# PLANNING A MECHANISED CAVE WITH COARSE FRAGMENTATION IN KIMBERLITE

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## ABSTRACT

Premier Diamond mine had to plan, develop and operate a low cost, mass mining method to recover extensive ore reserves below a 75 metre thick, dipping gabbro sill. The mining method had to preclude the extraction of as much of the 52 million tons of barren waste contained in the gabbro sill as possible and ensure the safety of personnel and operations. Consequent on the failure of the open stope mining method first attempted, a geotechnical investigation showed that ore recovery by caving methods was possible. The fragmentation that reported to the drawpoints would be coarse and the production horizon would be situated in relatively weak rock. The increased depth of mining, experience on the mine in using LHD's, changes in mining technology and the high production rate required, determined that Premier Diamond Mine could implement a mechanised cave using LHD's for ore extraction.

Parameters that need to be defined to successfully exploit any orebody considering cave mining methods include (Cummings et al., 1984):

* The area that must be undercut to induce continuous caving.
* The fragmentation that will result as the orebody caves and the fragmentation size distribution that will report to drawpoints. The size distribution will determine drawpoint spacing, secondary blasting procedures and equipment, ore pass diameters, as well as tunnel and LHD sizes.
* The rock mass response to the mining operations must be understood and used to optimise the mining sequence. Once the rock on the production level has been damaged by high abutment stresses, maintaining the stability of excavations can be expensive and time consuming.
* Support systems and time of installation must be carefully planned and controlled. The function and potential method of failure of the support elements must be understood.
* Draw control and analysis of draw control data is important to ensure that premature waste ingress is minimised and that stress related problems that can result in cave "sit-downs" do not occur.

Research was undertaken by Premier Mine personnel into aspects of cave mining prior to the implementation of a panel retreat cave in the BA5 mining block. This included visits to cave mines using LHD's for extraction in other areas of the world. Problems after initial implementation of the BA5 cave forced further investigations by the Geotechnical Department.

Premier Diamond Mine had experienced problems in predicting the area that would need to be undercut to induce continuous caving in caves above the gabbro sill. D.H. Laubscher's correlation of Mining Rock Mass Rating with hydraulic radius was found to be the most accurate method of predicting the area that needs to be undercut to induce continuous caving.

An expert system to predict the fragmentation that will result as ore caves and moves through the draw column to drawpoints below was developed and successfully calibrated at Premier Diamond Mine. Prediction of the fragmentation size distribution and hangup frequency have been used to plan several aspects of cave mining.

The rock mass response to cave mining has been investigated by detailed monitoring of stress changes, displacements and support damage associated with development and operation of the BA5 mining block. Detailed numerical models have been accurately calibrated and used to plan and implement an alternate mining sequence and improve support systems.

Draw control and draw control data analysis have shown that in a cave experiencing fragmentation as coarse as seen at Premier Diamond Mine, drawzones relate to drawbells and not drawpoints. Material flow can be influenced and controlled by the geometry of the extraction horizon. Drawpoints can be widely spaced with a considerable saving in support and development costs. Static columns were not created and there was no loss of ore reserves.

## SAMEVATTING

Die kimberlietpyp wat deur Premier Diamantmyn ontgin word, is die grootste in Suid-Afrika en beslaan 32 hektaar op die oppervlakte. Wat die pyp besonders maak, is die voorkoms van 'n 75 meter gabbroplaat bar rotslaag wat horisontaal deur die pyp sny op 'n diepte van 380 meters. Die diamantneerslae aan die bokant van hierdie rotslaag is aanvanklik ontgin as 'n oopgroef operasie. Later is daar ook ondergronds met oopbank- en roosterif-blokstortingmetodes begin.

Om die ontginning van diamante onder die gabbro rotsplaat moontlik te maak, moes Premier Myn 'n mynbou metode ontwikkel wat nie alleen koste effektief en veilig vir die werkers is nie, maar ook die minimum hoeveelheid nie-diamantdraende erts vanuit die 52 miljoen ton bar rotsplaat na die oppervlak bring.

Aanvanklik is die oopvlakafboumynboumetode aangewend, maar sonder sukses. Hierna het 'n geotegniese ondersoek gevolg wat getoon het dat die blokstortingmetode moontlik was. Growwe verbokkeling van ersts by trekpunte, onstabiele rotsformasies, sowel as erge afbreking van ersts wanneer in kontak met water, is as primere probleme geidentifiseer.

Alhoewel die De Beers Groep alreeds 24 blokinstortingmynoperasies suksesvol ontgin het met roostersif en skraper metode, en Premier Myn ook ondervinding van 4 soortgelyke ontginningsmetodes sedert die 1970's gehad het, het dit geblyk dat faktore soos growwe verbrokkeling, toename in diepte van die myn, die gebruik van laai-stort-waens (LSW's), veranderinge in tegnologie, en die noodsaaklikheid van verhoodgde produksie, vereis het dat Premier Myn 'n gemeganiseerde ontginningsmetode moes in werking stel.

Die volgende parameters moes goed gedefinieer en afgebaken word ten einde die ertsliggaam met die blokstortingmetode te ontgin (Cummings et al., 1984):

* Die gebied van ondersny moet word om storting te bewerkstellig, moet deeglik afgebaken wees.

* Die hoeveelheid verbrokkeling van erts by ineenstorting en die hoeveelheid verbrokkelde erts by trekpunte, moet bepaal word. Die graad van verbokkeling van die ertsliggaam sal die posisies van die trekpunte, die sekondere skietprosedures en skiettoerusting, die deursnitte van die ertsgange en die tonnels bepaal, sowel as die groottes van meganiese LSW's sal hiervan afhang.

* Die reaksie van die rotsformasie op die myn proses moet in ag geneem en ingespan word om die ontginningproses te bevorder. Daar moet altyd ag geslaan word op die feit dat onoordeelkundige ondersnying 'n duurder instandhoudings proses sal vereis.

* Die bepaling en tyds berekening van tonnelbestutting moet deeglik gedoen word. Die beperkinge en voordele van bestuttingmetodes moet goed bekend wees.

* Trekpuntbeheer en -ontleding is van groot belang om te verseker dat groot hoeveelhede afvalrots nie ontgin word nie. Sorg moet ook gedra word dat onoordeelkundige gewigsverspreiding nie storting sal teenwerk nie.

Aangesien Premier Myn en ander De Beers myne van voorneme is om van die gemeganiseerde blokstortingmynboumetode in die toekoms gebruik te maak, is daar by Premier Myn 'n uitgebreide ondersoek ingestel na hierdie mynbou metode.

Dr. D.H. Laubscher het met sy kennis en ondervinding 'n reuse bydrae gelewer. Ook die skrywer het wereld wyd myne wat blokstortingmetodes gebruik besoek om inligting in te win en probleme te bespreek. Kontak met hierdie myne word tans steeds behou. Myne wat besoek is sluit in El Teniente, Andina, Salvador (Chile), Henderson, Magma Copper (USA), Grasberg (Indonesia) en Philex (Philipines).

Daar is gevind dat Laubscher se korrelasie tussen rotsmassa klassifikasie en die hidrolise straal die beste metode is om te bepaal hoe groot die ondersnyde area moet wees om voortdurende, beheerde storting te bewerkstellig. Dit is bepaal deur 'n deeglike onleding van die data wat versamel is van verskeie mynblokke wat in verskillende rotstipes by Premier Myn ontgin is. 'n Omvattende studie is onderneem om 'n numeriese model te skep wat beheerde storting kan voorspel. Daar is gevind dat 'n numeries model wel vir voorspelling gebruik kan word, maar dat dit moelik is om presiese parameters te bepaal vir die opstel van so 'n numeries model.

'n Doelontwerpte sisteem is ontwikkel om die verbrokkeling te voorspel wat ontstaan wanneer die erts stort en deur die ertsglybane na die trekpunte beweeg. Hierdie model is by Premier Myn gekalibreer en word gebruik om die verbrokkeling te voorspel wat by trekpunte voorkom. Hierdie inligting is gebruik om die posisies van die trekpunte te bepaal, asook die sekondere skietwerk prosedures en die algemene produksietempo.

Die rotsmassareaksie op die storting is by Premier Myn noukeuring ondersoek. Daar is, in besonder, gelet op spanningsveranderinge en verdringing, sowel as skade aan bestutting tydens die toepassing van die mynbou operasie in die BA5 blok. Gedetailleerd numeriese modelle is met groot akkuraatheid gekalibreer en gebruik om alternatiewe mynbou prosedures en bestutting te evalueer. Na aanleiding van hierdie resultate het Premier Myn die vooruit ondersny begin gebruik. Bestuttingstelsels is in fases beplan en sodoende is daar groot besparing bewerkstellig.

Deur kontrolering en ontleding van erts by die trekpunte, het dit geblyk dat met die verbrokkelverdeling soos in Premier Myn se geval, trekslote meer betrekking her op trekregters as op trekpunte. Die ondervinding  wat by Premier Myn en ander myne opgedoen is, kon nuttig aangewend word om optimum posisies van trekpunte te bepaal sodoende is aansienlik koste bespaar by die ontsluiting- en bestuttingprosesse, sonder dat diamantdraende erts verloor is.

Studies wat uigevoer is tydens die voltooiing van hierdie proefskif het duidelik getoon dat baie aspekte van blokstortmynbou wat proefondervindelik in die verlede bepaal is, nou beter verstaan en voorspel kan word deur deeglike geotegniese studie van die rotsmassa en die storting proses. Die gebruik van 'n geskikte rotsmassaklassifikasie sisteem en die omvattende gebruik van kragtige wiskundige berekinge wat tans beskikbaar is, is noodsaaklik by die bestudering van blokstortingmynboumetodes. Tog vorm die tegniese oordeel van 'n opgeleide en bekwanme werkspan 'n integrale deel van die ontwikkeling van die blokstortingmynbouproses.

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# CHAPTER 1

# INTRODUCTION

**Statement:**

The De Beers mining group has implemented and successfully operated 24 caves in kimberlite pipes in the Kimberley area, at Premier Diamond Mine* and at Jagersfontein. Following technological advances in the 1970's the Kimberley mines moved to sub-level caving, Premier implemented open stoping and Finsch and Koffiefontien did not consider caving as a mining method when they moved from open pitting to underground ore recovery. Three Kimberley mines have since returned to cave mining, using scrapers. Premier planned and implemented a mechanised LHD cave below a massive gabbro sill and experienced numerous problems in terms of undercutting, support, layout, and fragmentation prediction.

Further, detailed, geotechnical investigations were undertaken by the author into all aspects of cave mining to improve the effectiveness of operations. The information and experience gained was used to develop a systematic approach to acquiring the necessary geotechnical data needed to plan a cave mine, as Premier intended to implement a second mechanised cave in weak kimberlite on the eastern side of the mine at greater depth. Moreover, a large, inferred, kimberlite resource exists at Premier to a depth of at least 1200 metres. A pre-feasibility study has shown that it should be practical to exploit this resource using cave mining methods. At least two other mines within the De Beers Group are undertaking geotechnical investigations to implement mechanised cave mines.

Existing planning tools were investigated and calibrated for use at Premier. Where planning tools were inadequate or did not exist, tools were developed to assist with planning and prediction to reduce risk and uncertainty in cave mining.

This chapter defines problems experienced and solutions found at Premier Mine as regards the planning, implementation and control of a mechanised caving operation. The method of addressing caving problems and the contribution that the author has made to defining and solving these problems is set out.

## 1.1. INTRODUCTION

The Premier Pipe is a unique kimberlite orebody in that the pipe is intersected by a 75 metre thick, dipping gabbro sill which cuts through the pipe at a depth of between 380 and 510 metres below surface. In plan the Premier kimberlite is kidney shaped and consists of an eastern and western lobe. The eastern lobe is dominated by the Brown Tuffisitic Kimberlite, whilst in the western lobe, the Grey and Black Tuffisitic Kimberlite Breccia's are present. This is shown in Figure 1.1. Below the sill, in the western lobe, the Grey Tuffisitic Kimberlite Breccia surrounds a core of Black Tuffisitic Kimberlite Breccia. Intruded into these Tuffisitic Kimberlite Breccias and bearing a cross-cutting relationship with them are the Hypabyssal Core Kimberlites.

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*Premier Diamond Mine referred to as Premier or Premier Mine

These consist of the Black Hypabyssal Kimberlite, the Pale Piebald Kimberlite, the Dark Piebald Kimberlite and a late stage Carbonate Dyke Complex. Contacts between all the kimberlite types are usually gradational. The geology of the pipe immediately below the sill is shown in Figure 1.2.

Rock mass classification shows that, geotechnically, the Tuffisitic Kimberlite Breccias are similar as are the Hypabyssal Kimberlites. In this thesis, therefore, kimberlites are divided into only two geological domains. Tuffisitic Kimberlite Breccia's are relatively soft, ultrabasic, poorly jointed rocks that originate from depths in excess of 100 kilometres and are intruded as a relatively cold, gas-rich phase. Ground mass minerals are always extremely fine grained. Deuteric alteration of this ultrabasic rock can lead to the occurrence of widespread montmorillonite clay. On exposure to water this clay can cause extreme squeezing conditions. Hypabyssal Kimberlite is intruded as a melt. Cooling of this melt results in a well jointed, reasonably competent rock. Deuteric alteration is minimal and the rock is not prone to decomposition and squeezing in the presence of water.

Premier Diamond Mine started mining operations in 1903, first by open pit methods and, later, with increasing depth, by underground methods. The underground method originally employed was open benching. Following on the successful implementation of block caving at other mines in the De Beers Group in Kimberley and at Jagersfontein, cave mining was implemented at Premier Mine. The B1, B2, B3 and B4 block caves, together containing in excess of 50 million tons of kimberlite, started operation in the early 1970's and were drawn to completion in the late 1980's. Extraction was by grizzly feeding into scraper drifts. The caves were positioned immediately above the sill and carried no overburden. Drawpoints were sited either in competent gabbro or in very competent metamorphosed kimberlite. Engineering geology problems relating to abutment stresses or brow wear, even though powder factors averaged 250 grams per ton for secondary blasting during the life of the caves, were minimal. LHD extraction replaced grizzly mining in the open bench mining method and has also been used for ore extraction in clean-up levels below the depleted block caves.

Mining below the sill started in 1978, whilst ore reserves above the sill were still being exploited. Three mass mining methods were considered. These were sub-level open stoping, sub-level caving and block caving. The open stope mining method was eventually adopted and implemented on a trial basis. Sub-level caving was discarded as caving of the sill could not be guaranteed. Block caving was discarded because of problems relating to the influx of water once the sill had been breached and concern over the life of drawpoint brows in weak, decomposing kimberlite.

The author arrived at Premier Mine in June 1983 to head the Geotechnical Department. In August 1983 the gabbro sill failed unexpectedly above a large open stope in the L1 mining block, depositing 50 000 tons of barren waste into the stope. Three large stopes, with plan dimensions of 120 metres by 80 metres and 90 metres high, had been developed immediately below the gabbro sill to prove open stoping as a mining method suitable for recovering ore reserves below this sill. During subsequent months, the sub-level open stope mining method failed as a result of premature collapse of gabbro sill crown pillars above the two remaining open stopes.

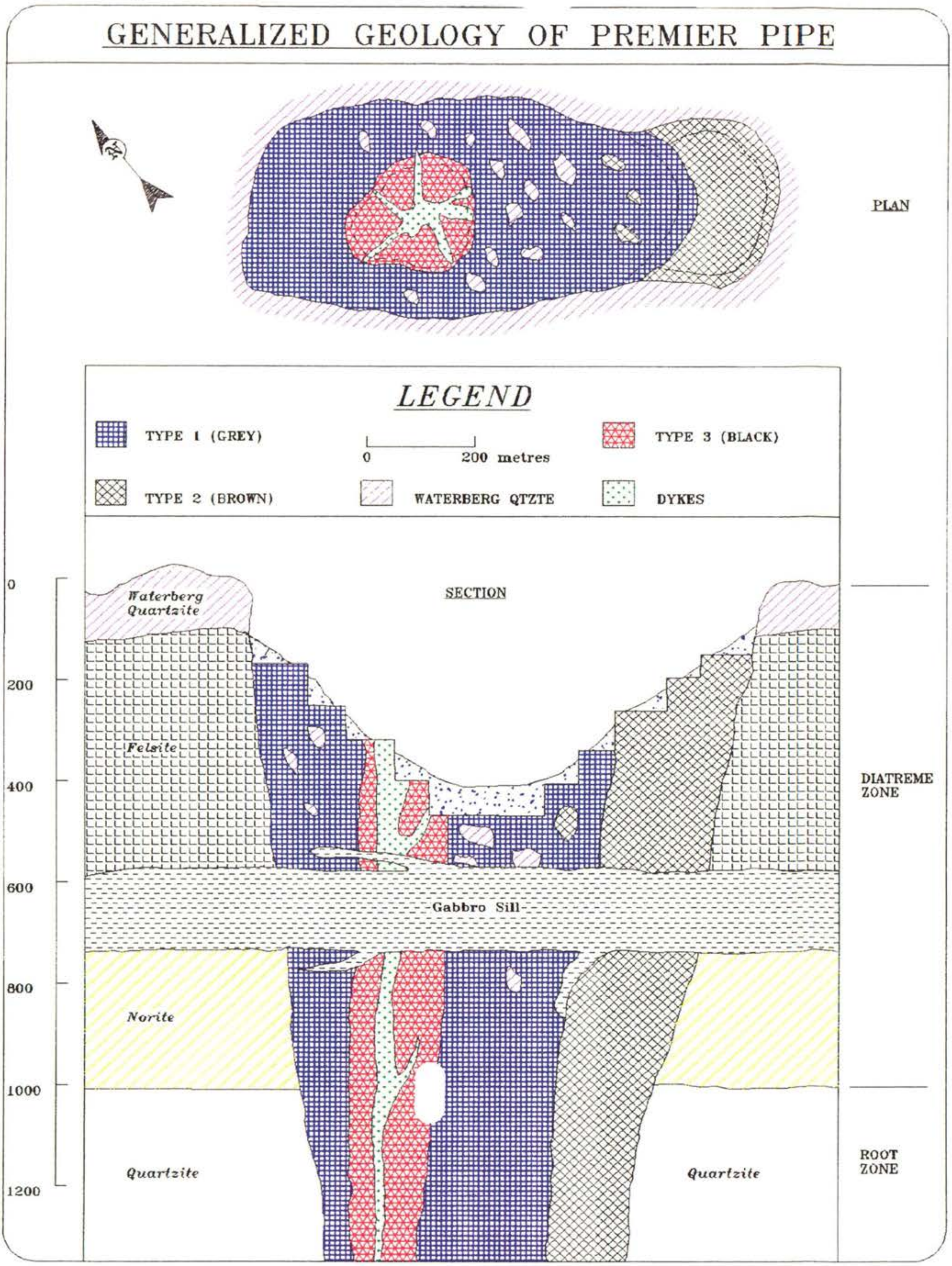


**Figure 1.1. Simplified Plan and Section of Pipe Geology**

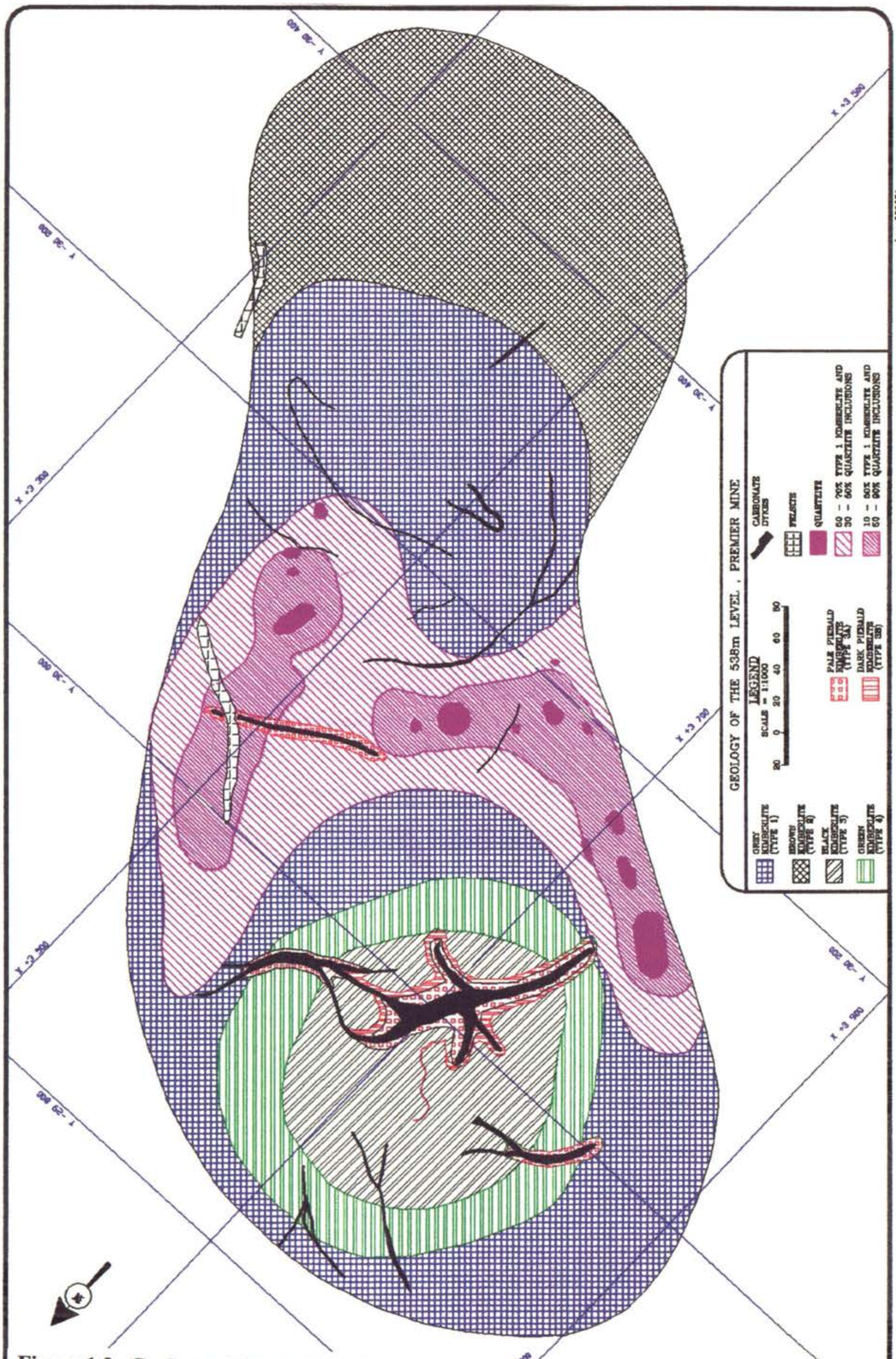


Figure 1.2. Geology of the 538 Sampling Level

Initial failure involved the collapse of several tens of thousands of tons of gabbro into the underlying stope, followed by periodic collapses as stope dimensions increased and caving propagated in the sill. This resulted in unacceptable levels of waste dilution by high density, extremely hard, competent gabbro. Problems of tunnel support in decomposing kimberlite aggravated brow wear and destruction by blasting in weak kimberlite, compounded by difficulty in drilling in highly stressed rock, contributed to the open stope mining method being abandoned.

The author, together with the head of the Planning Department, was charged with re-examining ways of mining the ore-body below the sill. The gabbro sill had been extensively mapped during the early 1970's, prior to implementation of the open stope mining method. Conclusions from this study, based partly on experience gained with dolerite sills in coal mines in South Africa, was that the gabbro sill would not fail at the dimensions of the planned 120 metres by 80 metres open stopes. Stress measurements showed a 4:1 horizontal to vertical stress ratio in the gabbro sill. Numerous rod extensometers and stress meters had been installed in the gabbro sill crown pillars above all 3 open stopes by the Geotechnical Department. After crown pillar failure occurred and caving propagated, regular surveys were conducted using a laser distomat to measure the shape and rate of failure of the sill above the open stopes. As the sill had started to cave above three open stopes, considerable monitoring data relating to the mode and dimensions of these collapses was available. This allowed a back analysis of the crown pillar failure to be carried out for the mine by personnel from the Mining Department at the University of Pretoria. This back analysis, using numerical models not available when mining below the sill was first considered, showed that collapse of the crown pillars was predictable and that caving of the sill above all three stopes would propagate through 75 metres of gabbro into the overlying open pit. Open stoping, using planned stope dimensions, was therefore not a viable option in mining the kimberlite ore reserve below the gabbro sill. Uncertainty regarding the span at which the gabbro sill could be expected to cave and the rate of caving once failure had initiated, made sub-level caving a high risk option as this could have created a large open void immediately below the sill that could have resulted in a catastrophic airblast.

Caving therefore remained the only practical mass mining option that could be used to exploit the extensive kimberlite ore reserve that was known to exist below the gabbro sill. It was decided that a mechanised LHD cave method of mining should be implemented. The planned layout of the cave was based on experience gained above the sill where undercutting was accomplished by ring drilling from tunnels sited between the production tunnels. This formed a continuous trough and drawpoints were developed from adjacent, parallel production tunnels into this continuous trough. The major difference below the sill would be that LHD's rather than grizzlies and scrapers would be used for ore extraction and transport to the tips. The geotechnical data used to plan the mining block below the sill was derived from a few borehole cores and was extremely limited.

A programme of additional core drilling was planned by the author to provide more detailed geotechnical information about the mining block to be caved. This revealed a zone of weak, poorly jointed, decomposing kimberlite around a core of hard, more closely jointed kimberlite. The strength of the extraction level became cause for concern and the entire mining method was reviewed. A cave mining method was once again decided upon, but it was realised that experience from above the sill, where the extraction level was sited in competent gabbro and where the cave carried no overburden, would be of limited value in designing the proposed mechanised cave.

An extensive literature survey was undertaken by the author and the Planning Department. Consultants with experience in caving were employed to assist with the design and implementation of the cave. Many design parameters were taken from the Henderson Mine in Colorado which, at that time, was judged to be one of the most efficient cave mines in the world. Political considerations, however, made it impractical to visit Henderson, Climax or San Manuel mines in the USA.

Once a preliminary design had been completed, the author and two mine officials from the Planning Department, visited the El Teniente, Andina and Salvador cave mining operations in Chile to observe practical aspects of LHD cave mining and better anticipate problems that might arise at Premier in the proposed mechanised cave. Observations and discussions in Chile emphasised that undercutting, drawpoint support, ore pass support, prediction of fragmentation, draw control and early waste ingress would all pose geotechnical problems that would need to be considered in planning, implementing and operating a mechanised cave at Premier.

The BA5 mining block was subsequently planned as a panel retreat, mechanised LHD cave. Drilling tunnels in an area sufficiently large to induce caving were developed together with the underlying drawpoints and post undercutting commenced. The detailed geology and layout of the undercut level of the BA5 mining block is shown in Figure 1.3. The partially mined L1 and L2 open stopes on the eastern side of the mine were converted to mechanised LHD caves. Preliminary planning was undertaken to extract the ore reserves in the BB1E mining block using a mechanised LHD cave. The relative positions of the B1,B2, B3 and B4 grizzly/scraper caves above the sill and the BA5, BB1E, L1 and L2 mechanised LHD caves below the sill are shown in Figure 1.4.

## 1.2. DEFINITION OF THE PROBLEMS THAT HAD TO BE ADDRESSED BY THE AUTHOR

Problems associated with cave mining have been discussed in numerous technical articles and are well documented in an extensive review published in 1984 (Cummings et al., 1984): Major problems defined are:

**The circumstances under which an orebody can be induced to cave. A full description of caveability is needed. The literature reveals no precise definition of caving. A reliable, predictive correlation between caveability and a range of rock types must be developed. This must relate to measureable parameters defining the rock mass.**

At Premier the caveability of the orebody to be mined was largely unknown. Two blocks above the sill had caved easily. A further two blocks failed to cave when a far larger area than originally planned had been undercut. Caving was only achieved after boundary weakening and blasting of the crown pillar had been undertaken.

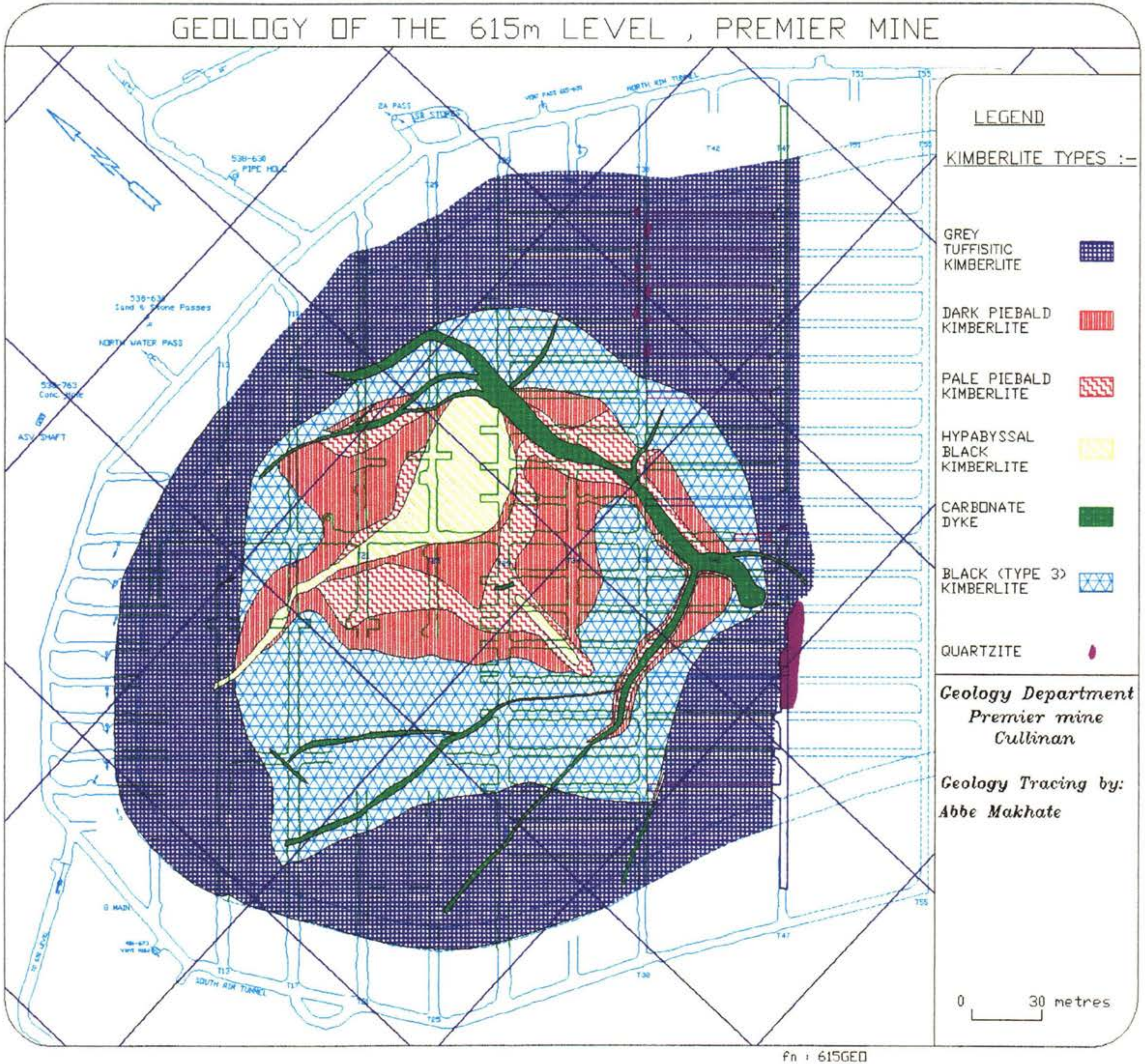


Figure 1.3. Geology of the BA5 Mining Block

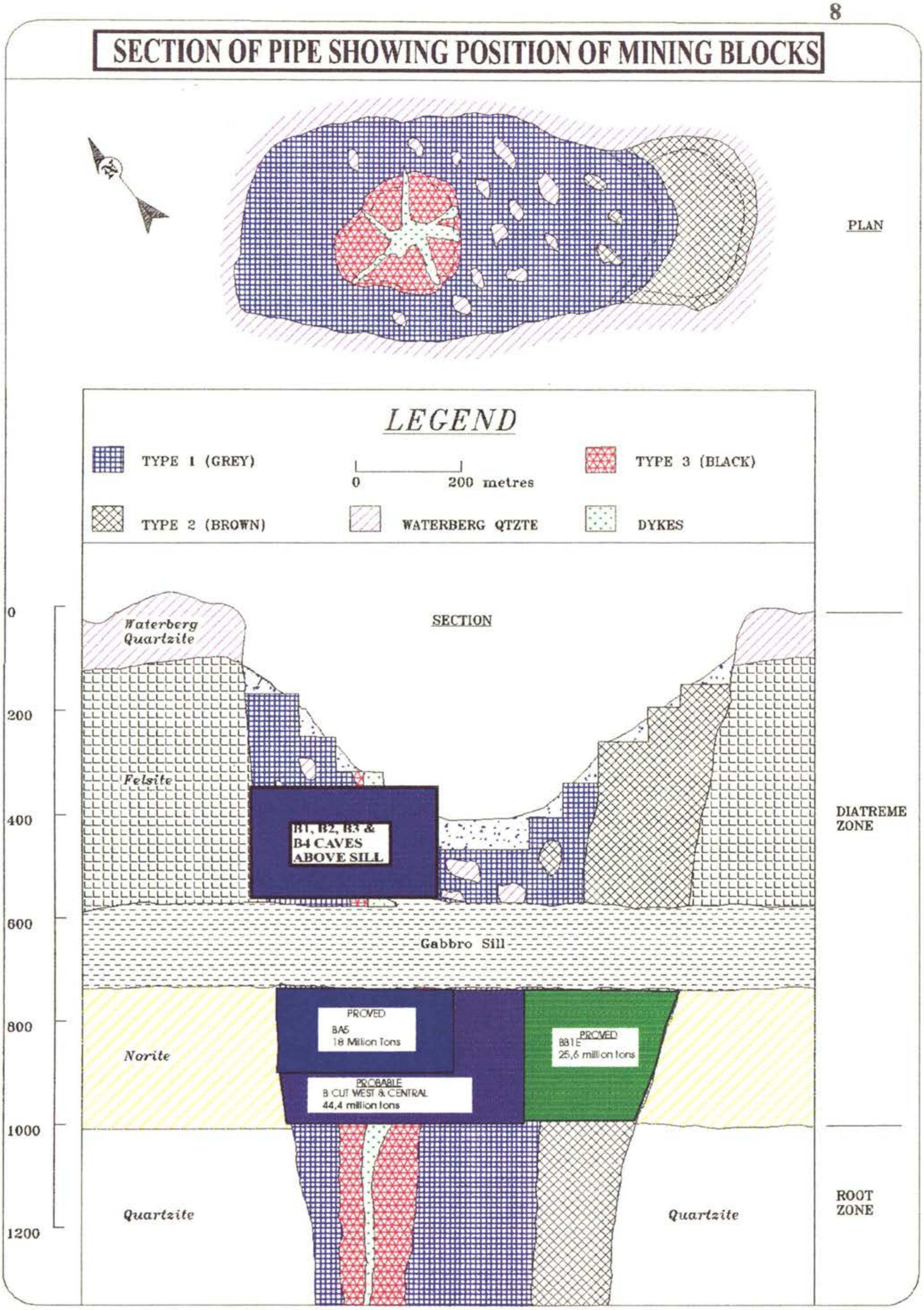


FIGURE 1.4. POSITION OF MINING BLOCKS

Caving then occurred with a minor airblast which fortunately caused no injuries and little damage. Production from these two blocks was delayed by 18 months. All four of these blocks carried no overburden and the effect of stress on caving of the kimberlite at increased depth was an unknown factor.

The original geotechnical assessment of the sill suggested that the gabbro would cave with difficulty. Subsequent experience above three open stopes showed that the sill caved when an unexpectedly small area of 80 metres by 60 metres was undercut. After initial failure, the rate of caving was a function of stope dimension. At the time of planning the BA5 cave which was sited on the western side of the mine, the sill above the open stopes sited on the eastern side of the mine had not yet caved through to the overlying open pit and it was uncertain whether caving would occur at a constant rate.

The dimensions of the undercut that would be needed to induce caving and rate of caving of the kimberlite, as well as the overlying gabbro sill, were unknowns that would need to be anticipated. If the rate of caving of the kimberlite was slower than planned, the production buildup from the BA5 cave would be adversely affected. If the area of undercut needed to be larger than predicted before caving initiated, development on both the undercut drilling level and underlying extraction level would not be available to allow the area of undercut to be expanded rapidly.

Based on experience gained from the three open stopes it was anticipated that caving of the gabbro sill would initiate at a smaller undercut area than the kimberlite. Once caving had propagated through the kimberlite to the base of the sill, caving of the sill would be immediate and widespread. This could result in early waste ingress. It was uncertain whether caving of the sill would propagate right through into the overlying open pit where mining was still in progress, or whether caving would be episodic, resulting in a large airgap which would pose the danger of a catastrophic airblast.

**Aspects of draw control such as the effect of fragmentation in the draw column and optimum column height are still poorly understood. Fragmentation impacts on equipment size, tunnel size, secondary breaking procedures and pass diameters.**

The column height chosen was dictated by available infrastructure and extent of defined ore reserves. Experience in the caves above the sill was that fragmentation, especially in the Tuffisitic Kimberlite Breccia, would be coarse. This resulted in a secondary blasting powder factor as high as 250 grams per ton. Secondary blasting was accomplished using lay-on charges and bombs. In competent gabbro and metamorphosed kimberlite, this level of explosive usage did not cause extensive rock mass damage. It was, however, predicted that secondary blasting with lay-on charges and bombs in weak kimberlite below the sill would result in drawpoint brow destruction. It was, therefore, important to predict the fragmentation size distribution that would report at drawpoints and use this prediction to plan secondary blasting equipment and procedures that would allow the planned production rate of 100 tons per drawpoint per day to be achieved. Large diameter passes sited in the surrounding incompetent norite resulted in extensive block fall out. Attrition and impacts by large fragments were expected to result in progressive damage to concrete lined passes. Production would be adversely affected as passes would need to be constantly repaired and relined.

**Stress levels need to be predicted in advance of cave mining as an input parameter for support design. Abutment loads, undercut unloading and cave stresses must be related to depth, stress field, geologic setting and mining geometry.**

Stress effects are the most problematic aspect of cave mining. The magnitude of stress change resulting from changes in abutment loading is usually the most damaging factor that needs to anticipated and, if possible, reduced. Abutment stress changes, the distance to which they will cause rock mass damage ahead of the abutment zone, their effect on the rock mass in and immediately below the abutment zone and the degree of damage that will be caused to the rock mass and support as the undercut moves overhead, must all be predicted to allow effective support to be planned and installed well ahead of the abutment zone.

Stress effects relative to rock mass strength can be so damaging that the mining sequence may need to be revised. Advance or pre-undercutting rather than post undercutting may have to be considered.

**It was realised that several additional aspects of cave mining could pose geotechnical problems. The height of the undercut was arbitrarily determined by the need produce ore tonnage early in the life of the cave and the height to which drilling could be practically undertaken. The shape of the undercut advance was similarly determined by practical considerations in the absence of clear geotechnical guidelines that could be used to plan the undercut.**

It was known that the rate of undercut advance, leads and lags between adjacent drilling tunnels, the ability to ensure that no remnant pillars (stubs) were left on the undercut level and the overall shape of the undercut advance would be important to the success of the cave. No quantitative way of determining these parameters existed.

**Material flow in the draw column, which is a function of fragmentation, depth of mining and rock type, impacts on drawpoint spacing and must be predicted to plan production output. Effective draw control relies on an accurate information system that can be used to plan and control LHD production on a continuous basis. Draw control is essential to avoid static columns, isolated draw from a single drawpoint and premature waste ingress.**

Coarse fragmentation was predicted in the BA5 and, consequently, drawpoint spacing was planned at 15 metres by 15 metres, the largest currently in use anywhere in the world. This spacing was chosen to keep the extraction ratio to an acceptable level in weak rock. It was not known whether this spacing would ensure overlapping drawzones and complete extraction of the ore reserve as the soft kimberlite, although initially coarsely fragmented, was known to comminute rapidly to produce abundant fines.

No effective draw control program was commercially available and one would need to be developed. The author, together with Dr. D.H. Laubscher and Dr.J.A.C. Diering defined the parameters that would need to be considered for an effective draw control programme. A draw control programme, PC-BC, was subsequently written for Premier Mine and made commercially available at a later stage by Gemcom and is being used on several cave mines. This programme has recently been substantially upgraded after wide ranging discussions involving Dr.J.A.C. Diering, Dr. D.H. Laubscher, the author and other persons engaged in cave mining.

Effective draw control requires an accurate information-gathering system that can be used to plan and control LHD production from numerous drawpoints. No such system was commercially available. The author, together with mine personnel and personnel from an electronics company initiated development of a microwave-based system to achieve an effective draw control system. After several years of development and numerous problems this system is operational at Premier and is now commercially available.

Draw control needs to be planned to achieve production targets and minimise geotechnical problems. At Premier it is important to plan the draw in such a way that early waste ingress is minimised. Inflow of excessive gabbro creates diamond recovery problems in the treatment plant. The high density gabbro reports to sink in the Dense Media Separation* and Heavy Media Separation** sections of the treatment plant and overloads X-ray and grease belt streams in the final recovery section of the diamond treatment plant. This can necessitate that drawpoints be closed long before ore in the drawpoints becomes uneconomic to extract. Failure to limit early waste ingress can result in considerable loss of ore reserves.

**Some 100 million tons of inferred kimberlite ore resource exist at Premier to a depth of 1200 metres. Cave mining could be used to extract this ore reserve. Finsch Mine, also part of the De Beers group, is planning to extract a considerable ore reserve from the Finsch kimberlite pipe using cave mining methods. Koffiefontein, also a De Beers kimberlite mine, has planned extraction of a large ore reserve using cave mining methods.**

The BA5 cave at Premier Mine would therefore provide practical experience and data that could be used to plan further caves both at Premier and other group mines. Methods of defining, monitoring and solving the problems set out above needed to be found at Premier Mine and communicated to other group mines.

Initial production from the BA5 mining block revealed a number of shortcomings in aspects of the planning and implementation of a mechanised cave as practised at Premier. A geotechnical programme that monitors displacements, stress changes, rates of caving in different rock types, draw control, support performance and fragmentation has been implemented on the mine. The effect of all these factors on the mining operation and on support performance is being assessed. Analysis of this data is being used to improve the planning, implementation and operation of new caving operations.

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* Dense Media Separation referred to as DMS

** Heavy Media Separation referred to as HMS

## 1.3. ROLE AND CONTRIBUTION OF THE AUTHOR IN ADDRESSING THE DEFINED PROBLEMS.

**Definition of geological and geotechnical parameters.**

The first task of the author was to carry out a detailed review of the existing geological and geotechnical information that was available on the mine. For the kimberlite orebody, this consisted of geological mapping from existing development above the sill and limited geological information from borehole cores drilled to define the orebody at depth. Uniaxial compressive strength for most rock types encountered was available, but little or no additional geotechnical information had been gathered and collated for other rock mass parameters. Some information was available from earlier geotechnical studies relating to joint spacing and joint condition in the gabbro sill, and data had been gathered from tunnel development and core logging that related to the state of weathering of the norite host rock which impacted on tunnel stability.

Few geotechnical problems were experienced in mining the 4 caves above the sill. Initial mine planning for extraction below the sill at increased depths was premised on the basis that mining operations below the sill would also be geotechnically simple. High abutment loadings at the base of open stope faces together with decomposing kimberlite soon resulted in major support problems in the open stopes and indicated that unforseen geotechnical problems could be anticipated below the sill.

The author planned a core drilling programme to investigate the orebody in the BA5 mining block and logged the core using Laubscher's rock mass classification. Rocks from the various geological domains were characterised to provide Rock Quality Indices that could be compared to rock mass parameters from other cave mining operations.

The author mapped the norite wallrock and gabbro sill to provide data that could be used to define rock mass indices for these geological entities. This data was used to predict fragmentation, caveability and rock mass strength for support design for all the rock types that would be influenced by cave mining in the BA5. As decomposing kimberlite posed a major problem, the author provided samples to the Council for Scientific and Industrial Research* for analysis aimed at determining the pressure and volume changes that were associated with this decomposition (Pellissier & Vogler, 1990). Numerous tests on commercially available products that could be used to seal and prevent water damaging the kimberlite were carried out.

The results of the geotechnical and geological investigations of the kimberlites in the BA5 have been summarised in a paper written and presented by the author at the Commonwealth Mining and Metallurgy Conference held South Africa in 1994 (Bartlett, 1994).

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* Council for Scientific and Industrial Research referred to as CSIR

**Determination of the area that needed to be undercut to induce caving of the kimberlite and overlying gabbro sill, and the rate of caving of these rock types.**

Failure to correctly predict the area that needed to be undercut to induce caving had resulted in costly production delays at Premier. Methods of prediction needed to be improved. Review of literature relating to cave intiation and propagation suggested that caving is largely joint controlled, especially if the rock mass is competent. Regional stresses have the effect either of assisting the caving process, or of creating clamping forces across vertical joint sets which inhibit caving. In weak rock, the caving process may be assisted by failure in compression or tension. At Premier, caving of both the kimberlite and overlying gabbro sill was important to the success of cave mining in the BA5 mining block. Laubscher's correlation of hydraulic radius against adjusted rock mass rating (MRMR) seemed to be the best predictor of caveability. In defining this MRMR, it is assumed that jointing plays a major role in caving and that it is possible to predict the effects of stress on caving the various rock types. It was uncertain whether caving would be joint controlled in the poorly jointed, incompetent Tuffisitic Kimberlite Breccia. The high horizontal to vertical stress ratio in the gabbro sill was expected influence caving of the sill, but no quantitative way of predicting the effect of this stress ratio was available. Moreover, in 1987 when the BA5 was first planned, little empirical data was available from caves sited in competent rock where the MRMR exceeded 50.

The author undertook a back analysis of the caving history of the four block caves above the sill, two of which were sited in Tuffisitic Kimberlite Breccia and the other two partly in Hypabyssal Kimberlite. Rock Quality Indices calculated from detailed mapping of the gabbro sill undertaken by the author, together with data gathered during caving of the sill above three open stopes, and a back analysis using a numerical stress model performed by G.S. Esterhuizen of the Mining Department of the University of Pretoria, were used to review the mode of caving and rate of cave propagation of the gabbro sill. A consulting group was employed by the author to review all the data gathered on the mine to determine whether numerical stress models, using various numerical codes, would be better predictors of the area that would need to be undercut to initiate caving than Laubscher's empirical model (Howell et al., 1993).

Numerous core holes were drilled from excavations within the overlying gabbro sill to determine the rate of caving of the kimberlite and sill, as well as to monitor the size and shape of any airgap that might form. Monitoring and detailed observation of joint movement and rock mass failure in the kimberlite and overlying gabbro sill was undertaken on a regular basis by the author as the cave propagated towards existing excavations located in the kimberlite and gabbro sill.

**Determination of fragmentation size distribution in various rock types in the BA5 and model calibration.**

Experience gained from mining above the sill had shown that fragmentation would be coarse, but no way of accurately predicting fragmentation size distribution was available. Work done by the Julius Kruttschnitt Mineral Research Centre on the mine had shown that detailed scanline mapping could be used to predict the in situ fragmentation size distribution for the kimberlite, and that this could be used to accurately predict the fragmentation size distribution for blasted material using various levels of explosive energy (Villaescusa , 1991). The author attempted to use results from scanline mapping carried out by himself to predict the fragmentation size distribution in the various kimberlite types, but realised that these predictions were inaccurate because the method assumed that all joints would define rock block boundaries. Predicted fragmentation size distribution made on this basis was finer than that experienced in caves above the sill. At this time Dr. D.H. Laubscher was experiencing similar problems in predicting fragmentation size distribution on several cave mines where he was acting as a consultant.

Prediction of the fragmentation size distribution is an essential first step in planning a cave mining operation. It becomes more important as the largest fragment size increases and secondary blasting becomes a more time consuming, expensive and essential part of the production process. It was decided that an expert system should be developed that would seek to accurately predict fragmentation size distribution. Extensive discussions involving the author, Dr. Laubscher and Mr. Esterhuizen followed to determine the way in which such prediction could be accomplished. Extensive reliance was placed on Dr. Laubscher's knowledge and experience in cave mining to understand the caving process and this, together with fundamental rock mechanic principles and probability theory, was used by Mr. Esterhuizen to develop an expert system capable of predicting fragmentation size distribution, based on detailed scanline mapping of an orebody and other measured rock mass parameters.

The author undertook to calibrate the model by using extensive scanline mapping results from the kimberlite types, norite and gabbro at Premier Mine. Cave mining was in progress in the BA5 and drawpoints, sited in both kimberlite types, from which various tonnages had been drawn and where varying degrees of comminution could be seen, provided data. Information relevant to the model, such as the aspect ratio of fragments and the number of joints that could still be observed in a fragment, was collected during detailed observation of the muckpiles in drawpoints. These observations were undertaken on a regular basis over an extended period of time. Three types of hangups were defined. Definition was based on the number of fragments involved in the hangup and the height at which the hangup occurred. The number of fragments involved in the hangup influenced the stability of the hangup and this together with the height of the hangup above the footwall, determined the method of secondary breaking used to bring the hangup down. This in turn determined secondary breaking equipment, personnel requirements and blasting procedures.

The fragmentation model has now been used on several mines to predict the fragmentation size distribution. Data collected by the author at Premier during the study has been used to characterise the type of hangup and to predict frequency for various types of hangup. It has also facilitated the planning of secondary blasting procedures and equipment requirements in proposed caves both at Premier and on other mines (Koffiefontein, Finsch, Palabora, Argyle, Northparkes). Details of the model development, data collected at Premier and how this was been used to calibrate the model has recently been presented at a symposium in South Africa (Esterhuizen et al., 1996)

**Determination of stress levels and the effect of stress on the rock mass adjacent to the cave excavation.**

The magnitude of the stress levels that occur around the cave excavation has a major effect on cave mining and must be predicted well in advance of the actual mining operations. Stress levels affect cave propagation, fragmentation, extraction level layout and stability, as well as support design.

The numerous support problems that were experienced in the open stope mining operations at Premier led to a number of empirical support systems and elements being tested in the open stope below the sill. Support elements that were found to be effective included resin grouted roofbolts, fully grouted cable anchors and mesh and steel-fibre reinforced shotcrete which were used in various combinations. It was realised that well designed support would be an essential part of ensuring effective mining in the BA5 cave. The philosophy that was adopted was that extensive numerical modelling would be carried out to determine the stress levels that would be encountered during successive stages of mining and that the installed support would be sufficiently robust to ensure that support rehabilitation would be minimised. Detailed numerical modelling was undertaken by consultants to the mine to determine rockbolt and cable lengths and strengths. Effective interbolt support in the form of steel fibre- or mesh-reinforced shotcrete was identified as an essential support element. An extensive programme to test the effectiveness of shotcrete as an interbolt support element was planned. Shotcrete panels using several thickness of shotcrete, reinforced with various types of steel fibre and mesh, were sprayed under the supervision of the author and subsequently tested at the University of the Witwatersrand. Details of this work have been published (Kirsten & Bartlett, 1992). The results of the numerical modelling and shotcrete testing were used to design support for the BA5. An extensive monitoring programme was planned by the author to determine actual stress levels and displacements in the cave during the various stages of mining. The behaviour of the support elements and the rock mass was carefully observed during successive stages of cave mining, especially the effect on the rock mass of abutment stresses as the undercut moved overhead. These results were used to assess and improve the cost effectiveness of the support system.

The monitoring programme provided extensive data on the pattern and level of stress changes and rock movement, as well as rock mass and support damage. Several unexpected problems were defined. Data from the monitoring programme was used to calibrate a detailed numerical model. The mining engineer who undertook the numerical modelling shared an office with the author for the space of six weeks and all aspects of the modelling were discussed in detail. Frequent trips were made underground to ensure good correlation between monitoring data, rock mass and shotcrete failure and numerical model results. The results of this modelling have been summarised (Mckinnon, 1992). Model results were used to plan the undercut face shape, as well as to improve aspects of support design. Further monitoring programmes which were designed and controlled by the author were subsequently used to provide data for calibrating additional numerical models that provided the basis for a change in the undercut sequence from post to advance undercutting.

Geotechnical strategy at Premier is to design, implement and maintain an extensive monitoring programme that involves measuring stress changes, displacements and consequent rock mass damage. The mine does not have the skills to develop the complex numerical models used to simulate various aspects of cave mining. Consultants with the required expertise are used to develop numerical models that can be used to assess aspects of the cave mining operation. Monitoring results are used to accurately calibrate the numerical models. The author has sufficient skill and experience in numerical modelling, using the FLAC code, to be able to vary important parameters in the numerical model to test the influence of changes in parameters such as the cohesion and friction angle of the various rock types, as well as parameters that have a bearing on support design.

This methodology has allowed the author to:

* plan an effective shape to advance the undercut in the BA5 and BB1E mining blocks.
* design and plan an effective and logical support code of practice that predicts the extent of rock mass damage on the basis of measured rock mass parameters and of expected stress changes. This allows the use of expensive cable anchors and shotcrete to be reduced but maintains the integrity of excavations. Rigid concrete and shotcrete linings are used only when and where they will not be destroyed by high stresses.
* site service excavations in areas away from abutment stresses.
* change the mining sequence from post undercutting to advance undercutting.

**Determination of parameters that impact on effective undercutting.**

The author devised and implemented a method of quantitatively measuring and recording the level of support and rock mass damage on the extraction and undercut levels. These were correlated with the rate of undercut advance and leads and lags between adjacent undercut drilling tunnels. The author was able to recommend the rate of undercut advance that should be used as well as determine the maximum permissible leads and lags that can be allowed to exist between undercut tunnels. A similar methodology may be applicable on other cave mines but actual results are unique to the Premier orebody.

**Material flow in the draw column and the effect of this on drawpoint spacing, static columns, early waste ingress, isolated draw and effective extraction of the ore reserve.**

Material flow in the draw column in cave mines is probably the least understood aspect of cave mining. Drawpoint spacing and relative rate of extraction from drawpoints which relate directly to poorly understood material flow, were empirically determined for the BA5.

The presence of the well defined gabbro sill which overlies the cave and, on caving, produces easily recognisable waste fragments which have to be monitored because they adversely impact on diamond recovery, together with accurate draw control information, has allowed Premier Mine to produce a unique draw control data set. Analysis of this data by the author has shown that the widely spaced drawpoints did not result in the creation of static columns and did not adversely affect extraction of the ore reserve. The draw control pattern used did not result in conditions of isolated draw or allow early waste ingress. Data show that ore, especially fines, migrates towards areas of high draw, allowing extraction in excess of 100 percent to be achieved in these areas of high draw. Surrounding areas, where the rate of draw is lower, are selectively depleted, producing waste at lower extraction percentages. Observation suggests that drawbells rather than drawpoints define overlying drawzones, provided that drawbells can be accessed from both sides and that both drawpoints are active. Observation and data similarly suggests that there is greater interaction across minor apices than across major apices. Waste will spread rapidly along a line of adjacent drawbells separated by minor apices but, in several areas, waste did not move easily across major apices. On the basis of this information, the author recommended that drawpoint spacing in the BB1E mining block be increased to 15 metres by

18 metres with a considerable saving on development and support and an increase in overall extraction level stability. In future, draw control will be based on drawbell reserves and not drawpoints and every endeavour will be made to keep lines of drawbells, separated by minor apices, active to ensure drawzone interaction. No loss of ore reserves is predicted. The effect of this increased drawpoint spacing on the extraction of the ore reserve in the BB1E will be carefully monitored and results used to plan drawpoint spacing in other caves in kimberlite ore bodies operated by De Beers Consolidated Mines.

**Detailed understanding of the principles involved in all aspects of cave mining will have to be developed to allow cave mining in coarsely fragmented ore using LHD extraction to be planned at greater depth.**

Premier has recently completed a pre-feasibility study into cave mining of the kimberlite ore reserve to a depth of 1100 metres. It is anticipated that undercutting and support of drawpoints that will have to produce 250 000 tons of ore over the space of several years in highly-stressed, weak rock will pose major challenges.

## 1.4. LOGIC AND LAYOUT OF THIS THESIS

This thesis is laid out so that it can be used to plan, develop and operate a cave mine predicting coarse fragmentation and using LHD's for extraction.

Geotechnical factors that affect most cave mining operations predicting coarse fragmentation and using LHD's for extraction are considered by reviewing the literature. Work carried out by the author, especially in the BA5, together with observations made by him relating to other cave mining operations recently visited, or that he has assisted in planning, are considered in detail to improve and elucidate geotechnical aspects of cave mining operations in caves experiencing coarse fragmentation and using LHD's for extraction.

Extensive literature exists relating to cave mining operations at relatively shallow depth in finely fragmented ore that allows extraction using slushers, grizzlies and scrapers. The effect of undercut stresses in such orebodies is similar and relevant to caves experiencing coarse fragmentation and using LHD's for extraction. Much geotechnical data gained from such caves is, however, not directly applicable to planning many other aspects of a cave using LHD's for extraction. Support requirements, secondary blasting techniques, drawpoint spacing, material flow and cave layout are substantially different. For this reason, much of the literature relating to cave mining in finely fragmented ores has only been reviewed where this has appeared relevant. The author accepts that this could result in some omissions

The author uses the thesis to correlate aspects common to mines predicting coarse fragmentation and planning LHD extraction into a comprehensive model that can be used to plan, develop and operate a cave mine. It was judged by the author that the thesis should concentrate on presenting a readable text that would be useful to mining practitioners interested in the general aspects of planning such a cave. Detailed data interpretation and discussion of aspects considered unique to Premier have therefore been set out in appendices and are only referred to in the main body of the thesis.

Fundamental implications of the thesis are that:

* A detailed geotechnical assessment of the orebody must be undertaken. The information that needs to be collected must be defined and methods of collection must be pre-planned.. The geology of the orebody and distribution of ore values must be known in detail. Rock mass parameters such as rock strength and distribution of joint patterns and their attributes must be obtained. Details of groundwater flow are important. As much information as possible regarding in situ stresses should be gathered.

* The geotechnical assessment will provide guidelines as to what mining methods should be considered practical in the exploitation of the orebody. Other aspects that will follow from the geotechnical investigation are set out in Table 2.1. The thesis only considers implementation of a mechanised cave mining operation consequent on the geotechnical assessment.

* Caving theory allows anticipation of the behaviour of the rockmass as a result of mining operations. The pattern of stress changes that occur as a result of the mining process are well documented, and monitoring of the rock mass during cave mining has been carried out on many mines. Computer simulation of the caving process has been carried out using elastic models. This information provides qualitative information pertinent to cave mining but the uniqueness of orebodies means that further investigations need to be carried out to establish the behaviour of the rock mass within a specific deposit. A detailed model of the caving process must be developed to define the information that needs to be collected. The author has commissioned extensive computer simulation to anticipate the behaviour of the kimberlite rock mass as mining progresses. He has also designed a monitoring programme and implemented continuous monitoring of the rock mass to calibrate and validate the results of the computer simulations. Information is used to evaluate and improve the mining sequence, and the effectiveness of the mining layout and installed support.

* The layout of a mechanised cave mine must consider the structural strength of the rock mass in which mining is to be done, the ease of developing the layout and ease of production from the final layout. The design, layout and operation of orepasses is an important aspect. The lift height of the cave has a major influence on the profitability of any cave operation and needs detailed consideration. The height of the undercut affects fragmentation, drilling and blasting costs and rate of advance of the undercut.

* The production and undercut levels are considered in detail in terms of the stresses and rock mass responses that develop on and around these levels as mining progresses. The results of detailed monitoring are interpreted in terms of the mining operation. Ways of undercutting are evaluated in terms of geotechnical and financial considerations. Effective support system designs are evaluated.

* Fragmentation of the ore is one of the parameters that needs to be established because it is fundamental to any cave mining operation. Recent development of fragmentation models allow initial block size to be anticipated from detailed structural mapping of the orebody. Simulation of the breakdown of initial fragmentation can then be used to

anticipate the size of fragments that will arrive at the drawpoints. The effect of the caving process and interaction between the rock in the cave back and the broken ore in the ore column needs to be understood and the information used to optimise fragmentation. This information can then be used to determine drawpoint spacing and size, type of mining equipment and secondary blasting procedures.

* Material flow as monitored and interpreted at Premier, is set out.

* Data interpretation in terms of caving, fragmentation, layouts, rock mass response to mining, induced stresses and support design, are used to form a comprehensive model that can be used to plan a mechanised caving operation where coarse fragmentation is expected.

TECHNICAL PAPERS AUTHORED AND CO-AUTHORED BY P.J. BARTLETT RELEVANT TO THIS THESIS:

1. Harverson, R., and Bartlett P.J. The Move to Block Caving Below the Sill at Premier Mine. Anglo American Corporation Group Mining Symposium pp. 29-34. September, 1987. The paper summarises the initial planning and implementation of mechanised cave mining in the BA5.

2. Stacey, T.R., and Bartlett, P.J. Probabilistic Evaluation of Ore Pass Stability and Support. Static and Dynamic Considerations in Rock Engineering. pp. 309-315 1990. Editor Brummer. Balkema, Rotterdam. ISBN 90 6191 1532. The paper summarises problems experienced with ore passes at Premier and a proposed support design process for ore passes.

3. Kirsten, H.A.D., and Bartlett P.J. Rigorously Determined Support Characteristics and Support-design Method for Tunnels Subject to Squeezing Conditions. S.A.I.M.M., Vol. 92 No.7., 1992. The paper summarises work done at Premier on reinforced shotcrete support and numerical modelling to determine the length and strength of steel tendon support and the way in which the was work was used to design support for the BA5.

4. Bartlett, P.J. Geology of the Premier Diamond Pipe. XVth CMMI Congress. Johannesburg, S.A.I.M.M., 1994, vol.3, pp 201-213. Editor H.W. Glen. The paper summarises the geology of the Premier diamond pipe and includes the mapping that was undertaken by the author as part of the detailed geotechnical investigation of the BA5 mining block.

5. Bartlett, P.J. Support in a Mechanised Cave at Premier Mine. MASSMIN 92. Johannesburg, S.A.I.M.M., 1992. Editor H.W. Glen. The paper considers the detailed support design process used in the the BA5 and preliminary results of the support monitoring programme installed by the author to assess support performance.

6. Bartlett, P.J. The Design and Operation of a Mechanised Cave at Premier Diamond Mine. MASSMIN 92. Johannesburg. S.A.I.M.M., 1992. Editor H.W. Glen. The paper details many of the practical problems encountered when the BA5 was implemented

below the sill using a post undercut, and preliminary attempts to overcome these problems.

7. Rood, H.R., and Bartlett, P.J. Mechanised Caving at Premier Mine. XVth CMMI Congress. Johannesburg, S.A.I.M.M., 1994, vol. I, pp 219-225. The paper details many of the geotechnical investigations that were undertaken and the way in which these were used to improve mining conditions in the BA5.

8. Esterhuizen, G.S., Laubscher, D.H., Bartlett, P.J., and Kear, R.M. An Expert System Approach to Predict Fragmentation in Block Caving. Massive Mining Methods. S.A.I.M.M Colloquium., Mintek., Randburg., South Africa, July 1996. The paper details the development of an expert system to predict fragmentation size distribution and the ways in which the author collected the information to calibrate the models developed, and used data collected in the BA5 to plan secondary blasting procedures.

9. Bartlett, P.J., Heap, P.A.J., and Matthews, V.J.E. Ore Pass Support Below the Sill at Premier Mine. Papers and discussions, 1992 - 1993. Association of Mine Managers of South Africa. Published by the Chamber of Mines of South Africa. 1994. The paper deals with methods of ore pass support, and management used in passes situated in norite at Premier after numerous problems were experienced in using steel tendon support alone to ensure the stability of passes.

10. Heap, P.A.J., and Bartlett, P.J. Mechanised Panel Retreat Caving at Premier Mine. Papers and discussions, 1993 - 1994. Association of Mine Managers of South Africa. Published by the Chamber of Mines of South Africa, 1995. This paper details the numerous problems that were encountered when the BA5 cave at Premier was originally commissioned. The ways in which planning and implementation had to be revised, changed and augmented, are detailed.

# CHAPTER 2

## GEOTECHNICAL ASSESSMENT

**Statement:**

The numerous parameters that need to be collected to plan a cave mining operation are detailed in the literature. This chapter aims at defining the subset of parameters that are needed to plan a mechanised cave in a kimberlite orebody with coarse fragmentation.

The author had to determine what parameters needed to be defined and an effective, systematic way to gather the data. The database then had to be assessed, correlated with data from other cave mines and used to plan the mechanised cave. It was decided to use Laubscher's rock mass classification for the purpose for reasons detailed in this chapter. Rock mass classification was augmented by analytical and observational design procedures where necessary.

The author planned and implemented a drilling programme and logged the core, as the first step in gathering the required geotechnical data from the BA5 rock types. The BA5 mechanised cave was planned and implemented on the basis of this and other data derived from earlier drilling programmes. The initial planning of the BA5 has been reported in the literature (Harverson & Bartlett, 1987).

Tunnel development on both the 615 undercut drilling level and the 630 extraction level was mapped by the author as exposures became available to provide additional geotechnical details. Several tens of metres of intensive scanline mapping, using the method developed by Villaescusa, were undertaken to provide the data necessary for the estimation of the fragmentation size distribution from the rock types in the BA5. The results of geological mapping have been reported in the literature (Bartlett, 1994. Stacey & Bartlett, 1990).

Geotechnical parameters for the gabbro sill, above the BA5 mining block, the norite surrounding the BA5 orebody and several types of kimberlite within the orebody were derived from the work done. Data collected by the author and test work on Tuffisitic Kimberlite Breccia rock samples submitted to the CSIR by the author indicated that this rock type would pose some unique problems because it was poorly jointed, decomposed when wet to exert high pressure and behaved in an inelastic manner..

The rock mass parameters were used in detailed numerical modelling commissioned by the author for Premier Mine. On the basis of geotechnical assessment and results of the numerical modelling, the author drew up the Premier Mine Support Code of Practice. This is a legally required document that specifies the type and installation procedures for all types of support at Premier Mine. The Support Code of Practice is appended to the thesis.

## 2.1 INTRODUCTION

Mine personnel need to be able to anticipate the behaviour of the rock mass in which they intend to mine for mine design purposes. Often local experience is sufficient to allow planning of new mining blocks. Where rock conditions change, or new mines are planned, engineers must base the mine design on a geotechnical assessment of the ore body that they will be mining. This involves detailed geotechnical mapping of the deposit aimed at characterising the important parameters that will affect the rock mass during mining.

Prediction of the undercut area required to induce caving, the rate of caving, support design, fragmentation, as well as material flow and comminution in the draw column, require a detailed characterisation of the rock mass, the stress and strain levels that lead to instability and the effect that this instability will have on the mining process. In the case of fragmentation and material flow in the draw column, the consequence of the instability must be predicted.

Geotechnical assessment of the mining environment is an essential first stage in the planning of any mining operation. The geology of the deposit defines the ore distribution within the rock mass and has the primary influence on the mining method. If the ore distribution defines a massive, low grade deposit at a depth that makes open pitting uneconomic, caving methods should be considered as these have proved to be the lowest cost underground mining methods. Good examples are the iron ore deposits of the Lake Superior area, the molybdenum porphyry at Climax, Urad and Henderson and the copper porphyry at Magma Copper in the USA, the Codelco mines at El Teniente, Andina and Salvador in Chile, chrysotile asbestos mines in Canada and Zimbabwe and the kimberlite pipes in the De Beers mines in South Africa.

Rock types define broad rock mass characteristics from which rock mass parameters are determined in more detail. Mines often cover an extensive area and several geological domains may be encountered within the area both laterally and vertically. Service and mining excavations must be developed in different rock types.

Bieniawski (Bieniawski, 1989) defines three design methods for assessing the stability of rock masses and tunnels:

* Empirical design methods assess the stability of rock masses and tunnels using statistical analysis of underground observations. Rock mass classification is the best known empirical approach to facilitate design.

* Analytical design methods rely on analysis of displacement and stress around mine openings. These methods include various computer modelling techniques, closed form solutions, analog simulations and physical modelling.

* Observational design methods monitor actual rock instability, ground movements and ground-support interaction. This approach often forms an integral part of both the other two methods.

## 2.2. GEOLOGICAL ASSESSMENT

The following needs to be defined in any geological investigation:

* Orebody shape, dimension, dip and extent at depth.
* Ratio of the surface area of the ore/waste interface to the contained ore as this will impact on the waste drawn into the system.
* The mineral and value distribution in the ore and dilution zone.

## 2.3. GEOTECHNICAL ASSESSMENT

The most important aspects that need to be defined in designing a cave mining operation are:

**The area that must be undercut to induce caving.** Parameters that need to be investigated to define this include:

* Rock mass strength
* Regional stress
* Caveability of the orebody and overlying capping. If the orebody is not vertical, caveability of the hangingwall must also be considered.
* Joint structure
* Groundwater

**Fragmentation.** Parameters that influence fragmentation are:

* Jointing which includes joint attributes
* Rock mass strength
* Regional stress
* Mining induced stress
* Mining sequence
* Rate of undercut advance
* Height of draw column
* Draw control strategy

**Rock mass response to mining and support requirements** are affected by:

* Rock mass strength
* Regional stress
* Joint structure
* Groundwater
* Location and strength of extraction horizon
* Mining sequence
* Geometry of undercut
* Mining induced stress
* Major structures including contacts, faults, dykes and shear zones.

**Draw control strategy** is dependant on:

* Method of draw (LHD, grizzly, scraper, slusher)
* Caving method (block, panel or mass)
* Competence of rock mass on the extraction horizon

* Drawpoint spacing
* Production tempo
* Fragmentation
* Secondary breaking requirements

All the above parameters need to be integrated into a comprehensive mining strategy. Rock mass classification methods have evolved which aim to gather much of the above information in a systematic manner. The information is then integrated and empirically applied to new mining systems based on experience gained in other, similar, mining environments. The information gathered can be used as the basis for design or to estimate parameters needed in analytical design and numerical models.

Essential data that needed to be collected, therefore, included information that could be gained directly from measurements in the orebody, or rock taken from the orebody and analysed in the laboratory. These included Poisson's ratio, uniaxial compressive strength, Young's modulus, joint attributes, groundwater, major geological structures and regional field stress. Some parameters can be directly derived from this data. These include caveability, rock mass strength and fragmentation. Most other parameters must be deduced on the basis of experience and engineering judgement of personnel involved in cave mining. The author undertook a detailed review of suitable rock mass classifications, analytical and observational design procedures. Results were used to plan the effective collection, categorization and analysis of geotechnical data used to plan the BA5 mechanised cave.

The following three sections consider aspects of this review as they relate to the use of rock mass classifications, analytical design procedures and observational design procedures.

## 2.4. ROCK MASS CLASSIFICATION

In the initial stages of mine planning only limited data is usually available. Typically this information is gathered from the analysis of borehole cores. Rock mass classification systems have been developed that allow this limited information to be used during the feasibility study and initial mine design for a new deposit. The planning engineer must know what information is needed as input parameters to derive an acceptable mine plan and how this information can be gathered. If insufficient information is available additional data must be gathered in the most cost effective way, or the additional risk involved as a result of inadequate information must be quantified. Four useful rock mass classification systems and ways of deriving necessary rock mass parameters for design purposes from rock mass classification correlations are briefly considered.

Rock mass classification is not a substitute for engineering design but, if used with engineering judgement, can be a powerful design aid. Bieniawski (Bieniawski, 1989) defines the objectives of rock mass classification systems as follows:

* Identify the important parameters influencing rock mass behaviour
* Classify rock mass formations into groups with potentially similar behaviour
* Provide a basis for understanding the characteristics of each rock mass class

* Allow experience of rock mass behaviour to be transferred between sites
* Compile quantitative data and guidelines for engineering design
* Provide a numerical basis for communication between engineers and geologists

The following rock mass parameters will influence the behaviour of the rock mass during mining related activities such as tunnel support and the creation of excavations for mining and other purposes:

* rock strength
* joint structures
* stress distribution
* groundwater

Joint spacing, joint surface expression and joint alteration together with the influence of water which usually acts along joints, all play an important role in the rock mass response to mining. The degree to which these parameters affect the rock mass response has been quantified with increasing exactitude as more people use rock classifications in mining and tunnelling operations and the data base grows. The influence of mining related activities such as drilling and blasting, stress changes and the proximity of excavations to one another, has led to the concept of adjustments being made to the original ratings to allow the influence of these factors to be taken into account. Rock mass classification is a way of quantifying mining experience and putting numbers to this experience. The ability to communicate the experience to planning engineers is thus made easier.

Rock mass classification has two aspects. The first of these is aimed at adequately describing the rock mass parameters. The influence of a specific parameter is turned into a number and the combined influence of all the parameters is combined into a single number or rating class which serves as a Rock Quality Index. The influence of mining related activities on the rock mass can be used to adjust the ratings. This first aspect of rock mass classification is becoming increasingly accurate as the data base grows and ratings are turned into numbers. Rock mass ratings can be correlated with rock mass parameters such as friction angle, fragmentation and rock mass cohesion. These parameters are widely used in computer modelling and other methods of engineering design.

The second aspect of classification is to infer rock mass behaviour on the basis of the rating. This step is empirical and is based on the experience of the people who have developed and used the classification systems. As the data base grows the empiricism becomes less experience based and more statistical. Powerful computer based programmes are becoming available that give rock mechanics practitioners the ability to design underground support systems and apply factors of safety to designs in the same way as has been done for rock slopes in open pits and pillars in underground coal mines.

A rocks mass is a complex structure and every joint within the structure has several attributes. Any attempt to describe the complete rock structure and define all the rock properties will be extremely difficult in terms of data collection. Accurate numerical simulation of jointing has not yet been achieved. Because it is impractical to provide sufficient data to allow accurate computer analysis, numerical models are used to test the sensitivity of the design to a range of input parameters. The rock mass in which an excavation is to be placed has parameters within the range simulated. The numerical model is used as a design tool rather than a design approach (Nicholson, 1988). The semi-empirical Hoek and Brown failure criterion has been incorporated into several numerical models and, if rock mass ratings could be added in the same way, this would represent a considerable advance in excavation design.

Rock mass properties peculiar to a particular deposit or class of rocks can make the direct application of rock mass classification difficult. The effect of free water on Tuffisitic Kimberlite Breccia can be such that it transforms a competent, poorly jointed rock into a medium that behaves plastically rather than elastically. In the space of a few hours the water enters into the montmorillonite clay structure and breaks down the cohesion of the rock. This results in indeterminate swelling of the rock into an excavation. The swelling effect is accompanied by considerable pressure that can destroy support (Kirsten & Speers, 1991). Such effects must be incorporated into the classification system when the system is used to design support and to estimate rock response where swelling rock conditions are expected. Barton's Q system is the only classification system that specifically considers squeezing ground conditions and he states that squeezing ground is " inadequately represented in the original data base" (Barton, 1988).

Geological domains where peculiar rock properties have an influence must be defined and rock behaviour analysed in detail. Excavation and support design are then made specifically for that rock type. Several potash and salt mines where rock properties are unique are obliged to use site specific excavation and support system designs. This does not allow solutions and experience to be transferred between mines.

Classification systems were originally developed for use in tunnelling and other civil engineering work. It was easy to extend these systems into mines for support estimation in tunnels and Barton's NGI rock mass classification system has gained wide acceptance as a design tool for support estimation. Laubscher's classification system which expanded on Bieniawski's Rock Mechanics Rating (RMR) classification has been aimed more specifically at mining operations and at cave mining in particular. Barton's NGI system and Laubscher's system are widely used to address aspects of mine design other than support. More recently a system designed specifically for caving rock mass classification and support estimation in cave mines has been suggested (Cummings et al., 1984). The latter is based on Bieniawski's RMR classification and uses Laubscher's logic in applying a series of adjustments to the original ratings in terms of the siting of excavations, method of excavation and the operation of the cave. This system is not yet in widespread usage.

Parameters that are important to the definition of rock mass behaviour and that can be used for aspects of mine design other than support are reviewed below. The two most widely used classification systems are those of Barton and Bieniawski. Bieniawski's classification forms the basis of Laubscher's classification system and the Modified Basic Rock Mechanics Rating of Cummings. These latter two classification systems are aimed specifically at cave mining and are considered in some detail. Further details can be found in the literature.

### 2.4.1. BARTON'S NGI ROCK MASS CLASSIFICATION SYSTEM

Barton's classification aims at assessing three basic parameters:

* rock block size ($RQD/J_n$)
* joint shear strength ($J_r/J_a$)
* confining stress ($J_w/SRF$)

The complete equation is:

$$Q = (RQD/J_n)*(J_r/J_a)*(J_w/SRF) \qquad \text{(eqn 2.1.)}$$

RQD = Deere's Rock Quality Designation
$J_n$ = Joint set number
$J_r$ = Joint roughness number
$J_a$ = Joint alteration number
$J_w$ = Joint water reduction factor
SRF = Stress reduction factor

Deere's Rock Quality Designation is an initial measure of fracture spacing. The joint number relates to the effect of joint sets on the geometry which in turn affects the freedom of movement of any block that daylights in an excavation. These two aspects taken together define the block size and shape and should correlate with fragmentation in blasting and caving. The joint roughness number and the joint alteration number relate to the shear strength of the joint which can in turn be related to interblock shear strength and define an approximate angle of friction for the rock mass. The stress reduction factor is a measure of active stress and focusses on three aspects of the rock mass that affect mining. The first is the presence of shear zones or other zones of weakness, the second relates to the level of stress anticipated in and around the excavations. Thirdly, squeezing and swelling rock conditions are considered. Although guidelines are laid down experience and engineering judgement play an important role in determining SRF.

### 2.4.2. BIENIAWSKI'S ROCK MECHANICS RATING CLASSIFICATION SYSTEM

Bieniawski's geomechanics classification was based on data collected mainly from sedimentary rocks in civil engineering applications in South Africa. The classification is based on five rock mass parameters:

* Strength of intact rock material. The uniaxial compressive strength is used. (For very weak rock the point load index is used).

* Deere's RQD

* Joint spacing (The term joint is used to describe all discontinuities.)

* Condition of joints rates joint separation, continuity, surface roughness, wall condition and gouge.

- Groundwater conditions play an important role in terms of observed flow rate into the excavation and the ratio of joint water pressure to major principal stress.

### 2.4.3. THE MODIFIED BASIC ROCK MECHANICS (MBR) SYSTEM

A rock mass classification system aimed specifically at support design in cave mines has been developed (Cummings et al., 1984). This classification uses Bieniawski's rock mass rating (RMR) to derive a " modified basic RMR" (MBR) as the starting point and applies two series of adjustments to the original rating. The logic is the same as that applied by Laubscher, but the adjustments consider additional aspects. The first set of adjustments considers the initial mining environment in terms of blast damage, induced stresses and fracture orientation for isolated tunnels to derive an "adjusted MBR". This is further modified in terms of major structures in the deposit, distance from an excavation to the cave line, and block or panel size to derive the "final MBR" which is used for support design in areas affected by cave mining operations.

The MBR classification has the following rating system:

- Strength of Intact Rock (IRS) 0 - 15. The rating is determined from a diagram that allows for a range of rock strengths rather than a single value.
- Discontinuity Density (RQD and spacing) 0 - 40. The RQD and discontinuity spacing are used to derive ratings from a diagram.
- Discontinuity Condition 0 - 30. A description of the joint condition is used to determine the rating from a table.
- Groundwater Condition 0 - 15. A description of the groundwater condition is turned into a rating using a table.

The rating values used in this system are the same as that used in Bieniawski's original rock mass rating system.

The modified basic RMR (or MBR) can have a value of between 0 and 100. Three adjustments are applied to the MBR to derive the adjusted MBR which is used for support design in excavations in service areas, or for drifts during the development stage of mining. Blast adjustments ($A^b$) are between 0,8 and 1 depending of the degree of damage done to the rock and are read from a table. Induced stresses ($A^s$) are calculated in terms of effective extraction ratio, depth and stress state. The adjustment is between 0,8 and 1,2. The fracture orientation ($A^o$) is computed next. The logic is that fractures perpendicular to an opening are more favourable than fractures parallel to the opening, that support design and excavation development are facilitated by fractures that dip away from rather than towards an opening and that a steep dip is preferable to a shallow dip. The rating is read from tables.

The three adjustments ($A^b$, $A^s$, $A^o$) are multiplied together to yield a value of between 0,45 and 1,2. The MBR is then modified by this value or 0,5 (whichever is the greater) to yield the adjusted MBR. The final MBR is then calculated. This step takes into consideration the effect of abutment loadings. Major structures (S, adjustment 0,7 - 1,1) are considered as are distance to the cave line (DC, adjustment 0,8 - 1,2) ) and the block or panel size (PS, adjustment 1,0 - 1,3). The three adjustments are multiplied together to yield a product that has a value of between 0,56 and 1,7. This value is then multiplied by the adjusted MBR to yield the final MBR which is used for final support design using a design chart. The support recommended by the system is discussed in more detail in Chapter 5.

### 2.4.4. LAUBSCHER'S CLASSIFICATION SYSTEM

Laubscher modified Bieniawski's RMR classification system in 1974 to suit mining operations in the chrysotile asbestos mines in Zimbabwe. The basic classification parameters are the same. The five classes of the RMR classification are divided into A and B subclasses. The significant difference was to determine an in situ classification of the rock mass (RMR). This rating is then adjusted to anticipate the effect of the proposed mining system on the rock mass (MRMR).

Different ranges for intact rock strength (IRS) are used. If rock of various strength occurs within the area under consideration an average value is assigned to the rock mass. The weak rock is judged to have a greater influence on the behaviour of the rock mass during mining than the strong rock. The weighting is therefore not linear and is read off an empirical chart. A rating range from 0 to 20 is necessary to allow the influence of the rock strength to be adequately represented in the calculation. A upper strength limit is taken as 185 MPa as for stronger rocks, factors other than rock strength determine rock mass behaviour.

Deere's rock quality designation (RQD) is used to rate the rock mass and a maximum rating of 15 can be given to rock with an RQD of 100 percent (%). This is combined with the joint spacing (JS) which has a maximum rating of 25. The rating is read off an empirical design chart that takes account of the number of joint sets and joint and/or fracture spacing. RQD must be determined by logging intact drill core.

The fracture frequency per metre (FF/m) can be used in the place of RQD and joint spacing to derive a rating with a maximum value of 40. Fractures are defined as both fractures and joints. Fracture frequency must be accurately determined by unbiased structural mapping of representative areas and/or core logging, and the data reduced to give an accurate picture of the rock mass structure in terms of the joint plane inter-relationships and continuity of structural features. The rating can then be determined from tables or charts.

The final rating parameter is joint condition (JC) which has a maximum value of 40. Joint condition is a measure of the frictional property of the joints (not fractures) and is a function of expression, surface properties, alteration zones, filling and water. Important aspects to consider are large-scale joint expression, small-scale joint expression, extent of joint wall alteration and joint filling.

When the individual parameters have been rated, the values are combined into a single number lying between 0 and 100 and the rock mass is placed in an A or B sub-class of five different class ratings. The rating class is much the same as that of Bieniawski's and is also called the rock mass rating (RMR). The rating is used in basic design procedures for mining excavations.

**Table 2.1. Correlations based on Laubscher's Mining Rock Mass Rating**

**MINING METHOD RELATED TO MRMR IN CAVE MINING OPERATIONS**

| Class rating | 5 (0-20) | 4 (21-40) | 3 (41-60) | 2 (61-80) | 1 (80 - 100) |
|---|---|---|---|---|---|
| **Block Caving** | | | | | |
| Undercut SI (m) | 1-8 | 8-18 | 18-32 | 32-50 | >50 |
| Caveability | very good | good | fair | poor | very poor |
| fragmentation (m) | 0,01-0,3 | 0,2-2,0 | 0,4-5 | 1,5-9 | 3-20 |
| 2nd lay-on blast/drill g/t | 0-50<br>0-20 | 50-150<br>20-60 | 150-400<br>60-150 | 400-700<br>150-250 | >700<br>>250 |
| hangups as % of tonnage | 0 | 15 | 30 | 45 | >60 |
| Dia of draw zone, m | 6-7 | 8-9 | 10-11,5 | 12-13,5 | 15 |
| Drawpoint span, m<br>grizzly<br>slusher<br>LHD, m | <br>5-7<br>5-7<br>9 | <br>7-10<br>7-10<br>9-13 | <br>9-12<br>9-12<br>11-15 | <br><br><br>13-18 | |
| Brow support | steel and concrete reinforced concrete | | concrete | blast protection | |
| Drift support | lining, rock reinf., repair techniques | | lining reinf. | | |
| Width of drawpoint | 1,5-2,4 | 2,4-3,5 | 2,4-4 | 4 | |
| direction of advance | towards low stress | | towards high stress | | |

The rock mass strength (RMS) can be determined from consideration of the intact rock strength (IRS) and the rock mass rating (RMR). The rock is rated in terms of the Rock Quality Designation and joint spacing (or fracture frequency per metre) and joint condition. The intact rock strength is reduced to 80% of its original value. The ratings are added together to arrive at the rock mass strength (RMS). The rock mass strength is adjusted in terms of the factors that relate to the mining environment: weathering, orientation of fractures and joints and blasting procedures. The adjusted rock mass strength is defined as the design rock mass strength (DRMS) and can be related to mining induced stresses.

The value of this procedure is that it forces the rock mechanics and planning engineer to recognise that the strength of the rock mass will be less than that given by laboratory samples and that this lower strength material could be further weakened by weathering, unfavourable orientations of joints and fractures and poor blasting practice. The effect of induced stresses on this weakened material will be more severe than the effect of the same stresses on pristine rock.

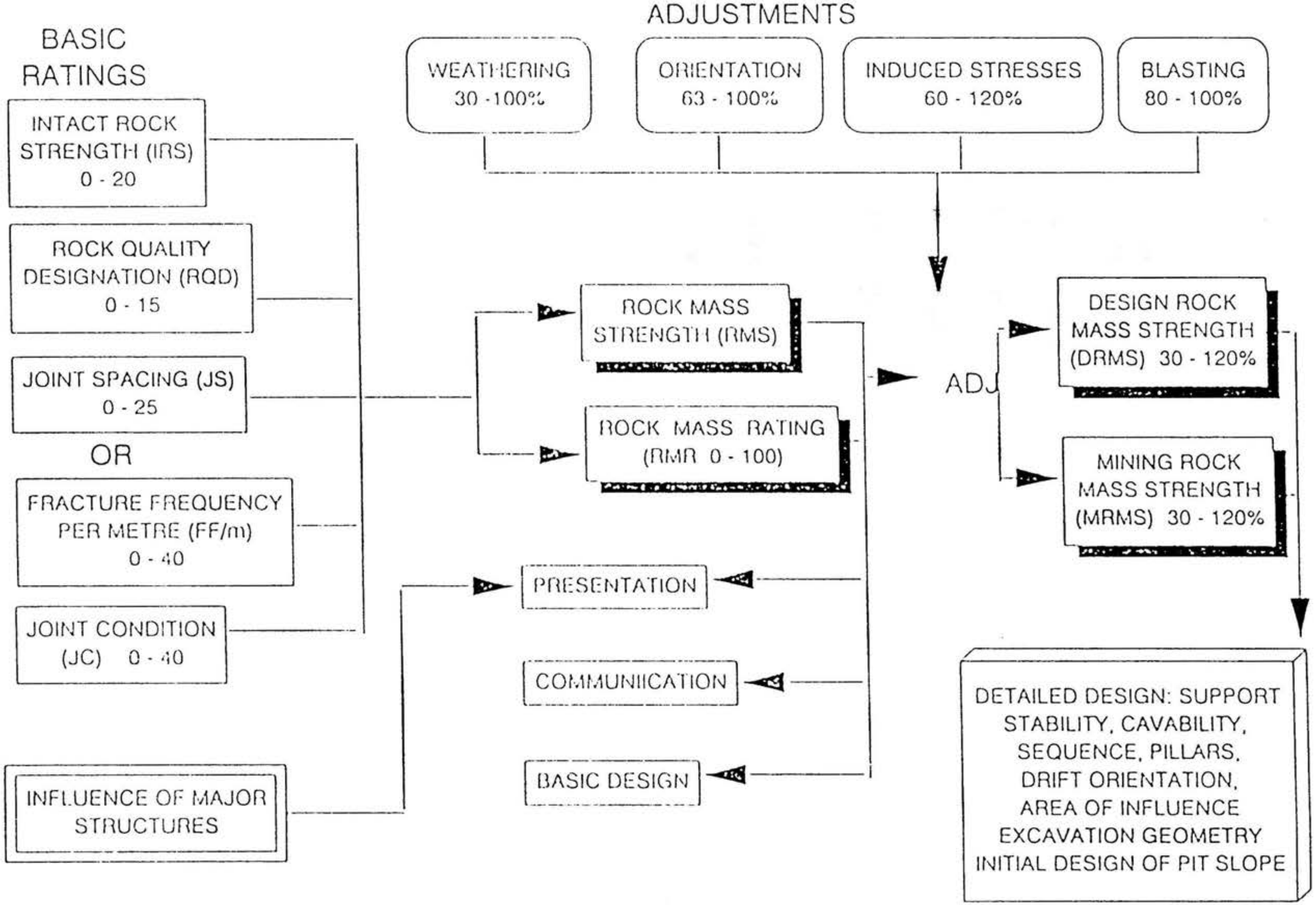



Figure 2.1. Laubscher's Rock Mass Classification

The originally calculated rock mass rating (RMR) is adjusted to take cognisance of potentially damaging factors that exist in the mining environment. These are:

* weathering effects which can reduce the strength of the rock or joint. This effect is rated between 30 and 100%.

* unfavourably oriented fractures and joints can affect the stability of the excavation. This effect is rated at between 63 and 100 %.

* induced stresses as a result of mining can have a variable effect on the rock mass. Stresses normal to a joint plane can actually increase the friction on the joint and increase the strength of the rock mass. This effect is rated at between 60 and 120%.

* Blasting can damage rock. This effect is rated at between 80 and 100%.

The combined multiplicative effect of these adjustments can reduce the rock mass rating (RMR) to as low as 30% of the originally calculated rating or actually increase the rating by 120%. These combined adjustments of between 30 and 120% multiplied by the original rock mass rating result in a new rating which is defined as the mining rock mass rating (MRMR). This adjusted rating is used in many aspects of mine design.

Figure 2.1 is a summary of Laubscher's rock mass classification. Reviews exist in the literature.

## 2.4.5. OTHER ASPECTS OF ROCK MASS CLASSIFICATION

Correlations have been developed between several aspects of mine design and specific rock mass classification systems. Several correlations that have been used at Premier are set out below.

**Bieniawski's RMR**

1. Hoek and Brown (1980) propose a method of estimating rock mass strength which uses the RMR classification:

$$(\sigma_1/\sigma_c) = (\sigma_3/\sigma_{c)} + m\,(\sigma_3/\sigma_c) + s \qquad \text{(eqn 2.2.)}$$

where $\sigma_1$ = major principal stress at failure
$\sigma_3$ = the applied minor principal stress
$\sigma_c$ = uniaxial compressive strength of the rock material
m & s are constants relate to the rock properties and condition of the rock

For undisturbed rock masses:

$$m = m_i \exp\{(RMR - 100)/28\} \qquad \text{(eqn 2.3.)}$$
$$s = \exp\{(RMR - 100)/100\} \qquad \text{(eqn 2.4.)}$$

where $m_i$ relates to intact or undamaged rock strength.

