Becorit yield valves, with possible resultant damage. Conversely, if the pump delivers a pressure significantly less than 29MPa, Unique props will be set at a support force of less than 120kN.

In retrospect, the specifications for 200/400kN hydraulic props should have included maximum and minimum limits in respect of the bore of props, or should have indirectly set these by specifying the relationship between internal pressure and external load generation, to facilitate compatibility between various prop types.

APPENDIX 2. HANGINGWALL STRUCTURE.

APPENDIX 2A(I).

HANGINGWALL MAPPING - 72-43W N7 PANEL (23/11/92).

[illegible]

APPENDIX 2. HANGINGWALL STRUCTURE.

APPENDIX 2A(iii).

HANGINGWALL MAPPING - 78A-42W N6 PANEL (15/4/93).

[illegible]

APPENDIX 2. HANGINGWALL STRUCTURE.

APPENDIX 2B(i).

BOREHOLE CAMERA SURVEY - 72-43W N8 GULLY (21/12/92).

DEPTH (mm)	SPACE (mm)	DISCON TYPE	HORIZ ORIENT	DIP	WIDTH (mm)	SURF TEXT	FILLING
0	Vaal Reef Horizon						
2000	Hangingwall Horizon, Start of Hole						
2300	300	Fracture	Dip	75°	3	Rough	Open
2370	70	Fracture	-	Horiz	5	Rough	Open
2460	90	Fracture	-	h	2	Rough	Open
3180	720	Joint	-	Horiz	-	-	Closed
3300	120	Bedding	-	Horiz	50	Smooth	Argillite
3500	200	Joint	-	Horiz	-	-	Closed
3700	200	Bedding	-	Horiz	2	Smooth	Open
5100	1400	Fracture	Dip	75°	-	-	Closed
5300	200	Band	-	Horiz	20	-	-
5400	100	Joint	Dip	70°	-	-	Closed
5650	250	Discont	-	Horiz	2	Smooth	Open
5680	30	Discont	-	Horiz	2	Smooth	Open
5800	120	Vein	-	20°	5	-	-
6800	1000	Stress	-	Horiz	15	Rough	Powdered
7400	600	Discont	-	Horiz	1	-	Closed
7810	410	Discont	-	Horiz	-	-	Closed
7900	90	Fracture	Dip	75°	-	-	Closed
8720	820	Fracture	Dip	75°	1	-	Closed
9550	830	Fracture	Dip	75°	-	-	Closed
9600	50	Band	-	Horiz	15	-	-
9650	50	Fracture	Dip	65°	1	-	Closed
10350	700	Joint	-	Horiz	-	-	Closed
10560	210	Bedding	-	Horiz	-	-	Closed
11000	440	Band	-	Horiz	25	-	-

DEPTH (mm)	SPACE (mm)	DISCON TYPE	HORIZ ORIENT	DIP	WIDTH (mm)	SURF TEXT	FILLING
11140	140	Fracture	Dip	75°	3	Rough	Open
11200	60	Fracture	Dip	90°	2	Rough	-
11220	20	Bedding	-	Horiz	10	Smooth	Argillite
12100	880	Vein	-	75°	20	-	-
12200	100	Fracture	Dip	10°	-	-	-
12300	100	Fracture	Dip	70°	2	-	Closed
12390	90	Fracture	-	Horiz	2	-	-
12420	30	Fracture	Dip	45°	2	-	Open
12460	40	Bedding	-	Horiz	20	Smooth	Argillite
12470	10	Fracture	Dip	45°	2	-	-
12530	60	Bedding	-	Horiz	25	-	Open
12700	170	Vein	-	20°	2	-	-
12780	80	Fracture	-	10°	-	-	Closed
13100	320	Bedding	-	Horiz	30	-	Open
13200	100	Discont	-	Horiz	3	-	-
13220	20	Discont	-	Horiz	3	-	-
13240	20	Discont	-	Horiz	3	-	-
13340	100	Discont	-	Horiz	3	-	-
13370	30	Fracture	Dip	45°	3	Rough	Closed
13750	380	Band	-	Horiz	50	-	-
13830	80	Fracture	-	10°	3	-	-
14100	270	Fracture	-	10°	10	-	-
14180	80	Zone	-	Horiz	15	Rough	Pebbles
14260	80	Fracture	-	70°	1	-	Closed
14860	600	Zone	-	Horiz	50	-	Pebbles
14940	80	Fracture	-	Horiz	15	-	Open
15450	510	Fracture	-	10°	2	-	-
15490	40	Fracture	-	10°	2	-	-

APPENDIX 2. HANGINGWALL STRUCTURE.

APPENDIX 2B(ii).

BOREHOLE CAMERA SURVEY - 78A-42W N7 GULLY (20/4/93).

DEPTH (mm)	SPACE (mm)	DISCON TYPE	HORIZ ORIENT	DIP	WIDTH (mm)	SURF TEXT	FILLING
0	Vaal Reef Horizon						
1000	Hangingwall Horizon, Start of Hole						
1900	900	Fracture	Dip	70°	1	-	Closed
2000	100	Bedding	-	Horiz	50	Smooth	Open
2200	100	Fracture	-	Horiz	3	Rough	Open
2250	50	Fracture	-	Horiz	5	Rough	Open
2280	30	Fracture	Dip	75°	2	-	Closed
2300	20	Bedding	-	Horiz	25	Smooth	Open
2500	200	Joint	-	Horiz	-	-	Closed
3600	1100	Fracture	Dip	60°	2	-	Closed
3750	150	Bedding	-	Horiz	20	Smooth	Argillite
4100	350	Fracture	Dip	75°	-	-	Closed
4500	400	Band	-	Horiz	30	-	-
4650	150	Joint	Dip	70°	-	-	Closed
4700	50	Discont	-	Horiz	2	Smooth	Open
5600	900	Bedding	-	Horiz	25	Smooth	Argillite
5690	90	Fracture	-	Horiz	5	Rough	Open
5710	20	Fracture	Dip	71°	2	Rough	Open
6200	490	Discont	-	Horiz	2	-	Closed
6600	400	Discont	-	Horiz	2	-	Closed
7040	1440	Bedding	-	Horiz	15	Smooth	Argillite
10000	1960	Discont	-	Horiz	2	-	Closed
10300	30	Discont	-	Horiz	2	-	Closed
10220	190	Discont	-	Horiz	1	-	Closed
10400	180	Discont	-	Horiz	2	-	Closed
10550	150	Discont	-	Horiz	1	-	Closed

APPENDIX 2. HANGINGWALL STRUCTURE.

APPENDIX 2B(iii).

BOREHOLE CAMERA SURVEY - 78A-42W CENTRE GULLY (20/4/93).

DEPTH (mm)	SPACE (mm)	DISCON TYPE	HORIZ ORIENT	DIP	WIDTH (mm)	SURF TEXT	FILLING
0	Vaal Reef Horizon						
1000	Hangingwall Horizon, Start of Hole						
1250	250	Fracture	Dip	65°	10	Rough	Open
1300	50	Fracture	-	Horiz	10	Rough	Open
1700	400	Bedding	-	Horiz	50	Smooth	Argillite
1840	140	Discont	-	Horiz	3	-	Closed
1900	60	Discont	-	Horiz	1	-	Closed
2060	160	Discont	-	Horiz	2	-	Closed
2150	90	Discont	-	Horiz	2	-	Closed
2300	150	Bedding	-	Horiz	25	Smooth	Argillite
3220	920	Bedding	-	Horiz	10	Smooth	Argillite
3880	660	Fracture	Dip	Vert	2	-	-
4070	190	Discont	-	Horiz	1	-	-
4550	480	Discont	-	Horiz	1	-	-
4681	130	Discont	-	Horiz	1	-	-
4840	160	Discont	-	Horiz	2	-	-
4880	40	Discont	-	Horiz	1	-	-
4940	60	Discont	-	Horiz	1	-	-
5000	60	Discont	-	Horiz	1	-	-
5090	90	Discont	-	Horiz	1	-	-
5330	240	Discont	-	Horiz	1	-	-
5500	170	Discont	-	Horiz	1	-	-
5840	100	Discont	-	Horiz	1	-	-
5940	100	Discont	-	Horiz	1	-	-
6370	430	Discont	-	Horiz	1	-	-
6680	310	Discont	-	Horiz	1	-	-

APPENDIX 2. HANGINGWALL STRUCTURE.

APPENDIX 2C(i).

DRILL CORE FRACTURE LOG - 72-43W N8 GULLY (11m HOLE).

DISTANCE (mm)	SPACING (mm)	DISCONTINUITY DESCRIPTION
0		Vaal Reef Horizon
2000		Start of Hole
2000		Start of Core
2000 - 2400	(400)	Broken pieces: 10-30mm; horizontal fractures.
2500 - 2580	180	Clean 70° fracture.
2600 - 2790	(210)	Broken fragments; horizontal, 70° & vertical fractures.
2790	-	Clean, horizontal fracture.
2800 - 2880	90	Clean, 70° fracture.
2900	20	Ground surface.
2950 - 3000	100	Clean, 45° fracture.
3050	50	Clean, horizontal fracture.
3130	80	Rough break.
3190	60	Ground surface.
3290	100	Grout-filled (bedding) discontinuity.
3320 - 3400	110	Clean, 75° fracture.
3450 - 3500	150	Broken pieces: 10-20mm; horizontal fractures.
3500	-	Ground surface.
3540	40	Ground surface.
3740	200	Rough, horizontal fracture.
3770 - 3820	80	Clean, 45° fracture.
3860	40	Ground surface.
3900	40	Ground surface.
4010 - 4030	130	Clean, 30° fracture.
4080	50	Smooth (bedding) surface.
4260	180	Clean, horizontal fracture.

DISTANCE (mm)	SPACING (mm)	DISCONTINUITY DESCRIPTION
4350 - 4410	150	Grout-filled, 60° fracture.
4450	-	Grout-filled fissure.
4540	90	Ground horizontal surface.
4630	90	Ground horizontal surface.
4740	110	Clean, horizontal fracture.
4850	110	Rough, horizontal fracture.
4930	80	Rough, horizontal fracture.
4990 - 5030	100	Clean, 55° fracture.
5100	110	Very rough, horizontal fracture.
5140 - 5180	80	Clean, 45° fracture.
5320	140	Ground horizontal surface.
5480	160	Clean, horizontal fracture.
5590	110	Ground horizontal surface.
5610	20	Ground horizontal surface.
5640 - 5650	30	Clean, 10° fracture.
5700 - 5750	-	Grout-filled joints.
5780 - 5790	140	Clean, 10° fracture.
5850 - 5870	100	Clean, 20° fracture.
5870 - 6120	250	Ground surfaces: 30-50mm pieces.
6200 - 6280	160	Clean, 75° fracture.
6580 - 6600	320	Rough, 20° fracture.
6780	180	Ground horizontal surface.
6850	70	Ground horizontal surface.
6930 - 6970	120	Rough, 45° fracture.
7100	130	Ground horizontal surface.
7100	Log terminated at the end of the first 5m of core.	

APPENDIX 2. HANGINGWALL STRUCTURE.

APPENDIX 2C(ii).

DRILL CORE FRACTURE LOG - 72-43W N8 GULLY (20m HOLE).

DISTANCE (mm)	SPACING (mm)	DISCONTINUITY DESCRIPTION
0		Vaal Reef Horizon
2000		Start of Hole
2000		Start of Core
2075	75	Rough, 10° fracture.
2200	125	Ground surface.
2270 - 2310	110	Rough, 70° fracture.
2390	10	Ground surface.
2320 - 2390	-	Rough, 80° fracture.
2400 - 2420	-	Rough, 45° fracture.
2420	100	Rough, horizontal fracture.
2460 - 2490	-	Rough, 30° fracture.
2490	70	Ground surface.
2525	35	Rough, horizontal fracture.
2525 - 2595	-	Rough, 80° fracture.
2595	70	Rough, horizontal fracture.
2600 - 2630	-	Grout-filled, vertical fracture.
2630	35	Smooth, horizontal (bedding) surface.
2630 - 2770	-	Rough, vertical fracture.
2770 - 2810	180	Rough, 30° fracture.
2810 - 2880	70	2x 70° closed fractures.
2880 - 2950	70	Rough, grout-filled, 30° fracture.
2950 - 2960	10	Clean, 40° fracture.
2960 - 3040	80	Rough, 80° fracture.
3100	60	Rough, 10° fracture.
3170	70	Rough, 10° fracture.

DISTANCE (mm)	SPACING (mm)	DISCONTINUITY DESCRIPTION
3370	200	Rough, 10° fracture.
3450	80	Rough, 10° fracture.
3500	50	Rough, 10° fracture.
3630 - 3670	130	Rough, 75° fracture.
3720	50	Jagged, closed fracture.
3750	30	Rough, horizontal fracture.
3810 - 3880	130	Rough, 75° fracture.
3905	25	Rough, 10° fracture.
4010	105	Rough, 20° fracture.
4010 - 4100	(90)	Broken pieces: 20-30mm; horizontal fractures.
4140 - 4180	80	Jagged, 45° fracture.
4250	70	Rough, 15° fracture.
4290	40	Rough, horizontal fracture.
4360 - 4370	80	Clean, 15° fracture.
4450	80	Clean, 10° fracture.
4510	60	Rough, 5° fracture.
4600	90	Rough, horizontal fracture.
4620 - 4680	-	Rough, vertical fracture.
4680	60	Rough, horizontal fracture.
4680 - 4740	60	Rough, 80° fracture.
4850	110	Smooth, horizontal (bedding) surface.
4850 - 4950	-	Vertical, calcite-filled joint.
4950	100	Ground surface.
4950 - 5220	(270)	Broken pieces: 20-40mm, grout-filled.
5290	70	Rough, 20° fracture.
5400	110	Rough, horizontal fracture.
5400 - 5550	(150)	5x 25mm pieces with 45° fractures.
5550 - 5600	50	Rough, 45° fracture.