For disturbed rock masses (blast damaged excavations):

$$m = m_i \exp\{(RMR - 100)/14\} \qquad \text{(eqn 2.4.)}$$
$$s = \exp\{(RMR - 100)/6\} \qquad \text{(eqn 2.5.)}$$

Numerical models including solution schemes such as FLAC and UDEC require that Mohr-Coulomb cohesion and friction be determined for disturbed rock masses that have been affected by successive stages of stress loading associated with cave mining. Correlations between Hoek and Brown empirically determined failure envelopes and Mohr-Coulomb failure envelopes are one of the few reliable ways of determining required parameters. The parameters can in turn be used to determine whether a rock will behave in an elastic manner, a plastic manner with determinate displacements or in a plastic manner with large, indeterminate displacements. Brittle failure is the most probable failure mode for an elastic rock mass. Brittle failure with strain softening can be expected in a plastic rock mass with determinate displacements. A plastic failure mode with large indeterminate displacements can be expected in a rock mass with low cohesion and friction.

The most commonly used correlation between Barton's and Bieniawski's classification is:

$$RMR_B = 9 \ln Q + 44 \qquad \text{(eqn 2.6.)}$$

$RMR_B$ = Bieniawski's RMR rating (eqn 2.7.)

A correlation based on 99 cases in a recent publication where RMR and Q values are given (Bieniawski, 1989) is:

$$RMR_B = 7 \ln Q + 47.9 \quad (r^2 = 0{,}67) \qquad \text{(eqn 2.8.)}$$

An extensive mapping exercise was undertaken by Taylor at 30 localities at Shabani and King chrysotile asbestos mines, using Laubscher's, Bieniawski's and Barton's Q systems. This produced the following correlations (Taylor, 1980):

$$RMR_B = 7.6 \ln Q + 51 \quad (r^2 = 0{,}81) \qquad \text{(eqn 2.9.)}$$

$$RMR_L = 8.2 \ln Q + 42.4 \quad (r^2 = 0{,}90) \qquad \text{(eqn 2.10.)}$$

$RMR_L$ = Laubscher's RMR rating

From the work, Taylor established the following correlation between Laubscher's and Bieniawski's systems:

$$RMR_L = 0.97\, RMR_B - 5.91 \quad (r^2 = 0{,}87) \qquad \text{(eqn 2.11.)}$$

The closest correlation between Laubscher's RMR and Barton's Q that can be determined from the available data is:

$$RMR_L = 9 \ln Q + 37 \qquad \text{(eqn 2.12.)}$$

It should be noted that all the above relationships are empirical and have been developed by observation in specific geological domains. There is no way to calculate constitutive parameters from rock mass ratings and it is risky to extrapolate beyond the confines of the available data base. The effect of stress is probably the most difficult to incorporate into rock mass ratings. Laubscher has introduced an adjustment to allow for the effect of induced stress and Barton uses the Excavation Support Ratio (ESR) concept to adjust for stress. In both classification systems the application of the stress adjustments remains difficult.

## 2.5. ANALYTICAL DESIGN PROCEDURES

Most analytical procedures that have been used in mine design assume that the rock mass is isotropic and homogenous and will respond elastically to induced stresses, and rock mass response is calculated in terms of stress and strain laws. Young's modulus and Poisson's ratio are important parameters that characterise the rock mass. In the type of model used, strains are small and the modulus of the rock large. The stresses can be inherent in the rock mass as a result of the tectonic history of the area or induced in the rock mass by mining operations in the vicinity of the excavation, or both. Few rock masses meet the requirements of homogeneity, isotropy and continuity required by elastic theory. Nevertheless, the advent of computers and elastic computer simulation models have allowed detailed analysis of the stresses and strains that can be expected around an excavation in most mining configurations. The theory of elasticity as applied to rock masses is well documented in the literature and has been invaluable in allowing mining engineers to better understand the behaviour of the rock mass in which they create their excavations.

A typical design procedure (Brady & Brown, 1985) starts with planning an excavation to satisfy its design requirements. The boundary stresses around the excavation are then determined. These stresses are compared to the uniaxial compressive strength and tensile strength of the rock mass in which the excavation is to be developed. If stress levels are unlikely to cause failure of the wall rock in compression or tension, the influence of major discontinuities is considered and the design modified to accommodate the effect of the discontinuities, if these create problems.

If stress levels are likely to lead to failure of the rock mass, the design of the excavation must be modified to minimise the effect of this failure and support must be designed to limit the rock mass failure to acceptable levels, if this is possible.

The methodology, simply stated here, is often fraught with difficulties. An excavation will interact with the rock mass in which it is developed and will have an influence on other excavations in the vicinity. The effect of the excavation on the rock immediately surrounding the excavation must be considered, as well as the effect of excavation on any other excavations nearby. For a single excavation of known dimension, this is a relatively simple calculation, especially with the aid of an appropriate computer programme. Where the excavation is large and of a largely indeterminate shape, as with many cave operations, the effect of the excavation is difficult to assess.

Planes of weakness, especially faults and shear zones, can have a major influence on the elastic stress distribution around an excavation. If displacement on a discontinuity, as a result of induced stress changes, can be predicted , the effect of the displacement on the stress distribution can be calculated. The shape of the excavation has an important bearing on the stress distribution around the excavation. In general, it can be stated that the smaller the radius curvature of the excavation boundary the greater will be the stress concentration in that area of the excavation sidewall. The effect of the excavation shape on the stress distribution is amenable to computer analysis.

An important part of the design process is to establish failure criteria that will allow correct prediction of whether or not a rock mass with known compressive and tensile strength will fail in a given stress regime. The Mohr-Coulomb failure criterion is well known. The model has only two active material constants, $c_m$ and $\phi_m$, the cohesion and friction angle of the rock mass respectively. The excess shear strength is then compared to the shear strength of the material through the equation

$$\tau_m = c_m + \sigma_n \tan \phi_m \qquad \text{(eqn 2.13.)}$$

where $\sigma_n$ is the normal stress across the potential failure plane. This model is really only applicable to intact rock and soils.

Failure takes place on joint planes and the strength of the joint needs to be defined. It is usually calculated in terms of joint cohesion $c_j$ and $\phi_j$. If the excess shear strength is greater than the strength of the material, failure can take place through the intact rock. Typically, however, failure is joint controlled. Several numerical models incorporate a "joint model" where the strength of the joint or joint system helps to define rock mass behaviour.

More recently, a semi-empirical failure criterion has been established (Hoek & Brown, 1980) which is widely used and has been incorporated into many computer programmes. This failure criterion is defined in terms of the major and minor stress effective at the time of failure.

$$\sigma'_1 = \sigma'_3 + (m\sigma_c\sigma_3 + s\sigma^2_c)^{1/2} \qquad \text{(eqn 2.14.)}$$

$\sigma'_1$ = major principal stress effective at time of failure
$\sigma'_3$ = minor principal stress effective at time of failure
$\sigma_c$ = uniaxial compressive strength of the intact rock
m and s are empirical constants.

The failure envelope is curved and the values of friction and cohesion are not unique but depend on the confining stress at the time of failure. The failure criterion is widely used in underground support design and accurate determination of m and s for both broken and intact rock underground is useful for a number of purposes.

Rock failure can be controlled by installing active support on the periphery of the excavation. A simple calculation (Brady & Brown, 1985) shows that in most cases the support does not fulfil its function by "opposing" the forces imposed on it. Typically a zone of broken rock will exist around an excavation. The support functions by generating and maintaining a state of triaxial stress in the fractured domain by mobilising friction between the surfaces of the rock fragments.

Mechanical principles are widely used to design support in layered rock strata. The principles governing the behaviour of the rock response in such situations are discussed in numerous articles in rock mechanics literature. Excavation design in jointed rock can be done by detailed mapping of the rock and determining the geometry of the blocks that make up the rock mass. Knowledge of friction coefficients of the joints, together with mechanical principles, can be used to determine the stability of individual rock blocks and the support needed to maintain stability. Concepts such as keyblocks and probability analyses can be used with these calculations to design effective support.

Most mechanised cave layouts create a regular pattern of complex pillars comprising major and minor apices on the extraction level. Extraction ratios as high as 50 percent can result. Any analytical design procedure must aim at calculating pillar strength and the gain in strength due to the support system installed. Methods of estimating the strength of square and rectangular pillars have been developed. Calculation of the strength of other pillar shapes and the increase in pillar strength due to the effect of installed support is still imprecise.

## 2.6. OBSERVATIONAL DESIGN METHODS

The best known and most widely used observational design method is the New Austrian Tunnelling Method (NATM). This method has been widely used in Europe over the past three decades, often in extremely poor ground conditions and its implementation was first comprehensively described by Rabcewicz (Rabcewicz, 1965). The first principle of the method is to harness the inherent strength of the rock mass to support itself. This involves inhibiting rock deterioration, joint deterioration and loosening due to movement toward and into the tunnel.

The NATM can be used in conjunction with rock mass classification systems, which are used to characterise the rock. If the classification indicates that potentially squeezing or swelling rock conditions exist, NATM principles can be used as the basis for support design.

Monitoring of the rock mass is an important aspect of the method. Support is installed and the behaviour and response of the rock mass is monitored. If excessive convergence of the tunnel continues, additional support is installed until the rock mass stabilises.

## 2.7. DESIGN PROCEDURE AT PREMIER MINE

It was decided by the author to use Laubscher's mining rock mass rating to gather and assess the geotechnical data needed to plan the BA5 mechanised cave. All four rock mass classification systems considered provided an accurate assessment of most of the major rock mass parameters that could be measured in the orebody, or obtained from samples taken from the orebody and analysed in the laboratory. Laubscher's mining rock mass rating has been used on many cave mines throughout the world and only this classification system provided useful correlation on the basis of the Rock Quality Index calculated. These correlations are set out in Table 2.1. Determination of a reliable correlation between Laubscher's, Barton's and Bieniawski's rock mass classification systems was considered important as correlations between specific aspects of the various Rock Quality Indices and rock mass behaviour have been developed and reported in the literature. The correlation between Barton's Q and Bieniawski RMR and the semi-empirical Hoek and Brown failure criterion is of particular importance.

None of the rock mass classification systems provided an accurate assessment of the stress levels that would be induced around the cave excavation.

Observational design methods were used extensively to measure the rate of caving, stress changes, displacements and support system effectiveness.

The failed open stope mining method had created a number of highly stressed pillars. Drawpoints and tunnels beneath these pillars, established to extract as much ore as possible from the pillars in a panel cave type mining method, collapsed and several million tons of high grade ore was lost.

It was considered essential that a method of support design should be developed that would ensure that excavations remained stable for the duration of their intended life. If it is impractical to support the area, the design method should define this. A detailed geotechnical investigation of the kimberlite orebody was undertaken by the author to characterise the various types of kimberlite and surrounding wallrock (Bartlett, 1994. Stacey & Bartlett, 1991). Although the geotechnical assessment was initially undertaken for support design, it was subsequently expanded to define parameters that were needed to determine the area that needed to be undercut to induce caving, the rate of caving, primary and secondary fragmentation, and production tempo from drawpoints. All these aspects of cave mining are linked and relate to defining rock mass and joint strengths, and the displacements that will occur as a result of rock mass or joint failure. These displacements are all induced by stress changes.

Detailed correlations were established between parameters derived in the laboratory and rock mass ratings, as well as correlations between rock mass ratings and m and s parameters used in the semi-empirical Hoek and Brown failure criterion. Extensive numerical stress analysis was commissioned by the author to calculate the stress changes that could be expected around the cave and analytical design procedures were used to calculate where these stresses would lead to rock mass and support system failures. Numerical modelling was used to determine optimum mining sequences, support design, the area that would have to be undercut to induce caving and to predict secondary fragmentation in the drawpoints.

**Determination of Stress Levels**

At Premier geotechnical assessment of the various rock domains has been done at different times and for different purposes. Stress determinations have been carried out in the gabbro and in the Hypabyssal Kimberlite. Stress determinations in the gabbro were carried out to assess whether the sill would act as a competent crown pillar when the originally planned massive open stopes were installed beneath it. These showed that mining in the open pit had unloaded the sill and a vertical stress of 4 MPa was measured 60 metres below the pit bottom at a depth of 450 metres below surface. The horizontal stress was anisotropic with a value of 17 MPa in the strike direction and 10 MPa in the dip direction.

**Table 2.2. Stress Parameters**

| STATION MINING STAGE | CALCULATED STRESS | MEASURED STRESS |
|---|---|---|
| In Situ Stress (Mpa) | | (MPa) |
| 445 Level - Gabbro Sill | H:V 10:4 | H:V 14:4 |
| 732 Level - Norite Wallrock | 19:13 | |
| 615 Level - Kimberlite | 16:6 | |
| Stress change - 615 and 630 Levels | | |
| Tunnel Development | 2.6 | 2 - 3 |
| Drawbell Development | 10 | -11 - 10 |
| Undercut Development | 30 | 20 - 30 |
| Cave Exhausted | 1.6 | ? |

A series of stress measurements in the hypabyssal kimberlite at the 645 metre level suggested that maximum principal stress was negative with the direction attributed to the influence of the high, near vertical pit sidewalls. Computer simulations using a three dimensional boundary element numerical model were carried out subsequently to predict how the proposed cave mining would affect stress distribution in the area affected by the cave. Table 2.2 is a summary of the predicted stress levels associated with the subsequent mining stages.

**Determination of Rock Strength**

Detailed line mapping of the norite country rock was undertaken as sidewall instability and block fallouts were experienced in ore passes sited in the norite. Problems were anticipated when the stress changes that accompanied the retreating undercut began to affect the norite (Stacey & Bartlett, 1991).

Mapping of the kimberlite defined a core of competent Hypabyssal Kimberlite within the BA5 mining block. Three distinct kimberlite types and a dyke system were mapped within this core kimberlite. A fourth type of kimberlite was mapped that showed gradational change from a hypabyssal to a tuffisitic kimberlite. This gradational change was evident both from the petrology of the rock and from the joint patterns mapped within this rock type. A fifth kimberlite was defined as a true Tuffisitic Kimberlite Breccia (TKB). During the core drilling programme which was carried out at an early stage in the BA5, it became evident that areas of the Tuffisitic Kimberlite Breccia would exhibit swelling ground characteristics if the rock was allowed to become wet (Bartlett, 1994).

The detailed mapping was used to determine the rock mass classification of the rocks using Laubscher's system. Table 2.4. sets out the results of laboratory tests of the effect of water on decomposing kimberlite in terms of swelling pressures and volume changes that was measured

Triaxial tests of core from the various rock types was undertaken at the CSIR to determine Mohr-Coulomb and Hoek and Brown failure criteria for intact rock. The strength of the broken rock was initially determined using RMR and Hoek and Brown empirical correlations. It should be noted that the cave mining method imposes several stress cycles on the rock mass around the undercut excavation. For effective support design, the strength of the rock after it has been affected by the harshest and most damaging stress cycle (usually the retreat of undercut) needs to be established. A detailed monitoring programme was planned and implemented to retrospectively analyse field data to ensure that calculated values of m and s for jointed and stress-damaged rock correlated with field measurements. Accurate determination of rock strength, taking into account both intact rock and joint strength after it has been affected by the undercut, is important in determining support design, fragmentation and area of undercut.

**Table 2.3. Rock Parameters at Premier Mine**

| PARAMETER | TKB | HYP | NORITE | GABBRO |
|---|---|---|---|---|
| UCS (MPa) | 50-110 | 110-160 | 50-160 | 284 |
| E (GPa) | 20 | 43 | 70 | 72 |
| μ | 0.25 | 0.28 | 0.350 | 0.3 |
| cm (MPa) | 30 | 120 | 130 | 180 |
| φm (degrees) | 35 | 15 | 35 | 45 |
| mi | 16 | 7 | 24 | 26 |
| si | 1 | 1 | 1 | 1 |
| m | 2 | 5 | 15 | 20 |
| s | 0.001 | 0.05 | 0.05 | 0.07 |
| cj (MPa) | 0.1 | 0.15 | 0.12 | 0.13 |
| φj (degrees) | 30 | 45 | 35 | 40 |
| RMR | 50 | 55 | 45 | 65 |
| MRMR | 25 | 33 | 33 | 48 |
| RMS | 35 | 59 | 36 | 126 |
| DRMS | 25 | 36 | 26 | 94 |

mi = Hoek and Brown m parameter for intact rock (core)
si = Hoek and Brown s parameter for intact rock (core)
m = Hoek and Brown m parameter for broken rock
s = Hoek and Brown s parameter for broken rock
TKB = Tuffisitic kimberlite breccia
HYP = Hypabyssal kimberlite

Numerical modelling was used extensively to determine mining sequence (McKinnon September 1992), support design (McKinnon, October 1992), area of undercut (Howell et al.,1993) and fragmentation (Esterhuizen, 1996). The parameters needed for this modelling, using the most suitable computer codes, are set out in the tables 2.2. and 2.3. The way in which the parameters were derived is set out where this is considered appropriate. Figure 2.2 (a), (b) and (c) set out the range of shear strengths, friction angle and cohesion of joints in kimberlite at Premier as determined by Barton's method. Figure 2.2. (d) and (e) set out the range of cohesion and friction angle of joints in kimberlite at Premier as a function of Laubscher's joint condition rating (JRC).

An important objective of this geotechnical assessment was to establish correlation between rock mass parameters that can be measured in underground excavations and then apply rock mass ratings to determine support design, area of undercut and fragmentation without having to go through the expensive and time consuming process of detailed laboratory testing and complex numerical modelling.

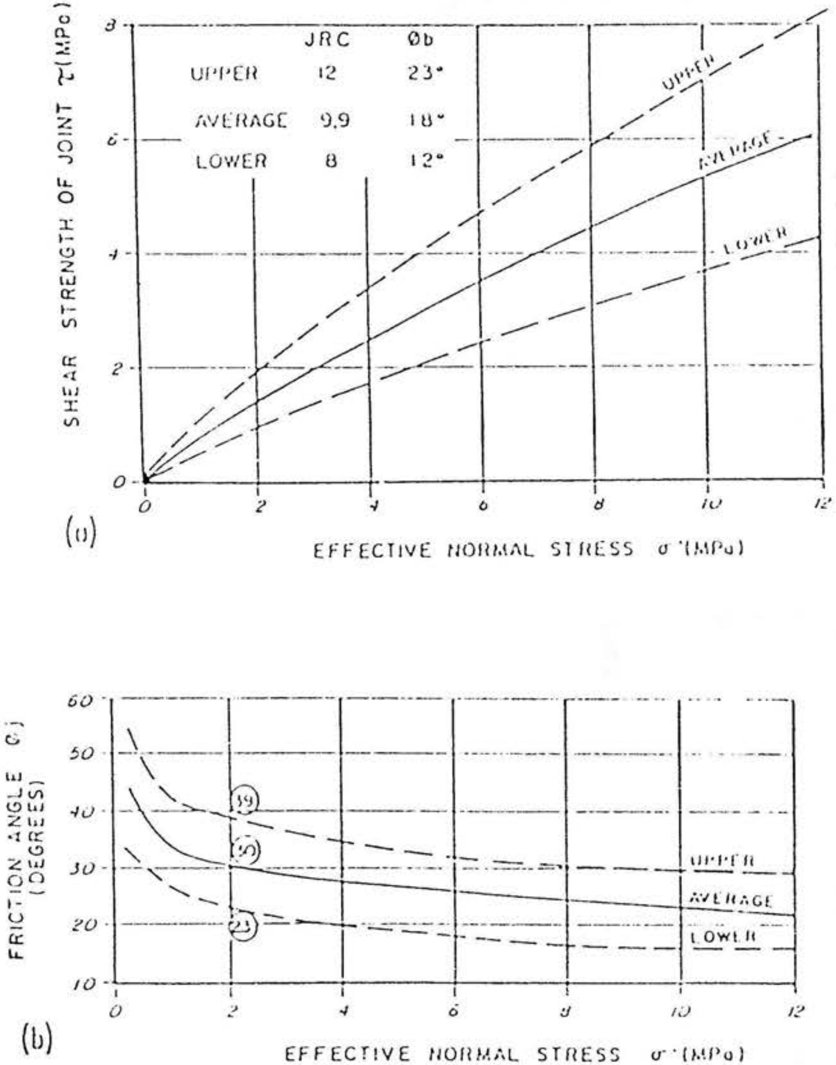
Much of this thesis deals with the methodologies of designing effective support, determining the undercut span needed to induce caving and the methodology of reliably determining fragmentation using detailed analytical design. The thesis also deals with the development of the methodology of correlating readily measurable rock mass parameters with expected rock mass behaviour based on detailed analytical design rather than experience as the intermediate step.

**Table 2.4. Pressure changes in squeezing rock**

| | Constant volume method | | Consolidation method | |
|---|---|---|---|---|
| Preload | Specimen No. | Swell Pressure | Specimen No. | Swell Pressure |
| **Kimberlite 1% water** | 03<br>04<br>10 | 25 Mpa<br>29,1<br>31,7 | 06<br>08 | 42,0 Mpa<br>40,0 |
| **Kimberlite 10% water** | 02<br>05<br>11 | 36,0<br>31,9<br>30,4 | 09<br>07 | 45,0<br>48,0 |
| **Tutuka Mudstone 1% water** | B<br>C<br>G | 6,4 Mpa<br>7,2<br>6,3 | I<br>E | 9,8 Mpa<br>7,7 |
| **Tutuka Mudstone 10% water** | A<br>D<br>H | 5,6<br>7,5<br>6,3 | I<br>F | 6,9<br>6,0 |

Taken from Pellissier & Vogler, 1990

The kimberlite used in the tests was supplied by Premier Mine. Swelling pressures developed in kimberlite are higher than in the Tutuka mudstone by a factor of 5. The mudstone is also known to create squeezing rock conditions.



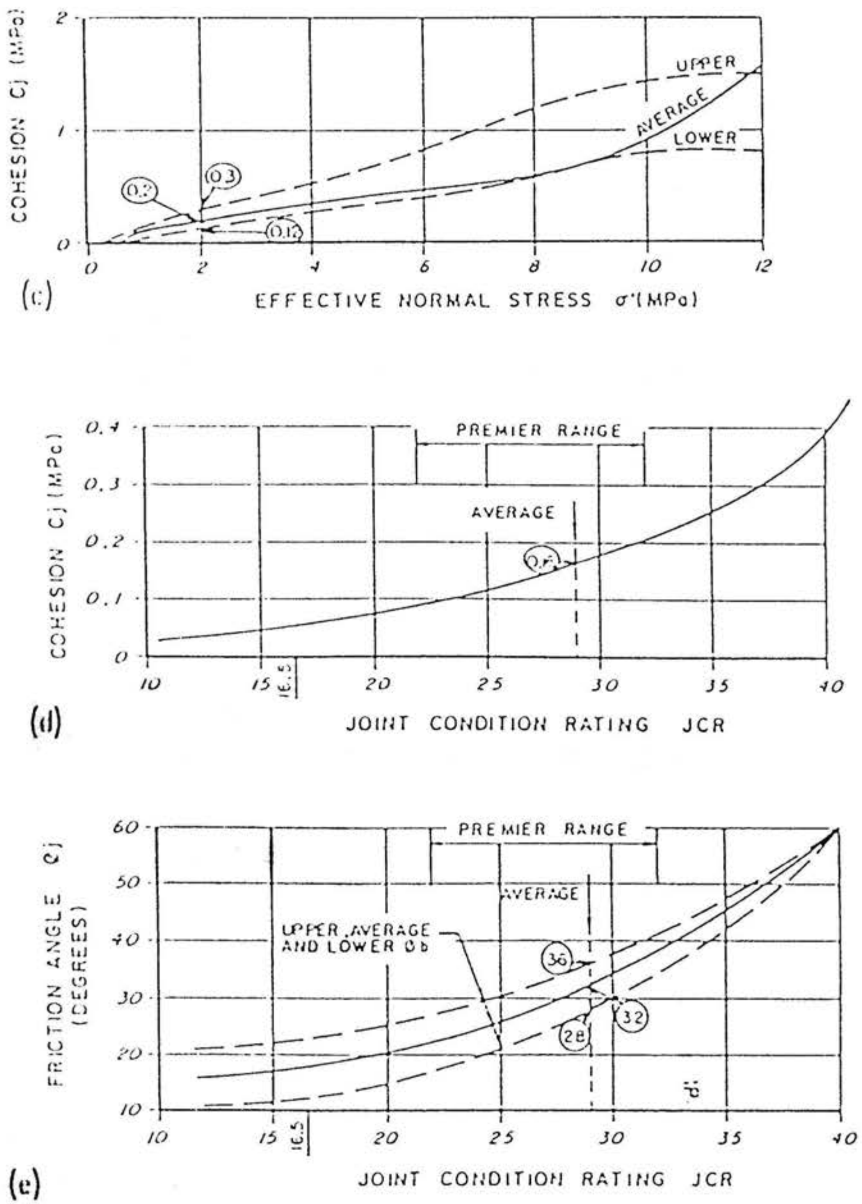




FIGURE 2.2. JOINT STRENGTHS AS CALCULATED USING BARTON'S AND LAUBSCHER'S METHODS

## 2.8 CONCLUSIONS

The geotechnical assessment defined two broad types of kimberlite within the oreblock to be extracted by mechanised caving in the BA5. These two kimberlite types are defined as:

- A core of competent, well jointed Hypabyssal Kimberlite. Several types can be differentiated petrographically within this core, but rock mass parameters which impact on mine design do not vary amongst the kimberlite types. The Hypabyssal Kimberlite is defined, geotechnically, as a single kimberlite type.

- The Hypabyssal Kimberlite core is surrounded by Tuffisitic Kimberlite Breccia. The latter is a poorly jointed, inelastic rock, which decomposes rapidly when wet as a result of its high montmorillonite content, causing large pressure and volume changes. Water has filtered into the pipe in time, along joints in both the core kimberlite and the norite, resulting in degradation of the Tuffisitic Kimberlite Breccia adjacent to these rock types. Away from the contact zones, the Tuffisitic Kimberlite Breccia is a reasonably competent rock except when it comes into contact with free water and decomposes.

A well-jointed, competent, 75-metre-thick, dipping gabbro sill overlies the entire BA5 cave.

Norite surrounds the entire orebody. Away from the kimberlite pipe, this norite is a competent, well jointed rock, creating support problems only where it is traversed by shear zones. Within 30 metres of the pipe, chlorite and serpentine have formed as gouge along joints resulting in weak joints. This gives the norite a blocky nature, with these blocks prone to movement if not constrained by adequate support. Immediately adjacent to the pipe a narrow zone of decomposing norite may exist.

All these rock types were mapped in considerable detail and the results assessed using Laubscher's rock mass rating. These ratings were used to make an initial assessment of the parameters that would influence mine design, such as area of undercut required to induce caving, support design and expected fragmentation. Ratings were used to estimate parameters required for numerical modelling, including cohesion and angle of friction. Modelling was used extensively to predict stress changes that could damage minor and major apices between drawbells and drawpoint brows to the extent that they possessed only a residual strength, irrespective of the support installed as well as stress changes in the far field of the abutment zone tens of metres away from the actual abutment.

The following chapters define how the geotechnical assessment, numerical modelling, and observational design methods, together with experience and engineering judgement were used to address problems associated with cave mining in a kimberlite with coarse fragmentation.

mass around the undercut is not well studied and documented and predictions as to the behaviour of this area of the rock mass and its effect on the mining operation are complex. The local geology and major structures in the area have an important bearing on the cave mining operation.

## 3.2. AREA THAT MUST BE UNDERCUT TO INDUCE CAVING

An area sufficiently large to induce continuous caving must be undercut to ensure the success of the cave mining operation. This area is a function of the rock mass strength and the regional stress field that prevails before mining starts and as mining progresses. The following aspects are considered important in cave mining (Taylor, 1980):

* The number, attitude and spacing of joint sets. A combination of flat dipping and sub-vertical joints are conducive to caving.

* The frictional properties of joints are important in the caving process.

* The magnitude and orientation of the principal stresses have an effect on the caving process. High lateral stresses in strong rock inhibit caving. If the stress:strength ratio is high enough, however, failure in shear can occur and assist fragmentation. Low confining stresses allow tensile failures to occur, but induce little shear failure. Primary fragmentation will be largely joint controlled.

* The relationship of the area being caved to other areas that have been mined, as this can diminish confining stresses that inhibit caving.

* The importance of the strength of the rock increases as the number of joints decreases as low strength rock can fail in compression if joints are not present.

Numerical simulation (Mahtab & Dixon, 1976) suggests the following:

* Lateral confinement decreases the intensity of the shear stresses in the periphery of the undercut and spreads them more evenly over the cave back.

* As the undercut size increases both the peripheral shear stresses and tensile stresses over the cave back increase.

* A decrease in the internal angle of friction of the rock results in an increase in the magnitude of the peripheral shear stresses.

* Boundary slots increase the intensity of both the peripheral stresses and those over the cave back.

* Low angle fractures and joints in the rock mass have a marked effect on peripheral and cave back stresses.

* Near vertical fractures also affect stresses at the periphery and over the cave back.

The most accurate and widely used methods of predicting the area that must be undercut to induce caving (or, conversely, to ensure that crown pillars above open stopes remain stable) are empirical correlations between a Rock Quality Index and the undercut area, as defined by the hydraulic radius. The shape of the undercut area plays an important role and the concept of hydraulic radius, defined as the ratio of the area of undercut to the length of the perimeter of the undercut, is used to modify the area of undercut. The most widely used correlations are:

1. Laubscher's correlation, where the adjusted rock mass rating (Mining Rock Mass Rating or MRMR) is correlated with hydraulic radius. Figure 3.1 is a diagram compiled by Laubscher (Laubscher, 1994) where the Mining Rock Mass Rating of numerous caves is plotted against hydraulic radius.
2. Matthew's correlation, where an adjusted NGI number is correlated with hydraulic radius. The Matthew's method was developed for open stope design, but has been extended to include caving orebodies. The method has recently been comprehensively reviewed (Stewart & Forsyth, 1993).

### 3.2.1. EXPERIENCE AT PREMIER MINE

Two ore blocks above the sill caved with difficulty and impacted adversely on production. The gabbro sill crown pillar above three large open stopes failed prematurely, again impacting adversely on production. Problems with caving of the sill above the BA5 cave were anticipated.

The inability to correctly predict the undercut area needed to induce caving had therefore proved a major problem for Premier. As the mine planned to use cave mining in the exploitation of its ore reserves, in future mining blocks, the author considered it vital to have a proven tool that could be used to accurately predict the undercut area needed to induce caving in various kimberlite types at increasing depth. Considerable effort was therefore expended by the author in back analysing caving history at Premier using Laubscher's empirical correlation between hydraulic radius and Mining Rock Mass Rating. It should be noted that when Premier first started to plan the BA5 cave in 1987, Laubscher's chart contained relatively few correlation points (Laubscher, 1987), especially where the Mining Rock Mass Rating was above 50 and a hydraulic radius in excess of 20 was predicted.

Table 3.1. shows the correlation found between adjusted rock mass rating and hydraulic radius found for the rock types caved at Premier Mine. Table 3.2. is a back analysis of the correlation between the parameters that go to make up Laubscher's classification and the hydraulic radius at which caving occurred. These tables show that correlation is good for caving in all types of kimberlite, both above and below the sill, but poor for caving of the 75 metre thick gabbro sill.

Numerical modelling was commissioned by the author to establish whether it would be more accurate than Laubscher's empirical correlation, especially in assessing the effect of stress on the caving process, as the mine was planning to implement caves at depths of up to 1000 metres below surface. The exercise clearly showed that caving in all rock types at Premier was joint controlled and that, provided accurate joint and rock mass parameters could be determined, numerical modelling could be used to predict the hydraulic radius needed to initiate caving

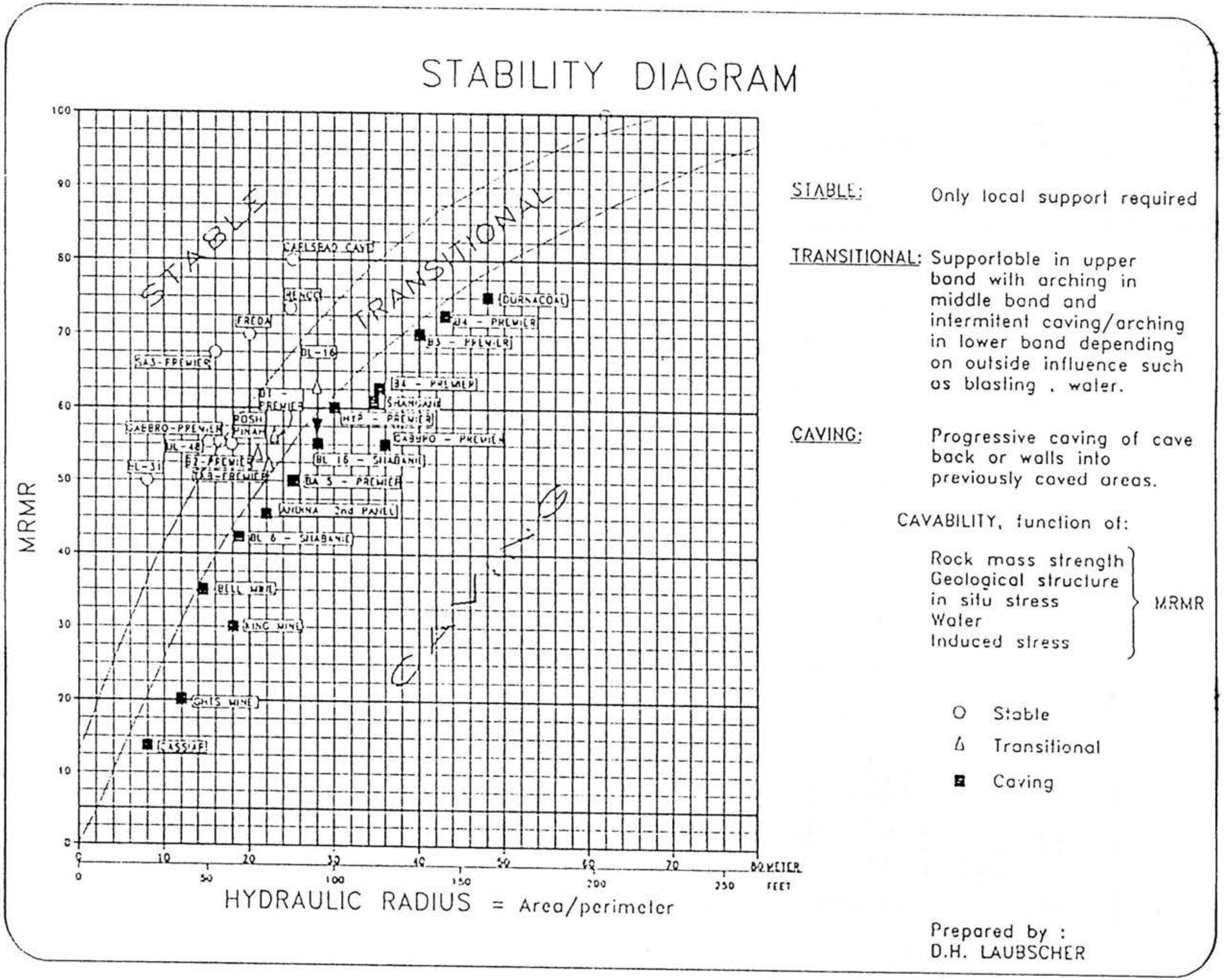



Figure 3.1. Mining Rock Mass Rating vs hydraulic radius

(Howell et al., 1992 & 1993). Numerical modelling, however, was no more accurate than rock mass classification correlation and the exercise was not pursued.

**Table 3.1. Dimensions of Caves and Stopes at Premier Mine**

| Mining Block | Lift Height | Area of undercut | MRMR | Hydraulic Radius | Comments |
|---|---|---|---|---|---|
| B1 (above sill) | 155 m | 82 x 100 m | 55 | 23 | Boundary weakening |
| B2 (above sill) | 155 m | 100 x 70 m | 53 | 21 | Caved easily |
| B3 (above sill) | 155 m | 200 x 130 m | 70 | 40 | Boundary weakening |
| B4 (above sill) | 115m | 200 x 150 | 72 | 43 | Boundary weakening |
| L1 STOPES | | | | | |
| SA3 | 90m | 80 x 55 | 66 | 16 | Open stopes |
| SA2 | 90m | 80 x 60 | 66 | 17 | |
| SA1 | 90m | 80 x 60 | 66 | 17 | |
| BA5 CAVE | | | | | |
| TKB | 100m | 90 x 90 | 52-56 | 22.5 | |
| HYP | 100m | 120 x 120 | 58-62 | 30 | |
| Gabbro | 100m | 60 x 60 | 42 | 16 | |
| Gabbro* | 100m | 130 x 120 | 65 | 36 | |
| Gabbro** | 100m | 200 x 220 | 75 | 52 | |

* Gabbro sill was initially 75 metres thick. Caving of the sill initiated at a low hydraulic radius of 16. Caving in the gabbro is almost entirely joint controlled and the horizontal stress, which increased as the gabbro sill caved - the same total stress was transmitted through a diminishing "beam" - had the effect of increasing the MRMR dramatically. When only 20 metres of sill thickness remained a hydraulic radius of at least 36 was required to propagate the cave.
** The gabbro sill eventually caved through into the overlying open pit when an area of 220 x 200 metres in plan view existed on the 500 metre level. The calculated hydraulic radius was 52. This shows that the MRMR in the gabbro changed from 42 to 75 as a result of a change in stress conditions and illustrates the importance of stress levels in accurately determining the MRMR.
Hydraulic Radius is defined as the area of the undercut divided by the length of undercut perimeter

Four block caves in kimberlite were operated above the gabbro sill. Two of the caves (B1 and B2) were sited largely in Tuffisitic Kimberlite Breccia and caved easily. The other two caves were sited largely in Hypabyssal Kimberlite and caved with difficulty. None of the caves carried any overburden and stress levels were minimal. Photograph P.3.1. illustrates the 170 metre wide undercut that existed in the B4 before caving was initiated by boundary weakening. Points that relate only to caving history at Premier from Table 3.1. have been plotted on Laubscher's diagram in Figure 3.2. This plot shows that Laubscher's empirical diagram is a good predictor of caveability in all kimberlite types, both above and below the sill. After initiation, caving continues to propagate with a progressive reduction in the undercut area needed to cause caving, as increasing horizontal stresses aid the caving process. In the gabbro, high horizontal stresses provide effective clamping forces across vertical joints which inhibits the caving process. A progressively larger undercut area is needed for caving to propagate in this competent rock. It

**Table 3.2. Back Analysis of Caving at Premier Mine**

| PARAMETER | TKB | HYP | GABBRO |
|---|---|---|---|
| UCS rating | 8 | 12 | 20 |
| RQD rating | 12 | 14 | 12 |
| FF rating | 16 | 9 | 13 |
| Jw/W rating | 18 - 20 | 23 - 27 | 10 - 12 |
| CALCULATED RMR | 52 - 56 | 58 - 62 | 53 - 57 |
| WEATHERING | 1 | 1 | 1 |
| BLASTING | 1 | 1 | 1 |
| INDUCED STRESSES | 1 | 1 | 0.9 |
| JOINT ORIENTATION | 1 | 1 | 0.8 |
| CALCULATED MRMR | 52 - 56 | 58 - 62 | 38 - 41 |
| HYDRAULIC RADIUS AT ONSET OF CAVING | 22.5 | 30 | 17 (stopes)<br>16 (BA5) |

emphasises that the effect of stress on the rock mass needs to be understood and used to establish the Mining Rock Mass Rating for accurate prediction of caveability.

The BA5 panel retreat cave was developed partly in Tuffisitic Kimberlite Breccia and partly in Hypabyssal Kimberlite. Both of these ore types were overlain by the 75 metre thick gabbro sill. The undercut area needed to induce caving in all rock types had to be estimated. Given the history of caving in kimberlite above the sill and in the gabbro above the L1 stopes, it was estimated that, given the increased stress levels, an undercut area of 100 x 100 metres (H.R.= 25) would be sufficient to induce caving in all rock types. Provision was made to increase the area to 120 x 120 metres (H.R.= 30) if it became necessary. The actual area that had to be undercut to induce caving are given in Table 3.1.

In four separate mining blocks caving of the sill initiated at a hydraulic radius of 17 or less. As shown in Table 3.2 the Mining Rock Mass Rating value suggested that a larger hydraulic radius would be required to initiate caving. In three of the areas monitored the gabbro sill formed the crown pillar above open stopes. The stopes were 80 metres wide and of variable length. In all three cases caving initiated when an area of 80 x 60 metres had been undercut. At no stage was the ore in the stope in contact with the base of the crown pillar. Above the first open stope (SA3) the crown pillar caved to a depth of 18 metres overnight, depositing 55 000 tons of gabbro into the open stope below. Above the second open stope, 45 metres of gabbro caved over a weekend. Apart from the area undercut immediately prior to caving, no data was available for the third stope. Rod extensometers to measure movement and stressmeters to measure stress change were installed in the hanging wall of all three stopes but showed no indication of stress change or

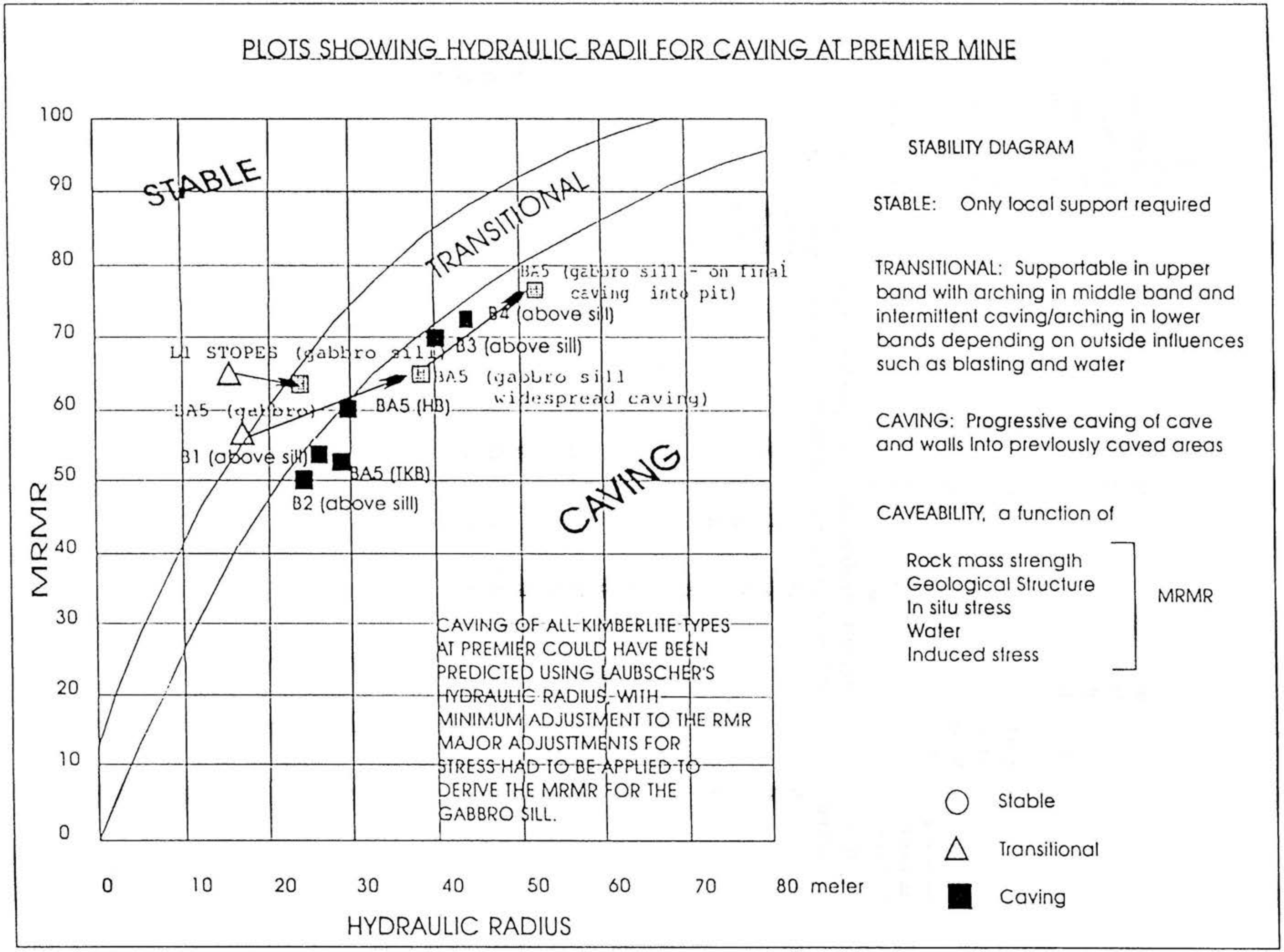
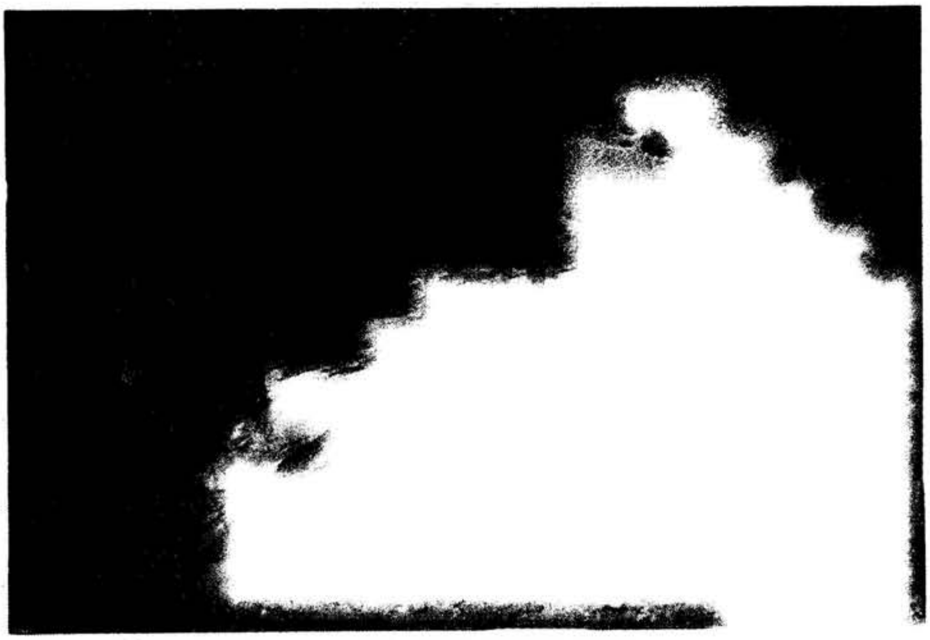



Figure 3.2. Mining Rock Mass Rating (MRMR) vs hydraulic radius for Premier caves.

movement prior to caving. It was possible to monitor the base of the crown pillar above one stope for several years using laser measurements. After the first major collapse, caving of the sill was joint controlled and intermittent, and could usually be correlated with the retreat of the stope below. Fallout was seldom to a depth of more than ten metres at a time. It took 2005 days for caving to propagate through the 75 metre thick sill to the open pit above, an average rate of caving of 37,4 millimetres per day. The last 20 metres of sill collapsed with a minor air blast.

Above the BA5 cave, once a hydraulic radius of 16 had been undercut, a 37 metre thickness of sill collapsed in the space of a few hours. At the time of collapse an airgap of 40 metres existed between the top of the draw column and the cave back. A further 7 metre thickness of sill collapsed during the next two weeks. Thereafter the sill remained stable for 4 months with no indication of caving whatsoever, as confirmed by careful monitoring. Monitoring from an observation point using laser measurements and core drilling showed that the area of sill that had caved was 14 300 square metres and roughly circular in shape. The calculated hydraulic radius was 34. The mass of gabbro involved in the collapse was calculated to be 1,5 million tons. At this stage the gabbro sill was in a transitional state between stable and caving. Stress measurements in the sill prior to caving gave a vertical stress of 4 MPa, a strike stress of 10 MPa and a dip stress of 17 MPa. The rock mass rating (RMR) of the gabbro sill was 55. The gabbro sill started caving at a hydraulic radius of 16 suggesting an Mining Rock Mass Rating of 42. A hydraulic radius in excess of 50 was needed before the sill finally caved into the overlying pit. In order to explain the caving process monitored in the sill and its sporadic nature, it is postulated that the competent sill behaved as a beam or plate. Joint dilation on horizontal joints in the tensile zone in the cave back area occurred and the two well developed sub-vertical joint sets allowed block fallout in the gabbro. Total horizontal stress in the gabbro sill remained unchanged but, as the sill caved the thickness of the gabbro "beam" diminished, stress levels per unit area increased and joint cohesion increased, firmly clamping the well defined blocks of gabbro. This situation temporarily halted the caving process until such time as the hydraulic radius was increased and the weight of undercut gabbro was sufficient to overcome joint cohesion on the weakest joint plane. Further caving occurred, beam thickness diminished, horizontal clamping forces per unit area increased and the cave back stabilised again. Back analysis of this data shows that the high horizontal stresses in the competent gabbro eventually had the effect of increasing the Mining Rock Mass Rating to 75. Photograph P.3.2., taken in February, 1994, shows the airgap that developed above the BA5 cave. The beam of sunlight shown in the photograph is about 30 metres wide. The size of the airgap measured 200 metres by 200 metres and sixty metres in height. Three weeks after the photograph was taken, an estimated 6 million tons of gabbro collapsed in the space of two weeks.

Observation and stress measurements in kimberlite pipes have consistently shown that there is negligible residual stress in a kimberlite pipe, that Tuffisitic Kimberlite Breccia is not capable of transmitting stress effectively and that stresses from the surrounding wallrock are not transmitted across pipe boundaries into the kimberlite. In contrast, the gabbro extends into the surrounding wallrock and stresses are markedly anisotropic, with a high 4:1 horizontal to vertical stress ratio. As the gabbro sill caves, the total horizontal stress transmitted through this rigid beam remains constant but over a diminishing area. Stresses normal to the near-vertical joint planes increase progressively and an increasing undercut areais needed to induce caving as caving of the sill progresses. Once 55 metres of the 75 metre thick had collapsed the horizontal to vertical stress ratio should have increased by a factor of 3,75 to give a horizontal to vertical



P.3.1. 170 metre wide arch in B4 cave prior to boundary weakening

P.3.2. Airgap of 200 metres by 200 metres by 60 metres in sill stress ratio of 15:1. In a weak rock this would have led to widespread shear failure in the cave back. In the competent gabbro sill, clamping forces across joints are increased and arching results. This suggests that more weight needs to be given to stress levels and joint condition rating in applying Laubscher's classification in extremely competent rock.

Experience at Premier proved that Laubscher's empirical correlation of Mining Rock Mass Rating against hydraulic radius provides the best estimate of the area that must be undercut to induce caving in all the rock types at Premier (Figure 3.2.). Jointing is the primary determinant of caveability and must be carefully assessed. Stress effects can inhibit caving, as in the gabbro sill, where high clamping forces in strong rock allowed a 200 metre arch to form and created a potentially dangerous airgap. If such a situation were to be encountered at greater depth where clamping forces would be even higher, unassisted caving would probably not be a viable mining option for Premier.

In the kimberlite, although jointing is the primary determinant of the undercut area, stress effects aid the caving process. Greater stress at increased depth will decrease the area that must be undercut to induce caving and enhance the caving process leading to finer fragmentation.

## 3.3. RATE OF CAVING

Caving of a rockmass results when large displacements, often accompanied by rock mass failure, start to occur. The mode of failure will be a function of rock strength, rock structure and stress levels. If stress levels are sufficiently great relative to the rock mass strength the rock can fail in compression. If the rock mass is well jointed and stress directions are such that shearing will occur along horizontal or sub-horizontal joint sets, the rockmass will fail in shear. The caving process will be assisted by near vertical joint sets which allow block fallout. If horizontal stress levels are low, the rock mass can fail in tension. Again the process will be assisted by the opening up of horizontal joints in the rock and vertical joints will aid block fallout. The rate of caving will change across lithological boundaries where rock mass strengths and/or joint characteristics vary. Boundary weakening will introduce additional planes of weakness and reduce horizontal stresses.

Observations regarding the mode of failure of crown pillars above large open stopes confirm that high stresses occur near the points of maximum curvature of the excavation. If the cave back is a well formed dome, stress concentrations occur above the dome and shear caving occurs. If the base of the crown pillar is flat, stress concentrations occur where the sidewall meets the hangingwall. High shear stresses are set up in this area and the crown pillar may fail catastrophically. This is directly analogous to the situation that can occur in a cave. As long as a well formed dome exists, shear caving occurs in the cave back. If a stable arch is allowed to form failure occurs at the edge of the excavation at the top of the arch abutment and subsidence caving, often accompanied by an air blast, occurs.

The caving process is dependent on the dynamic equilibrium that exists between the failure mode in the cave back and the broken rock in the ore pile. If the cave back is in contact with the ore pile, the broken rock acts as a rockfill and effectively inhibits caving. If the "support pressure" exerted by the ore column on the cave back is so great that failure is inhibited, caving will cease and only re-activate when ore is drawn on the extraction level. If a void exists between the ore pile and the cave back, the process is not inhibited and failure progresses unimpeded, often with damaging airblasts.

Constitutive and comminution characteristics of mined ore have been little studied due to the difficulty of observing the behaviour of ore in the draw column. The behaviour of rockfill used in the construction of dams has, however, been studied in some detail. Results show that the behaviour of granular material is affected by both compression and shearing. Particle crushing increases as the stress level increases. Three stages of particle comminution can be identified as the stress level increases. Particle crushing first begins at a stress level as low as 0,7 MPa in uniform rockfills containing large blocks. Particle shape, size and jointing play an important role at this stage. It can be shown mathematically that large particles are more susceptible than small particles to failure under compression. Particles are more susceptible to failure in tension than compression and particles have a high probability of failure at comparatively low levels of stress. Fragments with high aspect ratios break and jointed fragments fail along joints. At stress levels exceeding 3,5 MPa, particle crushing is advanced and dominates the behaviour of the material as blocks collapse to form more compact configurations. Particle failure occurs largely at the edges and corners of fragments at this stage. At stress levels in excess of 17,5 MPa structural breakdown and greatly increased compressibility occurs. The breakdown process at all stress levels in greatly intensified by relatively large shearing strains. Extensive interparticle crushing is achieved in the milling process on Premier Mine at a stress level of 5 MPa.

These studies provide a broad indication of the behaviour of broken rock under increasing pressure. The behaviour of the rock in the ore column is, however, complex. It has been shown that as the height to width ratio of an undercut area increases, stress levels on the extraction level increase but at a decreasing rate due largely to shear drag along the perimeter of the undercut. Internal arching and temporary abutments increase the complexity (Laubscher,1981).

The way in which the undercut is developed has an effect on the failure mode in the cave back. Shear caving occurs in the cave back as the result of failure of the rock in shear or tension (Laubscher, 1981). The undercut span at which shear caving occurs is a function of joint spacing and joint condition, as well as in situ stress level. Movement along joint planes as a result of shear stress lowers the cohesion of the joint and increases the likelihood that the joint will become a block boundary. As a result, shear caving produces smaller fragments than subsidence caving which occurs in the abutment zone as the undercut advances. Subsidence caving produces large, slabby fragments which are created in the abutment zone by the formation of extension fractures ahead of the abutment. Gravity plays a major role in subsidence caving in the cave back and failure is more likely to take place along well defined joints with a low cohesion. If the undercut moves slowly and the stress regime in the cave back is allowed to develop stress caving occurs. If the undercut advances rapidly, as a result of weak rock on the extraction level that has to be undercut to avoid damage as the undercut is run over it, subsidence caving rather than shear caving occurs. Rock mass classifications can be used to determine whether induced fractures are likely to form. This will affect the rate at which the undercut should be advanced and should also provide an indication as to whether shear or subsidence caving should occur. No correlations between rock mass ratings and these parameters have been defined in the literature.

A natural rate of caving of the ore body is quoted for some orebodies and figures of between 100 millimetres and 1200 millimetres per day are given as measured rates of caving. At Henderson the measured rate of caving in one area of the mine was 270 millimetres per day. The overall rate of caving as determined by the time at which caving broke through to surface was 700 millimetres per day (Brumleve & Maier, 1981). At Shabani 700 metres of caving occurred in 21 days, a rate of 33 metres a day (Taylor, 1980). A constant rate of caving implies that a stress regime develops in the cave back where the strength of the rock, either in shear, tension or compression, is exceeded and failure occurs. As a result of the changing stress patterns, caving is not arrested but continues at a constant rate throughout the block height.

Several authors refer to the extraction of a given volume or weight of ore resulting in a certain height of caving (Panek, 1981. Brumleve & Maier, 1981). This suggests a dynamic relationship between the uncaved ore and the ore in the draw column, implying that extraction of ore from the drawpoint creates a void which then allows the ore above to cave into it. The ore swells as it caves and caving is arrested as restraint from the ore in the draw column creates "support" pressure that inhibits the displacement of the uncaved ore. Caving then stops until more ore is drawn to provide a void.

Examples quoted in the literature suggest that mass caving can occur only if two conditions are satisfied:

* The hydraulic radius of the rock mass must be exceeded allowing the rock mass to cave freely.

* A sufficiently large void must exist to allow room for mass caving to occur. Bulking factors need not be large. At Shabani (Laubscher, 1981) a volume increase of only 6 percent associated with mass caving was measured. At San Manuel Mine, 1 ton of ore drawn resulted in 10 tons of ore caving implying a volume increase of only 10 percent (Panek, 1981). Volumes actually measured by Panek showed bulking of 26 percent and 13 percent. At Henderson, 1 metre of ore extracted resulted in 7 metres of caving giving a bulking factor of 14 percent (Brumleve & Maier, 1981). A bulking factor of 15 percent could allow mass caving though a column height of 115 metres into a 15 metre high undercut that had been pulled empty.

### 3.3.1. EXPERIENCE AT PREMIER MINE

#### Monitoring Methods

A total of 31 holes were drilled into the cave back over a period of three years to ensure the safety of personnel and monitor caving in the BA5 mining block.

A multiple point borehole extensometer was installed in several of the holes in order to measure the rate of caving and allow remote, electronic monitoring of the caving process. This monitoring was generally unsuccessful as a result of equipment failure or slight shear movement along joints that pinched the electric wires. Monitoring in open boreholes was more effective.

The rate of caving was measured in an open borehole by tying a steel rod to the end of a reel of trace-wire with thinner copper wire and lowering the trace-wire down the hole until the steel rod touched the top of the orepile. The trace-wire was then pulled up until the steel rod was drawn against the cave back. The installation was then anchored. Any further caving or large scale displacement of rock blocks broke the copper wire connection. This type of measurement was carried out on a weekly basis and provided good information as to the rate of caving and the size of void that existed between the cave back and the draw column.

**Rate of Caving**

The rate of caving was monitored to try to match the rate of production from a drawpoint with the rate of caving above the drawpoint and, by doing so, to ensure the stability of excavations above and adjacent to the area that had been undercut.

When an area of 14 000 square metres (hydraulic radius = 30) had been undercut, the rate of caving was measured at 266 millimetres a day in the Tuffisitic Kimberlite Breccia. This created 160 tons of ore over the area of influence of an individual drawpoint. The rate of caving accelerated to 2 000 millimetres per day as the undercut area increased to 20 000 square metres. When caving reached the base of the sill, caving ceased abruptly. The rate of caving measured from the time that the monitored area was undercut to the time when the sill was reached averaged 180 millimetres per day. This supplied 109 tons of ore to a drawpoint per day.

In the Hypabyssal Kimberlite, when an area of 14 000 square metres had been undercut, the rate of caving was only 60 millimetres per day, supplying 36 tons of ore over the area of an individual drawpoint. As the undercut area increased to 20 000 square metres, the rate of caving increased to 215 millimetres per day, supplying 130 tons of ore to drawpoints below.

Observation showed that, when monitoring started, a void of more than 20 metres existed between the ore column and cave back. The first signs of caving noted were borehole extensometer anchors becoming loose and able to move several tens of millimetres in the borehole. These anchor points had been static for several weeks. The freedom of movement of the anchor points was attributed to minor shear failure around the boreholes as the result of high horizontal stresses. The stresses were, however, not great enough to cause shear movement along the joints to the extent that boreholes closed.

Monitoring showed that caving was irregular and correlated well with increase in the size of the undercut. As much as six metres of kimberlite caved immediately after a ring blast on one occasion.

As a result of the restricted lift height and the large void that was allowed to form above the undercut level as a result of overdrawing, the draw column was never in contact with the cave back and free caving occurred until the base of the gabbro sill was reached.

When the cave back reached the base of the gabbro sill, caving stopped. Observation and monitoring showed that the caved area was steep sided and almost circular in plan. When observation was first possible (October 1992) it could be seen that a roughly circular area of sill 30 metres in diameter (area of 1000 square metres - hydraulic radius = 8) was exposed. Experience suggested that caving of the sill would not occur at this stage.

Observations in tunnels on the 500 metre and 538 metre level showed only minor shear movement along horizontal joint sets prior to caving progressing through the tunnels. As the undercut excavation was established and increased in size, stresses were re-oriented and increased around the undercut excavation. Joints and fractures define planes of weakness within the rock mass and stress changes usually initiate movement on these planes. Typically this leads to a lowering of joint cohesion. Shear movement on favourably oriented joint sets is important in lowering cohesion along joint planes that subsequently allow block fallout under the influence of gravity. This is the most important process in caving in kimberlite at Premier and, observation indicates, in most kimberlite caves.

Observation further showed that caving in the Tuffisitic Kimberlite Breccia was rapid and that almost vertical sidewalls had developed. Failure was largely by toppling of the sidewall into the void. Failure was sometimes on well-developed, aerially extensive joints and sometimes along fractures initiated in the Tuffisitic Kimberlite Breccia by induced stresses. Caving in the Hypabyssal Kimberlite was more gradual and an arch had formed in places. Caving was by block fallout from the cave back and largely joint controlled.

Draw control strategy in the early stages of caving was to shape the draw column by pulling hard at the western perimeter of the cave to create the correct angle of draw and a void into which the sill could cave when the gabbro began to fail. Drawing on the eastern margin was slow to ensure that an airgap did not form between the orepile and the uncaved ore in the cave back. On the northern and southern perimeters of the cave only the undercut ore was drawn to allow a clear undercut face for blasting purposes. The centre of the cave was not drawn to ensure that the monitored airgap did not increase in size.

Observation showed that the strategy of not drawing ore from the cave centre had resulted in uncaved ore resting on ore in the draw column. Caving was inhibited. Fortunately it had not resulted in damaging stress loads being transmitted to the extraction level. Finely blasted ore from the 12 metre high ring blast reported to the undercut level. The retreating undercut face was difficult to blast and load clean. The situation in the centre of the cave was now stable and static. Given the size of the airgap and the potential for a damaging air blast, it was decided that a slot should be drilled from within the sill through to the base of the sill where it formed the cave back. The slot would be blasted to break the high horizontal stresses in the sill to initiate the caving process.

The first phase of slot drilling in the gabbro sill was to drill at least seven 160 millimetre diameter holes around the proposed slot for monitoring purposes. Collapse of the sill started after three holes had been drilled and it was possible to monitor the collapse of the sill using these holes. Additional monitoring holes were subsequently drilled into the cave back and periphery of the cave to define the area that had caved as well as the thickness of the crown pillar as mining was still in progress above and in the vicinity of the caved area.

The core drilling showed that rock around the cave was little affected by the caving process until the hole approached to within 10 to 12 metres of the cave back. Complete loss of drilling water was then usually experienced, indicating open joints and/or fractures. Core barrels and drill rods were sometimes lost when movement occurred on joints. A vertical hole in the gabbro intersected an open, horizontal joint at a depth of 9 metres and experienced compete water loss, but the cave back was only intersected at a depth of 30 metres. A petroscope examination of the hole indicated joint opening on two prominent horizontal joints. No vertical holes drilled through the gabbro into the cave back were lost as a result of holes shearing closed on joints or fractures.

A further sill collapse occurred in November 1993. It involved 9 metres of sill at the apex of cave back and it is estimated that the collapse involved 0,5 million tons of gabbro. The collapse took the cave back to within 4 metres of the 445 metre level immediately below an extensive tunnel network that had been used for storage and parking. Observation of the rock mass behaviour in the area 4 metres above the cave back was possible for several months. Remarkably little damage occurred. Such damage as did occur was entirely due to movement along joints. Horizontal joints gradually opened and unpinned blocks fell from excavation sidewalls but there was no indication of failure in tension or shear through intact rock.

Drilling from the 390 metre level into the cave back showed the sill to be still remarkably intact, although some movement on pre-existing joints that intersected the development on the 390 metre level was noted. Water loss while drilling was experienced 15 metres from the cave back, indicating that the hole had intersected some open joints.

Observation at Premier suggested that slightly different modes of caving occurred in different rock types. Caving in the Tuffisitic Kimberlite Breccia was rapid. In the weak, poorly jointed tuffisitic kimberlite, movement along joints initiated block fallout a considerable distance form the cave back. This was accompanied by considerable tensile failure as well as some shear failure through intact rock.

In the more competent, well jointed, Hypabyssal Kimberlite, movement on joint planes was more restricted. Shear failure of intact rock only at points of maximum curvature of excavations whilst failure in tension was rare.

Caving in the gabbro sill was entirely episodic and joint controlled. No tensile or shear failure of intact rock was observed. Strain bursting, as rock bridges ruptured, was sometimes heard. The first indication of caving was minor shear movement and dilation along extensive, well developed, sub-horizontal joints, indicating that a tensile stress regime prevailed in the gabbro above the cave back.

Experience at Premier showed that it was essential to monitor the rate of caving of the various rock types and that caving rate should determine the rate of production. A high undercut and overdrawing of broken undercut ore allowed uncontrolled caving to propagate to the base of the sill once the hydraulic radius of the kimberlite was exceeded. This subsequently allowed a huge, potentially damaging, airgap to form. If there had not been a 70 metre thick layer of broken ore between the airgap and the production level, mining would have had to be suspended to ensure the safety of personnel.

The rate of caving was found to be a direct function of the area of undercut. The larger the undercut areathe greater the rate of caving when an airgap existed between the draw column and the cave back. When the draw column was in contact with the cave back, the rate of caving was determined by the rate of draw.

The rate and type of caving is a function of rock type. In the gabbro sill, only when a sufficient volume of rock had been undercut to induce failure on joints under the influence of gravity did caving of the sill occur. Mass, episodic caving occurred, involving large volumes of gabbro and potentially damaging airblasts. In the kimberlite, the rate of caving was more constant as failure in shear and tension assisted the caving process.

## 3.4. FRAGMENTATION

Determination of the primary fragmentation size is an important factor in selecting the mining method. For several decades, fine fragmentation was a prerequisite for cave mining. Such ore was found in the iron ore deposits of the Lake Superior area and in secondary copper oxide deposits. Advances in technology have allowed cave mining to be considered for more competent orebodies where once only mining methods such as sub-level caving, blast-hole open stoping or shrinkage stoping would have been used. Caving is the cheapest underground mass mining method and should be considered if fragmentation is acceptable. As the average size of fragment reporting to the drawpoint increases, costs rise and the production tempo from the drawpoints is adversely affected.

In block, panel or continuous caving systems, fragmentation affects

* Drawpoint spacing
* Draw control
* Drawpoint productivity
* Secondary blasting damage to drawpoints
* Secondary blasting/breaking costs
* Dilution entry into the ore column

The in situ rock mass is flawed by joints, fractures and other geological planes of weakness that determine the distribution of rock blocks in the unmined ore. Numerous attempts have been made to accurately determine in situ rock block distribution from core logging and mapping of exposures along tunnels and drifts underground and then to relate the original rock block distribution to the fragment size distribution that eventually reports to the drawpoint. The relationship is, however, by no means straightforward or easy to predict. The joint system and the attributes of the various joint sets such as continuity, orientation, strength and presence of water will be the primary determinant of the average fragment size distribution that eventually reports to the drawpoints. Joints are, however, not always continuous and joint strength can be such that the joint is stronger than the adjacent rock. RQD is a crude and inadequate measure of rock block size. The RQD/$J_n$ term in Barton's Q system is a measure of average rock block size, but it is the distribution of rock block size rather than average size that is important. Rocks larger than the drawpoint will present drilling problems, while rocks that cannot be moved by LHD's will determine the frequency of secondary blasting. This will impact on productivity.

Villaescusa, working at the Julius Kruttschnitt Mineral Research Centre (JKMRC), has developed a three dimensional model of rock jointing, a method of scanline mapping and computer software necessary to use the data so collected to predict the in situ rock block distribution. His method of scanline mapping is only slightly more complicated that normal line mapping.

Scanline mapping was used at El Teniente in the sub-level 6 area and has been used in mapping at Premier in Hypabyssal Kimberlite, norite and gabbro where it is believed that jointing will be the prime determinant of rock block size. Results are set out in Table 3.3. Results from Mount Isa, El Teniente and Koffiefontein have been included in the table for comparitive purposes. Mount Isa practices open stoping, while El Teniente Sub-6 block and Premier are mechanised caves. Koffiefontein practices a form of open benching, but caving has been considered as an option with increasing depth of mining.

**Table 3.3. Predicted average fragment dimensions**

| | ESTERHUIZEN | | | | VILLAESCUSA | |
|---|---|---|---|---|---|---|
| MINE | AVERAGE PRIMARY FRAGMENTS | AVERAGE SECONDARY FRAGMENTS | MAX SECONDARY FRAGMENTS | % SEC. LESS THAN 2 CUBIC METRES | AVERAGE RADIUS | AVERAGE VOLUME |
| Premier Norite | 0,82 | 0,47 | 10,7 | 62,54 | 0,445 | 0,088 |
| Premier Gabbro | 0,1 | 0,08 | 2,01 | 97,22 | 0.494 | 0,120 |
| Premier DP | 0,27 | 0,14 | 5,30 | 93,38 | 0,236 | 0,013 |
| Premier TKB | 0,29 | 0,12 | 6,31 | 86,93 | 0.232 | 0,012 |
| Koffiefontein | 0,06 | - | - | - | 0,250 | 0,015 |
| Mount Isa | 2,98 | 1,54 | 10631 | 34,67 | 0.354 | 0,045 |
| El Teniente | 0,01 | 0,015 | 5,191 | 95,39 | 0,267 | 0,055 |

Notes:

1. FIRST FOUR COLUMNS BASED ON WORK OF ESTERHUIZEN
   - 1.1. Average primary fragmentation size in cubic metres
   - 1.2. Average secondary fragmentation size in cubic metres
   - 1.3. Maximum size of fragment in secondary fragmentation
   - 1.4. Percentage of fragments less than 2 cubic metres in secondary fragmentation

2. LAST TWO COLUMNS BASED ON WORK OF VILLAESCUSA - GLOBAL JOINT SPACING
   - 2.1. Average radius of fragments in primary fragmentation
   - 2.2. Average primary fragmentation size in cubic metres (Villaescusa, 1991)

Fragmentation analysis using the methods described is a measure of most favourable fragmentation size distribution that could enter the draw column from the failure zone of the cave as the rock mass caves, at least in the initial stages of drawing. Experience has shown that prediction of fragmentation using this method results in a considerable underestimation of the fragmentation size distribution that reports to drawpoints.

Laubscher (unpublished notes, 1991) observes that caving results in primary fragmentation and rock blocks are formed in the failure zone of the advancing cave. The failure zone of the cave includes both the abutment zone and the cave back area. Secondary fragmentation is the reduction in size of the primary rock blocks by comminution in the draw column.

### 3.4.1. PRIMARY FRAGMENTATION

The controlling factors in primary fragmentation are the orientation, intensity, trace length, size and cohesion of the joints as these relate to the induced stresses in the cave back. The most commonly accepted model of joints is that they are elliptical or disc shaped discontinuities of varying diameter, location and orientation. Joints define the potential boundaries of rock blocks. Whether or not the joint will actually become a boundary of a rock block will depend on the stresses acting on the joint as well as the condition of the joint. There is a high probability that open joints will define rock block boundaries as caving occurs. Cemented joints, where the cement is stronger than the rock, have a low probability of becoming a rock block boundary. Blasting fractures are discontinuous and have little effect in defining rock block boundaries. The large scale expression of the joint (planar, irregular, wavy) as well as the small scale expression (rough, smooth, slickensided) together with the stresses acting on the joint plane will determine the cohesion on the joint.

Work by Panek (Panek, 1981) shows that extension fractures associated with the cave front can extend over tens of metres in weak rock (RMR 35). Observations made while mining remnants 115 metres above the extraction level against the country rock in a Kimberley mine confirm the presence of a zone of weak, poorly jointed rock with well defined extension fractures immediately adjacent to ore in the draw column. Where the rock is poorly jointed and incompetent, these extension fractures can have an important effect in determining rock block size. Typically, in this situation fragmentation is initially coarse as the fractures are sub-parallel and create large slabs of rock.

Observations show that stress levels play an important role in determining rock block size where rock is weak and poorly jointed. Determining the primary fragmentation size is, however, problematic. Observations show that joints plays an important role but tensile failure of the rock is also important. One possible method of determining rock block size in this situation is to numerically simulate the stresses that occur in the cave back and to use stress levels, in situ rock mass strength together with the Hoek and Brown failure criterion for damaged rock to determine the height to which failure will occur into the cave back. The degree of overstressing provides a measure of the fragmentation that will result. A problem with this approach is that it impossible to access the cave back to calibrate the model.

It is possible to assess primary fragmentation size with some exactitude in jointed rock if the following is taken into account:

> The stresses in the cave back will affect fragmentation. High horizontal stresses acting on predominantly vertical joints will inhibit caving and result in coarse fragmentation as only the weakest, favourably oriented joints will fail.
>
> Low angle joints in the same stress environment will fail in shear, resulting in finer fragmentation and more rapid caving.

Tensile stresses in the cave back will promote caving, but rock block size will be a function of joint spacing, orientation and joint condition, with failure occurring along favourably oriented joints and/or joints with low cohesion.

In the absence of well defined joints, the strength of the rock plays an important role in caving as the rock will fail in either compression or tension. Failure criteria rather than joint assessment will provide an indication of primary fragmentation.

### 3.4.2. SECONDARY FRAGMENTATION

Secondary fragmentation may be defined as the reduction in size of the primary rock blocks by comminution in the draw column. For such comminution to occur, stresses in the draw column must exceed the strength of the weakest portion of the rock block. If most of the rock blocks that report to the drawpoint are too large to pass through the drawpoint, or cannot be conveyed by the LHD bucket, or are larger than a third of the diameter of the ore pass, excessive blasting with resultant blast damage to support and production delays could make the cost of caving uneconomic.

The product of secondary fragmentation can be measured in the drawpoints once the cave is in operation. It is, however, important to predict fragmentation before the cave comes into operation as fragmentation size is one of the main criteria that must be defined before choosing a mass mining method. Drawpoint spacing, equipment size, tunnel size, secondary blasting procedures, rock breaker capacities, grizzly diameters, ore pass diameters and crushing capacities are directly related to fragmentation size distribution. The change in fragmentation size distribution with the tonnage drawn from a mining block should be considered as it can have a favourable effect on all the above items and increase the production potential of a mining block as a function of time. Drawpoints spaced at the correct interval for coarse fragmentation may be too far apart if comminution improves, allowing early ingress of waste dilution.

### 3.4.3. EXPERIENCE AT PREMIER MINE

Caving at Premier above the sill resulted in extremely coarse fragmentation and a secondary blasting efficiency of 250 gm per ton, almost as high as that required for primary breaking.

In the BA5, LHD's could remove rock fragments up to 2 cubic metres in size from the drawpoints but, in the initial design, no rockbreakers were installed on the extraction level and grizzly bars were spaced at 1 metre. Any rock larger than 1 metre along one side constituted "oversize" that had to be broken by drilling and blasting in the drawpoints. It was therefore important to determine the number of rocks that would report to the drawpoints in fragments larger than 1 cubic metre in size to determine secondary breaking requirements and have the required equipment and blasting procedures available to break these fragments so as to maintain the required production tempo from the block.

It was obvious to the author from both the literature survey and experience at Premier that accurate assessment of the fragmentation size distribution was not possible using existing geotechnical tools. Although both experience and the geotechnical assessment predicted that fragmentation at Premier would be coarse, it was impossible to predict type and frequency of hangups, or the equipment, time and labour that would be required to bring down the hangups. Costs and production tempo from the BA5 were therefore difficult to plan.

A simulation model would have to be developed and calibrated on the mine. The mine did not possess the expertise to develop such a model and commissioned the Mining Department at Pretoria University to write a programme that would allow simulation of the fragmentation size distribution that could be expected in drawpoints. Subsequent development of the program was funded by another mining group that had a similar requirement. Factors that were taken into consideration in the model were based on the experience of Dr. D.H Laubscher as well as the experience of other persons familiar with block caving in South Africa, Chile and North America and with whom Dr. Laubscher and Mr. Esterhuizen (author of the program) have exchanged ideas. The author was involved in defining the parameters that would allow accurate prediction of the fragmentation size distribution and developed both direct and indirect methods of measuring the fragmentation size distribution in the drawpoints to calibrate the model.

Based on observations by the author of the way in which hangups occur, this work has been extended and used to predict the type and frequency, and hence secondary blasting requirements in a mechanised cave mine experiencing coarse fragmentation (Esterhuizen et al., 1996).

Fragmentation is considered further in Chapter 8 of this thesis.

## 3.5. ROCK MASS RESPONSE AROUND THE UNDERCUT EXCAVATION

The rock mass response to a cave mining operation can be usefully considered in terms of three areas:

* the response immediately ahead of the undercut face on the drilling level where stresses are transitory and excavations are of a temporary nature

* the rock mass response on the extraction level where drawpoints must be maintained in extremely severe conditions as a result of stress changes, secondary blasting and LHD impacts

* the rock mass response above the undercut level and in the cave back

The rock mass response on the undercut level has been well studied. Relatively minor stress changes can force shear movement along joints, fractures, shear zones and major structures as much as 100 metres away from the actual abutment zone. There is usually a good correlation between rock mass rating and the distance to which rock damage can be measured away from the actual abutment zone (Cummings et al., 1984. Ferguson, 1977). Movement along major structures can result in large stress re-distribution and wedge failure.

The zone immediately adjacent to the undercut face is termed the abutment zone and both theory (Wagner, 1992. McKinnon, 1992. Esterhuizen, 1991.) and monitoring (Brumleve & Maier, 1981) show that stress levels in this zone are increased by a factor of 2 to 4 times the virgin stress as a result of the creation of the undercut. Confining stresses are low. Taken together, it usually results in failure of the rock mass in the abutment zone. Failure takes the form of shear failure on joints, as well as closely spaced, induced shear fractures through intact rock. The width of the abutment zone can vary from as little as three metres to as much as 15 metres, depending on rock strength and stress levels. Stress levels increase as the area of the undercut increases and are at their maximum immediately prior to continuous caving. Thereafter, stress levels remain constant and damage is constrained by the confining effect of the broken ore in the undercut excavation.

On the extraction, level the rock mass is subjected to much the same stresses as the undercut level as the abutment zone passes overhead. The rock mass response depends on the rock mass strength on the extraction level, as well as the amount of rock that has been mined out on the extraction level prior to the undercut being run overhead. Support effectiveness will also determine the rock mass response. Additional loading can occur on the extraction level as a result of incomplete undercutting that creates "stubs" or remnant pillars. Incorrect drawpoint spacing and/or draw control can result in pillars that create point loading. A slow moving undercut and large leads and lags between adjacent tunnels can all result in aggravated stress levels. Stress effects can result in rock mass damage which typically takes the form of shear movements along joints and fractures, especially in the blast damaged zone around excavations, shear failure through intact rock especially in bullnose areas, footwall heave and even cave "sit-downs".

The rock mass response above the extraction level has been little studied. Panek (Panek, 1981) reports displacements of 5 millimetres or more in monitoring holes drilled into the area around a cave at San Manuel (USA) which he interprets as tangential fracturing around the undercut excavation. Movement in drifts well above the extraction level was also monitored and attributed to the formation of the tangential fractures. It is theorised that these fractures extended over the cave back and contributed to the caving process. Stress caving and mass, or subsidence caving have been discussed (Heslop & Laubscher, 1981) as possible failure mechanisms in the cave back. Failure in tension and shear along joints and through intact rock, as well as failure in compression in weak rock have all been suggested as failure mechanisms in the cave back.

### 3.5.1. EXPERIENCE AT PREMIER MINE

Monitoring and observation of the behaviour of the rock mass around the undercut excavation have largely corroborated the principles of the rock mass response reported on the undercut and extraction levels from other cave mines. Extensive monitoring and good access have allowed a considerable insight into caving mechanisms in the cave back at Premier in the various rock types.

Experience at Premier showed that, although some parameters that influenced the rock mass response such as rock type and overall stress levels could not be controlled, many parameters that were vital to the successful implementation of the cave could be planned and implemented in such a way that rock mass and support damage was minimised. The literature survey defined many of the problems that could be expected, but an actual monitoring programme, planned and implemented by the author, was needed to quantify many of the mine design parameters that were important to the successful implementation of the cave. These included the distance to which the abutment stresses would be felt ahead of the undercut, the rate of advance of the undercut, the actual distance of leads and lags between adjacent tunnels that could be tolerated, the distance that the undercut needed to be beyond the drawbell before the drawbell could be opened, time lags that could be tolerated before compaction of undercut ore became a problem, timing of support installation and mining sequence. These parameters are a function of rock type and stress levels, and simple extrapolation from other caves or previous experience might not be adequate for mine planning.

The monitored rock mass response around the undercut excavation in the BA5 is discussed in detail in this thesis in the chapters that deal with development of the extraction level, the development of the undercut and the stresses that develop around the undercut excavation.

## 3.6. SUPPORT REQUIREMENTS IN A MECHANISED CAVE

Support in most cave mines is kept to a minimum on the undercut level where tunnels are sacrificial and of a temporary nature. On the extraction level, support is usually inadequate even though support specifications are greater than those suggested by most rock mass classification systems (Cummings et al, 1984). Most cave mines accept rehabilitation of support as part of the cave mining operation. The main reason for support problems is that installation of the production level achieves a high extraction ratio (typically as high as 50 percent). Poor blasting can increase the extraction ratio and damage the rock that remains in situ. The extraction level is then subjected to high and variable stress loads as the undercut is run overhead. These stresses are sufficient to force shear movement along joints and fractures to create stress induced shear fractures. The blast damaged zone around excavations can be extensive.

Many mechanised cave mines (Henderson, El Teniente, Salvador, Andina, Premier) develop extraction levels in competent rock. Active support in the form of grouted steel tendons and reinforced shotcrete is used in the initial stages of cave mining to harness the strength of the in situ rock for support purposes. Drawpoints that will be subjected to secondary blasting are often supported with steel reinforced concrete arches anchored into the surrounding rock. Lateral constraint in the form of steel cables and shotcrete or concrete linings is provided for bullnoses and camelbacks.

Experience has generally been that rigid linings such as concrete or shotcrete are damaged as the undercut abutment is run overhead. Hoop stresses lead to high thrusts developing within the rigid linings (Kirsten & Bartlett, 1992) and inclined stress fractures are common. Experiments and modelling have shown that a steel tendon typically needs a grouted embedment length of between 1 and 1,5 metres to withstand the forces imposed on the steel tendon during cave mining. If the embedment length is less than 1 metre the forces imposed on the installation will result in failure at the rock/grout, or steel tendon/grout interface. Movement on joints and induced shear fractures in the blast damaged zone around excavations leads to short embedment lengths on grouted steel tendons. High tensile stresses develop in the tendons and result in shearing at the rock/grout, or grout/steel interface and support becomes ineffective.

The process is illustrated in Figure 3.3. Variable stress levels in the rock around the tendons can also lead to the installed support being less effective than planned or calculated (Kaiser et al., 1992. Hyett et al., 1992).

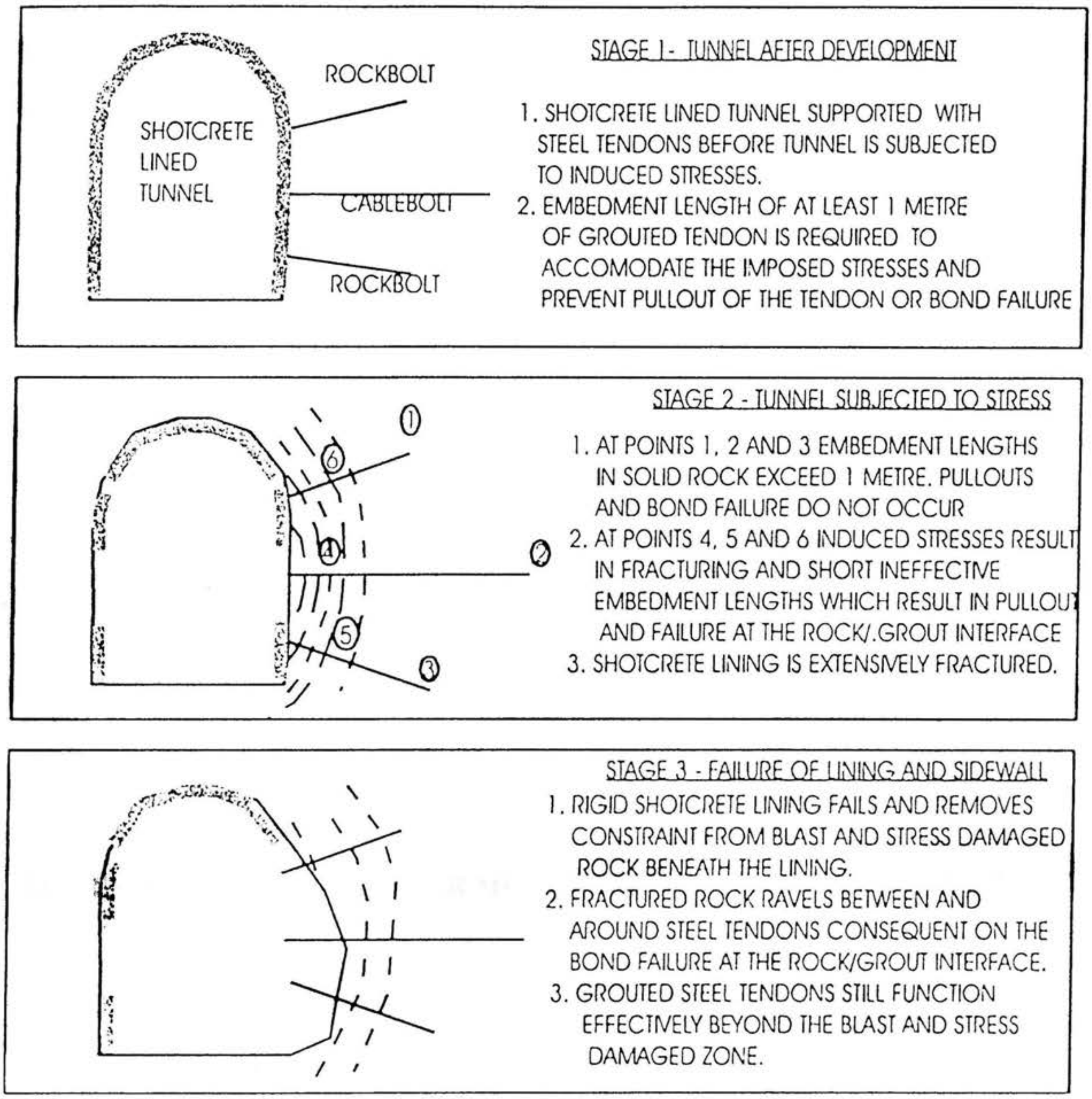
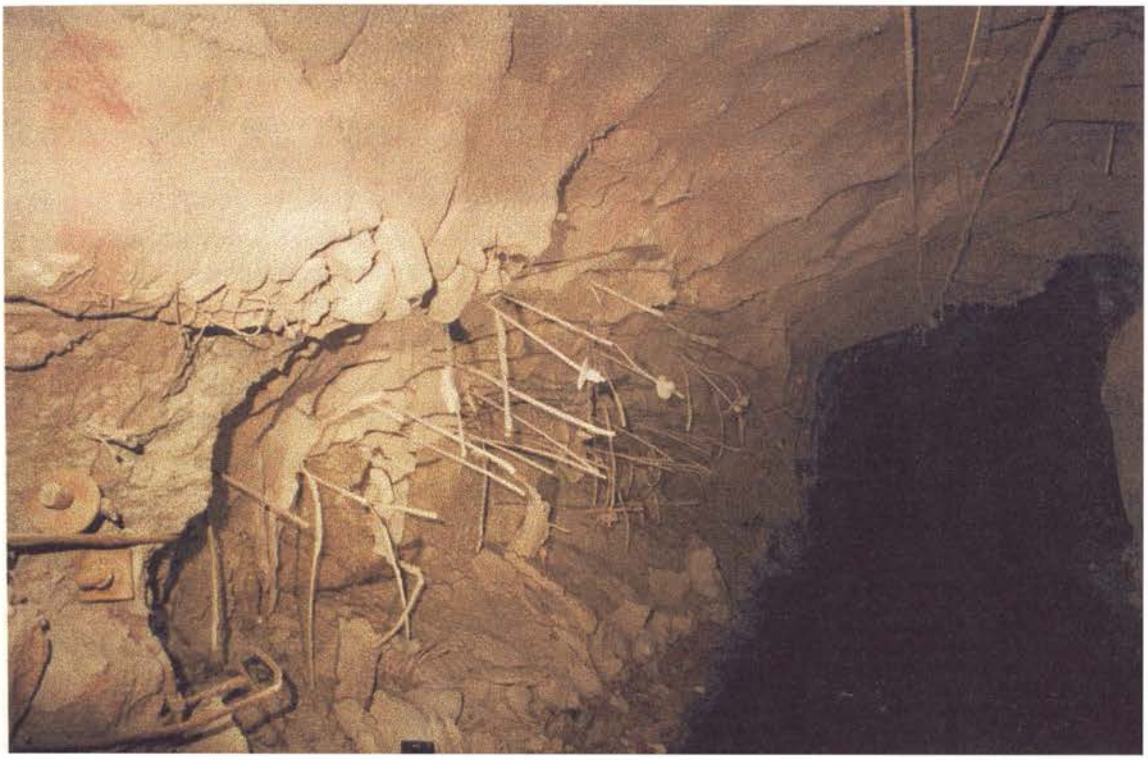



**Figure 3.3. Progressive Damage to supported tunnel.**

Once the rock and support has been damaged by the undercut stresses, it is then further damaged by further stress changes, secondary blasting and LHD impacts.

This experience has lead to a two phase approach to support advocated for cave mines (Cummings et al.,1984). Support to ensure the safety of personnel and excavations is initially installed and final support, especially rigid linings, are only installed after the undercut has passed over the area.



**P.3.3. Progressive damage to supported tunnel. 1,8m rock bolts and 6m cable anchors at 0,5 metre spacing cannot prevent ravelling after induced fracturing**

### 3.6.1. EXPERIENCE AT PREMIER MINE

Major problems were experienced with support at Premier (Bartlett, 1986, 1993) which corroborated the view of many cave mines that support recommendations derived from most rock mass classification systems are inadequate for cave mines that experience large, damaging stress changes (Cummings et al., 1984). A decision was made to carry out rigorous support design and install the required level of support to ensure that excavations remained stable and that rehabilitation support would not be routinely required during the life of the BA5 cave (Kirsten & Bartlett, 1992). Extensive research into the behaviour of shotcrete subject to large deformation was undertaken (Kirsten, 1992. Kirsten & Labrum, 1990) and the tensile stresses that develop in steel tendons around excavations subjected to the abutment stresses were calculated. The support system derived from calculation was then installed and a programme to monitor stress changes, displacements and support damage was instituted by the author to assess damage and improve support design.

The installed support has ensured that no drawpoints have been lost as a result of abutment stresses, but damage to the support and to the underlying rock has been extensive in some areas and a programme of support rehabilitation has had to be instituted. This takes the form of massive concrete linings as the rock around the drawpoints is often so extensively damaged that drilling becomes impractical and steel tendon reinforcement cannot be installed.

Support damage has been correlated with rock mass strength, stress levels, size of undercut, rate of undercut advance, leads and lags between adjacent tunnels, secondary blasting, LHD impacts and tons drawn. The distance to which support needs to be installed away from the undercut can be correlated with the rock mass rating.