DISTANCE (mm)	SPACING (mm)	DISCONTINUITY DESCRIPTION
5800	200	Clean 10° break on change of facies.
5800 - 5860	(60)	3x 20mm discs; rough, horizontal fracture.
5910	50	Clean, horizontal fracture.
5960	50	Clean, horizontal fracture.
6120	160	Ground surface.
6185	65	Rough, 5° fracture.
6185 - 6245	(60)	Small fragments.
6300 - 6340	95	Jagged, 25° fracture.
6380	40	Rough, horizontal fracture.
6440 - 6490	-	Grout-filled, vertical fracture.
6490 - 6520	-	Grout-filled, 60° fracture.
6520	140	Rough, horizontal fracture.
6690	170	Rough, horizontal fracture.
6720	30	Rough, horizontal fracture.
6760	40	Rough, horizontal fracture.
6830	90	Rough, horizontal fracture.
6920	70	Rough, horizontal fracture.
6960	40	Rough, horizontal fracture.
7030	70	Rough, 25° fracture.
7030	Log terminated at the end of the first 5m of core.	

APPENDIX 2. HANGINGWALL STRUCTURE.

APPENDIX 2C(iii).

DRILL CORE FRACTURE LOG - 78A-42W N7 GULLY (22m HOLE).

DISTANCE (mm)	SPACING (mm)	DISCONTINUITY DESCRIPTION
0		Vaal Reef Horizon
1000		Start of Hole
1000		Start of Core
1090	90	Rough, horizontal fracture.
1155	65	Rough, horizontal fracture.
1155 - 1250	(95)	Broken pieces: 10-20mm, horizontal fractures.
1280	30	Rough, 20° fracture.
1280 - 1340	-	2x rough, vertical fractures.
1340	60	Ground surface.
1350	20	Ground surface.
1380 - 1460	-	Closed, 70° fracture.
1460	100	Rough, horizontal fracture.
1460 - 1510	(50)	Broken pieces: 45° - 80° fractures.
1510	-	Rough, horizontal fracture.
1610	100	Grout-filled, 10° fracture.
1720	110	Rough, clean, 10° fracture.
1720 - 1910	(190)	Broken pieces: 30°, 60° & 70° fractures.
1960	50	Jagged, horizontal fracture.
2010	50	Rough, horizontal fracture.
2060	50	Rough, horizontal fracture.
2130	70	Rough, 30° fracture.
2170	40	Rough, horizontal fracture.
2200	30	Rough, 10° fracture.
2280 - 2300	100	Rough, 50° fracture.
2360	60	Jagged, 60° fracture.

DISTANCE (mm)	SPACING (mm)	DISCONTINUITY DESCRIPTION
2400 - 2440	-	Grout-filled, 60° discontinuity.
2450	90	Clean, 45° fracture.
2510	60	Ground surface.
2510 - 2550	-	Closed, vertical fracture.
2550	40	Ground surface.
2570	20	Jagged fracture surfaces.
2640	70	Rough, 20° fracture.
2640 - 2750	(110)	Broken pieces: 10-30mm; horizontal fractures.
2830	80	Ground surface.
2900	70	Ground surface.
2960	60	Ground surface.
2960 - 2990	-	Grout-filled, 70° fracture.
3080	90	Ground surface.
3180	100	Rough, 15° fracture.
3200 - 3310	130	Clean, 15° fracture.
3340	30	Grout-filled, 10° fracture.
3340 - 3440	(100)	Broken pieces & discs: 10-30mm; horizontal fractures.
3440	-	Jagged, horizontal fracture.
3480	40	Grout-filled, 40° fracture.
3530	50	Jagged, 60° fracture.
3580	50	Rough, 15° fracture.
3680 - 3730	150	Rough, 75° fracture.
3830	100	Smooth bedding plane with waxy filling.
3900	70	Rough, 10° fracture.
4110 - 4160	260	Rough, 70° fracture.
4270	110	Rough, horizontal fracture.
4310	40	Rough, 10° fracture.
4405	95	Rough, horizontal fracture.

DISTANCE (mm)	SPACING (mm)	DISCONTINUITY DESCRIPTION
4580 - 4610	205	Rough, 50° fracture.
4660	50	Rough, horizontal fracture.
4760	100	Rough, 15° fracture.
4810	50	Smooth bedding plane with waxy filling.
4890	80	Jagged, 15° fracture.
4890 - 5040	(150)	Broken pieces: 45° fractures.
5080 - 5110	70	Jagged, 50° fracture.
5200	90	Jagged, horizontal fracture.
5260	60	Rough, 30° fracture.
5270	10	30° disc.
5350	80	Ground surface.
5410	60	Ground surface.
5500	90	Ground surface.
5520	20	Ground surface.
5560	40	Ground surface.
5600 - 5660		Closed 80° fracture.
5720	160	Rough, 10° fracture.
5810	90	Rough, 10° fracture.
5820 - 5900	90	Rough, vertical fracture.
6020	120	Jagged, 10° fracture.
6020	Log terminated at the end of the first 5m of core.	

APPENDIX 3. ROCK QUALITY.

APPENDIX 3.

ROCK TESTING - 78A-42W N7 GULLY BOREHOLE CORE.

REF No	DISTANCE (wrt VR) (mm)	SIGMA 1 (σ_1) (MPa)	SIGMA 3 (σ_3) (MPa)	MODULUS (E) (GPa)	POISSONS RATIO (ν)	LINEAR LIMITS (MPa)
1-1U	1650-1770	169	-	62,8	-	61-140
2-1T	2550-2720	220	10	63,2	0,16	45-173
3-1T	2990-3050	243	20	63,3	0,18	30-112
4-1T	3500-3610	309	40	60,5	0,17	48-169
5-1U	3680-3900	178	-	66,4	-	56-126
5-2T	3680-3900	336	40	61,9	0,19	48-194
6-1T	4000-4100	265	10	64,0	0,18	49-148
7-1T	4100-4280	307	20	66,5	0,17	53-172
7-2T	4100-4280	462	40	71,9	0,15	88-276
8-1U	4360-4450	133	-	64,7	-	58-128
9	5640-5730	-	-	-	-	-
10	6670-6800	-	-	-	-	-
11	7000-7110	-	-	-	-	-
12	7110-7420	-	-	-	-	-
13	7940-8050	-	-	-	-	-
14	8230-8430	-	-	-	-	-
15	8660-8850	-	-	-	-	-
16	8850-9080	-	-	-	-	-
17	9080-9300	-	-	-	-	-
18	9330-9430	-	-	-	-	-
19	12260-12460	-	-	-	-	-
20	12590-12940	-	-	-	-	-
21	13210-13300	-	-	-	-	-
22	13370-13470	-	-	-	-	-
23	13500-13720	-	-	-	-	-

REF No	DISTANCE (wrt VR) (mm)	SIGMA 1 (σ_1) (MPa)	SIGMA 3 (σ_3) (MPa)	MODULUS (E) (GPa)	POISSONS RATIO (ν)	LINEAR LIMITS (MPa)
24-1T	13720-13920	210	10	59,4	0,20	40-128
25-1T	14140-14420	238	10	61,8	0,19	63-153
25-2T	14140-14420	328	20	67,3	0,15	50-187
26-1U	14610-14930	192	-	62,8	-	25-126
26-2T	14610-14930	410	40	68,7	0,16	51-197
27-1T	15000-15260	281	10	66,8	0,16	47-186
27-2T	15000-15260	300	20	66,0	0,18	46-184
27-3T	15000-15260	432	40	70,1	0,16	37-221
28	15260-15440	-	-	-	-	-
29	15740-15890	-	-	-	-	-
30	15980-16090	-	-	-	-	-
31	16500-16610	-	-	-	-	-
32	16610-16760	-	-	-	-	-
33	16760-16860	-	-	-	-	-
34	16880-17040	-	-	-	-	-
35	17690-17800	-	-	-	-	-
36	17840-17930	-	-	-	-	-
37	18000-18200	-	-	-	-	-
38	18200-18300	-	-	-	-	-
39	18310-18480	-	-	-	-	-
40	18900-19160	-	-	-	-	-
41	19340-19480	-	-	-	-	-
42	19720-19830	-	-	-	-	-
43	19950-20210	-	-	-	-	-
44	20210-20590	-	-	-	-	-
45-1U	20730-20840	95	-	57,7	-	20-55
46-1U	21000-21120	184	-	64,0	-	46-146

REF No	DISTANCE (wrt VR) (mm)	SIGMA 1 (σ_1) (MPa)	SIGMA 3 (σ_3) (MPa)	MODULUS (E) (GPa)	POISSONS RATIO (ν)	LINEAR LIMITS (MPa)
47-1T	22370-22600	279	10	65,0	0,13	54-204
47-2T	22370-22600	312	20	65,2	0,15	46-209
47-3T	22370-22600	393	40	66,5	0,14	51-205
48-1U	22720-23010	207	-	68,0	-	52-160
48-2T	22720-23010	262	10	64,0	0,15	42-203
48-3T	22720-23010	475	40	70,4	0,13	49-264
49-1T	23070-23240	283	20	66,7	0,16	54-191
49-2T	23070-23240	364	40	69,4	0,13	34-195

MEAN CHARACTERISTICS OF SAMPLE GROUPS

	UNIAXIAL COMPRESSIVE STRENGTH	CALCULATED TENSILE STRENGTH	INTERNAL FRICTION ANGLE
UPPER GROUP - 20,7m-23,3m	204 MPa	45 MPa	42,5°
MIDDLE GROUP - 13,7m-15,3m	199 MPa	42 MPa	43,8°
LOWER GROUP - 1,7m-4,5m	171 MPa	38 MPa	42,0°

APPENDIX 4. STRATA MOVEMENT.

APPENDIX 4A(i).

GULLY CLOSURE MEASUREMENTS - 78A-42W STOPE.

DATE	N8 GULLY - STN No 4			N7 GULLY - STN No 6			N6 GULLY - STN No 4		
	FACE DIST	WEEK CONV	PROG CONV	FACE DIST	WEEK CONV	PROG CONV	FACE DIST	WEEK CONV	PROG CONV
11/1/93	9	-	-	10	-	-	8	-	-
18/1/93	12	-	-	14	(1129)	-	12	(1041)	-
25/1/93	16	(1023)	-	19	111	111	17	65	65
1/2/93	19	51	51	23	118	229	21	71	136
8/2/93	22	46	97	27	76	305	25	50	186
15/2/93	25	40	137	29	110	415	27	30	216
22/2/93	27	32	169	30	100	515	29	18	234
1/3/93	29	4	173	32	97	612	30	14	248
8/3/93	31	10	183	33	65	707	31	16	264
15/3/93	33	14	197	35	42	749	32	11	275
22/3/93	35	31	228	38	47	796	35	21	296
29/3/93	37	9	237	41	9	805	38	15	311
5/4/93	39	28	265	44	28	833	41	25	336
12/4/93	41	8	273	47	12	845	44	2	338
19/4/93	44	14	287	50	18	863	48	9	347
26/4/93	46	11	298	53	12	875	53	11	358
3/5/93	48	10	308	56	11	886	57	8	366
10/5/93	50	7	315	59	8	894	61	8	374
17/5/93	52	6	321	61	7	901	64	6	380
24/5/93	54	.	338	64	11	912	67	13	393
31/5/93	57	13	351	66	11	923	70	7	400
7/6/93	59	18	369	68	3	926	72	6	406
14/6/93	61	3	372	70	4	930	76	4	410
21/6/93	64	6	378	73	4	934	79	2	412
28/6/93	66	3	381	76	5	939	82	3	415
5/7/93	68	3	384	78	4	943	85	2	417
12/7/93	70	2	386	80	3	946	88	2	419
19/7/93	70	2	388	80	2	949	88	2	421

APPENDIX 4. STRATA MOVEMENT.**APPENDIX 4A(ii).****FACE CLOSURE MEASUREMENTS - 78A-42W STOPE.**

	FACE DIST (m)	STOPE HEIGHT (mm)	STOPE CLOSURE (mm)	PROG CLOSURE (mm)	ELASTIC CONV (mm)
TOP	1,0	1200	-	-	15
	3,1	1045	-	0	30
	4,1	1002	43	43	35
	4,9	960	42	85	39
	5,7	921	39	124	41
MIDDLE	1,0	1200	-	-	15
	3,0	1057	-	0	30
	4,0	1007	50	50	35
	4,9	958	49	99	39
	5,9	912	46	145	41
BOTTOM	1,0	1300	-	-	15
	3,0	1121	-	0	30
	4,0	1056	65	65	35
	5,0	998	58	123	39
	6,0	943	55	178	41

APPENDIX 4. STRATA MOVEMENT.**APPENDIX 4A(iii).****CLOSURE AND RIDE MEASUREMENTS - 78A-42W STOPE.**

DATE	MEASUREMENTS			RESULTS			
	HW1-FW (mm)	HW2-FW (mm)	HW3-FW (mm)	DAILY CLOSE	PROG CLOSE	STRIKE RIDE	DIP RIDE
9/9/93	972	1296	1140	-	-	-	-
13/9/93	961	1287	1130	0	14	2	0
14/9/93	953	1283	1117	19	33	- 2	11
15/9/93	936	1278	1108	29	62	- 17	13
16/9/93	932	1274	1106	2	64	- 15	10
20/9/93	913	1267	1090	37	101	- 28	19
21/9/93	909	1264	1087	5	106	- 28	18
22/9/93	908	1263	1085	2	108	- 27	19
23/9/93	906	1261	1081	4	112	- 26	22
24/9/93	903	1258	1077	4	116	- 25	22
27/9/93	895	1233	1056	- 7	109	12	17
28/9/93	860	1231	1053	61	170	- 31	12
29/9/93	850	1227	1050	- 4	166	- 24	11
30/9/93	860	1221	1047	- 8	158	- 12	7
1/10/93	857	1219	1046	3	161	- 13	6
4/10/93	838	1209	1038	24	185	- 19	0

NOTE.**Hangingwall Base Triangle:**

Strike HW1 - HW2 = 650mm
 Dip HW2 - HW3 = 850mm
 Diagonal HW1 - HW3 = 1000mm

APPENDIX 4. STRATA MOVEMENT.