The author had to determine whether an advance undercut would ensure that rock and support is not damaged to the extent that a process of expensive rehabilitation becomes a ongoing aspect of the mining operation. Reinforced concrete was required for brow support in weak kimberlite, but if the lining failed, it became problematic to repair. This dilemma had to be resolved. Ways of adapting and using Laubscher's rock mass classification system to anticipate the level of stress induced damaged that could be expected in areas affected by abutment stresses had to be developed. Premier's Geotechnical Department's support experience and design philosophy is set out in subsequent chapters and appendices.

## 3.7. DRAW ANALYSIS AND CONTROL

Draw control on cave mines tends to differ widely as a result of the geological characteristics of the orebody. Draw control is also the least studied aspect of cave mining. Much of the current theory behind the flow of material in the draw column is based on analogue models where material ranging from sand to bricks has been used to simulate material flow (Heslop & Laubscher, 1981. McNeary, 1993). Underground observations have generally confirmed theory based on sandbox models. Where markers have been placed in rock that has subsequently caved and the markers moved through the draw column, the markers have often shown considerably more horizontal movement than anticipated. It is important to note that it is impractical to simulate the fragmentation size distribution found in some cave mines. The effects of stress can also not be replicated in sand box models. At Henderson Mine drawpoints experiencing extreme convergence during extraction are sometimes filled with concrete and adjacent drawpoints pulled until the stress effects are no longer evident. Calculations show that all the ore in the area has been extracted, including that attributed to the closed drawpoint (W.Reich - personal communication). In theory, the loss of a drawpoint should lead to ore loss as drawzones no longer overlap.

No numerical models exist that will allow simulation of the flow of material in the draw column.

Some empirical rules have been developed to avoid premature waste dilution and stress problems. These are that adjacent drawpoints should be pulled by at least a third of the amount at which the most heavily pulled drawpoint is pulled. For example, if 200 tons is pulled from one drawpoint, all the surrounding drawpoints should have at least 66 tons drawn during the same period to avoid isolated draw conditions and early waste ingress.

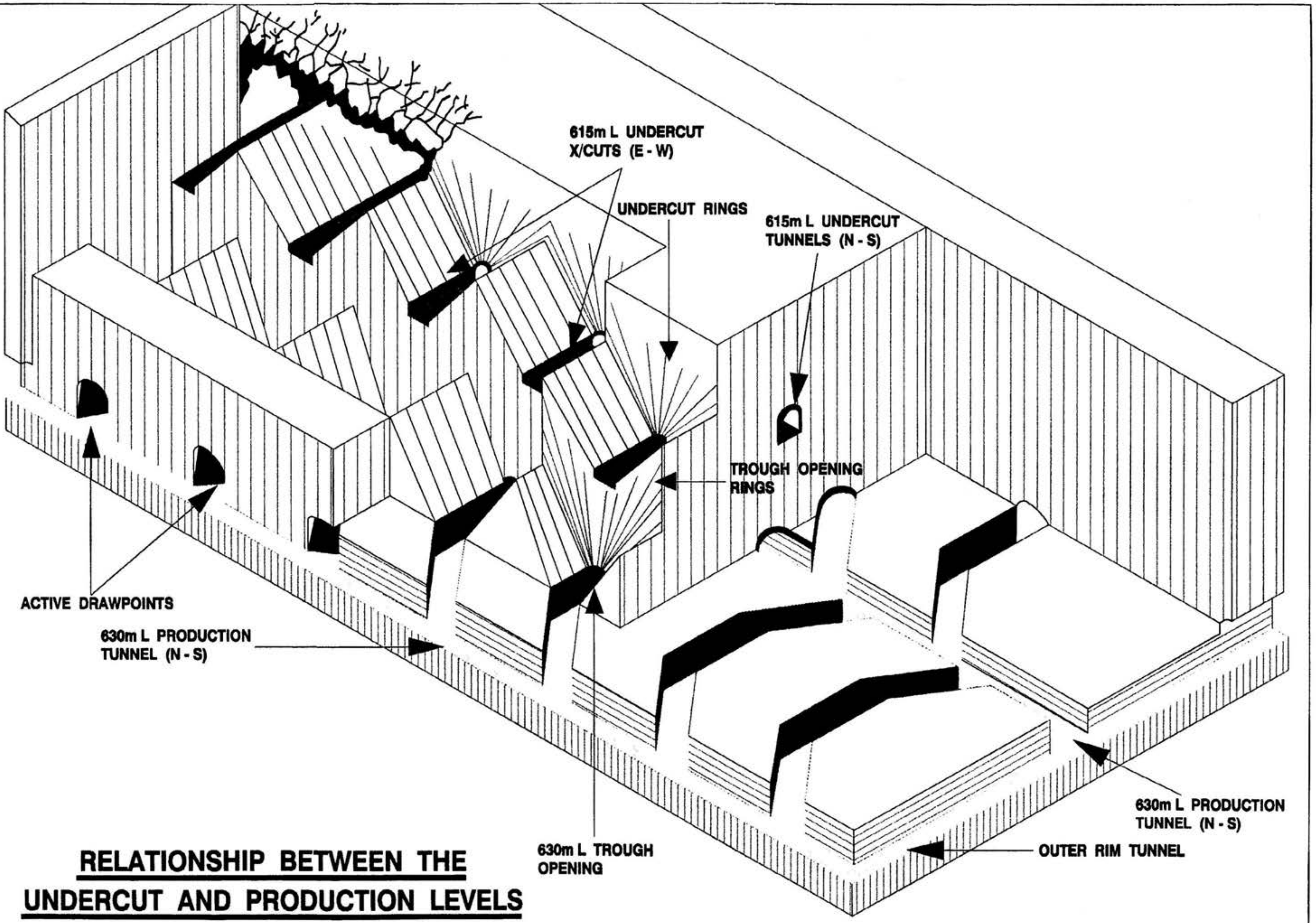
In several caves that have experienced coarse fragmentation, a "rock-the-block" method of mining has been used to set up shearing stresses in the draw column. In Kimberley, "strip mining" is practised to allow stresses to build up in areas that are not drawn for several months. As stresses approach critical levels in terms of concrete lining damage, the "strip" is moved and the area that had not been mined for several months is brought back into production. In this way, an active strip is constantly moved across the mining block.

In caves where grizzlies and/or slusher layouts are used, drawpoints are usually closer together than the maximum suggested by theory. Development of most mechanised cave layouts results in a system of complex pillars made up of minor and major apices on the extraction level. Resultant extraction ratios can be as high as 50 percent and pillars fail as the undercut is run overhead. In mechanised caves, there are major advantages to be gained in terms of structural stability and development cost by placing drawpoints as far apart as possible. If drawzone spacings are too far apart, drawzones can fail to interact and as much as 50 percent of the ore can be lost. Compact pillars can lead to stress problems. Influenced by these latter considerations, few attempts have, until recently, been made to space drawpoints at distances of more than 15 metres apart.

### 3.7.1. EXPERIENCE AT PREMIER MINE

At Premier, the importance of careful draw control below the sill was always appreciated. It was anticipated that fragmentation of the gabbro sill would be finer than in the kimberlite and that fine fragments of gabbro would move through the draw column quickly. Moreover, some mixing of gabbro and kimberlite would inevitably occur as drawing progressed. The gabbro is dense and creates problems in the recovery section of the treatment plant. For these reasons, it was anticipated that at least 15 percent of the in situ ore would be lost as a result of contamination. Microwave beacons were installed in each drawpoint, together with receiver and data storage units on each production LHD operating in the cave area. This allowed accurate information on tons drawn from each drawpoint to be available at all times to ensure that correct draw control procedures were followed.

The BA5 ore block was overlain by gabbro which could easily be distinguished from kimberlite. It was, moreover, important to monitor the gabbro waste as this high density rock adversely affected the diamond recovery process and had to be excluded from headfeed to the diamond recovery plant. Careful waste monitoring was instituted and good information was available from the LHD monitoring system that allowed the author to investigate several aspects of material flow and draw control. Aspects that were investigated included whether estimated reserves were better correlated with drawpoints or drawbells, the effect of minor and major apices on ore flow, the migration of ore towards areas of high draw, and the percentage of ore that could be drawn before gabbro waste started to report at drawpoints.




**RELATIONSHIP BETWEEN THE UNDERCUT AND PRODUCTION LEVELS**

# CHAPTER 4

# MECHANISED CAVING LAYOUTS

**Statement:**

This chapter reviews the geotechnical, practical and financial considerations of the author following on the literature review, geotechnical assessment and visits to other mechanised cave mines undertaken by Premier Mine personnel, prior to final design of the BA5 mechanised cave layout.

At the time Henderson Mine, using the offset herringbone layout, was operating drawpoints spaced at 12 metre centres and El Teniente, Andina and Salvador, using the El Teniente layout, were operating drawpoints spaced at 15 metre centres. The cost and stability advantages of a wide drawpoint spacing for Premier were obvious. Fragmentation size distribution was known to be coarse, but the fear that too widely spaced drawpoints would result in drawzones failing to interact, resulting in a loss of ore reserves and stress problems, constrained any increase in drawpoint spacing beyond what had been proven on other mines. The Premier geotechnical assessment had identified layout stability as a risk, but it was assumed that a support system could be designed that would ensure stability. Stress modelling, commissioned by the author and Andina personnel to assess the structural stability of several possible mechanised cave layouts, was undertaken at the University of Pretoria. The stress modelling, together with practicalities on other mines of developing and operating within these layouts, is reviewed in this chapter.

Experience in the BA5 mining block highlighted inadequacies and shortcomings in the Premier mine design process. These were quantified and the mine design process improved. The way in which the process was used to plan future caves is detailed in the thesis.

## 4.1. INTRODUCTION

The layout of a mechanised cave mining operation must take the following into account:

* The interaction of drawzones.
* The structural strength of the layout.
* The ease with which the planned layout can be developed.
* The interaction between the undercut and production level.
* The ease with which production LHD's can operate within the layout.
* The siting of service excavations.
* The cost of developing and operating within the layout.

## 4.2. STRUCTURAL STRENGTH OF LAYOUTS

The structural strength of the production level layout is a function of such natural factors as the rock mass competence and the prevailing regional stress field. The design engineer has little control over these, but must be aware of their potential effect. Far greater control can be exercised over excavation ratio, layout design, drilling and blasting practices and mining sequence (Cummings et al., 1984). Ideally the spacing between excavation openings should be such that the core of the confined major apex is sufficient to carry the imposed load. Drawpoint spacing, however, is dictated by fragmentation which might mean that openings are too close together and the plastically deforming envelopes around excavations overlap (Brumleve, 1987). Layout and support design can both help to compensate. Most cave "sitdowns" are the result of tunnel sidewall failure and back analysis of the failure mode often suggests "pillar" failure (Stevens et al., 1987). These "pillars" are complex, three-dimensional structures made up of major and minor apices of various shapes, depending on the mechanised cave layout used. Formulation of a universal pillar failure criteria for mechanised cave mine layouts is therefore difficult.

The importance of correct drawzone spacing is considered in detail in Chapter 9. Table 4.1. however, considers the theoretical benefits that can be gained by changing the drawpoint spacing and the extraction ratio. In a typical mechanised cave layout such as used at Premier, the planned extraction ratio on the production level, with drawpoints spaced at 15 metre centres, is 43 percent. The planned extraction ratio on the undercut level is 20 percent. Each drawpoint in this 15 metres x 15 metres layout would have an associated 26 metres of development on the production level and 15 metres on the undercut level. The combined development and support cost for each metre of development is approximately R6000. Forty five drawpoints can be developed per hectare at a cost of 11,07 million rand. A total of 540 drawpoints could be developed in the 12 hectare area of the BA5 using this layout. The undercut and production level development required would be 22 140 metres at an overall cost of 133 million rand. If drawpoint spacing could be increased to 18 metres x 15 metres, only 444 drawpoints would be needed. Each drawpoint would have an associated 46 metres of development. Development and costs for both the production and undercut levels would reduce to 20 424 metres and 122 million rand respectively, a saving of 8,3 percent. Realistically, as the extraction ratio reduces, support costs could probably be reduced.

**Table 4.1. Development cost and extraction ratios for various drawpoint spacings.**

| Spacing (metres) | Area of drawpoint (square metres) | Number of drawpoints per hectare | Associated development (metres) | Extraction ratio (percent) | Stability index | Cost per hectare (millions of rands) |
|---|---|---|---|---|---|---|
| 12 x 12 | 144 | 70 | 36 | 63 | 1,58 | 15,12 |
| 15 x 15 | 225 | 45 | 41 | 43 | 2,28 | 11,07 |
| 18 x 15 | 270 | 37 | 46 | 39 | 2,54 | 10,21 |

Figure 4.1. is a graph showing that development associated with each drawpoint increases linearly with drawpoint spacing by a factor of 2.6 and that the development and support cost per hectare decrease exponentially as a function of drawpoint spacing. Development is in linear metres. Assumptions are that the same level of support per metre would be installed for all layouts and that all sections of the layout will increase proportionately. For example, if drawpoint spacing increases from 10 to 15 metres, drawbell length will increase from 10 to 15

metres. If drawpoint spacing along the trough line is increased to 18 metres, the ratio of metres to drawbell length becomes 2.4. This ratio is graphed for comparison. The graph shows that the cost per hectare of support and development would decrease from 11,07 million rand to 10,12 million rand if the drawbell spacing was increased from 15 metres to 18 metres. Figure 4.2. graphs the increase in the safety factor with a decrease in extraction ratio. Strength and load on the Y-axis are in kPa. The safety factor has been calculated simplistically using pillar attribution theory, and merely serves to highlight the potential benefit that can be gained from increasing the drawpoint spacing. It emphasises that the determination of a failure criteria and accurate safety factor for major and minor apices for various mechanised cave layouts developed in rock with different Rock Quality Indices would be a valuable design tool for cave mining.

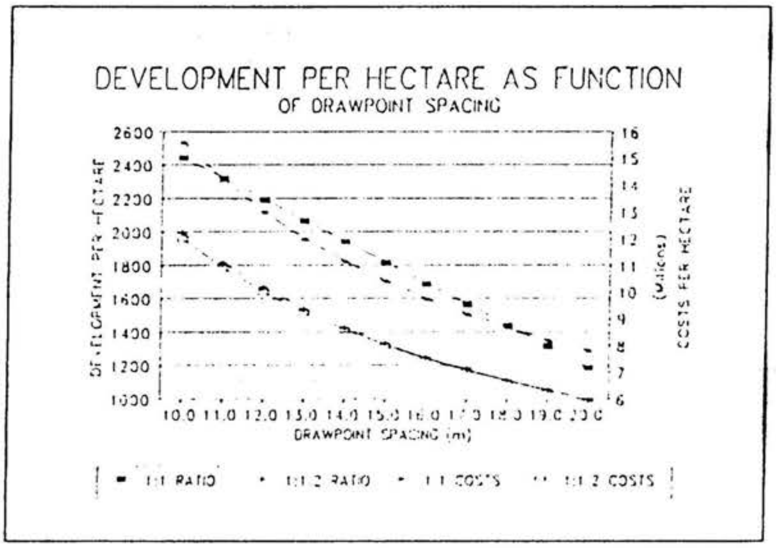



Figure 4.1. Development and Support Costs as a function of Drawzone Spacing (development in linear metres)

A problem that can arise with increased drawpoint spacing is that the number of drawpoints developed within the area of the cave might not be able to yield the required production. The literature review showed that the production tempo from a drawpoint should match the rate of caving of the orebody. At Premier, an initial rate of caving of 80 millimetres per day, rising to 160 millimetres per day (50 tons per drawpoint rising to 100 tons per drawpoint per day) was assumed. Figure 4.3. shows that tons per day increases exponentially as a function of drawpoint spacing. A drawpoint with an area of influence of 15 metres by 18 metres in an area experiencing a rate of caving of 200 millimetres per day (1:1,2 200mm) would yield 148 tons per day. A drawpoint with the same area of influence in an area caving at the rate 300 millimetres per day (1:1,2 300mm) would be expected to yield 225 tons per day.

In the initial stages of caving, fragmentation is coarse and production is largely a function of secondary blasting efficiency. Widely spaced drawpoints could, therefore, constrain the production potential of a mining block. If fragmentation remains coarse, secondary blasting remains an important part of the production process and too few drawpoints could limit production from a cave for the life of the mining block. If comminution in the draw column is effective, production from a drawpoint becomes a function of the rate of caving of the orebody, until such time as caving has progressed through to surface. Other factors such as tramming distance, LHD availability and orepass availability then constrain production, rather than rate of caving. Effective comminution was expected in the BA5. Figure 4.3. was therefore considered useful in determining the production potential of a drawpoint and of the BA5 mining block. If the increased drawpoint spacing resulted in loss of ore reserve and stress related problems, it would render the mining method uneconomic.

**Individual cones** leave the greatest amount of intact rock on the production level, i.e. have the lowest extraction ratio and should be considered where excavation stability problems are anticipated. Individual cones can pose problems of access to hangups and fragmentation size needs to be determined before implementation. El Salvador in Chile is a mechanised cave mining operation that has used individual cones in a well fragmented, copper oxide deposit. Numerous problems with high, inaccessible hangups were experienced. At Premier, single sided drawbells developed at the orebody margins where drawbell diameter is often only 9 x 15 metres, result in more hangups than in conventional double sided drawbells. This indicates that individual cones would create problems at Premier.

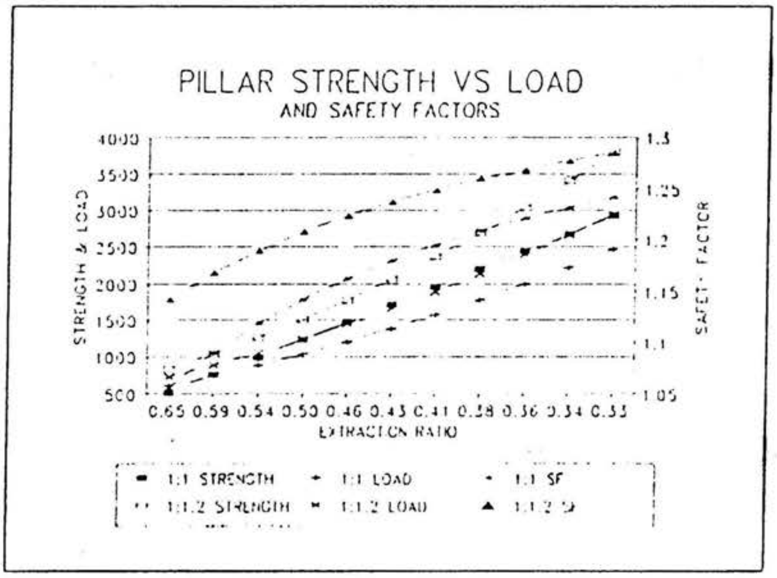



Figure 4.2 Pillar Strength vs Load and Safety Factors (strength and load on Y- axis in kPa.)

**Continuous troughs** have the highest extraction ratio and can be implemented where excavation stability is not a problem. Simulation (Esterhuizen, 1991) has shown that areas of tension develop in the drawpoint brows in this mining layout. The brows must therefore either be in competent rock or adequately supported. In a continuous trough layout, undercut drilling can be done from the production level. This leads to a considerable saving in development costs as no additional access or ground handling facility is required on a separate undercut level. At Premier, a continuous trough layout was implemented above the sill in a grizzly mining operation and produced for many years with very few problems. Stress levels were, however, low and the extraction level was sited in competent gabbro and metamorphosed kimberlite.

Several different drawbell layouts are in operation in mechanised caves worldwide. These include El Teniente, Andina, El Salvador, Bell, Henderson and Premier. Premier undertook simulation of three proposed layouts during the planning stage of the first mechanised cave implemented at the mine, using elastic boundary element modelling. The stress distribution on the production level of an offset herringbone, a herringbone and a continuous trough layout were modelled (Esterhuizen, 1987). At a later stage, further modelling using an elastic boundary element model was undertaken in conjunction with Andina, who were planning their dropdown into primary ore (Esterhuizen, 1991). The layouts modelled were all based on production drifts located 30 metres apart with drawbells at 15 metre centres along the drifts. The undercut was modelled at 18 metres above the floor of the production level. The layouts modelled are illustrated in Figure 7.10. From the figure it can be seen that each layout gives rise to a different stress distribution. The stress distributions resultant on the layout can enhance or oppose the regional stress field which should be measured and carefully considered when designing and orienting the layout.

Some important conclusions from the modelling are (Esterhuizen G.S. 1991):

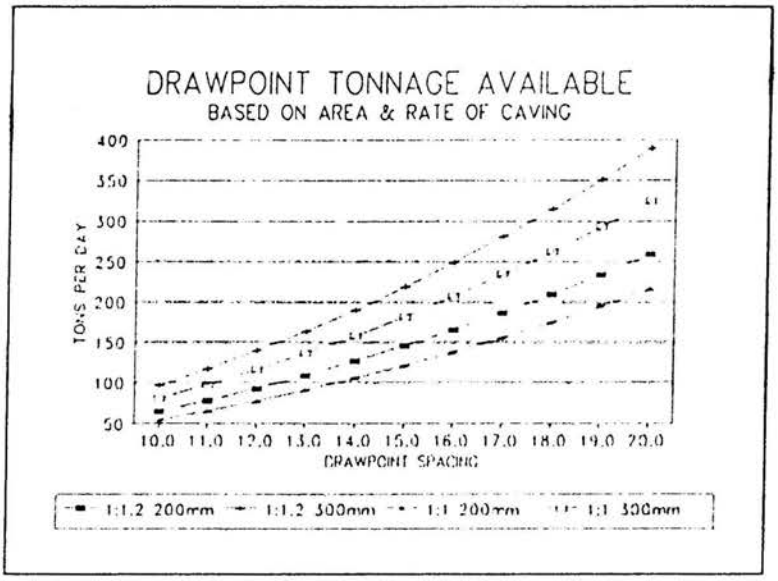



**Figure 4.3. Tons Available from Drawpoints with Different Areas of Influence and Rates of Caving**

* There is little difference in stability and induced stress levels between the layouts.
* The orientation of the layout within the regional stress field is important.
* In terms of overall stability the offset and El Teniente layouts are the most stable.
* The roof, floor and sidewalls of the El Teniente layout are the most stable.
* The drawpoint brow is most stable in the offset layout.
* The major apex of the El Teniente layout is less stable than for the other layouts.
* The minor apex of the El Teniente layout is the most stable.

The modelling was undertaken to compare the structural strength of several layouts, but provides additional information pertinent to developing and supporting the chosen layout:

* The obtuse and acute corners at the juncture of the drawpoint crosscuts and the production tunnels (termed, respectively, camelbacks and bullnoses at Premier) are subjected to high stresses in the sidewall and hangingwall which need to be carefully developed and well supported by providing lateral constraint to confine these areas.
* The drawpoint brows are subject to high stresses. They need to be well constructed and supported if drawpoint stability is to be maintained and wear minimised.
* Footwalls are subject to high stresses. If the footwall rock is weak, the footwall will need to be supported.
* The minor apices play an important role in ensuring that tunnel sidewalls remain stable. The drilling and blasting of drawbells should be done carefully so as to avoid damage to minor apices.
* The rock strength, modulus of elasticity of the rock mass, and stress directions are important parameters in the modelling and should be determined as accurately as possible. Monitoring of the rock mass shows that the modulus of elasticity of the rock is affected by the advancing cave front and that stress levels are variable. Rock strength can be considerably lower than that measured in the laboratory as a result of blast and stress induced damage.

Modelling of the stress distribution around a tunnel on the production level during the tunnel development, drawbell development, retreat of undercut and cave exhausted stages was undertaken using the FLAC finite difference programme. The simulation was used as the basis of support design at Premier. FLAC is a two dimensional programme and any cave layout is a complex three dimensional design. Further simulation was therefore undertaken using the three dimensional 3DEC programme to investigate stress distribution in and around the production level (McKinnon, 1992).

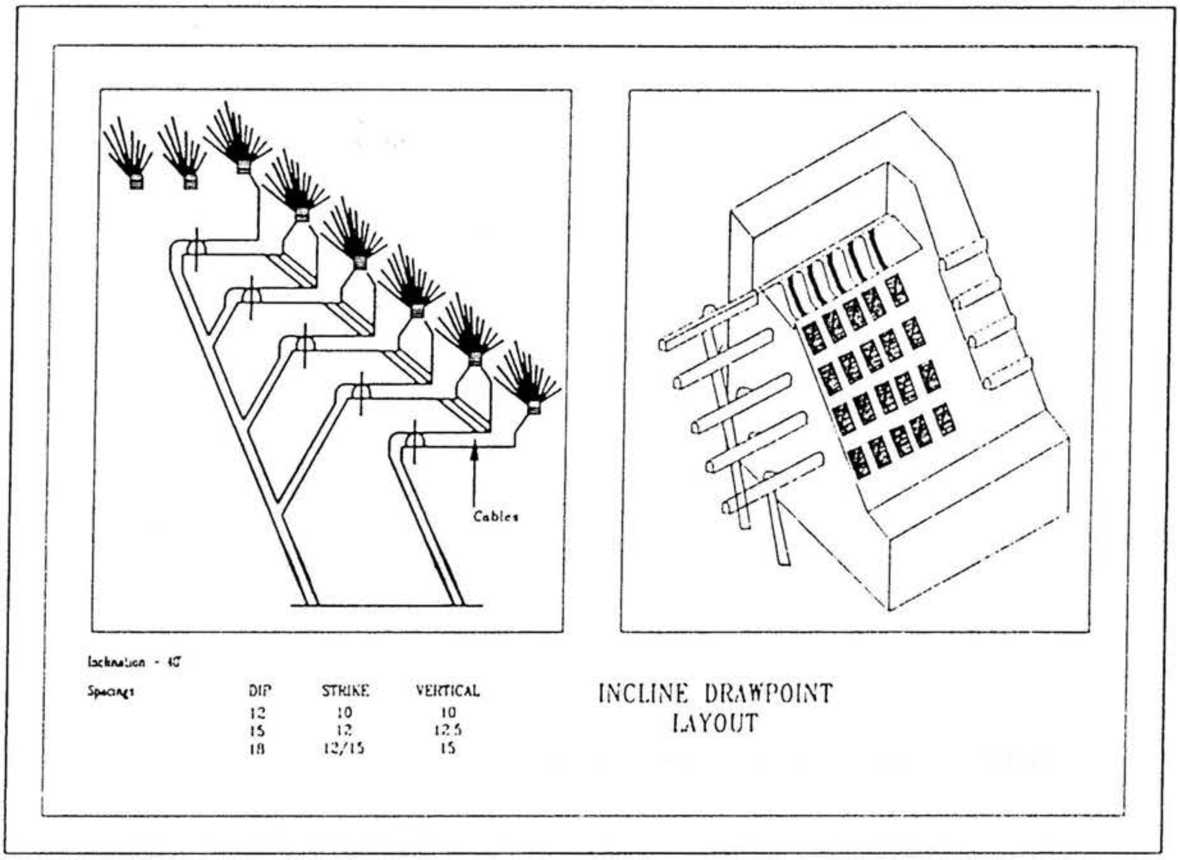



**Figure 4.4. False Footwall Layout (Laubscher, 1994)**

The false footwall layout for mechanised cave mining has been successfully implemented in asbestos mines in Zimbabwe (Heslop & Laubscher, 1981) and at Cassiar in Canada (Carew, 1992). A requirement for the layout to be implemented is a steeply-dipping, massive orebody. A typical layout for this method of cave mining is shown in Figure 4.4. Two inclined mechanised cave layouts are in use worldwide (Laubscher, 1994).

## 4.3. DEVELOPMENT OF LAYOUTS

Eight horizontal mechanised cave mining layouts have been designed and implemented (Laubscher, 1993). All layouts consist of parallel production tunnels with a maximum spacing of 30 metres along which LHD's tram ore to the transfer system. Crosscuts off the production tunnels have a maximum spacing of 15 metres and allow LHD's access to cones, drawbells or continuous troughs that gather ore from the cave above. The spacing of the drawbells, continuous troughs or individual cones are a function of fragmentation size. The crosscuts from the production tunnels to the drawpoints usually approach the drawbells, continuous troughs or individual cones obliquely to allow the LHD as much loading space as possible, as well as to accommodate brow wear and retreat.

Layouts planned for optimum production conditions can create problems during development. In panel retreat or mass caving, layouts must be developed and supported as part of an ongoing operation. If the production level is developed before undercutting starts and there is no ore handling facility on the undercut level, a typical development sequence is that three of four production tunnels are developed together with crosscuts to drawbells, continuous troughs or individual cones to facilitate handling of broken ore. Drawbells, troughs or cones are then developed. This is followed by undercutting.

The herringbone, offset herringbone or Henderson "z" layouts all create difficulties during development if mechanised drill rigs are used as turns are tight and drill rounds may need to be shortened to drill and blast the layout. A plan view of the various layouts discussed here is illustrated in Figure 7.11. Support needs to be installed immediately behind the advancing face in many rock types (Hoek & Brown, 1980). Weak rock allows the development of extension blast fractures a short distance behind the face (Guest, 1985). These factors dictate that development must be carefully done and development and support sequenced to allow successful implementation of the layouts. Simulation has shown the acutely angled "bullnoses" move into tension as soon as they are created. They can fail if not supported expeditiously. Failure of the bullnose can create an unduly large span in the adjacent tunnel. This in turn can result in hangingwall collapse. When the layout has been developed and supported, travelling access through the awkwardly shaped drawbell excavations with LHD's and auxiliary vehicles is difficult.

The "El Teniente" layout is easier to develop as all tunnels are straight. It makes development using mechanised drill rigs straightforward. Access for LHD's and auxiliary vehicles through crosscuts is easy. The support of the acutely angled "bullnoses" remains a problem. The drilling pattern originally used to develop the drawbell in the "El Teniente" layout is complex. The drill pattern has subsequently been modified by both El Teniente and Henderson mine, the latter having adopted the "El Teniente" layout on the 8800 level.

The "El Teniente" layout avoids the awkward geometry of the offset herringbone layout and creates fewer problems at the development stage of the extraction level. Bullnoses still have to be supported immediately after development. Trough cutting was facilitated in El Teniente by taking a steep, inverted-V blast cut and then doming the drawbell to its final dimension. The method is quick and easy but has the disadvantage that considerable damage is done to the surrounding rock by the blasting operations (G. Chitombo - personal communication). This method of drawbell development is not practical if advance undercutting is used.

A problem that affects development in all cave mining operations is that tunnels, crosscuts and service excavations must be developed and supported before the undercut starts to affect them. This is especially so if the excavation is developed before the undercut passes overhead. Stresses and strains associated with a limited undercut area are small. As the area of the undercut enlarges, induced stresses increase and damage to inadequately supported excavations increases in severity and is felt at greater distances from the undercut abutment. The extent of damage and distance to which the stress raising effects of the undercut are felt is largely a function of rock mass competence. In the near field of the abutment zone, extensive rock mass damage can occur up to 30 metres ahead of the undercut face (Cummings et al., 1984. Ferguson, 1977). This means that, as the area of undercut increases, the radius within which support and development must be completed increases, until caving starts to occur. At Premier, a planning constrain was that a zone 30 metres wide would be fully supported before the undercut was run overhead. This placed an increasingly high demand on support and development resources.

**Development of the Undercut**

The undercutting operation has a pronounced bearing on the effectiveness of the cave operation in terms of the damage to tunnels and tunnel support on the production level (Cummings et al., 1984). There are three approaches to the timing of the undercut relative to development of the extraction level. In post undercutting, the extraction level including drawbells is developed prior to the undercut being run over the area. A typical extraction ratio on the production level for both the "El Teniente" and offset herringbone layouts is 43 percent. An advance undercut is usually developed above a partially developed extraction level. The extraction ratio is of the order of 20 percent. If a pre-undercut is installed, no prior development exists on the extraction level. These methods of undercutting are discussed further in Chapter 5.

On most cave mines, drawbell development is, at least partially, accomplished by drilling downwards from the undercut level. In both an advance and pre-undercut, the drawbell must be developed upwards from the extraction level into the broken ore of the overlying undercut. This means that a new way of drawbell development has to be designed if an advance or pre-undercut is planned.

## 4.4. EASE OF OPERATION WITHIN THE LAYOUT

When development is complete and caving has initiated, access into the cave area is along service and production tunnels. The layout should be such that there is unhindered tramming from drawpoints to orepasses for production LHD's. Tramming distances should be minimised. Auxiliary vehicles must be routed around production areas.

The offset herringbone and herringbone layouts are ideal for electric LHD operation. The offset herringbone layout gives the most room for LHD's to manoeuvre. This is important if retreating drawpoint brows are a problem. With the herringbone layout, retreating brows can soon lead to problems of LHD loading access. The "El Teniente" layout is not suitable for electric LHD's and an area that allows the LHD to turn around must be incorporated into the layout as the LHD would have to go into the production tunnel bucket leading or bucket following depending on which side of the tunnel must be loaded. LHD's have considerable room to move and the layout is suitable if retreating brows are a problem.

Ventilation is a basic consideration in any mining operation. In most kimberlite mines, free water enters the montmorillonite clay lattice structure and causes large volume and stress changes which result in a dramatic reduction in rock mass strength. The use of water for dust suppression therefore poses a problem. This means that the LHD bucket, whether empty or full, should be "downwind" of the LHD driver at all times to prevent the driver travelling in dust laden air.

## 4.5. SITING OF SERVICE EXCAVATIONS

Service excavations include service and ventilation tunnels and ramps that provide access to the production tunnels as well as shafts, orepasses, workshops and stores.

Careful geotechnical assessment will indicate the level of stress change that can be expected resulting from undercutting, as well as the distance to which stress raising effects and excavation damage can be expected away from the undercut. This information can be used to locate excavations and, if necessary, design support of shafts, workshops, rampways and stores. Adequate support can usually prevent damage resulting from movement on joints in the far field of the abutment zone. Movement on major structures, such as faults, can lead to major instability. It is often difficult to support excavations that must serve for long periods in the near field of the abutment zone. Instability may extend to 50 metres from the actual abutment.

The siting of access and ventilation tunnels, as well as orepasses, is more difficult as they must often be developed in areas affected by the cave. Ideally, orepasses should be developed as close to drawpoints as possible to minimise travelling distances for LHD's. The orepass is then subjected to high and variable abutment stresses. This often means that orepasses must be lined or otherwise supported. Service and ventilation tunnels must be adequately supported to ensure that they remain serviceable during their required life under adverse and changing stress conditions.

## 4.6. OREPASS DESIGN

Important aspects of orepass design include:

* Rock mass competence of rock in which the orepass is sited.
* Fragmentation size and characteristics of ore produced by cave mining.
* Siting of orepasses relative to drawpoints.
* Orepass layout.
* Inclination of orepasses.

The competence of the rock mass in which the orepass is sited will have a major influence on maximum orepass size and will also determine the level of support, if any, that will be necessary to maintain the orepass in a serviceable condition during its planned life. The orepasses below the sill at Premier are sited in well jointed norite. Joint condition, joint frequency, where the rock is traversed by shear zones, and water play an important role in determining the stability of the orepass. Detailed work done at Premier as the result of the collapse of 6 metre diameter raisebored passes intended for ventilation purposes has resulted in guidelines being established for the size of hole that can safely be raisebored in rock of varying competence (M<sup>c</sup>Cracken & Stacey, 1988). Several orepasses in the norite collapsed shortly after being commissioned. The integrity of the orepass can be maintained by lining the pass with 1 metre thick, strong concrete. This was expensive and time consuming, and led to alternative methods of pass support being sought. The norite was mapped in detail to determine joint spacing and the information was used to apply probability concepts to determine support design in terms of bolt spacing and support (Stacey & Bartlett, 1990). This design concept has been used in the support of several orepasses with varying degrees of success. The probability approach can be improved by using in situ rock block size determination methods (Villaescusa, 1991).

The fragmentation size and characteristics of ore produced by the cave will determine the size of the orepass. Two types of hangups have been identified in pass systems (Hambley, 1987). Interlocking arches form as the result of large sized boulders that wedge together to form an obstruction. Interlocking arches also form where passes undergo an abrupt change in pass geometry. The probability of forming an interlocking arch depends on the percentage of large fragments handled, the size of the fragments relative to the orepass and outlet diameters, on the shape of the fragments and on the velocity profile across the flowing ore. The probabilistic approach is, however, poorly developed and empirical rules based on the ratio of particle size to pass diameter are used in practice. These rules are simply stated in the table below.

**Table 4.2. Interlocking Arch Formation**

| Ratio of Orepass Dimension to Particle Dimension | Relative Frequency of Interlocking |
|---|---|
| D/d >5 | Very low, almost certain flow |
| 5> D/d >3 | Often, flow uncertain |
| D/d <3 | Very high, little chance of flow |

**D = ore pass diameter**
**d = particle dimension**

After Hambley et al 1983.

The particle size distribution that will report to the drawpoint can be calculated or measured. This information can be used to design the orepass system so that the formation of interlocking arches is minimised. If the suggested orepass size is too large, the size of particle that is placed in the orepass can be restricted by a grizzly at the top of the pass. Oversize material can be reduced in size by secondary blasting or by impact hammer.

Cohesive arches form as a result of sticky, fine particles adhering to each other. Coarse particles resist motion as a result of interparticle friction. Fine particles exhibit cohesive resistance in addition to friction. This is made worse if water is present. The minimum orepass dimension to prevent a cohesive arch can be determined from the formula:

$D>(2k/g)(1+1/r)(1+\sin\phi)$ (eqn 4.1. Hambley et al., 1983)

D = orepass dimension
k = cohesion of fines (psf)
g = density of fines (pcf)
r = length/width ratio of opening
$\phi$ = angle of internal friction of fines (degrees).

Orepasses and transfer systems need to be designed to avoid hangups in as far as is practical.

**The siting of orepasses relative to drawpoints.**

The shorter the distance that the LHD has to tram between the orepass and nearby drawpoints the greater will be the tonnage that the LHD can transfer to the pass. Constraints are that passes usually need to be developed in competent rock. It may require that orepass be moved beyond the orebody boundary. Cave horizons are sited at progressively greater depths within the ore deposit. This could mean that the transfer system is sited in a mining block that is being prepared for the next dropdown and will be lost. Numerical modelling at both Premier and El Teniente has shown that stress levels can change in magnitude and direction as a result of the cave mining process. These stress changes can have an important effect in determining orepass support design and need to be considered.

**The design of orepasses is important.**

Experience and simulation have shown that a productivity increase of up to 30 percent can be achieved by creating a surge capacity of as little as 100 tons above a rockbreaker. This ensures that the rockbreaker continues to operate when LHD's are not loading into the pass and that LHD's can operate when the rockbreaker is out of action. Changes in geometry of the orepass can result in hangups. Orepasses at angles of less than 70 degrees can result in hangups. Chutes into passes on multiple levels need to be carefully considered and engineered. Dust can be a major problem and chutes into the orepass at lower levels may often be blocked if the orepass has limited capacity. The rock mass must be evaluated to ensure that the orepass will function for its planned life without major scaling and instability. Support of the orepass, using steel reinforcement, is possible if the pass is sited in competent but jointed rock. In incompetent rock, pass linings should be considered.

## 4.7. CONCLUSIONS

The geotechnical assessment, experience gained from other mines, engineering judgement and simulation of the layout was used to design the mining layout of the BA5. The mining layout that was planned is set out below. Hard experience, gained subsequently by the author as a result of monitoring and planned observations during the development and operation of the BA5 cave, showed how planning, development and operation of a mechanised cave could be improved. This is detailed in subsequent chapters in the thesis and has been used to improve the planning and development of future caves at Premier, as well as other caves with which the author has been associated both within the De Beers group and other mines.

1. The geotechnical assessment of the ore deposit will determine the geometry of the deposit, the probable fragmentation size distribution and the rock mass competence.

   In the BA5, the geotechnical assessment defined an extensive orebody suitable for caving with near-vertical sidewalls. Fragmentation was predicted to be coarse and the stability of the extraction was predicted to be a risk.

2. Geotechnical assessment can be used to determine the layout and method of ore extraction from the base of the cave and siting of service excavations.

   An offset herringbone layout with drawpoints spaced at 15 metres x 15 metres was chosen for the BA5 cave. Deciding factors were ventilation requirements and the decision to use electric LHD's as electric power is comparatively cheap in South Africa. Service excavations were sited at least 30 metres away from the cave.

3. Numerical modelling can be used to determine the stress levels that will result from the undercutting process.

   Numerical modelling at Premier was used to calculate expected stresses and assess the relative stability of the "El Teniente" and offset herringbone, and herringbone layouts within the regional stress field. This modelling showed relatively little difference in induced stress levels between the layouts, although areas of stress concentration were different.

4. Induced stress levels can be used together with failure criteria to determine the support system. Support costs can be lowered by decreasing the tunnel size and implementing quality control on development drilling and blasting as well as support.

   Experience and numerical modelling, together with Hoek and Brown failure criteria, were used by the author to design a support system for the BA5 for the various rock types. The experience gained from the literature, where it was stated that most support design in cave mines proves inadequate, was heeded. A system to monitor support effectiveness and improve support design was planned and installed by the Geotechnical Department.

5. The geometry of the undercut has a major bearing on the magnitude of the induced stresses. The size of the undercut excavation as well, as leads and lags between tunnels, impact on the level of induced stress.

   It was originally planned by the author to advance the BA5 undercut as a straight face. The maximum leads and lags that could be tolerated were only determined by experience. No quantitative guidelines could be gained from the literature or numerical modelling. Experience from other mines showed that widely varying leads and lags could be tolerated, depending on rock mass competence, speed of undercut advance and general mining competence.

6. Ideally, the undercut should be advanced at a rate that is slow enough to allow shearing stresses to develop in the cave back, but fast enough to avoid major damage to the rock mass and support on the extraction level. The geotechnical assessment will provide guidelines as to whether the rate of undercut advance needs to be rapid or not, but actual rate of advance must be determined by experience. If the geotechnical assessment indicates that induced stress levels could damage apices and drawpoint brows, pre- or advance undercutting should be considered.

The minimum speed of undercut advance that could be tolerated in the BA5 was gained from experience. The cave was originally planned as a post undercut, as detailed numerical modelling and limited experience suggested that a support system could be installed that would prevent rock mass and support damage as the undercut passed overhead.

7. If support costs are high, the cost of support should be weighed against some form of advance or pre-undercutting. It was assumed, after detailed support design had been undertaken, that support would not be damaged as the undercut passed overhead. Support rehabilitation was planned to be minimal. Support damage as the undercut was run overhead, the subsequent cost of support rehabilitation and the time taken to repair drawpoints which impacted on production from the cave, obliged the author to recommend a change in mining sequence to advance undercutting.

# CHAPTER 5

# EXTRACTION LEVEL

**Statement:**

In a mechanised cave the extraction level exists to allow LHD's to collect ore from drawpoints and move this ore to the passes. Depending on the mining sequence the rock forming the extraction level is subjected to a number of stress cycles which can result in rock mass and support damage. Rock and support can be further eroded by LHD impacts, secondary blasting and the drawing of ore. The structural strength of the layout as well as its suitability for efficient transport of ore is therefore important.

The geotechnical assessment had highlighted the risk of instability on the extraction level in the BA5. This risk was emphasised by the literature review which indicated that rock mass classification often recommended a level of support that was inadequate to ensure stability in cave mines. Inadequate support resulted in continuous, costly support rehabilitation which adversely affected the production. This chapter therefore focusses largely on the support design process used in the BA5, monitoring of the interaction between installation of the extraction level and the installed support, and ways in which this monitoring was used to improve support design and the overall mining operation.

The author undertook detailed support design for all excavations in the BA5 in an attempt to ensure that support would be adequate. It involved an investigation as to how and why various support elements function, the effect of the five phases of stress change associated with the cave on the support system and how support generally fails. The investigation involved testwork on the mine undertaken by consultants and the author, a detailed literature review and the application of previous experience gained on the mine by the author in adverse mining conditions in the failed open stopes below the sill.

It was considered important by the author to establish criteria that would objectively measure support effectiveness. A system that would measure parameters that determine support effectiveness was then designed by the author. This involved stress and displacement monitoring and regular planned observations by the author and other geotechnical staff of rock and support damage on the extraction level using a simple system developed by the author to quantify the damage. Damage levels were then correlated with mining operations.

The results of monitoring were used by the author to assess the effect of the five stress cycles on the support elements installed at Premier and to judge the effectiveness of the support system. The assessment was ultimately used to justify a change in the mining sequence and design the Code of Practice used at Premier to determine, plan and install effective support in all rock types.

## 5.1. INTRODUCTION

The extraction level in a cave mining operation is usually developed in ore that is relatively weak, or cave mining would not be considered as a mining option. As much as 50 percent of the rock on the level is extracted to create the drawbells, production tunnels and crosscuts needed for mining. The level is subjected to high and variable stress changes as drawbells are developed, as the abutment stresses associated with the advancing undercut move overhead and ore is drawn from the drawpoints. Damaged rock is subjected to further erosion by mining activities such as LHD impacts and secondary blasting. Layout design, mining sequence and support must ensure the stability of excavations in this harsh environment and that production is maintained at the required tempo.

This demands careful planning of the layout and support system on the level.

## 5.2. INTENDED FUNCTION OF THE EXTRACTION LEVEL

The intended function of the extraction level is twofold:

* To allow the efficient collection of ore from the base of the caved ore column that is being mined and to transport the ore to a system of orepasses for further treatment.

* In mechanised caves, the extraction level is used as an intermediate step in breaking the ore down to a size that can be accommodated in the rest of the extraction process.

It means that drawbells on the extraction level must be spaced in such a way that all the overlying ore is efficiently extracted. Ore must then be transported to the orepasses using optimum-sized LHD's with as short a tramming distance as possible. Constraints are imposed by fragmentation and a need to maintain the structural integrity of excavations needed for mining on the extraction level.

Experience and material flow theory suggest that the maximum practical drawpoint spacing is of the order of 15 metres, although at least two mechanised cave mines are using or have planned a slightly greater spacing. The layout is designed to allow access to the drawpoints using LHD's. Rock on the extraction level will be subjected to stress changes associated with tunnel and drawbell development as well as the undercut being run overhead and subsequent mining of ore.

A decision must be made as to whether post, advance or pre-undercutting will be implemented in terms of the magnitude of anticipated stress changes and rock quality on the extraction level. The level of stress change associated with the undercut being run overhead is 2 to 4 times the virgin stress but this can be increased and stress effects aggravated by the geometry of the undercut, a slow rate of undercut advance, and remnant pillars as a result of incomplete undercutting.

Support on the undercut level must be designed to ensure excavation stability during tunnel and drawbell development, and as the undercut is run overhead. Thereafter support must withstand erosion by mining activities.

Ventilation, roadway construction, pass design and the stability of surrounding service excavations are all important aspects in the design of the extraction level.

## 5.3. THE DEVELOPMENT OF EXTRACTION LEVEL

Experience above the sill had shown that fragmentation would be coarse with 30 percent of fragments reporting to drawpoints having at least 1 side longer than 3 metres. This suggested that the widest possible drawpoint spacing should be used. The widest successful drawpoint spacing that was being used at that time was 15 metres. The decision was made to implement an offset herringbone layout with drawpoints at 15 metre centres and production tunnels at 30 metre centres on the extraction level.

The decision to implement post undercutting in the BA5 was made largely on the basis of experience.

* Few problems had been experienced with the implementation of post undercutting above the sill although it was appreciated that the extraction level was sited in competent gabbro and stress levels were lower.

* The failure of the open stope mining method on the eastern side of the mine had resulted in four 40 metre wide, 90 metre high pillars being left behind. A decision was made to cave one of these using a pre-undercut. The pre-undercut was successfully blasted, but problems in developing the extraction level 15 metres below as a result of the high stress levels (30 MPa) that had developed in the pillar, resulted in the ore on the undercut level compacting. This resulted in the pillar "sitting-down". It emphasised one of several problems that can arise with a pre-undercut and fear of similar compaction was evoked to influence the choice of undercut methods used in the BA5.

* The Hypabyssal Kimberlite in the centre of the BA5 was known to be more competent than the Tuffisitic Kimberlite Breccia on the eastern side of the mine and stress levels were not expected to be as high as in the open stope pillars.

In all cave mines visited by the author, drawbell development involved some blasting from the undercut level. In the failed open stopes on the eastern side of the mine at Premier, no access on the undercut level was available to allow any blasting of the drawbells that were eventually installed below these open stopes. Methods of drawbell development used on other mines were therefore impractical in this situation. In the BA5, drawbells were developed by drilling a 12 metre long, 660 millimetre diameter blindhole bores from the centre of the drawbell tunnel on the extraction level to the undercut level. The blindhole bores were raised from below using a blind hole borer. Several sixty four millimetre diameter holes were then drilled close to, and parallel to, the bore. The blindhole bore acted as a free breaking face and the drawbell was cut by blasting first a single hole into the bore and slowly increasing the size of the raise to form a slot. This was then advanced in both directions towards the planned drawpoint brow by ring drilling. The sides of the drawbell were angled at 65 degrees and drawpoint brows were vertical. Drawbell development generally took 10 shifts to complete. Although drawpoint spacing was planned at 15 metres, the rate of brow wear was anticipated to be rapid and the original drawbell was only developed to 13 metres to allow for brow wear during the life of the drawpoint.

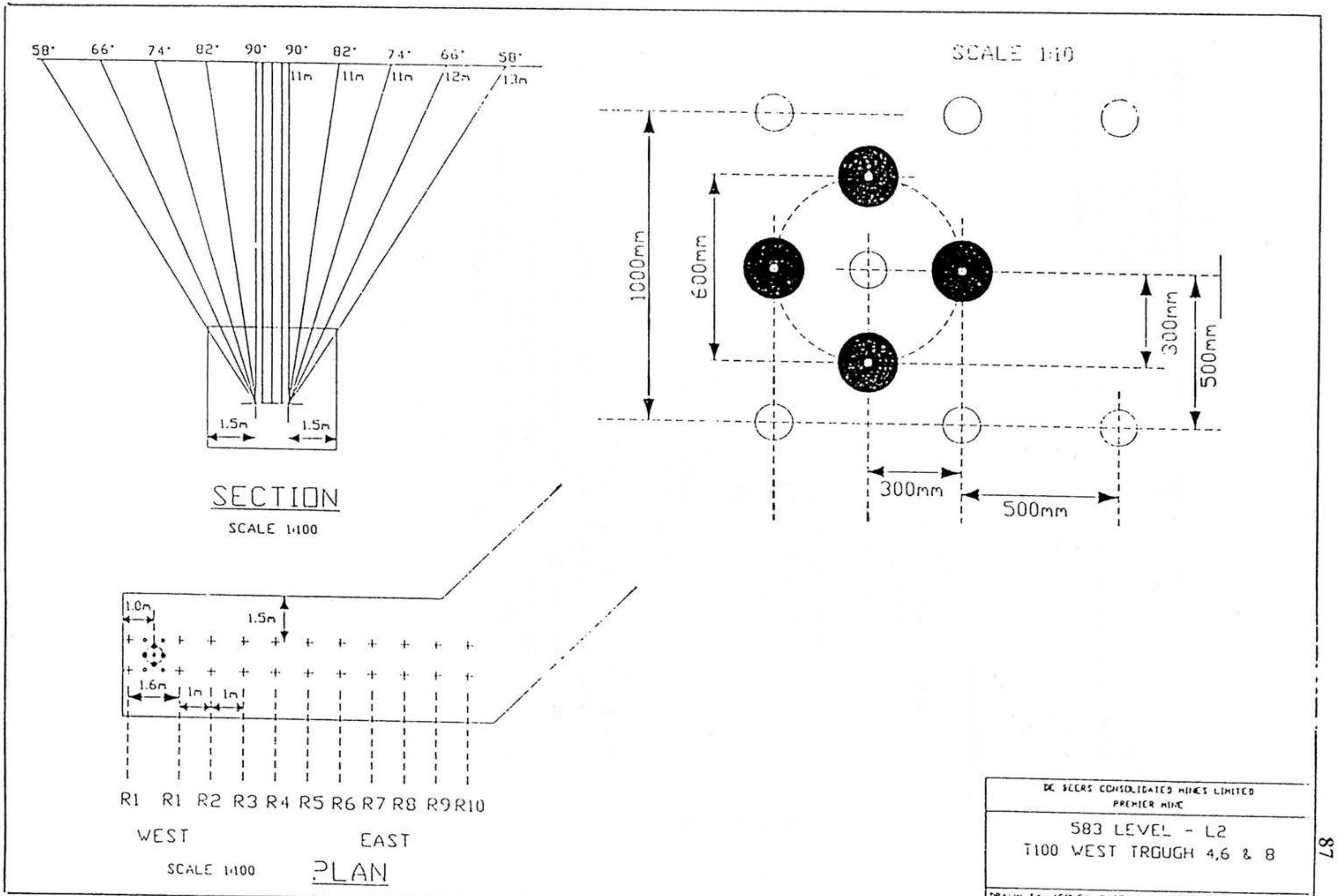



Figure 5.1. Development of Drawbells

In several instances, poor communication led to drawbells only 9,5 metres long being developed. The layout for drawbell development is given in Figure 5.1. The fact that Premier had been forced to develop a method of drawbell development that involved access from the extraction level only proved useful when a change in mining sequence was considered. El Teniente has since adopted and improved this method of drawbell development in an area where they have used an advance undercut.

## 5.4. SUPPORT

The geotechnical assessment had shown that weak rock on the extraction level could result in excavation instability when the rock was subjected to high abutment loads during undercutting. The author therefore made every effort to design a support system that would ensure stability even after the undercut had been run over pre-developed minor and major apices. It was planned to avoid continuous support rehabilitation during the life of the cave.

The objective of a support system is to provide sufficient confinement to limit the growth of the fracture zone around the excavation so that displacements remain within stable limits during initial development and the subsequent stress changes that characterise cave mining. The fracture zone must be protected by the support system from erosion by mining activities. If stress levels and rock mass strength are such that collapse of the excavation is probable, this should be known in advance. It demands a knowledge of rock mass behaviour during the stress changes and of the characteristics of the support system that is installed.

However, "there is an incomplete understanding of rock mass behaviour in caving areas which leads to a high degree of uncertainty in support design. The design process is still largely one of trial and error. Full analytical designs are not achieved because quantitative information on the behaviour of the rock masses involved is not available. Moreover the layout of cave production areas does not consist of isolated drifts amenable to simple analysis. The problem is fully three dimensional. The imposed loads are dynamic and variable in time and space" (Cummings et al., 1984).

For these reasons, support design on most caves mines remains empirical (Lacasse & Legast, 1981. Brumleve & Maier, 1981. Brumleve, 1987. Ferguson, 1977. Kvapil et al., 1989. Wilson, 1992). Specialised support has been developed for areas that are subjected to unusually high levels of stress damage, either as a result of awkward geometry (bullnoses and camelbacks) or erosion by mining activities such as LHD impacts and secondary blasting (drawpoint brows and crosscut sidewalls). The rock mass on the extraction level is often damaged by high abutment stresses and most cave mines accept regular support rehabilitation as a part of cave mining operations.

Two rock mass classification systems that specifically address cave mining methods both recognize that the support system must be tailored to the rock mass response to the mining process. The rock mass rating of the pristine rock is adjusted to take cognisance of the damaging effects of the cave mining process. Massive, passive, rigid support is required to ensure the stability of excavations in rock with a low rock mass rating. Active support that harnesses the residual strength of more competent rock is recommended for rock with a higher rating. A major problem is that rigid linings such as shotcrete or concrete are required to protect the rock on the extraction level from erosion by mining activities. These same rigid linings are damaged as the abutment stresses are run over the area if post undercutting is practised. This has led to a two phase system of support being advocated (Cummings et al., 1984) where rigid linings are only installed after the undercut has passed over the area. The undercut stresses, however, can cause substantial rock mass damage and the rock might need to be massively supported to ensure the stability of the excavation as the undercut is run overhead. At Premier, drawbell development produces 5 000 tons of rock and the 12 metre high undercut a further 14 000 tons of broken ore. Block fallout in the cave back inevitably produces some additional ore, often in the form of large blocks that must be blasted. At least 20 000 tons of ore is produced from a drawpoint before the undercut has passed over. Considerable erosion by mining activities therefore occurs before the undercut is run over the area. The support recommended for cave mining operations by the two rock mass classification systems is set out in Figures 5.2. and 5.3. Cummings' chart recommends support in terms of his adjusted rock mass rating with no specific reference to the stress regime although this is considered in deriving the adjusted rock mass rating. Laubscher's chart recommends support in terms of his MRMR. The stress regime is qualitatively considered in that rigid support is not recommended for high stress situations.

Problems with support in both kimberlite and norite below the sill had resulted in the loss of several million tons of high grade ore reserve. Experience had, nevertheless, shown that fully resin-grouted rockbolts, grouted cable anchors, mesh reinforced shotcrete and massive concrete arches could be effective support elements, under certain conditions.

It was judged that neither previous experience nor rock mass classification methods alone could be used for effective support design. The mine therefore employed consultants with a brief to assist the author in applying a rigorous analytical design process in a planning a support system that would not fail under conditions that were expected to prevail in the BA5 cave.

The logic employed in the support design process for the BA5 was:

* How would the support elements that had proved effective at Premier function when subjected to the five stages of stress change expected in the cave ?
* What objective criteria could be used to measure effectiveness of the support system and how would the support system be expected to fail, if this occurred ?
* How would these criteria be measured using a simple monitoring programme ?
* How could monitoring results be used to improve support design ?

### 5.4.1. HOW WOULD SUPPORT ELEMENTS THAT HAVE PROVED EFFECTIVE AT PREMIER FUNCTION IN A CAVE MINING SITUATION ?

A literature review highlighted a dearth of information on the way in which linings of concrete or shotcrete could be designed to function as interbolt support in excavations experiencing large convergence. High strength concrete linings are rigid and can be expected to fail at small levels of convergence when subjected to high stress. Shotcrete linings are relatively flexible and shotcrete linings reinforced with steel mesh or steel fibre are more flexible and have some residual strength and the ability to provide limited support even after failure.

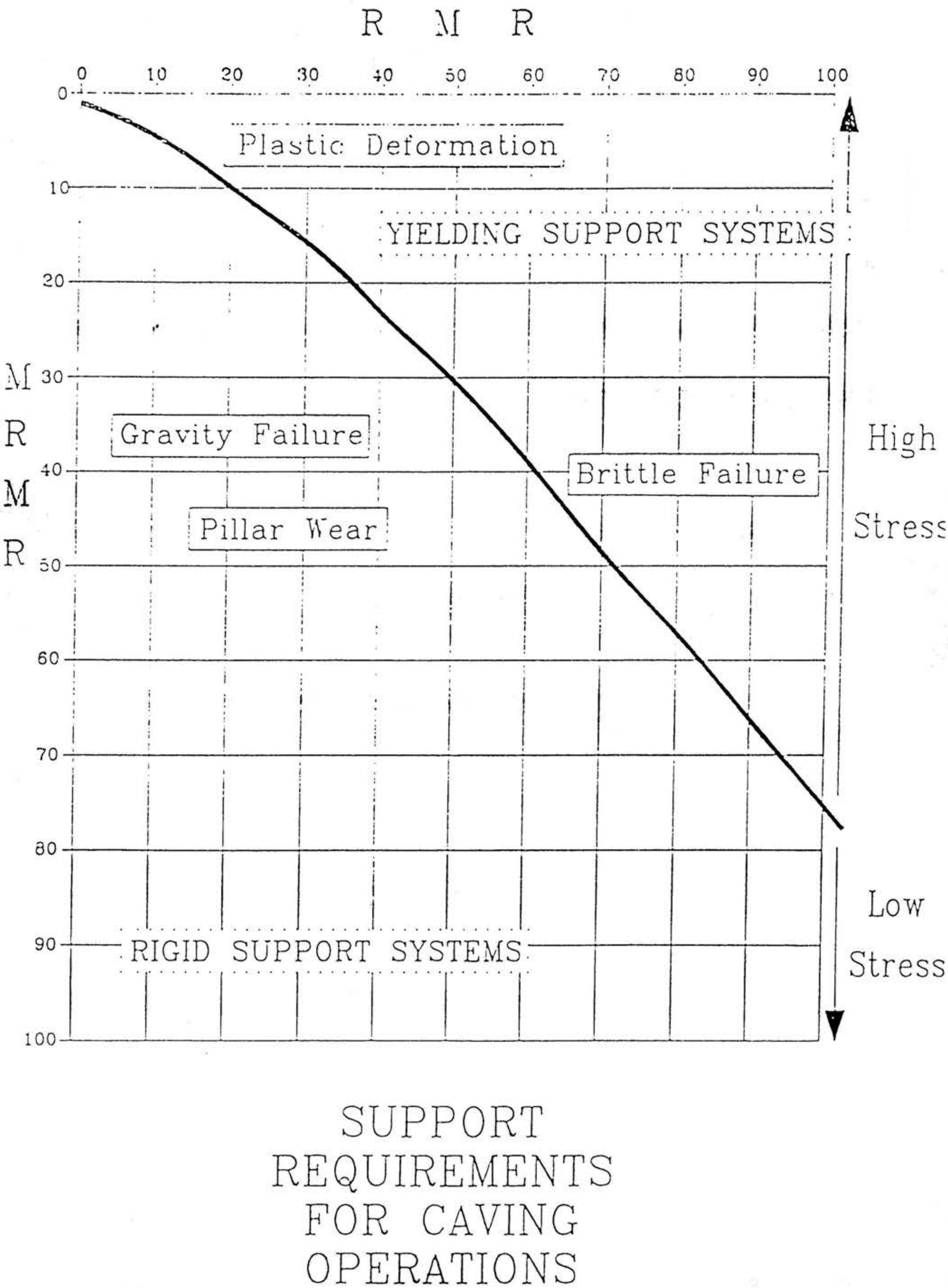



Figure 5.2. Laubscher's Support Recommendations (Laubscher, 1995).

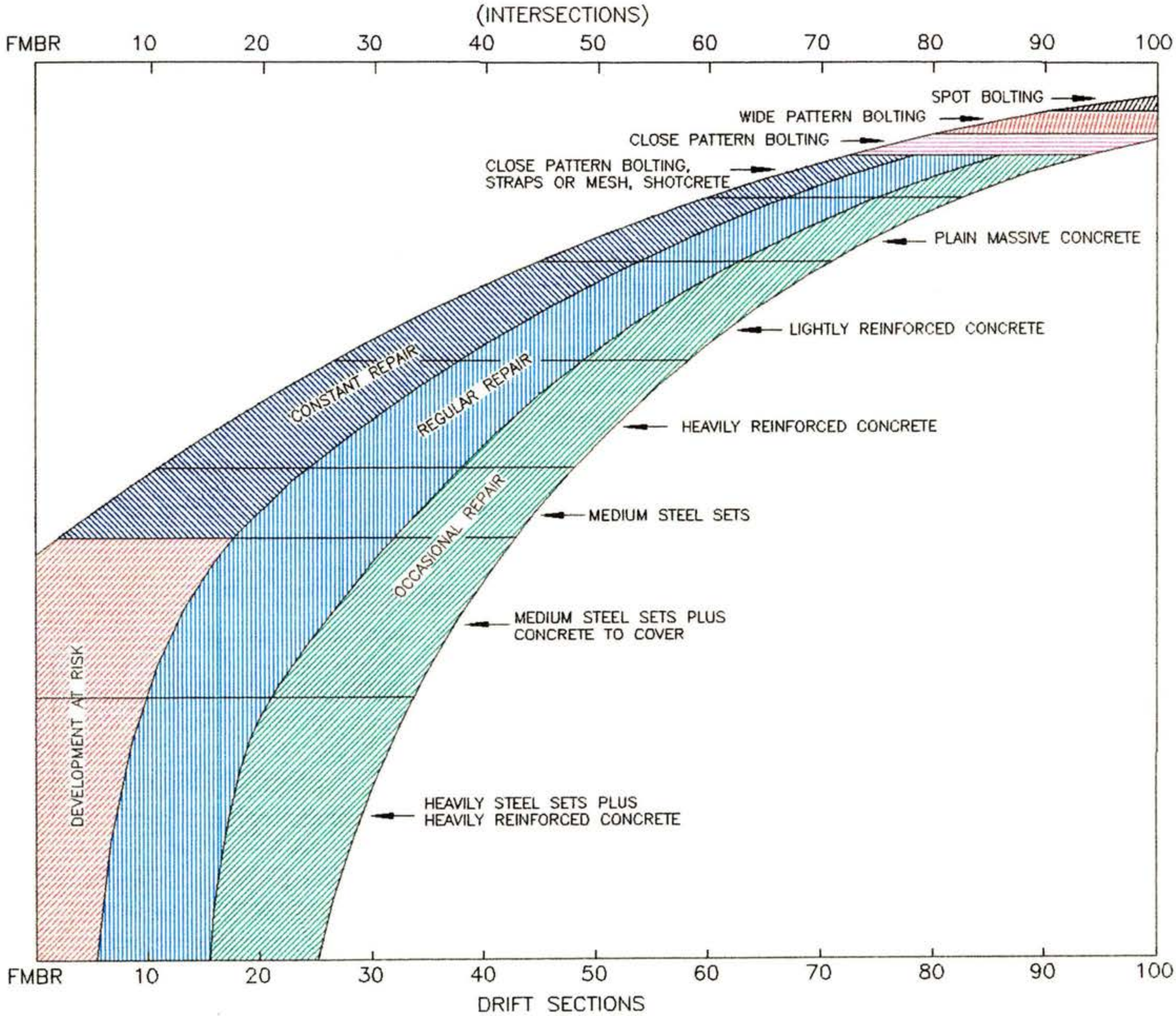



**Figure 5.3. Cumming's Support Recommendations (Cummings et al., 1984)**

Extensive testing to establish the support properties of shotcrete under load and at large deflections was undertaken for the mine (Kirsten & Labrum, 1991. Kirsten, 1992). Numerical models were correlated with monitored results and extensive numerical modelling was undertaken to establish the effect of the various stages of cave mining on displacements in the rock mass and the loads that would result in steel tendons used to support excavations. The ability of mesh- and fibre-reinforced shotcrete to fulfil a support function within this environment was studied. The effect of tunnel size, shotcrete thickness and strength, as well as the effect of various steel tendon lengths and strengths together with grout bond strengths, was considered.

**Table 5.1. Maximum Steel Anchor Forces for Various Tunnel Sizes**

| Tunnel Dimensions (m x m) | Maximum forces (kN/m) developed during retreat of undercut stage. (Figures in brackets are for additional 300 millimetres shotcrete lining) | | |
|---|---|---|---|
| | Hangingwall | Sidewall | Footwall |
| 3,0 x 3,5 | 17 32<br>(13) (9) | 58 63<br>(21) (32) | 38 45<br>(20) (45) |
| 4,0 x 4,2 | 20 37 67<br>(15) (12) (26) | 95 82 42<br>(34) (54) (25) | 48 44<br>(39) (41) |

After Kirsten & Bartlett, 1992

Important results from these studies were that extremely high loads could be expected to develop in the steel tendons as the undercut was run over support on the extraction level (Table 5.1). Loads were considerably higher in steel tendons installed in excavation sidewalls than in the hangingwall. Shotcrete fulfilled an important load spreading function, acting as a large face plate and helped to even out the forces that developed. In weak rock, it was impractical to install a sufficient thickness of shotcrete to ensure that the shotcrete lining did not fail. Although the strength of the shotcrete lining increases with increasing thickness (or modulus) so does the rigidity of the lining. The rigid lining attracts high stresses and results in failure of the shotcrete as high hoop stresses are set up in the lining which then fails in shear up to all practical thicknesses of application. The same logic applies to concrete linings. The usual sequence of destruction of support is failure of the rigid lining in shear at the points of maximum curvature of the tunnel. As the lining destructs, increasing load transfers to the steel tendons. Where these loads exceed the tensile strength of the tendon, the tendon fails (Kirsten & Bartlett, 1992).

Time constraints and resources made it impractical to undertake detailed testing of the steel tendon support in the form of fully resin-grouted rock bolts and cement grouted cable anchors used for support at Premier. Moreover, the behaviour of grouted rockbolts and cables as reinforcement elements has and is being widely studied worldwide. The brief review presented below identified aspects that were thought to be relevant to steel tendon support at Premier.

* A zone of radial compression is created in the area influenced by each bolt or tendon. If tendons are close enough, an arch or ring of radial compression will be formed around the tunnel.
* If tendons are installed before relaxation of the rock mass, a zone of circumferential compression can be maintained around the excavation.
* Tensioning of tendons will help to retain the level of strain in the rock mass if these are installed immediately after excavation.
* Tendons installed across discontinuities will have the effect of limiting dilatation and increasing friction on the interface.

Rockbolting has two effects. The first of these is to provide a support pressure aimed at creating a zone of radial and circumferential compression around the tunnel. Such support pressure can be calculated (Hoek & Brown, 1980) or simulated by numerical modelling. The objective is to create a state of triaxial stress in the blast damaged zone around an excavation.

The second effect is to prevent displacements by retaining the level of strain and limiting movement on discontinuities. It has the effect of increasing the in situ rock modulus of the supported rock and is an important role of steel tendon support in a hard rock environment. This reinforcement effect is increased by grouting and tensioning of cables and bolts and cannot be calculated or easily simulated. If high stress levels result in rock failure, new discontinuities are created and the initial level of strain may not be sufficient to prevent instability.

In terms of the second effect three stages of rockbolt-grout-rock behaviour can be distinguished (Spang & Egger, 1990):

1. Elastic stage: When the strength of the joint is exceeded, sliding starts to occur. The shear resistance of the bolted joint consists of the shear strength of the joint and the contribution of the bolt which consists of the elastic response of the bolt, grout and rock. This means that the bolt effect depends on Young's modulus of these three elements, as well as on the dimensions of the bolt and the grout sleeve.
2. Yield stage: In most rocks, the bolt has to deform before it offers shear resistance. During this deformation the yield strengths of the steel and grout are reached by bending and compression, respectively. It happens at very low displacements and forces (<1 millimetres). Behaviour during the yield stage is governed by the compressive strength of the rock and grout and the yield limit of the steel.
3. Plastic stage: All the materials yield at an early stage at low shear forces. Thereafter, the shear response of the bolted joint depends on the plastic behaviour of the materials. The contribution of the bolt to the total shear strength of the joint is a function of:

* angle of friction along the shear plane
* bolt inclination
* stiffness of grout and rock
* angle of dilation
* working capacity and deformability of the steel
* diameter of the bolt
* thickness of the grout collar.

Modelling and testwork shows that the contribution of fully cement-grouted bolts to the shear resistance of a rock joint depends strongly on:

* The friction along the shear plane
* Inclination of the bolt increases the shear resistance by up to 20 percent
* Dilation of the joint also increases the effect of the bolt by up to 20 percent
* The shear resistance of the bolted joint increases with the deformability of the rock and grout.

This result is of importance as it explains some of the discrepancy in results noted with regard to pretensioning of bolts and stress effects on joint surfaces. The quick dissipation of the applied force means that little normal stress is applied to a joint surface any distance from the face by the action of a pretensioned, grouted rockbolt or cable anchor.

As soon as any movement along joints occurs or the rock starts to fracture, a series of forces will be set up at each point of movement along the bolt. Any new fractures that start to form close to existing fractures will be inhibited by the force in the cable or rockbolt. In effect, the grouted steel reinforcement becomes more effective as the rock mass fails, as long as the steel and grout annulus retain their integrity. Both experiment and observation show that rock and grout are plasticised where shear displacements occur. Away from shear planes, rock and grout retain their integrity.

In a caving situation, the load imposed on the pillars on the production level are a combination of shear and tension. Steel reinforcement aims at maintaining the integrity of the rock mass by ensuring that the rock blocks remain locked together in as far as this is possible (Brumleve, 1987). The precise functioning of grouted bolts and cables is, however, poorly understood. Laboratory testing of the role of grout and rock-grout-steel interaction is complex and difficult to study in a laboratory or in the field. Computer simulation of the rock-grout-steel reinforcement interaction is similarly complex. Tensioning of the bolt or anchor adds further complexities (Spang & Egger, 1990).

Work done on the distribution of forces along a pretensioned, grouted rock bolt (Tadolini, 1990) shows that the applied force dissipates quickly with increasing distance from the face along the bolt, first linearly and then exponentially. Testwork further shows that "only a small fraction of the axial strain, and therefore axial load, applied at the proximal end of the bolt is transferred to the distal end of the anchorage" (Conley & Priest, 1992).

Recent work (Kaiser et al., 1992. Hyett et al., 1992) shows that cable installations act as frictional rather than adhesive support systems. The effectiveness of cable installation is a function of the grout properties, the embedment length of the tendon and the radial confinement acting on the outer surface of the grout. Stress changes, such as those experienced in a cave mining situation, can reduce the effectiveness of cable installations considerably. Cable installations achieve peak loads and maximum effectiveness at displacements of the order of 50 millimetres. In hard rock failure usually takes place at the steel/grout interface, or as a result of failure of the steel tendon. In kimberlite, tendons fail as a result of the rock disintegrating or the grout/rock bond failing. Occasionally the tendons are pulled through their plate washers.

Cable systems can be engineered in such a way as to meet difficult requirements. Pretensioned, fully grouted cable anchors are stiff and effective in maintaining the integrity of the rock mass. If cables can be installed before the rock starts to relax towards the opening it is not necessary to pretension the bolts. Monitoring of the rock mass response to blasting in kimberlite, however, shows that this relaxation starts within hours and it is impractical to install the required steel reinforcement before substantial relaxation has occurred (Guest, 1985). All elements of pretensioned support need to be equally strong to withstand any imposed support load as the system has little flexibility. If the imposed stress exceeds the strength of any element, including the rock, the support quickly becomes ineffective.

### 5.4.2. PRINCIPLES OF SUPPORT DESIGN ON THE BA5 EXTRACTION LEVEL

Rock mass strength was determined by laboratory testing and geotechnical mapping of underground exposures. Laboratory results and rock mass ratings were then correlated with parameters needed for accurate numerical modelling using the FLAC solution scheme (Cundall, 1993. Howell et al., 1993). Expected stress levels were derived from the numerical modelling and confirmed by monitoring underground. The proposed support system was modelled and parameters adjusted until good correlation was found between monitored and modelled results. The Hoek and Brown failure criterion and the correlations determined between this criterion and Bieniawski's Rock Mass Rating (converted from Laubscher's Rock Mass Rating) were extensively used by the author to calculate failure criteria and determine the necessary parameters for numerical modelling. The support design process is set out in more detail in Appendix 1.

The design process determined that the rock on the extraction level was generally sufficiently strong to withstand the stress changes that would be imposed by successive stages of cave loading. The rock would, however, need to be extensively reinforced to provide a support pressure aimed at creating a wide zone of radial and circumferential compression around any excavation sited in the cave area. This support pressure was designed to be sufficient to prevent displacements by retaining the level of strain and limiting movement on discontinuities. The zone would be created by installing sufficient fully grouted steel tendon reinforcement in the form of short 1,8 metre long, 20 millimetre diameter and 6 metre long, 12 millimetre diameter cable anchors. The length, density and strength of the steel tendons and grout strength was determined by simulation using calibrated numerical models.

Mesh reinforced shotcrete was used for interbolt support. The shotcrete was of uniform strength, but thickness and reinforcement was increased where high stresses and subsequent erosion by LHD impact and secondary blasting damage was expected. Shotcrete strength, thickness and reinforcement was determined by numerical modelling, engineering judgement and experience on the mine.

Using this support design process, typical support in production tunnelsd and drawpoint crosscuts consisted of 1,8 metre resin-grouted rockbolts installed on a 1 metre spacing down to footwall. The excavations were then covered with a mesh reinforced shotcrete lining of up to 120 millimetres down to footwall. Six metre long fully grouted 25 ton cable anchors were installed on a 1 metre within-ring and a 2 metre between-ring spacing through the shotcrete.

Footwalls were concreted with rails imbedded to act as a wearing surface and allow cleaning of the footwall using the LHD bucket.

Effective interbolt support is essential in most cave mining operations. Rigid linings and drawpoint brows are, however, damaged by high stresses. The level of support needed is often an order of magnitude greater than recommended by rock mass classification systems. (Speers, 1990. Cummings et al., 1984). Even with an increased level of support, deformations are large. The lower limit for shotcrete or concrete lining is a thickness great enough to cover installed steel support in the form of rockbolts, tendon straps and mesh. It is not practical to determine a theoretical upper limit for lining thickness. Practical constraints such as cost and tunnel size usually limit lining thicknesses to less than 1 metre. Even the most massive support cannot guarantee that excavations will not be crushed by high stresses if the area is not undercut, or if compaction occurs and allows stresses to be re-imposed. The adjustments suggested by Laubscher in deriving his MRMR need to be carefully considered and can exceed the maximum level of 0,9 of adjustment applied to reduce the RMR in terms of the expected stress change. Once interbolt linings have been damaged, support rehabilitation in the form of massive concrete linings, is often needed to ensure that the support system is sufficiently strong and flexible to withstand the imposed stresses during subsequent years of mining.

Bullnoses and camelbacks, because of their geometry, are subjected to unusually high stress loads and impacts by LHD's and need specially designed support. These areas require effective lateral constraint and a rigid lining to protect them from erosion by mining activities. At Premier these areas were constrained by wrapping 20, 40 ton, 6 metre long cable anchors around the bullnose or camelback in pairs at 300 millimetres vertical spacing. The ends of the cable anchors were grouted into drillholes, and a block and wedges used to tie the anchors together and tension them to 8 tons. The steel cables were then covered with mesh and shotcrete. Where large displacement occurs and lateral constraint is most needed, the steel ropes have proved effective.

Drawpoint brows are supported using concrete arches or steel reinforced concrete arches tied into the sidewall using steel tendons, in most mechanised cave mines. Experience at Premier in the open stopes had shown that drawpoints lined with 500 millimetres thick 55 MPa concrete in areas of high convergence were first damaged by high shear thrusts in the rigid lining and then destroyed by subsequent, continuous secondary blasting. In the BA5 cave, the decision was taken to increase the flexibility of the drawpoint lining by creating a formwork of two layers of flexible mesh and steel tendons tied into the rock mass using 3 metre long, resin grouted rockbolts. The steel was then covered with up to 300 millimetres of shotcrete. The inherent strength of rock was harnessed using 6 metre long 25 ton cables, installed in a ring pattern on a 1 metre spacing in the 5 metres of crosscut immediately behind the drawpoint brow. This pattern of support extended through the footwall which was further supported using at least 300 millimetres of concrete and steel rails. Minor apices adjacent to the drawpoints were strengthened by installing fully grouted 10 metre long 25 ton cable anchors through the apices. The cables were installed from the production tunnel over the crosscuts.

**P.5.1. 25 ton cables at 300 millimetres were tensioned around bullnoses and camelbacks and encased in shotcrete to provide efficient lateral support**

**P.5.2. Drawpoints were supported w ayers of mesh and 2 layers of steel tendon straps tied into the sidewall by rockbolts. This steel construction was encased in 300 millimetres of shotcrete.**

### 5.4.3. WHAT OBJECTIVE CRITERIA COULD BE USED TO MEASURE THE EFFECTIVENESS OF THE SUPPORT SYSTEM AND HOW WOULD THE SUPPORT SYSTEM BE EXPECTED TO FAIL, IF THIS OCCURRED ?

An integrated approach to support design which incorporated elements of analytical design, use of the Hoek and Brown failure criterion as well as the use of a rock mass classification to assess rock mass strength was undertaken in a high stress, hard rock, mining environment (Hepworth & Gay, 1986). The relatively weak rock and high stresses associated with cave mining make many of the principles established in this study relevant to support in a cave mining environment. The logic and methods applied in the study were useful to the author in determining what criteria could be used to judge support effectiveness and to plan a monitoring programme to measure these criteria at Premier.

This study found that the most important factors determining the deformation of the rock around a tunnel that was subsequently overmined were:

1. The stress field

   Damage to the rock mass correlated well with stress change only where the field stress was already close to a critical state of stress. Otherwise damage to the tunnel was better correlated with total stress. The duration for which the tunnel was subjected to an increased stress change, as well as a decrease in stress had a negligible effect on tunnel damage.

2. Geology

   Tunnel damage was well correlated with geology. It should be noted that the supported tunnel was in a fairly uniform geological domain.

3. Support performance.

   The support system was aimed at maintaining the integrity of the zone of fractured rock around the tunnel so that the zone is self supporting and creates a confining pressure that prevents the propagation of the fracture zone.

   Factors that were found to influence the support performance were:

   * stress change
   * tendon density
   * tendon length
   * yield loads and critical bond lengths
   * effectiveness of interbolt fabric in preventing fallout and ravelling

   It was found that a stress change of between 5 and 15 MPa resulted in little convergence. Between 15 and 30 MPa convergence was a linear function of stress change while once the tunnel had been subjected to a stress change of 30 MPa, any positive, induced stress continued to cause some convergence.

A hyperbolic relationship exists between normalised support resistance and convergence, with convergence very large below a ratio of 2 and small above a ratio of 10. Normalised support resistance is defined as a function of tendon density, yield loads and actual tendon length.

It was further noted that effective support limited dilation in the supported zone (i.e. created a more effective state of triaxial stress in this zone) and resulted in increased dilation beyond the installed tendons.

A good correlation between the calculated zone of dilation and measured dilation was achieved by using Bieniawski's classification to derive a rating for the damaged rock around the tunnel and then applying the Hoek and Brown failure criterion for damaged rock to calculate the expected zone of dilation.

It was concluded that tunnel support can be designed to limit dilation to an acceptable level provided that the level of stress change and the strength of the in situ the rock mass can be calculated. This study provided useful criteria for judging support design effectiveness at Premier Mine.

Experience and previous monitoring on the mine, the work carried out by Hepworth and Gay, the literature review on cave mining and numerical modelling, all indicated that high and variable stresses, especially those associated with the abutment, could result in displacements in spite of the installed support. This could lead to support and rock mass failure. Stress changes and displacements were, therefore, criteria that needed to be measured to assess the effectiveness of the support system. Aspects of the mining operations such as the creation of remnant pillars (stubs), large leads and lags, and the shape of the undercut face could lead to unexpectedly high stress levels. Correlations between these and the stress levels that they induced, and consequent aggravated rock mass and support damage, needed to be monitored to assess support effectiveness and improve support design.

## 5.5. MONITORING

A programme was designed by the author and installed by the Geotechnical Department to measure stress change and displacements underground. Regular, planned observations were carried out by the author and other geotechnical staff using a simple method to quantify the extent of damage in defined areas of the extraction level. This programme included detailed observation of the performance and mode of failure of support and underlying rock as well as monitoring of displacements using sonic probe extensometers and stress changes using solid vibrating wire stressmeters.

The usefulness of the detailed observation of support performance was based on the premise that most support consisted of grouted steel tendons and mesh reinforced shotcrete that had been consistently installed throughout the mining area. Difference in behaviour was attributed to varying stress levels and rock mass strength.

### 5.5.1 SUPPORT DAMAGE MONITORING

**Objective**

Support on the extraction level was monitored on a regular basis. The objectives of support monitoring were to establish the efficiency of the support system and to improve support design. The data was also used to improve secondary blasting procedures and reduce LHD impact damage.

**P.5.3. Footwalls were supported with a rail construction bolted into the footwall and encased in at least 500mm of concrete to provide good roadways for LHD tramming.**

In some areas it is possible to correlate support damage with monitored stress changes and displacements. In other areas the monitoring process is more subjective. However, given the consistency of the support damage pattern and continuous observation and monitoring of the support damage process, as well as correlation with numerical simulation and observations in other mines, the process provided data that allowed the objectives of the support monitoring process to be met.

**Monitoring Process**

The extraction level layout was divided into 6 separate areas or zones and support performance in each of these was observed. The areas defined were:

* Drawpoints
* Crosscuts
* Bullnoses and camelbacks
* Drawpoint portals

* Footwalls
* Production tunnels

The drilling tunnel was the only element defined on the undercut level. A standard damage scale ranging from 0 to 6 was defined in terms of degree of support and rock mass damage. The damage scale is explained and illustrated in Appendix 1. Where possible, damage is correlated with monitored stress change and displacement.

A damage matrix is calculated for each half tunnel and correlations between the degree of damage and parameters that were known to result in this damage were investigated.

**Table 5.2. Damage Analysis: Tunnel 13 South**

| Drawpoint No. | 30 | 31 | 32 | 33 | 34 | 35 | 36 | 37 | 38 | Total |
|---|---|---|---|---|---|---|---|---|---|---|
| Footwall | 0 | 1 | 1 | 1 | 2 | 2 | 3 | 2 | 3 | 15 |
| Hangingwall | 1 | 2 | 2 | 2 | 3 | 3 | 2 | 2 | 2 | 19 |
| Sidwewall | 2 | 2 | 2 | 3 | 3 | 3 | 3 | 4 | 4 | 26 |
| Brow | 1 | 2 | 1 | 3 | 3 | 3 | 3 | 4 | 3 | 23 |
| Bullnose | 2 | 3 | 3 | 3 | 4 | 3 | 4 | 4 | 4 | 30 |
| Camelback | 1 | 2 | 2 | 2 | 3 | 3 | 3 | 3 | 4 | 23 |
| Prod. Tunnel | 1 | 2 | 2 | 1 | 2 | 2 | 2 | 3 | 3 | 18 |
| Totals | 8 | 14 | 13 | 15 | 20 | 19 | 20 | 22 | 23 | |

Table 5.2.illustrates such a matrix for Tunnel 13 South. The matrix pattern is similar for all ten half tunnels currently used for production in the BA5. Drawpoints with lower numbers in the matrix are located at the centre of the cave and were never subjected to high abutment stresses. These drawpoints at the centre of the cave show little damage, but the damage level increases towards the margins of the cave. In terms of the zones defined, bullnoses show the greatest damage followed by sidewalls, brows and camelbacks. Tunnel 13 was sited in uniform Tuffisitic Kimberlite Breccia where rock mass classification ranged from 49 to 55. Damage ranged from 8 in drawpoint 30 to 23 in drawpoint 38.

## 5.5.2. CORRELATIONS

* The most important correlation noted was between the Laubscher's Rock Mass Rating, as determined from geotechnical mapping carried out by the author, and damage. In the Tuffisitic Kimberlite Breccia, all five stages of cave loading resulted in greater rock and support damage than in Hypabyssal Kimberlite. In the more competent, Hypabyssal Kimberlite DS 4 damage was rare and DS 5 damage was never observed.

* The second highest correlation was between damage level and induced stresses due to undercutting. The larger the size of the undercut, the greater the rock and support damage up until the time that caving initiated. The local geometry of the undercut increased the level of induced stress, and hence rock and support damage, considerably. Local geometry included leads and lags between adjacent tunnels, remnant pillars and corners. The speed of undercut was a major factor in support and rock damage. A slow moving undercut in weak rock led to extensive rock failure and support damage which included destruction of rigid linings and footwall heave.

* There was good correlation between support and rock damage, and the level of secondary blasting. This was most noticeable in the brow areas and where induced stresses had already damaged the rigid shotcrete linings. It should be emphasised that this correlation was only good where the rock and support had already been damaged by other factors. Secondary blasting aggravated, but did not cause extensive support and rock damage.

* LHD damage of support was most noticeable around bullnoses and camelbacks due to the impact of the LHD damaging the shotcrete and cutting and steel rope straps that were exposed. The process was usually initiated by induced stresses damaging the rigid lining. Thereafter LHD's aggravated the damage unless linings were repaired or replaced.

* The correlation between tons drawn from a drawpoint and level of damage was poor. In well constructed drawpoints in competent rock that had not been subjected to high induced stresses, rock and support damage was minimal after 30 000 tons of ore had been drawn.

### 5.5.3. MONITORING RELATING TO THE ROCK MASS RESPONSE TO MINING AND SUPPORT DESIGN

**Objective**

The objective of the monitoring programme was

* to measure the stress changes that were induced in tunnels on the undercut and extraction levels of the BA5. Stress measurements provide an early warning of potential problems such as incomplete undercutting. Stress change data is also used for back analysis to provide realistic parameters for use in failure criteria and support design using numerical modelling.

  Stress change was monitored using vibrating wire stress meters supplied by Irad and Geocon. All stressmeters were carefully installed by a consulting firm, who specialise in the installation of these stressmeters, together with geotechnical personnel from the mine. Stressmeters were subsequently monitored on a regular basis by geotechnical personnel. Each stressmeter was calibrated by the supplier and the calibration factor was used to convert microstrain units to kPa. The procedure used to convert microstrain readings taken underground to MPa was in accordance with the methods detailed in the manufacturers manual. The result of calculations at Premier showed that a microstrain unit was close to one kPa.

* to measure the dilation that occurs around excavations on the undercut and extraction levels and ensure that displacements remain within stable limits. Progressive dilation is a good indicator of incipient rock and support failure. Dilation can also be used to monitor the zone of failure around excavations and measure the efficiency of the support system. Displacement data, together with stress data, can be used to calibrate failure criteria and improve support design.

Displacements were measured using sonic probe extensometers. Sonic probe extensometers were installed by drilling a 54 millimetre diameter core hole into the rock in which displacement was to be monitored. A reference magnet was then installed at the end of the hole, at a maximum depth of 7 metres into the rock mass. Up to 10 auxiliary magnets can be installed in the same hole at the required distance from the reference magnet. These auxiliary magnets consist of several permanent magnets glued into in a spring loaded plastic ring. On installation, the spring anchors the auxiliary magnet firmly in position. Displacement was measured by inserting a flexible probe attached to a digitial readout unit through the auxiliary magnet rings until the end of the probe was firmly seated in the reference magnet at the end of the hole. A electronic signal was then passed down the probe. The auxiliary magnets modify the electronic pulse and this modification is interpreted by the instrumentation, and the distance measured in this way between the reference magnet and the auxiliary magnet is digitally displayed on the readout unit. Measurement is to an accuracy of 1 millimetre. These sonic probe extensometers were simple to install and monitor and were installed by the Geotechnical Department, and measured on a regular basis by mine staff. Standard procedure was to measure a sonic probe, installed near the shaft, at the start and finish of monitoring to ensure that the probe was not giving false readings. No movement was ever detected at the control site.

**Monitoring Stations**

1. A series of six boreholes per site were drilled at 12 monitoring stations. The monitoring stations were chosen to the north and South of the undercut where it was estimated that maximum stress levels would develop as the undercut area approached the hydraulic radius needed to ensure continuous caving. Two additional sites were instrumented. One of these sites (T25TR29) was in the area originally undercut and hence never subjected to high abutment loads. The other site (T29TR24) was 30 metres to the East of the advancing undercut in an area judged to be outside the range of high abutment loading, at least in the initial stages of undercutting.

2. Two of the six holes, one on either side of the crosscut, were 36 millimetres diameter, horizontal, diamond drill holes. Geocon vibrating wire stress meters were installed at a depth of 7 metres in each of the holes. The stress meters were oriented so as to read stress changes in the vertical direction. The meters were read at weekly intervals.

3. The four remaining holes, two on either side of the crosscut, were drilled upwards and downwards at an angles of between 15 and 25 degrees. The holes were 54 millimetres in diameter. The holes were used for sonic probe extensometer installations. The holes were seven metres long and either 4 or 5 monitoring magnets were installed. The reference point was at the end of the hole. These installations were used to measure displacements at various depths in the pillars to an accuracy of 1 millimetre. Readings were taken at weekly intervals.

4. At three sites sonic probe extensometers only were installed and used to monitor displacements. Results from these three extensometers are graphed in figures 5.11.a to 5.11.c. The eastern extensometer, installed below the lag pillar, shows the greatest degree of deformation.

A total of 66 stressmeters and 72 sonic probe extensometers were installed. All installations sited in both Tuffisitic and Hypabyssal Kimberlite yielded at least some data.. The layout of a typical monitoring station is shown in Figure 5.4. The position of monitoring stations on the 630 level is shown in Figures 5.5.

### 5.5.4. PRESENTATION OF DISPLACEMENT AND STRESS CHANGE DATA

In all the graphs, displacement in millimetres is scaled along the left margin of the graph. Displacement takes the form of sharp, reversible, spikes which are the result of movement along existing joints and fractures, abrupt displacement steps which are the result of the development of and dilation on new fractures, or irreversible movement on existing fractures. Gradual, continuous displacement is the result of strain, or the formation of induced fractures with little or no dilation. All graphs depict displacement data.