APPENDIX 4B(i).

EXTENSOMETER MEASUREMENTS - 72-43W N8 GULLY.

DATE	TIME LAPSE (Days)	FACE DIST (m)	CUBBY HEIGHT (mm)	STOPE CLOSE (mm)	2m TAPE (mm)	4m TAPE (mm)	8m TAPE (mm)	11m TAPE (mm)	15m TAPE (mm)	20m TAPE (mm)
16/9/92	0	4	3437	0	-	-	-	-	-	-
25/9/92	9	4	3373	64	-	-	-	-	-	-
2/10/92	16	4	3302	155	-	-	-	-	-	-
9/10/92	23	6	3266	191	-	-	-	-	-	-
16/10/92	30	10	3246	211	-	-	-	-	-	-
21/10/92	35	13	3215	242	-	-	-	-	-	-
28/10/92	42	16	3170	287	-	-	-	-	-	-
3/11/92	48	19	3129	328	-	-	-	-	-	-
9/11/92	54	22	3094	363	-	-	-	-	-	-
16/11/92	61	24	3052	395	-	-	-	-	-	-
23/11/92	68	26	3032	425	-	-	-	-	-	-
1/12/92	76	28	3004	459	-	-	-	-	-	-
9/12/92	84	30	2977	480	-	-	-	-	-	-
15/12/92	90	32	2968	489	-	-	-	-	-	-
22/12/92	97	34	2960	497	2000	4000	8000	11000	15000	20000
7/92	104	36	2945	512	2002	4003	8000	11002	15005	20007
3.1/93	111	38	2934	523	2001	4001	8001	11000	14999	20004
12/1/93	118	38	2926	532	2000	4001	8001	11000	15000	20007
21/1/93	127	38	2917	540	2000	4002	8001	11001	15000	20010
8/2/93	145	38	2904	553	1999	4001	7997	11000	15000	20009
22/2/93	159	38	2895	562	2001	4001	8000	11001	15000	20010
9/3/93	174	38	2888	569	2001	4001	8000	11001	15000	20013
23/3/93	188	38	2883	574	2000	4001	7991	11000	15000	20016
8/4/93	214	38	2878	579	2000	4002	8000	11002	15000	20015
20/4/93	226	38	2875	582	2001	4002	8001	11002	15000	20016
6/5/93	242	38	2872	585	2001	4003	8000	11002	15000	20016
20/5/93	256	38	2870	587	2001	4003	8000	11003	15001	20017

APPENDIX 4. STRATA MOVEMENT.

APPENDIX 4B(ii).

EXTENSOMETER MEASUREMENTS - 78A-42W N7 GULLY.

DATE	TIME LAPSE (Days)	FACE DIST (m)	CUBBY HEIGHT (mm)	STOPE CLOSE (mm)	2m TAPE (mm)	5m TAPE (mm)	10m TAPE (mm)	14m TAPE (mm)	18m TAPE (mm)
18/1/93	0	4	2549	0	-	-	-	-	-
25/1/93	7	7	2467	82	-	-	-	-	-
1/2/93	14	10	2120	429	-	-	-	-	-
12/2/93	24	14	2051	498	-	-	-	-	-
15/2/93	28	14	2023	526	-	-	-	-	-
24/2/93	37	14	2000	549	-	-	-	-	-
1/3/93	42	16	2000	549	-	-	-	-	-
8/3/93	49	19	1984	565	-	-	-	-	-
19/3/93	60	25	1975	574	-	-	-	-	-
29/3/93	70	29	1953	596	-	-	-	-	-
6/4/93	78	32	1943	606	-	-	-	-	-
12/4/93	85	35	1928	621	-	-	-	-	-
21/4/93	93	39	1884	665	2000	6000	10000	14000	18000
28/4/93	100	42	1869	680	1999	6000	9999	14000	18000
6/5/93	108	45	1864	685	1997	5997	9995	13993	17998
13/5/93	115	48	1859	690	1997	5996	9992	13991	17989
20/5/93	122	50	1853	696	1998	5998	9993	13989	17993
27/5/93	129	52	1837	712	2001	5996	9993	13989	-
3/6/93	136	55	1824	725	2001	5997	9994	13989	-
14/6/93	147	58	1812	737	2001	5993	9987	13989	-
23/6/93	156	60	1808	741	1998	5995	9989	13984	-
29/6/93	162	63	1803	746	1999	5995	9989	13984	-
6/7/93	169	66	1798	751	1999	5994	9989	13984	-
13/7/93	176	68	1793	756	1999	5994	9989	13985	-
19/7/93	182	68	1789	760	1998	5993	9989	13985	-
26/7/93	189	68	1786	763	1999	5994	9987	13985	-
3/8/93	197	68	1783	766	1999	5992	9991	13985	-
11/8/93	205	68	1781	768	2000	5992	9988	13980	-

APPENDIX 4. STRATA MOVEMENT.

APPENDIX 4B(iii).

EXTENSOMETER MEASUREMENTS - 78A-42W CENTRE GULLY.

DATE	TIME LAPSE (Days)	FACE DIST (m)	CUBBY HEIGHT (mm)	STOPE CLOSE (mm)	2m TAPE (mm)	4.5m TAPE (mm)	10m TAPE (mm)	14M TAPE (mm)	18m TAPE (mm)
8/4/93	0	4	2538	0	-	-	-	-	-
12/4/93	4	4	2530	8	-	-	-	-	-
19/4/93	11	4	2521	17	-	-	-	-	-
21/4/93	13	4	2520	18	2000	4500	10000	14000	18000
28/4/93	20	4	2513	25	2000	4503	10004	14005	18005
6/5/93	28	4	2507	31	2002	4508	10011	14011	18012
13/5/93	35	4	2505	33	2005	4515	10019	14019	18019
20/5/93	42	4	2502	36	2006	4515	10019	14021	18019
27/5/93	49	4	2493	45	2008	4521	10024	14025	18027
3/6/93	56	4	2487	51	2010	4524	10029	14030	18032
7/6/93	60	4	2484	54	2010	4525	10031	14032	18032
15/6/93	68	4	2476	62	2014	4530	10039	14037	18038
23/6/93	76	4	2456	82	2027	4545	10054	14065	18047
29/6/93	82	4	2445	93	2035	4568	10233	14288	18238
6/7/93	89	4	-	-	-	-	-	-	-
13/7/93	96	4	-	-	-	-	-	-	-
19/7/93	102	7	2221	317	2103	4680	10238	-	18240
26/7/93	109	9	2205	333	2117	4686	10241	-	18244
3/8/93	117	11	2153	385	2134	4723	-	-	18288
11/8/93	125	14	2094	444	2145	4735	-	-	-
19/8/93	133	15	2049	489	-	-	-	-	-
26/8/93	140	19	2010	528	-	-	-	-	-
2/9/93	147	22	1975	563	-	-	-	-	-
9/9/93	154	25	1944	594	-	-	-	-	-
16/9/93	161	29	1925	613	-	-	-	-	-
22/9/93	167	32	1912	626	-	-	-	-	-
28/9/93	173	36	1907	631	-	-	-	-	-

APPENDIX 5. STRESS MEASUREMENTS.

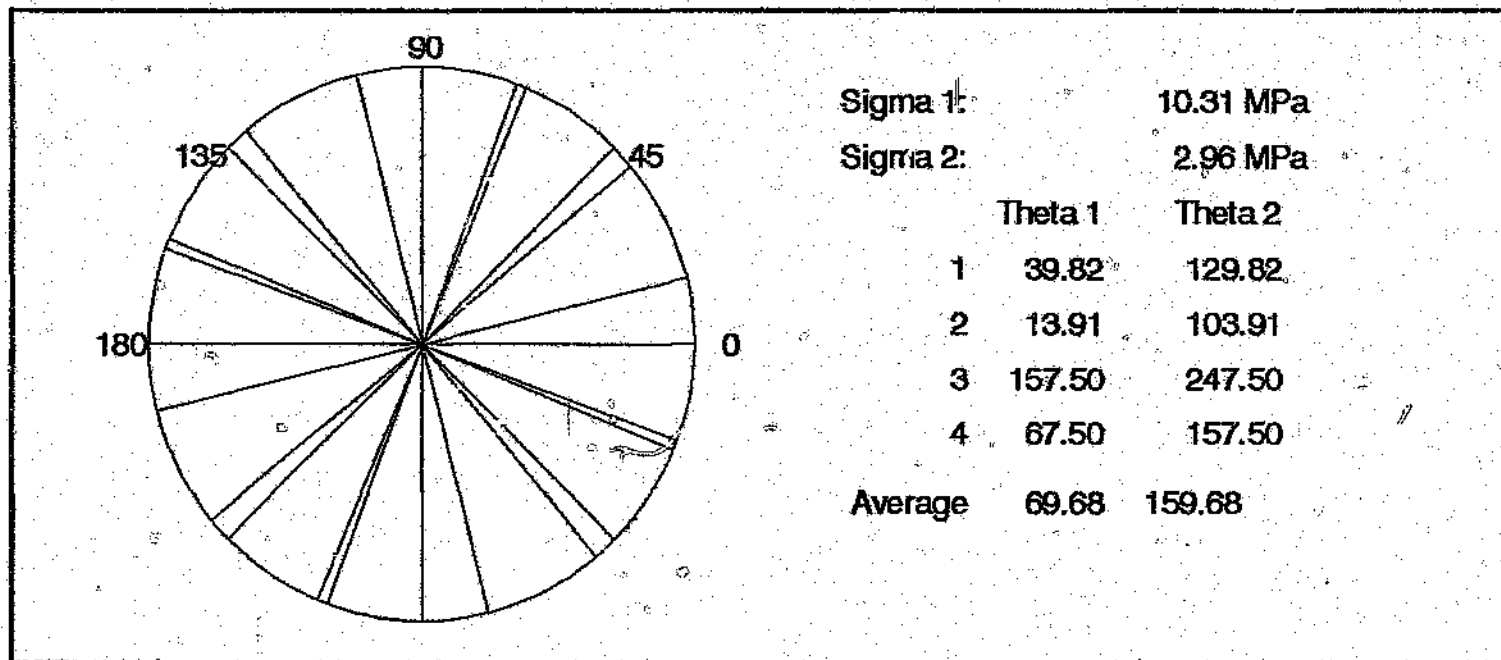
APPENDIX 5A(i).

HANGINGWALL STRESS - 72-43W N8 GULLY, (1st Result)

DATE: 22/12/92
 SITE: 72-43W N8 Gully
 FACE DISTANCE: 35m
 HOLE DEPTH: 2,30m
 ORIENTATION: 155°

STRAIN GAUGE READINGS

	BEFORE OVERCORING	AFTER OVERCORING	DIFFERENCE
HORIZONTAL GAUGE	+01 +791	+02 +029	
	+02 - 213	+03 - 972	
	- 01789	+02029	+240
45° GAUGE	+02 +293	+02 +489	
	+03 - 708	+03 - 512	
	- 02293	+02489	+196
VERTICAL GAUGE	+02 +958	+03 +002	
	+03 - 041	+04 - 999	
	- 02958	+03002	+044
135° GAUGE	+01 +930	+02 +022	
	+02 - 071	+03 - 980	
	- 01930	+02021	+091



**APPENDIX 5A(i) - Interpretation of Strain Gauge Readings.
72-43W N8 Gully. (1st Result).**

APPENDIX 5. STRESS MEASUREMENTS.

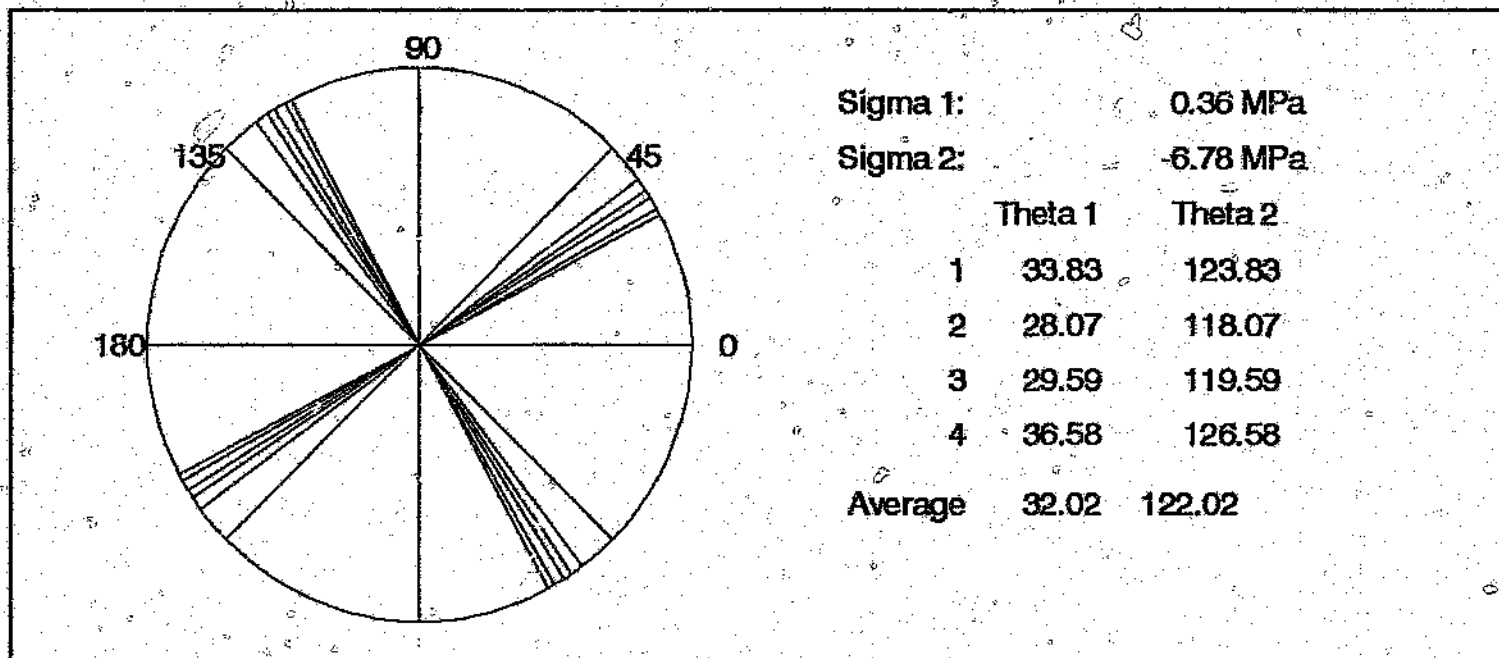
APPENDIX 5A(ii).