Initially, the sharp displacement spikes were thought to be incorrect probe readings as sonic probe extensometers installed in other areas of the mine, as well as on other De Beers kimberlite mines (Guest, 1985), never displayed these sharp spikes. It was speculated that movement in the hole made it difficult to insert the end of the probe snugly in the reference magnet at the end of the hole. For some time, it became standard for the author and other geotechnical staff to repeat any probe reading that indicated abrupt displacement. Repeating measurements seldom, if ever, indicated that the readings were wrong. Measurement in the same hole, often on the following day, showed that the displacement had disappeared. Careful observation of the rock mass often indicated movement on existing joints and fractures, and even new, induced fractures, when these displacement spikes were measured. The displacement spikes often correlated with abrupt stress changes. It was finally concluded by the author that these displacement spikes were the result of movement, often reversible, on existing and induced joints and fractures.

Stress change measured in microstrain units, is scaled along the right hand margin of the graphs. Stress change was often abrupt, but stress spikes analogous to the displacement spikes were never measured. Stress gauges were installed at depths of five metres into excavation sidewalls. Not all graphs include stress data.

### Interpretation of stress and displacement correlations

Figure 5.6. shows the result of displacement monitoring along a single sonic probe extensometer drilled into a minor apex in Tuffisitic Kimberlite Breccia on the 630 metre extraction level. The size of the undercut at this stage was about 14 000 square metres. Fracturing in zone 1 (the outer zone) occurred at the same time as a stress spike of more than 400 microstrain units was measured. Strain measurements thereafter were erratic but displacements deeper into the apex continued at an unabated rate.

Figure 5.7 shows monitoring results in tunnel 21 drawpoint 34 (T21DP34) as the undercut advanced over the monitoring station. North up refers to a sonic probe hole drilled upwards at 15 degrees, North down refers to a sonic probe installed in a hole drilled downwards at 15 degrees. Both holes were drilled in a Northerly direction. Stations South down and South up are mirror images drilled in a Southerly direction. The stressmeters were installed in horizontal hole drilled in a North and South direction (see Figure 5.4) This monitoring station is located in Hypabyssal Kimberlite with a uniaxial compressive strength of 150 MPa. At the time of monitoring, the area of undercut was 12 000 square metres.

All four probes show similar results. Initially there are some reversible displacements indicated by spikes showing peaks of up to 15 millimetres of displacement. Irreversible displacement of up to 20 millimetres occurred in the outer zone of both the South down and North up station. There is an abrupt increase in stress as the undercut advances over the installation with stress change of 3000 microstrain units monitored on the North side of the cross and 3500 microstrain units measured on the South side. There is a reversible displacement spike of 12 millimetres measured immediately after the stress increase at the North down installation and a similar spike at the South down installation immediately after the stress increase. Once the rock has been affected by the abutment stresses continuous displacements are measured up to 6 metres into the apex. Displacements range between 10 millimetres and 40 millimetres. The South down installation (4) was lost as the undercut passed over the station. Once the undercut had passed over the area, extensive cracking was monitored in the shotcrete and induced fractures in rock were observed in the bullnose area.

Figure 5.8 shows the movement measured at different depths along a single sonic extensometer hole drilled into a minor apex in Hypabyssal Kimberlite. Fracturing in zone 4 (the outer, blast damaged zone) occurred soon after monitoring started. Reversible spikes were noted in zones 2, 3 and 4 and a fracture in zone 1 within 6 weeks of the sonic probe being installed. Thereafter, displacement occurred at more or less the same rate at all depths into the apex, showing that support was not effective in limiting propagation of the fracture zone into the minor apex.

### Consideration of some Specific Results from Monitoring Stations

<u>CASE 1:</u> The highest stress change monitored on the extraction level was in tunnel 25 on the South side of drawpoint 34. Figure 5.10 shows an irreversible displacement of 12 millimetres in the outer two zones shortly after installation of the instrumentation. Little displacement was monitored for the next 7 months until an abrupt stress increase occurred on 10/7/91. The stress increase peaked at 7000 microstrain units about 2 months later on 11/9/91. This stress increase was accompanied by shortening of the extensometer hole by about 13 millimetres at depths of

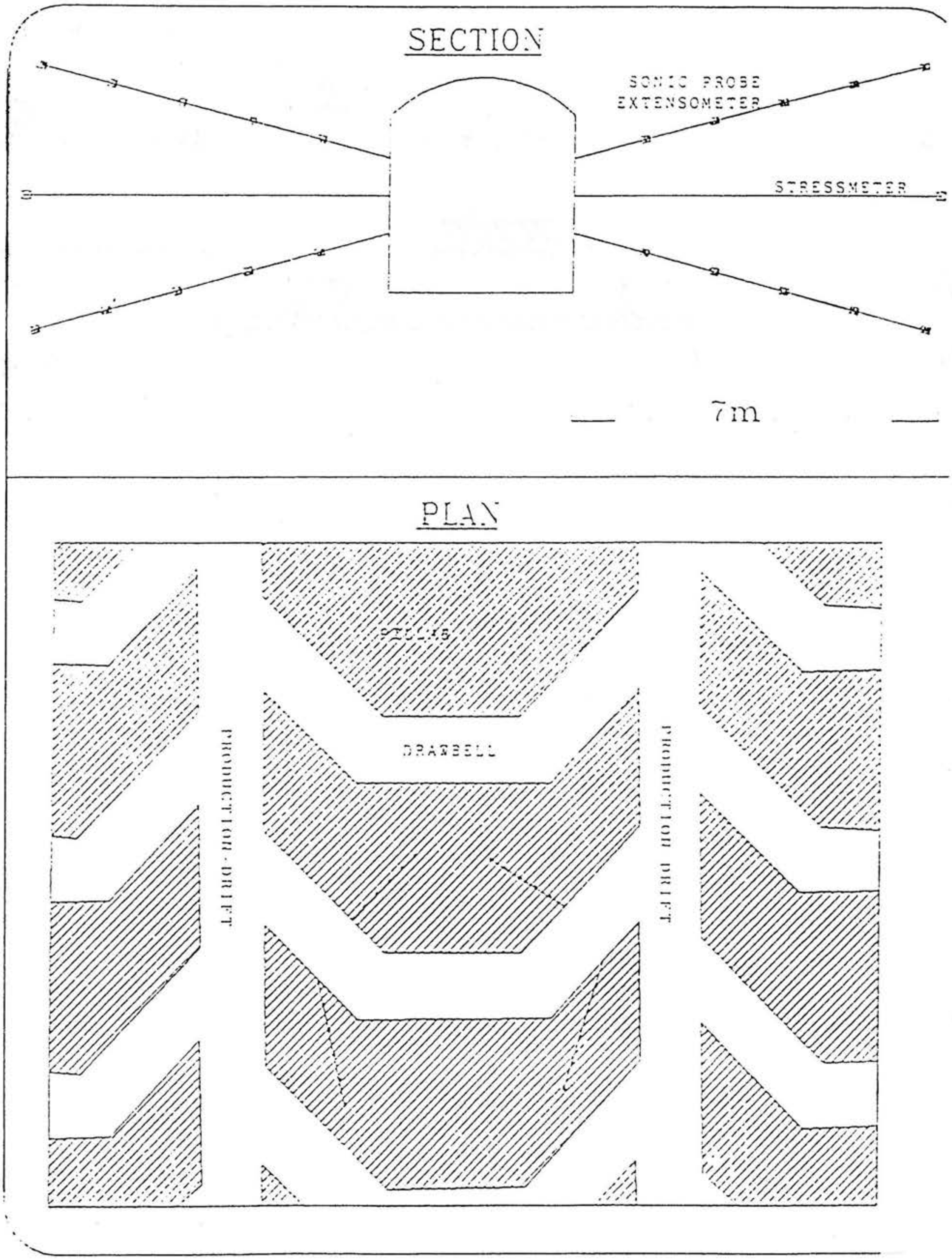



Figure 5.4. Section and Plan of Typical Monitoring Station

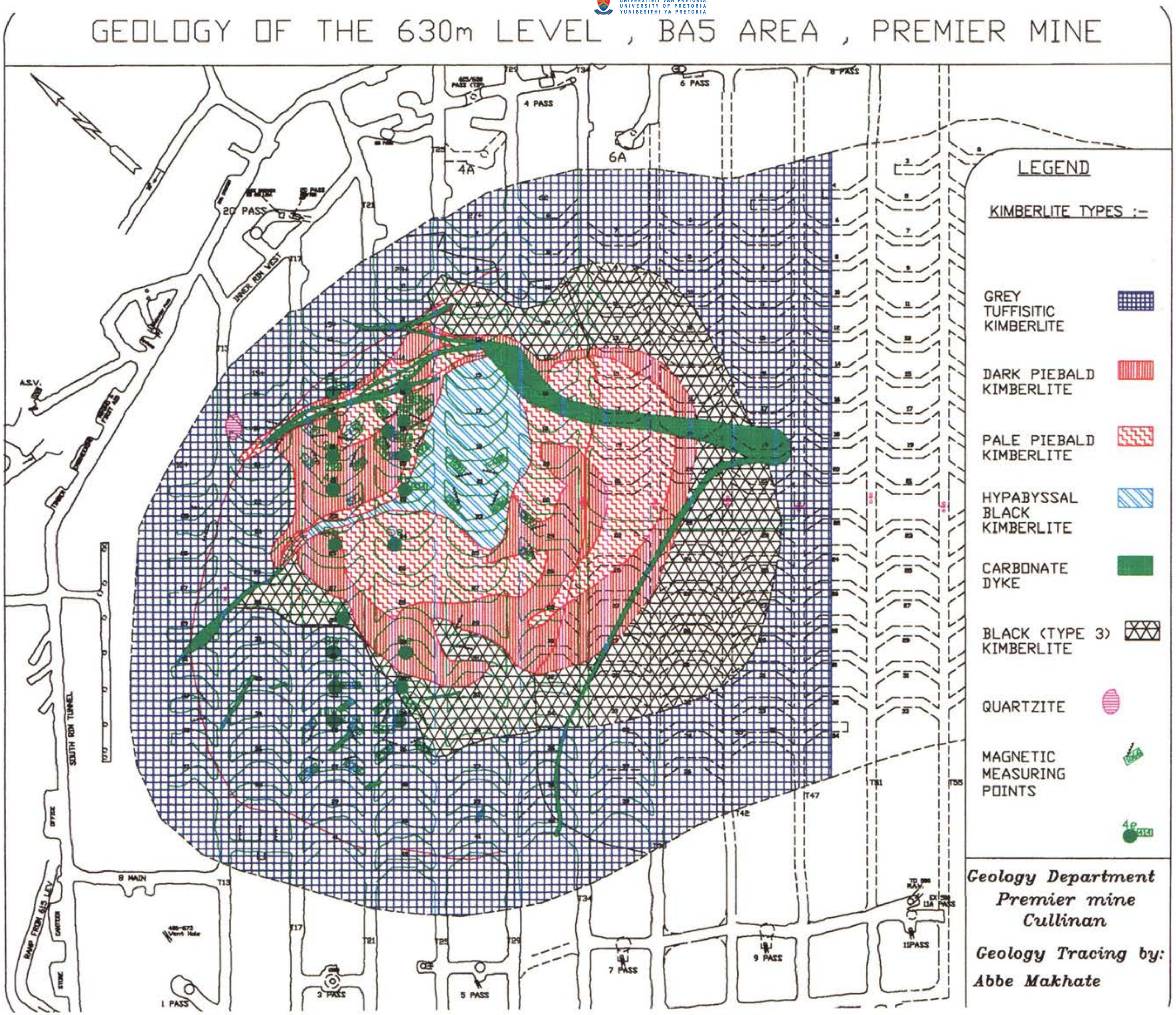



Figure 5.5. Layout of BA5 Production Level showing Monitoring Stations

up to 6 metres into the rock adjacent to the crosscut. As the stress decreased the rock relaxed by up to 30 millimetres. By this stage the shotcrete lining and underlying rock had been extensively cracked and damaged and the drawpoint had to be repaired using massive concrete. The monitoring installations were lost.

<u>CASE 2:</u> Three extensometers were installed on the Southern margin of the cave in a crosscut that was well away from the advancing undercut at the time of installation. A lead and lag situation existed between tunnels 13 and 17 and had a variable effect on the extensometers. The rock in which the extensometers were installed was Tuffisitic Kimberlite Breccia. The layout is shown in Figure 5.9. The extensometers were installed within 1 week of the crosscut being developed. Pattern bolting, using fully resin grouted 1,8 metre long rockbolts on a 1 metre spacing and 6 metre long, fully grouted 25 ton cable anchors also on a 1 metre spacing, was installed at the same time. Mesh reinforced shotcrete with a minimum thickness of 120 millimetres was installed 3 weeks thereafter.

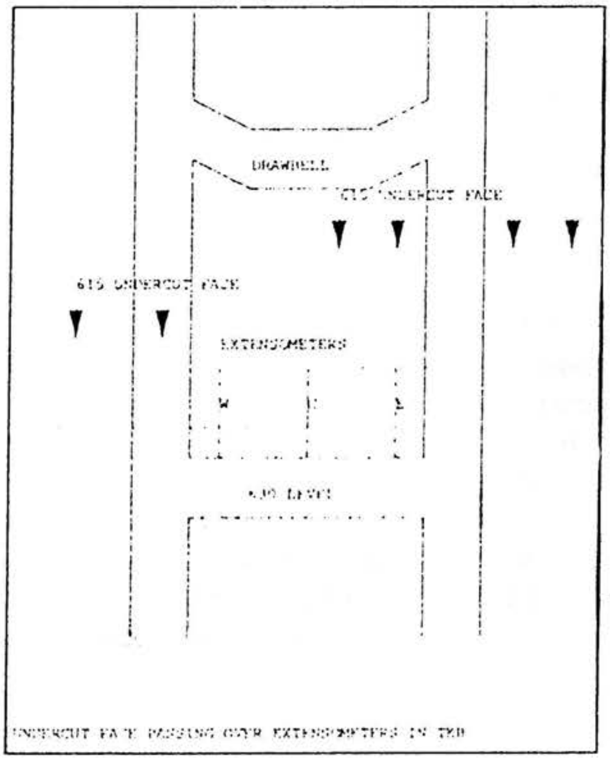



**Figure 5.9. Layout of extensometer holes relative to advancing undercut.**

Examination of the results of the plots from the three extensometers shows that movement occurred at all three monitoring sites from the time of installation. At two sites the rate of movement was initially 5 millimetres per week. The installation of the mesh reinforced shotcrete had the effect of slowing this rate of movement to 0,8 millimetres per week. Convergence continued at this rate for 35 weeks until movement of 35 millimetres had occurred subsequent to the application of the shotcrete. At this stage, cracks were noted in shotcrete. Reversible cracking was now monitored in the rock and this continued at an increasing rate as the shotcrete fracturing increased. The first extensometer (Figure. 5.11.a) was approached by a straight face and displacement was least at this extensometer As the undercut passed overhead, the outer zone (4) fractured and showed large displacement. Deeper into the rock (zones 1,2 and 3) relaxation occurred. At the second extensometer (Figure 5.11.b), which was closest to an advancing "corner", displacement slowed following the application of shotcrete. Crack initiation at a depth of 7 metres was monitored as the undercut passed overhead and the shotcrete lining fractured once displacement of 15 millimetres had been monitored. The two deeper zones (1 and 2) suffered less displacement than zones adjacent to the crosscut, but the zone immediately adjacent to the crosscut (4) suffered less displacement than the deeper zone (3) up until the time that the shotcrete began to fracture and loose strength. The third extensometer (Figure 5.11.c) was situated beneath a "lag pillar" and suffered most displacement. Shotcrete application slowed the rate of displacement, but once the undercut had passed over the area, the rate of strain accelerated and reversible displacements increased in frequency. At all stages, the deeper zones suffered less displacement than zones closer to the crosscut, indicating that the support was not effective. All these extensometers were equidistant from drawpoints that had already been installed. Geology was the same and no major slips or joints were identified in exposure or during drilling. The difference in monitoring results is attributed to the lead and lag geometry of the undercut face as it approached the extensometers.

CASE 3. Displacement history in T13TR33 is illustrated in Figure 5.12.a & b. Stress monitoring in the Tuffisitic Kimberlite Breccia was almost never successful. The graph shows that from the time when monitoring started displacement at depths up to 7 metres into the apex continued at an undiminished rate. Both the sonic probe extensometer installed in the upper corner of the tunnel (Figure 5.12. a) and that installed near the footwall of the tunnel (5. 12. b.) showed dilation of the rock mass into the tunnel excavation. The overstressing and fracturing of the rock mass which caused this dilation resulted in considerable convergence of the entire tunnel which then set up high thrusts in the rigid shotcrete lining. At a displacement of 20 millimetres, the shotcrete lining cracked and was subsequently destroyed by LHD's and secondary blasting. Support needed to be repaired before 15 000 tons had been drawn. Two repair cycles have been carried out in this drawpoint since drawbell development. A total of 50 000 tons has been taken from the drawpoint.

CASE 4: A monitoring station was established in the slot area, in Hypabyssal Kimberlite, after slot cutting had been completed (T25TR29 - Figure 5.13.a & b). This station was never subjected to high abutment stresses. Both vertical and horizontal stress changes were monitored for more than 2 years. The undercut and minor apices were developed together. A negative stress change in the vertical direction of 280 microstrain units and 400 microstrain units was measured on the North and South side, respectively, of the drawbell crosscut as this occurred. Horizontal stress changes of 180 microstrain units and 400 microstrain units were measured at the same time. An irreversible displacement of 30 millimetres was measured shortly after the drawpoint crosscut was developed and a number of reversible displacement spikes of up to 12 millimetres were monitored during 2 years of monitoring. The first graph shows that only the zone of rock within 1,5 metres from the sidewall was affected by these displacements. Magnets installed at 3, 4,5 and 6 metres showed insignificant (<5 millimetres) displacement. The second graph shows that displacement spikes occurred in all four extensometer holes installed around crosscut. In all holes, the displacement was confined to the outer, blast damaged zone of rock. The rock and support in this drawbell suffered minimal damage even after 30 000 tons of ore had been extracted from the drawpoint.

### 5.5.5. INTERPRETATION OF STRESS CHANGES AND DISPLACEMENTS AS RELATED TO DEVELOPMENT OF THE EXTRACTION LEVEL

Stress changes are notoriously difficult to measure accurately in an underground situation. The author does not claim that all stress results are accurate or that these stress changes would be reproducible at an acceptable level of accuracy. Stress was, however, measured at 66 positions and the pattern of stress change correlated well with underground observations and modelled stress changes. Stress change monitoring was therefore judged to be a reasonably accurate measure of actual stress changes underground in the BA5 during the various stages of cave mining.

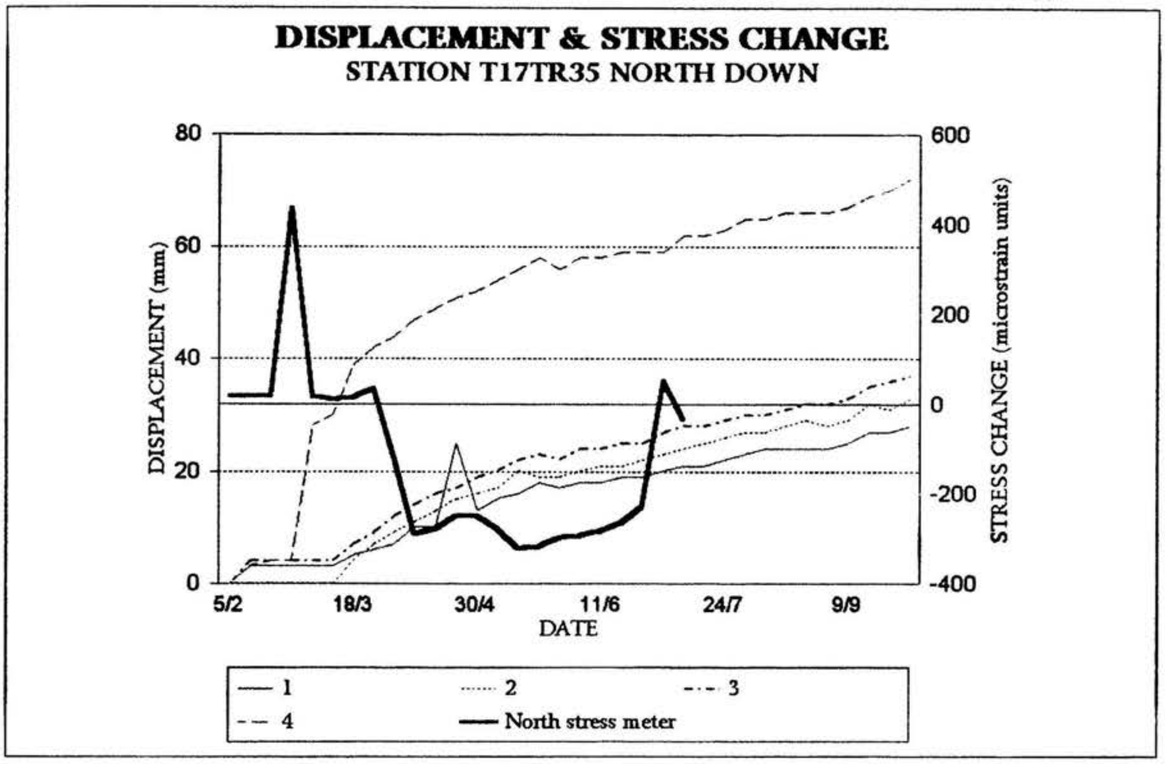



**FIGURE 5.6. STRESS CHANGE AND DISPLACEMENT AT STATION T17TR35**
A stress change of only 500 microstrain units in TKB was sufficient to result in fracturing in the outer zone (zone 4). Thereafter displacements of up to 30 mm were felt 6 metres into the outer apex

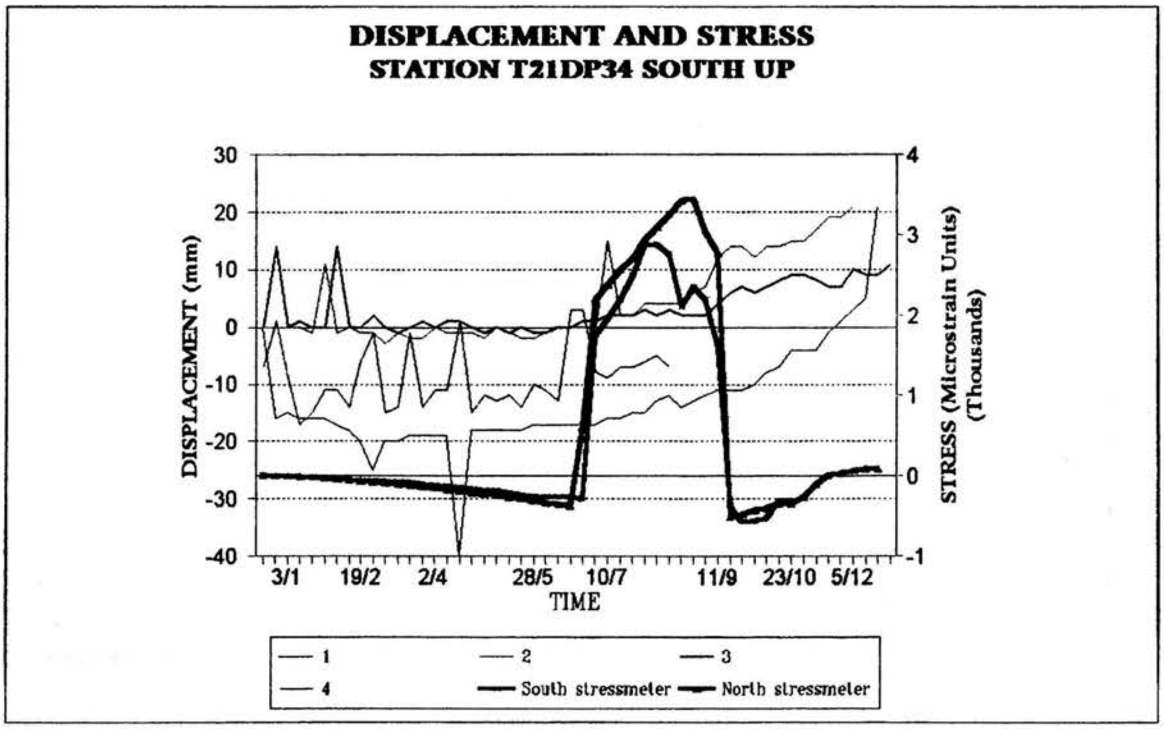



**FIGURE 5.7. DISPLACEMENT AND STRESS CHANGE AT STATION T21DP34 SOUTH UP**
The outer zones for 4 extensometers and 2 stressmeters are graphed. As the undercut passes overhead, stress changes of up to 3000 and 3500 microstrain units are measured. this was accompanied by considerable movement on joints and deformation

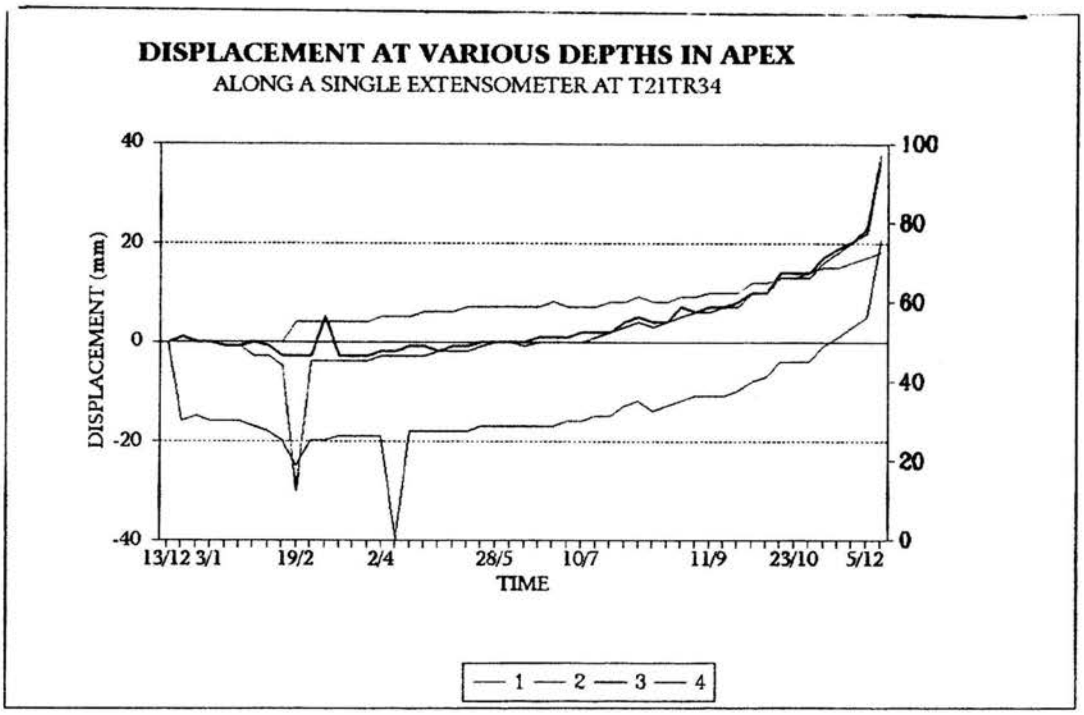



**FIGURE 5.8 DISPLACEMENT ALONG A SINGLE EXTESNOMETER AT STATION T21TR34**
Movement at various depths along a single extensometer. The outer zone (zone 4) shows a total movement of 44mm. The zone deepest into the minor apex (zone 1) shows the smallest displacement (18mm).

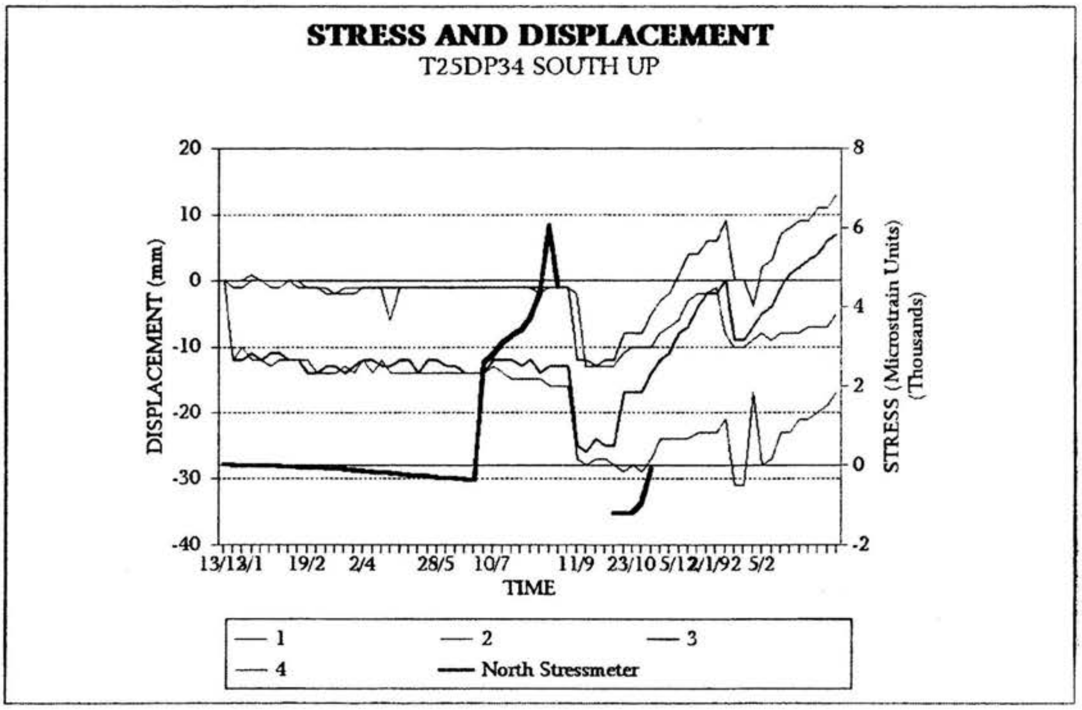



**FIGURE 5.10. DISPLACEMENT AND STRESS CHANGE AT STATION T25DP34 SOUTH UP**
This monitoring station was situated beneath a remnant pillar (stub) and showed extreme stress change (70 microstrain units) and rock deformation up to 6 metres into the minor apex.

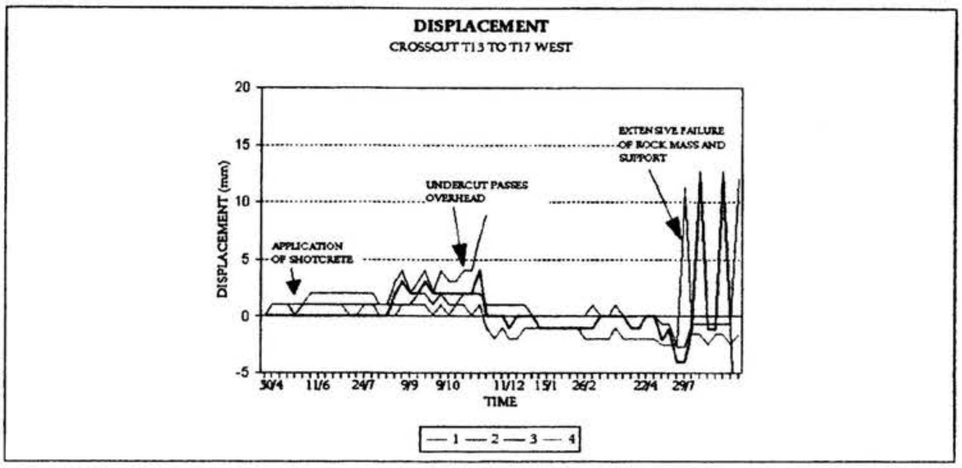



**FIGURE 5.11.a. CASE 1: WESTERN EXTENSOMETER**

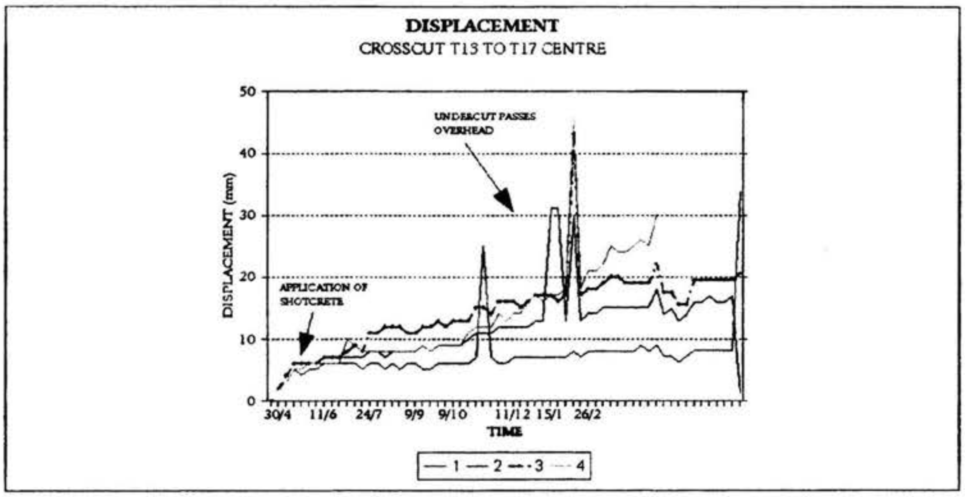



**FIGURE 5.11.b. CENTRAL EXTENSOMETER**

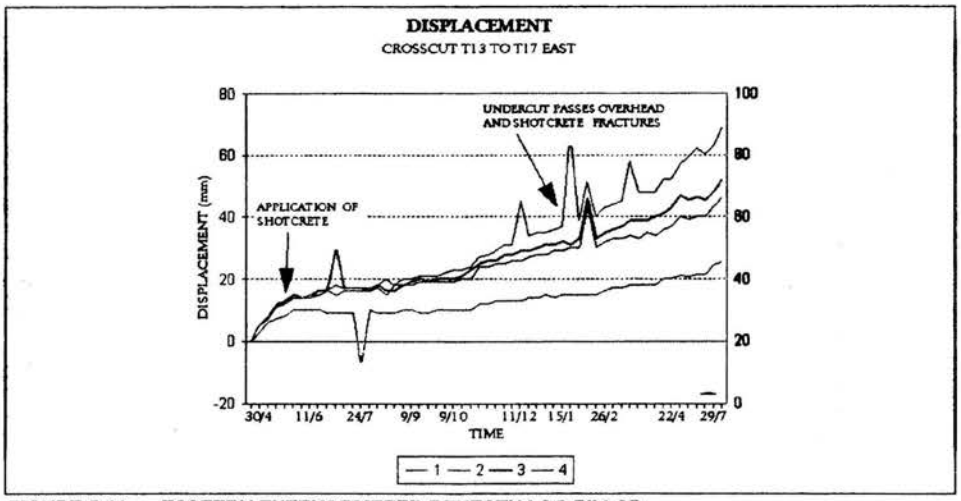



**FIGURE 5.11.c. EASTERN EXTENSOMETER BENEATH LAG PILLAR**

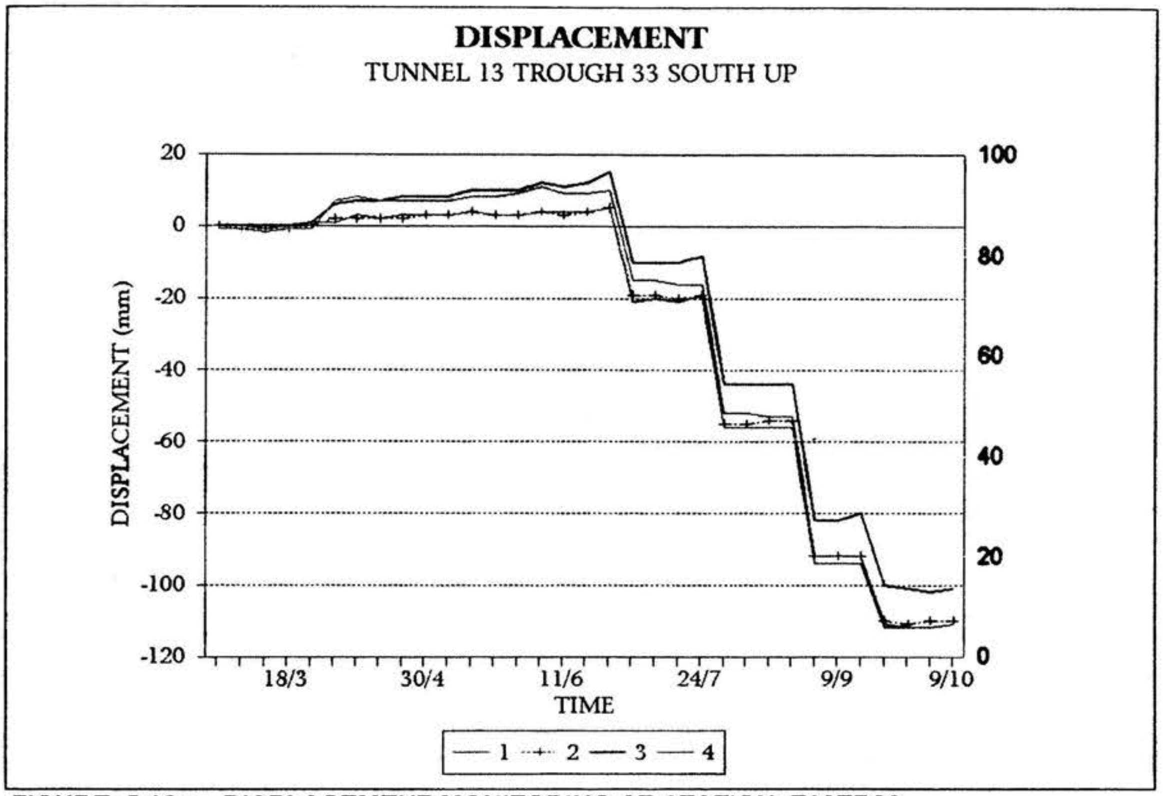



**FIGURE 5.12.a. DISPLACEMENT MONITORING AT STATION T13TR33**
Progressive compression of rock in minor apex that correlated well with ring blasting to advance the undercut face on the level above.

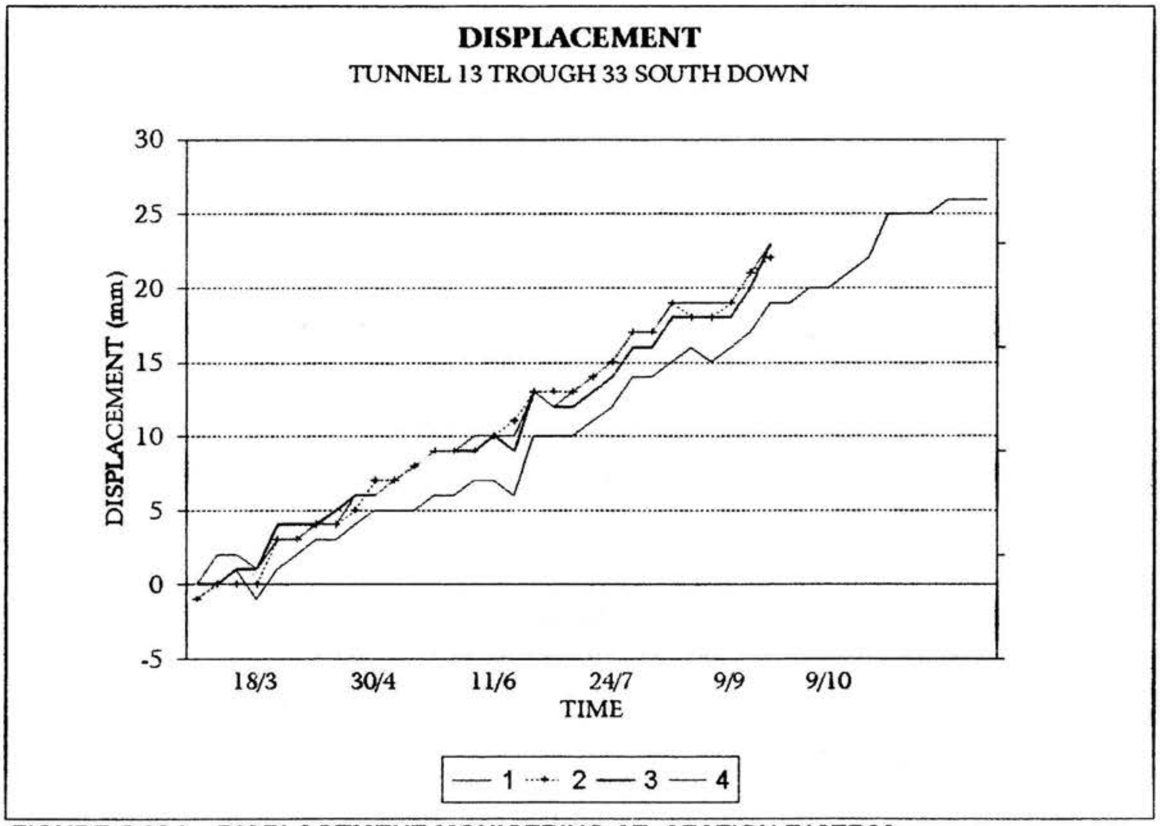



**FIGURE 5.12.b. DISPLACEMENT MONIOTRING AT STATION T13TR33**
Progressive convergence of the rock mass at the base of the tunnel into the tunnel excavation as the undercut advanced on the level above.

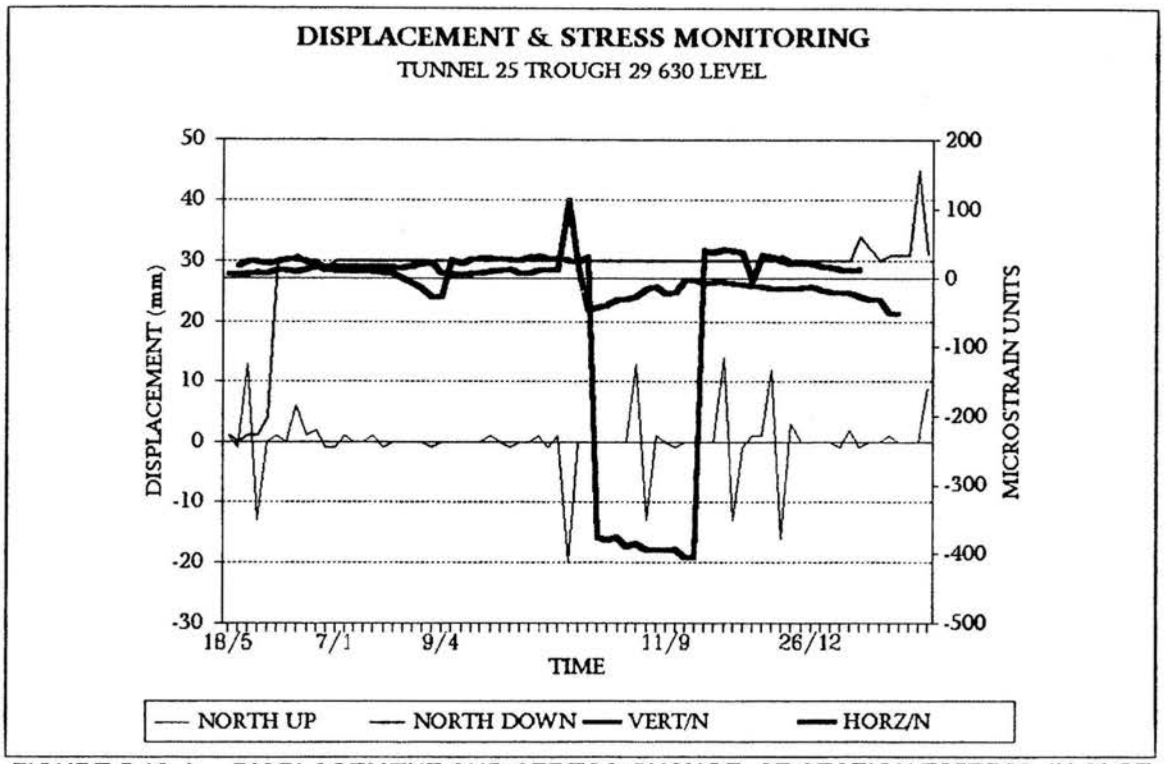



**FIGURE 5.13. b. DISPLACEMENT AND STRESS CHANGE AT STATION T25TR29 IN SLOT AREA**
This area was never subjected to high abutment stress changes. Stress changes equivalent to between 200 and -400 microstrain units were measured as the undercut was initiated above this station. Displacement spikes were confined to the outer, blast damaged, zone, as a result of movement on joints and fractures.

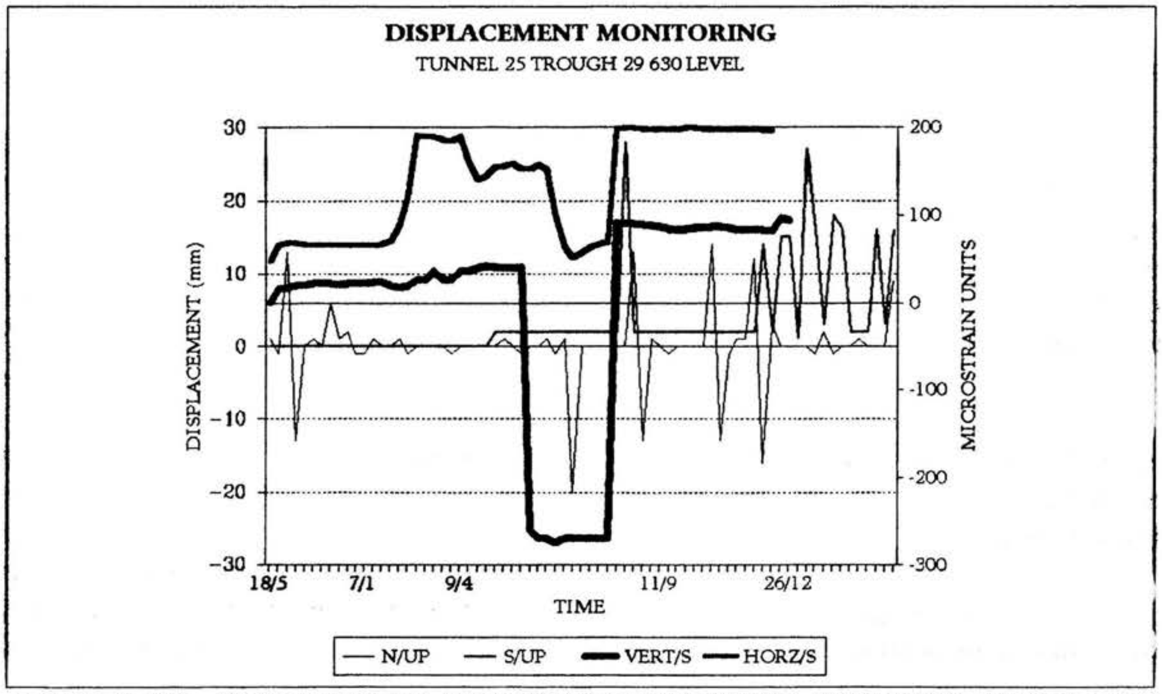



**FIGURE 5.13.b. DISPLACEMENT AND STRESS MONITORING AT STATION T25TR29 IN SLOT AREA**
This area was never subjected to high abutment stress changes. Stress changes equivalent to between 200 and -300 microstrain units were measured as the undercut was initiated above this station. Displacement spikes were confined to the outer, blast damaged, zone, as a result of movement on joints and fractures.

Measurement of displacement underground using sonic probe extensometers is straight forward, involving only the accurate calibration of the probe. The numerous spikes were unexpected but again correlated well with other underground observations and stress change results. Displacement monitoring was therefore judged by the author to be an accurate measure of actual movement underground.

### Tunnel Development

Overall results from the monitoring stations showed that tunnel development on the extraction level consistently resulted in a small stress decrease of around 2 MPa and dilation of the rock towards the excavation. It was simply the result of the rock mass relaxing as occurs after development in most rock types and is not peculiar to the cave mining method. In most areas final support was installed before instrumentation was installed in the drawpoints. To monitor the relaxation of the rock around excavations, separate monitoring stations were established. In weak Tuffisitic Kimberlite Breccia, support was not sufficiently stiff to prevent some relaxation of the rock toward the excavation. This was accompanied by a negative stress change which resulted in some of the severest fracturing monitored both in terms of amplitude of the cracks and frequency of cracking. Cracking was confined to the outer, blast-damaged zone of rock adjacent to the excavation. The brittle failure was attributed to tensile failure, typical Hoek and Brown failure in weak, jointed/fractured rock with low or no confining stresses and to shear failure on existing joints. An upper bound of 3 MPa has been calculated for the shear strength of joints in the kimberlite (Howell et al., 1993). This emphasised the urgency of installing support as soon as possible after tunnel development especially in weak Tuffisitic Kimberlite Breccia.

### Drawbell development

Drawbell development lead to unexpectedly severe stress changes being monitored in the minor apices. Drawbell development was often carried out well ahead of the undercut abutment and these stress changes were the result of the manner in which the drawbells were originally developed (see Figure 5. 14). The legal requirement is that drilling may not be carried out within 2 metres of a previously charged hole. This resulted in a series of 2 metre wide pillars between drawbells until holes from both drawbells were drilled prior to charging. Stress changes measured in the minor apex during drawbell development ranged from -10 MPa to 15 MPa (0 MPa to 25 MPa total stress). If drawbell cutting resulted in an excavation above the minor apex, an abrupt stress decrease (-10 MPa) was monitored. If drawbell cutting resulted in a narrow pillar being left in place above the minor apex, an abrupt stress increase (up to 15 MPa) was monitored. If this pillar was allowed to remain intact as a remnant pillar, large stress increases were monitored (Figure 5.14). As these remnant pillars interfered with the undercutting process, every effort was made to destroy and avoid their creation and their occurrence became less with time. These minor apices were minimally supported as the drawbell tunnel was only required for access to allow drilling and blasting of the drawbell. The severe stress changes, blast damaged rock consequent on tunnel development, and subsequent ring blasting, together with minimal support resulted in extensive damage to these minor apices. Monitoring showed extensive brittle failure and strain softening of the apices, even in competent Hypabyssal Kimberlite. This was attributed to shear failure along joints in the Hypabyssal Kimberlite, and shear failure of joints and intact rock in the weaker Tuffisitic Kimberlite Breccia.

unstable displacements. These high stresses were felt up to 10 metres ahead of the abutment in Tuffisitic Kimberlite Breccia and 5 metres in Hypabyssal Kimberlite. Once the abutment had passed overhead stress levels decreased abruptly so that by the time the abutment was 2 metres past the monitoring station stress levels had decreased to pristine values. The apex showed little further deformation until the effects of drawing in the ore column started to occur. The abrupt stress decrease is attributed to the inability of the kimberlite breccia to effectively transmit stress and to the geometry of the production level layout that created a series of independent pillars.

It should be noted that total stress rather than stress change is the determinant of rock mass behaviour. In the BA5 virgin stress values were between 10 and 12 MPa. A stress change of 30 MPa therefore represented a total stress acting on the rock mass of 40 MPa. As stress change rather than total stress was measured at monitoring stations, stress change rather than total stress is reported in the thesis.

There was usually good correlation between crack formation and stress levels, with stress increases preceding crack formation. Often, but not always, crack formation resulted in a decrease in stress. Continuous caving initiated during the week preceding the monitoring that was done on the 19th June 1991. Large stress changes of up to 30 MPa were monitored at most stations around the undercut. This was accompanied by major cracking and several instruments were lost as holes sheared closed. It was only when this occurred that the undercut face became a true abutment. Prior to this, stress changes were typically less than 6 MPa and displacements in the rock affected by the stress changes took the form of movement and dilation in the blast damaged zone around excavations. Once caving initiated, stress levels increased to at least 20 MPa and resulted in widespread induced fracturing ahead of the undercut face and in apices on the extraction level as the undercut passed overhead.

Table 5.3.is a summary of the damage scale experienced in the kimberlite types of the BA5. Rock and support damage can be correlated with stress changes and displacements. The pattern of damage is repetitive and amenable to interpretation. The monitoring programme has shown that the support is generally effective and Class 5 damage has not yet been observed in any of the 125 drawpoints that have been established to date in the BA5 or in the overlying undercut drilling tunnels.

### 5.5.6. DETERMINATION OF SUPPORT EFFECTIVENESS

Based on the monitoring data, criteria were established that allowed an objective assessment of the effectiveness of the installed support in the various rock types monitored.

The aim of the installed support system is set out below.

1. The objective of an active support system is usually to create a reinforced ring around the excavation so that the ring becomes self- supporting. The ring of reinforced rock around the excavation should limit convergence to levels that can be sustained by the installed support system. The reinforced zone should also limit the propagation of the fracture zone.
2. Interbolt support should stop extensive ravelling and brittle failure in the zones closest to the excavation. It means that reversible and irreversible displacements should be

**Table 5.3. Damage Scale**

| DAMAGE SCALE | SUPPORT DAMAGE | ROCK DAMAGE | MEASURED DISPLACEMENT | STRESS CHANGE |
|---|---|---|---|---|
| 1 | Minor shotcrete damage | Block fallout | 0-5 mm | 0-2 MPa |
| 2 | Shotcrete damage | Block fallout | 0-15 mm | 0-2 MPa |
| 3 | Major shotcrete damage | Minor brittle failure | 15-50 mm | 2-5 MPa |
| 4 | Destruction of linings. Some damage to steel tendons | Brittle failure | 20-80 mm | 5-20 MPa |
| 5 | Tunnels crushed | Extensive rock failure around excavations | 8--300 mm | >20 MPa |
| 6 | Large scale cracking of rigid linings and ravelling around steel tendons | Rock disintegrates and ravels with time | 80-1000 mm | small |

infrequent and limited to sustainable levels. Where extensive brittle failure near the

surface of the excavation was monitored closer pattern bolting using 1,8 metre long steel tendons together with an adequate mesh-reinforced shotcrete lining was recommended. An adequate lining included increasing the strength of the steel wire in the mesh and ensuring that mesh apertures were large enough to prevent "blinding" by shotcrete, which resulted in gaps behind the mesh.

3. Total displacement or convergence should be limited to levels that can be sustained by the most rigid element in the support system which, in the support system used at Premier, is shotcrete or concrete. Laboratory testing and monitoring has shown that, for shotcrete, this means displacement of 50 millimetres or less. Where total displacements of 50 millimetres or more were monitored, support and rock damage was extensive and any positive stress change thereafter resulted in convergence. Displacements could only be limited to sustainable levels by increasing the thickness of the reinforced ring by installing long cable anchors on a closer pattern and adding a thick shotcrete or concrete lining to provide the required support pressure to the excavation sidewall. Monitoring and observation showed that support was inadequate in many areas when post undercutting was the chosen undercutting sequence. These areas were stabilised by installing the massive, passive support in the form of concrete linings, usually at a high cost in terms of lost production, labour and materials. This experience was the main reason for changing the mining sequence to an advance undercut.

Monitoring data at each site was judged for effectiveness in terms of these criteria. If overall displacement was less than 50 millimetres but reversible and irreversible displacements were frequent and/or more movement was measured in the zones closer to the excavation than away from the excavation interbolt support was judged to be inadequate. Based on this criteria it can be seen from Table 5.4. that support installed in Hypabyssal Kimberlite at monitoring station T21DP34 was effective. Table 5.5. shows that support installed at monitoring station T17DP35 in Tuffisitic Kimberlite Breccia was inadequate.

Data from the various monitoring stations has been summarised in a series of tables similar to Tables 5.4. and 5.5. and used to assess support effectiveness, in the vicinity of these monitoring stations. These tables can be found in Appendix 1.

### 5.5.7. THE USE OF MONITORING DATA TO IMPROVE SUPPORT DESIGN

Monitoring data from the sonic probe extensometers was analysed to determine the actual zone of failure. Four zones were defined in terms of the sonic probe data and the installed support system. On the extraction level, in Tuffisitic Kimberlite Breccia, steel tendon support consisted of 1,8 metre long 12 ton rockbolts with an effective embedment length of 1,6 metres, installed on a 1 metre spacing and 25 ton cable anchors with an effective embedment length of 5,5 metres installed with a spacing of 1 metre. A grouted rock anchorage may fail in one or more of the following modes:

* failure within the rock mass
* failure of the rock/grout bond
* failure of the grout/tendon bond
* failure of the steel tendon or anchor head

Observation showed that rockbolts and cable anchors very seldom failed and that failure of the rock mass usually occurred only after water or unusually high stresses had affected the kimberlite. Failure at the rock/grout bond is the most common failure mode followed by failure at the grout/tendon bond. In order to define an index for the effectiveness of the installed tendon support a uniform bond distribution is assumed and the pull out capacity ($T_f$) can be calculated from the equation (Littlejohn, 1992):

$$T_f = \pi D L \tau_{ult}$$

where $\tau_{ult}$ = ultimate bond at the rock/grout interface
$L$ = length of fixed anchor
$D$ = diameter of fixed anchor

This gave a rock grout/rock bond strength of 800 kN/m for Hypabyssal Kimberlite and 500 kN/m for Tuffisitic Kimberlite Breccia for grouted rebar and 600 kN/m and 380 kN/m for cable anchors in the respective rock types. Similar values were obtained for the grout tendon interface. It has been shown that bond strength is at least a partial function of confining stress (Kaiser, 1992) and it can be expected that bond strength deeper into the rock mass will be greater than near the surface of excavations where low confining stresses can reduce bond strength by between 30 percent and 60 percent. These effects were ignored as values were only used as indices for comparative purposes.

**Zone 1** is closest to the excavation sidewall and extends from the first magnet to the second magnet. It is supported with shotcrete, 1,8 metre long rockbolts on a 1 metre spacing and cable anchors on a 1 or 2 metre spacing and extended, on average, 2,5 metres into the rock. Numerical modelling showed that the confining stress at a depth of 1,25 metres into the sidewall is 5 MPa whilst reinforcement, in terms of steel tendon density (rockbolts and cable anchors), is calculated at 252 kN/m/m$^3$ for Tuffisitic Kimberlite Breccia and 400 kN/m/m$^3$ for Hypabyssal Kimberlite. Mesh reinforced shotcrete provides a support pressure of 1 MPa for both rock types.

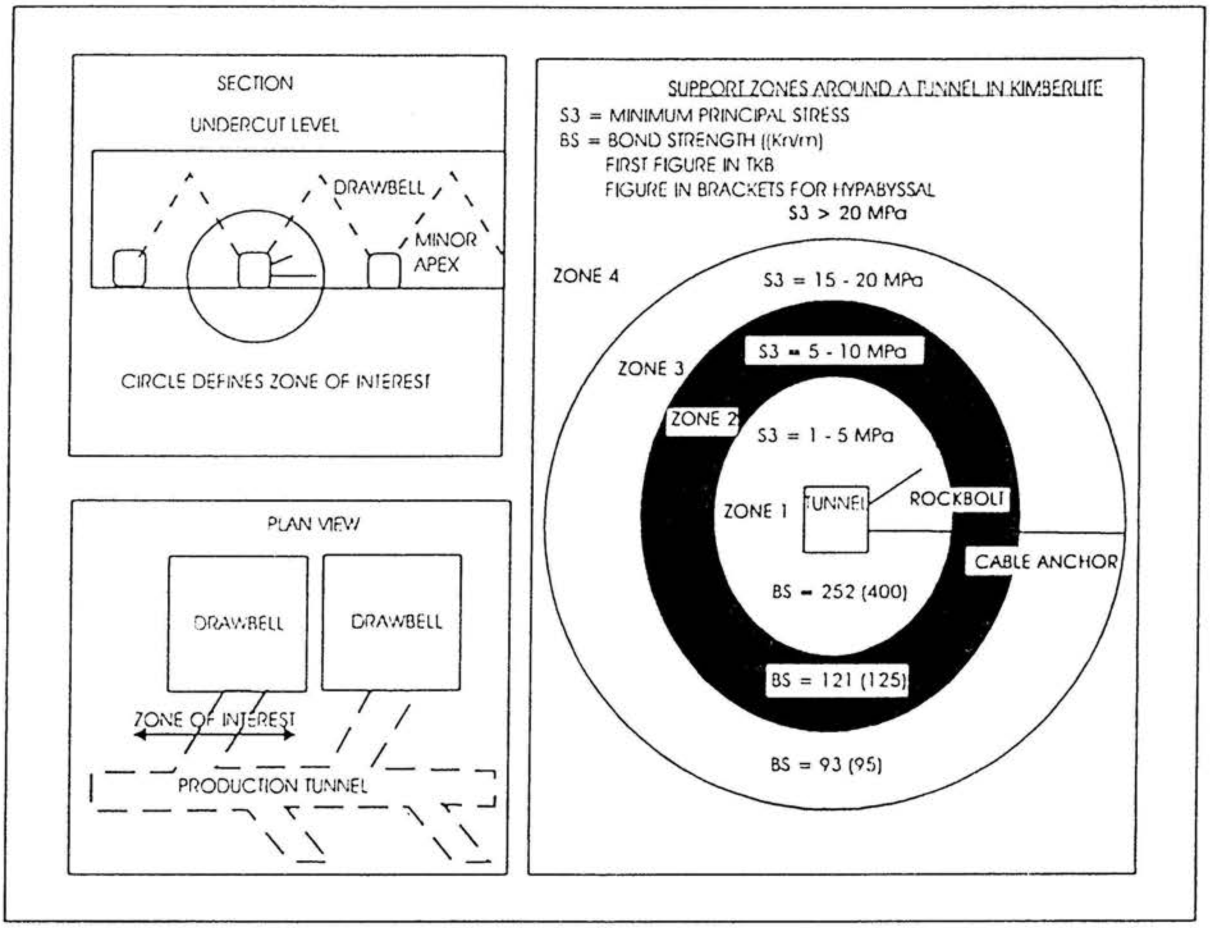



**Figure 5.15. Support Zones around a Tunnel**

**Zone 2** extends from the second to the third magnet and is supported with cable anchors on a 3 metre spacing radially and (2 metres along the length of the tunnel) and extends from 2,5 to 4 metres into the rock. Modelled confining stress at the centre of this zone is 10 - 15 MPa and tendon density is 121 kN/m/m$^3$ for Tuffisitic Kimberlite Breccia and 128 kN/m/m$^3$ for Hypabyssal Kimberlite.

**Zone 3** extends from the third to the fourth magnet at an average depth of 4 to 5,5 metres into the rock and is supported with cables on a 4 metre spacing radially. Modelled average confining stress for the zone is 15-20 MPa and tendon density 93 kN/m/m$^3$ for Tuffisitic Kimberlite Breccia and 98 kN/m/m$^3$ for Hypabyssal Kimberlite.

**Zone 4** extended from the fourth magnet to the reference magnet at an average depth of 5,5 to 7 metres into the rock mass and is not supported. The modelled confining stress is 25 MPa and reinforcement 0.

The above calculations show that, as a result of the greater rock strength of the Hypabyssal Kimberlite the Hypabyssal Kimberlite is more effectively supported than the Tuffisitic Kimberlite Breccia. The calculation also suggests that support in zone 1 is twice as effective as in zone 2 and 3 for Tuffisitic Kimberlite Breccia, and at least three times as effective for Hypabyssal Kimberlite. Even if the effect of confining stress at depth is taken into account, tendon support in zone 1 remains more effective than in zone 2 and 3, on this simplified, comparative basis. The defined zones, minimum principal stresses calculated for these zone and bond strengths relative to the installed rockbolts and cable anchors are illustrated in Figure 5.15.

**Table 5.4. Typical displacement and stress change results in Hypabyssal Kimberlite**

| HYPABYSSAL | STRESS CHANGE (MPa) | DILATION (mm) | | | | |
|---|---|---|---|---|---|---|
| T21DP34 | | 0-2,5m | 2,5-4m | 4-5,5m | 5,5-7m | MAXIMUM |
| NU | 28 | 18 | 38 | 38 | 36 | 38 |
| ND | 28 | 17 | 20 | 18 | 20 | 20 |
| SD | 34 | 13 | -7 | 7 | 12 | 13 |
| SU | 34 | 11 | 10 | 14 | 12 | 14 |

SUPPORT EVALUATION
1. Outer zones converge by less than deeper zones
2. Minor brittle failure
3. Total convergence less than 50 mm
CONCLUSIONS: SUPPORT GENERALLY EFFECTIVE
RECOMMENDATION: INCREASE ROCKBOLT DENSITY TO STABILISE OUTER ZONE. CABLE ANCHORS NOT NEEDED

**Table 5.5. Typical displacement and stress change results in TKB**

| TKB | STRESS CHANGE (MPa) | DILATION (mm) | | | | |
|---|---|---|---|---|---|---|
| T17DP35 | | 0-2,5m | 2,5-4m | 4-5,5m | 5,5-7m | MAXIMUM |
| NU | 4.2 | 13 | 17 | 18 | -50 | -50 |
| ND | 4.2 | 28 | 33 | 37 | 72 | 72 |
| SU | -14 | 6 | 10 | 14 | 160 | 160 |
| | | | | | | |

SUPPORT EVALUATION
1. Greatest dilatation nearest tunnel
2. Minor brittle failure
3. Total dilation 160 mm
CONCLUSION: INEFFECTIVE IN LIMITING CONVERGENCE TO WITHIN STABLE LIMITS.
RECOMMENDATION: ADVANCE OR PRE-UNDERCUT TO BE INSTALLED

The above observations and calculations showed that it was far more effective to increase tendon density than to increase tendon length. The use of steel cable anchors was limited to problem areas, resulting in large cost savings and an improvement in support performance.

Support damage monitoring showed a consistent pattern of support damage. Typically, kimberlite immediately adjacent to the tunnel sidewalls was extensively fractured as a result of high stresses and blast damage. The hangingwall was little damaged. If the sidewall fracture zone was not effectively confined by effective interbolt support in the form of an adequate shotcrete lining, the fractures propagated. The rigid lining, however, often failed as a result of high stresses. These observations led to the design of flexible steel tendon support over the entire tunnel profile. Only the sidewalls, however, were shotcreted. This created effective interbolt support where it was required, but did not complete an arch. High thrusts could not develop as the shotcrete lining was not continuous.

## 5.6. NORITE WALLROCK

The norite is a well jointed, 400 metre thick, igneous rock layer that surrounds the kimberlite orebody on both the undercut and extraction levels. All service excavations such as shafts, workshops and offices, as well as numerous orepasses are sited in this rock. Fifty percent of the 27 000 metres of development needed for the BA5 cave was developed in norite.

Three types of norite are recognised. The type of norite is generally correlated with distance from the kimberlite pipe contact. Immediately adjacent to the pipe contact is a zone of decomposing norite where the norite has been weathered to secondary minerals. This zone is not continuous along the pipe contact and seldom wider than 5 metres. Decomposing norite behaves in a plastic manner when wet and can lead to extreme squeezing ground conditions.

A zone of blocky norite exists along the pipe contact or behind the decomposing norite, if this is present. The zone of blocky norite varies in width from 15 to 50 metres. The seven joint sets present in the norite dictate its blocky nature. Secondary talc and chlorite mineralisation form a smooth, slickensided gouge on joint planes in the norite close to the pipe contact and in shear zones away from the pipe contact. The norite immediately adjacent to the joints is also weakened by weathering, usually as result of the orthopyroxene, clinopyroxene and feldspar being altered to clay. The cohesion and angle of friction on the joint planes is determined by the characteristics of the gouge, but even rough joint planes and asperities on these planes are formed by weak rock. Joint planes generally have an extremely low angle of friction. Stress changes of the order of 2 MPa are sufficient to cause shear failure along the persistent, well defined joint sets in the blocky norite. This causes deformation of the tunnel and block fallout in the norite up to 100 metres from the undercut face. If water is allowed to permeate the joints in the blocky norite, the gouge plasticises and the angle of friction on the joint planes reduces further. Block fallout of unpinned blocks is common and tunnels collapse as a result of large deformations. In dry, blocky norite, damage rated at DS3 was seen. In the presence of water, this routinely increased to DS5 and tunnels were lost.

The competent norite is well jointed with four well developed and two poorly developed sub-vertical joint sets and a sub-horizontal joint set. Typically little or no secondary mineralisation develops along joint planes and the joint surface can be described as rough-planar.

In the well jointed norite, stress changes of up to 4 MPa were felt as far as 60 metres from the undercut. The stress induced by the undercut was inclined and had the effect of deforming the tunnel. Deformation results in shear movement along joints. This in turn results in fallout of unfavourably oriented, unbolted blocks. Development support in most tunnels in the norite consisted of 1,8 metre long rockbolts on a 1,5 metre staggered spacing. Most shear movement was noted in the hanging on the side of the tunnel away from the undercut. Where the norite was supported by mesh reinforced shotcrete, some 30 metres from the undercut minor stress fractures were noted along the hinge line of the tunnels. This level of rock and support damage was assigned a damage scale of 1. Recommended support to prevent this type of damage was 6 metre long fully grouted, tensioned 25 ton 6 metre long steel cable anchors to stabilise the rockmass and more intense interbolt support to prevent block fallout between the widely spaced development rockbolts. It was usually sufficient to increase support only on that side of the tunnel that showed incipient "dogearing". The norite is traversed by occasional, near-vertical, shear zones. Block size within these shear zones is small. Extensive block fallout in these zones as a result of the increased stress resulted in closure of the tunnels which had to then be resupported. One hundred millimetres of mesh reinforced shotcrete, together with rockbolting decreased to a 1 metre spacing and grouted, tensioned, cable anchors on a 2 metre between-ring spacing and 1 metre within-ring spacing was sufficient to stabilise most shearing along joints. Corners were strapped using ten 40 ton cable anchors at a 300 millimetres horizontal spacing. Anchors were tensioned to 8 tons and covered with shotcrete to prevent LHD damage. As the undercut advanced, stress fractures along the hinge line formed, but seldom developed beyond a damage scale of 1.

In summary: a RMR can be calculated for each of the norite types. Stress change as the result of the advancing undercut results in shear movements along joints and affects joint condition. The norite is not subjected to all the stress cycles noted in the kimberlite, but must sustain relatively high stress changes due to the advance of the undercut in the adjacent ore. These stresses must be carried by the norite wallrock for a considerable period as the adjacent orebody is mined out. Blocky norite is affected by greater shear movement along joints, plasticisation of gouge and possible ingress of water. The intact norite is a competent rock and instability is almost entirely correlated with joint attributes. The adjusted RMR can be correlated with the distance to which stress is felt and the intensity of support damage.

## 5.7. PROBLEMS

Several problems that beset mining in the BA5 are discussed below.

1. The most time consuming, expensive and destructive problem was incomplete undercutting. Initially it was the result of minor apices not being undercut during drawbell development on the extraction level. Highly stressed pillars, often surrounded by broken ore, were formed. Once this problem had been overcome and continuous caving had initiated, ring drilling on the undercut had to be carried out in a choke blasting situation and it was impossible to check that complete undercutting had been achieved. In both cases, the probability of unidentified remnant pillars remained and, where this occurred, it resulted in damage to excavations on the extraction level. When the remnant pillar (stub) was identified, this had to be drilled and blasted. Drilling to wreck the pillar was complicated by high stresses which resulted in the loss of drillholes. The delay in advancing the undercut resulted in further rock damage which often included footwall heave. Planned leads and lags between adjacent drilling tunnels were difficult to maintain. This increased rock and support damage further.

Incomplete undercutting could only be remedied by carefully checking that all holes were drilled and charged as planned. Problems with the closure of drillholes were overcome by drilling, charging and blasting the ring in the same shift. In some holes, plastic tubing was inserted immediately after drilling which allowed the hole to be charged at a later stage.

2. Drawpoint crosscuts had to be profiled, and the resultant bullnoses and camelbacks supported as the production drift was developed. Failure to do so allowed blasting fractures to develop to a depth of between 2 and 3 metres into the rock surrounding the tunnel. If crosscuts were developed through these blasting fractures, bullnoses disintegrated and had to be reconstructed.

3. Block fallout was experienced up to 60 metres ahead of the undercut in Tuffisitic Kimberlite Breccia, and up to 30 metres in the Hypabyssal Kimberlite. Induced fractures were noted up to 10 metres ahead of the abutment zone on the extraction level in some areas. This meant that a zone of at least two, and sometimes four drawbells had to be fully supported ahead of the abutment zone. As the size of the undercut increased, this became a most onerous constraint. If the undercut geometry had been more carefully planned, fewer drawpoints would have had to be supported in advance of the undercut and undercut stresses could have been reduced.

4. In weak rock, footwalls had to be fully supported with cable anchors and rockbolts, together with concrete and a steel rail construction, to ensure good roadways for LHD tramming. As much as two metres of steel reinforced concrete often exhibited footwall heave as a result of remnant pillars and/or a slow moving undercut.

5. The undercut was initiated in the centre of the cave and production was often required from fully supported and operational drawbells at the centre of the cave, whilst other drawpoints in the same tunnel still needed to be developed and supported. It was impractical to schedule these two operations so that they did not interfere and support or production inevitably suffered.

# CHAPTER 6

# UNDERCUTTING IN A MECHANISED CAVE

**Statement:**

Undercutting is the most important aspect of any cave mining operation. Undercut drilling and blasting is inherently difficult, as drilling must usually be done in fractured rock and choke blasting conditions often prevail. The creation of remnant pillars, large leads and lags between adjacent tunnels and an undercut face that moves too slowly can result in extensive rock mass and support damage on the extraction level below. Undercutting operations can similarly impact on caving of the oreblock, affecting fragmentation and abutment stresses. Airgaps that can result in wedge failures, uncontrolled caving and even airblasts, can be created.

The objective of undercutting should be to undercut the oreblock to be caved as effectively as possible. This implies that undercut drilling and blasting should be as simple as possible, undercutting operations should allow the undercut face to advance at the planned rate, using the planned undercut face configuration. The mining sequence and rate of advance of the undercut face should be determined by the rock mass rating of the underlying extraction level. Practical mining and geotechnical considerations should define the undercutting operation. Early production tonnage from the cave should not conflict with these considerations.

In the BA5 mining block a 21 metre high undercut was planned to achieve production tonnage from the cave as soon as possible. Post undercutting was planned as this minimised development and meant that a costly ore handling system did not need to be installed on the undercut level. These decisions created numerous, costly geotechnical problems. It was the task of the author to analyse and quantify the cost of these problems and offer cost effective solutions for future undercuts.

This was done by assessing the results of stress change and support damage monitoring. Various undercut heights using different drilling and blasting patterns, different undercut layouts, mining sequences and configurations for the undercut face advance, were all considered and evaluated by the author and the Planning Department. Experience on both Premier and other cave mines was used to plan alternative ways of undercutting. The author then commissioned Itasca (McKinnon, 1992) to undertake numerical modelling to plan an optimum shape for the undercut advance and assist with the evaluation of support used on the undercut.

This chapter reviews the mining and geotechnical implications of undercutting sequences, undercut height and shape of the undercut face advance. The problems that were experienced at Premier as a result of developing a high undercut using a post undercut mining sequence, monitoring results and support recommendations are discussed.

## 6.1. INTRODUCTION

In any cave mining operation an area sufficiently large to induce continuous caving of overlying ore must be created. The area needed to induce the continuous caving can be estimated from experience or, where this is not available or where stress levels or rock conditions change, the area can be calculated using empirical correlations that have been developed between rock mass ratings and the hydraulic radius needed to induce caving. The most widely used correlation for cave mining is that developed by Laubscher (Laubscher, 1995). Barton's NGI rating has been correlated with the hydraulic radius to determine the stability of crown pillars in open stopes in several Canadian mines and this correlation has been extended to cave mining situations (Stewart, 1993). Numerical stress models such as FLAC or joint related models such as UDEC or QUAD can be used to predict hydraulic radius. Rock mass parameters must be accurately defined and the mode of failure anticipated before these models can be used. Numerical stress models have the advantage that they can be used to model stress effects more accurately than rock mass ratings but margins of error remain large (Howell et al., 1993).

## 6.2. METHODS OF UNDERCUTTING

The undercutting operation has a pronounced effect on the effectiveness of the cave operation in terms of the damage to tunnels and tunnel support on the production level (Cummings et al., 1984) and can influence stress levels in the cave back, thereby impacting on fragmentation (McKinnon, 1992). Overdrawing undercut ore can lead to uncontrolled caving and airgaps. There are three approaches to the timing of the undercut relative to development of the extraction level.

**Post-undercutting.**

Undercut drilling and blasting takes place after development of the underlying production level has been completed. Cones, drawbells or continuous troughs are prepared ahead of the undercut and are ready to receive the ore blasted from the undercut level. This approach has been adopted at El Teniente, Salvador, Andina and in the original layout in the BA5 at Premier Mine.

Advantages of the method are:

* No separate ore handling facility on the undercut level is needed.
* Tunnels on the undercut level are required for drilling and blasting only and can be a considerable distance (30 metres) apart. Development on the undercut level is reduced.
* Ore can be pulled through the drawbells that have been prepared on the production level. The probability of compaction of the broken ore is negligible.

Disadvantages are:

* The extraction ratio on the production level using 4 metre x 4 metre tunnels on a 15 metre x 15 metre drawpoint spacing is 43 percent and rises to 50 percent 3 metres below the undercut level as a result of drawbell development. Blasting damage, overbreak and overstressing can lead to extensive fracture propagation in the blast damage zone around excavations as the undercut passes overhead. This increases the planned extraction ratio. Pillar attribution theory shows that stress levels will be doubled in the remaining rock as a result of drawbell development.
* The rock between the undercut and production level is subjected to high and variable stress levels. It causes strain and displacements which result in damage to, and strain

softening of, the rockmass. It also decreases the in situ rock modulus and can damage rigid support elements such as shotcrete or concrete linings. Rock damaged by induced stresses often allows aggravated wear by subsequent mining activities and demands massive, and continuous, support rehabilitation.

* Support must be installed well ahead of the abutment zone. In weak rock it can mean that a radius of support of a hundred metres or more must be completed before the approach of the undercut. The installation of this support can be a major constraint on the rate of advance of the undercut level. A phased approach to support is possible so that the final application of rigid linings is delayed until the undercut has passed over the area (Cummings et al., 1984). In mechanised mining the shotcrete lining serves to protect the steel reinforcement from LHD damage and secondary blasting and must be installed promptly to be effective. This can preclude a phased approach to support.
* Ore can be extracted faster than the rate of caving resulting in a void developing between the ore in the draw column and the cave back. The ore in the undercut excavation cannot act as an effective "rockfill". The potential for damaging wedge failures into the airgap is high.
* Where the undercut height is great, ore from the draw column can continuously flood the undercut drilling tunnels and hamper charging and blasting of the ring drilling required to retreat the undercut.

**Pre-undercutting.**

The undercut can be completed ahead of development on the production level as a completely separate operation. This approach has been used in the past for several block caves in Kimberley and Premier. No major mechanised mine uses the approach currently although Premier has installed small caves on the eastern side of the mine 15 metres below areas that were partly stoped out. The time lag between stoping and implementation of the cave below was 2 or more years in most areas.

Advantages of this approach are:

* The production level is developed in a destressed situation. Potentially damaging abutment stress loads are avoided. It is important where large stress changes are anticipated as a result of the depth of mining or a great lift height. At El Teniente, tunnel collapse necessitated that a second extraction level be developed immediately below the collapsed area, in destressed rock. On the level above, in rock that had been damaged by undercut stresses, only a few thousand tons had been drawn resulting in considerable brow wear. Drawpoints developed in the undamaged rock below pulled up to 150 000 tons with no rock damage or brow wear recorded (D.H. Laubscher - personal communication).
* The extraction ratio on the production level, prior to undercutting, is zero.
* Support on the production level does not have to be done ahead of the abutment zone and is not a constraint on the rate of retreat of the undercut. The level of support required when the extraction level is developed in destressed rock is lower.
* Ore cannot be drawn from the production level. This allows the ore to be subjected to stress and allows comminution. The broken rock on the undercut level acts as a rockfill and reduces stress on the undercut face. Airgaps cannot be created by overdrawing.

* The undercut level can be mined independently of the extraction level.

Disadvantages are:

* A separate ore handling facility has to be developed on the undercut level.
* Tunnels are required for drilling and blasting as well as loading purposes. Experience and theory suggest that if compaction is to be avoided, ore must be drawn as soon as it has been undercut or, that a considerable volume of ore must be abstracted on the undercut level. Tunnels must be developed at a maximum distance of 15 metres apart to allow drawzones to interact and ensure that complete undercutting is achieved. It usually requires additional development compared to that needed for post undercutting. Theoretically, a situation can arise where ring blasting throws broken ore beyond the reach of loading equipment. This ore can compact and create stress problems when the extraction level below is developed subsequently.
* Rock around the undercut to a distance of 15 metres or more can be damaged by the advancing undercut. Drilling must often be undertaken in highly fractured rock. Pre-drilled holes can be lost. At least two mines - El Teniente and Du Toit's Pan - are currently experiencing drilling and blasting problems with their undercut rings. If stubs are left, resultant stress problems can be severe.
* Compaction of the broken ore on the undercut level is a potential problem. At Premier ore has been found to compact within 6 months. The compacted ore has the same effect as a stub and creates severe stress problems. Drilled and blasted kimberlite in pillar areas on the eastern side of the mine has recompacted in places and led to cave "sit downs" on the extraction level. It means that drawbell cutting on the level below cannot lag undercutting by more than 6 months in some ore types at Premier.
* If the pre-undercut is taken out as a sub-level cave, voids can be created on and immediately above the undercut level. Massive wedges can slide or fall into these voids.

**Advance undercutting.**

Undercut drilling and blasting takes place above a partially developed extraction level. In this case production tunnel, crosscut and drawpoint support can be completed prior to undercut blasting. The actual development completed on the extraction level prior to undercutting should be based on an assessment of MRMR as the more competent the rock the greater the extraction ratio that can be sustained. If only production tunnels are installed, the extraction ratio is 20 percent. If production tunnels and drawpoint crosscuts are installed, the extraction ratio increases to 33 percent. Once the undercut has been blasted, the drawbell is quickly developed. It ensures that compaction problems do not occur. The undercut height can be as low as 4 metres and LHD loading on the extraction level can be undertaken to ensure that complete undercutting is being achieved. The method of undercutting was practised at Bell Mine in Canada and at King Mine in Zimbabwe. At Henderson a variation of advance undercutting is practised in that the production tunnel and crosscuts are completely developed. Only half the height of the drawbell tunnel is excavated. Once this development has been completed, trough blasting is done from the undercut level and the drawbell completed as soon as possible. The result is that trough blasting and undercutting are simultaneous. The apices are subjected to one less cycle of loading and unloading.

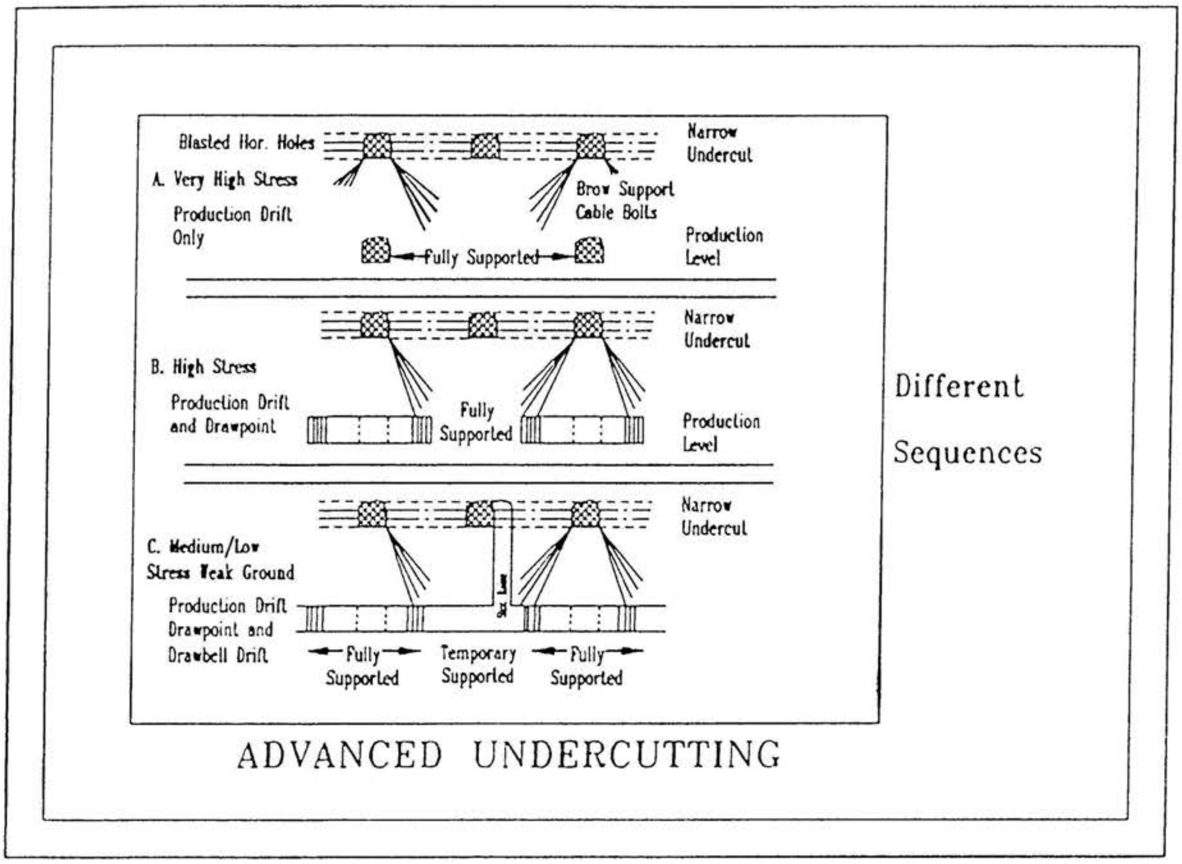



Figure 6.1. Advance Undercutting (Laubscher, 1995)

Advantages of this approach are:

* Tunnels on the undercut level are only required for drilling and blasting and not ore extraction. Only a limited ore handling facility is required on the undercut level.
* The extraction ratio on the production level decreases from 43 percent to less than 20 percent prior to undercutting. Large, unsupported drawbell excavations are developed only after the undercut has passed overhead and are never subjected to high abutment stresses. Fracture propagation and rock damage around the drawbell excavation is limited. This is important as experience has taught that brow damage leads to rapid brow failure. Brow wear as the result of attrition due to the drawing of ore is often negligible, if the brow has not be damaged, even after tens of thousands of tons of ore have been drawn. Minor apices are only developed after the undercut has passed overhead and are not subjected to point loading. The net result is a rockmass that is more confined during undercutting and less susceptible to undercut stress damage and subsequent stress damage during the drawing of ore.
* Active support is aimed at creating a zone of reinforced rock around the tunnels that will withstand the imposed stresses and limit fracture propagation. As most of the production level is developed after the undercut has passed overhead, less support is required. At Premier the installation of long, grouted cable anchors was confined to drawpoint brow areas and the thickness of the rigid shotcrete lining reduced in areas where undercutting

preceded development of the production level. Support cost were reduced by between 40 and 50 percent depending on the RMR.

* Less support rehabilitation is required in areas where undercutting precedes production level development. It reduces support rehabilitation costs and increases the availability of drawpoints. Less drawpoints are required to meet production targets and draw control is made easier as drawpoints are not withdrawn from production for extended periods of time.

Disadvantages are:

* Close control is needed to ensure that trough development follows immediately behind undercutting. Problems on the undercut level affect the extraction level and visa versa.
* Compaction problems can occur if the broken rock on the undercut level is not extracted quickly.
* The development of drawbells must be accomplished from the extraction level into broken rock on the undercut level. The process is more expensive, time consuming and demands greater control than conventional drawbell development.

**Undercut Height**

Theoretically, there is no constraint on the height to which the undercut can be drilled and blasted. The height is usually determined by practical considerations such as the length to which holes can be drilled and effectively charged and blasted. Mines often start with a maximum height of undercut so that ore is available early in the life of the mining block from drilling and blasting operations. With time, the speed with which the undercut needs to be moved to avoid damaging stress loads on the extraction level below becomes the overriding consideration and the height of the undercut is lowered to the practical minimum. Experience (D.H. Laubscher, personal communication) shows that, if the undercut is to be self cleaning, the minimum height is half the width of the major apex. This height allows an angle of 45 degrees to form and ensures that no potentially damaging pillars are formed by static ore columns. Ore is allowed to flow into the adjacent drawbells. Most undercut drill patterns include downholes that create a self cleaning "shoulder" above the drawpoint brow that allows blasted ore to slide into the drawbell as the undercut is blasted. It decreases the risk of static pillars of broken ore forming above the major apices.

At Premier, experience has shown that bulking factors during caving are less than 15 percent. A void of 4 metres on the undercut level will allow 20 metres of rock to cave. The broken rock can move laterally at an angle of 45 degrees or more towards the drawbell, once it is developed. Static columns of broken rock above major apices have not proved a problem.

The decision to mine a high undercut is often an expensive one. The high undercut is developed to provide ore at an early stage in the life of the block. A considerable void is allowed to develop between the ore pile and the cave back. Typical bulking factors for caved ore, especially if subsidence caving occurs, are less than 15 percent. This means that a void of 20 metres could allow 134 metres of ore to cave. If subsidence caving occurred, this could take place within a few weeks and be accompanied by a damaging air blast. Fragmentation could be coarser than anticipated. The time required and cost of drilling long holes are further considerations. Provided that ore can be efficiently removed and a void clear of remnant pillars can be guaranteed, the height of the undercut can be minimal. The smallest undercut height known to the author is 2 metres. This was achieved in undercuts on the Kimberley mines by taking the undercut out as a typical goldmine stope using scrapers to extract the ore through boxholes that were subsequently used as drawpoints once caving had occurred. Andina and El Teniente originally implemented an undercut using a 18 metre high long hole drilling pattern. Both mines have subsequently reduced the height of the their undercuts and El Teniente is successfully using a low undercut in the Regimento Section and have done the drilling that will allow them to carry out an advance undercut over an area of 20 000 square metres in the Isla Section. El Salvador undercuts with a 6,4 metre high undercut. These mines all do post-undercutting. Bell and Gath's Mine have used a 3 metre high advance undercut.

Analogue modelling (McNearny, 1993) suggests that the width of the draw zone increases with increasing undercut height. There is no field evidence to support these modelling results.

**Shape of undercut advance**

The initial shape of the undercut, as well as the face configuration that will be used to advance the undercut to its final position must be considered. Experience in most mining situations for example gold mine stopes, suggests that a straight face is the most stable configuration possible. Experience on several cave mines is that a concave shaped undercut advance is optimal. Corners have stress raising effects and the longer the leads and lags between tunnels the greater the stress effects. If the shape of the undercut leads to the creation of pillars, stress levels are raised in and below the pillars. It becomes increasingly difficult to mine in these progressively smaller pillars and damaging stress levels are felt on the extraction level below. Ideally, the undercut should advance from weak towards more competent rock. The rate of undercut advance has an effect on the stress levels. The rate of advance should be slow enough to allow stresses to develop in the cave back but fast enough to avoid damaging excavations on the extraction level below (Laubscher, 1995).

A circle is the most economical geometric shape of undercut to achieve the maximum hydraulic radius. The hydraulic radius for a circle with a diameter of 100 metres is 25 and an undercut area of 7854 square metres is needed to achieve this hydraulic radius. The most practical straight-sided geometrical shape to achieve the maximum hydraulic radius for the least amount of undercut development is a square. A hydraulic radius of 25 can be achieved by undercutting an area of 100 metres x 100 metres. To achieve the same hydraulic radius a 27 percent larger area must always be undercut if a square rather than a circular shape is used for undercut development. The advantages of a circular undercut are compared to the undercut shape used in the BA5 in Figure 6.2. An circular or square undercut shape is only practical in the initial stages of undercutting.

Insufficient panel retreat caves exist worldwide to determine an optimum shape of the undercut advance from experience. Rock mechanics principles, together with the geotechnical investigation of the orebody to be caved, will provide the best guidelines. Three dimensional numerical stress modelling should form part of the geotechnical investigation.