HANGINGWALL STRESS - 72-43W N8 GULLY. (2nd Result)

DATE: 30/12/92
 SITE: 72-43W N8 Gully
 FACE DISTANCE: 35m
 HOLE DEPTH: 2,50m
 ORIENTATION: 040°

STRAIN GAUGE READINGS

	BEFORE OVERCORING	AFTER OVERCORING	DIFFERENCE
HORIZONTAL GAUGE	+01 +486	+01 +466	
	+02 - 515	+02 - 536	
	- 01486	+01465	- 021
45° GAUGE	+02 +268	+02 +300	
	+03 - 734	+03 - 702	
	- 02267	+02299	+032
VERTICAL GAUGE	+02 +436	+02 +342	
	+03 - 565	+03 - 660	
	- 02436	+02341	- 095
135° GAUGE	+01 +747	+01 +627	
	+02 - 252	+02 - 373	
	- 01747	+01627	- 120



APPENDIX 5A(ii)

**Interpretation of Strain Gauge Readings.
72-43W N8 Gally. (2nd Result).**

APPENDIX 5. STRESS MEASUREMENTS.

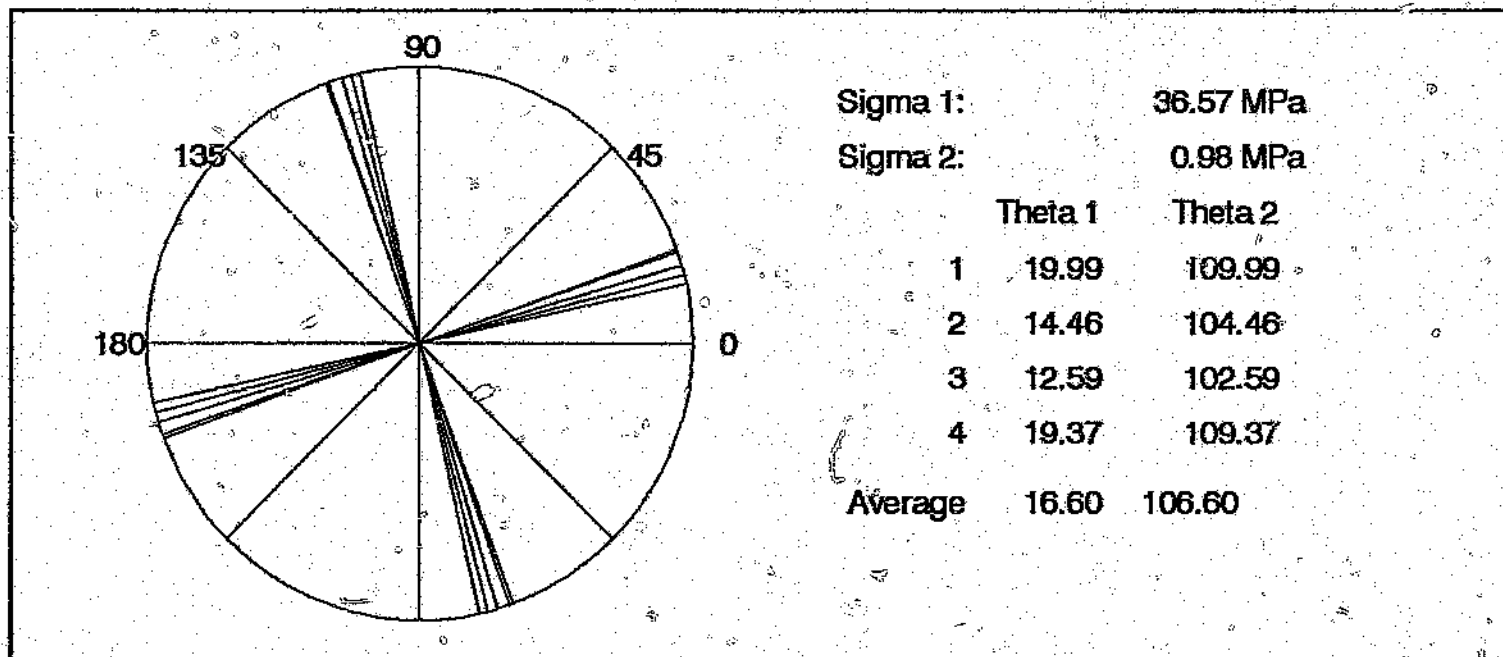
APPENDIX 5A(iii).

HANGINGWALL STRESS - 78A-42W N7 GULLY. (1st Result)

DATE: 22/2/92
 SITE: 78A-42W N7 Gully
 FACE DISTANCE: 20m
 HOLE DEPTH: 4,50m
 ORIENTATION: 000°

STRAIN GAUGE READINGS

	BEFORE OVERCORING	AFTER OVERCORING	DIFFERENCE
HORIZONTAL GAUGE	+01 +098	+01 +282	
	+02 - 901	+02 - 716	
	- 01098	+01717	+619
45° GAUGE	+00 +025	+00 +543	
	+01 - 982	+01 - 460	
	- 00020	+00542	+522
VERTICAL GAUGE	+01 +031	+00 +946	
	+02 - 969	+01 - 059	
	- 01031	+00944	- 087
135° GAUGE	+01 +563	+01 +665	
	+02 - 434	+02 - 335	
	- 01565	+01665	+100



APPENDIX 5A(iii)

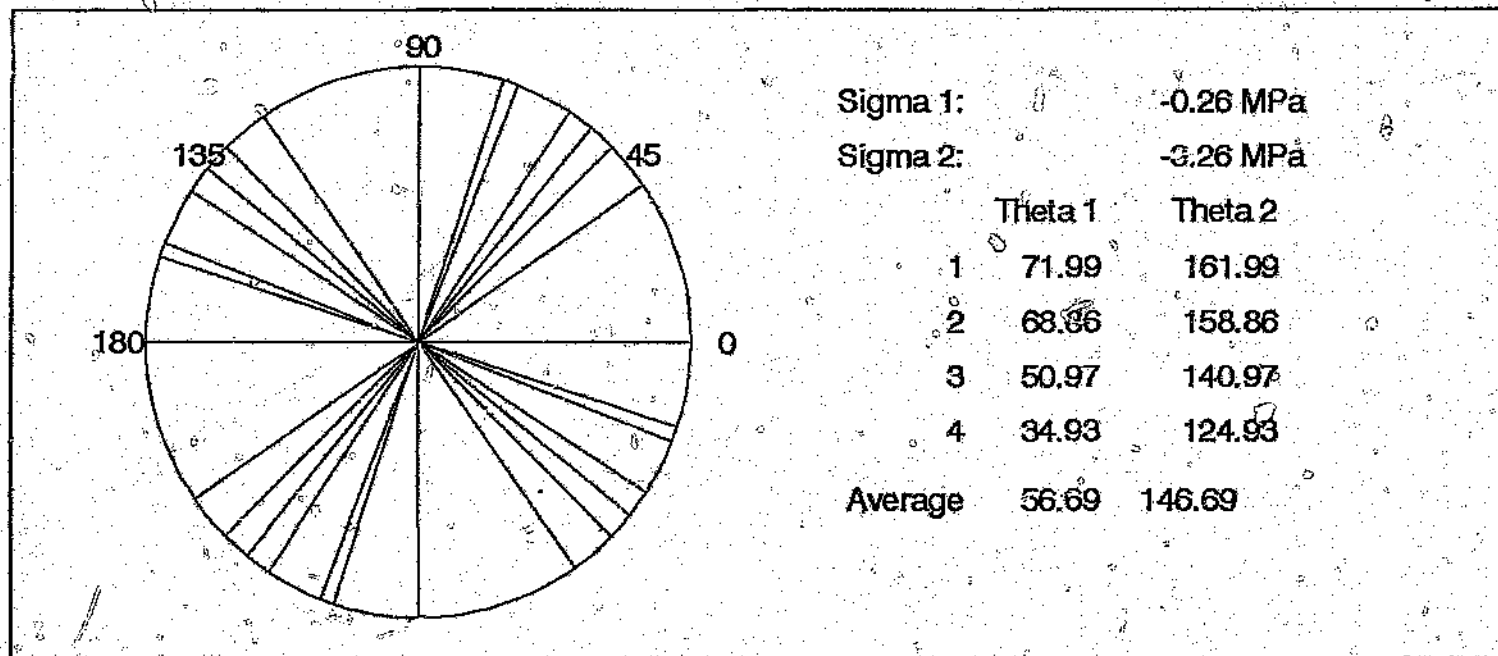
**Interpretation of Strain Gauge Readings.
78A-42W N7 Gully. (1st Result).**

APPENDIX 5. STRESS MEASUREMENTS.**APPENDIX 5A(iv).****HANGINGWALL STRESS - 78A-42W N7 GULLY. (2nd Result)**

DATE: 23/2/92
SITE: 78A-42W N7 Gully
FACE DISTANCE: 20m
HOLE DEPTH: 5,00m
ORIENTATION: 315°

STRAIN GAUGE READINGS

	BEFORE OVERCORING	AFTER OVERCORING	DIFFERENCE
HORIZONTAL GAUGE	+01 +279	+01 +251	
	+02 - 721	+02 - 749	
	- 01279	+01251	- 028
45° GAUGE	+02 +159	+02 +134	
	+03 - 843	+03 - 836	
	- 02158	+02149	- 009
VERTICAL GAUGE	+02 +316	+02 +310	
	+03 - 686	+03 - 692	
	- 02315	+02309	- 006
135° GAUGE	+01 +561	+01 +492	
	+02 - 438	+02 - 508	
	- 01561	+01492	- 069



APPENDIX 5A(iv)

Interpretation of Strain Gauge Readings.
78A-42W N7 Gully. (2nd Result).

APPENDIX 5. STRESS MEASUREMENTS.

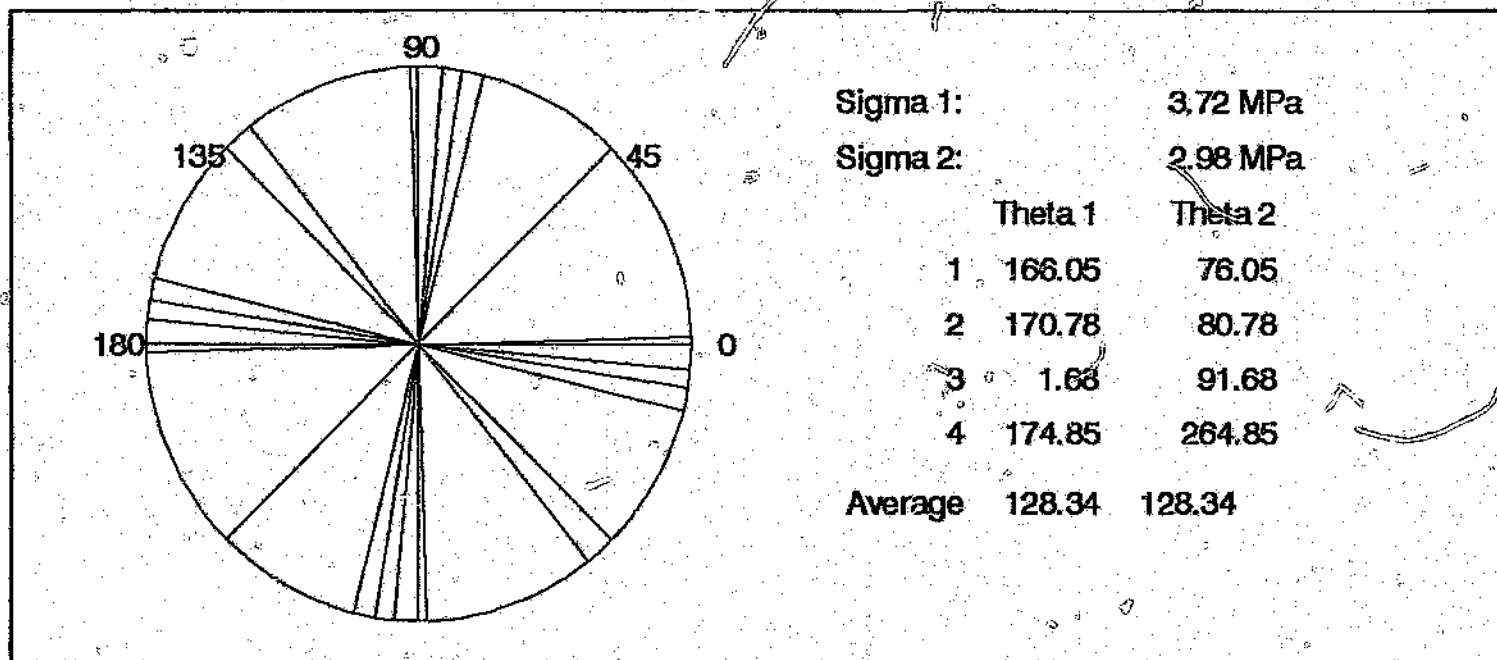
APPENDIX 5A(v).

HANGINGWALL STRESS - 78A-42W S4 CENTRE GULLY. (1st Res 116)

DATE: 19/4/93
 SITE: 78A-42W S4 C/Gully
 FACE DISTANCE: 4m
 HOLE DEPTH: 3,80m
 ORIENTATION: 135°

STRAIN GAUGE READINGS

	BEFORE OVERCORING	AFTER OVERCORING	DIFFERENCE
HORIZONTAL GAUGE	+01 +272	+01 +336	
	+02 - 726	+02 - 663	
	- 01273	+01336	+063
45° GAUGE	+02 +713	+02 +763	
	+03 - 286	+03 - 238	
	- 02713	+02763	+050
VERTICAL GAUGE	+02 +445	+02 +492	
	+03 - 558	+03 - 513	
	- 02444	+02490	+046
135° GAUGE	+01 +920	+01 +975	
	+02 - 078	+02 - 025	
	- 01921	+01975	+054



APPENDIX 5A(v)

Interpretation of Strain Gauge Readings.
78A-42W C/Gully. (1st Result).

APPENDIX 5. STRESS MEASUREMENTS.

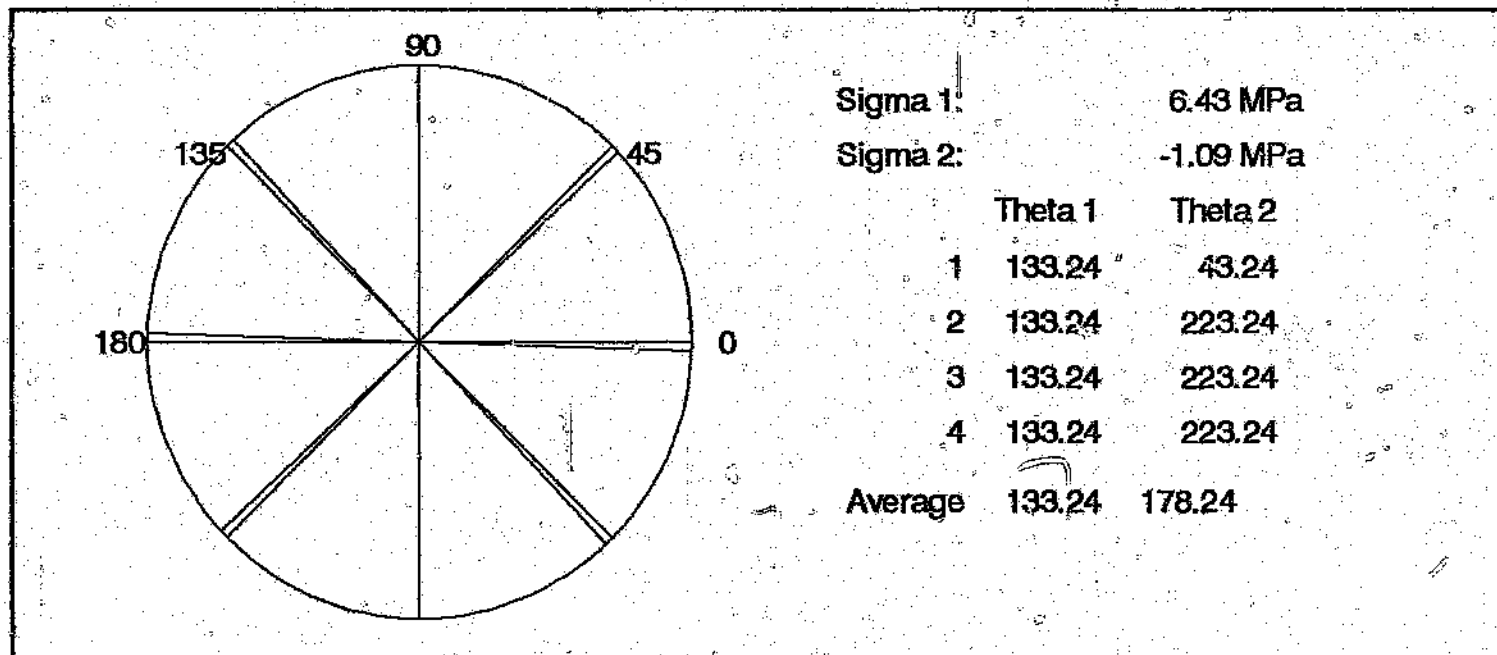
APPENDIX 5A(vi).

HANGINGWALL STRESS - 78A-42W S4 CENTRE GULLY. (2nd Result)

DATE: 28/4/93
 SITE: 78A-42W S4 C/Gully
 FACE DISTANCE: 4m
 HOLE DEPTH: 1.55
 ORIENTATION: 315°

STRAIN GAUGE READINGS

	BEFORE OVERCORING	AFTER OVERCORING	DIFFERENCE
HORIZONTAL GAUGE	+01 +445	+01 +481	
	+02 - 555	+02 - 519	
	- 01444	+01481	+037
45° GAUGE	+02 +314	+02 +268	
	+03 - 687	+03 - 734	
	- 02314	+02267	+ 047
VERTICAL GAUGE	+02 +341	+02 +389	
	+03 - 663	+03 - 614	
	- 02339	+02387	+048
135° GAUGE	+02 +299	+02 +431	
	+03 - 705	+03 - 573	
	- 02297	+02429	+132



APPENDIX 5A(vi)

**Interpretation of Strain Gauge Readings.
78A-42W C/Gully. (2nd Result).**

APPENDIX 5. STRESS MEASUREMENTS.

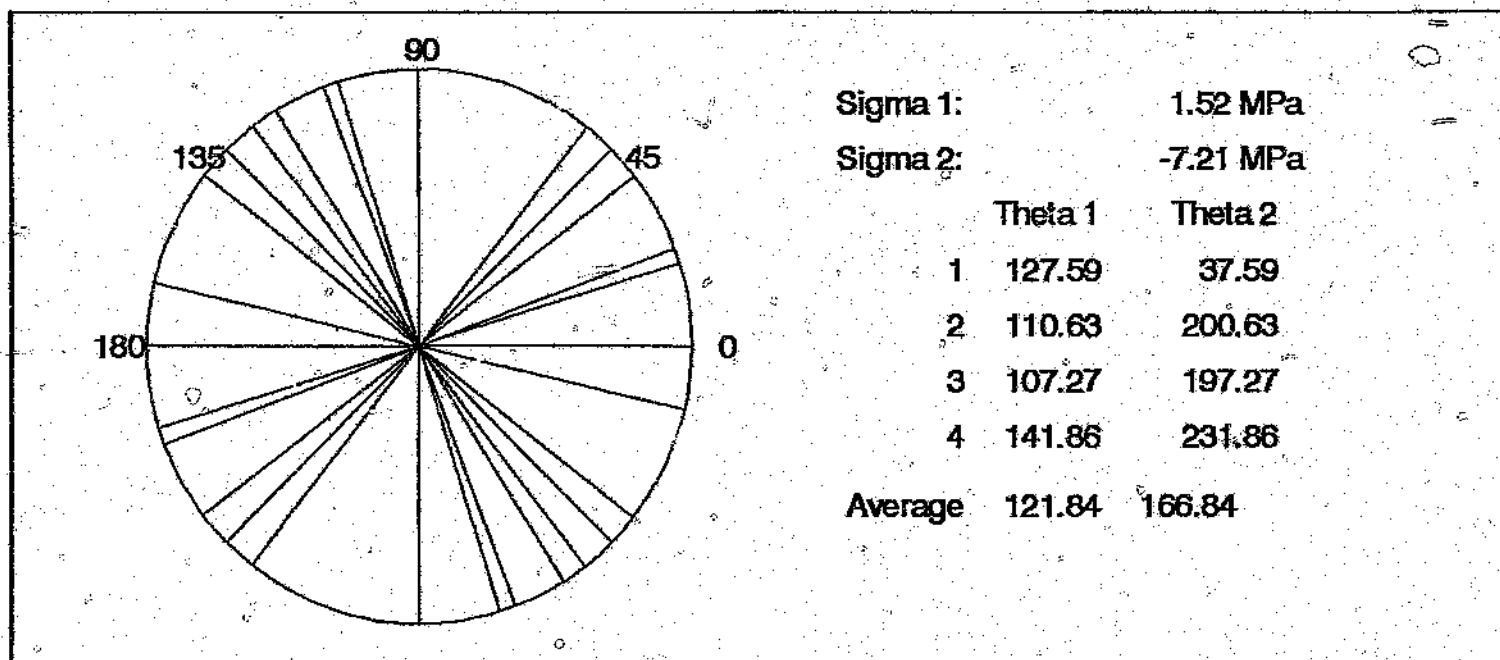
APPENDIX 5A(vii).

HANCINGWALL STRESS - 78A-42W S4 CENTRE GULLY. (3rd Result)

DATE: 29/4/93
 SITE: 78A-42W S4 C/Gully
 FACE DISTANCE: 4m
 HOLE DEPTH: 2,05m
 ORIENTATION: 225°

STRAIN GAUGE READINGS

	BEFORE OVERCORING	AFTER OVERCORING	DIFFERENCE
HORIZONTAL GAUGE			
	+01 +71	+01 +732	
	+02 - 227	+02 - 282	
	- 01779	+01725	- 054
45° GAUGE			
	+02 +026	+01 +876	
	+03 - 965	+02 - 138	
	- 02037	+01869	- 161
VERTICAL GAUGE			
	+02 +797	+02 +819	
	+03 - 192	+03 - 168	
	- 02802	+02825	+023
135° GAUGE			
	+02 +687	+02 +696	
	+03 - 304	+03 - 292	
	- 02691	+02702	+011



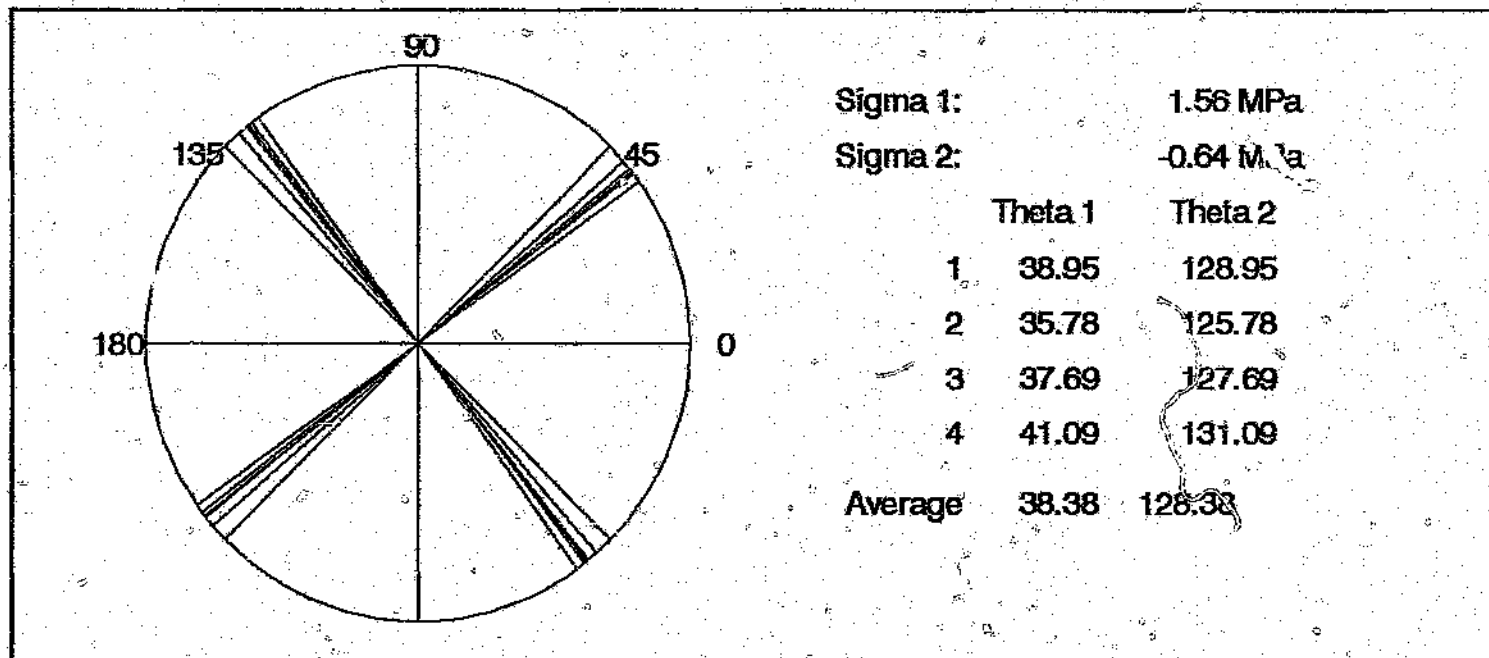
**APPENDIX 5A(vii) - Interpretation of Strain Gauge Readings.
 78A-42W C/Gully. (3rd Result).**

APPENDIX 5. STRESS MEASUREMENTS.**APPENDIX 5A(viii).****HANGINGWALL STRESS - 78A-42W S4 CENTRE GULLY. (4th Result)**

DATE: 29/4/93
SITE: 78A-42W S4 C/Gully
FACE DISTANCE: 4m
HOLE DEPTH: 2,30m
ORIENTATION: 135°

STRAIN GAUGE READINGS

	BEFORE OVERCORING	AFTER OVERCORING	DIFFERENCE
HORIZONTAL GAUGE	+00 +815	+00 +827	
	+01 - 203	+01 - 192	
	- 00806	+00818	+012
45° GAUGE	+02 +140	+02 +173	
	+03 - 844	+03 - 810	
	- 02148	+02182	+034
VERTICAL GAUGE	+02 +394	+02 +393	
	+03 - 589	+03 - 591	
	- 02402	+02402	0
135° GAUGE	+02 +277	+02 +261	
	+03 - 706	+03 - 724	
	- 02285	+02268	- 017



**APPENDIX 5A(viii) - Interpretation of Strain Gauge Readings.
78A-42W C/Gully. (4th Result).**

APPENDIX 5. STRESS MEASUREMENTS.