## 6.3. PROBLEMS AND SOME SOLUTIONS EXPERIENCED IN UNDERCUTTING AT PREMIER

At Premier it was calculated that a hydraulic radius of 30 would be needed to induce caving. Post-undercutting was the chosen method of development and tunnelling to allow the development of 64 drawbells to accept the ore from the undercut was undertaken. The undercut was initiated as a 120 metre long, 30 metre wide slot. The undercut was advanced both north and south from this slot in the shape of an stepped fan until continuous caving initiated. In the Tuffisitic Kimberlite Breccia caving initiated when a hydraulic radius of 22.5 (8 100 square metres) had been achieved. The contiguous area of Hypabyssal Kimberlite that had been undercut was 6 000 square metres. The total area undercut was therefore 14 100 square metres. If the shape of the undercut had been circular only 7 850 square metres would have had to be undercut to induce caving in the Tuffisitic Kimberlite Breccia and 30 drawbells prepared to receive the undercut ore. The time and expense needed to bring the cave into production would have been halved and abutment stress levels would have been considerably less. This is illustrated in Figure 6.2.

In the BA5 block the options were to advance the face eastwards as a straight line or in a configuration that approximated a V-shape. The numerical stress modelling undertaken by Itasca (McKinnon, 1992) had as one of its objectives the determination of an optimum face shape for the advancing undercut. This modelling showed that a V-shape with the acute angle of the V advancing towards the east would be the optimum shape. Stress levels on the undercut level immediately ahead of the abutment with either a straight line or V-shape were much the same but the V-shape had the advantage that shearing stresses in the cave back above the V were increased leading to finer fragmentation. On this basis, a V-shaped undercut advance was accepted and observation shows that no stress related problems have occurred and fragmentation has improved marginally relative to that which has been monitored in the rest of the BA5.

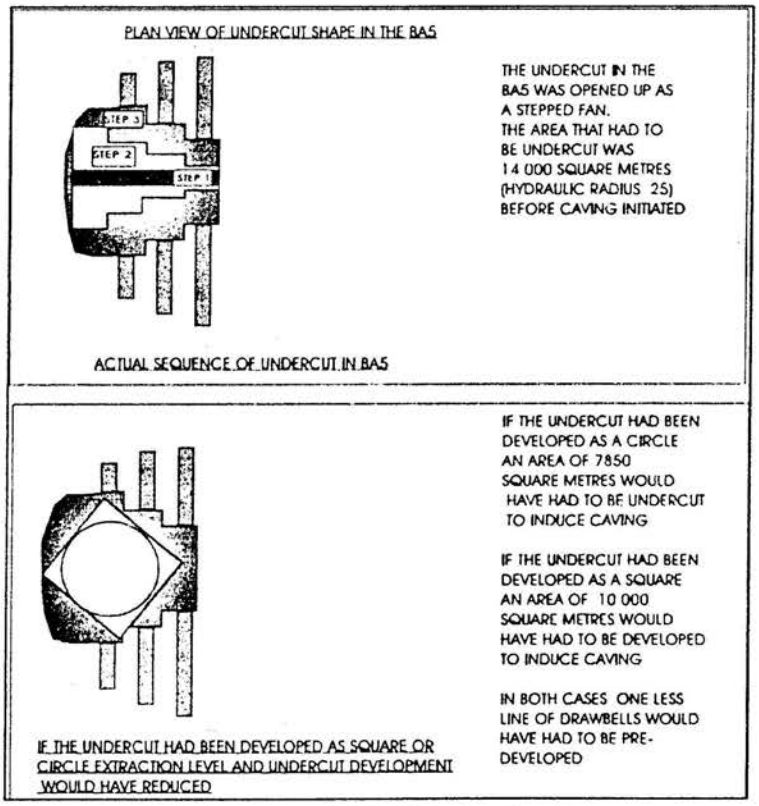



**Figure 6.2. Undercut options for the BA5**

It is possible to advance the V-shaped undercut face in an easterly direction using tunnels that are oriented east-west or north-south. It was decided that east-west oriented tunnels should be used as only the western ends of these tunnels would be within the abutment zone. A considerable length of any tunnel oriented north-south would be within the stress field of the abutment zone for an appreciable time as the undercut advanced towards the tunnel. This did, however, mean that undercut drilling tunnels and extraction tunnels on the level 15 metres below were oriented at right angles to one another. It resulted in complex undercut geometry relative to extraction level development and complicated drawbell development.

The most vexing problem during undercutting was the creation of remnant pillars or stubs. It was primarily the result of mining regulations which do not permit drilling within two metres of a previously charged drillhole for reasons of safety. In the preparation of the drawbells this often resulted in narrow pillars being left immediately above minor apexes. Subsequent undercut drilling in these highly stressed pillars was ineffective, often as a result of hole closure in drillholes angled at less than 45 degrees to the horizontal, due to the high stresses (see Figure 5.14). The situation was aggravated by large tunnels and a 21 metre high undercut which allowed broken ore to flood into the drilling tunnels blocking access to drillholes. Holes could only be charged once a hangup had occurred and the tunnel could be cleaned. A remnant pillar problem inevitably resulted in undercut drilling advancing at a slower rate than planned which, in turn, caused stress related damage, including footwall heave, on the extraction level below. At no stage was a lead and lag situation allowed to develop between adjacent tunnels that exceeded 15 metres. The ruling meant that a slow rate of advance in one tunnel impacted on the rate of advance in all tunnels on the same side (north or south) of the advancing undercut.

The problem of not drilling within two metres of a charged hole was overcome by predrilling in areas where overlapping holes were needed to ensure that pillars were not left above minor apices. Flooding of undercut tunnels with caved ore was minimised by decreasing the undercut height from 21 metres to 12 metres and decreasing tunnel size from 4 metres x 4 metres to 3 metres x 3 metres. This resulted in a better rate of undercut advance and fewer holes were lost as a result of hole closure or shearing.

Stress related problems on the extraction level which resulted in extensive support and rock damage still occurred and the author recommended that an advance undercut be implemented to extend the cave in an easterly direction. The justification for this change in mining sequence which had considerable cost implications is discussed further in Chapters 7 and 10. The pre-undercut has been designed and implemented as a typical sublevel cave layout.

## 6.4. SUPPORT

Tunnels on the undercut level are sacrificial and should therefore be kept to the minimum size commensurate with their intended purpose. This purpose will always include drilling and, where an advance undercut or pre-undercut is developed, access for rubber-tyred loading equipment. Support should be kept to the minimum needed to ensure the safety of personnel and equipment. Little reference to support on undercut levels exists in the literature. Personal observation at Bell Mine, Gath's Mine, Andina, El Salvador, El Teniente, Bultfontein, Du Toit's Pan and Wesselton Mines is that support is kept to a minimum and is far less than that found on the extraction level due usually to smaller tunnel sizes, a lower extraction ratio and the temporary nature of the undercut tunnels. The abutment stresses that develop on the undercut level are at least as high as those monitored on the extraction level but, although drilling and blasting must be carried out in rock that is often severely damaged by abutment stresses, these activities are of a temporary nature and prolonged mining operations are not required in the area of damaged rock.

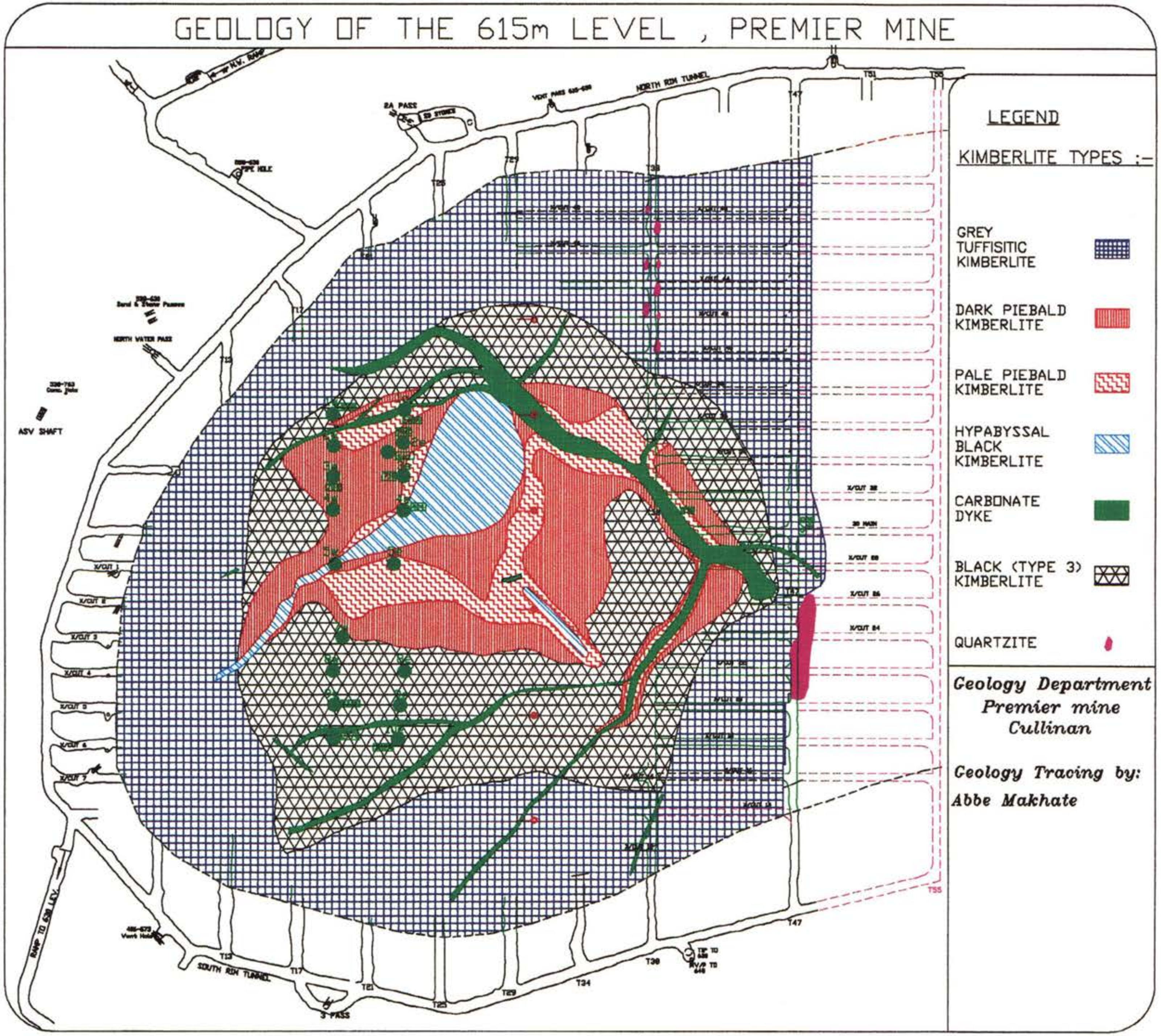



Figure 6.3. Layout and Geology of BA5 Undercut Level

Rock damage away from the abutment zone is well correlated with rock mass rating. Lowering of joint cohesion in the abutment zone as a result of shearing on joint planes can result in open joints which can allow the ingress of water into the area further lowering cohesion and the rock mass rating.

At Premier, competent Hypabyssal Kimberlite is minimally supported with 1,8 metre long 16 millimetre diameter, resin-grouted rockbolts on a 1 metre spacing continued down to grade (spring) line, installed within 5 metres of the development face. Tuffisitic Kimberlite Breccia is similarly supported. As soon as possible thereafter, the tunnels in Tuffisitic Kimberlite Breccia are sprayed with a sealant to prevent decomposition of the rock. Interbolt support in the form of chainlink wiremesh covered with tendon straps is installed down to grade line to control any ravelling that might occur in the hangingwall. Originally, the mesh and tendon straps were covered with 80 millimetres of shotcrete to prevent damage from LHD impacts during loading and tramming and as a result of development and ring blasting. Observation shows that these forms of support damage have been minimal and shotcrete is only applied where rock conditions are exceptionally poor. Six-metre-long 25 ton fully-grouted 7 strand cable anchors are installed in rock with a rock mass rating of less than 35. Typically these areas occur close to the pipe contact and are characterised by decomposing kimberlite and persistent, slickensided joints.

In blocky norite, which surrounds the kimberlite orebody, minimal levels of stress change force movement along the well developed, persistent joints. Rockbolts, cable anchors and mesh-reinforced shotcrete are required to ensure tunnel stability.

## 6.5. MONITORING

A programme of stress change, displacement, support- and rock-damage monitoring was undertaken on the undercut level to ensure the safety of personnel, assess the distance to which undercut stresses would be felt ahead of the undercut, the width of the abutment zone and the extent and mode of rock damage in the abutment zone. This information was used to assess and improve support design. Six-metre-long cable anchors were only used in defined areas and shotcrete linings were only used where the RMR indicated that extensive fracturing and ravelling could be expected. Tendon straps were used only to support the hangingwall and corners of tunnels rather than extended down to the grade line.

Stress change monitoring was undertaken by installing 40 Geocon solid inclusion, vibrating-wire stressmeters in 20 horizontal 36mm diameter core holes, drilled to a depth of 15 metres on either side of an undercut drilling tunnel. One stressmeter was oriented to read stress change in the vertical direction and the other was oriented to read stress change in the horizontal direction. The drilling tunnel was at right angles to the original undercut slot and holes were spaced at 15 metres and stretched to a distance of 60 metres on either side of the slot. Stress changes of less than 1 MPa were recorded until the stress meters were within one or two metres of the undercut face when there was an abrupt increase in stress. The pattern of stress change was often erratic with a stress increase in the vertical direction accompanied by a stress increase or decrease in the horizontal direction. A vertical stress increase of 60 MPa was measured at one station as the undercut face approached a tunnel and a remnant pillar was created. The average vertical stress change recorded was 20 MPa or 2 times the calculated vertical stress of 10 MPa. The level of stress change increased as the size of undercut increased up until the time that continuous caving initiated. The monitored stress change when a remnant pillar was created was more than sufficient to create the induced shear fractures through intact rock that were subsequently monitored.

It was anticipated that the undercut face would remain static along tunnel 29 for some time as the direction of undercut advance swung from a north-south to an eastwards advance. Six monitoring stations were established on the eastern side of tunnel 29. Two 7-metre-deep sonic probe extensometers and one stressmeter were installed at each site. Installations were in both Hypabyssal and Tuffisitic Kimberlite. These show that a minimal stress change was recorded up until the time that the undercut advanced over the installation and that displacement spikes were confined to the blast damaged zone adjacent to the tunnel. Displacements were larger in the Tuffisitic Kimberlite Breccia than in the Hypabyssal Kimberlite. Installations were generally lost before the large stress increases modelled were reached (McKinnon, October, 1992) but did indicate that the stresses in the abutment zone did not extend beyond five metres from the actual undercut face.

Once the V-shaped undercut advance had been established, 3 monitoring stations were installed ahead of the advancing undercut abutment and monitored on a weekly basis. An additional hole was drilled to allow petroscope monitoring. The sonic probe extensometers at these monitoring stations showed no indication of the displacement spikes (associated with movement on existing joints and fractures) or strain as monitored in the other areas of the undercut. Stress changes never exceeded 5 MPa showing that, immediately ahead of the V, the undercut face advance did not create a typical abutment zone. Abutment stresses were carried only in areas where caving had already occurred. This allowed the V-shaped undercut to advance with minimum of support and drilling problems. The support and rock damage monitoring method is detailed in Appendix 1. Monitoring showed that, on the undercut level, minor shear movement occurred along joints and fractures up to 100 metres from the abutment zone in well jointed norite and where joint planes had low condition ratings. This movement lowered the cohesion on joints and resulted in block fallout of unpinned blocks. A rating of 1 was generally assigned to this type of damage. In shear zones, or where water lowered the joint condition rating, block fallout and frittering of the rock around steel tendons was extensive. A damage rating as high as 3 could occur here. In the weak, poorly jointed Tuffisitic Kimberlite Breccia, movement along joints and minor shear failure in the blast damage zone around tunnels occurred up to 60 metres ahead of the abutment zone. A damage rating of 2 or 3 was generally assigned in these areas. Minor shear failure through intact rock together with minor block fallout and destruction of unreinforced shotcrete lining occurred in Hypabyssal Kimberlite up to 20 metres ahead of the abutment zone. This was assigned a damage rating of 1. At Premier, experience has been that the stresses associated with the abutment can force movement along joints and fractures up to eighty metres away from the actual abutment zone. This results in lowering of cohesion on the joints and, if blocks are not pinned by steel tendons or interbolt support, block fallout under the influence of gravity can occur. Within sixty metres of the abutment zone, failure of weak Tuffisitic Kimberlite Breccia can occur at the point of maximum curvature of tunnels. Rigid shotcrete lining often cracks and can show extensive scaling. The level of stress change associated with type of rock and support damage is usually less than 2 MPa.

The abutment zone in which substantial rock and support damage occurs is found between 3 and 15 metres from the undercut face on the actual undercut level. In the well jointed Hypabyssal Kimberlite shear movement along joints is often noted up to fifteen metres ahead of the abutment zone and closely spaced shear fractures are found up to 3 metres from the abutment face. These shear fractures are inclined at an angle of between 25 and 35 degrees to the vertical and fracture spacing is typically 100 millimetres or less. In the poorly jointed, less competent Tuffisitic Kimberlite Breccia, shear fractures are found up to 15 metres ahead of the undercut face. Mapping shows that these fracture have an average spacing of 400 millimetres and are inclined at between 25 and 35 degrees to the vertical.

Extensive support damage of rigid linings such as shotcrete or mesh reinforced shotcrete occurs in the abutment zone. Steel tendons loose their ability to knit the rock together as a result of the fracturing which results in short, ineffective embedment lengths of grouted steel tendon (See Figure 3.1). This in turn results in frittering of the rock around the tendons and faceplates become loose and do not provide effective restraint.

Support in the abutment zone is therefore problematic. Numerical stress and displacement modelling, together with underground observation, has shown that in the Hypabyssal Kimberlite it is more effective to increase tendon density and reinforce the rock mass rather than install long cable anchors into solid rock beyond the blast damaged zone around the tunnel on the undercut level. The stress induced fractures in the abutment zone result in short embedment lengths for the grout used to install the tendons rendering these less effective. Normally, the rock has sufficient residual strength to ensure that widespread instability does not result. On the production level, long cable anchors are needed to knit the minor and major apices together and prevent pillar failure where large displacements are predicted. This logic does not apply on the undercut level.

In Tuffisitic Kimberlite Breccia blast fractures which form shortly after development can extend up to 3 metres into the Tuffisitic Kimberlite Breccia (Guest, 1985). For support to be effective longer tendons that penetrate through the blast damaged zone are needed. Again, induced fractures result in short, ineffective embedment lengths for the grouted tendons which is often followed by failure at the rock/grout interface. Frittering of the rock around the collar of the tendon installation results and faceplates become ineffective. Any additional stress changes, secondary blasting, ring blasting or LHD loading results in aggravated erosion of the rock around the tunnel. Effective interbolt support in the form of tendon straps, butterfly face plates and wire mesh must be installed to ensure the safety of men and equipment. Due to the expense of installation concrete and shotcrete linings are avoided as much as possible as, where stress levels are high, this form of support is only marginally effective.

## 6.6. CONCLUSIONS RELATING TO MONITORING AND SUPPORT DESIGN

Displacement and stress change monitoring showed that high stresses only developed on and beneath the undercut face once continuous caving had initiated and the undercut face became a true abutment. Both kimberlites are relatively inelastic rock types and stress changes are felt at relatively small distances ahead of the abutment. The abutment zone of extensively damaged rock is therefore not wide. The change in shape of the undercut advance from a straight face to a V-shape lowered undercut stresses at the apex of the V. The level of stress change was generally less than twice the virgin stress which agreed well with both theory and monitoring on other cave mines. Remnant pillars created extremely high stresses, resulting in extensive rock mass damage and need to be avoided at all costs.

Support damage showed that it was impractical to install support that would not be damaged by undercut stresses associated with the abutment zone. Long cable anchors were consequently used less widely, being replaced by more rockbolts at a smaller spacing. Shotcrete was only used as interbolt support where it was expected that decomposing kimberlite would result in extensive frittering between rockbolts.

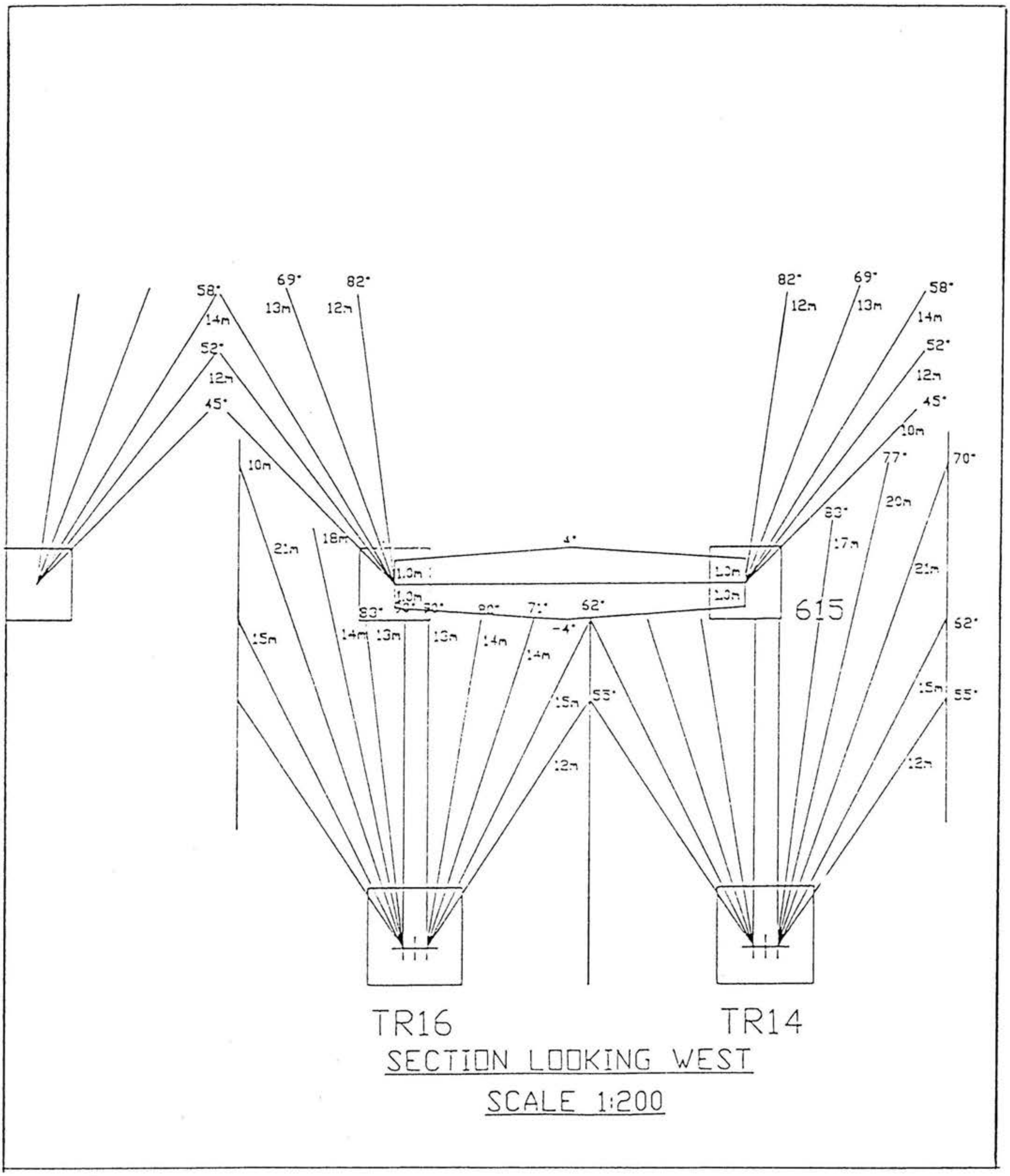



Figure 6.4. Undercutting and Drawbell drilling at Premier

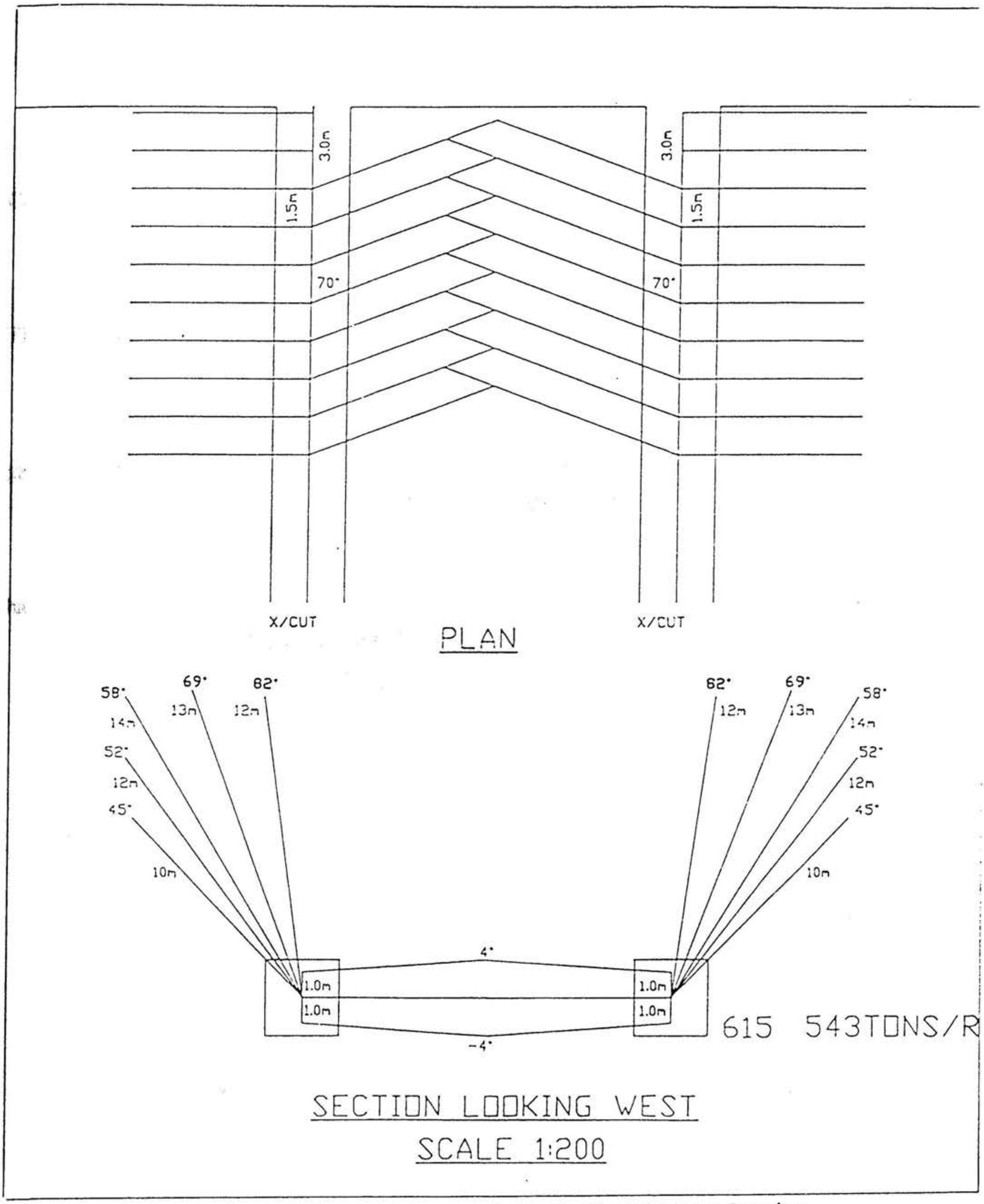



Figure 6.5. Undercut Layout in Trial Block at Premier

# CHAPTER 7

## STRESSES AROUND THE UNDERCUT EXCAVATION

**Statement.**

Cave mining differs from other mining methods in that it deliberately creates a large, unstable excavation. Stress effects and subsequent rock movement around this excavation, in both the near field and far field, can become uncontrolled. Many mining engineers are therefore extremely cautious and even reluctant to introduce cave mining.

The pattern of stress distribution around an excavation is a function of the rock mass characteristics. Several methods of characterising a rock mass which are not exclusive to cave mining, such as rock mass classification and laboratory analysis, have been developed. Rock mass classifications have difficulty in predicting the effect of stress on the rock mass. The prediction of the strength of broken rock is one of the most difficult problems facing geotechnical and mining engineers, even where laboratory tests have been performed.

The stress distribution set up around a cave excavation is a function of the size, shape, depth and regional stress field in which the excavation is created. Numerical codes that allow the accurate simulation of the stress distribution associated with cave mining methods are a useful tool which have given geotechnical engineers the opportunity to better understand and predict these stress distribution patterns.

Detailed observation, monitoring and numerical modelling at Premier, together with data gathered by the author on numerous caves mines worldwide, discussions with mining personnel involved in cave mining and study of existing literature, has allowed the author to develop a stress model for the BA5 mining block at Premier that will be useful in predicting the behaviour of the rock mass in future caves in kimberlite at Premier and on other cave mines in kimberlite in the De Beers mining group. This model is more detailed and relevant to a cave mine using LHD's for extraction and implementing the mining sequence and methods of drawbell development and undercutting used at Premier than other models reported in the literature. The model has been used to predict stress parameters for the calculation of hydraulic radius, layout, an advance undercut mining sequence, support design and fragmentation in the new BB1E mining block currently being developed at Premier.

## 7.1. INTRODUCTION

Any cave mining method relies on undercutting a sufficient area to induce continuous caving. The creation of the undercut sets up a pattern of stress redistribution around the undercut excavation. Resultant stress levels are a function of the primitive stress in the orebody and the excavation sequence. The mining operation therefore has the potential to favourably or adversely affect stress levels during cave mining. It is useful to study these stress effects above the undercut level, on the undercut level and on the extraction level below the undercut.

In a recent review (Wagner, 1992) of the stresses that develop around mining excavations, "depth" is defined in terms of both rock strength and pre-mining rock stresses rather than absolute metres. If the excavation is in a situation where the primitive stress is half the value of the rock mass strength the excavation is defined as deep. Stress concentrations around such excavations range from 2 $q$ in the vicinity of truly three dimensional excavations such as caves to 50 $q$ at the face in tabular excavations such as gold mine stopes. Here $q$ is the vertical component of the primitive stress. Fracturing of the rock ahead of stope (or undercut) faces is an unavoidable consequence of the extraction of narrow tabular (excavations) mineral deposits " (Wagner, 1992 p52). Applying a typical equation (Wagner, 1992 eqn 5 p52) used in gold mines to calculate the theoretical stresses that would develop ahead of a 12 metre high undercut face with a span of 120 metres at a depth of 630 metres shows that stress levels of up to 56 MPa could develop. This situation would occur immediately prior to continuous caving. Once caving initiated, the undercut span would reduce and stresses at the face would reduce to 36 MPa if a half span of 25 metres is assumed.

Stress changes measured as the undercut was run over the extraction level at Henderson Mine showed stress changes of 40 MPa or 3 times the primitive stress value (Brumleve & Maier, 1981). More recent work at Henderson (Rech & Lorig, 1992) showed that in situ stresses range from 16 MPa in the abutment zone on the 7700 level in the stress shadow of the 8100 level to 63 MPa in drifts ahead of the undercut on the 8100 level. Modelled results in the abutment zone on the 8100 level are 60 MPa.

In a cave mining situation, an area sufficient to induce continuous caving must be undercut. An area of 120 metres x 120 metres would be a large undercut area in most cave mining situations. As the size of the excavation increases so do stresses ahead of the undercut face. Stress levels from 2 (equidimensional excavation) to 4 times (tabular stope) the primitive stress are suggested by theory and numerical modelling and confirmed by in situ stress measurements. These stress levels can be high enough to induce shear fractures in the rock ahead of the abutment. The magnitude of these stresses reduce exponentially with distance from the face.

Stresses on the extraction level, usually 12 to 18 metres below the undercut level, have much the same effect on the rock mass as they do on the undercut level. If the extraction level is developed prior to the undercut being run over the area (post-undercut), high extraction ratios on the production level have the potential to further increase stress levels. This can result in aggravated support and rock mass damage.

Prior to the initiation of caving, a tensile zone can exists above the undercut. Caving above the undercut will move towards creating a stable, spherical or arch, configuration. Theory (Wagner, 1992 eqn 3 p51) suggests the primitive stress level immediately adjacent to the cave excavation will increase by a maximum ratio of 1,5 in a hydrostatic stress field. In an area such as the BA5 mining block, tangential stress levels could therefore be as high as 15 MPa.

It should be noted that theoretical stress levels do not take local geometry into consideration and stress levels can be raised to damaging levels by the creation of remnant pillars during undercutting, wedge failures and compaction of broken ore. The mining sequence can be planned in such a way as to take advantage of the stress levels to assist fragmentation in the cave back. Advance and pre-undercutting can limit damage to the rock on the extraction level.

A knowledge of the stress pattern around a cave excavation can be used to calibrate failure criteria and predict rock mass behaviour both in the near field and far field of the excavation. In the near field, rock mass response involves failure in shear, compression and tension, induced fracturing and the large scale movement of rock fragments, including caving. The width of the near field zone is a function of the stress field, rock mass characteristics and the position of the zone relative to the excavation. The near field associated with the abutment zone on the undercut level is usually wider, (and rock mass damage more intense) than the near field in the cave back. The width of the near field typically varies between 5 and 15 metres. In the far field rock mass response is normally the result of movement on existing planes of weakness such as faults, fractures, joints and shear zones. This response can be felt up to 200 hundred metres away from the abutment zone. Movement on large structural features such as faults can be damaging and needs to be considered in implementing the cave. A knowledge of the stress pattern can be used to improve the prediction of hydraulic radius, mode of caving, fragmentation and the magnitude of the stress changes associated with the successive stages of cave mining.

## 7.2. STRESSES ABOVE THE UNDERCUT LEVEL.

Little published data is available relating to stress and rock damage monitoring above the undercut level. Panek (Panek, 1981) undertook extensive monitoring around several panel caves at the San Manuel Mine. He reports strain relief in the rock surrounding the active cave and extensive development of extension cracks tangential to the cave that allowed the rock to displace towards the cave. At one site, cracks that displaced the rock by 5 millimetres or more were measured from 2 metres below to 55 metres above the undercut level. Cracks were noted up to 59 metres from the nearest mining. At another panel site, monitoring carried out 69 metres above the undercut level detected cracks developing up to 111 metres away (horizontally) from the nearest active drawpoint. Angles of draw of up to 58 degrees (average 52 degrees) were measured. Major concrete liner damage, deformation and cracking were monitored 69 metres above the undercut level at distances of up to 122 metres from the nearest active drawpoint. Angles of draw varied from 31 to 61 degrees. Table 7.1. summarises the results of monitoring at the San Manuel Mine.

For the San Manuel Mine the zone of major cracking around an active cave can be approximated by a circle of diameter S + 2w where S is the long side of the active cave and w the width of the active cave. Panek concludes that (1) the larger the active cave the greater the deformation at a given distance from the cave and (2) the greater the angle of draw to the limit of major concrete liner damage at a given elevation above the undercut level. Deformation response to the extraction of ore is swift and a simple elastic-plastic model should be appropriate for most structural analyses. Monitoring results suggest a yield zone surrounding a caved block.

**Table 7.1. Monitoring around Panel Caves at San Manuel Mine**

| **Panel site** | 25 | 7 | 2 | 41 | 41 |
|---|---|---|---|---|---|
| Location: facing end or side of cave | End | Side | End | End | Side |
| **Limit of major damage** | | | | | |
| Horizontal distance | >60m | 45m | 98m | 62m | 80m |
| Distance above undercut | -5m | 69m | 69m | 69m | 69m |
| Angle of draw (degrees) | * | 33 | 55 | 42 | 49 |
| **Dimensions of active cave** | | | | | |
| Maximum height | S | 400m | S | S | S |
| Length | 230m | 54m | 110m | 64m | 64m |
| T.O.D. closest block | 1m | 40m | 6m | 32m | 47m |
| block number(s) | b.1 | b.1 | b.1 | b.1 | b.2,3 |

Within this yield zone, extension cracks form tangential to the cave excavation and allow the rock to displace towards the cave. It is suggested that these tangential extension cracks continue around the cave back. Beyond the yield zone, a region of raised stress exists. Panek does not report on the competence of the rock mass in which the monitoring was carried out. A later report (Cummings et al., 1984) assigns a rock mass rating of 35 to the San Manuel rock. This later report shows that the rock mass response to mining is very much a function of the rock mass rating that is affected by the caving operation.

Panek's work taken together with that of Engineers' International and Laubscher suggests that a potential damage zone can be defined around the undercut. The diameter of the damage zone on the extraction level and undercut level will be a function of the hydraulic radius of the undercut and the rock mass rating. The damage zone will extend upwards from the undercut level at an angle. The angle will be a function of rock mass rating.

Work done by Brummer around a gold mine stope in rocks of great strength and at great depth show that the behaviour of the rock mass around the stope is much the same as that monitored by Panek (Brummer, 1988). The distance to which the effects of a stope in a gold mine are felt are considerably less than that monitored by Panek. The rock is competent quartzite with a high rock mass rating. Simulation of fracture growth around openings in highly stressed, brittle rock (Napier & Hildyard, 1992) replicate the fractures seen around some undercuts and show that the shear fractures that develop around and above a stope (or undercut) are a function of the applied horizontal stress.

Laubscher (Heslop & Laubscher, 1981. Laubscher, 1995) states that two types of caving can occur in the cave back. Where the undercut moves slowly stresses are allowed to develop in the cave back and "stress caving" is more likely to occur. Where the undercut moves quickly, often to prevent damage to the extraction level where it is sited in weak rock, stresses are not given time to develop and "subsidence caving" results. This usually results in coarser fragmentation.

At Kimberley, a mining block in the Bultfontein mine was planned as a vertical crater retreat (VCR). Three drilling tunnels were established 95 metres above the extraction level and a series of 160 millimetre diameter holes were drilled from the tunnels. Shortly after the VCR operation was started, continuous caving initiated and progressed rapidly through the drilling tunnels. At one stage, it was possible to approach along a drilling drift to within a few metres of the cave excavation where a considerable airgap had formed. At this stage, caving was occurring at the rate of 2 metres per day. Extensive joint dilation followed by block fallout was occurring to a depth of 2 metres into the rock mass around the cave. Shear movement along vertical joints together with some joint dilation could be seen up to 20 metres from the cave. Observation down numerous drill holes showed no indication of extensive shearing and holes remained open even, where the cave had progressed to within 2 metres of the drilling drift. The RMR of the kimberlite through which caving was occurring was estimated at 50.

## 7.2.1. EXPERIENCE AT PREMIER MINE ABOVE THE UNDERCUT LEVEL

Several mining levels existed above the undercut level at Premier and allowed access to the area adjacent to the cave as well as into areas immediately above the cave back as caving progressed. It was possible to monitor and observe the rock mass around the cave excavation in weak, poorly jointed Tuffisitic Kimberlite Breccia, competent Hypabyssal Kimberlite, well jointed norite and jointed, competent gabbro.

In the Tuffisitic Kimberlite Breccia (RMR 55), stress changes adjacent to and immediately above the cave back, in the near stress field, forced movement along joints and fractures. Minor shear movements in the far stress field, especially along horizontal joints, were noted up to 60 metres from the cave excavation. Joint dilation in the near stress field often resulted in open fractures. As open fractures started to form tensile fractures, block fallout and some shear fracturing through intact rock was noted (see photographs). Extensive tensile fractures which allowed toppling failures to occur were noted up to 10 metres from the cave. Observation showed that stress levels were not great enough to induce the extensive shear fracturing that was noted in the abutment zone on the undercut level in this same rock type. Localised, induced, shear fracturing was noted where joint dilation allowed large rock masses to move. Minor movement of a large rock mass often resulted in surrounding blocks being highly stressed and this resulted in complete failure of the adjacent blocks in shear. Monitoring on the extraction level shows that stress changes of 15 MPa are required to induce shear fracturing in this rock type.

P.7.1. Dilation (20mm) on joint in TKB prior to caving

P.7.2. Block fallout under the influence of gravity once joint cohesion has been overcome

In the Hypabyssal Kimberlite (RMR 65) stress changes forced shear movement along fractures and horizontal joints up to 20 metres from the cave excavation. Minor, induced shear fracturing was noted at the point of maximum curvature of tunnels up to 20 metres from the cave excavation. Within five metres of the cave tangential fractures and extensive block fallout was noted. A "pre-conditioned zone" always existed in the proximity of the cave and is an important prelude to caving progressing through an area. Minor tensile failure was noted, but caving was largely joint controlled. Stress levels were not high enough to result in extensive shear fracturing through intact rock as was seen ahead of the abutment on the undercut level. Monitoring in this rock type on the extraction level showed that stress changes in excess of 20 MPa are necessary to induce shear fractures in this rock type.

**P.7.3. Tensile failure through intact rock immediately adjacent to cave (UCS of TKB 100 MPa)**

In the norite (RMR 45) movement along joints was noted up to 20 metres from the cave. This resulted in joint dilation and block fallout under the influence of gravity. Tensile or shear failure through intact rock was seldom observed.

An aerially extensive parking bay allowed access into the gabbro sill (RMR 70) and monitoring and observation was possible here for several months as the cave back approached the excavation. The first indication of the approaching cave was joint dilation on a well defined continuous, horizontal joint. Dilation, accompanied by shear movement on the joint plane of less than 2 mm, continued to increase as the cave back approached and resulted in rock bridgess on the joint plane rupturing. Minor block fallout from the excavation sidewall occurred. Even when the cave back had approached to within three metres of this excavation there was no indication of any tensile or shear failure through intact rock. In the gabbro the caving process was entirely joint controlled. "Pre-conditioning" of the joints in the form of dilation and shear movements are an important prelude to the cave progressing through an area. Loss of water during core drilling into the cave showed that the "pre-conditioned zone" extended to at least 10 metres from the cave back. Monitoring showed that, in some cases, this "pre-conditioning" could occur up to 37 metres above the cave back.

A substantial airgap existed between the ore column and the cave back during the entire caving process in the BA5. Logically, if the broken rock had been in contact with the cave back it would have had an influence on the extent of the "pre-conditioned zone" and the stresses that developed around the cave excavation above the undercut level.

At Premier there was little evidence of the extensive extension fractures forming tangentially around the cave excavation as noted by Panek at San Manuel. There was no indication of these extension fractures immediately above the cave. The absence of these fractures could be a function of the rock mass competence and the primitive stress state at Premier Mine as compared to San Manuel.

**Numerical modelling**

Three dimensional numerical stress modelling was undertaken of the BA5 mining block to simulate successive stages of mining to predict the stress distributions that would develop as caving progressed and the sill collapsed (McKinnon, 1992).

Figure 7.1. shows various undercut geometries, in plan view, that were modelled. Figure 7.2.a shows the geometry of the undercut excavation as this existed in 1992 and Figure 7.2.b. the predicted distribution of stresses around this excavation. This shows that stress levels in excess of 30 MPa only occur at points of maximum curvature around the undercut excavation. Figure 7.3.a. shows the geometry of the undercut excavation if the cave is advanced eastward with a straight face. Figure 7.3.b. is a section through this geometry and shows the predicted level of stress around such an excavation. Stress levels generally do not exceed 39 MPa ($3q$) except at the point of maximum curvature of the undercut excavation, i.e. in the abutment zone. Stress levels of 31 MPa are felt up to 20 metres ahead of the undercut, and stress levels of up to 23 MPa are felt up to 40 metres ahead of the undercut. (It should be noted that these are total stress values and not stress changes.)

Figure 7.4.a. shows the geometry of a stepped cave line (or V-shaped cave line) in plan view. The three dimensional geometry is illustrated in Figure 7.4.a. The predicted stress distribution if the undercut face advances as a V-shape, is illustrated, in plan view, in Figure 7.4.b. This shows that stress levels on the undercut level are much the same as for the straight cave face. Further consideration of this model (not illustrated here) shows that stress level generally do not exceed $3q$ except in the region above the undercut face. The V-shape of the undercut, however, creates a considerable area of raised stress in the unstable brow above the V. The principal stress direction lies close to the horizontal and will force shear movement along existing horizontal joint sets. Stress levels are sufficiently high to cause shear failure through intact kimberlite. The unstable brow will allow considerable tensile failure.

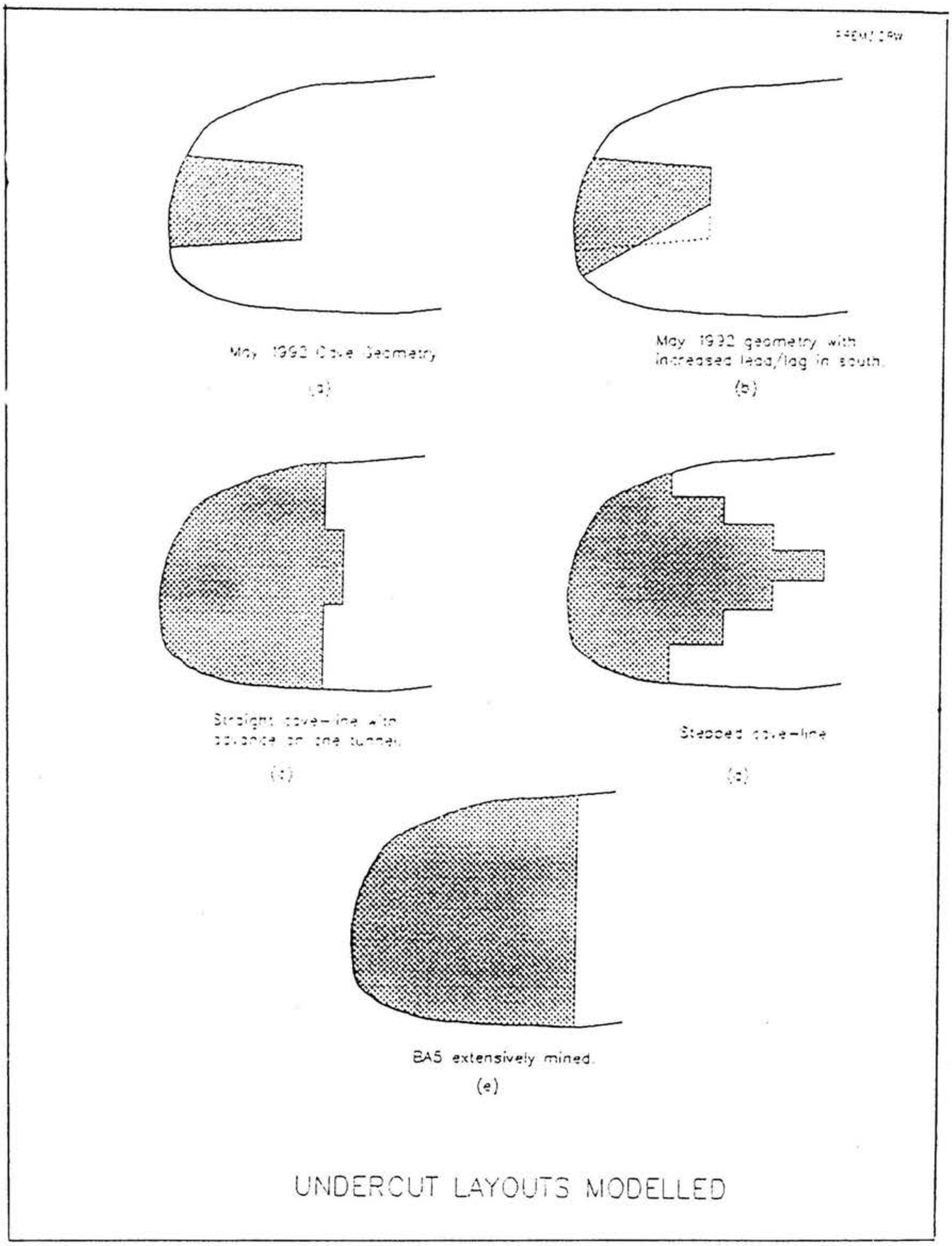



Figure 7.1. Alternate Layouts that were Modelled

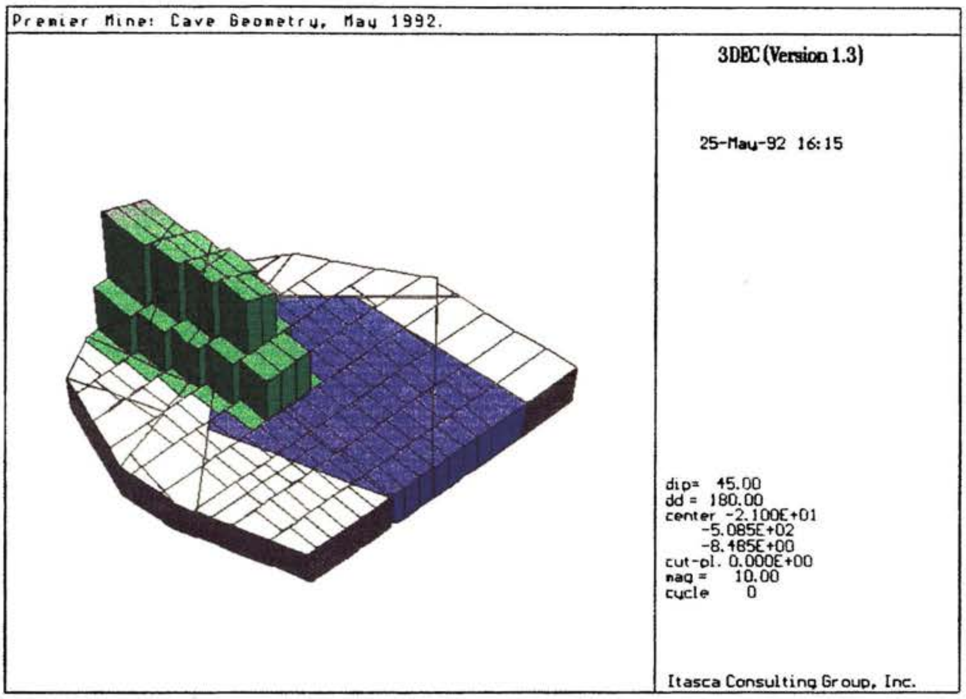



**Figure 7.2.a. Actual Undercut Geometry in 1992**

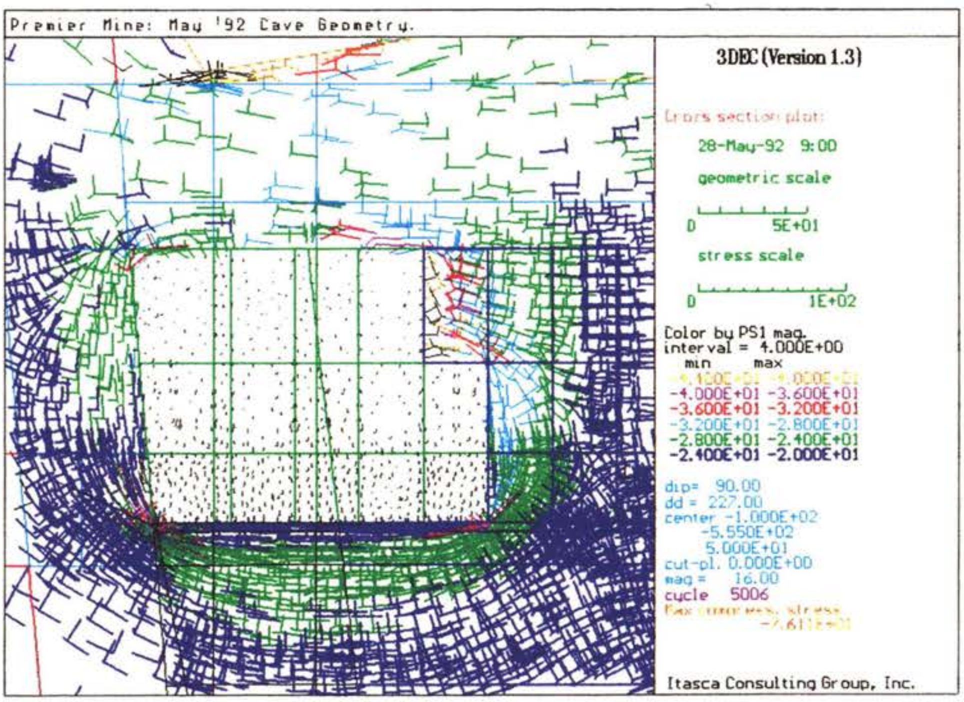



**Figure 7.2.b. Predicted Stresses around Actual Excavation**

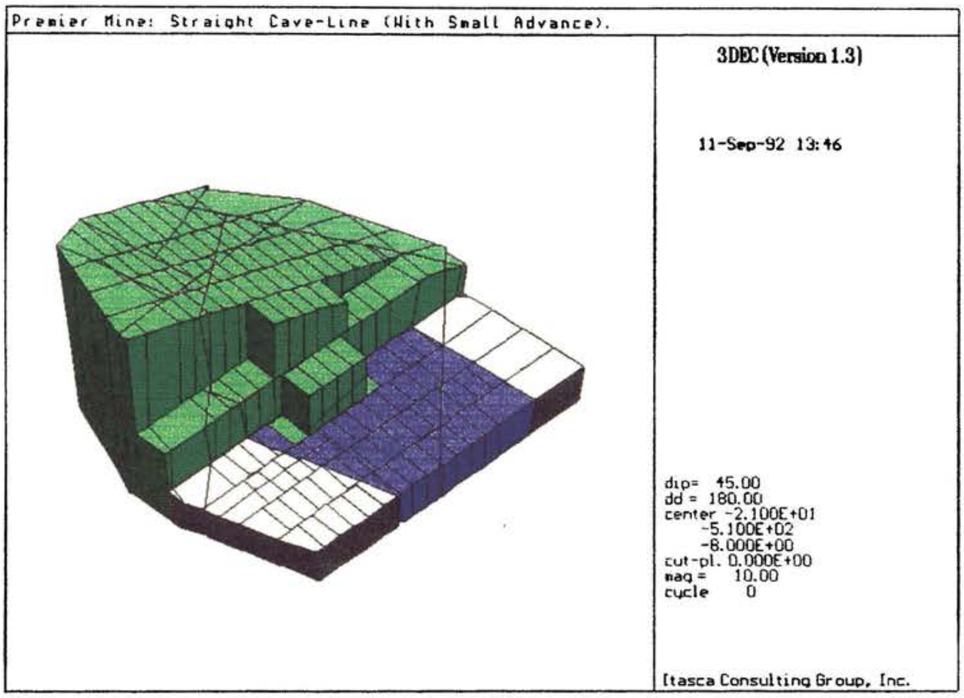



**Figure 7.3.a. Advancing with a Straight Face**

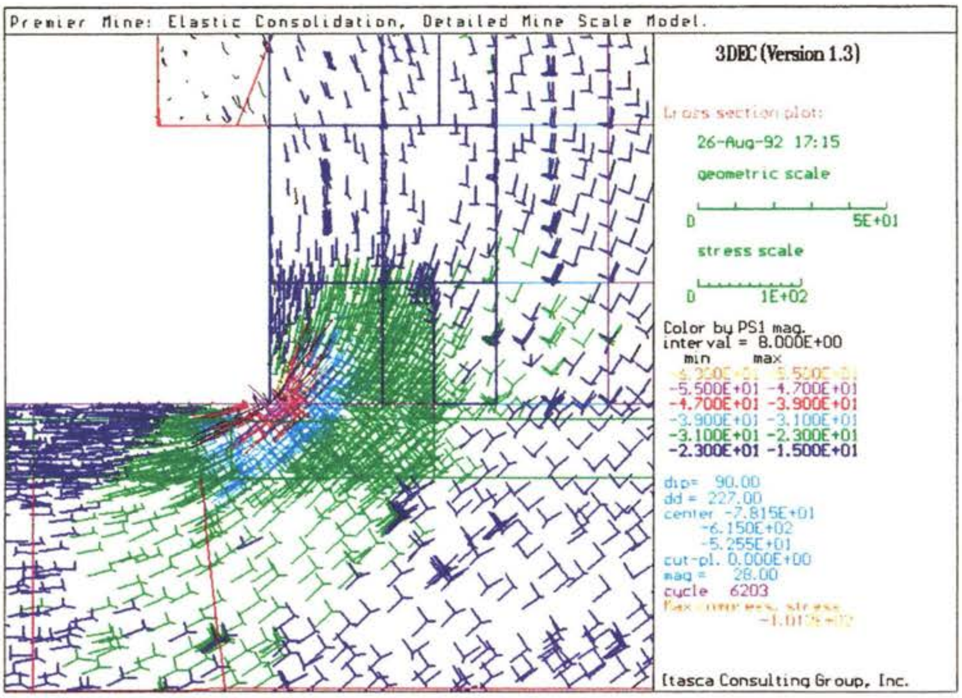



**Figure 7.3.b. Predicted Stress Levels around "Straight Face" Excavation**

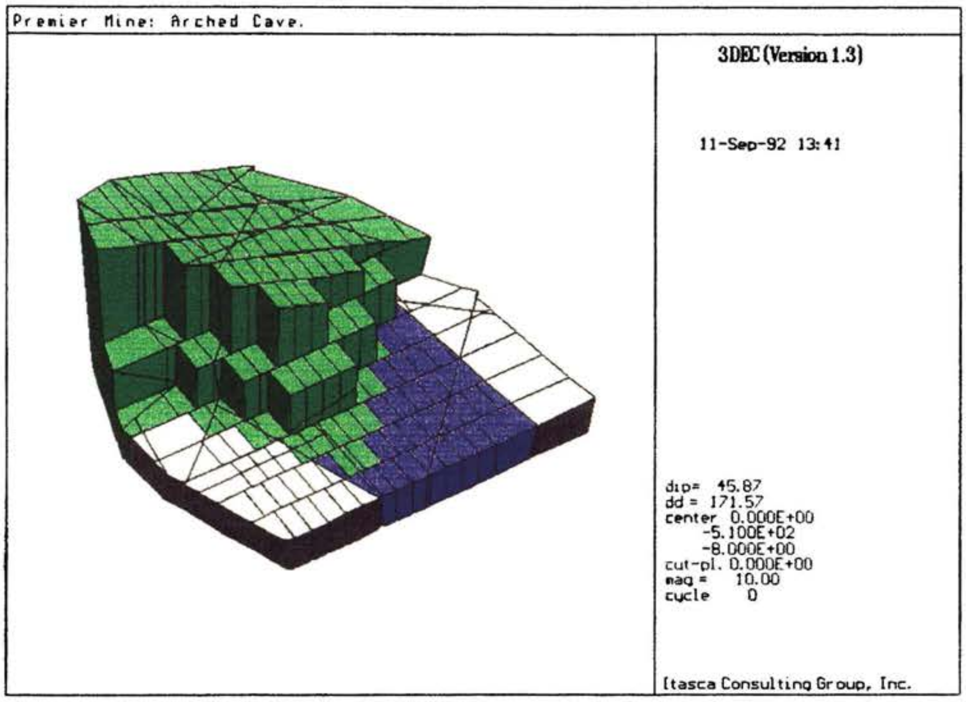



**Figure 7.4.a. Advancing with a V-shaped face**

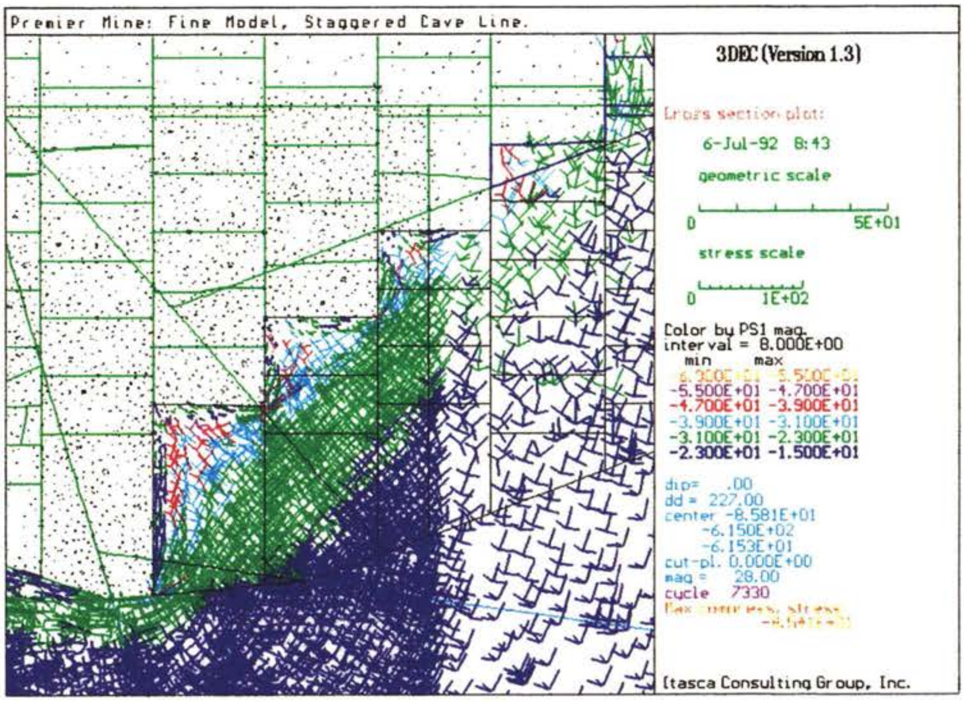



**Figure 7.4.b. Predicted Stress Levels around Undercut Excavation when this advanced as a V-shape**

## Monitoring

Convergence monitoring in concrete lined drifts on the 500 metre level, 115 metres above the undercut level showed both convergence and expansion of the order of 2 mm as the cave back approached. Hoop stresses resulted in minor shear failure of the rigid lining. There was little damage to the drifts prior to the cave progressing through the area.

Vertical holes drilled into the cave back through gabbro and kimberlite usually showed water loss within 10 metres of the cave back, but no other drilling problems were encountered. Core holes drilled into areas of maximum curvature of the undercut excavation resulted in difficult drilling as a result of the shear movements that occurred within a few metres of the cave excavation. This confirmed the modelled results.

## Discussion

The progressive size of the undercut excavation, even after continuous caving initiates, influences the rate of caving. The size of the undercut excavation needed to initiate caving in the BA5 on the undercut was 14 000 square metres. Monitoring showed that the size of the undercut excavation that eventually reached the gabbro sill 115 metres above was less than 1 000 square metres in area. Similar caving histories have been recorded on several other De Beers mines where the area that was needed to initiate caving was greater that the area needed to propagate caving and that this area diminished as the cave propagated upwards. At Bultfontein, a hydraulic radius of 15 was needed to initiate caving. When the cave eventually propagated through a drilling level 95 metres above, the area of undercut was only 240 square metres and caving was occurring at the rate of 2 000 millimetres per day. At De Beer's Mine, the area of undercut needed to initiate caving was in excess of 1 000 square metres and the eventual area of cave that broke through into overlying workings was less than 500 square metres (Hartley, 1981). In both mines in Kimberley and in the BA5 at Premier, this diminishing size of excavation created problems. At Bultfontein and De Beer's mines when caving propagated through to the overlying waste capping, finely comminuted shale and kimberlite flooded into the undercut excavation and prevented further caving. This resulted in early ingress of waste and substantial wedges of uncaved ore being left behind that had to be mined by other methods. In the BA5, the size of undercut that reached the sill was too small to induce caving of the sill and a substantial airgap formed.

The progressive decrease in size of the undercut area needed to propagate caving is best explained by recent work undertaken on crack initiation and propagation above an open stope (Napier & Hildyard, 1992). Modelling shows that horizontal stress levels play an important role in determining the inclination of shear fractures around an opening such as an undercut or stope. The larger the ratio of the horizontal to vertical stress, the more induced shear fractures will incline away from the vertical. If the major principal stress is horizontal and vertical stress levels are low, these stress fractures would be even more steeply inclined away from the vertical. In a kimberlite pipe, the vertical to horizontal stress ratio is usually 1:1, but mining of the overlying ore by open pitting reduces the vertical stress ratio. Horizontal stress are increased above the undercut excavation as caving progresses and the "crown pillar" above the cave reduces in thickness. Above the undercut excavation, therefore, induced stress fractures will incline towards the horizontal and increase in magnitude as the cave propagates. The size of undercut excavation will progressively reduce. In the gabbro sill where stress levels are not sufficient to result in induced fracturing, this process cannot occur. If the cave propagates into weaker overlying rock, the caving process will accelerate and, once the cave propagates through to surface or into an overlying open pit, the size of the cave excavation will increase by the processes experienced in a normal open pit which include toppling and wedge failures.

Prediction of the stress pattern above the undercut excavation and the effect that it will have on the rock mass, in both the near and far field, allows a better estimate of the hydraulic radius needed to induce caving, the rate of caving and the primary fragmentation size distribution. Uncertainty regarding the stress pattern in, and its effect on, the gabbro sill, the Hypabyssal Kimberlite and Tuffisitic Kimberlite Breccia led to uncertainty in all of the above in the BA5. This adversely affected planning of the cave.

**Stress effects above the undercut level: conclusions.**

1. The overall pattern of rock mass response above the undercut showed that minor stresses were felt ahead of the cave to a distance of between 10 and 60 metres. These stress changes forced shear movement along joints and fractures and resulted in joint dilation which lowered the cohesion on joint planes. Rock mass response was joint controlled and was better correlated with joint condition rating than total rock mass rating. Pre-conditioning of joints, especially horizontal joints, which involved movement that lowered cohesion and the angle of friction and ruptured rock bridges, was an important prelude to caving progressing through an area.

2. As the cave initiated block fallout under the influence of gravity was the most common failure mode in all rock types and remained the almost exclusive failure mode in the well jointed gabbro and norite. In the poorly jointed, weak Tuffisitic Kimberlite Breccia tensile and shear failure through intact rock was common. In the more competent, but better jointed Hypabyssal Kimberlite shear and tensile failure was less widespread.

3. Results show that stress changes above the undercut in the BA5 are of the order of 15 MPa (1,5 $q$) or less within 10 metres of the cave and decrease to less than 2 MPa 80 metres from the cave excavation. This agrees with both theoretical and modelled stress predictions. Heightened stress levels are the result of unstable geometry such as the brow area above the undercut V and at the points of maximum curvature around the undercut excavation.

4. Observations suggest that two distinct zones occur within the tensile zone above the cave back. Immediately adjacent to the cave excavation large displacements occur in the near stress field. Here, radial and tangential stresses can be sufficient to induce failure though intact rock aiding the fragmentation process indicating stress levels in excess of 20 MPa. Blocks defined by both natural and induced joints and fractures slide and fall under the influence of gravity onto the top of the draw column. The depth of the zone is a function of rock mass structure, strength and the total stress field and varies from 5 metres in Tuffisitic Kimberlite Breccia and norite to 3 metres in Hypabyssal Kimberlite. In competent gabbro this zone is poorly defined as no induced fracturing occurs.

5. Beyond the zone in the far field, stress levels are only sufficient to induce movement along joints and fractures in the rock types that occur at Premier Mine. Joint condition rating is important in defining the depth of the zone. In norite, the zone extends to 60 metres from the cave excavation, in Tuffisitic Kimberlite Breccia 30 metres and in Hypabyssal Kimberlite 15 metres. Due to the nature of the caving process in the sill, the zone is poorly defined in the gabbro.

## 7.3. STRESSES ON THE UNDERCUT LEVEL

Stresses on the undercut level and the behaviour of the rock mass in the abutment zone have been studied in several cave mines and are directly analogous to the stress regime and induced fracturing that develops ahead of a gold mine stope. As the undercut is developed, stress levels in the near field are sufficiently high to result in extensive shear fracturing of the rock ahead of the undercut face. The fractured rock in the abutment zone is not able to carry much stress and a zone of increased stress is found immediately ahead of the abutment zone. The width and extent of fracturing of the abutment zone is a function of the total stress field and the rock mass competence. Typical widths for the abutment zone are between 5 and 15 metres and fracture frequency ranges from 20 fractures per metre to less than 1 fracture per metre within this zone.

### 7.3.1. EXPERIENCE AT PREMIER MINE ON THE UNDERCUT LEVEL

#### Monitoring

Stress problems were anticipated on the undercut drilling level and a monitoring programme was set up to measure the magnitude and the distance to which stress changes were felt ahead of the undercut.

Details of this monitoring are set out in Chapter 6, sections 6.5 and 6.6. The layout of the 615 metre undercut drilling level and the location of the monitoring stations is set out in Figure 6.3. Measured horizontal and vertical stress changes on the undercut level are set out in Table 7.2.

Monitoring showed that prior to the onset of continuous caving stress changes were less than 2 MPa. When continuous caving initiated and the undercut face became a true abutment, stress changes in the near field of the abutment zone, up to 20 metres ahead of the undercut face, ranged between 10 and 20 MPa. A stress change of 20 MPa was critical in the Hypabyssal Kimberlite and 15 MPa critical in the Tuffisitic Kimberlite Breccia as this level of stress change led to widespread induced fracturing. It impacted heavily on support efficiency.

**Table 7.2. Undercut stress change monitoring**

| | | STRESS CHANGE | | |
|---|---|---|---|---|
| STATION | DISTANCE TO FACE | VERTICAL MPa | HORIZONTAL MPa | REMARKS |
| 1 E | 20 | +15 | +9 | Caving |
| 1 W | 20 | +12 | -5 | Caving |
| 2 E | 20 | +20 | -5 | Caving |
| 2 W | 30 | +5 | +3 | Caving |
| 3 E | 20 | -2 | -2 | Prior to caving |
| 3 W | 25 | +1 | -2 | Prior to caving |
| 4 E | 20 | +7 | -6 | Prior to caving |
| 4 W | 2 | +2 | 0 | Prior to caving |
| 5 E | 8 | +2 | +1,5 | Prior to caving |
| 5 W | 6 | +2 | +1,5 | Prior to caving |
| 7 E | 6 | +1,5 | +2 | Prior to caving |
| 7 W | 6 | +2 | +1 | Prior to caving |
| 8 E | 15 | +20 | +3 | Caving |
| 8 W | 12 | +2 | +2 | Prior to caving |
| 9 E | 15 | 0 | -35 | In abutment zone |
| 9 W | 6 | -25 | +2 | In abutment zone |
| 10 E | 18 | +40 | -2 | Pillar |
| 10 W | 12 | +60 | +25 | Pillar |

Notes:
Monitoring continued until instrumentation failed

## Numerical modelling

Detailed numerical modelling was undertaken to predict stress levels ahead of the undercut face on the undercut level for different mining sequences and at various hydraulic radii (McKinnon, 1992). This predicted that stress levels would increase by 25 percent as the area of undercut increased from 14 400 square metres (at which stage caving initiated) to 34 650 square metres (by which stage the gabbro sill would have collapsed). When an area of 14 500 square metres had been undercut, stress levels were predicted to be between 2,5$q$ and 3,5$q$ (25 and 35 MPa). As the area of undercut increased to 34 650 square metres, stress levels were predicted to increase to between 3$q$ and 4,7$q$ (30 to 47 MPa). Awkward geometry such as protruding corners or re-entrant angles had the potential to increase stress levels by a further 10 MPa.

The effect on stress levels of advancing the undercut as a straight face, or as a V-shape, was modelled and showed that the shape of the undercut face had little impact on the total stress field, at least on the undercut drilling level. Leads and lags between adjacent tunnels did not increase stress levels, but did increase the width of the zone of heightened stress appreciably.

Figure 7.5.a. shows the predicted extent of Hoek and Brown failure in a 3 metre x 3 metre tunnel on the undercut level 20 metres ahead of the undercut (i.e. outside the abutment zone). Figure 7.5.b. shows the predicted maximum principal stresses around the same tunnel. The stress field that develops is eccentric and stress levels do not exceed 30 MPa. In the immediate vicinity of the tunnel stress levels do not exceed 10 MPa. Widespread failure is nevertheless predicted by the Hoek and Brown failure criterion. Figure 7.6. shows the same tunnel in an area within 5 metres of the undercut face, i.e. in the abutment zone. Figure 7.6.a. shows that the Hoek and Brown failure criterion indicates more extensive rock mass failure. In Figure 7.6.b., maximum principal stresses are still eccentric, but have doubled to a level of 60 MPa away from the excavation and reach 20 MPa at the corners of the tunnel.

**Stress effects on the undercut level: conclusions.**

Monitoring confirms the pattern of stress distribution predicted by numerical modelling and highlights some important considerations.

1. Prior to the initiation of caving, stress changes are low within one or two metres of the undercut face. Severe abutment stress loading will therefore not be felt on the extraction level excavations within the hydraulic radius. An advance undercut mining sequence and a timed approach to the installation of rigid interbolt linings is, therefore, not strictly necessary within this radius.

2. As continuous caving initiates, stress changes increase abruptly and stress changes of up to 2 $q$ (20 MPa) are felt up to 20 metres ahead of the undercut face. Adequate support must be installed well ahead of the anticipated abutment on both the undercut and extraction level as soon as the undercut face become as true abutment.

3. The broken rock within the abutment zone is unable to carry much stress and even negative stress changes are recorded here up to 15 metres from the undercut face. Support and drilling and blasting operations can be expected to be problematic.

4. Pillars can create unusually high stress levels that lead to aggravated induced fracturing that damage support and complicate ring drilling. These should be avoided by careful control of undercut drilling and blasting operations.

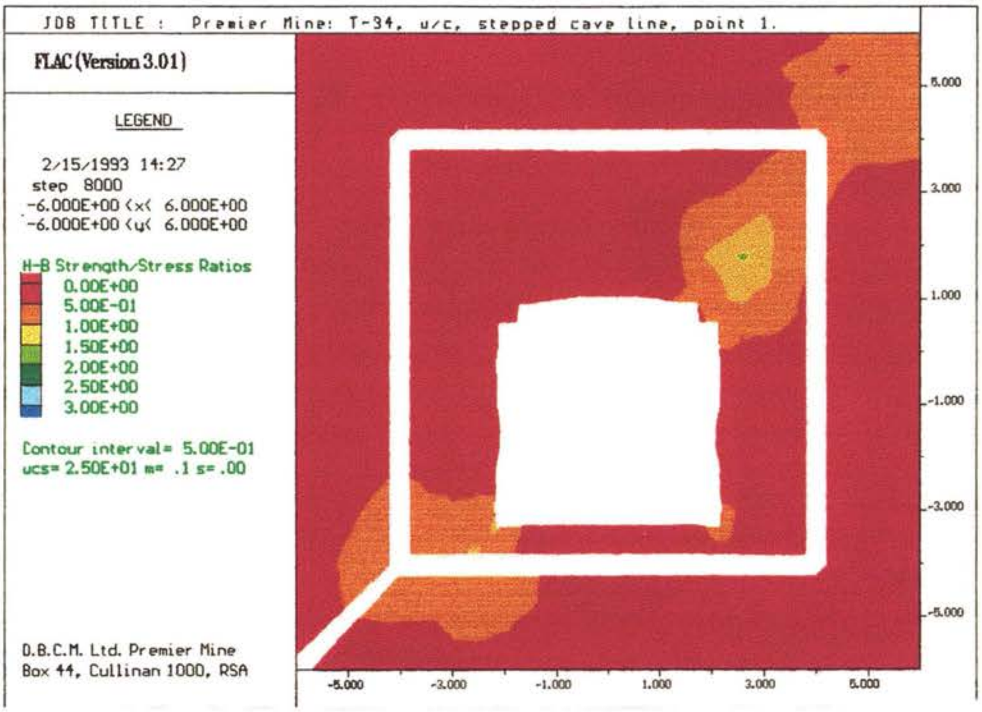



**Figure 7.5.a. Extent of Hoek and Brown Failure around an Undercut Tunnel 20 metres ahead of the Undercut showing Failure at Corners**

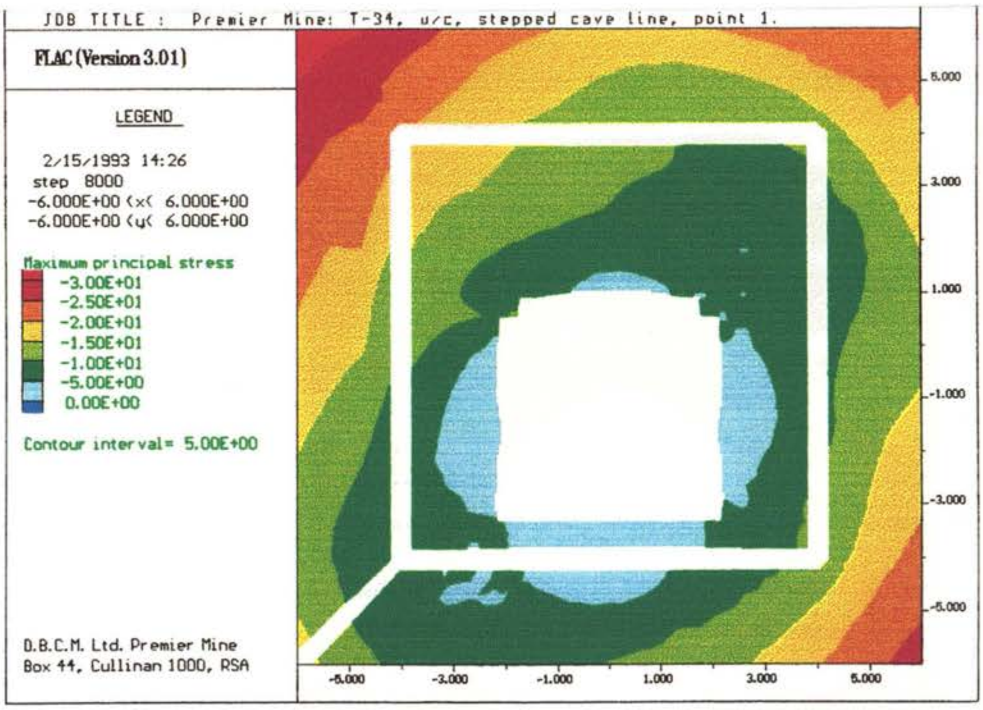



**Figure 7.5.b. Predicted Maximum Principal Stresses around Undercut Tunnel 20 metres ahead of Undercut**

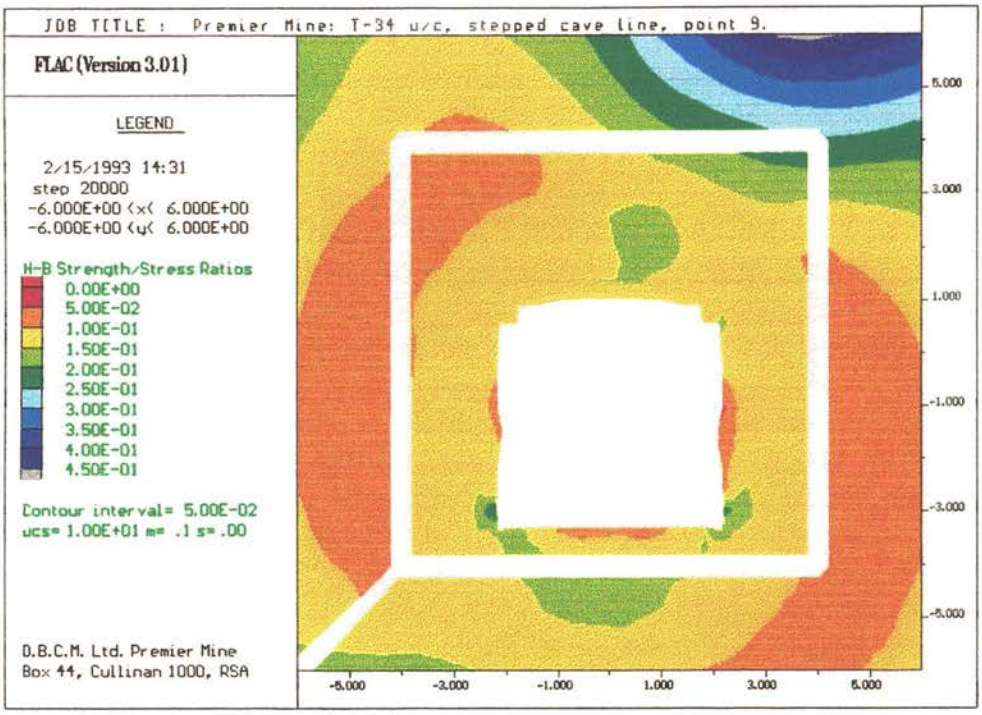



**Figure 7.6.a. Extent of Hoek and Brown Failure around an Undercut Tunnel 5 metres ahead of the Undercut showing widespread Failure**

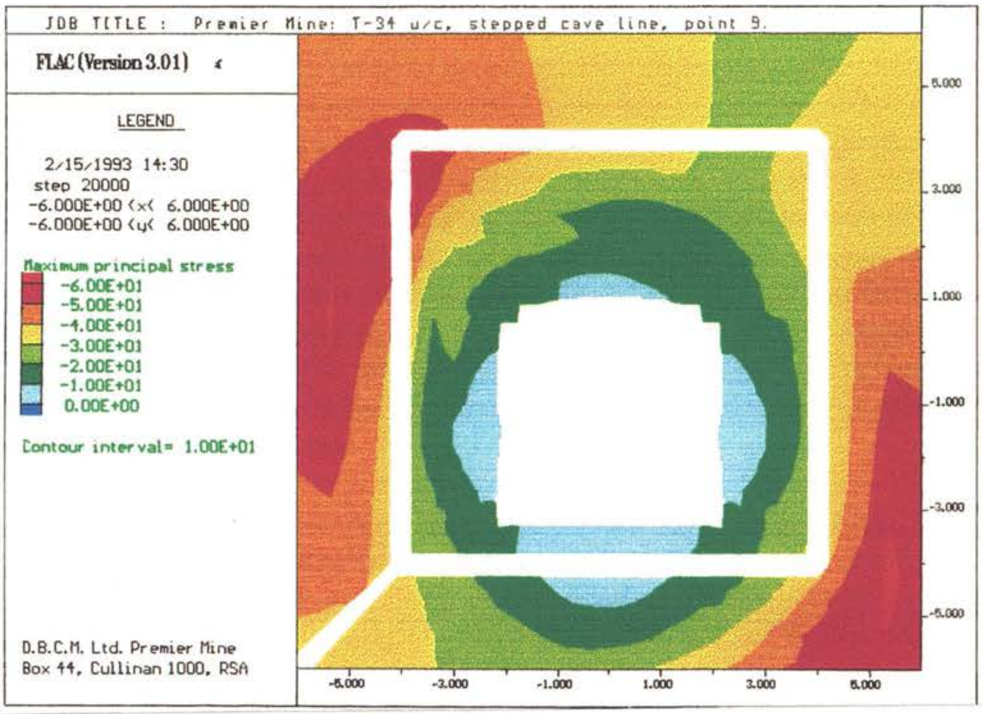



**Figure 7.6.b. Predicted Maximum Principal Stresses around Undercut Tunnel 5 metres ahead of Undercut**

## 7.4. STRESSES ON THE EXTRACTION LEVEL.

Stress changes on the extraction level have been widely studied and at least five levels of stress change have been recorded

1. The first level of stress change is that associated with tunnel development. This type of stress change is not unique to caving operations and modelling shows that stress levels around a spherical excavation can increase by a ratio of up to 1,5 times the primitive stress in the area. The geometry of a tunnel can have a stress raising effect especially at sharp corners. Support interaction analysis provides a clear picture of the stress changes that follow tunnel development and the support system required to ensure the stability of excavations in this situation. The stress changes that accompany tunnelling are similar in most mining operations and well documented (Hoek & Brown, 1981).

2. Drawbell development increases the extraction ratio on the production level to as much as 45 percent and to 50 percent immediately below the undercut level in a typical mechanised cave layout. Pillar attribution theory shows that drawbell development will increase stress levels by a factor of 2. The stress raising effect of awkward geometry can raise this considerably. The stress changes that result from drawbell development are poorly documented and there is little evidence of monitoring or modelling of these stress changes. This is largely because drawbell development is accomplished in a different way on almost every cave mine and the geometry of the drawbell is complex and does not lend itself to simple numerical modelling.

3. The stress changes associated with the undercut being run over an area are usually the most destructive in terms of support and rock mass damage on the production level. Stress changes of at least 3 $q$ are predicted and have been monitored on several mines. Again, awkward geometry can increase stress changes by up to 5 $q$. The stresses and rock mass damage that results in production drifts as the undercut is run over the extraction level are well documented (Brumleve & Maier, 1981. Brumleve, 1987. Ferguson, 1977. Kirsten & Bartlett, 1992. Cummings et al, 1984). Stress levels are raised by between 2 and 4 times the primitive stress level as the undercut is run over the production level. It results in movement along joints and fractures in the rock mass and, if stress levels are high enough, extensive induced fracturing. The blast damaged zone around excavations is often extensively damaged and even the core of minor and major apices might possess only a residual strength. Installed support, especially rigid interbolt linings are often destroyed. Induced fractures lead to short, ineffective embedment lengths of grouted steel tendons and extensive shearing takes place at the grout/rockbolt or grout/rock interface. This is followed by ravelling of the rock around the anchor heads and faceplates become ineffective

4. Once the extraction level has been developed and mining is in operation, various stress changes can occur. Poor draw control, compaction problems as a result of static pillars and wedge failures in the cave back can increase stress levels by several orders of magnitude. As drawing of the overlying orepile progresses, there is a reduction in stress levels. The effects of poor draw control and static pillars leading to compaction problems vary widely from mine to mine and are poorly documented.

5. Movement along major structures such as faults and contacts can lead to large stress changes well away from the actual cave area. The potentially damaging effect of wedge failures along major structures is recognised and numerical modelling techniques allow these effects to be anticipated (Rech & Lorig, 1992).

### 7.4.1. EXPERIENCE AT PREMIER MINE ON THE EXTRACTION LEVEL

#### Monitoring

The results and interpretation of monitoring results on the BA5 extraction level are discussed in detail in Chapter 5, Section 5.5. Monitoring results largely confirmed the generalised stress change and rock mass response graphically depicted in Figure 7.12.on page 170. Some stress measurement results are summarised in Table 7.3.

Stress change associated with drawbell development is not discussed in block caving literature, although drawbell development can impose a cycle of stress increase and decrease on minor and major apices at least as damaging as undercutting. The level of stress change at Premier was directly related to the way in which development was done. In drawbells where a remnant pillar was allowed to remain above the minor apex stress concentration occurred and a stress change of up to 20 MPa (total stress field 30 MPa) resulted. This was sufficient to cause extensive induced fracturing and made these pillars difficult to drill subsequently. Where these pillars went undetected, major problems resulted. Where development of adjacent drawbells undercut the minor apex destressing occurred and a negative stress change of between -4 and -14 MPa was recorded. It often allowed relaxation of the rock mass and resulted in considerable displacement on joints and fractures.

Prior to continuous caving initiating, stress changes of less than 4 MPa were measured on the extraction level in both the vertical and horizontal directions as the undercut was run overhead. Stress changes of up to 30 MPa (total stress 40 MPa) were measured beneath the undercut face on the extraction level immediately prior to the onset of continuous caving. Once caving had occurred stress changes of between 15 and 28 MPa were routinely measured. As the abutment moved away the primitive stress state was re-established. Both stress increase and decrease were remarkably sharp and initiated when the undercut face was only one or two metres from the monitoring station. The geometry of the extraction level undoubtedly influenced these abrupt stress changes. A stress change of 70 MPa was measured immediately below the pillar area where a stress change of 60 MPa had been measured on the undercut level.

**Table 7.3. Measured Stress Changes on the Extraction Level**

| STATION | ROCK TYPE | STRESS CHANGE (MPa) | COMMENTS |
|---|---|---|---|
| T21DP34 | HYP | 28 | UNDERCUT STRESS CHANGE |
| | | 34 | |
| T25DP34 | HYP | 18 | UNDERCUT STRESS CHANGE |
| | | 65 | REMNANT PILLAR |
| T17DP19 | TKB | -2 | APEX UNDERCUT |
| | | -4 | APEX UNDERCUT |
| T29DP21 | HYP | -4.8 | APEX UNDERCUT |
| | | 12 | APEX NOT UNDERCUT |
| T25DP18 | HYP | 18 | APEX NOT UNDERCUT |
| | | 3 | APEX PARTLY UNDERCUT |
| T21DP19 | HYP | -5 | APEX UNDERCUT |
| | | 27 | APEX NOT UNDERCUT |
| T25DP29 | HYP | -2.5 | UNDERCUTTING IN SLOT AREA |
| | | -4.0 | |

## Numerical modelling

Extensive numerical modelling was undertaken to predict stress levels in shotcrete linings and steel tendons around excavations and on the extraction level in the BA5 (Esterhuizen, 1987. Kirsten & Bartlett, 1992). Further modelling (Esterhuizen, 1991) identified areas of stress concentration and potential weakness around the production level in 4 popularly installed mechanised cave layouts. Figure 7.11. illustrates the result of numerical modelling using a three-dimensional, elastic, numerical code (BEEP) of 4 commonly used mechanised cave extraction level layouts. The plan view depicted is a plane taken 3 metres above the hangingwall (Esterhuizen, 1992). It shows that all layouts will experience high stress concentrations in the drawpoint brow areas and above the drawpoint crosscuts. The herringbone, offset herringbone and Henderson layouts all experience stress concentrations above the "bullnoses". This modelling was very useful in predicting zones of stress concentration and rock mass damage, using the Hoek and Brown failure criterion. Stress concentrations are not equally distributed around excavations on the extraction level. Bullnoses, camelbacks, drawpoint brows, crosscut portals and production tunnel footwalls are all areas of stress concentration and/or potential weakness as a result of geometrical considerations. Table 7.4. sets out the expected levels of stress change associated with various stages of cave mining in the BA5. The stress changes associated with drawbell development were not predicted.

**Table 7.4. Stresses on Tunnels on 630 Production Level at Premier Mine**

| STRESS CYCLE | VERTICAL STRESS (MPa) | HORIZONTAL STRESS (MPa) |
|---|---|---|
| TUNNEL DEVELOPMENT | 3.12 | 6.24 |
| DRAWBELL DEVELOPMENT | 6.24 (-4 - 15) | 6.24 |
| RETREAT OF UNDERCUT | 12.48 (14-65) | 6.24 |
| POINT LOADING | 26 | 6.24 |
| CAVE EXHAUSTED | 0.6 | 6.24 |

NOTES:
1. Figures are calculated total stresses
2. Figures in brackets are measured stress changes

The predicted stress levels were used in subsequent FLAC modelling to determine the extent of rock mass failure around a typical drawpoint crosscut on the production level (Kirsten & Bartlett, 1992). Modelling confirmed the pattern of stress distribution around a production tunnel and showed that considerable rock mass failure could be expected in weak Tuffisitic Kimberlite Breccia, and even in more competent Hypabyssal Kimberlite.

Stress change monitoring, the extent of induced fracturing, the large displacements suffered, together with extensive rock mass and support damage, resulted in high maintenance costs and production losses when drawpoints and tunnels had to be rehabilitated. A numerical model of the extraction and undercut levels was developed using FLAC3D (Leach, 1995). Monitoring data was used to accurately calibrate the model and predict the post peak behaviour of the damaged rock mass. The model was then used to determine an extraction ratio that would not result in unacceptable rock mass damage on the production level during advance undercutting.

The results of modelling are set out in Figures 7.7. through 7.9. Figure 7.7.a. shows the extensive rock mass damage that is induced on the extraction level when all development including drawbells is completed prior to the undercut being advanced overhead as a post undercut. The extraction ratio on the production level is calculated at 50 percent in this case. Figure 7.7.b. shows the limited rock mass damage induced when the same rock mass is subjected to the same stress levels but only production tunnels and drawbell crosscuts are developed prior to the undercut being advanced overhead. The extraction ratio in this latter situation is 23 percent. The view plane is horizontal and passes through the centre of the production level in both cases.

Figure 7.8. is of the same two models discussed above, but the view plane is vertical, parallel to and immediately adjacent to the production tunnel. Figure 7.8.a. shows the extensive rock mass damage found above, below and between the undercut and production levels when post undercutting is practised. Figure 7.8.b. shows that rock mass damage is found only immediately adjacent to excavations if an advance undercut is implemented.