APPENDIX 5B.

STRESS INTERPRETATION SUMMARY.

	FACE DIST	HOLE DEPTH	σ_1 (MPa)	σ_3 (MPa)	MAX STRESS ORIENTATION	REEF STRIKE
72-43W N8 GULLY						
1st	35m	2,30m	+10,3	+ 3,0	155 - 070 = 085°	045°
2nd	35m	2,50m	+ 0,4	- 6,8	040 - 032 = 008°	045°
Mean			+ 5,7	- 1,9	046°	
78A-42W N7 GULLY						
1st	20m	4,50m	+36,6	+ 1,0	000 - 017 = 163°	045°
2nd	20m	5,00m	- 0,3	- 3,3	315 - 057 = 078°	045°
Mean			+18,2	- 1,1	120°	
78A-42W C/GULLY						
1st	4m	3,80m	+ 3,7	+ 3,0	135 - 128 = 007°	045°
2nd	4m	1,55m	+ 6,4	- 1,1	315 - 133 = 002°	045°
3rd	4m	2,05m	+ 1,5	- 7,2	225 - 122 = 104°	045°
4th	4m	2,30m	+ 1,6	- 0,6	135 - 038 = 097°	045°
Mean			+ 3,3	- 1,5	052°	

NOTES.

Calculation assumptions:

Elastic Modulus = 70GPa.

Poisson's Ratio = 0,2.

Maximum stress and reef strike orientations are measured:

Clockwise from North = 000°

APPENDIX 5. STRESS MEASUREMENTS.

APPENDIX 5C.

STRESS MEASUREMENT METHOD AND PROBLEMS.

A number of problems were encountered while attempting to measure the state of stope hangingwall stress. Out of a total of some fifty attempts made during a six month period between 7 November, 1992 and 4 May, 1993, which cost in the region of R50 000 and an inordinate number of man-hours, only eight met with moderate success. A brief outline of the procedure employed and a list of some of the problems encountered, follows:-

PROCEDURE.

- * A diamond drill hole is drilled to the required distance into the hangingwall using a BX bit, taking care to avoid the introduction of grease into the hole.
- * The integrity of the rock must be reasonably sound to facilitate stress measurement. This may be judged from the tone and performance of the drilling machine.
- * The end of the hole is flattened and polished by removing the drill string and replacing the drilling bit with appropriate flattening and polishing bits.
- * The hole is flushed of fine material and dirt using a hose connected to the mine water supply which is Rand Water Board quality. Once again the introduction of grease is avoided.
- * The hole is dried with a clean cloth attached to the end of a string of flexible rods. The polished end of the hole is further treated with acetone using a clean cloth or brush applicator.
- * The end of the hole is visually inspected for defects.
- * A CSIR "doorstopper" strain gauge is prepared by lightly roughening the gauge surface with emery paper and cleaning with acetone. This greatly assists the adherence of glue to the gauge.
- * The doorstopper is fitted to the connection socket on the front end of the insertion trolley which houses a compensating dummy gauge, a trailing cable, and a spring arrangement for applying constant placement pressure.
- * An appropriate two component glue is prepared to the correct consistency commensurate with the prevailing temperature conditions to allow just sufficient time to insert and locate the doorstopper.
- * The glue paste is applied evenly to the top of the strain gauge which is then inserted into the drill hole and glued onto the polished end of the hole using the insertion trolley and rods.

- * Pressure is maintained on the installation apparatus by wedging the rods at the mouth of the hole until the residue glue in the mixing thimble has set, indicating likely bonding of the strain gauge and rock surface.
- * The appropriate wires of the trailing cable are connected to an electrical resistance meter (DMD 20 strain indicator apparatus).
- * Initial (resistance/strain) readings are taken of the horizontal, 45°, vertical and 135° gauge circuits respectively once the gauge and circuits are at equilibrium.
- * The insertion trolley is disconnected from the strain gauge by carefully tugging the rods.
- * The rods, cable, and trolley are withdrawn from the hole, taking careful note of the orientation of the trolley. This is checked by visually inspecting the installed orientation of the doorstopper.
- * The drill string fitted with an overcoring barrel is reintroduced and the strain gauge is carefully overcored for an adequate distance.
- * The core with doorstopper attached is broken off by applying sudden, momentary torque to the drill string.
- * An adequate flow of drilling water, fed via a swivel up the inside of the drill stem, is maintained to keep the core and doorstopper in "suspension".
- * The drill string is withdrawn from the borehole and the doorstopper with attached core is recovered from the core barrel.
- * The doorstopper, now with core attached, is reconnected to the socket on the insertion trolley and the final relaxed state equilibrium strain readings are recorded.

PROBLEMS.

- * Scheduled and unscheduled interruption of the air and/or water supply may delay or disrupt stress measurement activities.
- * Similarly, the demands of the stope working environment, such as scraping operations, may cause interruptions and delays.
- * Worn flattening and polishing bits may result in an unacceptably concave or insufficiently polished measuring surface.
- * Cleaning cloths may get stuck in the borehole and may also become detached from the insertion rods.

- * The trolley may become jammed in the borehole before reaching the back of the hole.
- * Loose rocks or dislocations in the borehole may prevent penetration of the installation apparatus.
- * High temperature and humidity conditions may result in deterioration of doorstopper and glue component quality if stored or taken underground repeatedly.
- * Chilled washing water may reduce rock temperature and affect the glue curing process.
- * The glue may set too quickly and harden before the strain gauge makes contact with the back of the hole.
- * The glue consistency may be too thin resulting in the glue being squeezed out completely between the strain gauge and the rock surface by the application pressure.
- * A satisfactory bond between the gauge and the rock may not be achieved because of an incorrect glue mix, dirty gauge and/or rock surfaces, an excessively rough rock surface, or an insufficiently rough gauge surface.
- * The doorstopper may be destroyed by attempting to overcore while a flattening or polishing bit is still attached to the drill string.
- * The doorstopper may become dislodged if the overcoring barrel is misaligned when it reaches the back of the hole, or if the drill string is pressed home too violently.
- * The doorstopper may be damaged if the overcoring barrel is misaligned while overcore drilling takes place.
- * The doorstopper may become dislodged by drilling vibrations or chattering during overcoring.
- * The core may break prematurely on a joint, bedding surface or other discontinuity.
- * Rock cracks and flaws which intersect the measurement surface may invalidate results.
- * The core stem may not break off and immediately come away when the core barrel is withdrawn, thereby increasing the risk of failure during subsequent recovery attempts.
- * The doorstopper and rock core may become separated or damaged by milling action within the core barrel subsequent to successful overcoring.

- * The female socket on the trolley may become worn or damaged through the repeated action of connecting and disconnecting doorstoppers.
- * One or more of the electrical circuits may be faulty causing a set of readings to be incomplete and thus useless.
- * The trailing cable may suffer damage as a result of accident or manhandling.
- * The strain indicator apparatus may malfunction as a result of inadequately charged batteries.

The wide scatter of results obtained during the course of this project may in part be due to inaccuracies introduced by inconsistencies in the measuring procedure described above.

APPENDIX 6. CAVE PRESSURE.

APPENDIX 6A.

CAVE PRESSURE LOAD CELLS - 78A-42W N6 PANEL.

LOAD CELL CALIBRATION (DEVICE No.1).

APPLIED LOAD (kN)	LOADCELL PRESSURE (kPa)				
	CELL No.1	CELL No.2	CELL No.3	CELL No.4	CELL No.5
0	0	0	0	0	0
50	500	500	475	500	575
100	1000	1000	950	1000	1150
150	1450	1450	1400	1425	1575
200	1900	1900	1850	1850	2000
250	2350	2350	2325	2300	2475
300	2800	2800	2800	2750	2950
350	3225	3200	3200	3150	3375
400	3650	3600	3600	3550	3800
450	4075	4000	4000	4050	4250
500	4500	4400	4400	4550	4700
550	4900	4800	4800	4900	5100
600	5300	5200	5200	5250	5500
650	5650	5650	5650	5625	5850
700	6000	6100	6000	6000	6200

APPENDIX 6. CAVE PRESSURE.

APPENDIX 6B(i).

CAVE PRESSURE LOAD CELLS - 78A-42W N6 PANEL.

SITE No.1 - LOAD CELL READINGS.

DATE	FACE DIST.	LOADCELL PRESSURE (kPa)				
		CELL No.1	CELL No.2	CELL No.3	CELL No.4	CELL No.5
19/05/93	4m	0	0	0	0	0
19/06/93	14m	0	0	0	0	0
26/06/93	19m	400	270	0	0	0
29/06/93	21m	1000	1100	0	0	0
19/07/93	25m	1100	2000	0	0	0

NOTES.

1. No load was registered by any of the load cells at this site for more than a month after they were installed.
2. Thereafter only Cells No's. 1 and 2 recorded an increase in pressure associated with stress generated in the cave.
3. Cells No's. 3, 4 and 5 failed to record any increase in load for the duration of monitoring, and were assumed to be faulty.
4. After two months all the gauges, which were attached to a gully edge pack, were destroyed by slope closure.

APPENDIX 6. CAVE PRESSURE.

APPENDIX 6B(ii).

CAVE PRESSURE LOAD CELLS - 78A-42W S4 PANEL.

SITE No.2 - LOAD CELL READINGS.

DATE	FACE DIST.	LOADCELL PRESSURE (kPa)				
		CELL No.1	CELL No.2	CELL No.3	CELL No.4	CELL No.5
19/08/93	4m	0	0	0	0	0
09/09/93	14m	0	0	0	0	0
16/09/93	20m	0	0	0	200	10
20/09/93	24m	0	0	0	660	50
22/09/93	26m	0	0	0	810	75
28/09/93	29m	0	0	0	850	82
08/10/93	34m	0	0	0	880	85
18/10/93	40m	0	0	0	1205	0
21/10/93	43m	0	0	0	1250	0
22/10/93	44m	0	0	150	1380	0
25/10/93	46m	0	0	200	1400	0
05/11/93	51m	0	0	200	1480	0
08/11/93	53m	0	0	200	1590	0
09/11/93	54m	0	0	200	1600	0
17/11/93	58m	150	0	530	1800	0
19/11/93	60m	200	0	580	2000	0
24/11/93	62m	200	0	610	2090	0
25/11/93	63m	205	0	610	2100	0
29/11/93	64m	210	0	650	2100	0
01/12/93	66m	230	0	670	2120	0

DATE	FACE DIST.	LOADCELL PRESSURE (kPa)				
		CELL No.1	CELL No.2	CELL No.3	CELL No.4	CELL No.5
03/12/93	68m	240	0	680	2150	0
06/12/93	70m	280	0	780	0	0
08/12/93	72m	290	0	780	0	0
09/12/93	73m	300	0	800	0	0
15/12/93	73m	300	0	810	0	0
20/12/93	73m	360	0	850	0	0

APPENDIX 6. CAVE PRESSURE.**APPENDIX 6B(iii).****CAVE PRESSURE LOAD CELLS - 78A-42W S4 PANEL.****SITE No.3 - LOAD CELL READINGS.**

DATE	FACE DIST.	LOADCELL PRESSURE (kPa)				
		CELL No.1	CELL No.2	CELL No.3	CELL No.4	CELL No.5
09/09/93	6m	0	0	0	0	0
16/09/93	12m	0	0	0	0	0
20/09/93	16m	0	30	0	115	70
22/09/93	18m	0	400	5	190	78
18/10/93	32m	0	780	170	455	160
19/10/93	33m	0	780	170	460	160
21/10/93	35m	0	930	165	555	170
22/10/93	36m	0	970	320	655	325
25/10/93	38m	0	1000	430	755	455
05/11/93	43m	0	152	480	900	650
08/11/93	45m	0	1600	415	990	760
09/11/93	46m	0	1620	525	1000	800
17/11/93	50m	0	2100	170	1180	0
19/11/93	52m	0	2200	200	1210	0
24/11/93	54m	0	2200	220	1350	0
25/11/93	55m	0	2200	220	1400	0
29/11/93	66m	0	2280	230	1440	0
01/12/93	58m	0	2300	260	1470	0
03/12/93	60m	0	2350	300	1500	0
06/12/93	62m	0	2380	300	1500	0

DATE	FACE DIST.	LOADCELL PRESSURE (kPa)				
		CELL No.1	CELL No.2	CELL No.3	CELL No.4	CELL No.5
08/12/93	64m	0	2390	300	1500	0
09/12/93	65m	0	2400	300	1500	0
15/12/93	65m	0	2430	310	1530	0
20/12/93	65m	0	2500	360	1570	0

APPENDIX 7. COMPUTER MODELLING.

APPENDIX 7A(i).

INTUITIVE CAVING MODEL - UDEC INPUT PARAMETERS.

SCENARIO No.1 : INTACT HANGINGWALL MODEL.