Figure 7.9. is of the same two models but the view plane is vertical and at right angles to the production tunnel. The plane passes through the drawbell and production tunnel. Figure 7.9.a. shows that the entire rock mass between the production and undercut levels is in a state of potential shear failure and that zones of tensile and shear failure exist in the drawpoint brow and in the vicinity of the production tunnel. Figure 7.9.b. shows that, when an advance undercut is implemented, the zone of potential shear failure is limited to within a few metres of excavations and that tensile failure is minimal.

The results of this modelling are well correlated with experience and monitoring underground. In practical terms:

* dark blue zones represent intact rock with no support required.
* green zones need rockbolt reinforcement to prevent instability where these zones are adjacent to excavations.
* red represents zones where movement on joints and induced fracturing is found. The rock has some residual strength. Steel tendon support is needed to reinforce the rock mass and limit instability.
* white represents zones where the rock has only limited residual strength. Rockbolt reinforcement and extensive interbolt support in the form of reinforced shotcrete or concrete is needed to ensure excavation stability.
* other colour zones represent rock that in all cases needs substantial support to ensure long term stability.

Zones other than blue, green or red represent rock that is so badly damaged that it will be subject to extensive erosion as a result of secondary blasting and LHD impacts and will need continuous, expensive, support rehabilitation throughout the producing life of that mining area.

**Stress effects on the extraction level: conclusions**

Monitoring confirms the pattern of stress distribution predicted by numerical modelling and allows some conclusions.

1. The pattern of stress change modelled and monitored at Premier largely duplicates the generalised stress changes and rock mass response found in the literature. This confirms that the pattern of stress distribution in a cave mine is repetitive and predictable.

2. The development of large drawbell excavations needed for LHD extraction adds a stage of stress change to the model reported in the literature. The way and sequence in which drawbells are developed therefore needs to be planned to minimise rock mass damage.

3. Monitoring of the stress changes and rock mass response can allow the development of an accurate numerical model that can be used as a tool in planning layouts and mining sequences in a cave mine.

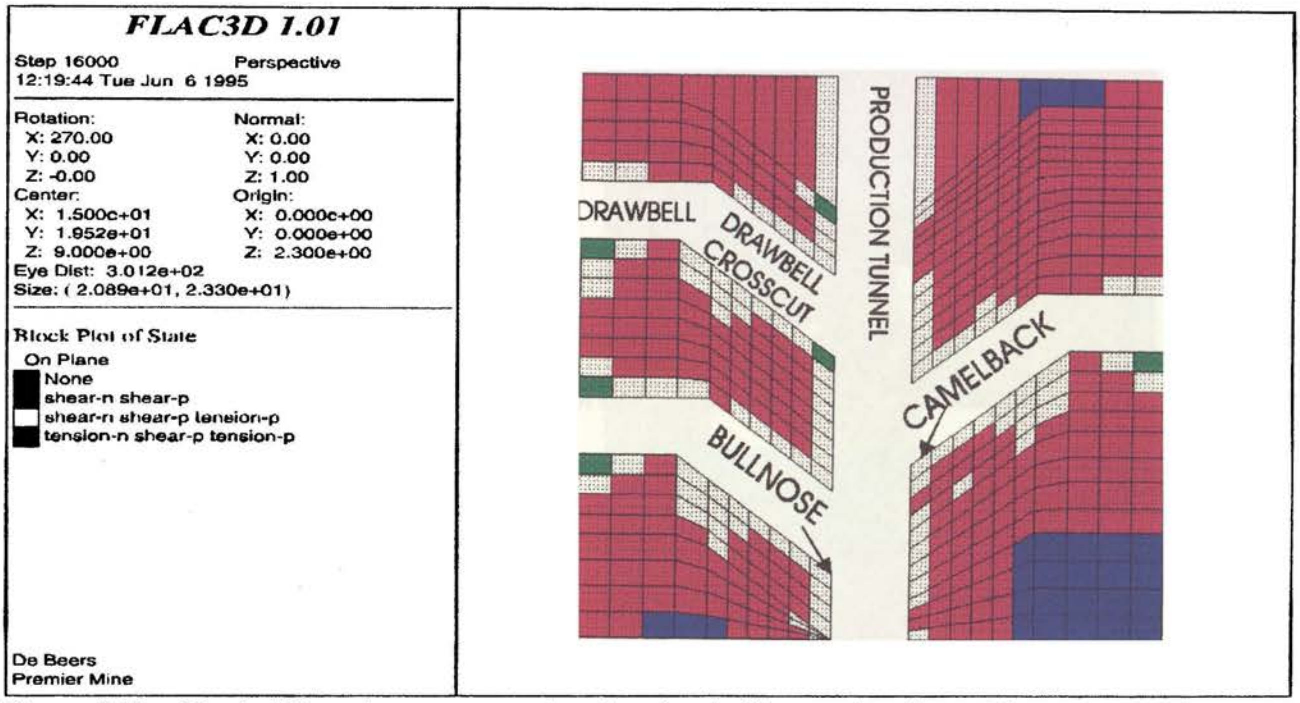



**Figure 7.7.a. Rock Mass damage on extraction level after post undercutting**

Horizontal plane through extraction level showing extent of rock mass damge after rock has been damaged by abutment stresses. All development on the extraction level is completed before the undercut is run overhead. The numerical model was accurately calibrated using stress and displacement data from monitoring underground

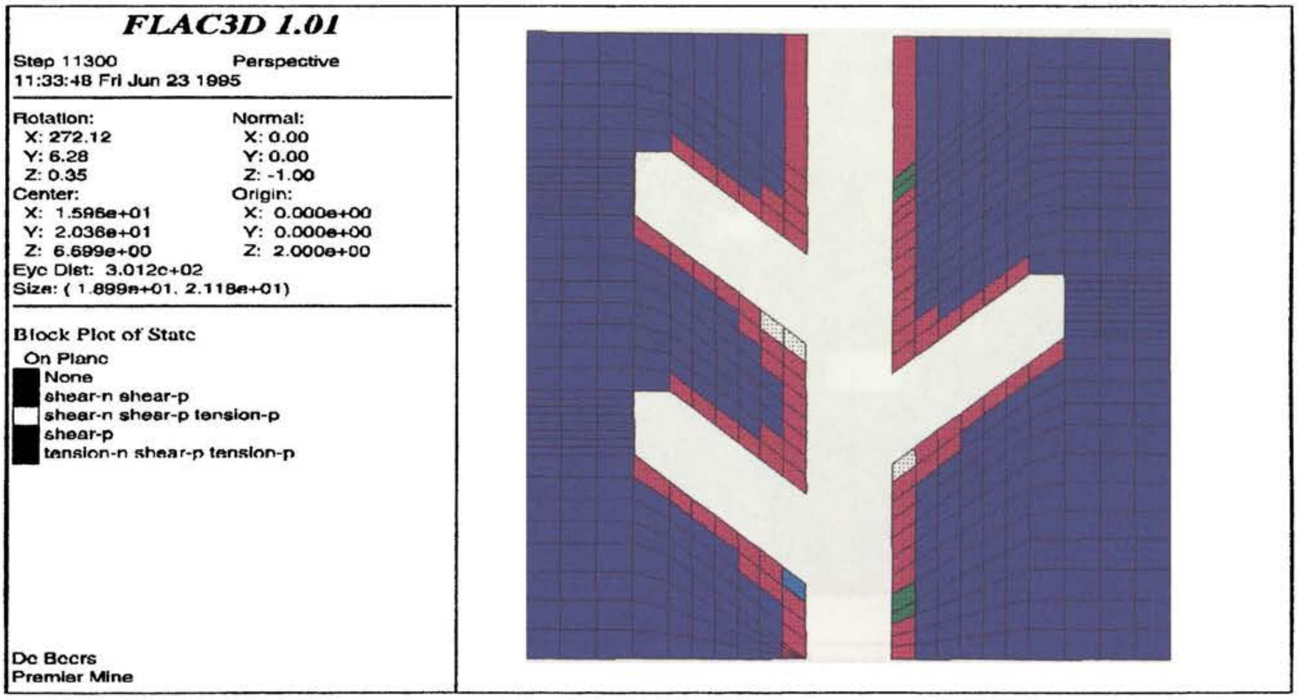



**Figure 7.7. b. Rock mass damage on extraction level after advance undercutting.**

The rock mass damage on the extraction level is largely confined to the blast damaged zone around tunnels when an advance undercut is used. Bullnoses and camelbacks are slightly damaged. The plane of view is the same as in Figure 7.7.a.

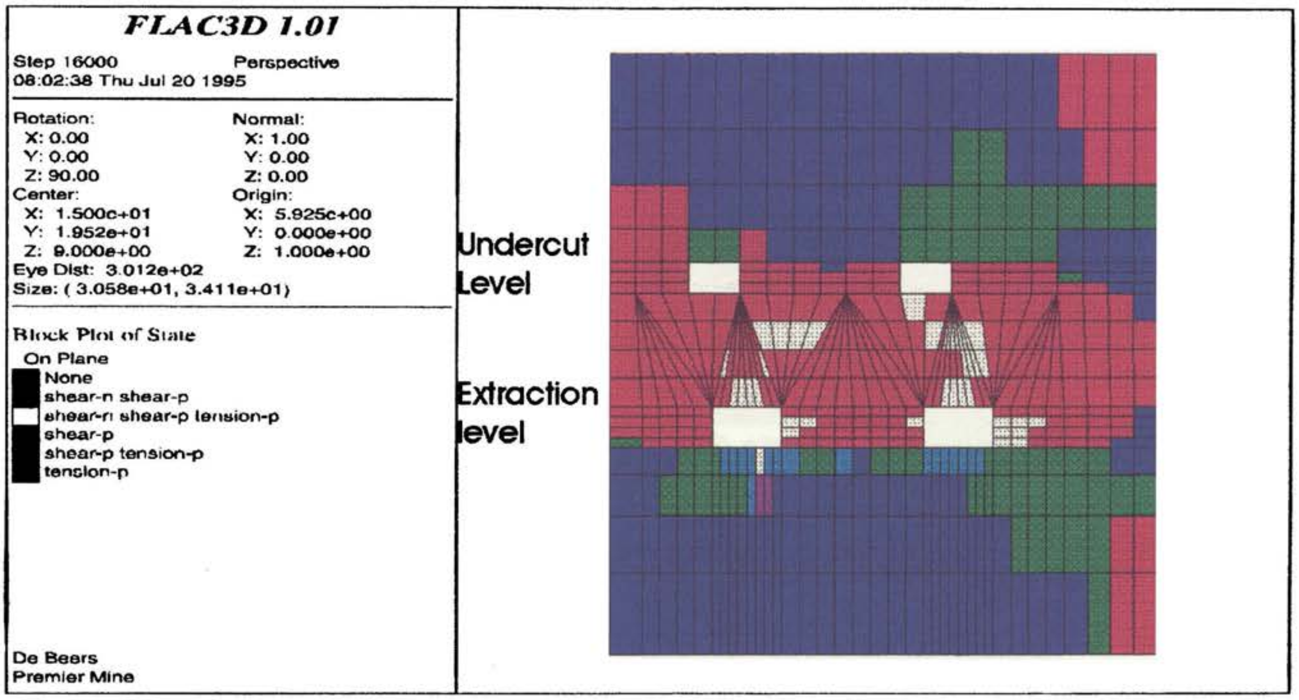



**Figure 7.8.a. Rock mass damage on undercut and extraction level after post undercutting**
This vertical section through the undercut and extraction levels shows the extent of rock mass damage predicted around excavations on these levels after they have been subjected to the abutment stresses predicted in the BA5. Monitoring showed that after the abutment had moved over the area, apices only had a residual strength and drawpoint brows were damaged.

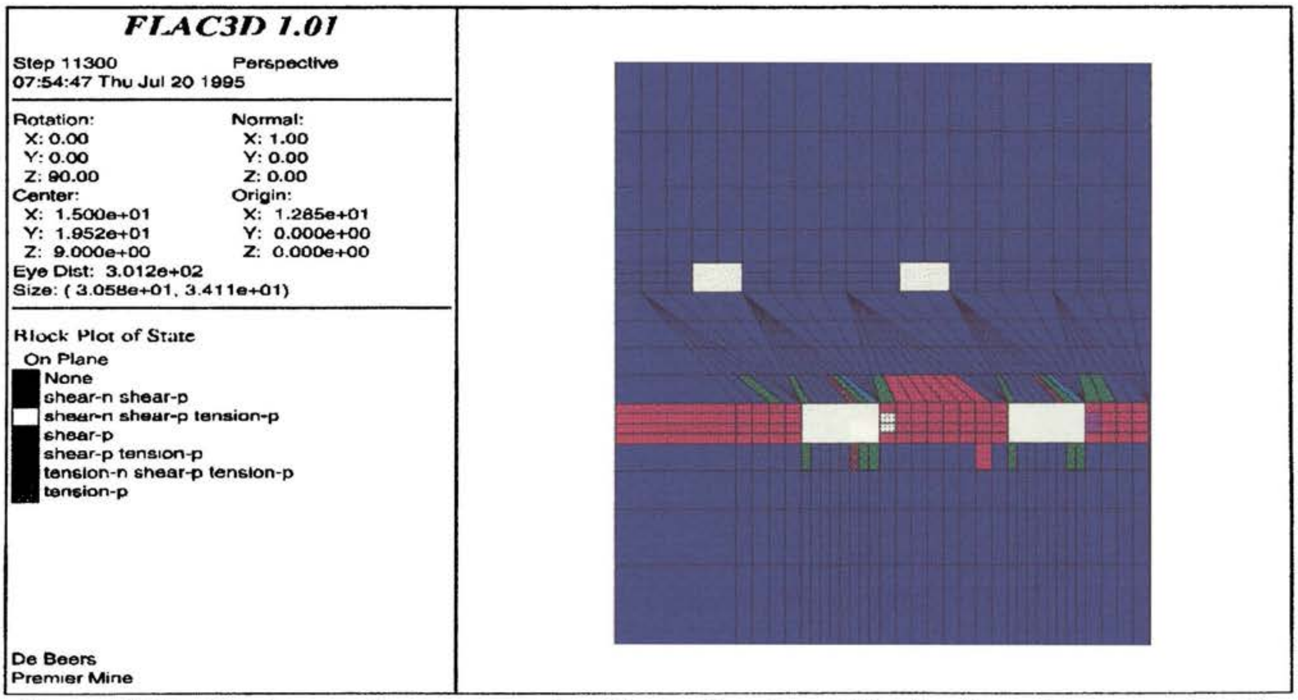



**Figure 7.8.b. Rock mass damage on extraction and undercut levels after advance undercut.**
This vertical section through the undercut and extraction level shows that numerical modelling predicts far less damage to the rock mass when an advance undercut is used. Apices and drawpoint brow areas are never subjected to high, damaging, abutment stresses. These predictions have been confirmed by monitoring and observation.

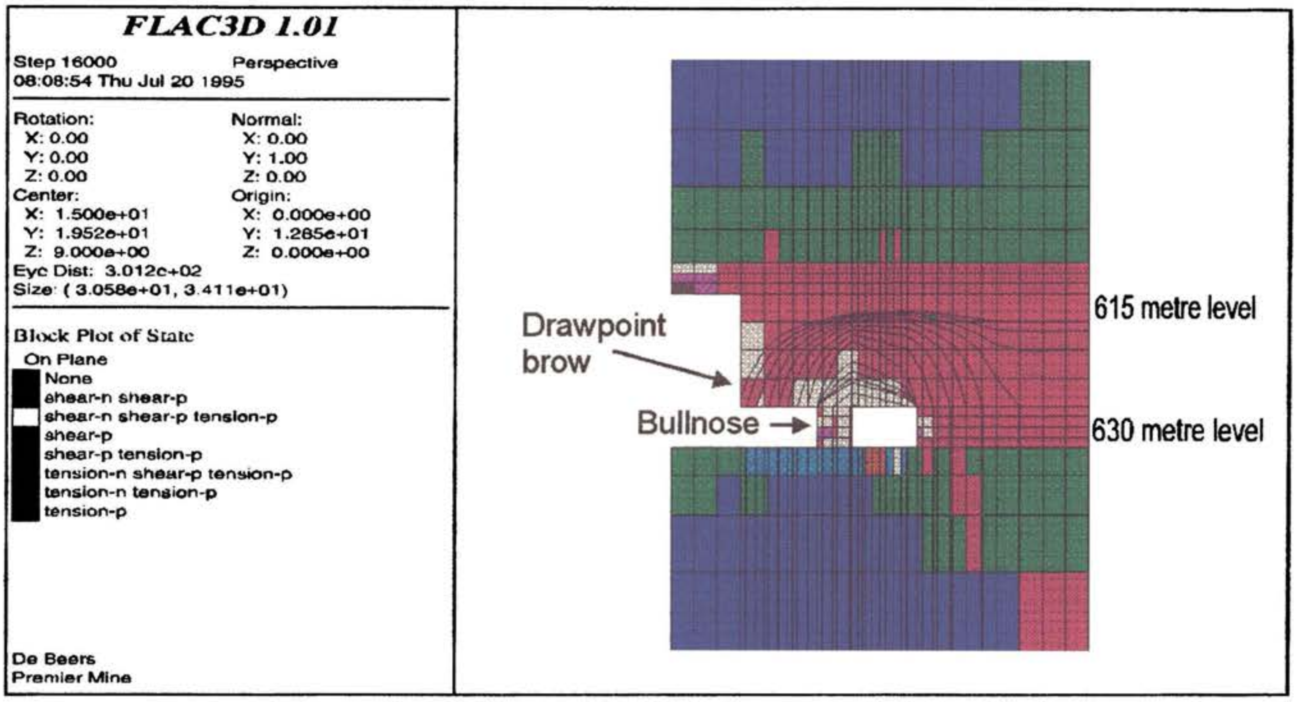



**Figure 7.9.a. Rock mass damage to undercut and extraction level after post undercut.**

This vertical section through the undercut and extraction levels is at right angles to the previous section. The calibrated numerical model predicts the extensive rock mass damage that has been monitored in drawpoint brows and bullnoses once these have been subjected to abutment stresses as the undercut passes over the fully developed extraction level.

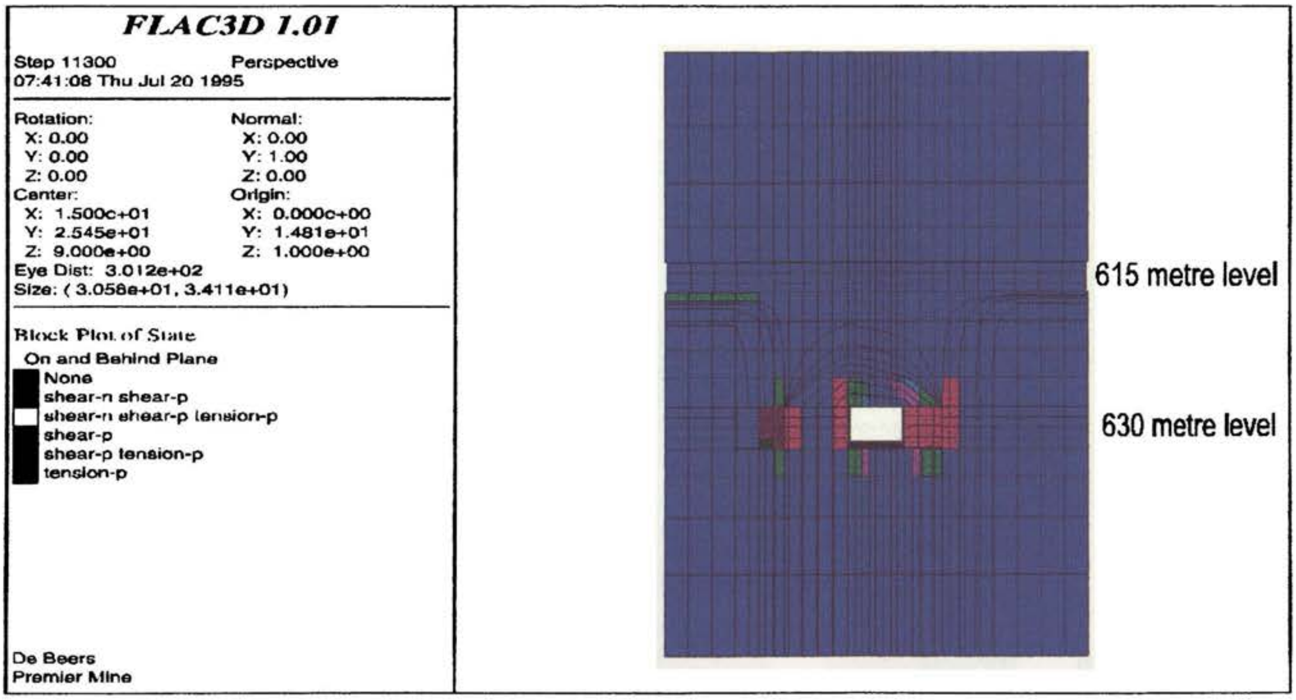



**Figure 7.9.b. Rock mass damage on undercut and extraction level after advance undercut**

This vertical section through the undercut and extraction level is at right angles to the vertical section in figure 7.8.a. and from the same viewpoint as the section above. Here, however, an advance undercut is used. Drawpoint brows and bullnoses are developed only after the undercut has passed overhead and are never subjected to abutment stresses. Observation and monitoring underground have shown minimal damage in these areas, confirming model results.


**Figure 7.11. Three dimensional modelling of several mechanised cave layouts**

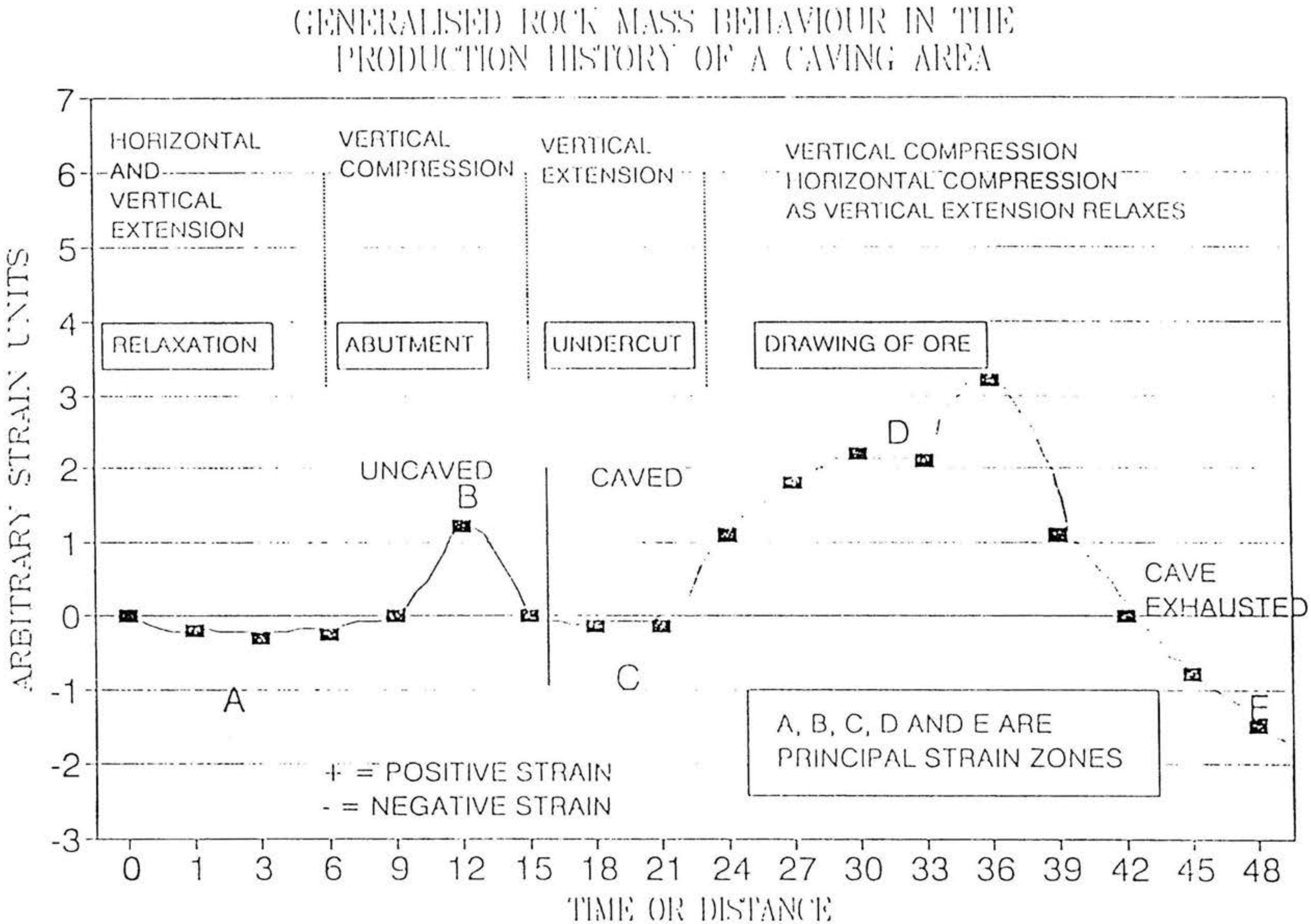



Figure 7.12. Generalised Stress Changes and Rock Mass Behaviour during the History of a Cave

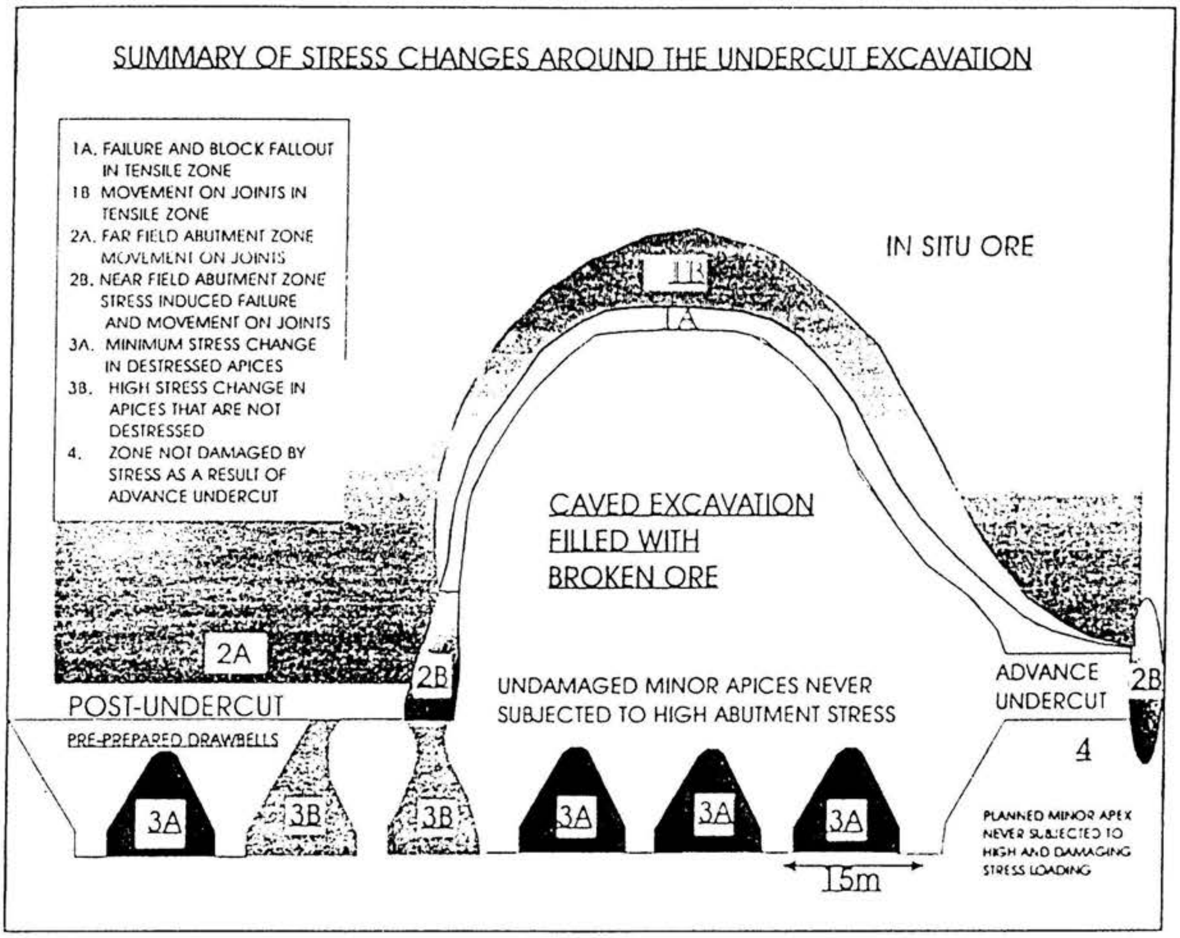



Figure 7.10. Summary of stress changes around the undercut excavation

## 7.6. CONCLUSIONS

Monitoring and numerical modelling at Premier, observations by the author on other cave mines, discussion with mining personnel on these mines and earlier models reported in the literature, have allowed the author to develop a general model of the stress pattern than can be expected around a cave excavation in kimberlite and the rock mass response that this stress pattern will elicit. Rock mass classification provides an essential Rock Quality Index in characterising the rock mass.

* Two zones that exhibited different responses to stress change can be defined in the cave back, above the undercut excavation. The first of these zones, the far field, (Figure 7.10. zone 1B) is defined by relatively low stress changes that are only sufficient to force movement along existing planes of weakness such as joints and fractures. These shear movements result in a lowering of cohesion and friction angle on the joint planes prior to caving occurring in an area. The distance to which these shear movements occur is a function of the condition rating of the joints in the affected rock mass. In Tuffisitic Kimberlite Breccia, this distance is 30 metres, in Hypabyssal Kimberlite 15 metres, in norite 20 metres and in gabbro 10 metres. The second zone (Figure 7.10. zone 1A), the near field, is immediately adjacent to the cave back. In this zone block fallout, shear

Taylor (Taylor, 1980) and the numerical modelling by Mahtab and Dixon (Mahtab & Dixon, 1976), summarised in Chapter 3, Section 3.2. characterises the rock mass behaviour that can be expected in this zone. In the Hypabyssal Kimberlite this zone is about 5 metres deep, in the gabbro about 1 metre and in the Tuffisitic Kimberlite Breccia 10 metres. As failure in this zone is through intact rock, the Hoek and Brown failure criterion is a good indicator of the depth of the zone. Stress levels in the zone are of the order of 1,5$q$. In the situation where the cave back approaches a pit bottom and a crown pillar is formed, as occurred in the gabbro sill at Premier mine, substantially higher stress levels can be expected as stresses are concentrated around the base of the open pit and compounded by stress concentrations in the crown pillar above the cave. Actual stress levels will depend on regional stresses and open pit depth. Stress changes around the undercut excavation and their effect on the rock mass is set out in Table 7.5.

* On the undercut level, two zones of stress change are defined which correlate with the zones noted above the undercut level. In the first zone (zone 2B), the near field, stress changes vary between 2 and 4 times the primitive stress level and can be substantially increased by unfavourable geometry. This level of stress change is sufficient to result in extensive induced fracturing ahead of the undercut face. These fractures are inclined and fracture spacing ranges from 50 to 600 mm. The distance to which fracturing is felt ranges between 5 and 15 metres. Again the distance to which the stress changes induce fracturing is a function of rock strength. In the second zone (zone 2A), stress levels are only sufficient to force movement along existing joints and fractures. The distance to which stress levels are felt is a function of the joint condition rating in the various rock types. Monitoring shows that this level of stress change is between 1 and 5 MPa. In the Tuffisitic Kimberlite Breccia the zone extends to 60 metres ahead of the undercut face, in the Hypabyssal Kimberlite 30 metres from the undercut face and, in the norite, up to 100 metres from the undercut face.

* On the extraction level, the same pattern of stress change and rock mass damage is noted. The level of stress change will vary and the extent of the damage caused by the stress changes will depend on the mining sequence (post undercut or advance undercut), the rock mass strength and the extraction ratio. If the undercut is moved over a pre-developed extraction level, induced fracturing is usually widespread and massive support is needed to ensure that the area is stabilised and that further erosion by secondary blasting and LHD impact is limited. Stress changes that result in movement on joints and in the blast damaged zone around tunnels and in drawbells are felt well ahead of the advancing undercut and can create major support logistical problems.

**Table 7.5. Stress Changes Around the Undercut Excavation**

| ZONE | THEORETICAL STRESS CHANGE | MEASURED STRESS CHANGE | ROCK MASS RESPONSE | COMMENTS |
|---|---|---|---|---|
| 1 | 0,5q (5MPa) | 2-4 | Movement on joints<br>Block fallout<br>Tensile and shear failures | TENSILE ZONE IN CAVE BACK |
| 2a | 0,5q (5MPa) | 0-4 MPa | Movement on joints<br>Block fallout | ABUTMENT ZONE PRIOR TO CONTINUOUS CAVING |
| 2b | 3-4q (30MPa) | 10-70MPa | Movement on joints<br>Extensive induced fracturing | ABUTMENT ZONE AFTER CONTINUOUS CAVING INITIATES |
| 3a | -10MPa | -10-12MPa | Relaxation - movement on joints and fractures | STRESS CHANGE IN APICES WHERE UNDERCUTTING IS ACHIEVED DURING DRAWBELL DEVELOPMENT |
| 3b | 0-20MPa | 4-30MPa | Movement on joints and fractures<br>Induced shear fractures | STRESS CHANGE IN APICES WHERE UNDERCUTTING IS NOT ACHIEVED DURING DRAWBELL DEVELOPMENT |
| 4 | -1q (-10MPa) | -2-14MPa | Relaxation on joints and fractures | STRESS CHANGE IN APICES PRIOR TO CONTINUOUS CAVING |

* Stress levels of between 2 and 4 times the primitive stress will occur on the undercut and extraction levels as the undercut moves through the area. If this level of stress change approaches one third of the uniaxial compressive strength of the rock, induced fracturing will occur and result in a greatly reduced rock mass rating in the affected rock. Excavations in this damaged rock on the undercut level are of a temporary nature.
On the extraction level, drawpoints must be maintained in this damaged rock for a considerable period of time. As much as 50 percent of the rock on the extraction level is taken out during the development of production tunnels, drawbells and drawpoint crosscuts. The minor apex defined as zone 4 is destressed during drawbell opening. Here stress changes of between -4 MPa and -15 MPa are measured. In the situation shown in zone 3B, apices are subjected to abutment stresses as the undercut is run overhead. It can only occur during post undercutting where the problem is compounded by a high extraction ratio. Thereafter, the damaged rock mass possess only a residual strength and the level of support needed will be an order of magnitude greater than that suggested by

most rock mass classifications and a programme of continual support rehabilitation will be needed. One side of the diagram shows development on the extraction level prior to the undercut being run over the area. On the other side of the diagram an advance undercut is illustrated with no pre-development on the extraction before the undercut is run overhead.

* It is important to note that stress changes of 2 to 4 times the primitive stress are only monitored once caving has initiated. This means that damaging stress changes are not felt on either the undercut or extraction levels within the hydraulic radius needed to induce caving. Apices within the zone designated 3A will not be subjected to high, variable, damaging stress loads. Drawbells supported with massive concrete lining to withstand erosion by secondary blasting and LHD impact can be installed within this radius.

* On the production level, observation at Premier and on several other cave mines (Stevens et al, 1987)) shows that cave "sit-downs" are the result of "pillar" failure with the sidewalls of the excavation failing. Typically it is the result of zones such as 3A failing as the undercut moves overhead. Hangingwall and footwall often suffers minimal damage. A universal failure criterion for the complex "pillar" made up of minor and major apices in a cave layout is difficult to develop.

* Stress levels are aggravated by a slow moving undercut, leads and lags between adjacent tunnels and unfavourable undercut geometry.

* The first four stages of stress change associated with cave mining were noted at Premier mine. Movement on faults and massive wedge failures fortunately never occurred.

* The average magnitudes of stress change were those predicted by numerical modelling, theory and experience on other cave mines.

* Large stress variations within each mining stage occurred. These could be related to rock mass strength as defined by the rock mass rating, unfavourable local geometry (stubs, leads and lags) and a slow moving undercut.

* The rock mass response to the various mining stages and support effectiveness is a function of the level of stress change and the rock mass structure.

* A small stress change induced movement on joints up to 100 metres from the abutment.

* A characteristic stress change threshold was needed to induce fracturing on the undercut and extraction level. In the Tuffisitic Kimberlite Breccia it was 15 MPa and in the Hypabyssal Kimberlite 20 MPa. This level of stress change was seldom felt more than 15 metres ahead of the undercut face in Tuffisitic Kimberlite Breccia and 5 metres ahead in the Hypabyssal Kimberlite.

The details of the model developed here by the author might not be directly applicable to other cave mines. The model does, however, provide guidelines in anticipating stress levels and rock mass response in a cave mine. A detailed geotechnical assessment of the orebody that can be used for rock mass classification and to establish accurate rock mass parameters for numerical simulation together with monitoring to establish stress levels and rock mass response can be used to develop an accurate model that can be a useful tool in planning in any cave mine using LHD extraction.

The ability to accurately predict the effect of stress on a cave mining operation is a useful planning tool as many cave mines move to exploiting ore reserves in coarsely fragmented ore at greater depths.

# CHAPTER 8

# FRAGMENTATION

**Statement.**

Fragmentation size distribution in a cave impacts on drawpoint spacing, the type of hangups that will occur and the frequency of these hangups. This in turn defines secondary blasting equipment, procedures and personnel requirements. The frequency of hangups and secondary breaking requirements determines production costs and the production tempo from the mining block. Fragmentation size distribution can be used to plan grizzly spacings, rock breaker requirements and pass diameters. If the above parameters can all be accurately predicted, it allows simulation of the mining operation, a powerful mine design tool.

In the initial stages of caving in the BA5 fragmentation was coarser than anticipated and this impacted adversely on production. It became important for the author to predict the fragmentation size distribution that would occur throughout the life of the cave in order to purchase additional secondary breaking equipment. A similar need had been identified by other cave mining operators. In collaboration with D.H. Laubscher and G.S. Esterhuizen, the author helped to define the parameters that would be needed to produce such an expert system.

Detailed scanline mapping by the author in the BA5, prior to caving and data collected from drawpoints in the BA5 by the author, allowed calibration of the expert system. A method of collecting the data that was needed for calibration and validation of the system had to be developed by the author as existing methods were of little use in measuring the frequency and dimensions of the large fragments that caused hangups and required secondary breaking. This information, together with data collected on the type and frequency of hangups as a result of varying fragment configurations in drawpoints was useful in allowing R. Kear (Esterhuizen et al., 1996) to develop a model that allowed the type and frequency of hangups to be predicted. The model was validated at Premier Mine and used in mine planning for the proposed Palabora cave.

Observation of the caving process at Premier and data collected by the author has allowed the author to validate and calibrate models that predict fragmentation size distribution and hangup frequency in a cave mine. It has allowed the author to recommend equipment requirements for secondary breaking in the new BB1E cave at Premier. It has further allowed mining simulation and prediction of the production potential of the BB1E mining block.

## 8.1. INTRODUCTION

In ore deposits where cave mining methods originated fragmentation was generally fine. Drawpoints were closely spaced and did not approach the limits at which theory indicated that ellipsoids would fail to interact and cause draw control problems. There was little need to predict the fragmentation size distribution to plan drawpoint spacing and secondary blasting procedures. As caving methods are used to mine orebodies with increasingly coarse ore, it has become important to predict the fragmentation size distribution that will report to the drawpoints at various stages of draw in order to determine optimal drawpoint spacings, secondary drilling and blasting procedures, equipment size as well as grizzly spacings and pass diameters. Where coarse ore is anticipated cave mining remains a drilling and blasting operation throughout the life of the cave and the prediction of fragmentation becomes important to plan secondary blasting procedures in order to maintain the required production tempo.

**Table 8.1. Fragmentation Size Distribution in some Cave Mines.**

| MINE | DRAWPOINT SPACING | % OF ORE SIZE | | | |
|---|---|---|---|---|---|
| | | +1,5m (1,3m3) | 0,6-0,9m | 0,3-0,6m | -0,3m |
| CREIGTON | 9,1 x 12,2 | 30 | 30 | 40 | |
| THETFORD (BELL) | 15 x 15 | 20 | 25 | 25 | 30 |
| MATHER | 4,0 x 8,5 | 5 | 10 | 15 | 70 |
| EL SALVADOR | 15 x 15 | VARIES | | | |
| SAN MANUEL | 4,6 x 5,3 | FINE ORES | | | |
| CLIMAX | 10,2 x 10,4 | 7 | 24 | 23 | 46 |
| GRACE | 6,1 x 9,1 | 50 | 20 | 20 | 10 |
| CORNWALL | 7,6 x 12,2 | 10 | 20 | 20 | 50 |
| URAD | 9,1 x 9,1 | 40 | 15 | 15 | 30 |
| HENDERSON | 12,2 x 12,2 | 30 | 25 | 15 | 30 |
| DE BEERS | 4,6 x 6,9 | 40 | 20 | 15 | 25 |
| PREMIER | 15 x 15 | 60 | 10 | 20 | 10 |
| PALABORWA * | 15 x 15 | 65 | 20 | 10 | 5 |

Adapted from Richardson 1981
* Planned
Thetford (Bell) 15% + 2m3
Premier originally 50% + 2m3, 20% + 2m3 after 50% draw
Palabora 15% + 2m3 (predicted)

The effects of fragmentation on cave mining have long been known qualitatively. " In summary, the resulting fragmentation from a structural domain depends on the original rock block size, the comminution effect, the strength of the rock, the thickness and nature of fracture fillings, the tendency of large blocks to float down to lower levels, and the fines likely to be formed by grinding or by travelling in from elsewhere." (Kendorski, 1982). RQD has been suggested as a measure of fragmentation size (Kendorski, 1982) but this has been refuted (Laubscher, 1981). A system to determine the "size distribution of ore fragments" is discussed in the literature, but no results are provided (Panek, 1981). Table 8.1. sets out the measured fragmentation size distribution at several cave mines. It should be noted that it is difficult to objectively determine the fragmentation size distribution accurately in a drawpoint and figures quoted here are estimates. The percentage of fragments that are greater than 2 cubic metres is useful as this is the size of rock that can be transported readily by an LHD with a 5 cubic yard (3,8 cubic metres) bucket capacity. The literature contains no way of quantitatively predicting the fragmentation size distribution that will report to drawpoints in a cave mining situation.

A requirement to provide an accurate estimate of the fragmentation size distribution that would report to drawpoints in order to plan secondary blasting requirements was identified at mines where coarse ore is being experienced (Premier Mine) and anticipated (Palabora). Based largely on the work and experience of Dr. Laubscher with assistance from the author and others engaged in cave mining, a computer programme was written by Mr. Esterhuizen to predict the fragmentation size distribution that would report to drawpoints at various stages of draw. Scanline mapping by the author in the BA5 at Premier was used to predict primary fragmentation. Methods of determining secondary fragmentation size distribution in drawpoints in the BA5 were developed by the author and results used to calibrate and validate the expert system. This data, together with data collected on the type and frequency of hangups at Premier has allowed numerical simulation to be used to predict the frequency of hangups that can be expected given the geometry of the drawbells and the predicted fragmentation size distribution that reports to the drawpoints (Esterhuizen et al., 1996). Simulation results have been validated by the author at Premier. This chapter sets out the results of the numerical simulations using these programs and the correlation found between predicted and measured results. The way in which these predictions can be used to plan and control secondary blasting is illustrated.

## 8.2. FRAGMENTATION SIZE DISTRIBUTION

The fragmentation size distribution that results as an orebody caves and blocks enter the top of the draw column and that which reports to the drawpoints have been defined as primary and secondary fragmentation, respectively (Laubscher, 1995).

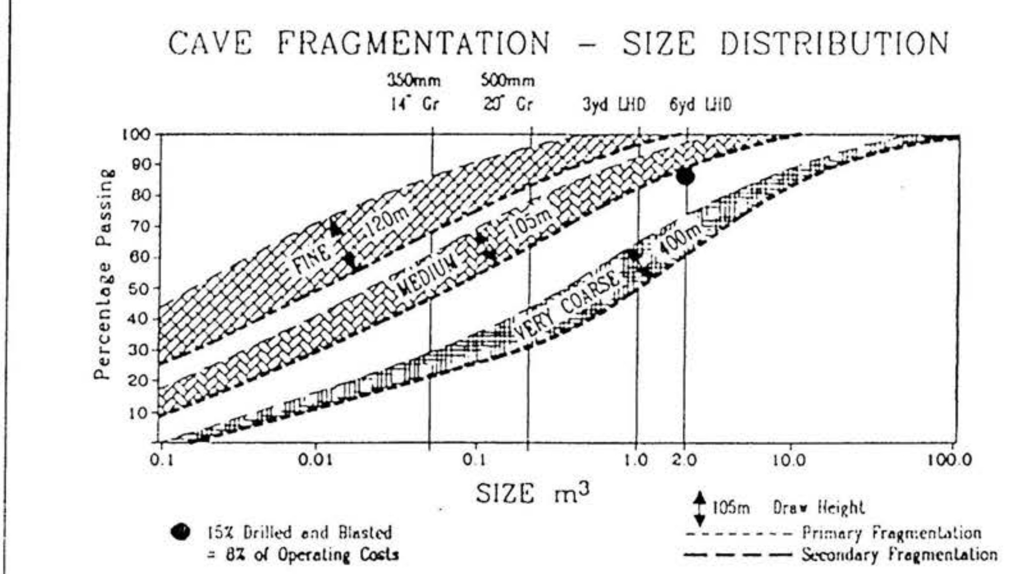



Figure 8.1. Primary and Secondary Fragmentation Size Distribution (Laubscher, 1994)

The fragmentation size distribution of in situ ore has been predicted (Kleine & Villaescusa, 1990. Villaescusa, 1991) based on the assumption that all joints and fractures will become rock block boundaries. Although this assumption has been shown to be valid in a blasting environment, many rocks that have travelled 100 metres or more in a draw column have been shown to contain numerous joints on arrival at the drawpoint. The primary fragmentation that will result as an orebody caves is a function of the in situ stress regime, rock strength and joint condition of the identified joint sets and these parameters are used to modify the assumption that all planes of weakness will become rock block boundaries (Esterhuizen, 1993).

As the fragments move through the draw column comminution occurs and the secondary size distribution that reports to the drawpoints is finer than that feeds into the top of the draw column. It is this secondary fragmentation size distribution that determines drawpoint spacing and blasting requirements.

Figure 8.1 shows the fragmentation size distribution in cave mines with coarse, medium and fine fragmentation and the effect of comminution on this fragmentation size distribution as it passes through a 105 metre high draw column (Laubscher, 1995). Important points to note are that the coarse ore is less comminuted than the fine ore as it moves through the draw column and that some extremely large fragments are predicted.

## 8.3. PRIMARY FRAGMENTATION

The controlling factors in primary fragmentation are the orientation, intensity, trace length, size and cohesion of the joints as these relate to the induced stresses in the cave back. The most commonly accepted model of joints is that they are elliptical or disc shaped discontinuities of varying diameter, location and orientation. Joints define the potential boundaries of rock blocks. Whether or not the joint will actually become a boundary of a rock block will depend on the stresses acting on the joint as well as the condition of the joint. Open joints have a high probability that they will define rock block boundaries as caving occurs. Cemented joints where the cement is stronger than the rock have a low probability of becoming a rock block boundary. Blasting fractures are usually not continuous and have only a localised effect in defining rock block boundaries. The large scale expression of the joint (planar, irregular, wavy) as well as the small scale expression (rough, smooth, slickensided) together with the stresses acting on the joint plane will determine the cohesion on the joint.

The initial data required to determine the primary fragmentation is gathered by detailed geotechnical scanline mapping of exposures, preferably in three orthogonal directions, to determine the orientation and frequency of joint surfaces in the orebody. Both the micro- and macro- attributes of the joint sets need to be recorded and are used to assign a joint condition rating to the joints in terms of Laubscher's RMR (Laubscher, 1991). In any rock mass joints represent planes of weakness and usually fail at relatively low levels of stress change. The stress component acting normal to the plane on the joint sets needs to be determined to predict the likelihood of various joints failing. If stress levels are high enough induced fractures will occur through intact rock and these need to be predicted as they influence the fragmentation size distribution.

The extent to which regional and induced stresses will fracture the rock mass in the cave back are also a function of rock mass strength. Rock strength will increase with a decrease in fragment size until the fragment possesses no inherent planes of weakness such as joints, fractures or veinlets. Rock strength is assessed in terms of Laubscher's MRMR. In order to predict whether failure will occur in terms of the prevailing stresses and rock mass strength, Hoek and Brown's failure criterion is used.

The programme uses the defined input parameters, rock mechanics principles and logic to predict the primary fragmentation size distribution that will occur as the orebody caves.

## 8.4. SECONDARY FRAGMENTATION

Fragments that detach from the cave back are comminuted as they progress through the draw column to the drawpoints below. Fragments with a high aspect ratio are more likely to break than fragments with a low aspect ratio. Fragments containing joints, fractures or veinlets are more likely to break than rocks without these features. Rocks involved in arching within the draw column will be subjected to point loading conditions and are liable to fail in shear. As the rocks move through the draw column grinding takes place and fines are generated. The distance of travel therefore impacts on the probability of comminution in the draw column whilst the fines generated move quickly through the draw column and skew the measured fragmentation at drawpoints. Fines provide a cushioning effect and inhibit comminution. Stress levels within the draw column as a result of gravity loading also need to be considered. Failure criteria must be applied in terms of rock strength and stress levels to assess the degree of failure of fragments within the draw column. All these factors are considered in the numerical simulation to predict secondary fragmentation in the draw column.

## 8.5. FRAGMENTATION SIZE DISTRIBUTION IN THE BA5 AT PREMIER MINE

The major determinants of fragmentation size distribution are rock type, which determines rock structure, stress levels and rock strength.

### Rock Types

Two distinct kimberlite types exist in the BA5 mining block at Premier. The Hypabyssal Kimberlite is a competent, reasonably well-jointed rock with a uniaxial compressive strength (UCS) of between 120 and 150 MPa and an RMR that ranges between 55 and 65. Monitoring and observation in this rock types indicates that caving is largely joint controlled. The second kimberlite type, Tuffisitic Kimberlite Breccia, is a relatively weak (UCS 80 -110 MPa), poorly jointed rock with an RMR of between 50 and 55. It should be noted that the average uniaxial compressive strengths quoted here are from selected core. Values have been derated to derive calculated MRMR values. Observation shows that caving is joint controlled, but that tensile and shear failure through intact rock contributes to the caving process.

Figure 8.2. illustrates the predicted primary fragmentation mass distribution of the Tuffisitic Kimberlite Breccia in the BA5. The dotted line graphs the simulated primary fragmentation size distribution. This predicts that fragments smaller than 2 cubic metres will make up 70 percent of the rock that enters the draw column. Six percent of fragments will occur as fragments larger

than 10 cubic metres. The solid line graphs the simulated secondary fragmentation size distribution after fragments have moved through 80 metres of draw column. The simulation predicts that 91,76 of the rock by mass will occur in fragments smaller than 2 cubic metres when they report to drawpoints on the extraction level. The brecciated nature of the kimberlite impacts on the fragmentation size distribution. Observation shows that the boulder/kimberlite interface is well defined as a result of secondary mineralisation on this interface. Locally this leads to improved fragmentation. Monitoring in drawpoints in the BA5 shortly after caving initiated showed that 66 percent of the ore occurred as fragments longer than 1 metre on one side. The average size of these fragments was calculated at 2 cubic metres.

Simulation of the Hypabyssal Kimberlite is illustrated in Figure 8.3. It shows that during primary fragmentation 55 percent by mass will occur in fragments smaller than 2 cubic metres and that, after about 30 000 tons have been drawn, 71 percent of this kimberlite type by mass will occur in fragments smaller than 2 cubic metres.

The BA5 is overlain by a 75 metre thick gabbro sill with typical columnar jointing. Caving in the gabbro sill is entirely joint controlled. The RMR in this rock types ranges between 65 and 75 and the average UCS of the rock is 284 MPa. Predicted cumulative mass percent of primary and secondary fragmentation is graphed in Figure 8.4. Caving of the gabbro results in the finest fragmentation size distribution but there is little predicted comminution as this hard rock moves through the draw column.

The country rock around the orebody is well jointed, competent norite with a UCS of 160 and an RMR of between 40 and 50. The norite is cut by a sub-horizontal joint set and 5 near-vertical joint sets. Shear zones in the norite are common. Joint condition ratings in the norite are low as a result of secondary mineralisation on joint planes and in shear zones and water percolating through the norite in places. Caving is joint controlled. Caving of the norite results in relatively coarse fragmentation, but still finer than the kimberlite. Simulation results are shown in Figure 8.5.

### Stress levels

Stress distribution around the cave has been calculated by extensive numerical modelling (McKinnon, 1992). It shows that stress levels immediately adjacent to the cave are sufficient to cause shear failure on all favourably oriented joint sets and failure through intact rock in both kimberlite types. Fragmentation size distribution in both kimberlite types is predicted to be finer than would be predicted by joint mapping alone. A progressively larger hydraulic radius has been needed to induce caving in the gabbro sill. This has been attributed to progressively larger stresses being applied across near vertical joint planes as the thickness of the gabbro "plate" reduces. Due to the competent nature of the gabbro, little failure through intact rock is predicted.

### Rock strength

The rock strength, as determined by the MRMR, together with modelled stress levels predicts considerable stress induced failure in the Tuffisitic Kimberlite Breccia in the cave back. Some stress induced failure is predicted in the Hypabyssal Kimberlite and none in the gabbro.

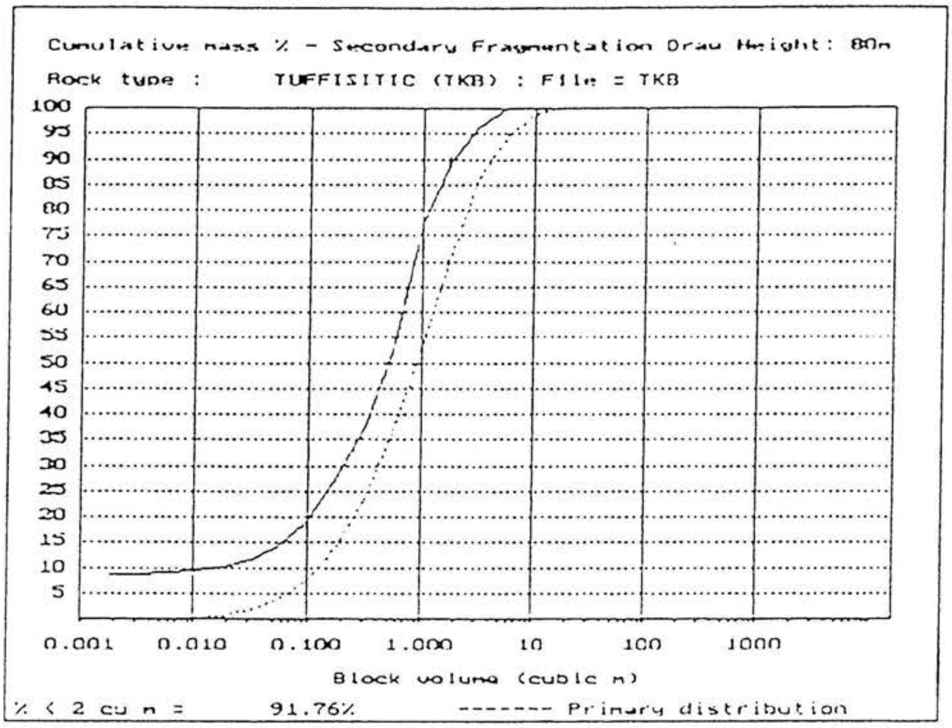



Figure 8.2. TKB Cumulative Mass % - Primary and Secondary Fragmentation

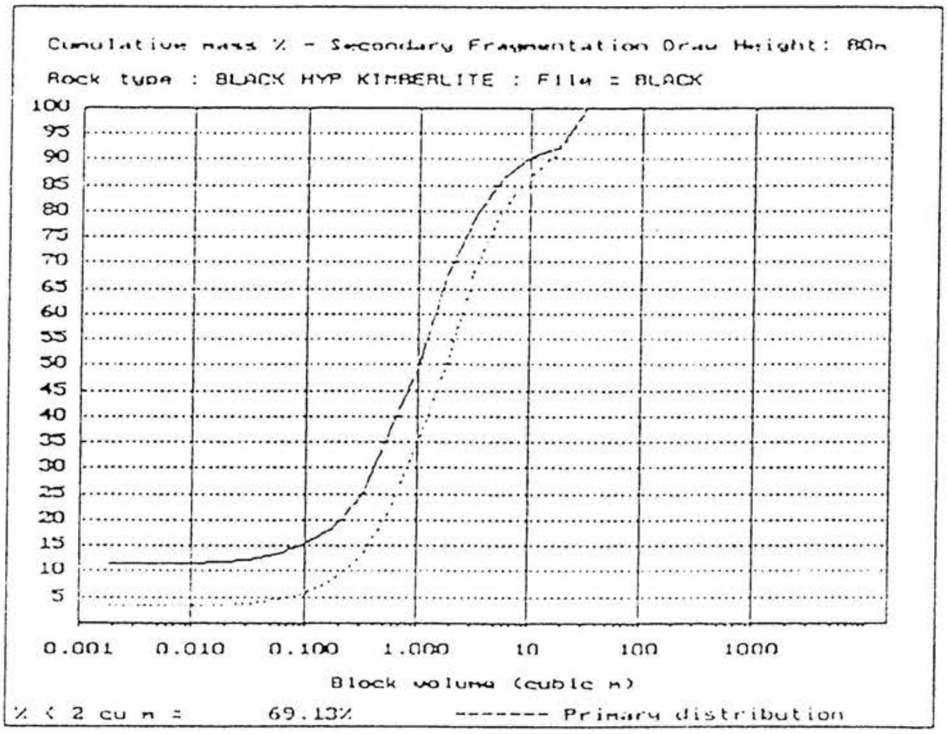



Figure 8.3. Hypabyssal Kimberlite Cumulative Mass % - Primary and Secondary Fragmentation.

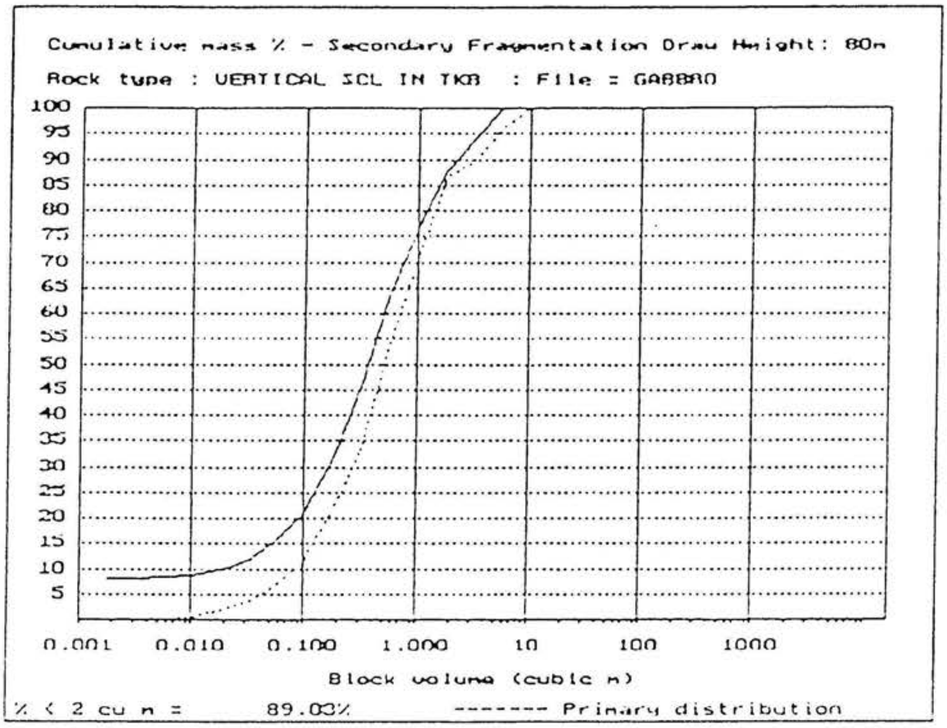



Figure 8.4. Gabbro Cumulative Mass % - Primary and Secondary Fragmentation

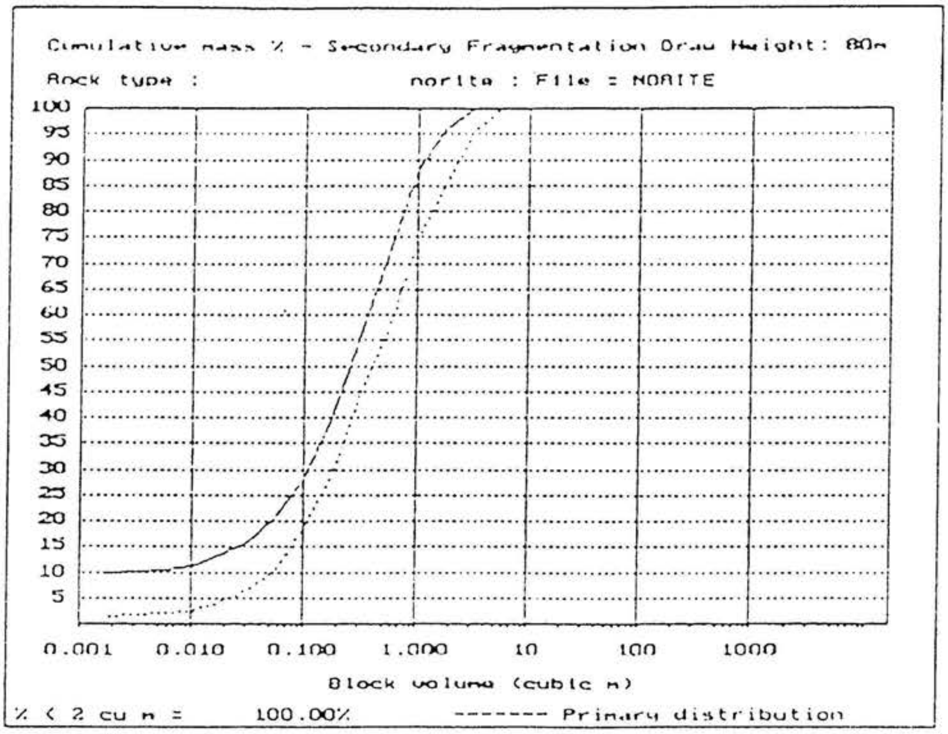



Figure 8.5. Norite Cumulative Mass % - Primary and Secondary Fragmentation.

**Determination of fragmentation size distribution in drawpoints.**

It is difficult to observe primary fragmentation distribution in the cave back as the area is usually inaccessible. The high undercut allowed a considerable airgap to form during caving of the kimberlite and the top of the draw column could be observed intermittently. The author observed that some extremely large fragments, tens of metres in dimension, detached from the cave back in Tuffisitic Kimberlite Breccia and entered the draw column. These soon broke into smaller fragments. Rock fragments detaching from the cave back in Hypabyssal Kimberlite were usually smaller. There was insufficient light in the airgap to allow photographs to be taken, nor was there any scale available. Immediately prior to the cave propagating through the gabbro sill into the open pit above, an intense beam of sunlight illuminated the top of the draw column daily. This allowed photographs of primary fragmentation in kimberlite to be taken (P.8.1). Several weeks later, after a large hole had appeared in the pit bottom, it was possible to take further photographs of primary fragmentation in the gabbro at the top of the draw column (P.8.2). Observation and the photographs allowed the author to see that caving of the gabbro had resulted in relatively fine fragmentation. Most blocks had low aspect ratios. No scale was available. Quantitative determination of primary fragmentation size distribution in the kimberlite and gabbro was therefore not possible.

The secondary fragmentation size distribution in the muckpile in drawpoints was first measured by digitising photographs and video footage of drawpoints. This gave a reasonable estimate of fragmentation in the finer size ranges, but provided little information on the frequency and size distribution of fragments larger than 2 metre along one side. Data on these larger fragments were required as the fragments caused most hangups and need to be blasted before they could be moved by LHD's. This data was finally collected by simple observation in drawpoints by the author and, later, to ensure reproducibility, by other geotechnical staff. The number of joints in fragments was noted, as were aspect ratios, the number of faces bounded by clear joint planes and the percentage of fines. Fines were defined as fragments larger than 100 millimetres in maximum dimension. A laser distomat was used to assist in estimating dimensions of fragments lodged in high hangups. Fragmentation data was collected at regular intervals. The results of this monitoring are set out in Table 8.2.

In preliminary work undertaken in the BA5 (Esterhuizen & Thompson, 1992) it was found that 30 percent of fragments reporting to drawpoints were larger than 1 metre but less than 3 metres along one side and that 25 percent of fragments were larger than 3 metres along at least one side. At this stage, fragments from the cave back were reporting almost directly to drawpoints with very little distance of travel or comminution in the draw column. Less than 5 000 tons of caved ore had been taken from the drawpoints.

Of the fragments that were larger than 3 cubic metres in size, the average aspect ratio was 1,52. Block volumes of the fragments ranged from 3 cubic metres to 17 cubic metres. Table 8.2. summarises the data collected on fragmentation size distribution in drawpoints as tons drawn increased from 5 000 tons to 30 000 to 60 000. The most noticeable effect was a decrease in fragments that measured 3 cubic metres or more from 55 percent to 12 percent. The number of fragments that were bounded by joint planes showed a substantial change probably as a result of rounding in the draw column.

P.8.1. Gabbro Primary Fragmentation in airgap above cave

P.8.2. Primary fragmentation into top of ore column after collapse of sill

**Table 8.2. Measured Parameters of Fragments in Drawpoints**

| ATTRIBUTE | 5 000 TONS DRAWN | 30 000 TONS DRAWN | 60 000 TONS DRAWN |
|---|---|---|---|
| Fragmentation Distribution in terms of frequency | 20% <100mm<br>50% <2000mm<br>50% >2000mm | 30% <100mm<br>75% <2000mm<br>25% >2000mm | 30% <100mm<br>86% <2000mm<br>14% >2000mm |
| Bounded by Joint Surfaces | 8% show 1 joint<br>24 show 2 joints<br>18 show 3 joints<br>6% show 4 or more joints | 40% show 1 joint<br>5% show 2 joints<br>0% 3 or more joints | 30% show 1 joint<br>5% show 2 joints<br>0% show 3 or more joints |
| Percentage of fragments containing joints | 8% | 5% | 5% |
| Percentage fines estimated | 5% | 5-10% | 30-40% |
| Average aspect ratio | 1,52 | 1,5 | 1,5 |
| Maximum aspect ratio | 10 | 5 | 3 |

The amount of fines showed a marked increase especially in the Tuffisitic Kimberlite Breccia. The number of fragments that contained joints did not change much as drawing progressed nor did the average aspect ratio of the blocks that reported to the drawpoints show much change. There was a marked decrease in the maximum aspect ratio showing that large blocks with high aspect ratios were quickly broken into finer fragments.

Monitoring was undertaken in drawpoints in both Hypabyssal Kimberlite and Tuffisitic Kimberlite Breccia at various stages of draw. Detailed observations were undertaken underground and photographs were taken of selected drawpoints at regular intervals to record any changes in fragmentation. Photographs P.8.3. and P.8.4. illustrate secondary fragmentation in both kimberlite ore types after 30 000 tons of ore had been drawn. The discs shown in the photographs are 0,5 metre in diameter.

Table 8.3. sets out a comparison between predicted fragmentation parameters as derived from the block cave fragmentation expert system using the data derived from the geotechnical investigation using Laubscher's rock mass classification and Hoek and Brown failure criterion. The correlation between measured and predicted parameters is generally good.

Analysis of secondary blasting records, together with draw control information, which records the types and frequency of the various types of hangups, and also with observation, provides a reliable estimate of the size frequency and mass of rock contained in size fragments larger than 2 cubic metres. Rocks larger than 2 cubic metres cannot be moved by an LHD and must be drilled and blasted using a short reach drilling rig.

**P.8.3. Fragmentation in Hypabyssal Kimberlite drawpoint after 30 000 tons drawn**

**P.8.4. Fragmentation in drawpoint in Tuffisitic Kimberlite Breccia after 30 000 tons drawn**

**Table 8.3. Predicted and Measured Fragmentation Distribution of Rock Types in the BA5**

| | PRIMARY FRAGMENTATION (% LESS THAN 2 CUBIC METRES) | | SECONDARY FRAGMENTATION (% LESS THAN 2 CUBIC METRES) | |
|---|---|---|---|---|
| | PREDICTED FREQUENCY | MEASURED FREQUENCY | PREDICTED FREQUENCY | MEASURED FREQUENCY |
| TKB | 92,7 | 90 | 96,4 | 92 |
| HYP | 93,5 | 92 | 96,87 | 92 |
| GABBRO | 94,5 | - | 98,84 | - |
| NORITE | 96,6 | - | 95,56 | - |
| | PREDICTED MASS GREATER THAN 2 CUBIC METRES | % BROKEN BY SECONDARY BLASTING | PREDICTED MASS GREATER THAN 2 CUBIC METRES | % BROKEN BY SECONDARY BLASTING |
| TKB | 30 | 25 | 10 | 12 |
| HYP | 48 | 45 | 31 | 21 |
| GABBRO | 47 | - | 39 | - |
| NORITE | 35 | - | 16 | - |
| SOME ADDITIONAL PREDICTED PARAMETERS (MEASURED PARAMETERS IN BRACKETS) | | | | |
| SECONDARY FRAGMENTATION | TKB | HYP | GABBRO | NORITE |
| % < 1 cu. metre | 50 | 50 | 39 | 64 |
| % > 10 cu metre | 0 (3) | 10 (4) | 0 | 0 |
| AVERAGE VOLUME | 0,37 | 0,38 | 0,20 | 0,39 |
| AVERAGE JOINTS PER BLOCK | 0,27 (0,40) | 0,75 (0,65) | 0,57 | 0,53 |
| MAXIMUM VOLUME | 9,42 | 13,33 | 9,43 | 9,43 |

On average one hole is drilled per cubic metre of rock for efficient breaking. The number of holes drilled by short reach rigs which are capable of drilling all the blockages that they can reach on a shift basis, provides a good estimate of the cumulative mass of rock larger than 2 cubic metres in size. The predicted and measured amount of material greater than 2 cubic metres does not correlate well and is attributed to fines enhancement at the base of the cave as a result of the preferential flow of fines to the drawpoints. This has been observed on numerous occasions.

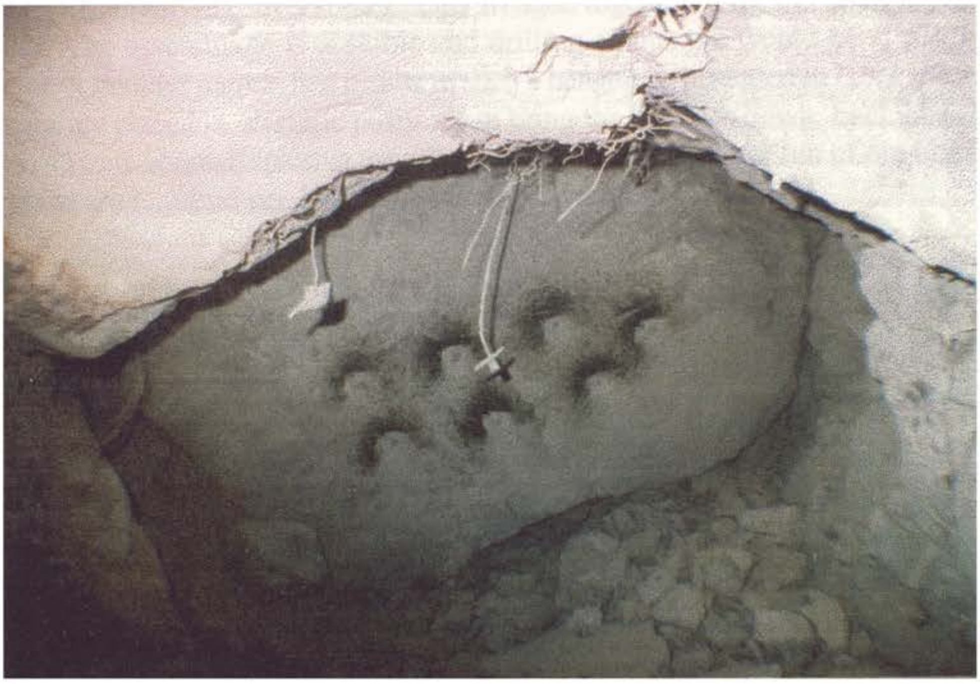
Another important parameter that is underestimated by prediction is the amount of material occurring in rocks greater than 10 cubic metres in the Tuffisitic Kimberlite Breccia. The cumulative mass of ore contained in fragments larger than 10 cubic metres in size is again measured indirectly. Observation shows that the size of fragments that cause high hangups which must be drilled by a long reach rig, are larger than 10 cubic metres in size. A hole drilled by a long reach rig allows 5 tons of rock, on average, to be fragmented. The number of holes drilled by the long reach rig therefore correlates well with the mass of ore contained in fragments larger than 10 cubic metres in size. Although this is an indirect way of estimating the volume of ore contained in large fragments, the method has been shown to provide consistent results and has been accepted as statistically accurate by the author. Large fragments lead to high hangups. Accurate prediction of the frequency of occurrence is therefore important. Table 8.3 illustrates an important aspect of fragmentation size distribution. In terms of the predicted frequency of occurrence only 7,3 percent of fragments will be larger than 2 cubic metres. In terms of mass, however, 52 percent of the material reporting to drawpoints will be in the form of fragments larger than 2 cubic metres which will have to be drilled and blasted.

## 8.6. HANGUPS AND SECONDARY BLASTING

Hangups, blockages and secondary blasting create problems in all caving mines. An average of 300 blasts are required for a production rate of 735 000 tons (2450 tons per blast) per month at Henderson Mine (Keskimaki K.W. & Wagner R., 1994). At Premier Mine, an average of 3 500 blasts are required for a production rate of 160 000 tons (45 tons per blast) per month. Tons per blast has varied from 35 tons per blast in the initial stages of caving to the current level and secondary blasting is reducing progressively with tons drawn. It is predicted, however, that secondary drilling and blasting will remain an important part of the mining operation in the BA5 until the cave has been drawn to completion. Secondary blasting must therefore be efficient and cost effective.

Three types of blockage have been identified at Premier and confirmed by observation at several other cave mines using LHD's for extraction. High hangups can occur when a single large fragment lies across the major apices that define the top of the drawbell. Such a hangup can occur 15 metres above the extraction level and is difficult to bring down. In the BA5 only one such hangup has occurred in mining 5 million tons of ore. More typically, a single large fragment lies across the minor apices or some distance into the drawbell. This has been termed a high hangup and requires a long reach (9 metres at Premier ) drill to drill holes into the fragment. Explosive and a detonator are placed through hollow, extendable aluminium rods using an Anfex loader and a compressor on the drill. On average, such a hangup is 10 cubic metres in size and six holes are required to break such a rock. Each hole is 2 metres long and 32 millimetres in diameter and contains 100 grams of ammonium nitrate fuel oil explosive (Anfex) to give an explosive usage of 22 gm of explosive per ton of ore blasted. On average 27 tons (3 percent) are broken in this way per 1000 tons of ore produced. Photograph P.8.5. shows a single, large kimberlite fragment across a drawpoint throat into which seven 2 metre long, 32 millimetres diameter holes have been drilled. The holes were subsequently charged with 100gm Anfex using hollow, extendable, aluminium rods and an Anfex blower. It is estimated that the fragment contained 30 tons of kimberlite.

A blockage occurs when a large rock reports at the throat or on the floor of a drawpoint and effectively prevents loading by the LHD's. At Premier, these rocks are more than 2 cubic metres in size. Rocks are drilled and blasted using a short reach (5 metre) drill. One stick of 20 millimetres by 25 millimetres 63 gram Dynagel cartridge explosive and detonator is placed by hand in each hole. On average, each hole breaks 3 tons of ore to give an explosive usage of 21 grams per ton of ore blasted. On average, 36 tons (4 percent) are broken in this way per 1000 tons of ore produced. Low hangups are illustrated in Photograph P.8.6.



**P.8.5. High Hangup with single large fragment across throat of drawpoint**

**P.8.6. Low hangup with several large fragments in drawpoint that prevent loading.**

Boulder jumbles involving several fragments wedged together in the drawbell to form an arch often occur. The assemblage is unstable and drilling equipment would be at risk in such an environment. A boulder jumble that results in such a hangup is illustrated in P.8.7. 12,5 kilogram Anfex bombs are placed in strategic positions to bring the hangups down. Each blast produces, on average, 43 tons of ore. Explosive usage is therefore 290 grams per ton of ore blasted. Three or four bombs are required per 1000 tons of ore produced.

**P.8.5. Unstable boulder jumble with several fragments arching across drawpoint**

It is therefore estimated that 45 kilograms of explosive is used in secondary blasting per 1000 tons of ore produced. Explosive usage in ore that is drilled and blasted is 19 grams per ton. On average, 213 tons of ore is directly blasted per 1000 tons of ore produced to give an explosive usage of 210 grams per ton of ore blasted. This shows the extreme inefficiency of bombing as a means of secondary breaking. On average, 45 grams of explosive is used per ton of ore produced. This is a substantial improvement on secondary blasting above the sill where lay-on charges and bombs were used and explosive usage was 250 grams of explosive used per ton of ore produced.

Figure 8.6 graphs the frequency of the various types of hangups experienced at Premier. Large high (large/h) hangups are drilled with a long reach Atlas Copco (AC) rig and explosive is placed through aluminium rods. Large low (large/l) hangups are drilled with a short reach Tamrock Commando (TC) rig and hand charged. High and low rock clusters (cluster/h and cluster/l) are brought down using 12,5 kilogram Anfex bombs. The graph can be used to plan secondary blasting procedures and the production tempo from any planned cave at Premier. Based on Premier's blasting efficiencies, Figures 8.7 and 8.8 set out drilling equipment requirements, explosive usage and tons that need to be blasted per time period. This can be used to plan secondary blasting procedures and production tempos. For example, Figure 8.8. shows that if 8000 tons per shift is required from the BA5, it will require 87 Atlas Copco (AC) holes drilled into high hangups, 63 Tamrock Commando holes drilled into low hangups and 28 bombs

**Table 8.4. Measured and Predicted Secondary Blasting Parameters**

| PARAMETER | PREDICTED | MEASURED |
|---|---|---|
| Secondary fragmentation size distribution | TKB: 30% by mass | TKB: 21% by mass |
| | HYP: 31% by mass | HYP: 21% by mass |
| Hangups per 1000 tons of ore produced<br><br>Calculated on basis of hangup frequencies<br><br>Calculated on basis of oversize mass | Total: 22<br>High Hangups: 1<br>Blockages: 16,5<br>Boulder jumbles: 5<br>Total Tonnage: 292 tons (29%) | Total: 32,5<br>High Hangups: 1<br>Blockages: 12<br>Boulder jumbles: 3,5<br>Total Tonnage: 214 tons (21%) |
| | 32% by mass of the rock will reside in fragments larger than 2m3 | |
| Explosive usage per 1000 tons of ore produced | 600 gm Anfex loaded<br>825 gm dynagel<br>62 500 gm in bombs<br>Total: 63 gm/ton | 600 gm Anfex loaded<br>600 gm dynagel<br>43 750 gm in bombs<br>Total: 45 gm/ton |
| Drilling required per 1000 tons produced<br>Atlas Copco<br>Tamrock Commando | <br>12 metres<br>16,5 metres | <br>12 metres<br>12 metres |

Please note: high hangups, blockages and boulder jumbles are collectively referred to as hangups in this thesis for ease of expression.

will have to be placed. Figure 8.7. shows the secondary blasting explosive usage associated with this production rate. Table 8.4. summarises predicted and measured secondary blasting parameters. The data set out in tables 8.3. and 8.4. show that sufficient accuracy can be obtained from numerical simulation that predicts the fragmentation size distribution and the number of hangups of various types to purchase equipment and plan and control secondary blasting procedures. Table 8.4. illustrates the poor blasting efficiency of bombs as compared to drilling and blasting. The explosive energy that does not go into breaking rocks damages drawpoint brows and support.

The author has compiled data relating to the frequency and types of hangup, together with secondary drilling equipment performance and secondary blasting procedures for Premier Mine. Graphs such as Figures 8.7 and 8.8 can be used to plan equipment requirements and explosive usage in new caves.

This data would not be available for a proposed mine. The expert system described in this chapter, together with Kear's simulation can be used to estimate the type and frequency of hangups that can be expected for a given fragmentation size distribution. Secondary drilling equipment performance and blasting procedures taken from operating mines can then be used to plan secondary blasting operations and production.

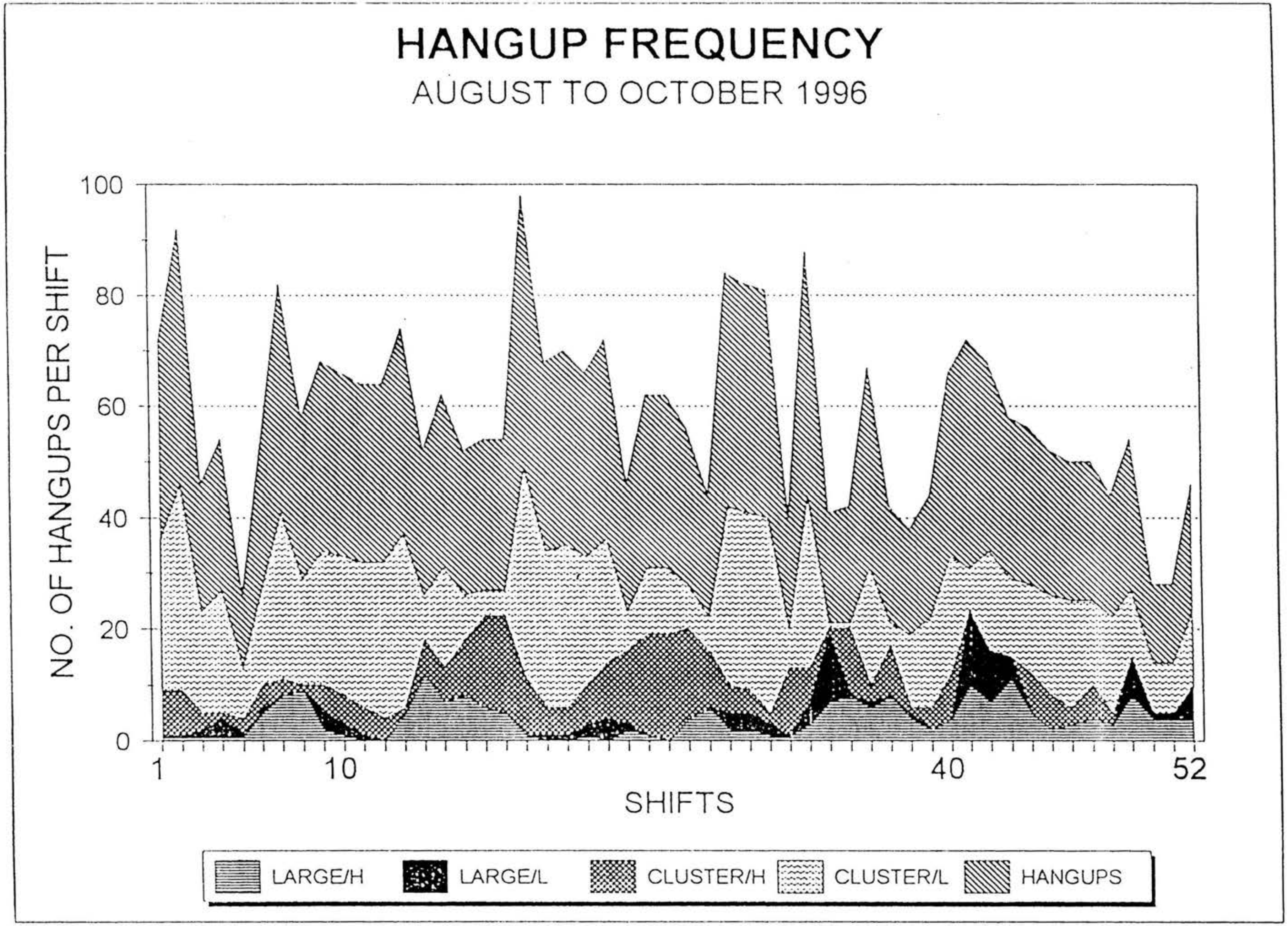



Figure 8.6. Hangup types and frequency

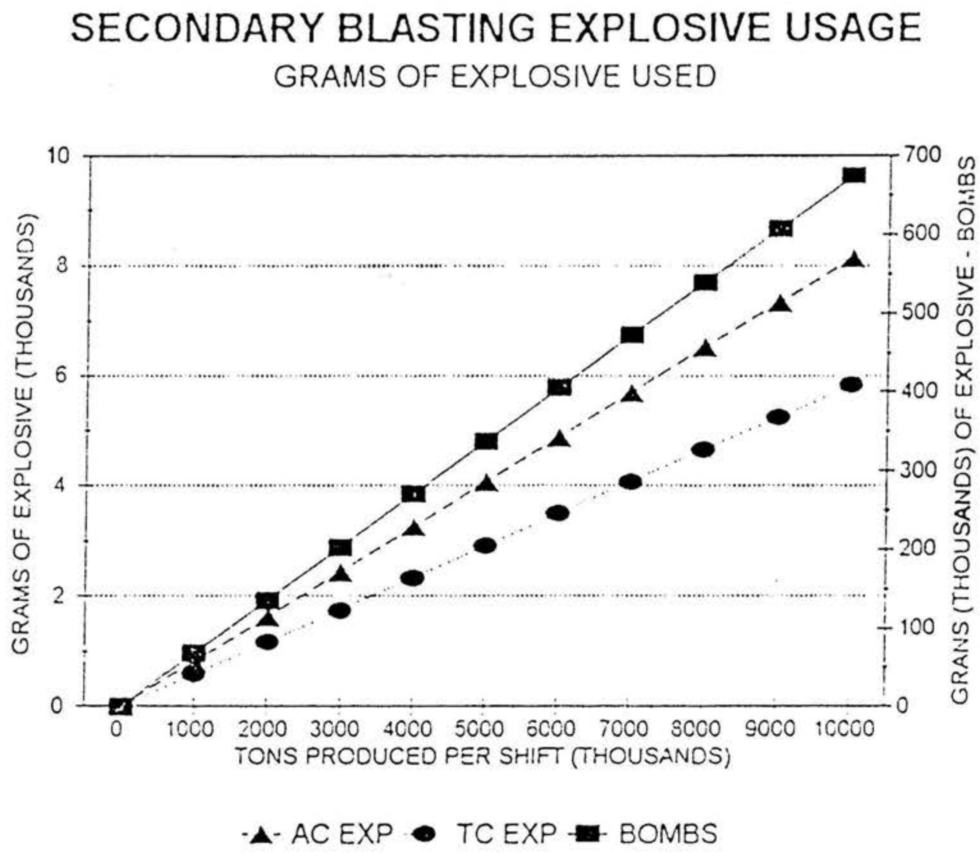



Figure 8.7. Secondary Blasting Explosive Usage

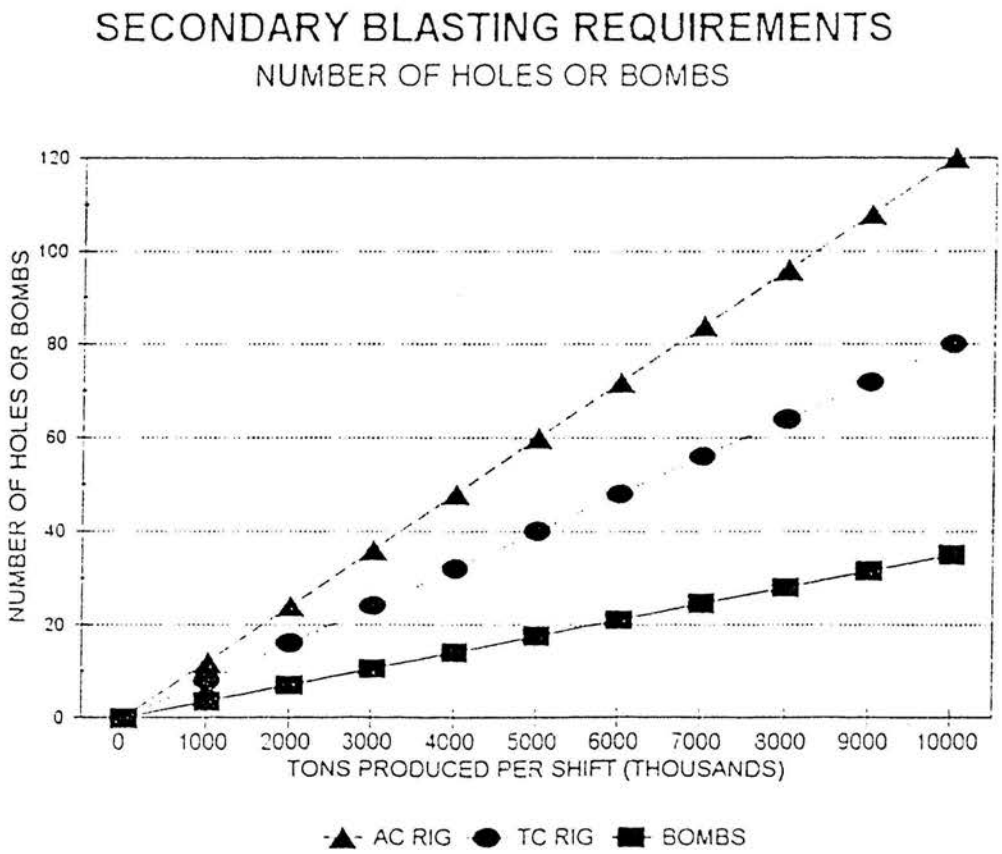



Figure 8.8. Metres to be drilled and bombs placed for secondary blasting in the BA5

## 8.7. CONCLUSIONS

1. A geotechnical investigation using Laubscher's rock mass rating will provide the information needed to simulate the fragmentation size distribution that will report to the drawpoints at various stages of draw, using the expert system described in this chapter.

2. Simulation using the fragmentation size distribution can be used to predict the frequency and type of hangup that will occur. The information can be used to provide the required drilling and blasting equipment and plan secondary blasting procedures to meet the required production tempo.

3. Predicted fragmentation size distribution was finer and secondary blasting requirements were lower than predicted. This is attributed to fines enhancement on the extraction level as a result of the preferential movement of fines through the draw column. It increases productivity, but has the potential to allow early waste ingress where the orebody has a finely fragmented, weathered zone at the top of the orebody.

4. The stress changes that result from the cave mining process result in shear movement on joints and fractures, as well as induced fracturing and are important in determining the fragmentation size distribution that enters the top of the draw column. The stress that occur in the draw column, as well as the rock mass parameters determined in the geotechnical investigation, can be used to predict comminution in the draw column with considerable accuracy.

5. The predicted fragmentation size distribution can be used to plan drawpoint spacings. It should be noted that fragmentation size distribution will become finer as drawing progresses and drawpoints correctly spaced for initial coarse ore may be too far apart for the comminuted ore that eventually reports to the drawpoints. This can lead to ore loss, early waste ingress and compaction pillar problems.

# CHAPTER 9

## DRAW ANALYSIS AND CONTROL

### Statement.

The objective of draw control and analysis in a cave mine is to ensure that the mining block can meet production targets on a continuous basis. The maximum percentage possible of ore contained in that block must be extracted at the planned grade. Rock engineering problems must be minimised to ensure continued production and the safety of men and equipment. Practically it means that drawpoints must be developed at the correct spacing and operated in such a way as to achieve interactive draw.

Material flow in a cave mining situation has been extensively studied and reported in the literature. Sufficient information is available from observation in cave mines, research and theoretical considerations to prove that if ore is abstracted from the base of the cave correctly, all the above objectives can be achieved.

Several practical problems that could not be easily answered from existing research and information had to be addressed. For reasons set out in Chapter 4, drawpoints were spaced at a nominal 15 metres x 15 metres in the BA5 mining block. The author needed to determine whether this spacing was optimum. The increased structural stability that could be gained by spacing drawpoints further apart had to be balanced against a potential loss of ore reserves and rock mechanics problems if drawpoints were too far apart.

Two conclusions from recent research (McNeary & Abel, 1993) are that the drawzones that develop in a cave mine differ from those predicted and observed in sub-level caves and that too closely spaced drawzones could lead to early waste ingress of overlying fines. The extreme ranges in fragmentation size distribution and an inability to recreate realistic stress conditions make it difficult to extrapolate from research results and theory to mines experiencing coarse fragmentation at increasing depths of mining.

Observation at Premier suggested that drawzones related to drawbells rather than drawpoints and that migration of ore across major apices was slower than movement across minor apices. The layout of the extraction level might therefore have an important bearing on material flow in the cave. It was also observed that caved ground migrated towards areas of high draw, with fines migrating more quickly than coarse fragments. This could impact on the calculation of ore reserves and the way in which the cave was drawn to achieve maximum ore extraction with minimum waste dilution.

The 75 metre thick gabbro sill which overlayed the BA5 mining block provided a clear marker horizon. The dense gabbro creates problems in diamond recovery and gabbro waste must be identified, and drawpoints stopped to prevent excessive gabbro waste being fed to the diamond recovery plant. These circumstances, accurate draw control information, and observations by the author underground in drawpoints, provided a unique data set that could be analysed in an attempt to answer some of the questions posed above.

This analysis is set out in Appendix II. The analysis provides an insight into material flow in a cave with coarse fragmentation and shows layout and draw control can have a marked effect on material flow. The information will be used to plan and control the draw in the BA5 and future caves at Premier Mine.

## 9.1. INTRODUCTION

The theory of draw control as it currently exists is that drawzones are created above a drawpoint as ore is drawn. In a situation of closed draw, i.e. where the drawzone does not break through to surface, the drawzone has most of the characteristics of a draw ellipsoid as defined in the literature (Just, 1981). Where the drawzone breaks through to surface, or into a void where the draw column is not in contact with the cave back, a draw cone of considerable extent can be created. Draw control is essential to ensure that the drawzones of adjacent drawpoints interact. If drawzones do not interact, isolated draw conditions can arise, and premature waste ingress and compaction pillar problems can be anticipated. Most of the information regarding the effects of poor draw control has been derived by experience in cave mines and by using "physical" models with the size of fragments used in the "draw column" ranging from sand to building bricks. Rules of thumb have been established. This information forms the basis of most draw control planning (Heslop & Laubscher, 1981).

Accurate information relating to the tons drawn from each drawpoint is essential for draw analysis and only if this is available can draw control be achieved. Production pressures often force personnel to draw as much ore from drawpoints that are "running" easily and/or drawpoints that are close toorepasses. Strict secondary blasting discipline aimed at ensuring that drawpoints are brought down as soon as hangups occur is important to maintain effective draw control and material flow.

A literature survey shows that draw control and analysis varies widely on most cave mines and it is difficult to plan a draw control programme based on theoretical considerations. Premier accepted that, based largely on physical model results as reported in the literature (Heslop & Laubscher, 1981) and as experienced on other mines, the maximum spacing for drawpoints should not exceed 15 metres. Given the risks associated with increasing drawpoint spacing beyond known limits, no planned attempt was made to experiment with this spacing. The migration of fines resulting in fines enhancement at the production level, as noted at Premier, is an aspect of cave mining that has been recognised (Just, 1981) but not explained in any of the theory of draw ellipsoid or material flow as defined in the literature.

## 9.2. DRAW CONTROL THEORY

### 9.2.1. MODEL RESEARCH IN MATERIAL FLOW IN CAVE MINES.

McNeary in a recent PhD thesis provides a comprehensive review of the research work done into material flow in caving methods dating back to 1913. It evaluates physical modelling, including kinematic model and granular flow research, physical scale models, photo elastic models and previous block model research at the Colorado School of Mines. Numerical models using finite element, finite difference and distinct element methods are similarly evaluated (McNeary, 1991).

Most physically scaled modelling research applied to drawing behaviour can be classified as granular flow research, where the majority of the particles are fine. The flow behaviour of the particles dominates the performance of the test and the resultant conclusions. Numerous studies have been carried out on granular flow. It includes work by Wesler (Wesler, 1932) and McNicholas (McNicholas et al., 1945). Physical models, based on granular flow, have provided most of the data on which drawpoint spacings in cave mines are based. Widely used and reported results include those conducted in Zimbabwe (Heslop & Laubscher, 1981).

These models clearly show that, if drawpoints are pulled in a controlled manner, it is possible to draw material down as a horizontal layer. If drawpoint spacing is progressively increased, the horizontal layer is increasingly disrupted. Eventually, there is no interaction between adjacent drawzones and a situation of isolated draw prevails. Where an isolated drawzone breaks through to surface, a funnel shaped draw-cone forms and there is an enhanced flow of fine material into the draw-cone and this moves preferentially through the drawzone to the drawpoint below. The size of the drawpoint influences the diameter of the drawzone above. It is essential that models are fully three dimensional in order to accurately model drawzone interaction and ensure that the sides of the model do not influence material flow within the model. If the effects of the model sidewall are felt, the model is essentially two dimensional and results must be treated with caution.

Much of the research has found practical application in understanding material flow in draw columns in cave mines. The situation in a cave mine is complicated by the extreme range of fragmentation size distribution and relatively high stresses found in the draw column. Scale effects also make it risky to move from small scale physical models to an underground mining situation. No proven numerical models exist that allow material flow in a draw column to be realistically simulated. The Particle Flow Code developed by Itasca for De Beers and other interested parties is currently being calibrated in an attempt to achieve such simulation.

### 9.2.2. DRAW ELLIPSOID THEORY

Draw ellipsoid theory is well developed in the literature (Just, 1981. Laubscher & Heslop, 1981. Brady & Brown, 1985). The theory was developed for sub-level cave mining, and the limited field evidence that exist is based on material flow monitored in silos. No direct field evidence to support draw ellipsoid theory is available from any cave mining operation.

### 9.2.3. MARKER STUDIES

Marker studies have generally confirmed physical model results, but usually show that the migration of ore to a specific drawpoint can come from a greater horizontal distance than anticipated and that the flow pattern in the draw column is more complicated than suggested by theory (Kvapil 1965. Jenike 1966). A study recently conducted at El Teniente in the LHD section, where drawpoint spacings are a nominal 15 metres by 15 metres, produced results typical of most marker exercises. Markers were old 0,8m diameter tyres. Only 19 of the 60 markers placed were recovered. Some conclusions are:

* The horizontal distance of travel ranges between 2 and 42 metres, with an average of 14 metres. This implies a drawzone with an average diameter of 28 metres.
* Angle of draw varies between 60 and 88 degrees with an average of 80.
* Marker flow is towards areas of high draw rate, suggesting that considerable horizontal migration is possible.
* Several variables such as rate of draw, geology, fragmentation size distribution, drawpoint spacing and moisture all affect movement in the drawzone.

Marker studies from the De Beers mines in Kimberley and asbestos mines in Zimbabwe have produced similar results, suggesting that material flow in the draw column is complicated and influenced by several variables.

Study of research reported in the literature and experience on other mines proved useful in determining initial drawpoint spacing and draw control planning for the BA5, but could not provide answers as to whether the drawpoint spacing was optimum, or whether draw control planning would allow maximum ore extraction from the block, limit waste ingress and avoid static columns in the draw column. Orebodies are unique geological features and the answers required could only be gained from experience on the mine, analysis of existing draw control data and informed underground observation, always bearing in mind the guidelines provided by research and experience on other mines.

## 9.3. EXPERIENCE AT PREMIER MINE

### 9.3.1. PLANNED DRAWPOINT SPACING

Premier has operated four block caves using a grizzly and scraper layout above the sill. The caves were undercut using continuous troughs with the distance between the centre line of adjacent troughs 27,4 metres. Drawpoint spacing parallel to the direction of the continuous troughs was 9 metres. Drawpoints were holed into the troughs from tunnels that had been developed in the major apices between troughs. Initial drawpoint spacing for the drawpoints was 3,4 metres, the width of the original tunnel that was used to develop the continuous trough. The distance measured between drawpoints across the major apex was 24 metres. Pit sidewalls were near vertical and remarkably stable and it was possible to observe the lowering of the draw column until the caves were drawn to completion and the continuous troughs on the extraction level and the drawpoints that had been developed into the troughs were exposed (see P.9.1). Survey measurements from aerial photographs once the cave had been drawn to completion showed that the distance between drawpoints across the major apex ranged between 18 and 22 metres, whilst actual drawpoint across the continuous trough ranged from 6 to 10 metres. Effective drawpoint spacing during the life of these caves, measured across the continuous trough, ranged from an initial 3,4 metres to as much as 10 metres. Drawpoint spacing across the major apex ranged from an initial 24 metres to a lower limit of 17 metres. As can be seen from photograph P.9.1. this cave was successfully drawn to completion with an average of 53 000 tons being drawn through each drawpoint. It resulted in between 2,6 and 6,6 metres of brow wear in competent gabbro and/or metamorphosed kimberlite. The column heights of the caves varied between 100 metres and 155 metres. The caves extended from the base of the open pit to the top of the dipping gabbro sill.

P.9.1. Continuous troughs and drawpoints exposed in glory hole

P.9.2. Exposed apices between troughs are 27,4 metres apart. Thirty percent of ore by mass is contained in fragments larger than 10 metres along 1 side

Experience showed that fragmentation was extremely coarse by world standards (see Table 8.1) with at least 30 percent of fragments that reported to the drawpoints larger than 3 metres in maximum size, 20 percent of fragments were between 3 and 1 metre in size and 50 percent of fragments were smaller than 1 metre in maximum dimension. Secondary blasting costs were high throughout the life of the cave with an average explosive usage of 250 gm per ton of ore produced. Initial explosive usage was as high as 600 grams per ton. Cave loading was never high and the final stages of draw were marked by the frequent occurrence of large boulders, often in excess of 10 metres in diameter. Photograph P.9.2. shows the typical fragmentation size distribution experienced in the caves. The exposed major apices between the continuous troughs are spaced at 27.4 metres and provide a scale. A cursory examination of the photograph shows that at least 30 percent by mass of the broken ore visible in the photograph is contained in fragments larger than 10 metres along one side.

The observations indicated that drawpoints could be widely spaced in the Premier kimberlites, given their coarse fragmentation. It was realised, however, that material flow in the caves was in a free flow situation with the top of the draw column exposed to surface and no waste overburden. Material flow beneath the gabbro sill capping would be more complex.

A geotechnical survey based on core drilling showed that weak decomposing kimberlite would be a problem in some areas of the BA5, especially if water reached the extraction level once the sill was breached. The structural stability of the extraction level, especially the potential for rapid brow wear, was therefore a source of concern. This concern, together with coarse fragmentation experienced above the sill, suggested that drawpoints should be as widely spaced as possible. A literature survey showed that physical models (Heslop & Laubscher, 1981) indicated that the maximum practical spacing for caves experiencing coarse fragmentation was 15 metres by 15 metres. At that stage Henderson had drawpoint spacings of 12,2 metres by 12,2 metres and El Teniente had some drawpoints spaced at 15 metres by 15 metres in the primary ore. Andina, with fine, oxidised, primary ore, was experiencing problems with drawpoints spaced at 17 metres by 14 metres. A recent, preliminary review of the maximum drawzone spacing actually implemented on cave mines experiencing coarse fragmentation by Laubscher shows that several cave mines have successfully implemented caves with drawzones as widely spaced as 21 metres (D.H. Laubscher - personal communication).

## 9.3.2. ACTUAL DRAWBELL SPACING

Drawpoints were planned at 15 metre by 15 metre centres in the BA5 with the length of an individual drawbell 15 metres. As rapid brow wear was expected, it was decided that actual drawbell development would be 13 metres. The last metre of each end of the drawbell was drilled but not blasted as it was reasoned that rapid brow wear would soon widen the drawbell to its planned dimension of 15 metres. In all cases drawpoints were supported prior to drawbells being developed. In some drawbells support inadvertently extended into the planned drawbell and mining crew were reluctant to extend the drawbell to its planned distance of 13 metres, destroying newly-installed, expensive support.

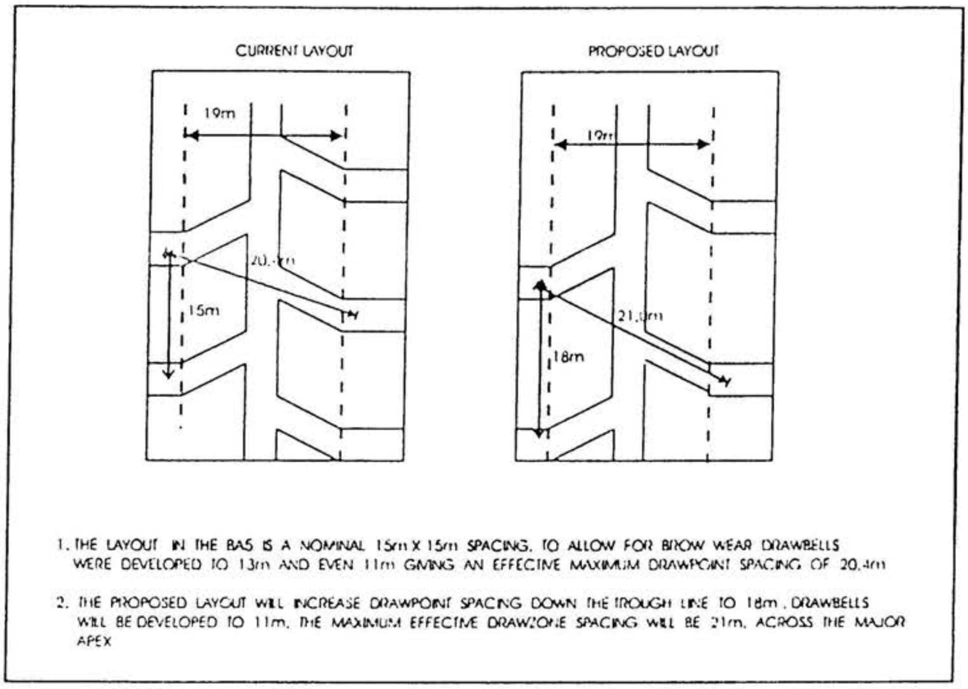



**Figure 9.1. Current, proposed and ideal layouts for cave mining at Premier.**

Actual drawpoint spacing as calculated from survey offsets are set out in Table 9.2. All drawzones were spaced at 15 metres across the minor apex. Figure 9.1. shows the geometry of the drawpoint spacing in the BA5 mining block and the increased spacing being implemented in the BB1E mining block. Brow wear can change drawpoint spacing markedly during the life of a drawpoint. It should be noted that the term brow wear as used here implies brow retreat rather than actual wear by attrition. Where the brow is damaged by high stresses or weakened by geological structures, block fallout can result in brow destruction. Where the brow is sited in competent rock and secondary breaking is achieved by drilling and blasting, minimal brow wear has been recorded after tens of thousands of tons of ore have been drawn. Premier has found no statistical correlation between tons extracted from a drawpoint and "brow wear".

A major problem that arises with increased drawpoint spacing is that drawzones can fail to interact and a significant proportion of the planned ore reserve is sterilised within the cave. Some measure of the effectiveness of the implemented drawpoint spacing must therefore be adopted before the drawpoint spacing can be accepted as being effective. If the calculated tonnage is drawn from several drawpoints in the area, the drawpoint spacing is usually accepted as being correct. The manner in which the ore reserve is calculated must be considered. Typically the ore reserve tons for a drawpoint in a cave is calculated for a single rectangular or square column extending from the production level to the top of the defined ore block. An average grade is then assigned to this column. As caving occurs and drawing progresses, both vertical and horizontal mixing occur and fines from adjacent columns or from overlying waste migrate to or away from the theoretically defined, now broken, column. The definition of both ore reserve and grade for such a column is complex. The calculation of the tons and grade that can be expected from a block cave must be considered in terms of the rock that both surrounds and overlies the mining block. The definition of the ore reserve in a single drawpoint must be treated with caution as it is well established that considerable migration can occur towards areas of high draw.

As indicated in Table 9.1, several adjacent drawpoints in the BA5 mining block with a maximum spacing of 22,5 metres, diagonally across the major apex, have pulled more than 70 percent of the expected tonnage. Gabbro waste has been recorded in only three of the drawpoints. This is a good indication that the widely spaced drawzones have not resulted in early waste ingress or isolated drawzones. Fourteen adjacent drawpoints in the L2 were pulled to at least 120 percent of calculated ore reserve before high levels of gabbro waste were monitored showing that no early ingress of waste resulted from the widely spaced drawzones. Column height in the block is only 90 metres. Premier has therefore operated numerous drawzones spaced at a maximum of 22,5 metres across the major apex with little or no ore loss of ore reserves in small caves beneath failed open stopes on the eastern side of the mine. One hundred percent of the planned ore was recovered. Final assessment of the effect of the widely spaced drawpoints in the BA5 and the effect on total extraction will only be possible once numerous adjacent drawzones have been pulled to completion.

Discussion often centres around whether drawzones relate to individual drawpoints or to drawbells, and whether the size of the drawpoint opening plays an important part in determining the diameter of the drawzone above it. Observation at Premier shows that, at times, a large fragment will lie over the top of one or more drawbells. On other occasions, an entire drawbell will hangup several metres above the extraction horizon and LHD's remove all the ore below the

**Table 9.1. Recovery from drawpoints with a maximum spacing of 22,5 metres.**

| MINING BLOCK | No. OF DRAWBELLS AND PERCENTAGE DRAWN | % WASTE DRAWN |
|---|---|---|
| BA5 CAVE | 4>90% DRAWN | 10% |
| | 4>80% DRAWN | 10% |
| | 15>70% DRAWN | 10% |
| L2 CAVE | 13>100% DRAWN | 10% |
| | 4>90% DRAWN | 10% |
| | 4>80% DRAWN | 10% |

Notes:
1. Drawpoints that produce more than 10 percent waste are abandoned
2. In the BA5 several adjacent drawpoints have drawn more than 70 percent of their expected tonnages.
3. In the L2 cave 14 adjacent drawpoints have pulled more than 100 percent of their expected tonnage.

hangup. The size of the excavation, in plan view, within the drawbell that is then empty of ore can be as large as 13 metres long by 8 metres wide. This leaves little doubt that, in a cave experiencing fragmentation as coarse as at Premier, drawzones relate to drawbells and not drawpoints and that the width of the drawpoint excavation plays a marginal role in determining the width of the drawzone, as long as large fragments that can cause this type of hangup exist in the draw column. Observation during mining above the sill clearly showed that material flow related to trough spacing and not drawpoint spacing. In the L2 mining block incomplete undercutting in highly stressed pillars resulted in one tunnel being crushed. It prevented access to drawbells from both sides as originally planned. Between 94 percent and 218 percent of the calculated drawbell ore reserve was nevertheless recovered from these drawbells which could only be accessed from one side.

**Table 9.2. Drawpoint spacing, tonnages drawn and secondary blasting statistics from Premier caves**

| MINING BLOCK | MAXIMUM DRAWPOINT SPACING | SECONDARY BLASTING EXPLOSIVE USAGE (gm/t) | TONS DRAWN | BROW WEAR |
|---|---|---|---|---|
| ABOVE SILL | 612 @ 27,4m | 600 --> 250 (11 to 17 tons) | 44 000 TO 68 000 | 2,6 - 6,6m |
| BA5 | 2 @ 18m | 100 - 50 (2,4 to 4,8 tons) | 48 000 TO 84 000 | 0 - 7m |
| | 5 @ 20m | | | |
| | 64 @ 22,5m | | | |
| L2 | 67 @ 22,5m | 50 (2 tons) | 41 000 | 0 - 9m |
| SA1 | 26 @ 22,5m | 50 (3,4 tons) | 53 000 | 0 - 5m |
| BB1E (Planned) | 134 @ 21m | 50 (5,4 tons) | 107 000 | ? |

Notes:

1. Brows above the sill were sited in competent gabbro
2. Tons quoted for secondary blasting refers to explosive usage during the life of the drawpoint
3. Drawpoint spacing refers to the maximum spacing diagonally across the major apex. The first number refers to the number of drawpoints installed at this spacing e.g. 612 @ 27,4m
4. Brow wear refers to total brow retreat

Based on experience at Premier and on Laubscher's review of drawzone spacings in cave mines worldwide (Laubscher D.H. personal communication), Premier has made the decision to develop future drawpoints at a spacing of 15 metres by 18 metres. It will result in effective trough spacings of 19 metres across the major apex and 21 metres, diagonally, across the major apex between staggered drawpoints. If drawzones move further apart, the height above the production level at which drawzones interact becomes greater until, eventually, drawzones no longer interact. Theoretically, considerable ore reserve could be sterilised and lost in this way. Observation shows that the column of broken ore that exists above the major apex usually has an angle of about 45 degrees, implying that a situation of free flow prevails, at least at various times, in the cave. Columns of static ore 50 or more metres high above apices have never been observed at Premier Mine.

### 9.3.3. PLANNED RATE OF DRAW

The rate of caving in the two types of kimberlite was unknown and the production tempo from each drawpoint was tentatively planned at a rate of 300 millimetres of caving per day. Over the area of a 15 metres by 15 metres drawpoint, this provided 182 tons per drawpoint per day. It was further assumed that drawpoints would only produce for 66 percent of the available time as a result of hangups, drawpoint repairs and other production constraints. It was therefore calculated that, when it was available, a drawpoint would have a production tempo of 180 tons per day but that, over its planned life, a drawpoint would have an average production rate of 120 tons per day. Due to the dip of the sill, column height varied from a maximum 140 metres to a minimum of 100 metres. Drawpoint tonnage varied from 85 000 to 62 000 tons per drawpoint. It was further calculated that gabbro waste would start to ingress once a drawpoint had been 70 percent depleted and that, by the time a drawpoint had drawn 85 percent of the available ore, gabbro waste would render a drawpoint uneconomic. At an average production rate of 120 tons per drawpoint per day planned, drawpoint life varied between 2 and 4 years.

Overall draw control strategy was initially aimed at pulling drawpoint lines adjacent to the cave front at a rate of 30 tons per drawpoint per day to prevent an airgap forming between the broken ore and the advancing caving face. The next row of drawpoints was planned at a production rate of 60 tons per drawpoint per day. Away from the caving face, drawpoints were pulled at a conservative rate of 100 tons per drawpoint per day.

Problems were soon encountered.

* Drawbells adjacent to the undercut face had to be pulled hard to clear the face and reduce the effects of choke blasting. Only once caving had occurred and the undercut face had advanced away from the drawbell could the rate of draw be reduced. Drawbells were overdrawn and this allowed a considerable airgap to form between the cave back and the top of the draw column.

* Caving in the Tuffisitic Kimberlite Breccia at the centre of the cave was rapid until the base of the sill was reached. In order to reduce the effect of an airblast when the sill caved, the centre of the cave was not pulled and production tonnage was derived largely from the advancing undercut. The "shoulders" of the cave soon rested on broken ore and prevented further caving. Draw control had to pull ore from beneath the "shoulders" to allow further caving of the kimberlite, which in turn increased the area of the base of the sill that was exposed and allowed caving to occur. As caving of the sill occurred the "shoulders" of the cave again rested on broken ore which had to be drawn before further caving could initiate. Caving was inhibited by the high horizontal stresses that accumulated in the arch as the sill reduced in thickness and clamping forces increased across joints. The area and shape mined on the undercut level was no longer the determinant of the hydraulic radius, but rather the area defined by the "shoulders" of the cave resting on broken ore. This complicated draw control and analysis (see Figure 9.2).

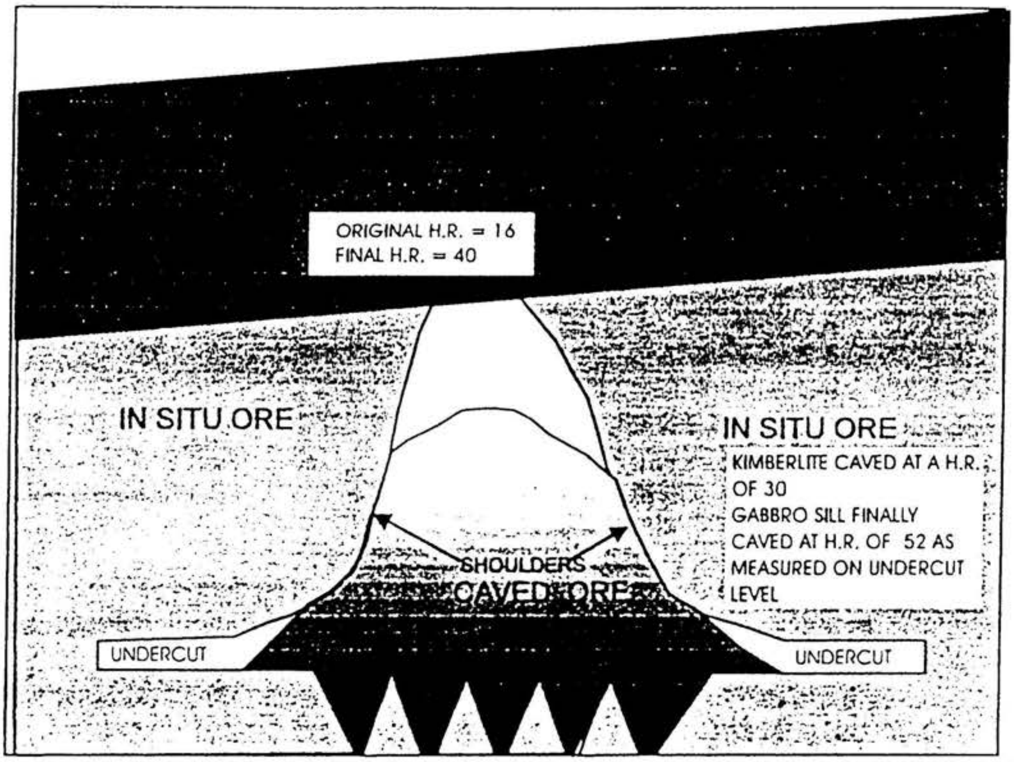



**Figure 9.2. Hydraulic radius of sill defined by "shoulders".**

### 9.3.4. FRAGMENTATION REPORTING TO DRAWPOINTS.

The predicted fragmentation size distribution in the kimberlites in the BA5 is set out in Figures 8.2. and 8.3. Monitoring showed that, once blasted undercut ore had been drawn and continuous caving initiated, the fragmentation size distribution that reported to drawpoints situated in both Hypabyssal and Tuffisitic Kimberlite Breccia was coarse with more than 50 percent of ore fragments having a diameter of more than 1 metre (Table 8.2). Predicted fragmentation size distribution correlated well with that predicted for primary fragmentation in both the Tuffisitic Kimberlite Breccia and Hypabyssal Kimberlite.

As drawing progressed, comminution of the incompetent Tuffisitic Kimberlite Breccia was rapid and fines concentration was noted in drawpoints sited in Tuffisitic Kimberlite Breccia. Large fragments with an abundance of fines were typical. In the well jointed, competent Hypabyssal Kimberlite disintegration of fragments along joints occurred resulting in a finer fragmentation size distribution reporting to the drawpoints as drawing of ore progressed. Fines concentration was noted but not to the extent monitored in the Tuffisitic Kimberlite Breccia.

After caving had been in progress for three years it was decided to carry out a rim loading exercise on the western side of the oreblock and to change the method of undercutting from post undercutting to advance undercutting. Both decisions required that tunnels on the undercut level be developed into caved ore in the cave excavation. A total of 17 tunnels on 15 metre centres were subsequently holed into the active cave 15 metres above the extraction level close to operating drawpoints. This allowed observation of the material flow in the cave 15 metres above the drawpoints. The amount of ore drawn from drawpoints that could be observed, varied from 70 000 to less than 5 000 tons. Tunnels were sited in both Tuffisitic Kimberlite Breccia and Hypabyssal Kimberlite. Three stages of comminution could be distinguished from these observations.

Above drawpoints where only a few thousand tons of ore had been drawn fragmentation was generally coarse and the aspect ratios of the fragments were high. As much as 50 percent of the draw column consisted of large voids, often a metre or more in diameter, that anastomosed to create channelways. These allowed fines to flow to the drawpoints and resulted in substantial fines enhancement on the extraction level at an early stage of draw. It was often possible to observe a flow of fines from such a channelway for as long as ten minutes at a time. Radiating fans in drawpoints at the base of such channelways were common. The flow of fines was stopped if a drawpoint was not pulled for a while and compaction occurred or if larger fragments moved and blocked the channelways. Secondary blasting was then required to extract ore from the drawpoint. As much as 50 percent of the ore extracted from such a drawpoint was gained by drilling and blasting. At this early stage of draw, fragments rested directly on one another. Simulation of the contact forces experienced near the base of a 100 metres high 30 metre wide drawzone showed that these were of the order of 1,8 MPa, sufficient to cause rounding at corners and along the edges of fragments, producing a considerable quantity of fines. Arch formation and considerable breakage of fragments as a result of point loading was widespread. Arches that formed above drawpoints were broken by secondary blasting or by accumulated stresses. Often the collapse of an arch would block channelways and further blasting was required to allow the fines to flow. At this stage, only 20 tons was recovered on average subsequent to each secondary blast. There were few other differences noted between the ore characteristics in the draw column above Hypabyssal Kimberlite or Tuffisitic Kimberlite Breccia.

Above drawpoints in the Tuffisitic Kimberlite Breccia where 30 000 to 40 000 tons of ore had been drawn, an intermediate stage of comminution was noted. Channelways no longer existed and as much as 30 percent of the draw column consisted of fragments greater than one metre in diameter. Aspect ratios were lower and most fragments were reasonably well rounded. Relatively large areas of fines mixed with some coarser material existed in pockets interspersed throughout the draw column. Presumably, these had previously been channelways through which fines flowed, before they were isolated by movement of larger fragments in the draw column. Secondary blasting of large fragments and arches sometimes resulted in these pockets being ruptured and a considerable flow of fines to the drawpoints. The effect in Hypabyssal Kimberlite was similar, but not as marked as in Tuffisitic Kimberlite Breccia.

Near drawpoints in the Tuffisitic Kimberlite Breccia where considerable tonnage had been drawn, a third phase of comminution was noted. Large fragments still existed in the draw column but these were cushioned and "floated" in a bed of fines. Attrition and comminution of large fragments at this stage was negligible. Secondary blasting was necessary when one or more large fragments blocked the throat of the drawpoint. Cushioning in the Hypabyssal Kimberlite was considerably less marked.

Draw control should aim at ensuring that large fragments are broken as they arrive at the drawpoints by secondary blasting. Drawpoints that "run" should be pulled at the planned rate only. Taken together this means that all drawpoints should be pulled at the planned production rate irrespective of the blockages in the drawpoints. To achieve this secondary blasting must be efficient. An inevitable consequence of the fines enhancement at the base of the cave is that fines will eventually become depleted and fragmentation size distribution will become coarser as the cave nears exhaustion. The coarsening has been noted in several kimberlite caves and was clearly evident and recorded in photographs in caves above the sill at Premier. Coarsening of the fragmentation size distribution as the cave is depleted is predicted in Esterhuizen's fragmentation simulation programme.

If the cave is such that the cave back can eventually break into an overlying, weathered, waste capping with fine fragmentation, the fines will flow preferentially to the base of the draw column. If voids exist, the angle at which such fines will flow to a drawpoint that is running freely can be as low as 30 degrees.

Observations from the undercut level gave no indication of interacting drawzones where the ore was coarse. Fines migrated through voids to the drawpoints in a free flow situation. Little lateral movement of large fragments was seen. As the ore became comminuted, zones of accelerated movement were noted immediately above drawbells. Large fragments sometimes extended across two or more drawpoints causing hangups. When the hangups were brought down, ore flow was restarted in all the affected drawpoints showing that drawzones, at least in the affected areas, extended across drawbells and even across minor apices. In drawbells with drawpoint access from both sides, if only one drawpoint was pulled for a period of time to achieve draw control, the drawpoint that was not pulled often had coarse fragmentation when drawing restarted here, showing that fines started migrating towards the operational drawpoint.

As defined in the literature (Brady & Brown, 1985) the concept of gravity flow in a sub-level cave is best explained in terms of a flow ellipsoid. After draw has been in progress for a period of time, all discharged material from a drawpoint will have originated from an ellipsoidal zone termed the ellipsoid of motion. Surrounding the ellipsoid of motion, a larger ellipsoid, termed the limit ellipsoid exists and within this limit ellipsoid, material will be loosened and displaced but will not yet have reached the discharge point. As drawing proceeds a defined draw cone will be created above each active drawpoint.

These concepts are useful but need to be extended to incorporate observations made at Premier and on other cave mines. Any cave with coarse fragmentation will experience three zones of comminution at least until such time as the draw column breaks through to surface. Boundaries between stages will be gradational and can vary according to ore type.

* At the top of the draw column coarse, primary fragmentation is laid down. The zone is characterised by high aspect ratios, low stresses, large void ratios, limited fines creation and fast fines flow. Fines will be created by contact between fragments and could be as low as 2 percent of material in this zone. Void ratios will typically be between 15 and 30 percent and many voids will anastomose to form channelways that allow a fast fines flow. The definition of "fines" will be a function of the primary fragmentation size which

will, in turn, determine the size of the voids that exist and allow fines to flow preferentially.

* With increasing height of the draw column the column is progressively loaded and comminution of fragments with high aspect ratios occurs within the column, creating larger surface areas. Closer spacing of fragments is achieved with increased contact areas between fragments. In soft ores, fines production is at a maximum in this zone. Void ratios are high enough to result in anastomosing channelways and fines flow.

* The fines that are created move rapidly through the draw column and result in fines enhancement at the base of the draw column at an early stage of draw in the lowermost zone. Voids and channelways become progressively blocked with the fines starting from the base of the cave (the production level). Large fragments at the base of the cave are effectively cushioned and "float" in a matrix of fines. The effectiveness of the process will depend on the height of the draw column and the hardness of the ore. In the BA5, fines enhancement and cushioning of large fragments was more noticeable in the Tuffisitic Kimberlite Breccia than in the Hypabyssal Kimberlite. The geometry of the extraction level plays a role in creating high arching stresses as fragments move into drawbells and hangups are created. If drawpoints that "run" are overdrawn, rapid fines migration can occur from both overlying and adjacent areas.

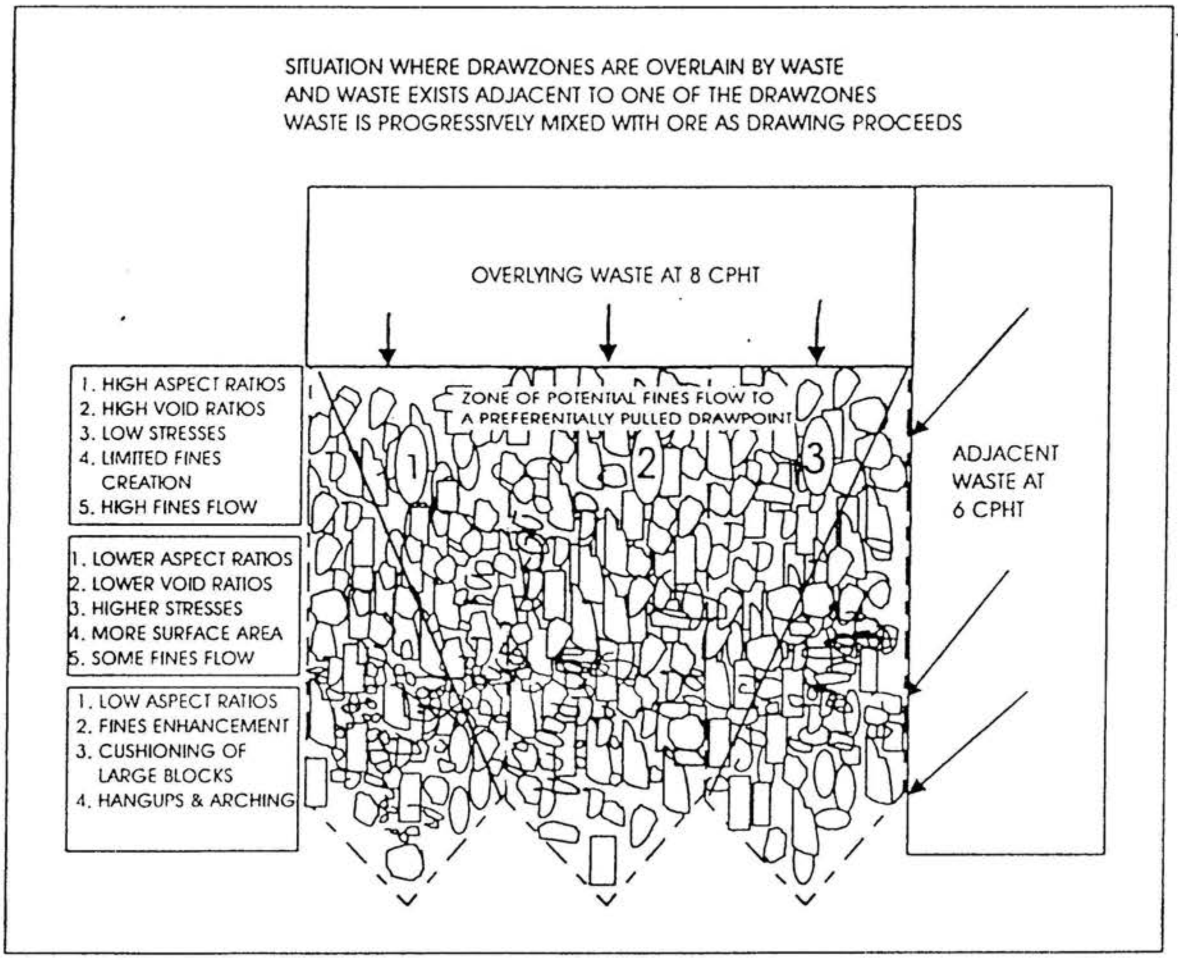



**Figure 9.3. Zones observed in the draw column in an active cave**

Figure 9.3 depicts several conclusions that result from these observations.

* Large fragments will move towards drawpoints under the influence of gravity and could migrate to adjacent drawpoints especially if column heights are great. The probability of such fragments moving more than the maximum drawpoint spacing, horizontally, from their original position in three dimensional space is low.

* Fines are created in the draw column and will move towards areas of high draw. "Fines" are a function of the size of material that can move through anastomosing channelways. At Premier, material smaller than 100 millimetres in maximum dimension was seen to flow freely at angles as low as 30 degrees.

* Where the drawzone breaks through to surface, a draw cone with large fragments moving at 45 degrees (angle of repose of broken ore) and fines moving at 30 degrees (angle of repose of sand) can be created. This "surface" can be a temporary airgap, an overlying open pit or the earth's surface.

* If the drawzone breaks through into an overlying zone of finely comminuted material (secondary ore in porphyry copper deposits, overlying waste capping in kimberlites), the fine material can move rapidly through channelways to the production level.

* Effective draw control and mining sequences must be employed to ensure that air gaps are not created and that only the planned ore is mined from each drawpoint to avoid the rapid ingress of waste. This requires effective secondary blasting procedures.

### 9.3.5 DRAW CONTROL INFORMATION

Accurate data is an important part of draw control and analysis. Observation in several cave mines had shown that accurate data on tons taken from drawpoints was important but difficult to achieve. Production pressures usually meant that drawpoints that flowed freely and/or were close to passes were pulled heavily. When hangups occurred, but sufficient drawpoints were available to meet targets, secondary blasting of the hangups was delayed. If a programme of planned draw control is not implemented and maintained, isolated draw can result above drawpoints that are preferentially pulled. Conversely, hangups can lead to drawpoints not being loaded and point loading can result.

At Premier battery-powered microwave beacons were installed in all operating drawpoints, at the entrance to production tunnels and at overpasses. These beacons gave out a four digit binary code at the rate of seven signals per second. Three of the binary digits identified the beacon and the fourth digit indicated the battery status. Battery packs lasted between 12 and 18 months. Data was received and stored by microwave receiver/radio transmitter units mounted on all production LHD's as these passed beneath the beacons. The data was transmitted via the mine's leaky feeder communication system to a base station sited underground and from there to a personal computer (PC) in a control room on surface. The unit on the LHD is capable of storing up to 5 000 records. The system was designed in such a way that the required draw control information could be downloaded from the LHD units, from the base station situated underground, or collected in the PC on surface. The system allowed good draw control and provided additional information relating to LHD travelling times and data required for operational control of passes.

### 9.3.6. WASTE MIGRATION AND CONTROL.

Figure 9.3 illustrates the situation where waste in a cave mining operation both overlies and exists adjacent to operating drawpoints. If drawzone 2 is preferentially drawn, fines will flow to the drawpoint at an angle as low as 30 degrees through voids and channelways. The tonnage drawn from the drawzone will exceed the calculated reserve by a large margin. Drawzone 3 has both overlying and adjacent waste. It is assumed that 100 000 tons of clean ore exists in a square 15 metre x 15 metre column above drawpoint 3. As drawing progresses, waste is increasingly drawn into the operating drawpoint, initially from the adjacent waste, and then from both the adjacent waste column and the overlying waste capping. Figure 9.4 shows a typical grade profile for drawzone 3 in this situation. It shows that by the time 60 percent of the calculated ore reserve has been drawn, waste from the adjacent column starts to ingress at an increasing rate and, by the time 70 percent of the calculated ore reserve has been drawn, waste from the overlying capping starts to report at the drawpoints. By the time that 100 percent of the calculated ore reserve has been drawn, only 79 percent of the expected ore has been drawn and 21 percent waste; 13 000 tons of this is from adjacent waste and 8 000 tons of overlying waste. Only 81 percent of the calculated carats have been recovered. The rate of waste ingress will depend on draw control, draw column height and on the fragmentation size distribution of the waste relative to the ore. If the waste is fine, as with an oxidised waste capping or comminuted shale and kimberlite as in the Kimberley mines, waste ingress can start when only 35 percent of the in situ ore reserve has been drawn. In most cave mining operations economic considerations determine that the cut off grade equals the marginal cost of mining and treating a ton of ore. If both the hoisting and treatment capacity of the mine is being utilised, the opportunity cost of mining and treating low grade ore must be considered. At both Koffiefontein and Premier, cut off grade is determined by the criteria that headfeed to the plant cannot exceed 20 percent dolerite and 10 percent gabbro respectively, as the dense waste displaces diamonds.

Waste ingress is a source of concern on all seven De Beer's underground kimberlite mines where caving has, or will be implemented. At Koffiefontein, front caving is planned and, once the initial slot material has been drawn dolerite waste from overlying pit sidewall failure will both overlie and exist adjacent to ore that is being drawn. In the Kimberley mines, all caves are overlain by a 300 metre capping made up of fine shale waste and comminuted Tuffisitic Kimberlite Breccia. A premature ingress of this fine waste has been noted on several of the Kimberley caves.

Waste ingress was a source of concern in the planning of the BA5. Table 8.3. shows that primary and secondary fragmentation of the gabbro is considerably finer than that of either of the two types of kimberlite. The gabbro sill is a layered intrusion with a layer of cumulate olivine at the base of the sill. Fragments from the cumulate layer can have a density as high as 3.2, although the average density of the sill is 2.9. It was expected that fine, dense fragments would filter down through the draw column at a relatively early stage of draw and that waste would start to report once 70 percent of the ore had been drawn. By the time 85 percent of the ore in the draw column above a drawpoint had been pulled, it was anticipated that the drawpoint would cease to be economic. The density of the gabbro causes it to report to the sink in the HMS and DMS sections of the recovery plant in large volumes. The plant has proved incapable of effectively treating headfeed ore that contains more than 10 percent gabbro or norite and this must also be taken into consideration when planning production.

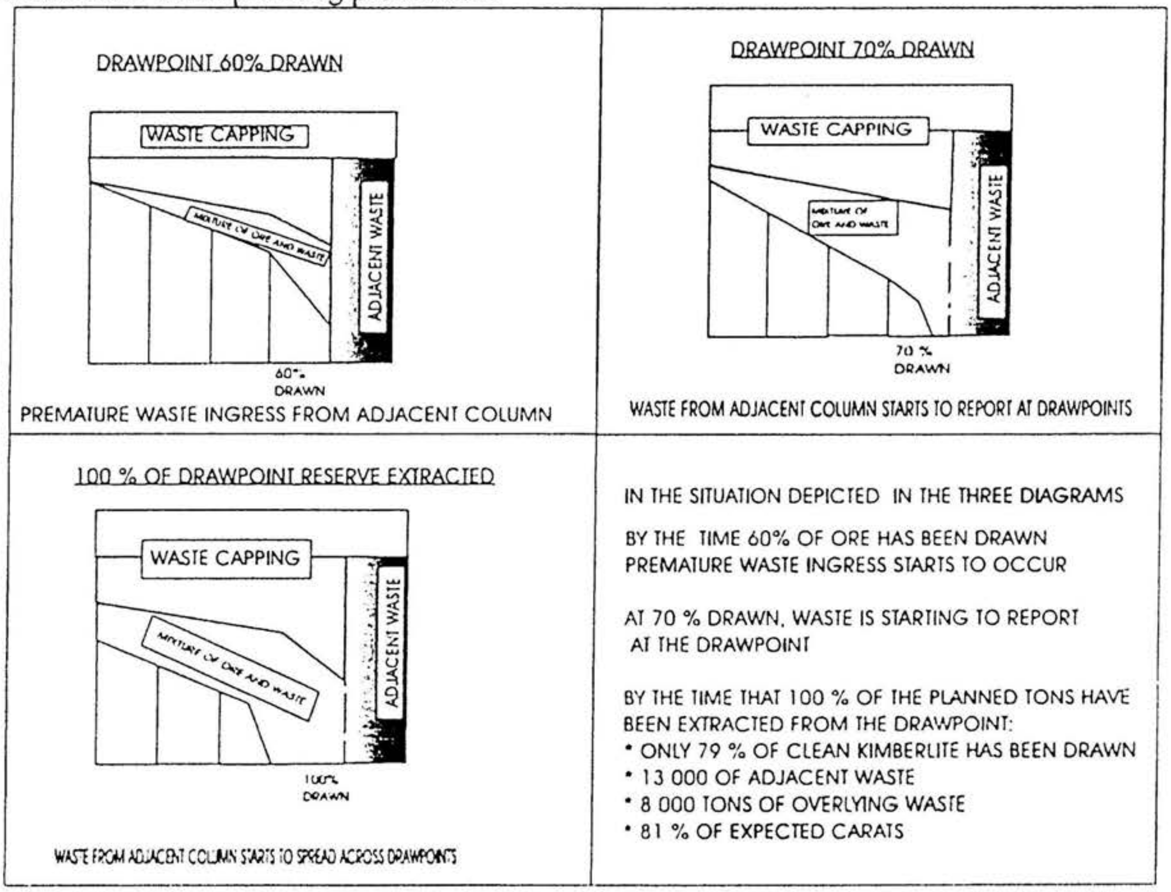



**Figure 9.4. Draw pattern showing premature waste ingress**

The coarse kimberlite ore in the BA5 necessitated regular blasting and gabbro was first noted in the BA5 in drawpoints that were almost 80 percent drawn. Observation was that fragments generally had to be smaller than 100 millimetres diameter before they moved through voids and flowed freely. Although the kimberlite had considerably more fragments in the size range smaller than 100 millimetres than the gabbro, the percentage of material less than 100 millimetres in size in the kimberlite was greater than in the gabbro and there was generally no enhancement of gabbro in the fines. Overall the gabbro did not move more quickly through the draw column than the kimberlite. This indicates that it is the void ratio and size of channelways together with the amount of fines that will move through the channelways that exist in the waste that will determine the rate of fines flow, rather than the average fragmentation size distribution. The skewness of fragmentation size distribution, measured by the amount of material less than 100 millimetres in size relative to the amount of material larger than 2 cubic metres, is a good indicator of the effectiveness of the process illustrated in Figure 9.3.

Norite, which forms the wallrock around the pipe has a density of 2,9 and creates problems similar to the gabbro. Fortunately norite fragmentation is coarser than the kimberlite and much of the norite can be excluded from the ore by leaving a barrier pillar of ore around the edge of the orebody.

The gabbro sill did not cave immediately and allowed a considerable airgap to form. Gabbro waste was not fed into the top of the draw column until a few drawpoints had been 70 percent drawn. This meant that gabbro fines took a considerably longer period to filter down to the drawpoints than would have been the case if the gabbro sill had caved immediately.

### 9.3.7. DRAW ANALYSIS AND CONTROL.

Overall draw control strategy was to pull drawpoints in such a way that the western side of the oreblock was pulled down more rapidly so that when caving of the sill initiated, gabbro would move down the slope that had been created towards the western side of the mine. It was intended that the ore adjacent to the undercut would be pulled at a slow rate to avoid an airgap forming into which collapsing gabbro could migrate.

For purposes of calculation it was assumed that the ore reserve above a given drawpoint was the ore contained in an orthogonal block measuring 15 metres by 15 metres situated immediately above each drawpoint and extending to the base of the sill. Draw control analysis started with collecting accurate information on tons drawn per time period from all operating drawpoints. The tons to be taken from each drawpoint were planned on a monthly basis. At the end of each week draw analysis was carried out and the draw control plan reviewed. Drawpoints that had been overdrawn were pulled at a reduced rate or even stopped. Drawpoints that had not been sufficiently mined were accelerated. If more ore was drawn from one drawpoint than from an adjacent drawpoint it was assumed that ore would migrate to the drawpoint from which most ore had been drawn. An algorithm that considered vertical and horizontal mixing was run each month to reassign tonnages to the various drawpoints. If, after redistribution of the tonnages, drawpoints were overdrawn, the rate of draw in these drawpoints was slowed. A constraint was that all drawpoints had to be mined at a minimum rate of a least a third of the tonnage of the most heavily pulled adjacent drawpoint.

It was assumed that only 85 percent of the ore tonnage above each drawpoint would be pulled before waste ingress made the drawpoint uneconomic to mine. Generally, this draw control strategy worked well and most drawpoints yielded more than their 85 percent of ore before being abandoned.

Figures 9.5., 9.6. and 9.7. show typical draw control statistics. Figure 9.5 shows the tons mined from a drawpoint for a given month. The average production rate during March 1995 was 49,7 tons per day for a draw down rate of 82 millimetres per day. Figure 9.6. shows the planned rate of depletion against the actual rate of depletion. Figure 9.7. shows the percentage depletion for each drawpoint. Apart from draw control these figures are essential for mine planning so that new drawpoints are made available as old drawpoints are depleted. The overall objective is to ensure that sufficient drawpoints are available to maintain the required production tempo from the mining block at the planned, average rate of draw of 100 tons per day from each drawpoint.

## 9.4. CONCLUSIONS

* Observation of material flow in the draw column showed that initial stages of draw were marked by fines filtering through anastomosing voids with little movement of larger fragments. It resulted in fines concentration at the base of the draw column but larger fragments had to be broken by secondary blasting. Fines moved in a free flow situation and at low angles. Only once voids were filled did movement of ore in defined drawzones become apparent.

* Ore in the draw column must be pulled continuously to avoid pillars forming. Compaction pillars in the BA5 did not create stress problems on the extraction level due largely to a limited lift height prior to caving of the sill. Caving was, however, inhibited. The effective hydraulic radius was defined by the "shoulders" of the cave area and not the actual area of undercut. The sill did not cave until the draw column beneath the "shoulders" was pulled and caving reinitiated. The fact that the sill did not cave for a considerable period of time resulted in a large airgap forming and posed the danger of an airblast. Draw control strategy and production were affected.

* Good information relating to tons drawn from each drawpoint, a knowledge of the fragmentation size distribution in the draw column, the rate of caving in the various rock types and the application of a few relatively simple principles allowed considerable control to be exercised over the cave mining process. This ensured that planned production targets were met, that the planned ore reserve was mined to completion, that early waste ingress was limited and that no drawpoints were lost as a result of cave "sitdowns".

* Review of the literature on research into material flow in cave mines, discussion with personnel from other cave mines, observation in drawpoints underground, good draw control information and a clear, gabbro sill marker horizon, allowed the author to analyse the material flow pattern in the BA5. The analysis shows that, in a mine experiencing coarse fragmentation, drawzones relate to drawbells and not drawpoints. Interaction across minor apices is better and waste migration faster across minor apices than major apices. It implies that drawbell spacing rather than drawpoint spacing is important. Apices act as a barrier to material flow. Drawbells need to be pulled to achieve interaction across minor apices and that entire lines of drawbells, separated by minor apices, need to be drawn hard to achieve interaction across major apices. Caved material migrates towards areas of high draw with the rate of migration a function of the relative rates of draw. This implies that to achieve uniform draw, drawbells should be pulled at equal rates. Unequal draw could be used force ore migration across a cave. Tonnage redistribution would need to be calculated accurately to ensure that reserves were not underestimated in areas of high draw and grossly overestimated in areas of low draw. The inadvertent creation of areas of high draw could result in an early waste ingress, especially if waste is finely fragmented.

* In new mining blocks and mining areas at Premier, ore reserves will be calculated relative to drawbells rather than drawpoints. Ore will be redistributed to areas of high draw. The observation that there is better interaction across minor apices than major

apices will be used to improve the algorithm used to redistribute tonnages at Premier. Draw control will be planned to ensure interaction between drawbells across minor apices. The fact that interaction between drawzones across major apices is less effective than interaction across minor apices will be used to control waste migration.

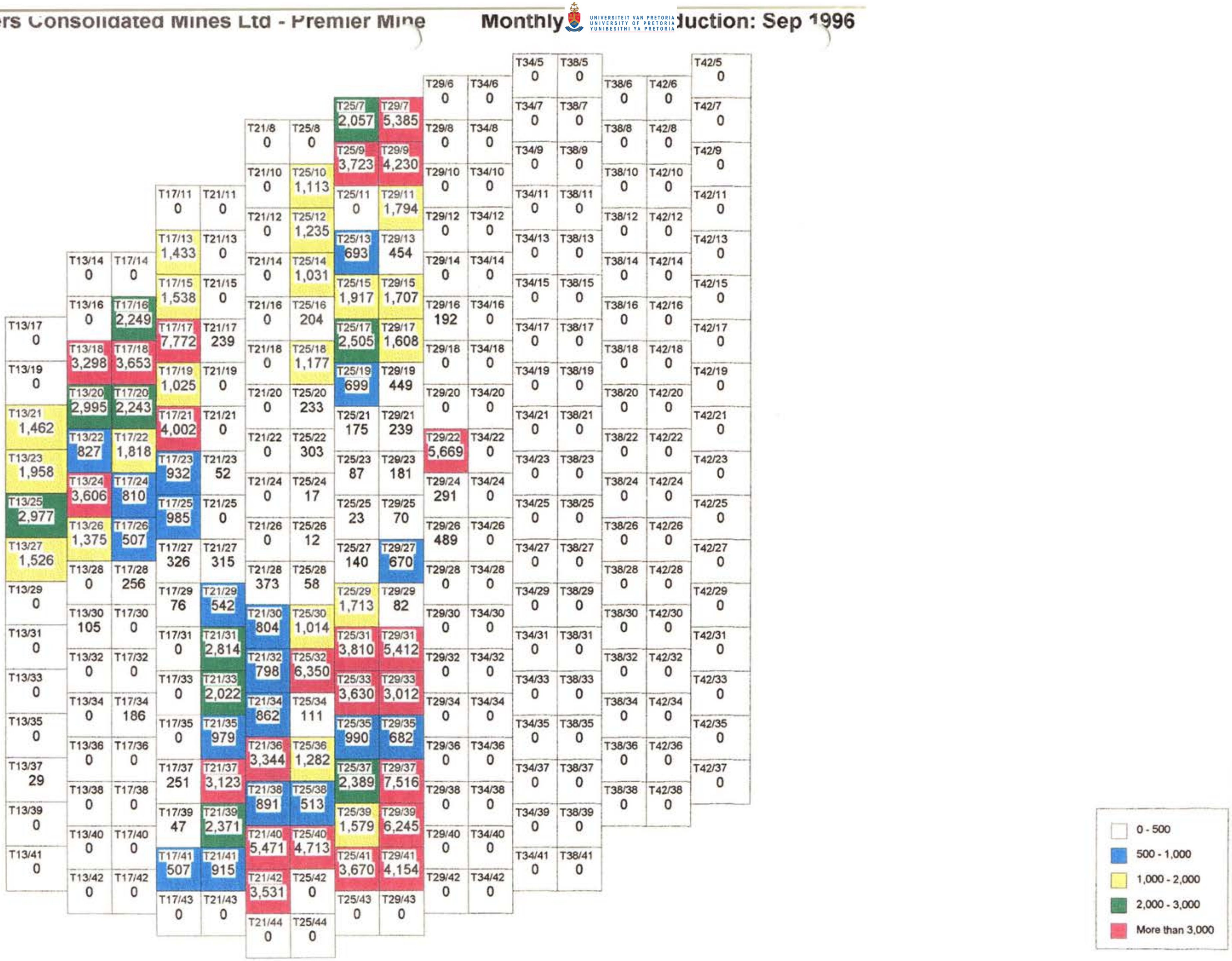



Figure 9.5. Tons drawn for a given month in the BA5

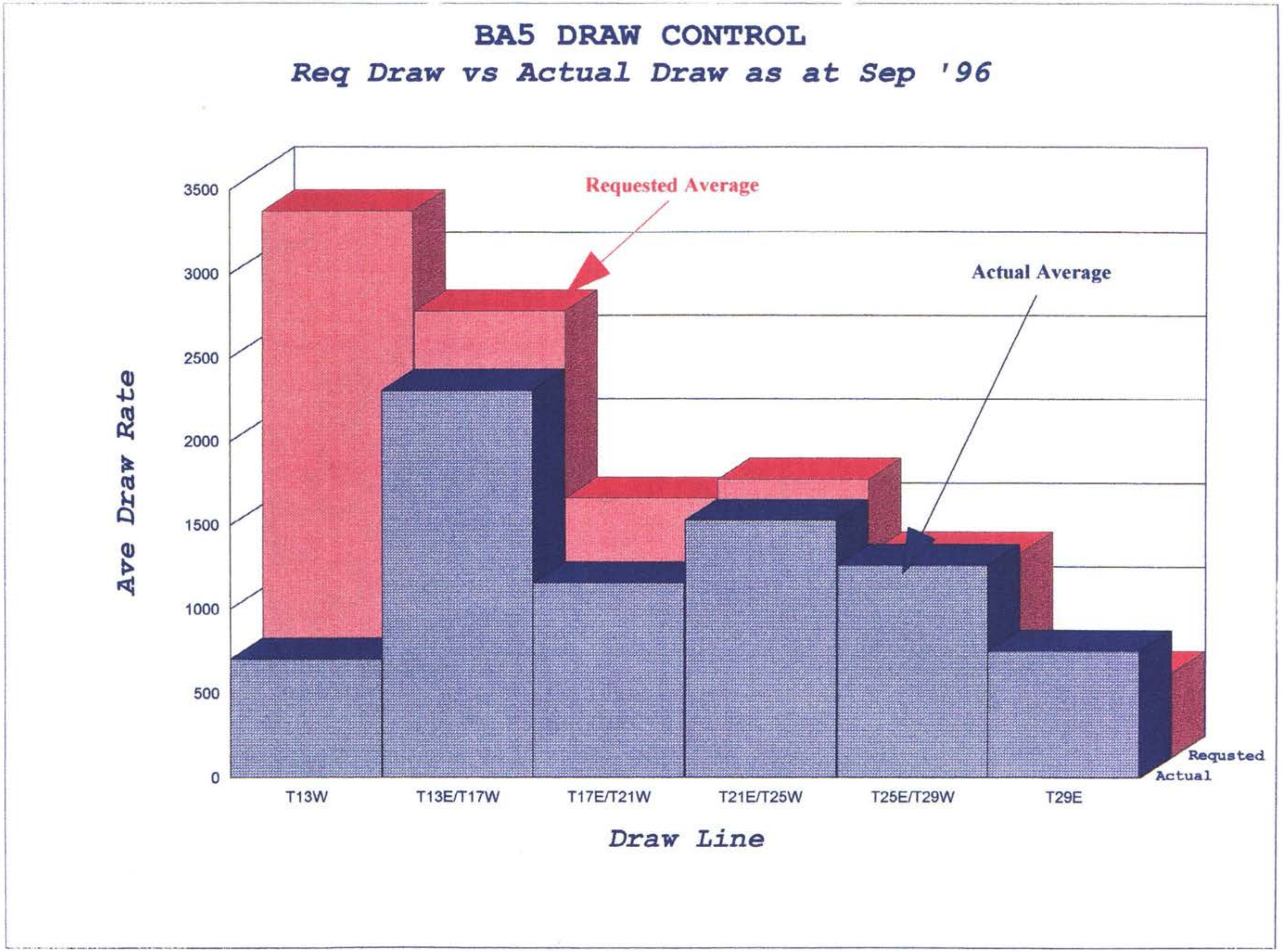



Figure 9.6. Planned vs Actual Depletion

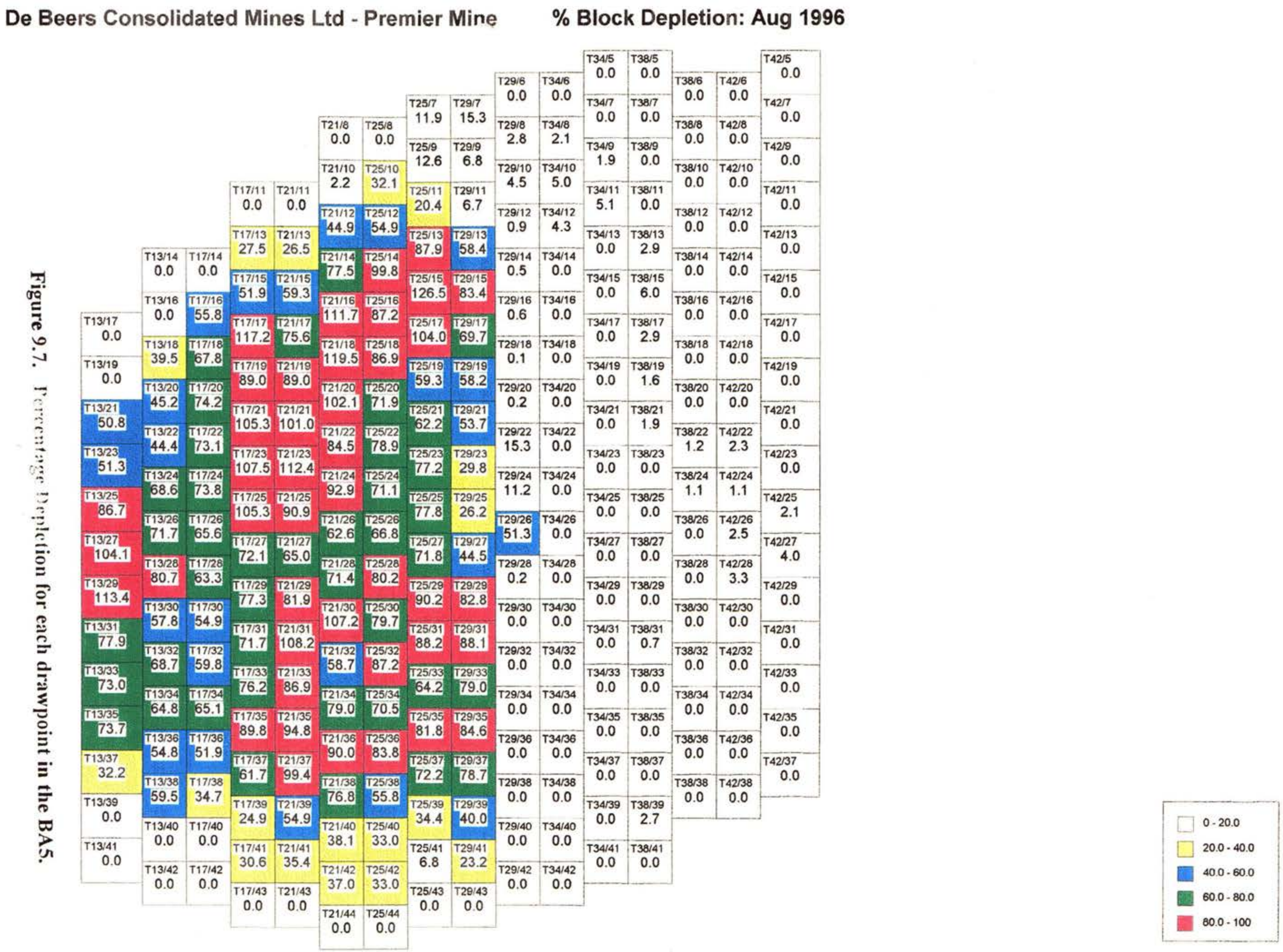



Figure 9.7. Percentage Depletion for each drawpoint in the BA5.

# CHAPTER 10

# PLANNING A MECHANISED CAVE WITH COARSE FRAGMENTATION

## Statement

Five major parameters need to be defined to plan a cave mining operation. In the BA5 at Premier Mine, all these parameters were addressed in terms of the mine's previous experience in cave mining above the sill, by reviewing cave mining literature and visiting mechanised caving mining operations in Chile. The BA5 was planned and implemented. Although the BA5 achieved its planned production targets, several problems were experienced. Undercutting failed to advance at the planned rate and it was difficult to maintain the face shape. Unexpectedly coarse fragmentation affected production adversely. The expensive support system, designed to ensure the integrity of excavations and minimise support rehabilitation, was only partly successful. Failure of the draw control strategy had the potential to create stress problems and result in the loss of ore reserves, but it was not possible to determine the effectiveness of the draw control strategy until numerous drawpoints were drawn to completion.

The author undertook a detailed review of the geotechnical aspects that impacted on the above problems. The objective of this review was to improve the author's understanding of the effect of cave mining on the rock types present in the BA5 cave. The knowledge was systematically quantified to improve mining operations in the BA5 and to develop a structured geotechnical approach that could be used in cave mining in future caves, at greater depth, at Premier and in other De Beers' cave mining operations.

This review involved installing, measuring and analysing the data from a monitoring programme planned to measure rock mass and support response to the cave mining process in the BA5. Additional geotechnical investigations involved extensive numerical modelling by outside consultants, detailed underground mapping by the author and laboratory analysis. Data collection, informed observation and analysis by the author on a regular, planned basis of fragmentation size distribution and its impact on the mining operation, the effect of undercutting on the extraction level and the effectiveness of support was an important part of the process.

This methodology and the way in which it has been applied to improve planning in the BB1E, a mechanised cave planned at a depth of 732 metres below surface in weak kimberlite on the eastern side of the mine, is set out in this chapter.

## 10.1. INTRODUCTION

Major aspects that need to be planned or predicted in any mechanised caving operation are:

1. The area that needs to be undercut to induce caving, the optimum shape, in plan, of the initial undercut excavation and the sequence in which this will be advanced.

2. The fragmentation size distribution that will result as the orebody caves and the fragmentation size distribution that will report to the drawpoints during the life of the drawpoint.

3. The rockmass response to the cave mining operation. The layout should be such that the extraction ratio on both the production and undercut levels leaves sufficient rock in situ to withstand the abutment stresses that will be imposed, as well as the stress effects of subsequent drawing of ore. The mining sequence and undercutting technique should be chosen to minimise stress changes on the extraction level and allow the undercut to be advanced at the planned rate. Support must be sufficient to ensure that excavations remain stable and that drawpoints continue to produce safely during their required life.

4. Draw analysis and control to ensure that waste dilution and stress levels are kept to a minimum and that the cave produces at the required production tempo and extracts the maximum ore reserve at the target grade.

## 10.2. CAVEABILITY AND AREA OF UNDERCUT

As a prerequisite for cave mining it must be established that the orebody will cave. The shape and dimensions of the undercut area that will be needed to induce caving must be predicted.

The most reliable way of estimating the area that needs to be undercut is the empirical correlation that has been determined between rock mass rating and hydraulic radius (Laubscher, 1987. Laubscher, 1995).

Back analysis of experience at Premier showed that the caving process in kimberlite pipes is largely joint controlled and dependent on block fallout under the influence of gravity assisted by shear and tensile failure through intact rock. Once a sufficiently large area has been undercut the weight of rock above the undercut excavation creates a tensile zone which results in joint dilation primarily along horizontal and sub-horizontal joint sets. Immediately above the cave back a negligible minor principal stress and high tangential stresses create a restricted fracture zone that allows the rock to fail in shear where stresses exceed the shear strength of the rock mass. The extent of this zone can be reliably estimated by applying the Hoek and Brown failure criterion. The shear stresses are greatest at the point of maximum curvature of the cave back and result in enhanced propagation of the fracture zone at the apex of the cave back. A progressively smaller hydraulic radius is required to maintain continuous caving. In most kimberlite pipes, the size of the cave that eventually breaks through into old workings is considerably smaller than the hydraulic radius needed to initiate caving at the base of the block. Fines then flood into the cave excavation preventing further caving. Wedge shaped remnants are left against the pipe sidewalls. At Premier, this sequence was repeated but the cave back encountered the competent gabbro sill and caving stopped abruptly. The stresses that occurred in the cave back were never high enough to result in failure in shear of the competent gabbro and caving of the sill occurred only when the weight of sill that had been undercut was sufficient to result in joint dilation along sub-horizontal joints and to overcome the clamping forces on sub-vertical joint sets.

In a kimberlite orebody the following needs to be planned in detail to predict and then accomplish effective caving.

* The shape of the undercut excavation must be optimised to ensure that the least area is undercut to achieve the maximum hydraulic radius. The shape of the undercut advance can be planned to reduce abutment stress levels.

* The undercut sequence must preferably be from weak rock that caves easily to strong rock. Pillars, created when the undercut approaches the kimberlite contact, must be avoided.

* The three dimensional shape of the undercut must be considered, as some caves have the propensity to decrease in area as they propagate towards the surface. The shape of the undercut excavation can assist shear caving or allow uncontrolled, mass caving.

* The rate of undercut advance must be slow enough to allow shearing stresses to propagate in the cave back and fast enough not to damage underlying extraction level excavations.

* The rate of draw must match the rate of caving or voids will be created. This allows the rate of caving to be controlled and decreases abutment stresses. Voids can lead to uncontrolled caving, wedge failures and airblasts.

In the BA5 the shape of the undercut resulted in a planned area of 14 800 square metres to achieve a hydraulic radius of 25. A square undercut shape would have achieved the same hydraulic radius for an undercut area of 10 000 square metres. A high undercut was planned to achieve a fast production buildup. The rate of draw, partly as a result of the high undercut, exceeded the rate of caving and resulted in a substantial void. The rate of undercutting was erratic, large leads and lags developed between undercut drilling tunnels and the shape of the undercut face advance was not optimal.

In the planned BB1E, the predicted hydraulic radius is 20 and will be developed as a square. The undercut face will be advanced as a V-shape. The undercut sequence is planned as an advance undercut and it will not be possible to extract ore from the base of the cave faster than the rate of caving. Once drawbells have been developed, required production rates are conservative and will be monitored to ensure that the undercut excavation remains full at all times.

## 10.3. FRAGMENTATION SIZE DISTRIBUTION

Experience above the sill, together with the geotechnical assessment, based on drill core from the BA5, predicted coarse fragmentation. The information was used to plan drawpoints in the BA5 at a spacing of 15 metres. Secondary blasting in small drawpoints equipped with grizzlies spaced at 700 millimetre in the caves above the sill had been accomplished using Anfex bombs and lay-on charges. Secondary blasting had averaged 250 grams per ton of ore produced throughout the life of the caves, but as the extraction levels were sited partly in competent gabbro and partly in competent, metamorphosed kimberlite minimal destruction of drawpoint brows had occurred.

Drawpoint brows in the BA5 were in weak kimberlite and it was realised that secondary blasting would have to be carefully controlled. It was, however, assumed that the large drawpoints below the sill would result in a reduced incidents of hangups. One metre square grizzlies were placed over ore passes, above surge bins that fed impact breakers which were sited below the production level. Unexpectedly coarse fragmentation and numerous hangups immediately impacted on production. It was soon appreciated that secondary blasting might remain an important part of the cave production process. An accurate estimate of the fragmentation size distribution together with a prediction of the type and frequency of hangups that could be expected was required by the planning department to plan secondary breaking equipment requirements, blasting procedures and production tempo.

No accurate way of predicting fragmentation size distribution or hangup type and frequency, in cave mines, existed. The author assisted in the development of an expert system to predict fragmentation size distribution based on a geotechnical assessment of an orebody. Observation and a method of data collection developed by the author allowed three types of hangups to be defined and their frequency of occurrence to be measured. This allowed the development of a model to predict type and hangup frequency by R. Kear (Esterhuizen et al., 1996). Both models have been calibrated and validated by data collected at Premier in the BA5. It is believed that these models will reduce risk and uncertainty in planning new cave mines expecting coarse fragmentation. The way in which fragmentation size distribution for various rock types can be determined and how this information can be used to plan secondary breaking requirements has been detailed in Chapter 8.

An inability to predict fragmentation size distribution in the BA5 caused problems with grizzly spacings and orepass diameters. Heavy bombing to bring down hangups damaged drawpoint brows and support. Two years lapsed before the required secondary drilling equipment was designed and purchased. Only then could effective secondary blasting procedures be developed and implemented. Secondary blasting breaks in excess of 12 percent of production ore, which occurs in the form of hangups and blockages, on a daily basis. Secondary drilling equipment in the mine consists of 4 short-reach, hydraulic drills and 2 long-reach, pneumatic drills. Cone packs and bombs are still used. Secondary blasting staff exceeds 20 persons and a drill and blast engineer is employed on a full time basis to co-ordinate, audit and improve secondary drilling and blasting techniques on the mine. Secondary breaking costs average more than US$2,0 per ton of ore produced.

In the BB1E, fragmentation size distribution has been predicted. As a result, drawpoint spacing has been increased to 18 metres by 15 metres. Impact breakers have been moved to the production level and grizzly spacings reduced to 600 millimetres by 600 millimetres to reduce hangups and orepass lining damage. Hangup type and frequency have been predicted and the required secondary drilling and blasting equipment ordered in advance. Staff have been trained and secondary drilling and blasting procedures are in place. Detailed planning of secondary breaking requirements has allowed mining simulation to predict production LHD requirements and production tempo from the BB1E cave.

Most cave mines have progressed from mining finely fragmented, often weathered, ore at shallow depth using slusher, grizzly and scraper systems to coarser ore at greater depths using LHD's for extraction. Their experience in many aspects of caving has usually stood them in good stead in successfully achieving this change. Cave mining has proved to be the cheapest underground mining method, but an inability to accurately predict fragmentation size distribution and its impact on the mining of an orebody has introduced an unacceptable degree of uncertainty and risk to new mines contemplating caving, especially in coarser orebodies at depth.

## 10.4. ROCK MASS RESPONSE TO CAVE MINING

Six stages of induced stress loading have been monitored during the implementation of cave mining systems (Cummings et al., 1984). Monitoring at Premier has defined an additional stage of stress change, associated with the development of drawbells, if these are developed prior to the undercut being run over the drawbells.

The problem in most cave mines is to predict the magnitude of stress change and predict the strains that will accompany these stress changes. The rock mass response to the cave mining operation is a function of the rock mass strength and the total stress field imposed on the rock mass during mining. Mechanised cave layouts result in extraction ratios as high as 65 percent on the production level, depending on the layout and drawpoint spacing used. The strength of the minor and major apices that are left situ may not be sufficient to withstand the subsequent stress loading as the undercut is advanced over the apices, if a post undercutting sequence is used. Ore extraction can also lead to harsh stress loading. This can lead to continuous support rehabilitation and even total collapse of drawpoints and production drifts. Figure 7.3. summarises the stress changes that can be expected around the cave excavation.

A detailed geotechnical assessment can be used to predict rock mass strength and determine the parameters required for the three dimensional numerical modelling needed to predict the levels of stress change associated with the various stages of cave mining.

Two undercutting sequences should be considered depending on the strength of the extraction level relative to the stresses that will be imposed on the rock around excavations on the extraction level. Post undercutting is the cheapest undercutting method to implement and operate, at least initially. Development is minimised and no ore handling system is required on the undercut level. Post undercutting can, however, lead to extensive damage to major and minor apices on the extraction level, which results in continuous, expensive support rehabilitation throughout the life of the cave. Most mines implement a post undercut and assume that they will be able to install sufficient support to ensure the stability of the extraction level. As mining depths increase, this assumption is often wrong (Cummings et al., 1984).

Advance undercutting can be used to reduce the extraction ratio on the production horizon to a level that limits rock mass damage, as a result of the undercut passing overhead, to an acceptable level. Numerical modelling is a useful tool that can be used to assess stress levels and predict rock mass damage for progressively increasing extraction ratios on the production level, prior to the undercut advancing overhead. If the extraction level is sited in particularly weak rock and stress levels are high, it might be decided that no development can be tolerated on the extraction level prior to undercutting. The time constraints imposed by this approach need to be evaluated.

Monitoring of stress changes and displacements during the implementation and subsequent operation of the cave can give early warning of stress related problems and assist in support design.

### 10.4.1. EXPERIENCE IN THE BA5

If several aspects of the rock mass response to the cave mining operation had been predicted in advance, time and resources could have been saved in bringing the BA5 into production. Monitoring in the BA5 showed that prior to cave initiation the stress levels associated with the undercut were less than 6 MPa. Only once caving had initiated did the undercut face become a true abutment. Stress levels then increased abruptly by more than 20 MPa.

An area of 100 metres by 100 metres was needed to start the continuous caving process and at least 20 drawbells could have been developed in this area as well as 320 metres of production tunnel, 320 metres of undercut tunnel and 440 metres of drawpoint crosscuts. All this development would have required less support and rigid concrete linings could have been installed and would not have been damaged by mining stresses. The time taken and cost incurred in bringing the cave into production could have been considerably reduced. The BA5 extraction level was massively supported, but even this level of support could not prevent rock mass damage. If the extent of rock mass damage had been predicted, an advance undercut would have been used in the BA5. The undercut sequence has been changed to an advance undercut to extend the BA5 towards the east.

The ability to undercut at a faster rate would have increased the rate of production buildup and limited support damage on the extraction level. The increased flexibility that could have been gained from a faster moving undercut could have been used to minimise leads and lags between adjacent tunnels.

### 10.4.2. PLANNED IMPROVEMENT IN THE BB1E

Correlations established in the BA5 between mining operations and the rock mass response to undercutting, together with numerical modelling, has allowed better prediction of the rock mass response that can be expected in the BB1E.

* An advance undercut has been implemented in the BB1E, with the extraction ratio on the production level limited to less than 20 percent by developing only production drifts and drawbell crosscuts prior to the undercut advancing overhead.

* Final support, within the area of the predicted hydraulic radius, will be installed immediately as support within this area will not be subjected to damaging abutment stress loading. This will allow drawbells within the hydraulic radius to be brought into production quickly and accelerate the production buildup.

* Outside the predicted hydraulic radius, rigid shotcrete linings and concrete drawpoint support will only be installed after the undercut has moved away. Rigid linings will not be subjected to harsh abutment loading.

* The undercut will advance at a rate of not less than 6 metres a month and leads and lags between adjacent undercut tunnels will be kept to less than 6 metres.

## 10.5. SUPPORT REQUIREMENTS AND DESIGN

Support design in most cave mines remains a major problem, especially on the production level. Observation in the United States (Cummings et al, 1984) shows that the level of support installed in cave mines is considerably greater than that recommended by rock mass classifications. Even massive support requires constant rehabilitation. Numerical modelling (Speers, 1990) shows that support often needs to be an order of magnitude greater than is routinely installed in most cave mines to be effective. The approach to support design adopted on most cave mines is to install massive support to limit rock mass damage and accept that support repair and rehabilitation will remain a continuous part of the cave mining operation. At shallow depths where stress levels are low this approach is often acceptable.

At greater depths the rock mass response to the cave mining process creates greater problems. Induced shear fractures as well as movement on joints inevitably has the effect of decreasing the overall tunnel size and this tunnel convergence sets up high thrusts or "hoop stresses" in any rigid lining such as concrete or shotcrete, installed for interbolt support. These thrusts often lead to destruction of the linings. Paradoxically, only massive, rigid linings provide the support pressures needed to prevent growth of the fracture zone around an excavation in a high, variable stress field.

Steel tendon bonds are largely frictional in character and bond effectiveness is affected by stress levels. Grouted steel tendons in a cave mining situation will be adversely affected by up to six levels of stress change. Furthermore, induced fracturing and shear movement on joints leads to short, ineffective lengths of grouted steel tendon in the blast and stress damaged zone around excavations. Under these conditions a philosophy of installing massive support and constantly repairing this may not be a practical approach.

Appreciation of these problems had led to a two phase approach to support in cave mines being advocated. Support sufficient only to ensure the stability of excavations as the undercut is run overhead is installed prior to undercutting. Grouted or anchored steel tendons are often used for active support at this stage. Once the undercut has been run overhead the massive support needed to withstand prolonged secondary blasting, LHD impacts and further stress changes is installed. The problem is to install only the needed level of support as often drawbell development and ore from undercut blasting must be removed before the undercut is finally run overhead.

A third approach to support design is to implement an advance undercut to destress the production level. Excavations are created and support installed in most areas only after the undercut has passed overhead, and are never subjected to harsh abutment loading. The approach has been used successfully at Bell Mine in Quebec and in Gaths Mine in Zimbabwe. Henderson uses a method of simultaneous drawbell and undercut development that imposes one less cycle of stress change on the extraction level. Two De Beer's mines in Kimberley that are developing scraper caves at depths of more than 800 metres in ore that is expected to yield coarse fragmentation have used the principle of an advance undercut and only developed every second production drift prior to undercut development. Once the area has been undercut and caving has initiated the alternate drifts that were originally left out are developed and supported in destressed conditions. Both Palabora and Finsch Mine have planned advance undercuts in feasibility studies to cave mine their orebodies. El Teniente has completed the drilling that will allow advance undercutting over 21 000 square metres in their Isla section.

### 10.5.1. SUPPORT DESIGN AT PREMIER MINE

In installing the BA5 cave at Premier a post-undercut was planned and implemented. The philosophy was that the extraction level would be sufficiently well supported that no pillars would crush and that little additional support rehabilitation would be needed during the life of drawpoints. Crushing on the extraction level did not occur but support rehabilitation has been a continual operation.

It was found that support rehabilitation costs can run as high as R5 000 per linear metre of tunnel. About 20 percent (2000 metres) of the tunnel development in the cave where post-undercutting was practised had to be rehabilitated at least once at an average cost of R3 000 per linear metre of tunnel. A total of 10 percent of development in the cave had to be rehabilitated a second time at a cost of R1000 per metre. The total cost of support rehabilitation was therefore R7 million or 29 cents per ton of ore mined. The original cost of support is R1 per ton. The cost of lost productivity is not considered but can be substantial.

The author was obliged to prove, on geotechnical grounds, that it was more cost effective to install an advance undercut to reduce initial support costs and minimise support rehabilitation in the future, in spite of the additional development required on the undercut level and the time delay that an advance undercut would cause. This option had to be costed against increasing initial support on the extraction level and accepting continuous support rehabilitation as part of the cave mining process. In order to achieve this, the level of rock mass damage during post undercutting and advance undercutting would have to be predicted to design the required support. Support costs for both mining sequences would then have to be compared. The cost of support rehabilitation for both mining sequences would similarly have to be predicted and compared.

Detailed observation and monitoring of rock mass and support damage in the BA5 has allowed the author to predict the extent of rock mass damage that can be expected as a result of the stress changes associated with cave mining operations in various rock types at Premier. Rock mass damage is quantified using Laubscher's Mining Rock Mass Rating, which allows rock mass strength to be derated as a result of the damage imposed by mining operations. If the extent of rock mass damage can be accurately predicted, adequate support can be designed. The details of the support design process and the way in which Laubscher's Rock Mass Rating has been used to derate the rock mass as a function of stress is detailed in Appendix I.

Figure 10.1. summarises the effects of cave mining on the rock mass and the support recommendations developed at Premier Mine. This figure shows that mining activities such as tunnel development and proximity to the undercut excavation can result in a stress change of up to 5 MPa. In well jointed norite this level of stress change can be felt up to 100 metres ahead of the abutment zone and forces shear movement along joints and can result in joint dilation.

This results in a decrease in Rock Mass Rating as a result of a change in joint condition rating. The line designated NOR in Figure 10.1 shows that typically a stress change of 5 MPa will result in a decrease of Rock Mass Rating from 45 to 35. Depending on the support installed and the level of stress change the Mining Rock Mass Rating can be reduced to less than 10 at which stage development is at risk irrespective of the level of installed support. It is important that pillars are not created by the mining layout if this level of stress is expected. Water aggravates the situation.

The $TKB_{sup}$ line shows that Tuffisitic Kimberlite Breccia with a Rock Mass Rating of 66 can be reduced to an Mining Rock Mass Rating of 63 as a result of a change in joint condition rating due to tunnel development or proximity to the undercut excavation. Monitoring shows that the undercut excavation will cause a stress change that will force movement along joints and fractures up to 60 metres from the actual undercut excavation. Induced fracturing starts to occur at a stress change level of 15 MPa by which stage the Mining Rock Mass Rating has been reduced to 40. Further stress changes can reduce the Mining Rock Mass Rating still further. The diagram shows that if the Mining Rock Mass Rating is reduced below 25 the support installed at Premier is unlikely to ensure that excavations remain stable. Monitoring shows that by this stage the level of displacement that occurs is too great to be sustained by the most rigid support element in the support system, usually shotcrete. Fracturing of the rigid lining occurs, followed by ravelling of the rock around the steel tendons. As the effectiveness of the support interbolt confinement reduces, ravelling of the rock around and between steel tendons occurs and major and minor apices fail. At this stage, drilling is impractical and the only effective support that can be installed to confine the rock mass is massive concrete.

The $HYP_{sup}$ line shows that tunnel development and proximity to the undercut excavation can decrease the Rock Mass Rating from 70 to 68 as a result of a change in the joint condition rating. At a level of stress change of 20 MPa induced fracturing starts to occur. By this stage the Mining Rock Mass Rating has decreased to 48.

An important aspect of the slope of the Rock Mass Rating line is the extraction ratio and size of pillar developed in the rock type. The layout at Premier resulted in an extraction ratio of 43% for both Tuffisitic Kimberlite Breccia and Hypabyssal Kimberlite. The extraction ratio in the norite was only 20% but a number of small pillars were developed in this rock type. Once monitoring has established the level of rock damage that can be expected in the various zones as a result of stress changes, numerical modelling can be used to calculate the expected level of stress change as a result of various possible mining sequences. The extent to which the Rock Mass Rating will be decreased as a result of the various anticipated stress changes can be read off the diagram and the level of support needed to ensure excavation stability can be designed using numerical simulation or from support charts linked to an appropriate rock mass classification.

It is important to note that high stresses can reduce the rock mass rating to the extent that it is impractical to support excavations on the extraction level effectively and tunnels can be crushed. A critical level of stress exists for any rock mass. At this level of stress induced fracturing and aggravated movement on joints and fractures becomes widespread and the level of support needed to be effective is an order of magnitude higher than predicted by most rock mass ratings. If it is predicted that this could occur an advance or pre-undercut should be considered and the

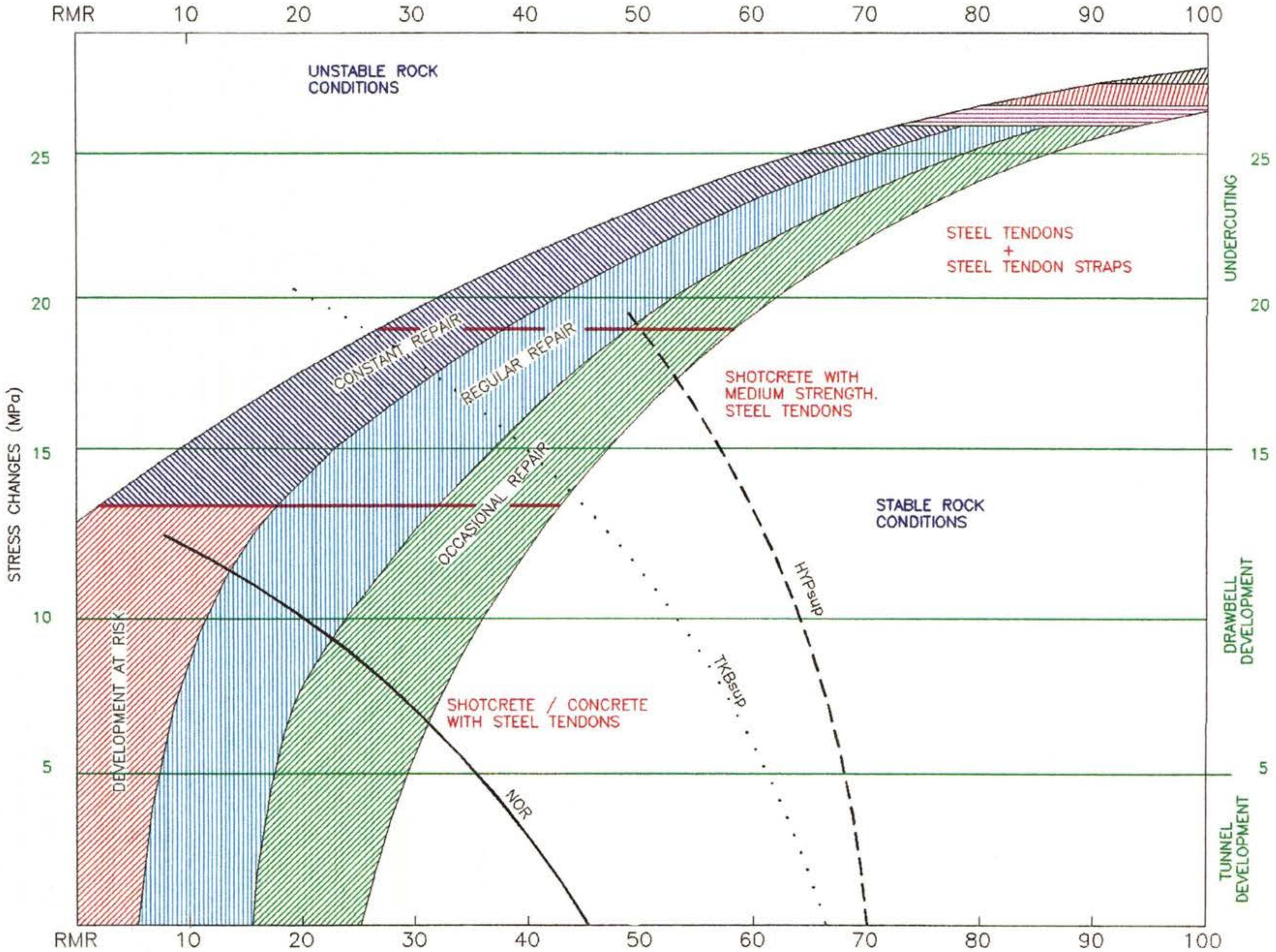



**Figure 10.1. Effects of Cave Mining and Support Recommendations at Premier**

additional cost of installing this compared to the massive support and support rehabilitation that will be needed to allow drawpoints on the extraction level to be drawn to completion.

Once the cave is in production, point loading can still result in stress damage to the extraction level. Extensive secondary blasting can result in aggravated brow wear. A programme of support rehabilitation remains essential on most cave mines. Experience at Premier showed that at least two phases of support were necessary where post-undercutting was practised. The extraction level is supported as it is developed to a pre-determined standard. As drawbells are developed and the undercut is run overhead support and the underlying rockmass are damaged. The damage is further aggravated by extensive blasting of the coarse ore that reports to the drawpoints in the initial stages of draw. After about 30 000 tons of ore have been extracted from a drawpoint, re-support is often necessary in drawpoints that have been subjected to high abutment stresses. Extensive fracturing in the blast damaged zone prevents drilling and support rehabilitation takes the form of massive concrete linings. The extent of support rehabilitation can be minimised by a programme of continuous support damage monitoring that ensures that support is undertaken before drawpoints become unsafe, or are irretrievably damaged and that support resources are optimally allocated.