SPECIFYING INITIAL SET-UP DATA.

new
heading
INTUITIVE CAVING: SCENARIO 1
set vga
round 0.01
set ovtol 0.01
damp auto

CREATING A 60m x 60m MODEL WINDOW.

block 0 0 0 60 60 60 60 0

CREATING THE HANGINGWALL PLANE.

split 0 30 60 30

CREATING THE BEDDING PLANE.

split 0 31 60 31

CREATING THE LEFT-HAND FACE.

crack 10 30 10 29

CREATING THE RIGHT-HAND FACE.

crack 50 30 50 29

CREATING THE STOPE FOOTWALL.

crack 10 29 50 29

CREATING THE LEFT-HAND SUPPORT ZONE.

crack 14 30 14 29
crack 17 30 17 29

CREATING THE RIGHT-HAND SUPPORT ZONE.

crack 46 30 46 29
crack 43 30 43 29

DEFINING THE ROCKMASS PROPERTIES.

prop mat=1 b=20e9 s=18e9 d=2700

DEFINING THE SUPPORT PROPERTIES.

prop mat=2 b=20e7 s=18e7 d=1000

DEFINING THE BEDDING PLANE PROPERTIES.

prop jmat=1 jkn=1e11 jks=1e11 jcoh=0 jfric=30

DEFINING THE SUPPORT CONTACT PROPERTIES.

prop jmat=2 jkn=1e11 jks=1e11 jcoh=0 jfric=15

GENERATING FINITE DIFFERENCE ZONES.

gen edge 5

SPECIFYING GRAVITATIONAL LOADING.

grav 0 -10

SPECIFYING STRESS CONDITIONS.

boundary stress -25e6 0 -50e6

insitu stress -25e6 0 -50e6

FIXING THE BOUNDARY.

bound corner 63 24 yvel=0

bound corner 63 24 xvel=0

REQUESTING HISTORIES.

hist unbal

hist damp

hist yvel 30 30

CYCLING TO EQUILIBRIUM AND SAVING EQUILIBRIUM STATE.

c 5000

sav caveleql.sav

MINING THE STOPE.

del blo 647 1267 1590

SPECIFYING THE SUPPORT AND SUPPORT CONTACT PROPERTIES.

change 13.9 17.1 28.9 30.1 mat=2

change 13.9 17.1 28.9 30.1 jmat=2

change 42.9 46.1 28.9 30.1 mat=2

change 42.9 46.1 28.9 30.1 jmat=2

NULLIFYING EQUILIBRIUM SUPPORT STRESS.

insitu 15 17 29 30 stress 0 0 0

insitu 43 45 29 30 stress 0 0 0

CYCLING AND SAVING MODEL STATE AT 0.5 SECOND INTERVALS.

c t 0.5

save cavels1.sav

c t 0.5

save cavels2.sav

c t 0.5

save cavels3.sav

c t 0.5

save cavels4.sav

ct 0.5
save cavels5.sav
ct 0.5
save cavels6.sav

APPENDIX 7. COMPUTER MODELLING.**APPENDIX 7A(ii).****INTUITIVE CAVING MODEL - UDEC INPUT PARAMETERS.****SCENARIO No.2 : RIGID HANGINGWALL MODEL.****SPECIFYING INITIAL SET-UP DATA.**

new
 heading
 INTUITIVE CAVING: SCENARIO 2
 set vga
 round 0.01
 set ovtol 0.01
 damp auto

CREATING A 60m x 60m MODEL WINDOW.

block 0 0 0 60 60 60 60 0

CREATING THE HANGINGWALL PLANE.

split 0 30 60 30

CREATING THE BEDDING PLANE.

split 0 31 60 31

CREATING THE LEFT-HAND FACE.

crack 10 30 10 29

CREATING THE RIGHT-HAND FACE.

crack 50 30 50 29

CREATING THE STOPE FOOTWALL.

crack 10 29 50 29

CREATING THE LEFT-HAND SUPPORT ZONE.

crack 14 30 14 29

crack 17 30 17 29

CREATING THE RIGHT-HAND SUPPORT ZONE.

crack 46 30 46 29

crack 43 30 43 29

GENERATING VERTICAL FRACTURES.

jreg 8 30 8 31 52 31 52 30

jset 93 0 1.25 0 0 0 0.5 0 8 30 0

DEFINING THE ROCKMASS PROPERTIES.

prop mat=1 b=32e9 s=28e9 d=2700 dil=0

DEFINING THE SUPPORT PROPERTIES.

prop mat=2 b=20e6 s=18e6 d=1000

DEFINING THE BEDDING PLANE PROPERTIES.

prop jmat=1 jkn=1e11 jks=1e11 jcoh=0 jfric=90

DEFINING THE SUPPORT CONTACT PROPERTIES.

prop jmat=2 jkn=1e11 jks=1e11 jcoh=0 jfric=90

DEFINING THE VERTICAL FRACTURE PROPERTIES.

prop jmat=3 jkn=1e11 jks=1e11 jcoh=0 jfric=0 jdl 0

GENERATING FINITE DIFFERENCE ZONES.

gen edge 5

SPECIFYING GRAVITATIONAL LOADING.

grav 0 -10

SPECIFYING STRESS CONDITIONS.

boundary stress -25e6 0 -50e6

insitu stress -25e6 0 -50e6 szz -15e6

FIXING THE BOUNDARY.

bound corner 63 24 yvel=0

bound corner 63 24 xvel=0

REQUESTING HISTORIES.

hist unbal

hist damp

CYCLING TO EQUILIBRIUM AND SAVING EQUILIBRIUM STATE.

c 5000

sav cave2eql.sav

MINING THE STOPE.

del blo 647 1267 1590

SPECIFYING SUPPORT AND SUPPORT CONTACT PROPERTIES.

change 13.9 17.1 28.9 30.1 mat=2

change 13.9 17.1 28.9 30.1 jmat=2

change 42.9 46.1 28.9 30.1 mat=2

change 42.9 46.1 28.9 30.1 jmat=2

SPECIFYING VERTICAL FRACTURE PROPERTIES.

change angle 88 92 jmat=3

APPLYING HORIZONTAL DILATIONARY STRESS.

bound int 7.9 8.1 29.9 30.6 xload 5e6

bound int 51.9 52.1 29.9 30.6 xload -5e6

REDUCING EQUILIBRIUM SUPPORT STRESS.

insitu 14 17 29 30 stress 0 0 1e6

insitu 43 46 29 30 stress 0 0 1e6

CREATING A SLOT TO INITIATE A CAVE.

del 30 30.5 30 31

CYCLING AND SAVING MODEL STATE AT 1.0 SECOND INTERVALS.

c t 1

save cave2s1.sav

c t 1

save cave2s2.sav

c t 1

save cave2s3.sav

c t 1

save cave2s4.sav

c t 1

save cave2s5.sav

c t 1

save cave2s6.sav

c t 1

save cave2s7.sav

c t 1

save cave2s8.sav

c t 1

save cave2s9.sav

c t 1

save cave2s10.sav

APPENDIX 7. COMPUTER MODELLING.**APPENDIX 7A(iii).****INTUITIVE CAVING MODEL - UDEC INPUT PARAMETERS.****SCENARIO No.3 : PSEUDO-PLASTIC MODEL.****SPECIFYING INITIAL SET-UP DATA.**

new
 heading
 INTUITIVE CAVING: SCENARIO 3
 set vga
 round 0.01
 set ovtol 0.01
 damp auto

CREATING A 60m x 60m MODEL WINDOW.

block 0 0 0 60 60 60 0

CREATING THE HANGINGWALL PLANE.

split 0 30 60 30

CREATING THE BEDDING PLANE.

split 0 31 60 31

CREATING THE LEFT-HAND FACE.

crack 10 30 10 29

CREATING THE RIGHT-HAND FACE.

crack 50 30 50 29

CREATING THE STOPE FOOTWALL.

crack 10 29 50 29

CREATING THE LEFT-HAND SUPPORT ZONE.

crack 14 30 14 29

crack 17 30 17 29

CREATING THE RIGHT-HAND SUPPORT ZONE.

crack 46 30 46 29

crack 43 30 43 29

GENERATING VERTICAL FRACTURES.

jreg 8 30 8 31 52 31 52 30

jset 90 0 1.25 0 0 0 0.5 0 8 30 0

DEFINING THE ROCKMASS PROPERTIES.

prop mat=1 b=32e9 s=28e9 d=2700 dil=3

DEFINING THE SUPPORT PROPERTIES.

prop mat=2 b=25e7 s=22e7 d=1000

DEFINING THE BEDDING PLANE PROPERTIES.

prop jmat=1 jkn=1e11 jks=1e11 jcoh=0 jfric=0

DEFINING THE SUPPORT CONTACT PROPERTIES.

prop jmat=2 jkn=1e11 jks=1e11 jcoh=0 jfric=60

DEFINING THE VERTICAL FRACTURE PROPERTIES.

prop jmat=3 jkn=1e11 jks=1e11 jcoh=0 jfric=1 jdil=10

GENERATING FINITE DIFFERENCE ZONES.

gen edge 5

SPECIFYING GRAVITATIONAL LOADING.

grav 0 -10

SPECIFYING STRESS CONDITIONS.

boundary stress -25e6 0 -50e6

insitu stress -25e6 0 -50e6 szz -15e6

FIXING THE BOUNDARY.

bound corner 63 24 yvel=0

bound corner 63 24 xvel=0

bound corner 50 63 xvel=0

bound corner 24 37 xvel=0

REQUESTING HISTORIES.

hist unbal

hist damp

hist yvel 30 30

CYCLING TO EQUILIBRIUM AND SAVING EQUILIBRIUM STATE.

c 5000

sav cave3eq1.sav

MINING THE STOPE.

del blo 647 1267 1590

SPECIFYING SUPPORT AND SUPPORT CONTACT PROPERTIES.

change 13.9 17.1 28.9 30.1 mat=2

change 13.9 17.1 28.9 30.1 jmat=2

change 13.9 17.1 30.9 31.1 jmat=2

change 42.9 46.1 28.9 30.1 mat=2

change 42.9 46.1 28.9 30.1 jmat=2

change 42.9 46.1 30.9 31.1 jmat=2

SPECIFYING VERTICAL FRACTURE PROPERTIES.

change angle 88 92 jmat=3

APPLYING HORIZONTAL DILATORY STRESS.

bound int 7.1 29.9 30.6 xload 5e6
bound int 5.5 32.1 29.9 30.6 xload -5e6

REDUCING EQUILIBRIUM SUPPORT STRESS.

insitu 14 17 29 30 stress 0 0 1e6
insitu 43 46 29 30 stress 0 0 1e6

CREATING A SLOT TO INITIATE A CAVE.

del 30 31 30 31

CYCLING AND SAVING MODEL STATE AT 1,0 SECOND INTERVALS.

ct 1
save cave3s1.sav
ct 1
save cave3s2.sav
ct 1
save cave3s3.sav
ct 1
save cave3s4.sav
ct 1
save cave3s5.sav
ct 1
save cave3s6.sav
ct 1
save cave3s7.sav
ct 1
save cave3s8.sav
ct 1
save cave3s9.sav
ct 1
save cave3s10.sav

APPENDIX 7. COMPUTER MODELLING.

APPENDIX 7B.

CAVING SIMULATION MODEL - INPUT PARAMETERS.

CAVE MINING IN PSEUDO-BRECCIA : DYNAMIC MODEL.

SPECIFYING INITIAL SET-UP DATA.

new
heading
CAVE MINING IN PSEUDO-BRECCIA
set vga
round 0.05
set ovtol 10
damp auto

CREATING A 40m WIDE x 15m HIGH MODEL WINDOW.

This window was originally 40m wide x 30m high but the size of the model had to be reduced to decrease calculation run times. Because symmetry is used the model window is actually 80m wide x 30m high.

* Create block
block 0 -1 0 15 40 15 40 -1

CREATING THE H/W & F/W PARTINGS AND THE BEDDING PLANES.

Four bedding planes were created 0,5m; 1,0m; 2,0m and 3,0m above the stope hangingwall. Given the constraints on calculation run times, this is a reasonable approximation of typical strata spacing measured in the 5A North area.

* Create bedding planes
split 0 0 40 0
split 0 1 40 1
split 0 1.5 40 1.5
split 0 2 40 2
split 0 3 40 3
split 0 4 40 4

CREATING THE INITIAL STOPE FACE.

The future stope face was specified to commence at, and advance from, the 7,25m ledging half-span position.

* Create stope face
Split 7.25 0 7.25 1

CREATING THE IMMEDIATE FACE FRACTURES.

The immediate face fractures were also spaced 0,25m apart but dip at 90° and extend 3,5m ahead of the face. They are truncated by the hanging and footwall parting planes.

* Create immediate face fractures
jreg 20 0 20 1 23.5 1 23.5 0
jset 90 0 2 0 0 0 0.25 0 20 0 0

CREATING THE IMMEDIATE HANGINGWALL FRACTURES.

The immediate hangingwall fractures were spaced 0,25m apart, extending 1m up into the hangingwall rock. They dip towards the face at 80° and are formed up to 3m ahead of the face.

* Create immediate hangingwall fractures

```
jreg 0 1 0 2 23 2 23 1
```

```
jset 100 0 100 0 0 0 0.25 0 0 1 0
```

CREATING THE FAR HANGINGWALL FRACTURES.

The fractures above the 1m bedding plane were spaced 0,5m apart, extending 3m up into the hangingwall rock. They dip towards the face at 70° and are formed up to 2m ahead of the face.

* Create far hangingwall fractures

```
jreg 0 2 0 4 22 4 22 2
```

```
jset 110 0 10 0 0 0 0.5 0 22 2 0
```

GENERATING FINITE DIFFERENCE ZONES WITHIN THE BLOCKS.

The blocks within the model were allowed to deform in accordance with the elastic properties specified in subsequent commands.

* Generate finite difference zones

```
gen edge 10
```

DEFINING THE DEFAULT ROCKMASS AND DISCONTINUITY PROPERTIES.

The following commonly accepted specifications for deep hard rock conditions served mainly as the default properties for materials and joints within the model. Most of these properties were changed after the model had been brought to initial equilibrium.

* Default rockmass and discontinuity properties

```
prop mat=1 b=30e9 s=28e9 d=2700 coh=50e6 fric=40 dil=1
```

```
prop jmat=1 jkn=1e11 jks=1e11 jfric=40 jcoh=5e6 jdil=1
```

DEFINING THE HANGINGWALL PROPERTIES.

The following values for bulk modulus, shear modulus, density, cohesion, friction and dilation were specified for the hangingwall rockmass during stoping operations. The somewhat lower material properties were specified in an attempt to simulate the weakening effect of blasting on particularly the strength, integrity and stability of the hangingwall rock.

* Define rockmass properties of the immediate hangingwall

```
prop mat=2 b=28e9 s=26e9 d=2700 coh=2e5 fric=15 dil=2
```

DEFINING BEDDING PLANE PROPERTIES.

The following properties were specified for the bedding planes.

* Define bedding plane properties

```
prop jmat=2 jkn=1e11 jks=1e11 jcoh=10 jfric=20 jdil=4
```

DEFINING FRACTURE PLANE PROPERTIES.

The following values for normal stiffness, shear stiffness, friction, cohesion and dilation were specified for the fracture planes during stoping operations. The somewhat lower joint properties were specified in an attempt to simulate the weakening effect of blasting on particularly the strength, integrity and stability of the hangingwall rock.

* Define fracture properties of the immediate hangingwall

```
prop jmat=3 jkn=1e11 jks=1e11 jcoh=0 jfric=20 jdil=2
```

SPECIFYING GRAVITATIONAL LOADING.

The direction and magnitude of gravity were specified.

* Set gravity
set grav 0 -10

SPECIFYING INSITU STRESS CONDITIONS.

The vertical insitu stress was set at 55MPa. The horizontal insitu stress was set at 18MPa. The Poisson's ratio was set at 0.33.

* Set insitu stress
insitu stress -18e6 0 -55e6 sxx -18e6

SPECIFYING BOUNDARY STRESS CONDITIONS.

The same stress conditions were applied to the boundaries of the blocks.

* Set boundary stress
boundary corner 37 50 stress -18e6 0 -55e6

FIXING THE BOUNDARY.

The boundaries of the block were fixed with regard to providing two axes of symmetry.

* Fix boundaries to simulate symmetry
bound corner 24 37 xvel 0
bound corner 50 63 xvel 0
bound corner 63 24 yvel 0

DELETING SMALL BLOCKS.

Small blocks which do not have a large effect on the results but have a significant effect on calculation run times were deleted.

* Delete small blocks
del area 5e-2

REQUESTING HISTORIES.

A variety of histories which are helpful in determining the state of the model and which assist in the interpretation of results, were requested.

* Request histories

his unbal	*Hist. 1
his dmp	*Hist. 2
his yvel 9 1.3	*Hist. 3
his xvel 9 1.3	*Hist. 4
his ydisp 9 1.3	*Hist. 5
his xdisp 9 1.3	*Hist. 6
his ydisp 9 3.5	*Hist. 7
his xdisp 9 3.5	*Hist. 8
his ydisp 12 1.3	*Hist. 9
his xdisp 12 1.3	*Hist. 10
his syy 20 0.5	*Hist. 11
his sxx 20 0.5	*Hist. 12

CYCLING TO EQUILIBRIUM.

The model was required iterate through 5000 time-steps under field stress conditions to establish equilibrium prior to mining. * Cycle for equilibrium
cyc 5000

SAVING THE EQUILIBRIUM STATE.

The state of equilibrium was saved.

* Save state of equilibrium

sav caveDeq1.sav

COMMENCING STOPING OPERATIONS.

The equilibrium state was recalled and ledging was carried out to create an initial stope span of 14.5m which was sufficient to allow the support specified below to be installed on either side of the plane of symmetry. This was mining step 1.

* re caveDeq1.sav

* Mining step 1

del blo 992

SPECIFYING SUPPORT PROPERTIES.

Three rows of 400kN hydraulic props spaced 1.5m apart on strike and installed 3.5m from the face were simulated. Individual support units were required to perform according to the specifications set out in Table 1 which depicts typical load-compression characteristics for a 40t hydraulic prop.

* Install support

support 2.25 0.5 wid=3 seg=2 mat=4

prop mat=4 kn=1e9 ks=1e9 c=1000 b=4e6 s=2e6 coh=1e35 cten=40e3

table 1 0 0 0.01 180e3 0.03 400e3 0.3 400e3

SPECIFYING ROCK, BEDDING PLANE AND FRACTURE PROPERTIES.

The properties of the hangingwall rock material, fracture surfaces and bedding planes, to a distance of 2m ahead of the advancing face, were changed from the default values to simulate the weakening effect of blasting on the strength, integrity and stability of this rock. The default fracture properties are so high that for all practical purposes the program perceives the area beyond this as being solid.

* Change properties of immediate hangingwall after blast

change angle -5 5 jmat=3

change reg 0 0.99 0 4.1 9.5 4.1 9.5 3.99 jmat=1

change reg 0 0 0 3.9 9.5 3.9 9.5 0 mat=3

change angle 60 110 jmat=2

change reg 9.5 0 9.5 4 40 4 40 0 jmat=1

change 0 9.5 0 3.5 jcons=5

CYCLING AND SAVING MODEL STATES.

The model was allowed to iterate for intervals equivalent to one second of real time (each requiring between 27000 and 29000 iteration cycles). At each "plot block" instruction, for this and subsequent steps (ie. after each one second interval), the model state was saved to a "movie" file capable of plotting the sequence of events in the model for later viewing. The model was allowed to run for two one-second intervals to bring the first mining step to equilibrium. All calculated data was saved at the end of the mining step for analysis at a later stage. Actual modelling time for this single mining step amounted to six hours. The later, more complex steps took up to thirty hours each.

* Cycle, save state and generate movie

movie file dyncave1.mov

movie on

window -1 24 -1 17

c t 1.0

plo blo supp blue
 c t 1.0
 plo blo supp blue
 sav step1.sav

ADVANCING THE STOPE FACE.

The stope face was advanced by 1.5m. This was mining step 2.

* Mining step 2
 del 7.25 9 0 1

ADVANCING THE SUPPORT.

The back line of prop support was moved forward by removing the previous support and installing three new lines of props 1.5m ahead of the previous position.

* Move support
 supp 2.25 0.5 rem
 supp 3.75 0.5 wid 3 seg 2 mat=4
 prop mat=4 kn=1e9 ks=1e9 d=1000 b=4e6 s=2e6 coh=1.35 cten=40e3
 table 1 0 0 0.01 180e3 0.03 400e3 0.3 400e3

ADVANCING THE WEAK ROCK ZONE.

The blast weakened rock zone was advanced ahead of the stope face.

* Change rock material properties
 change reg 0 0 0 3.9 11 3.9 11 0 mat=3

ADVANCING THE FRACTURE ZONE.

The stress fracture zone was advanced ahead of the stope face.

* Change stress fracture properties
 change angle -5 5 jmat=3
 change reg 0 3.99 0 4.1 11 4.1 11 3.99 jmat=1
 change angle 60 110 jmat=2
 change reg 11 0 11 4 40 4 40 0 jmat=1
 change 0 11 0 3.5 jcons=5

CYCLING AND SAVING MODEL STATES.

The model was once again brought to equilibrium by allowed it to iterate for two intervals equivalent to one second of real time each. Model state information was saved to the previously created movie file after each interval. The calculation data was saved to a new unique file at the end of each mining step.

* Cycle, generate movie and save state
 c t 1.0
 plo blo supp blue
 c t 1.0
 plo blo supp blue
 sav step2.sav

REPEATING THE STOPING PROCESS.

The process of advancing the stope face, the support and the fracture zone, and of saving the movie and the model state files was repeated for nine mining steps. The stope face was thus advanced just short of 20m. The entire model took roughly 150 hours to run. For completeness, the balance of the input data batch file is continued below:-

* Mining step 3
del 9 10.5 0 1

* Move support
supp 3.75 0.5 rem
supp 5.25 0.5 wid 3 seg 2 mat=4
prop mat=4 kn=1e9 ks=1e9 d=1000 b=4e6 s=2e6 coh=1e35 cten=40e3
table 1 0 0 0.01 180e3 0.03 400e3 0.3 400e3

* Change rock material properties
change reg 0 0 0 3.9 12.5 3.9 12.5 0 mat=3

* Change stress fracture properties
change angle -5 5 jmat=3
change reg 0 3.99 0 4.1 12.5 4.1 12.5 3.99 jmat=1
change angle 60 110 jmat=2
change reg 12.5 0 12.5 4 40 4 40 0 jmat=1
change 0 12.5 0 3.5 jcons=5

* Cycle, generate movie and save state

c t 1
plo blo supp blue
c t 1
plo blo supp blue
c t 1
plo blo supp blue
c t 1
plo blo supp blue
sav step3.sav

* Mining step 4
del 10.5 12 0 1
del blo 60944

* Move support
supp 5.25 0.5 rem
supp 6.75 0.5 wid 3 seg 2 mat=4
prop mat=4 kn=1e9 ks=1e9 d=1000 b=4e6 s=2e6 coh=1e35 cten=40e3
table 1 0 0 0.01 180e3 0.03 400e3 0.3 400e3

* Change rock material properties
change reg 0 0 0 3.9 14 3.9 14 0 mat=3

* Change stress fracture properties
change angle -5 5 jmat=3
change reg 0 3.99 0 4.1 14 4.1 14 3.99 jmat=1
change angle 60 110 jmat=2
change reg 14 0 14 4 40 4 40 0 jmat=1
change 0 14 0 3.5 jcons=5

* Cycle, generate movie and save state

c t 1

pl0 blo supp blue

c t 1

pl0 blo supp blue

c t 1

pl0 blo supp blue

c t 1

pl0 blo supp blue

sav step4.sav

* Mining step 5

del 12 13.5 0 1

* Move support

supp 6.75 0.5 rem

supp 8.25 0.5 wid 3 seg 2 mat=4

prop mat=4 kn=1e9 ks=1e9 d=1000 b=4e6 s=2e6 coh=1e35 cten=40e3

table 1 0 0 0.01 180e3 0.03 400e3 0.3 400e3

* Change rock material properties

change reg 0 0 0 3.9 15.5 3.9 15.5 0 mat=3

* Change stress fracture properties

change angle -5 5 jmat=3

change reg 0 3.99 0 4.1 15.5 4.1 15.5 3.99 jmat=1

change angle 60 110 jmat=2

change reg 15.5 0 15.5 4 40 4 40 0 jmat=1

change 0 15.5 0 3.5 jcons=5

* Cycle, generate movie and save state

c t 1

pl0 blo supp blue

c t 1

pl0 blo supp blue

c t 1

pl0 blo supp blue

c t 1

pl0 blo supp blue

sav step5.sav

* Mining step 6

del 13.5 15 0 1

* Move support

supp 8.25 0.5 rem

supp 9.75 0.5 wid 3 seg 2 mat=4

prop mat=4 kn=1e9 ks=1e9 d=1000 b=4e6 s=2e6 coh=1e35 cten=40e3

table 1 0 0 0.01 180e3 0.03 400e3 0.3 400e3

plo blo supp blue

ci 1

blo blo supp blue

ct 1

plo blo supp blue

sav step8.sav

* Mining step 9

del 18 19.5 0 1

*** Move support**

supp 12.75 0.5 rem

supp 14.25 0.5 wid 3 seg 2 mat=4

```
prop mat=4 kn=1e9 ks=1e9 d=1000 b=4e6 s=2e6 coh=1e35 cten=40e3
```

table	1	0	0	0.01	180e3	0.03	400e3	0.3	400e3

* Change rock material properties

change reg 0 0 0 3.9 21.5 3.9 21.5 0 mat=3

* Change stress fracture properties

change angle -5.5 jmat=3

```
change reg 0 3.99 0 4.1 21.5 4.1 21.5 3.99 jmat=1
```

```
change angle 60 110 jmat=2
```

```
change reg 21.5 0 21.5 4 40 4 40 0 jmat=1
```

change 0 21.5 0 3.5 icons=5

*** Cycle, generate movie, save state and close movie file**

ct i

plø blo supp blue

et l

plo blo supp blue

ct 1

blo blo supp blue

ct 1

plo blo sarp blue

cti

plo blo supp blue

ct 1

blo blo supp blue

ct 1

blo blo supp blue

ct 1

blo blo supp blue

ct 1

plo blo supp blue

```
sav step9.sav
```

movie off

return

* Change rock material properties

change reg 0 0 0 3.9 17 3.9 17 0 mat=3

* Change stress fracture properties

change angle -5 5 jmat=3

change reg 0 3.99 0 4.1 17 4.1 17 3.99 jmat=1

change angle 60 110 jmat=2

change reg 17 0 17 4 40 4 40 0 jmat=1

change 0 17 0 3.5 jcons=5

* Cycle, generate movie and save state

c t 1

plo blo supp blue

c t 1

plo blo supp blue

c t 1

plo blo supp blue

c t 1

plo blo supp blue

c t 1

plo blo supp blue

c t 1

plo blo supp blue

c t 1

plo blo supp blue

sav step6.sav

* Mining step 7

del 15 16.5 0 1

* Move support

supp 9.75 0.5 rem

supp 11.25 0.5 wid 3 seg 2 mat=4

prop mat=4 kn=1e9 ks=1e9 d=1000 b=4e6 s=2e6 coh=1e35 cten=40e3

table 1 0 0 0.01 180e3 0.03 400e3 0.3 400e3

* Change rock material properties

change reg 0 0 0 3.9 18.5 3.9 18.5 0 mat=3

* Change stress fracture properties

change angle -5 5 jmat=3

change reg 0 3.99 0 4.1 18.5 4.1 18.5 3.99 jmat=1

change angle 60 110 jmat=2

change reg 18.5 0 18.5 4 40 4 40 0 jmat=1

change 0 18.5 0 3.5 jcons=5

* Cycle, generate movie and save state

c t 1

plo blo supp blue

c t 1

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