Where a pre- or advance undercut is implemented a single support cycle is often sufficient to allow a drawpoint to be operated to completion. Experience showed that where the rock had not been damaged by high abutment stresses planned support costs could be halved.

Limited experience at Premier has shown that as a result of pre-undercutting both initial and rehabilitation support costs are reduced by between 30% and 50%. Initial support costs on the production level were calculated at R1-00 per ton and the cost of support rehabilitation at R1-25 per ton. These costs are based on a column height of 120 metres and 80 000 tons per drawpoint. Where an advance undercut was implemented support cost decreased to R0-50 and support rehabilitation costs to R0-15 per ton respectively.

The production and financial implications of installing an advance undercut, a pre-undercut and a post-undercut were compared. The development and infrastructure that was common to all methods of undercutting was not considered. For all three methods of undercutting the first four years of operation, divided into eight half year periods, were costed. The final development on the production level was the same for all three methods. Less development was required on the undercut level for post-undercutting. Results are summarised in Table 10.1. Major considerations that made an advance undercut the most favourable method financially was that post-undercutting resulted in a 30 percent increase in support costs on the production level and increased drawpoint maintenance thereafter. More infrastructure was required on the production level at an earlier stage to bring the post-undercut into production. This required more capital expenditure early in the life of the project for the post-undercut. The pre-undercut resulted in a considerable production delay. Detailed planning showed that a saving of at least R4 million could be achieved over four years by installing an advance undercut. The cost of decreased drawpoint maintenance and increased availability of drawpoints beyond four years was not considered, but will be considerable. If the time value of money is taken into consideration an advance undercut becomes more attractive, financially.

**Table 10.1. Financial Summary of Post-, Pre- and Advance Undercutting**

| Activity | Ist Half | 2nd Half | 3rd Half | 4th Half | 5th Half | 6th Half | 7th Half | 8th Half |
|---|---|---|---|---|---|---|---|---|
| **ADVANCE UNDERCUT** | TOTAL COST OF IMPLEMENTATION | | | | | R13,69 million | | |
| Undercut tunnels | 2,79 | 2,79 | 0,93 | | | | | |
| Undercut | | | | 0,20 | 0,2000 | 0,2000 | 0,20 | 0,20 |
| Production Tunnels | 3,01 | 3,01 | 3,01 | 0,13 | | | | 1,45 |
| Drawbell Development | | | | | 1,4597 | 1,4597 | 1,46 | 1,653, |
| Costs | 5,80 | 5,80 | 3,94 | 0,33 | 1,6597 | 1,6597 | 1,66 | 78 |
| Revenues | | | | | 0,6300 | 1,2600 | 3,15 | |
| **Total cost/revenue** | -5,80 | -5,80 | -3,95 | -0,33 | -1,03 | -0,40 | 1,49 | 2,12 |
| **PRE-UNDERCUT** | TOTAL COST OF IMPLEMENTATION | | | | | R17,37 million | | |
| Undercut tunnels | 2,79 | 2,79 | 0,93 | | | | | |
| Undercut | | | | 0,20 | 0,20 | 0,20 | 0,20 | 0,20 |
| Production Tunnels | | | | | | 3,01 | 3,01 | 3,02 |
| Drawbell Development | | | | | | | | 1,46 |
| Costs | 2,79 | 2,79 | 0,93 | 0,20 | 0,20 | 3,21 | 3,21 | 4,67 |
| Revenues | | | | | | | | 0,63 |
| **Total cost/revenue** | -2,78 | -2,78 | 0,93 | -0,20 | -0,20 | -3,21 | -3,21 | -4,04 |
| **POST UNDERCUT** | TOTAL COST OF IMPLEMENTATION | | | | | R17,25 million | | |
| Undercut tunnels | 2,79 | 2,78 | | | | | | |
| Undercut | | | 0,20 | 0,20 | 0,20 | 0,20 | 0,20 | 0,20 |
| Production Tunnels | 3,92 | 3,92 | 3,92 | 0,16 | | | | |
| Drawpoint Maintenance | | | | | | 0,03 | 0,03 | 0,03 |
| Drawbell Development | | | | 1,90 | 1,90 | 1,90 | 1,90 | 0,23 |
| Costs | 6,70 | 6,70 | 4,12 | 2,26 | 2,10 | 2,13 | 2,13 | 3,78 |
| Revenues | | | | 0,30 | 0,63 | 1,26 | 0,32 | |
| **Total cost/revenue** | -6,70 | -6,70 | -4,11 | -1,96 | -1,47 | -0,87 | 1,02 | 3,5 |

Note: All monetary values are expressed in millions of rands (R1 = US$0,31)

Design of massive concrete arches for drawpoint brow support in the BA5 had shown that these concrete arches would need to be constructed of 70 MPa concrete to withstand imposed stresses if the concrete was placed prior to the undercut advancing overhead. Placing concrete of this strength underground is difficult. Even if these arches were reinforced, they would probably be extensively damaged as the abutment moved overhead. Protruding steel from the damaged arches would then pose a major problem. For this reason, massive reinforced concrete arches were not installed in the BA5 drawpoints.

Drawpoint brows sited in weak kimberlite are a major concern in the BB1E, especially as each drawpoint will produce an average of 100 000 tons of ore. Concrete arch design in the BB1E, however, shows that, if these rigid linings are not subjected to abutment loading, concrete with a strength of only 25 MPa will be required. The probability of extensive damage to steel reinforced arches is low. Reinforced concrete arches are therefore planned for brow support in the BB1E, placed after the advance undercut has passed overhead. Within the predicted hydraulic radius, high levels of stress change are not expected and drawpoints here will be lined with concrete as soon as they are developed. This will allow these drawpoints to be brought into production as soon as they have been undercut and the drawbells developed.

A good understanding of the expected rock mass response in kimberlite allowed the author to prove the cost benefits of an advance undercut on geotechnical grounds. An advance undercut has been used to extend the BA5 eastwards and, although only a limited tonnage has been drawn through drawbells installed after the undercut had passed overhead, rock mass and support damage monitoring has shown negligible damage to bullnoses and drawpoint brows. In the BB1E, in weak Tuffisitic Kimberlite Breccia, production drifts and drawbell crosscuts, which result in an extraction ratio of 20 percent on the extraction level, have been installed at the same time as an advance undercut is being implemented overhead. Monitoring, as detailed in Appendix I, will be used to quantify rock mass and support damage as the undercut advances overhead and as drawbells are developed. If necessary the extraction ratio will be decreased.

## 10.6. DRAW CONTROL AND ANALYSIS

The objectives of draw control analysis are threefold:

* To minimise the ingress of waste.
* To avoid or minimise stress problems
* To ensure that the required production tempo is maintained.

Essential elements of draw control and analysis are anticipation of the fragmentation size distribution reporting to a drawpoint during its planned production period, drawpoint spacing sufficiently close to avoid isolated drawzones, an efficient draw control information system that allows draw control analysis and planning, effective secondary blasting equipment and procedures that ensure that planned draw control can be achieved and the required production tempo maintained.

### 10.6.1. WASTE INGRESS

Experience has taught that, if good draw control is maintained, as much as 70 percent clean ore can be recovered from a cave before waste dilution starts to report at the drawpoints. Where poor draw control is practised, waste ingress can occur when only 35 percent of the calculated ore has been drawn (Stevens et al., 1994). Sand box experiments have shown that optimum extraction is achieved when all drawpoints are pulled at an equal rate. Ideally, therefore, all drawpoints should be pulled at an equal rate on a per shift basis. Rock mechanics and operational requirements often preclude this.

In block caves in Kimberley and Crestmore, drawpoints were deliberately pulled at a variable rate as a function of time and drawpoint distribution to create a differential shearing stress in the draw column that assisted comminution. In panel caves, the rate of draw in drawpoints adjacent to the advancing face are pulled at a rate that allows caving to propagate but still ensures that an airgap that would allow waste ingress does not form between the cave "brow" and the broken ore (deWolfe, 1981). Drawpoints in areas that have already caved are pulled hard. Experience has similarly taught that, if an isolated drawzone above a drawpoint is to be avoided, surrounding drawpoints must be pulled at a rate of at least one third the rate of the drawpoint that has the highest rate of draw in the area (Heslop & Laubscher, 1981).

Fragmentation and rock hardness can create problems over which the mine has little control. Fine ore will move faster than coarse ore. Experience at Premier has demonstrated this very clearly. In the early stages of draw, open channelways exist between large fragments and allow fine material to move to the drawpoints below. The author has often observed a cascade of fine material (-25 mm) pouring out of a drawpoint for several minutes yielding several tons of fines. Fragments that were visible on the undercut can report to the production level 15 metres below minutes later and be drawn. Large fragments in the same drawpoint only move when they are blasted. If the overlying waste is fine, rapid waste ingress is a problem. The situation has been encountered at El Teniente where oxidised, near-surface, diluted primary ore reports to drawpoints from levels above very quickly and cuts off the primary ore. In Kimberley, distinctive "pag" from the 300 metres of overlying waste capping can report to drawpoints that are less than 50 percent drawn.

Once a situation of isolated draw has been created, either as a result of drawpoint spacing or poor draw control and waste has been introduced into the draw column, waste now exists above and adjacent to other operating drawpoints. The rate at which waste reports to operating drawpoints accelerates. If the waste is finely fragmented, waste will report more rapidly than ore in the drawpoints and the waste problem will be aggravated. Draw control becomes complicated.

### 10.6.2. STRESS PROBLEMS

Observation shows that, in caves experiencing coarse fragmentation, large fragments are in contact with one another in the draw column at least during the initial stages of mining. The contact forces that result are sufficient to result in rounding of angular fragments and some breakage through rocks with high aspect ratios. Contact forces are generally not high enough to result in widespread comminution of rock in the draw column.

Most stress related problems in cave mines occur as a result of incomplete undercutting which results in remnant pillars or stubs being left behind and punching through to the extraction level below. The second most common reason for stress related problems in cave mines is compaction of ore above the extraction level either as a result of ore not being drawn or because drawzones of adjacent drawpoints do not overlap and pillars of broken ore are left between drawzones. The pillars compact and eventually punch through to the extraction level in the same way as remnant pillars or stubs. More localised stress problems can result due to arching in the draw column, or the corner of a single large fragment resting on a drawpoint brow. This type of point loading can raise stress levels by between 10 and 25 times the virgin stress level, improving fragmentation or damaging drawpoint brows and apices. If the minor and major apices have already been damaged by abutment stresses and possess only a residual strength, compaction and point loading can be sufficient to cause "pillars" to fail.

### 10.6.3. PRODUCTION TEMPO

In mines with fine fragmentation, a major part of the mining problem is to keep excavations on the production level open so that these can act as conduits to allow the finally fragmented ore to move to rail haulages. Ore flows easily and secondary blasting is a minor, albeit important part, of the extraction process. Secondary blasting is needed to keep the ore moving, but is not an essential factor in maintaining the production tempo. Where the ore is coarse, secondary drilling and blasting remains an integral part of the extraction process and is central to maintaining the production tempo. Secondary drilling and blasting must be carefully planned to handle the hangups that occur both in terms of frequency of occurrence and type of hangup. A system of accurate information gathering to allow effective draw control such as has been developed at Premier is essential to develop the database that is required for control of production and trend analysis of historical data. Typically, the productivity of drawpoints increases as a function of time as comminution improves up until large fragments that have been cushioned in the drawpoints start to report to drawpoints as the cave nears 100 percent extraction.

Premier used the experience gained on other mines to develop their draw control strategy. A vehicle monitoring system and effective secondary breaking equipment and procedures allowed good draw control. The gabbro sill provided a good marker horizon that allowed the onset of waste ingress to be easily monitored. Once waste started to report in several drawpoints, this data was analysed and proved conclusively that the wide drawpoint spacing used in the BA5 had not resulted in early waste ingress, loss of ore reserves or static columns. Observation underground and data analysis also showed that the geometry of the layout influenced ore flow in the draw column. Drawzones related to drawbells rather than drawpoints in the double-sided drawbells installed in the BA5, and the height of drawzone interaction was greater above major apices than across minor apices. Lines of adjacent drawbells, separated by minor apices, became zones of accelerated draw. Ore flow across major apices was restricted.

This data analysis has given confidence that drawpoint spacing can be increased to 18 metres by 15 metres in the BB1E and will not result in early waste ingress or loss of ore reserves. The size of minor apices will increase and the extraction ratio on the extraction level decrease. Guidelines provided by the analysis on ore and waste flow will be used in ore reserve calculation and to improve redistribution algorithms used at Premier.

## 10.7. CONCLUSIONS

Experience gained by the author and other mining personnel has allowed a better understanding of cave mining in coarse fragmentation at Premier Mine. Data collection and analysis by the author have allowed existing techniques and tools to be confirmed for use in a cave experiencing coarse fragmentation at Premier. Additional tools have been developed that will be used at Premier, in caves now being developed and planned at greater depth, to reduce uncertainty and risk, improve mining conditions and reduce costs. These tools can be used on other cave mines.

1. The effectiveness of using an adjusted rockmass rating to predict the hydraulic radius needed to induce caving in various rock types has been tested and accepted at Premier

Mine. This method has been found to be at least as accurate as using a numerical model. Adjusting the rock mass rating to account for stress changes is problematic. Accurately establishing the parameters needed for numerical modelling is at least as difficult.

2. Methods to predict primary and secondary fragmentation have been developed for Premier and tested on the mine. The ability to accurately predict the fragmentation size distribution that will report to drawpoints at various stages of draw has been established. This information can be used to plan drawpoint spacing. If secondary drilling and blasting will be an important part of the production process, the various types and frequency of hangups that are likely to occur can be predicted and used to purchase secondary blasting equipment and plan secondary blasting procedures. Simulation of the mining plan can then accurately predict production tempos.

3. The rock mass response above, below and adjacent to the undercut during post- and advance or pre-undercutting has been investigated in detail by monitoring, planned observations and numerical modelling. It has shown that both rock mechanics and financial benefits can be gained by implementing an advance undercut. The information has been used to optimise cave layout design in terms of both stress changes and financial considerations.

4. Support design methods, developed by Laubscher and Cummings, have been extended to provide a method of support design that can be used in cave mines. The method uses the concept of adjusted rock mass rating with stress and its effect on the rock mass explicitly defined. It is impractical to determine an optimum support system for any individual mine as the support that is installed is a function of the requirements of the support system (passive, active, rigid, flexible), the experience of the mining personnel and the local availability of support elements. It is important that mining personnel understand the stress effects associated with cave mining, the rock mass response of their particular ore and the characteristics of the support elements that they install. Reliable criteria that allow the strength of extraction and undercut level extraction ratios and pillar strengths to be calculated for general application have not been established, but numerical modelling together with accurate rock mass parameters, where these are available, can be used to assess these problems.

5. Careful observation and monitoring of material flow in the draw column have allowed this aspect of cave mining to be better understood and applied to draw control. Important findings here have been fines concentration found at the base of the draw column and observations that show that the drawbell rather individual drawpoints should be considered as defining the drawzone. It suggests that drawzones, in caves experiencing coarse fragmentation, have a larger diameter than originally calculated. Production tunnels and drawbells can be spaced further apart, leaving more rock in situ on the production level and decreasing the likelihood of failure of the minor and major apices. Premier's experience in terms of drawzone spacing, together with the experience of several other cave mines, has allowed the mine to increase drawbell spacing to 18 metres by 15 metres for an effective drawpoint spacing diagonally across the major apex of 21 metres. It has resulted in a cost saving of several million rand in terms of saved development and support costs.

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# APPENDIX 1.

## SECTION 1. SUPPORT DAMAGE MONITORING

This appendix details how support damage monitoring was carried out and how support damage was related to monitored stress changes and displacements. Data was subsequently correlated with results from numerical modelling. Back analysis allowed accurate values of cohesion and angle of friction to be derived for use in 3-DEC and FLAC modelling. At the same time rock mass ratings using Laubscher's classification were carried out on the rock that had been affected by varying stresses and displacements. Laubscher's MRMR was converted to Bieniawski's RMR and used to derive rock mass cohesion and angle of friction values. These values were in turn correlated with those derived for FLAC and 3-DEC by back analysis of monitoring data.

The objective was to carry out rock mass classification underground to derive a rock mass rating of the pristine rock. Numerical modelling was used to calculate the stresses to which the rock affected by the caving process would be subjected. A given stress change would have the effect of decreasing the rock mass rating by progressively causing movement on joints and fractures, dilation and, finally, induced fracturing. The rock mass rating could therefore be objectively adjusted if the level of stress change was known. The adjusted rock mass rating could be used for support design using an empirical chart. Conversely, if the adjusted rock mass rating was known accurate values of cohesion and angle of friction could be calculated using the equations derived by Hoek and Brown (Hoek & Brown, 1988) for numerical modelling and support design.

### MONITORING PROCESS

The extraction level layout was divided into 7 separate areas or zones and support performance in each of these was observed. The areas defined were:

* Drawpoints
* Sidewalls
* Bullnoses
* Camelbacks
* Drawpoint brows
* Footwalls
* Production tunnels

A simple damage scale ranging from 1 to 6 was developed and used to describe support damage.

* **DS1: Minor damage to shotcrete support**. Induced stresses initiated shotcrete fracturing at the point of maximum curvature of the tunnel sidewall (see photograph P.A.1). As the undercut passed over the area inclined fractures were often noted in the shotcrete. These inclined fractures were angled at between 35 and 25 degrees to the vertical. This failure plane corresponds to the failure plane inclination predicted by the Mohr circle construction:

**P.A.1. DS1 on Damage Scale. Shotcrete scaling in tunnel 21 on undercut level 30 metres from abutment (RMR 55).**

$$ß = 45 - ½\phi_i' \quad \text{(eqn a1)}$$

where ß = failure plane inclination
$\phi_i$ = angle of internal friction of the material

An alternate prediction in terms of maximum and minimum effective stress levels is given by Hoek in the same publication (Hoek E., 1983). Similar fractures, described as diagonal fractures have been noted on other cave mines both on the undercut and extraction level (Cummings et al., 1984). At Premier the angle of friction is between 30 and 35 degrees. The predicted inclination of these inclined fractures is therefore between 30 and 33 degrees to the vertical. Minor movement along joints and distortion of excavations led to concrete damage up to 60 metres away from the undercut in surrounding norite (see photograph P.A.2).

* **DS2: Noticeable damage to shotcrete support**. Induced stresses cause extensive fracturing of the rigid shotcrete lining especially along the sidewalls of drawpoints and around camelbacks and bullnoses (see photograph P.A.3 & 4). Damage is the result of movement along joints and fractures.

* In hypabyssal kimberlite this type of support damage is the result of movement on joints and fractures. In the weaker TKB damage is as a result of movement along joints **and** induced shear fractures.

* **DS3: Severe damage to shotcrete support and minor rock mass failure.** Typically, much of the rock damage is at the point of maximum curvature of the tunnel (stress induced damage), near drawpoint brows (secondary blasting) and crosscut sidewalls (LHD impact damage). This type of support damage is illustrated in photographs P.A.5 & 6.

* **DS4: Destruction of rigid lining, damage to steel tendons and rock mass failure.** This type of failure is initiated by induced stresses and aggravated by LHD impacts and secondary blasting. Rigid concrete linings are needed for effective support rehabilitation (see photographs P.A.7 & 8). This level of damage is usually the result of some shear failure through intact rock.

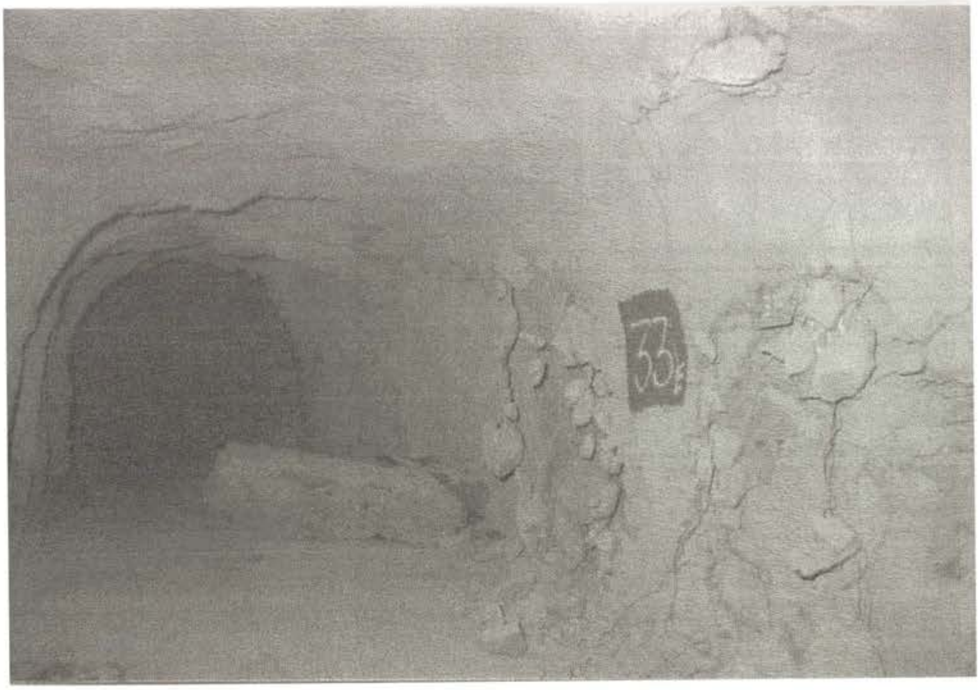
* **DS5: Tunnel collapse.** Tunnel collapse has only been monitored in areas of unusual stress concentration such as beneath pillars and in weak rock. The area has to be abandoned and only re-entered once mining has occurred in adjacent drawpoints and the area has been de-stressed (see photographs P.A.9 & 10). This level of support damage is associated with failure of the rock in tension, shear and compression as well as movement along joints and fractures.

* **DS6: Decomposing kimberlite.** If water is allowed to enter an area of decomposing kimberlite, rapid decomposition due to the expanding montmorillonite clay in the kimberlite can follow which leads to complete loss of strength of the affected rock. These lead to ravelling of the kimberlite around steel tendons and fracturing and disintegration of rigid concrete or shotcrete linings. These effects are illustrated in photographs P.A.11 & 12.

Photographs P.A.13, P.A.14 and P.A. 15 illustrate how the mining process damaged the rock mass and reduced the strength of original rock mass rating (RMR) to a far lesser value (MRMR).

**P.A.2. DS1 on Damage Scale. Concrete damage in refuge bay 60 metres from abutment on 630 level as a result of movement on joints in norite (RMR 45).**

**P.A. 3. DS2 on Damage Scale. Extensive cracking of rigid shotcrete lining in Tuffisitic Kimberlite Breccia on 630 metre level (RMR 50) after abutment stress has passed overhead.**



**P.A.4. DS2 on Damage Scale. Extensive cracking of rigid shotcrete lining on 630 level after undercut has passed overhead (RMR 50).**

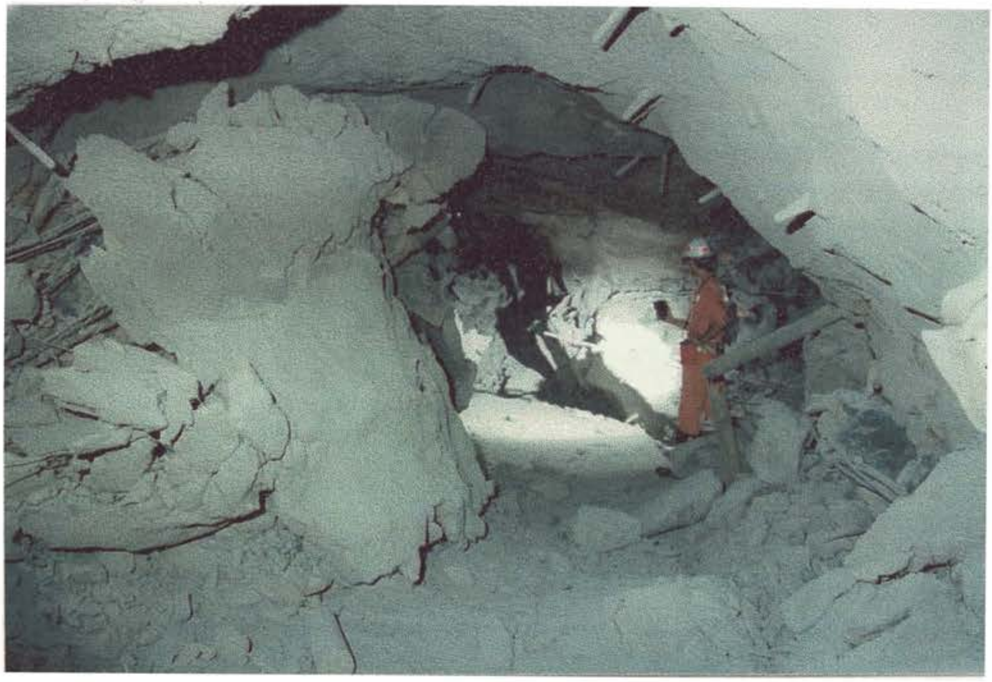
**P.A.5. DS3 on Damage Scale. Damage to steelwork and underlying rock in bullnose on 630 metre level (RMR 52).**

**P.A.6. DS3 on Damage Scale. Damage to support and underlying rock on 630 metre level after undercut has passed overhead.**

**P.A.7. DS4 on Damage Scale. Extensive damage to support and rock on 630 metre level in Tuffisitic Kimberlite Breccia (RMR 50).**

**P.A.8. DS4 on Damage Scale. Extensive damage to support in Tuffisitic Kimberlite Breccia on 630 metre level as a result of induced fracturing after undercut has passed overhead.**

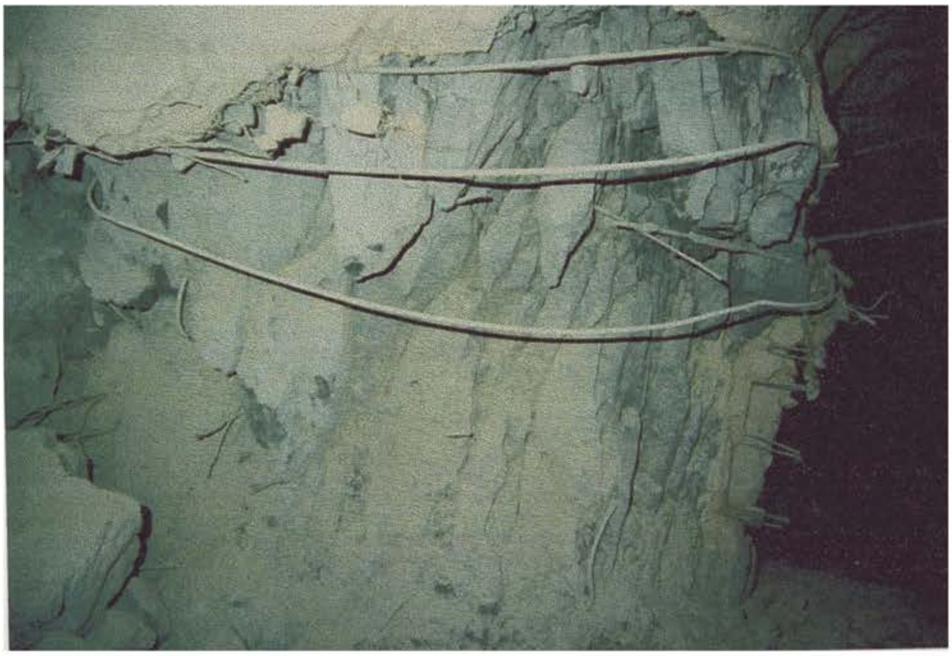
**P.A.9. DS5 on Damage Scale. Collapse of steel arches, induced fracturing and dilation on pristine and induced fracturing around tunnel 120 on 568 metre level in Tuffisitic Kimberlite Breccia (RMR 48) in stressed pillar.**



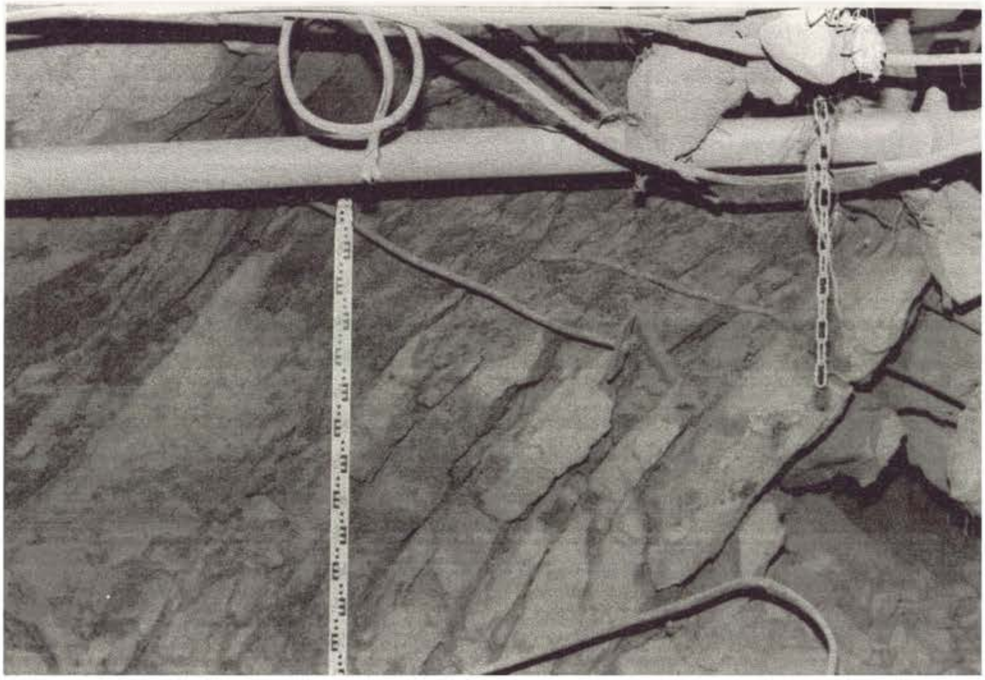
**P.A. 10. DS5 on Damage Scale. Tunnel 109 collapse on 583 metre level in pillar in Tuffisitic Kimberlite Breccia. (RMR 52)**

**P.A.11. DS5 on Damage Scale. Water damage results in decomposition and complete loss of rock mass strength (RMR 52) Location: 30 Main on 630 metre level.**

**P.A..12. DS6 on Damage Scale. Decomposition of kimberlite results in ravelling and fallout of kimberlite around rockbolts with no damage to steel tendons or plate washers. Location;tunnel 117 on 601 metre level (RMR 53).**



**P.A. 13. Induced fracturing in Hypabyssal Kimberlite after drawbell development and undercutting (MRMR 34).**



**P.A.14. Tuffisitic Kimberlite Breccia after undercut has passed overhead (MRMR 27)**

**P.A. 15. Tuffisitic Kimberlite Breccia after it has been affected by water (MRMR 20).**

The geotechnical investigation shows that stress levels as low as 2 MPa can force shear movement along joints and fractures. Damage was correlated with stress changes and displacements in areas where stressmeters and sonic probe extensometers had been installed. Support and rock damage was also correlated with the effect of the undercut being run over the area, area of undercut, the effect of a slow moving undercut, varying leads and lags between adjacent tunnels, rock type, LHD and secondary blasting damage and tons taken from drawpoints.

**Table A.1. Damage Scale**

| DAMAGE SCALE | SUPPORT DAMAGE | ROCK DAMAGE | MEASURED DISPLACEMENT | INDUCED STRESS CHANGE |
|---|---|---|---|---|
| 1 | Minor shotcrete damage | Block fallout | 0-5 mm | 0-2 MPa |
| 2 | Shotcrete damage | Block fallout | 0-15 mm | 0-2 MPa |
| 3 | Major shotcrete damage | Minor brittle failure | 15-50 mm | 2-5 MPa |
| 4 | Destruction of linings. Some damage to steel tendons | Brittle failure | 20-80 mm | 5-20 MPa |
| 5 | Tunnels crushed | Extensive rock failure around excavations | 8–300 mm | >20 MPa |
| 6 | Large scale cracking of rigid linings and ravelling around steel tendons | Rock disintegrates and ravels with time | 80-1000 mm | small. Decomposition causes large stress changes |

**Table A.2. Typical Monitoring Results in Tuffisitic Kimberlite Breccia**

| ROCK:TKB | STRESS CHANGE (MPa) | DILATATION (mm) IN SUCCESSIVE DEPTH ZONES | | | | |
|---|---|---|---|---|---|---|
| T13TR33 | | 0-2,5m | 2,5-4m | 4-5,5m | 5,5-7m | MAX |
| NU | n/a | -112 | -98 | -97 | -96 | -112 |
| SU | n/a | -112 | -110 | -102 | -98 | -112 |
| SD | n/a | 23 | 24 | 25 | 27 | 27 |

n/a = no stressmeter installed

**Table A.3. Typical Monitoring Results in Hypabyssal Kimberlite**

| ROCK: PK | STRESS CHANGE (MPa) | DILATATION (mm) IN SUCCESSIVE DEPTH ZONES | | | | |
|---|---|---|---|---|---|---|
| T25DP34 | | 0-2.5m | 2.5-4m | 4-5.5m | 5.5-7m | MAX |
| NU | 17 | 10 | 10 | 7 | 7 | 10 |
| ND | 17 | 14 | 16 | 28 | 17 | 28 |
| SD | 17 | 2 | 2 | 2 | 2 | 2 |
| SU | 17 | -13 | -13 | -26 | -28 | -28 |

Table A.1. summarises the support and rock mass damage that was observed as well as levels of stress change and displacement that were monitored during this support damage. The overall pattern is one of increasing displacement and rock mass damage with increasing stress levels for all damage apart from that caused by decomposition of the kimberlite. Tables A.2. and A.3. show typical monitoring results from the two kimberlite types. The support damage monitoring showed a consistent pattern of rock and support damage that could be correlated with the abutment passing overhead, the rate of undercut advance, undercut geometry, rock type, secondary blasting and LHD impact.

The planned rate of undercut advance, mining geometry and geology were all available in terms of the mine plan. Stress levels and displacements were taken from the calibrated numerical simulations that had been carried out and used to corroborate the extent of support and rock damage. It was therefore possible to predict the with a high degree of accuracy the derating of the RMR after the various rock types had been affected by the mining process. This information was used to improve support design.

## SECTION 2: CALCULATION OF ROCK MASS STRENGTH AND SUPPORT REQUIREMENTS IN KIMBERLITE IN THE BA5 MINING BLOCK

Monitoring and observation (Cummings et al., 1984. Brumleve, 1988. Jakubec J., 1992) shows that support in cave mining is problematic and usually inadequate. Although much work has been done in applying rock mass classifications to rock affected by caving operations and applying a series of adjustments (Laubscher, 1993. Cummings, 1984. Bieniawski, 1989) a clearly defined empirical correlation between the derived Rock Mass Rating and effective support for all rock types in mechanised cave layouts has not yet been clearly established. Tuffisitic Kimberlite Breccia in particular still poses many support problems that need to be solved if cost effective, long lasting support is to be implemented in caves in kimberlite.

Observation at Premier has shown that the extent to which the rock mass is damaged correlates well with abutment stress changes experienced on the extraction level as the undercut passes overhead. Given levels of stress result in a consistent pattern of rock mass and support damage. Numerical modelling allows accurate calculation of the levels of stress that will occur in terms of the planned mining configuration. Various support elements can only sustain a given level of displacement. For example shotcrete shows extensive damage once 50 millimetres of displacement has been monitored. If a numerical model can be accurately calibrated numerical modelling can provide an accurate estimate of the displacement that will result for a given stress change and mining geometry.

Several numerical models exist which allow the large displacements monitored in kimberlite to be simulated. The most effective of these is the FLAC numerical code. The problem still remains to accurately calculate values of cohesion and angle of friction needed in the model to realistically simulate the rock mass behaviour. Once this has been achieved the effect of the support systems used at Premier such as steel tendons and shotcrete can be simulated to design a cost effective support system. Laboratory data and monitoring are needed to accurately calibrate the models. Experience and data from other cave mines play an important role in the design process.

The support design objective is to apply a rock mass classification to the various rock types affected by cave mining at Premier mine and design cost effective support that will ensure that the extraction level (drawpoints, bullnoses and camelbacks) will continue to function for their planned life with the minimum of support repair. As the required correlation between the rock mass rating (RMR) and an effective support system does not exist for kimberlite, this has to be derived using a suitable numerical model capable of simulating the monitored rock mass response and the effect of the support system. Once an effective support system has been designed and correlations between RMR and the support system have been established support design can be based directly on the RMR as mapped underground and a knowledge of the effect of successive stress levels and displacements will have on the rock mass. The logic of the system developed for the BA5 is set out in Table A.8.

## ROCK MASS CLASSIFICATION

The model developed in the literature (Cummings et al, 1984), monitoring and the geotechnical assessment at Premier had shown that successive stages of cave mining have well defined effects on the rock mass. Laubscher's rock mass classification was used in the geotechnical investigation as this has been tested in numerous cave mining situations and allows adjustments to be made based of the effect of mining on the rock mass.

**Tunnel development** has only a minor effect on the rock mass rating if blasting is carried out efficiently. Poor blasting techniques which result in overbreak and backbreak can have an adverse effect on the size of tunnel and rock strength around the excavation. Tunnel development in the BA5 is usually well controlled and blasting effects are minimal.

**Drawbell development** results in as much as 45 percent of the rock on the extraction level being removed. This results in stress concentration in the remaining major and minor apices. Stress changes of between -10 MPa and +15 MPa are associated with drawbell development.

The monitored effect of drawbell development is to cause displacements of as much as 10 mm. Rock mass classification of an area once the rock has been affected by drawbell development shows that open joints often result. The rock mass rating can be considerably reduced as a result of the change in joint strength.

**Undercutting** can result in stress changes as high as 70 MPa, although 10 MPa to 30 MPa is more usual. This can result in further shear movement along joints and fractures which reduce cohesion and cause dilation on joints and fractures. Induced fractures which range from isolated fractures right through to the creation of closely spaced, en echelon fractures are also created by this level of stress change. This reduces the rock mass rating still further.

This rock must then be supported so that it can withstand the effects of drawing of ore, LHD impacts and secondary blasting with the minimum of support repair, until the drawpoint is drawn to completion.

**Table A.4. Rock Mass Classification after Drawbell development and Undercutting**

| RATING | TKB | | | HYPABYSSAL | | |
|---|---|---|---|---|---|---|
| | Orig. | Draw. | U C | Orig. | Draw | U.C |
| Intact rock | 7 | 7 | 7 | 12 | 7 | 7 |
| RQD | 17 | 8 | 3 | 12 | 7 | 3 |
| Joint Spacing | 25 | 20 | 10 | 20 | 20 | 10 |
| Joint Condition | 6 | 0 | 0 | 12 | 6 | 6 |
| Water | 10 | 10 | 7 | 10 | 10 | 7 |
| RMR | 65 | 45 | 27 | 66 | 50 | 33 |

Orig. Original rock mass rating after tunnel development or from core
Draw. - Rock mass rating after drawbell development
U C - Rock mass rating after undercutting

Photographs A.13. and A.14. illustrate Tuffisitic Kimberlite Breccia and Hypabyssal Kimberlite, respectively, that have been affected by drawbell development and undercutting. Photograph A.13. clearly illustrates the formation of induced fractures with dilation on these fractures that lowers RMR from 65 to 27. Table A.4. sets out the results of rock mass classification of the rock shown in these two photographs. These RMR values are typical of Tuffisitic Kimberlite Breccia and Hypabyssal Kimberlite affected by induced stresses associated with cave mining. The effect of undercutting on the drilling level was much the same as that on the extraction level. The effect of drawbell development was not as marked.

Monitoring therefore indicates that there will be a marked decrease in RMR of the affected rock as a result of the mining process. In order to simulate this decrease using the FLAC numerical code the cohesion and angle of friction of the rock before and after it has been damaged by the abutment stresses must be accurately determined and correlated with the RMR as mapped underground.

**Laboratory data and calculation of rock mass strength (cohesion and angle of friction ) using the semi-empirical Hoek and Brown formulae.**

Correlations between Bieniawski's RMR and Hoek and Brown *m* and *s* values for disturbed and undisturbed rock have been determined.

The most accurate relationship established between that of Laubscher and Bieniawski is:

$$RMR_L = 0.95\ RMR_B - 5.91 \quad \text{(eqn A.2.)}$$

Using this correlation Laubscher's RMR values ($RMR_L$) were converted to Bieniawski's RMR values ($RMR_B$). (Table A.5.)

**Table A.5. Conversion of Laubscher's RMR**

| LAUBSCHER'S RMR | BIENIAWSKI'S RMR |
|---|---|
| 27 | 34 |
| 33 | 41 |
| 45 | 53 |
| 50 | 59 |
| 65 | 74 |
| 66 | 75 |

One of the most difficult problems in rock mechanics is to calculate the strength of a broken or disturbed rock mass and the semi-empirical method of Hoek and Brown of calculating this strength is the most widely used.

The formulae derived by Hoek and Brown (Hoek & Brown, 1988) were used to calculate values of *m* and *s* for disturbed and undisturbed rock masses for the different rock mass ratings mapped underground after drawbell development and undercutting. The cohesion, tensile strength and angle of friction for successive RMR's at different confining pressures as taken from FLAC modelling of the retreat of the undercut (McKinnon, 1992) were calculated using further formulae derived by Hoek and Brown (Hoek & Brown, 1988).

**For disturbed rock masses:**

$$m/m_i = \exp\{(RMR-100)/14\} \quad \text{(eqn A.3.)}$$

$$s = \exp\{(RMR-100)/6\} \quad \text{(eqn A.4.)}$$

**For undisturbed or interlocking rock masses**

$$m/m_i = \exp\{(RMR-100)/28\} \quad \text{(eqn A.5.)}$$

$$s = \exp\{(RMR-100)/9\} \quad \text{(eqn A.6.)}$$

**To calculate the cohesion and angle of friction for a specified confining stress the following formulae have been derived:**

$$h = 1 + \{16(m\sigma_n + s\sigma_c)/(3m^2\sigma_c)\} \quad \text{(eqn A.7.)}$$

$\Theta = 1/3\{90+\arctan(1/\sqrt{h^3} - 1)\}$ (eqn A.8.)

$\phi_i = \arctan\{1/(\sqrt{4h\cos^2\Theta} - 1)\}$ (eqn A.9.)

$\tau = (\cot\phi_i - \cos\phi_i)*(m\sigma_c/8)$ (eqn A.10.)

$c_i = \tau - \sigma_n \tan\phi_i$ (eqn A.11.)

Values calculated from these equations for various RMR's and values of *m* and *s* are tabled in Table A.6. These values of $c_i$ and $\phi_i$ provided order of magnitude values for FLAC numerical modelling.

**Monitoring Results**

Stress change and displacement was monitored at numerous stations on both the extraction level and on the undercut level. These results have been set out in Chapter 5 and in Appendix I. This monitoring showed that, in Hypabyssal Kimberlite, displacements up to 50 millimetres and, in Tuffisitic Kimberlite Breccia, displacements greater than 50 millimetres routinely occurred in rock affected by undercutting. Stress changes of more than 20 MPa were routinely measured during undercutting. Numerical modelling therefore had to replicate these modelled results if the model was to be used as a design tool.

**Numerical Modelling**

Numerical modelling of both the extraction level (Kirsten & Bartlett, 1992) and undercut level (McKinnon, 1992) was undertaken to simulate the stress changes and displacements monitored underground. Modelled stress levels were similar to those monitored but displacements modelled were very dependent on the cohesion (and, to a lesser extent, the angle of friction) chosen for the rock mass. It was therefore important to determine these parameters accurately.

It was also important to determine to what depth around the excavation the rock was effectively "broken" or disturbed. Ideally use of the Hoek and Brown failure criterion should have allowed this to be determined. In terms of monitored results and underground mapping displacement of more than 5 millimetres usually resulted in dilation and a substantial lowering of joint cohesion. Stress to strength ratios using the Hoek and Brown failure criterion of less than 1 were taken to indicate "broken" rock with only a residual strength in terms of modelled results. The Hoek and Brown failure criterion is incorporated in the FLAC model.

**Table A.6. Calculated values of cohesion and angle of friction**

| RMR | TUFFISITIC KIMBERLITE BRECCIA | | | | |
|---|---|---|---|---|---|
| | COHESION | ANGLE OF FRICTION | CONFINING STRESS | *m* | *s* |
| 74 | 1.3 MPa | 58.48 | 0.5 MPa | 2.34 | 0.01 |
| 53 | 0.34 | 50.8 | 0.5 | 0.52 | 0.0004 |
| 34 | 0.19 | 39.3 | 0.5 | 0.13 | 1.67E-05 |
| 74 | 2.18* | 62.7* | 0.5 | 5.92* | 0.055* |
| 53 | 0.7* | 61.9* | 0.5 | 2.8* | 0.0054* |
| 34 | 1.03 | 18.6 | 5 | 0.13 | 1.67E-05 |
| 34 | 1.67 | 13.8 | 10 | 0.13 | 1.67E-05 |
| | HYPABYSSAL KIMBERLITE | | | | |
| 75 | 2.21 | 43.79 | 0.5 | 0.84 | 0.015 |
| 59 | 0.59 | 41.3 | 0.5 | 0.26 | 0.001 |
| 41 | 0.21 | 32.4 | 0.5 | 0.07 | 5.35E-05 |
| 75 | 6.06* | 48.8* | 0.5 | 2.04* | 0.06* |
| 59 | 2.19* | 52.1* | 0.5 | 1.15* | 0.01* |
| 41 | 0.88 | 14.4 | 5 | 0.07 | 5.35E-05 |
| 41 | 1.38 | 10.5 | 10 | 0.07 | 5.35E-05 |

* Values for undisturbed, interlocking rock.
Other values are for disturbed, broken rock.

Underground mapping showed that after the undercut had affected the rock mass it possessed only a residual strength (see photographs A.13. and A.14.) as a result of movement and dilation on joints and induced fracturing. Values of cohesion and angle of friction for broken rock as derived from the original RMR using equations for disturbed rock masses determined by Hoek and Brown should have provided the parameters needed for accurate modelling of the residual strength of the kimberlite after it had been affected by undercutting.

Practically, it was found that residual values of cohesion and angle of friction at low confining stress (0,5 MPa) were reasonable estimates and could be used to correlate FLAC simulations with monitored results. Calculated values of *m* and *s* were too high and did not allow the "failure zone" around the excavation to be accurately simulated. At higher confining stress (5.0 MPa) calculated residual values provided reasonable estimates of cohesion and angle of friction, but values of *m* and *s* were too low to allow accurate simulation of the zone of broken rock around excavations affected by undercutting. This suggests that the cycles of stress change and displacement imposed by cave mining are unexpectedly harsh and rock mass damage is considerably greater (and rock mass strength considerably less) than anticipated. Near the excavation surface the support system provides a support pressure greater than 0,5 MPa and, even though the rock has only a residual strength, it is effectively confined. Further into the rock mass the residual strength is greater than calculated.

It was concluded that the Hoek and Brown failure criterion and correlation with RMR provide only order of magnitude values of cohesion and angles of friction and the more accurate values needed for numerical modelling must be derived by back analysis of monitoring data. The parameters derived from back analysis of monitoring data are set out in Table A.7.

It is important to note, however, that once the rock has been affected by the undercutting process it possesses only a residual strength and support needs to be designed accordingly. The adage "why replace rock with a high strength by concrete with a far lower strength" is not true in a cave mining situation where the rock mass has been affected by drawbell development and high undercut stresses.

As monitoring and modelled stress results correlated well and fitted with the overall model established in the literature for cave mining, modelled stress results were accepted as being accurate. Rock mass strength (cohesion and angle of friction) as calculated from the Hoek and Brown formulae was accepted as less accurate. Values of cohesion and angle of friction were adjusted until modelled displacements matched monitored displacements. A spread sheet was set up in which Hoek and Brown failure criterion was calculated using stress values taken from numerical modelling but with a range of values of $m$ and $s$ values until a good correlation was established between the zone of Hoek and Brown failure and the zone where modelled results showed displacement of more than 5 millimetres.

Steel tendons of various lengths and strengths, various levels of grout bond strength, as well as shotcrete lining of various thickness were then simulated using FLAC until the modelling showed that displacements were less than 50 mm, and neither steel tendons nor grouted bonds broke (Kirsten & Bartlett, 1992; McKinnon, 1992).

**Table A.7. Rock Mass Parameters derived from back analysis of monitoring data**

| RMR | TUFFISITIC KIMBERLITE BRECCIA PARAMETERS | | | |
|---|---|---|---|---|
| | COHESION | ANGLE OF FRICTION | *m* | *s* |
| 50 | 1,2 (0,36) | 35 (50) | 0,3 (0,52) | (0,0004) |
| 25 | 0,5 (0,19) | 30 (40) | 0,01 (0,12) | (1,67e-05) |
| | HYPABYSSAL KIMBERLITE PARAMETERS | | | |
| 55 | 1,5 (0,5) | 35 (40) | 0,4 (0,6) | (0,001) |
| 33 | 1,0 (0,22) | 30 (30) | 0,012 (0,15) | (0,8e-05) |

Notes. 1. A confining stress of 0,5 MPa was assumed in all cases.
2. Figures in brackets are calculated parameters based on equation A.1. through A.10.

In this way the FLAC model was accurately calibrated and correlations were established between $RMR_L$, cohesion, angle of friction, *m* and *s* and support design. An empirical correlation between $RMR_L$ and support design was accurately established.

Support design still drew heavily on experience as gained from the literature, from other mines and from domestic experience Premier . Aspects such as tunnel size, length of steel tendons and shotcrete thickness, which cost the mine money and time, were derived largely from numerical modelling.

Another important aspect of the modelling was that it showed that in some cases it would be impractical to support a drawpoint for its planned life. In other drawpoints a continuous cycle of support repair was indicated. A pre-undercut or advance undercutting was necessary in these areas. This conclusion becomes particularly important as caves move to greater depths and greater lift heights such as is planned in the BB1E and proposed for the C-cut mining block.

## SUPPORT DESIGN SUMMARY

1. Characterise rock mass by laboratory testing (uniaxial compressive strength, triaxial tests) for values of cohesion and angles of friction.
2. Undertake detailed geotechnical mapping using an accepted rock mass classification to determine a reliable rock quality index that can be used to determine rock mass strength (Hoek E., 1983)
3. Use empirically established correlations, as set out in the literature (Hoek. E., 1983), to determine cohesion and friction angles as well as *m* and *s* values for the various, stress-damaged rock types during successive stages of cave mining.
3. Carry out detailed numerical modelling to determine stress levels, displacements as well as failure zones using Hoek and Brown failure criterion.
4. Carry out back analysis of monitoring data until a good correlation is established between numerical simulations and monitoring data during successive stages of mining.
5. Note ϕ, c, *m* and *s* values that allow accurate simulation of monitored data and correlate this with RMR values..
6. Use calibrated model to test support systems in new areas so that displacements can be kept within stable limits.
7. Implement and monitor support system or change layout design and/or mining sequence, if extensive rock mass failure is predicted.

Once accurate correlations have been established between rock mass classification ratings., ϕ, c, *m* and *s* for various rock types affected by various mining stages, and adequate support designed using the numerical models, support design can be based directly on the rock mass rating.

**Table A.8. Summary of Support Design Methodology**

| Rock Mass Classification | Laboratory Testing | Monitoring | Numerical Modelling |
|---|---|---|---|
| 1. Geotechnical mapping<br>e.g. RMR = 65<br>**2. Drawbell Development**<br>*Effect: joint dilation*<br>Geotechnical Mapping<br>e.g. MRMR = 58<br>**3. Undercutting**<br>*Effect: induced fractures*<br>Geotechnical mapping<br>e.g. MRMR = 45 | Triaxial testing<br>Uniaxial testing<br>Derive cohesion and angle of friction<br>1. Calculate m and s for intact rock<br>2. Calculate m and s for broken rock. Compare to RMR and modelled failure zone<br>3. Calculate m and s residual values. Compare to RMR and modelled failure zone | 1. Determine stress change and displacement during tunnelling<br>2. Determine stress change and displacement during drawbell development<br>3. Determine stress change and displacement during undercutting<br>4. Determine zone of dilation around tunnel.<br>5. Determine zone of Hoek and Brown "failure". | 1. Develop accurate geometrical model.<br>2. Use cohesion and friction angle as determined from test data<br>3. Adjust cohesion and angle of friction until modelled results match monitored results.<br>4. Adjust m and s values until modelled results match dilation zone.<br>5. Design support |
| 1. Correlate values of RMR, m and s and cohesion and angle of friction for successive mining stages<br>2. Use numerical model with accurate cohesion and angle of friction parameters to determine effective support design<br>3. Use RMR numerical model correlation for support in rock with predicted MRMR of mining damaged rock.<br>4. Use calibrated numerical model to determine displacements and for modelled stress level and decide on undercutting method<br>5. After decision on advance undercut or post undercutting, decide support system. | | | |

The support design process at Premier culminated in a Support Code of Practice with support in various rock types designed on the basis of calculated stress change and expected displacements. In all cases the support was simulated at various levels of stress and for various displacements using the FLAC numerical code.

| TKB | STRESS CHANGE (MPa) | DILATATION (mm) | | | | |
|---|---|---|---|---|---|---|
| T17DP35 | | ZONE 1 | ZONE 2 | ZONE 3 | ZONE 4 | TOTAL |
| NU | 4.2 | 13 | 17 | 18 | -50 | -50 |
| ND | 4.2 | 28 | 33 | 37 | 72 | 72 |
| SU | -14 | 6 | 10 | 14 | 160 | 160 |
| | | | | | | |

1. Geatest dilatation nearest tunnel
2. Minor brittle failure
3. Total dilatation 160 mm

| TKB | STRESS CHANGE (MPa) | DILATATION (mm) | | | | |
|---|---|---|---|---|---|---|
| T17DP19 | | ZONE 1 | ZONE 2 | ZONE 3 | ZONE 4 | TOTAL |
| NU | -2 | 0 | 0 | 0 | 0 | 0 |
| ND | -2 | -25 | -320 | -320 | -320 | -320 |
| SD | -4 | 0 | 0 | 0 | 0 | 0 |
| SU | -4 | 0 | 0 | 0 | 0 | 0 |

1. Little dilatation
2. Some brittle failure
3. Total dilatation more than 50 mm

| TKB | STRESS CHANGE (MPa) | DILATATION (mm) | | | | |
|---|---|---|---|---|---|---|
| T13-T17 | | ZONE 1 | ZONE 2 | ZONE 3 | ZONE 4 | TOTAL |
| WEST | | 0 | 0 | 3 | 8 | 8 |
| CENTRE | | 7 | 13 | 17 | 21 | 21 |
| EAST | | 16 | 32 | 34 | 44 | 44 |
| | | | | | | |

1.Dilatation greatest in cut zones
2. Increasing brittle failure
3. Total dilation exceeds 50 mm

| PK | STRESS CHANGE (MPa) | DILATATION (mm) | | | | |
|---|---|---|---|---|---|---|
| T29DP21 | | ZONE 1 | ZONE 2 | ZONE 3 | ZONE 4 | TOTAL |
| NU | -4.8 | 0 | 0 | 0 | 0 | 0 |
| ND | -4.8 | 0 | 0 | 0 | 4 | 4 |
| SD | 12 | -3 | -3 | 10 | 36 | 36 |
| SU | 12 | 0 | 0 | 0 | 50 | 50 |

1. Greatest dilation in outer zones
2. Some brittle fialure
3. Total dilatation less than 50 mm

| TKB | STRESS CHANGE (MPa) | DILATATION (mm) | | | | |
|---|---|---|---|---|---|---|
| T13TR23 | | ZONE 1 | ZONE 2 | ZONE 3 | ZONE 4 | TOTAL |
| NU | | 15 | -80 | 15 | | -80 |
| SU | | -78 | -82 | -80 | 38 | -82 |

1. No clear pattern
2. No bittle failure
3. Total dilatation more than 50 mm

| TKB | STRESS CHANGE (MPa) | DILATATION (mm) | | | | |
|---|---|---|---|---|---|---|
| T13TR33 | | ZONE 1 | ZONE 2 | ZONE 3 | ZONE 4 | TOTAL |
| NU | | -112 | -98 | -97 | -96 | -112 |
| SU | | -112 | -110 | -102 | -98 | -112 |
| SD | | 23 | 24 | 25 | 27 | 27 |

1. Dilatation greatest beyond support
2. Minor brittle failure
3. Total dilatation greater than 50 mm

| PK | STRESS CHANGE (MPa) | DILATATION (mm) | | | | |
|---|---|---|---|---|---|---|
| STATION | | ZONE 1 | ZONE 2 | ZONE 3 | ZONE 4 | TOTAL |
| | | | | | | |
| | | | | | | |
| | | | | | | |
| | | | | | | |

| PK | STRESS CHANGE (MPa) | DILATATION (mm) | | | | |
|---|---|---|---|---|---|---|
| T21DP34 | | ZONE 1 | ZONE 2 | ZONE 3 | ZONE 4 | TOTAL |
| NU | 28 | 18 | 38 | 38 | 36 | 38 |
| ND | 28 | 17 | 20 | 18 | 20 | 20 |
| SD | 34 | 13 | -7 | 7 | 12 | 13 |
| SU | 34 | 11 | 10 | 14 | 12 | 14 |
| 1. Outer zones converge by more than deeper zones<br>2. Minor brittle failure<br>3. Total convergence less than 50 mm | | | | | | |

| PK | STRESS CHANGE (MPa) | DILATATION (mm) | | | | |
|---|---|---|---|---|---|---|
| T25DP34 | | ZONE 1 | ZONE 2 | ZONE 3 | ZONE 4 | TOTAL |
| NU | 17 | 10 | 10 | 7 | 7 | 10 |
| ND | 17 | 14 | 16 | 28 | 17 | 28 |
| SD | 17 | 2 | 2 | 2 | 2 | 2 |
| SU | 17 | -13 | -13 | -26 | -28 | -28 |
| 1.Outer zones and inner zones dilate at the same rate<br>2. Minor brittle failure<br>3. Total dilatation less than 50 mm | | | | | | |

| PK | STRESS CHANGE (MPa) | DILATATION (mm) | | | | |
|---|---|---|---|---|---|---|
| T17DP18 | | ZONE 1 | ZONE 2 | ZONE 3 | ZONE 4 | TOTAL |
| NU | | | | | | |
| ND | | | | | | |
| SD | -9.5 | 12 | 10 | 12 | 11 | 12 |
| SU | -9.5 | 0 | 0 | 13 | 15 | 15 |
| | | | | | | |

| PK | STRESS CHANGE (MPa) | DILATATION (mm) | | | | |
|---|---|---|---|---|---|---|
| T21DP35 | | ZONE 1 | ZONE 2 | ZONE 3 | ZONE 4 | TOTAL |
| NU | 8.6 | 20 | 27 | 24 | 26 | 27 |
| ND | 8.6 | 0 | 0 | 0 | 0 | 0 |
| | | | | | | |
| | | | | | | |
| | | | | | | |

| PK | STRESS CHANGE (MPa) | MAXIMUM DILATATION (mm) | | | | |
|---|---|---|---|---|---|---|
| T25DP18 | | ZONE 1 | ZONE 2 | ZONE 3 | ZONE 4 | TOTAL |
| NU | 18 | 0 | 5 | 5 | 7 | 7 |
| ND | 18 | 0 | 0 | 0 | 0 | 0 |
| SD | 3 | 0 | 0 | 12 | 0 | 0 |
| SU | 3 | 0 | 0 | 7 | 65 | 65 |

1. No dilatation beyond support
2. Intense brittle failure
3. Maximum dispalcement of 65 mm

# APPENDIX II

## ANALYSIS OF DRAW CONTROL DATA FROM THE BA5

### 1. INTRODUCTION

The objectives of the draw control strategy implemented in the BA5 are:

* To meet production targets
* To extract as much of the in situ ore reserve as practical
* To mine to established ore reserve grades
* To minimise waste ingress
* To avoid stress related problems
* To maintain the overall draw column profile as planned

The tools that are available to achieve this strategy are rules of thumb that have been developed by other mines and reported in the literature and predicted by physical modelling of material flow. The rules are that, to avoid premature waste ingress and the creation of static columns within the draw column, conditions of isolated draw must be avoided. Practically it means that, within a given area, all drawpoints must be pulled at a rate not less than a third of the most heavily pulled drawpoint. This will achieve interactive draw which will allow the draw column to move down as a continuous layer, provided that drawpoint spacing is correct.

### 2. OBSERVATIONS UNDERGROUND IN THE BA5

1. Drawpoints that might be considered too far apart did not result in conditions of isolated draw. Observation underground in drawpoints in the BA5 showed that, even though some drawpoints were more than 20 metres apart diagonally across the major apex, this did not result in premature waste ingress, or result in stress related problems due to the creation of static columns.

2. Material flow in drawzones related to drawbells rather than drawpoints. Coarse fragments often spanned one or more drawbells. Fragmentation has improved, but coarse fragments, although less frequent, remain a feature of the cave even though some drawbells are more than 100 percent drawn. When support rehabilitation allowed only one drawpoint of the double sided drawbells to be pulled, clusters of rocks formed interlocking arches up to 20 metres above the extraction level footwall and resulted in major hangups. Adjacent drawbells, diagonally across the major apex, were pulled hard in an effort to bring these hangups down, without success. When these drawbells could again be accessed from both sides, the arches soon collapsed and allowed large rocks to move closer to the drawpoints. This sequence of events occurred on several occasions.

3. Fines flowed more readily than coarse fragments and confirmed that material flow related to drawbells rather than drawpoints. It was noticeable that when drawpoints that had been made inaccessible due to support rehabilitation restarted, they produced a

dearth of fines, showing that fines had migrated to the drawpoint on the opposite side of the drawbell. Fines enhancement was not so obvious across major apices.

4. The three dimensional geometry of the cave layout influenced material flow in the draw column close to the extraction level. The height of interaction of drawzones across major apices was considerably higher than across minor apices. When gabbro waste started to report at drawpoints, waste appeared within days at drawpoints on the opposite side of the drawbell after several hundred tons had been drawn from the drawpoints. Waste then spread to adjacent drawbells across minor apices after several thousand tons had been drawn from the adjacent drawbells. Waste migration across major apices often took several months to occur. More than 10 000 tons often had to be drawn from drawbells adjacent to a drawpoint producing waste, but separated from the drawpoint by a major apex, before waste started to report at the adjacent drawbells. Observation showed that many minor apices were extensively damaged and worn after drawbells were 80 percent drawn, reducing the influence of the minor apex on the draw pattern. Brow wear in drawpoints that had not been damaged by undercut stresses and subsequent secondary blasting was often minimal. This suggested that major apices were often little damaged and was sometimes confirmed by observation. Major apices had a considerable effect on material flow towards drawpoints.

These observations proved to the satisfaction of the author that drawzones related to drawbells rather than drawpoints, that drawpoints spaced at 20 metres diagonally across the major apex were not too far apart to allow drawzone interaction in ore as coarsely fragmented as that in the BA5, and that major and minor apices influenced material flow in the draw column.

It did not, however, prove that the wide drawpoint spacing had not resulted in early waste ingress or a loss of ore reserves which might still exist in the form of extensive static columns throughout the cave. The existence of a layer of competent, easily recognised gabbro, provided a good marker horizon. The gabbro has to be carefully monitored when it reports as waste in drawpoints as it impacts on diamond recovery in the treatment plant. These circumstances, together with good draw control information, have provided a unique data set that has allowed the author to prove that the widely spaced drawpoints in the BA5 have not resulted in early waste ingress or substantial loss of ore reserves. The data has been used by the author to corroborate observations made underground that drawzones relate to drawbells rather than drawpoints and that major and minor apices of the extraction level influence material flow.

## 3. DATA ANALYSIS

Draw control provides information as to the tonnage taken from individual drawpoints. The information is collected from the installed vehicle monitoring system and checked against mining records on a daily basis. Tonnages drawn are depleted from the ore reserve that exists in a 15 metre x 15 metre block above the drawpoint that extends to the base of the sill. If the draw is unequal, ore can move from all less heavily drawn drawpoints towards more heavily drawn drawpoints at a uniform rate of flow, influenced only by the relative rates of draw in adjacent drawpoints. Redistribution is done on a monthly basis. Waste started to report to some drawpoints in the centre of the cave during July, 1996 and has become more widespread since then. The percentage waste in each drawpoint is monitored and reported on a daily basis.

Data analysis has been carried out to achieve a best fit between the percentage depletion in each drawpoint and the waste distribution pattern as monitored. The first plot (Figure II. 1) is straightforward depletion with no redistribution of tonnage at the end of August 1996. Table II.1 shows the number of drawpoints that are more than 70 percent drawn and how many of these are producing waste. The correlation between percentage depletion in drawpoints and the appearance of waste is not good.

If tonnage is redistributed on the assumption that ore flows in a uniform manner from less heavily drawn to more heavily drawn drawpoints, influenced only by the relative rates of draw of adjacent drawpoints, the correlation between percentage depletion in drawpoints and the appearance of waste is little improved.

**Table II.1. Depletion of drawpoints as at end of August, 1996.**

| Percentage Depletion | Number of Drawpoints | Number of drawpoints containing waste |
|---|---|---|
| More than 120 % | 0 | 0 |
| 110 - 120% | 4 | 2 |
| 100 - 110% | 8 | 3 |
| 90 - 100% | 7 | 2 |
| 80 - 90 % | 18 | 0 |
| Less than 80% | 95 | 3 |

If the assumption is made that ore will flow towards more heavily pulled drawpoints, but that this flow will be controlled by the major and minor apices, there is good correlation between the percentage of ore drawn from a drawpoint and the appearance of gabbro waste. The assumptions made to achieve the best correlation are:

* Ore is extracted from drawbells and not drawpoints. Individual drawbells are depleted by the total tonnage taken from both drawpoints that access the drawbell.

* Tonnage is redistributed across minor apices when a given drawbell has had 5 000 more tons drawn than the adjacent drawbell. This implies that there is ore flow across minor apices when draw heights between adjacent drawbells exceed 10 metres. Ore then flows toward the more heavily drawn drawpoint, across the minor apex, until the difference in draw height is only 3 metres, between adjacent drawbells.

* It is assumed that ore will only flow towards the more heavily drawn drawbells, across major apices, when 10 000 more tons have been drawn than in the adjacent drawbell. This implies that there is ore flow across major apices to adjacent drawbells when draw heights, diagonally across major apices, differ by more than 20 metres. Ore flow continues until the difference in draw height reduces to six metres.

Relative differences of 10 metres (5 000 tons) across minor apices and 20 metres (10 000 tons) across major apices resulted in a best fit between drawbell depletion and the appearance of gabbro waste. Figure II.2 shows that this scheme of ore depletion resulted in several adjacent drawbells, separated by minor apices, being most depleted. Some of these drawbells were more than 100 percent depleted before waste was started to appear. Waste in adjacent drawbells, separated by minor apices, started to appear when these drawbells were less than 80 percent depleted. It suggests migration of ore toward the most heavily drawn area, followed by the appearance of waste in this area. Adjacent drawbells, separated by minor apices, were depleted of ore, and waste started to appear in the drawbells at a lower percentage depletion. This suggested that the algorithm used to redistribute ore across minor apices needed to be adjusted, and that ore would flow across minor apices when relative draw heights differed by less than ten metres. The algorithm was changed and correlation between percentage drawn from drawbells and the appearance of waste did improve slightly in some areas but not in others. This shows that lines of drawbells, separated by minor apices, can result in areas of accelerated draw, but material flow is more complex than modelled by the simple algorithm used. Flow of ore across major apices is much more restricted.

Since this analysis was carried out in August, ore depletion in production drawpoints in the BA5 has been rapid. September, 1996, statistics are:

**Table II.2. Drawpoint depletion and waste content as at end of September, 1996.**

| Percentage Depletion | Number of Drawpoints (Straight depletion) | Number of drawpoints (Layout influences flow) | Number of these drawpoints containing waste |
|---|---|---|---|
| More than 120 % | 3 | 0 | 1 |
| 110 - 120 % | 4 | 1 | 2 |
| 100 - 110% | 10 | 3 | 6 |
| 90 - 100% | 15 | 9 | 8 |
| 80 - 90% | 10 | 24 | 6 |
| 70 - 80% | 25 | 26 | 4 |
| Less than 70% | 67 | 71 | 4 |

Note: Of the 4 drawpoints that are less than 70 percent depleted and produce waste, 3 are close to the pipe contact and waste has derived from the sidewall rather than from above. The 4th drawpoint is separated from a drawbell that is 80 percent drawn by a minor apex.

Table II. 2. and Figure II.3. show that 47 percent of drawpoints are now more than 70 percent depleted. On average, producing drawpoints are more than 60 percent drawn and waste ingress has been minimal. The overall pattern has not changed much, but is becoming more complicated. In overdrawn areas, 2 or more drawbells are usually more than 100 percent depleted before waste is detected. Ore has migrated towards the overdrawn areas and drawbells adjacent to the overdrawn areas produce waste when they are between 70 and 100 percent drawn. Three such areas now exist and waste flow patterns are starting to anastomose, creating a complicated pattern of waste migration. Figure II.4. shows waste percentages monitored in drawpoints on 4th November, 1996. Four drawbells between production drifts 17 and 21 have been pulled to completion. Waste is starting to appear in drawbells separated from these drawbells by minor pillars. A small amount of waste (5 percent) is being recorded at drawpoint 21/24, across the major apex. Waste has been monitored intermittently in drawpoints separated from waste producing drawpoints by major apices. The overall waste pattern is consistent, but varies in detail on a day to day basis.

Data analysis confirms that the drawpoint spacing used in the BA5, which measures between 19 and 22 metres diagonally across the major apex, has not resulted in early waste ingress or loss of ore reserves. Although gabbro fragmentation is finer than kimberlite fragmentation, the percentage of gabbro in the size fraction less than 100 millimetres is less than in the kimberlite. There are few waste fines produced to cause early waste contamination.

Ore from all areas of the mine is fed to plant and it is not possible to prove that the recovered grade from the BA5 alone is as predicted. The BA5 cave produces in excess of 70 percent of headfeed on a daily basis and mine call factor for the mine during 1996 has been above 100 percent. It is therefore assumed that the BA5 has produced to its target grade.

Implications for Premier of this data analysis are:

1. The drawpoint spacing of up to 22 metres, diagonally across the major apex , in the BA5 has not resulted in a loss of ore reserves, the creation of static pillars or early waste ingress. The proposed drawpoint spacing of 18 metres by 15 metres in the BB1E (21,6 metres across the major apex) should not cause problems.

2. The geometry of the layout has an impact on material flow in the draw column that must be taken into account in calculating ore reserves and understanding interaction between adjacent draw bells . This is illustrated in Figure II.5.

    * Drawzones relate to drawbells and not drawpoints if these drawpoints can be accessed from both sides. Section A-A' shows that the height of interaction of the two drawzones associated with a single drawbell is virtually at the same height as the extraction level. This is observable in drawbells underground.

    * Ore and waste flow more easily across minor apices than major apices. Section B-B' shows that the height of interaction of drawzones separated by minor apices is the height of the minor apex, which can be considerably eroded with time. Interaction between adjacent drawbells separated by minor apices is good. This can be seen underground. Section A-A' shows that interaction across the major apex occurs at a considerable height above the extraction level. Evidence for this is that waste does not migrate across the major apex until several thousand tons of ore have been drawn.

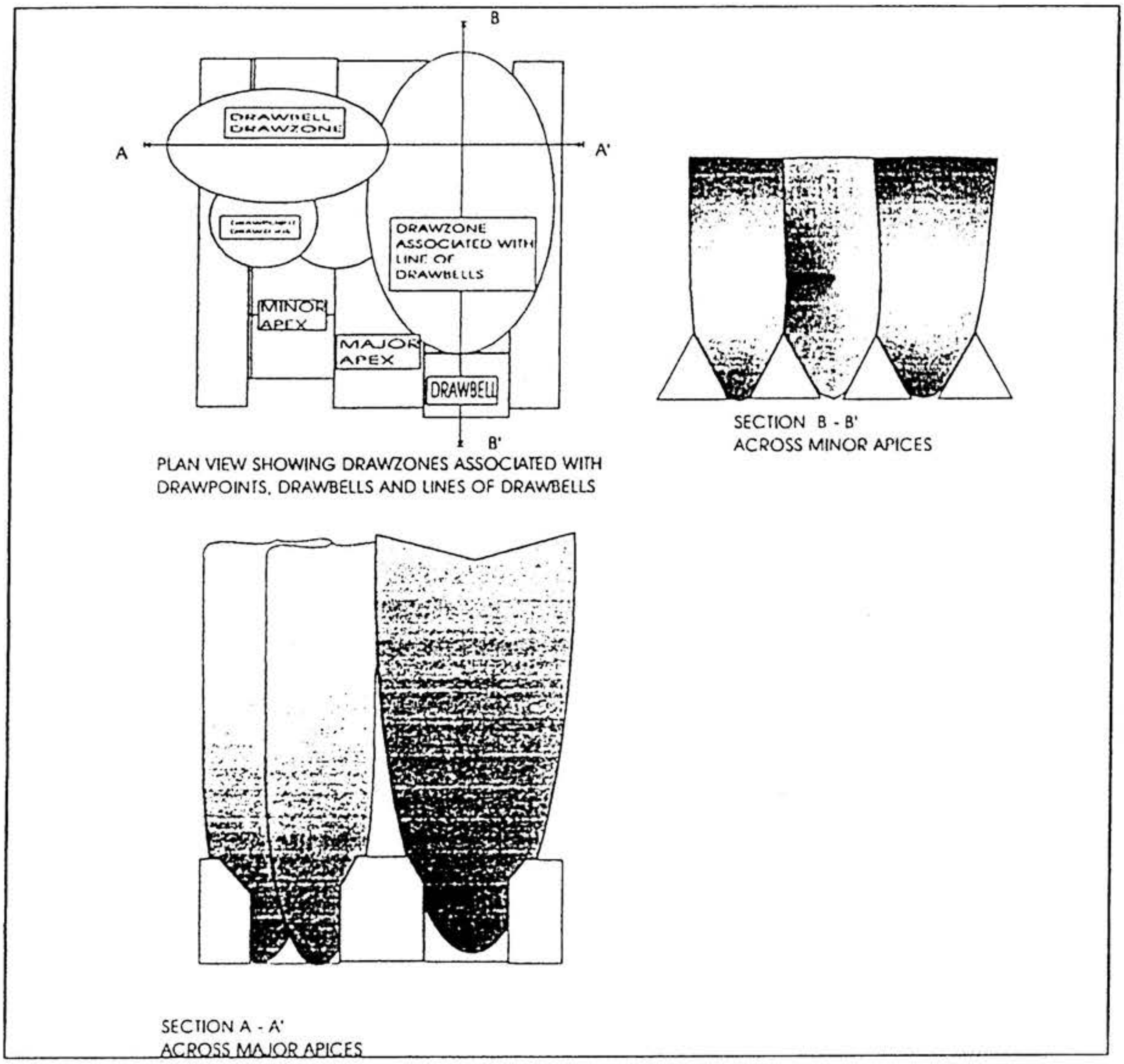



Figure II. 5. Drawzones of various sizes associated with cave layout

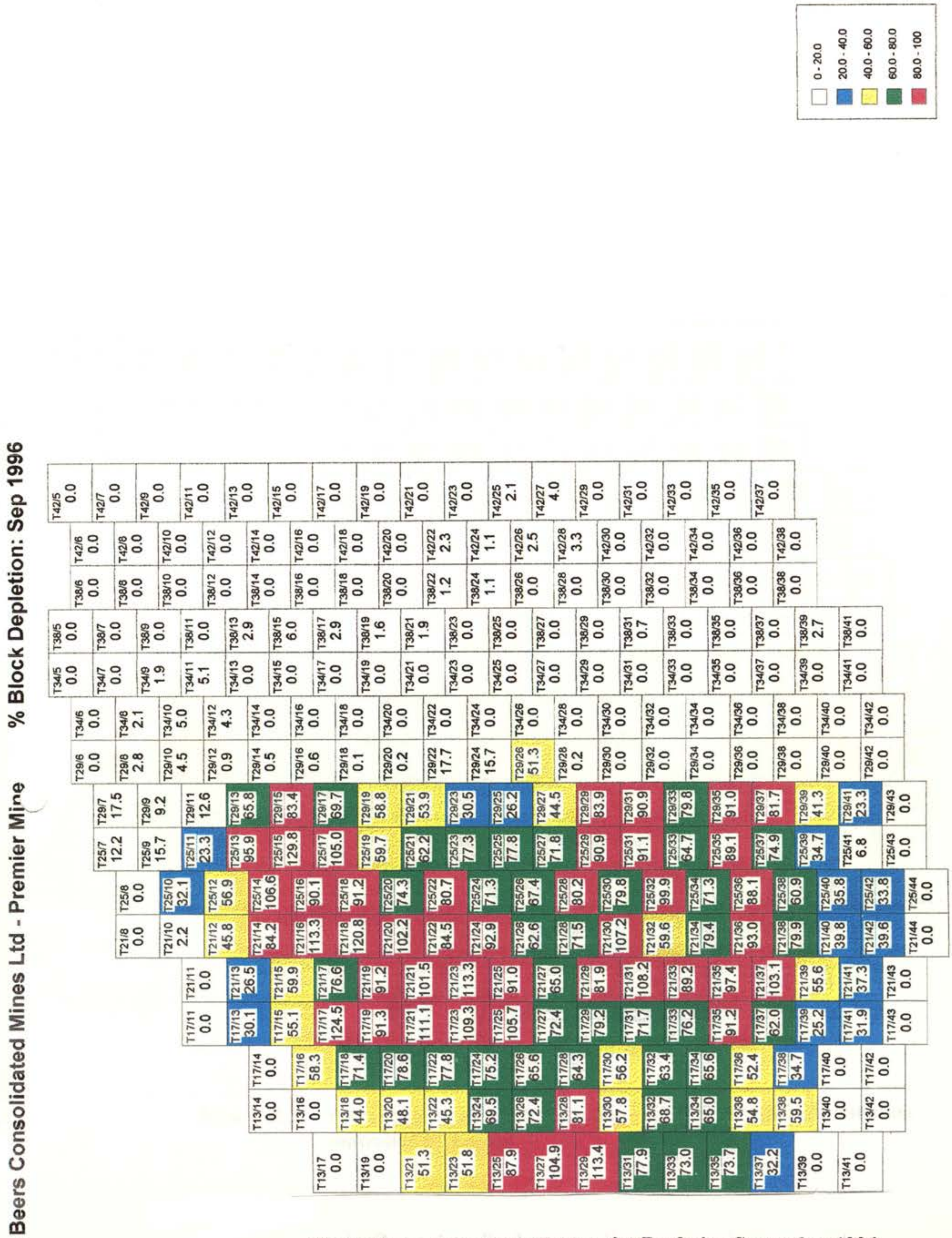
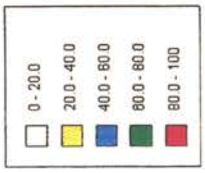



**Figure II.1. Percentage Drawpoint Depletion September 1996**



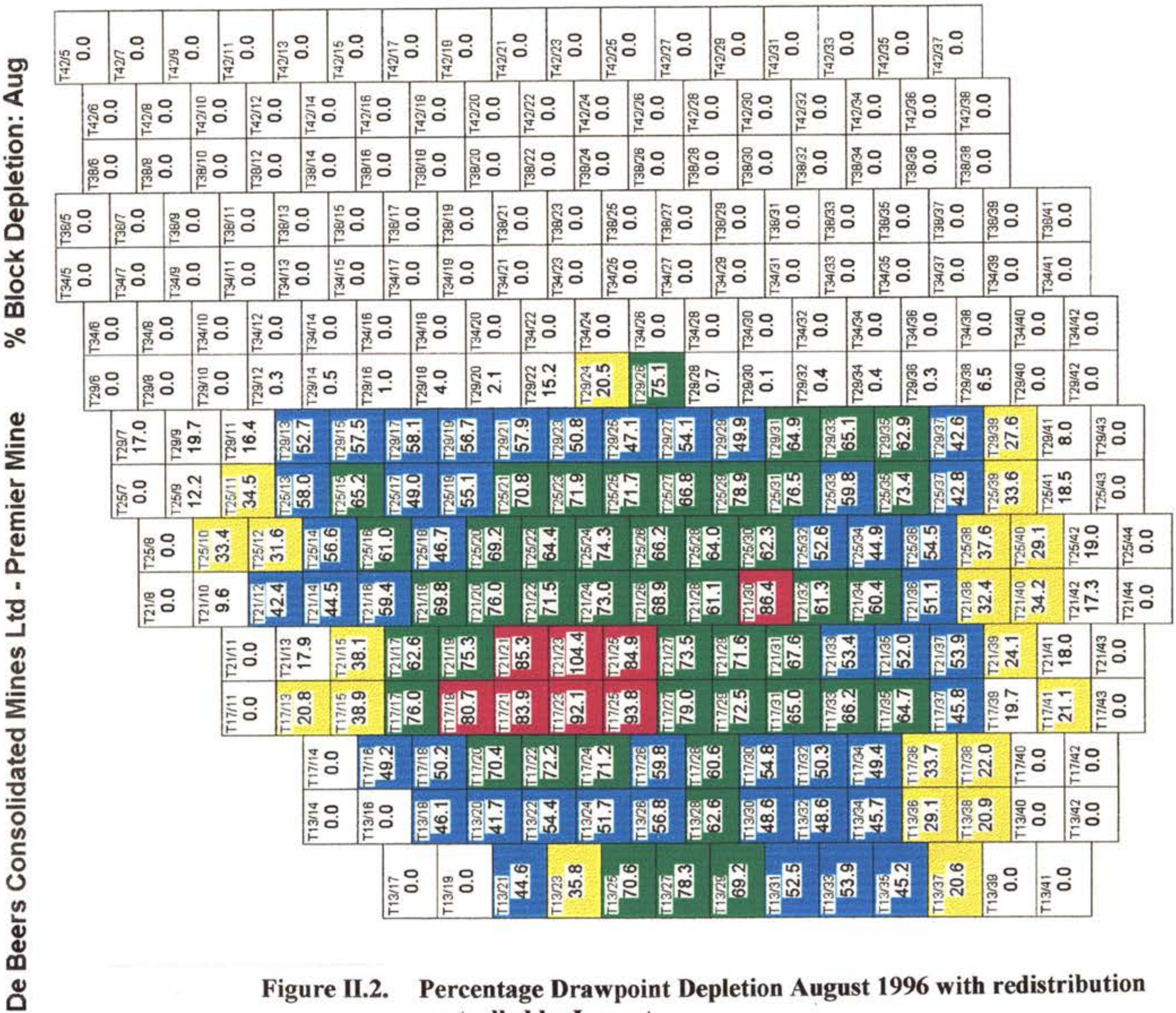




**Figure II.2. Percentage Drawpoint Depletion August 1996 with redistribution controlled by Layout**

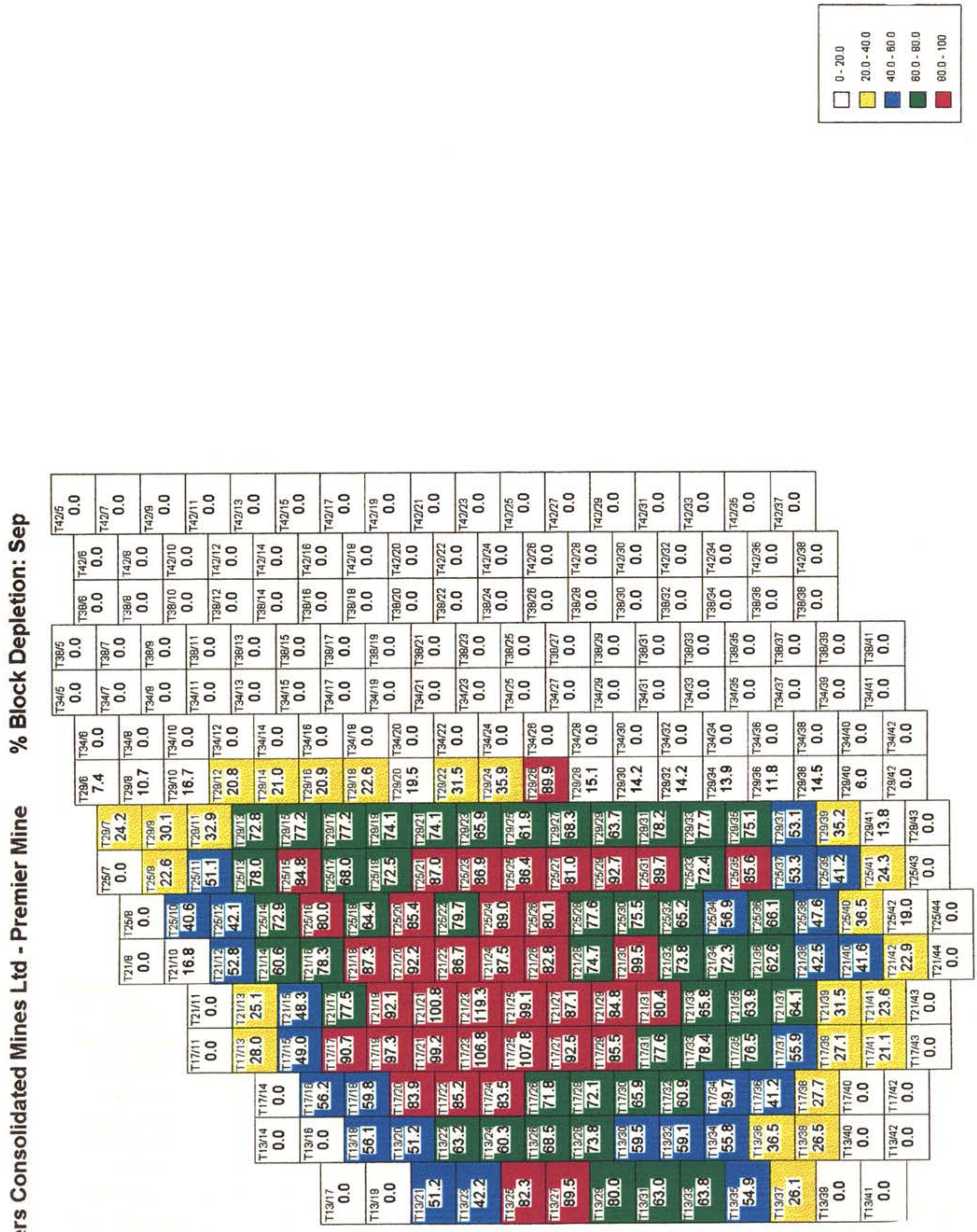
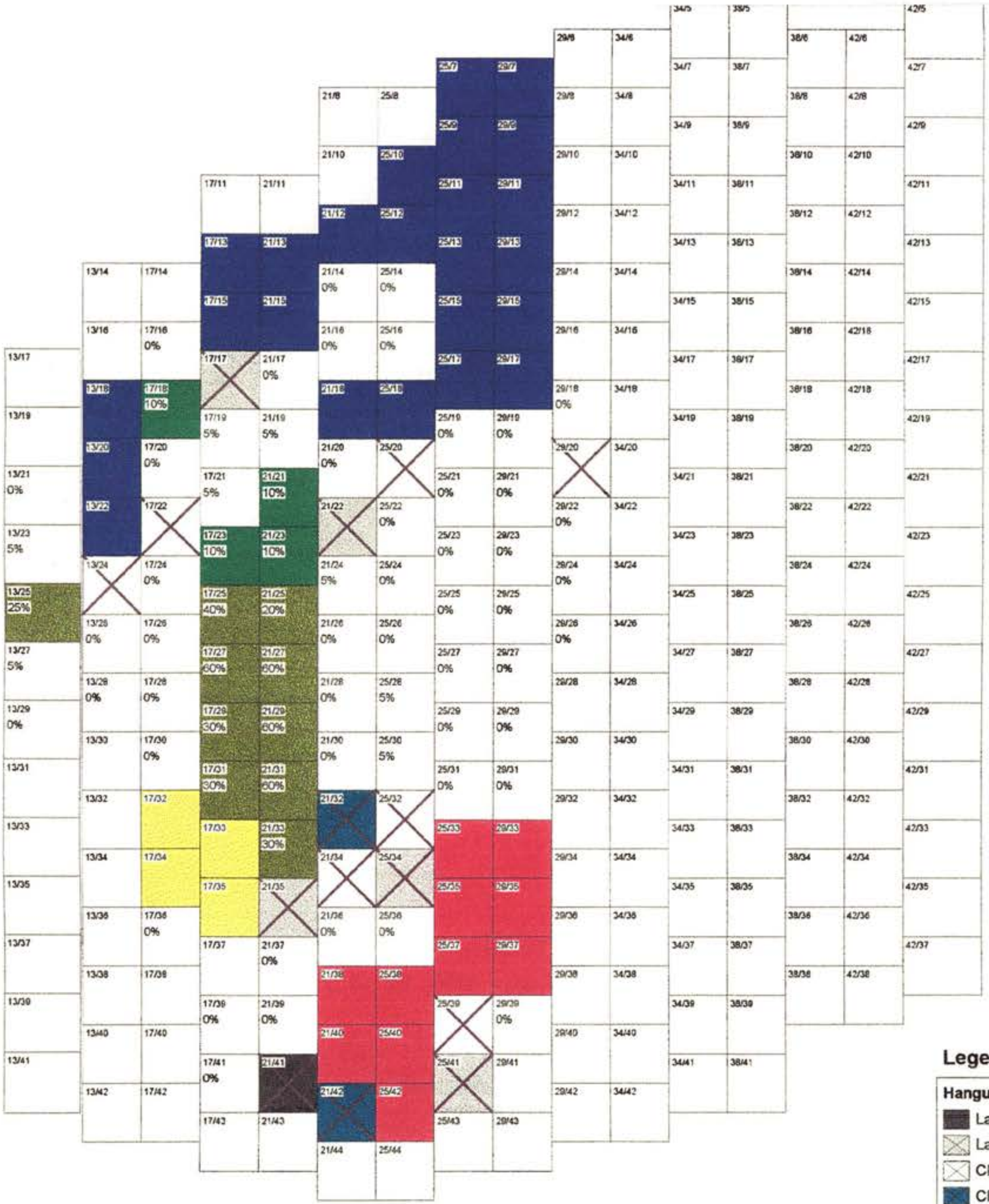



**Figure II.3.** **Percentage Drawpoint Depletion September 1996 with redistribution controlled by Layout**

# De Beers Consolidated Mines Ltd - Premier Mine
# Daily Block Status: BA5 West (Monday, 4 Nov 1996)

Appendix 2

X



**Legend:**

**Hangups:**
- Large High
- Large Low
- Cluster Low
- Cluster High

**Waste Content:**
- < 5%
- 5% - 10%
- 10% - 15%
- 15% - 20%
- > 20%

**Other:**
- Barricaded
- Drilling
- Charging
- Support
- Stopped + BR
- Stopped / Not BR
- Not Available

**Summary:**

| Tunnel | 13 | 17 | 21 | 25 | 29 | 34 | 38 | 42 | Total |
|---|---|---|---|---|---|---|---|---|---|
| Total Hangups | 1 | 2 | 6 | 5 | 1 | 0 | 0 | 0 | 15 |
| Large High | 0 | 0 | 1 | 0 | 0 | 0 | 0 | 0 | 1 |
| Large Low | 0 | 1 | 2 | 2 | 0 | 0 | 0 | 0 | 5 |
| Cluster High | 0 | 0 | 2 | 0 | 0 | 0 | 0 | 0 | 2 |
| Cluster Low | 1 | 1 | 1 | 3 | 1 | 0 | 0 | 0 | 7 |
| Tr's Open | 8 | 19 | 27 | 26 | 16 | 0 | 0 | 0 | 96 |
| Tr's Support | 0 | 4 | 0 | 0 | 0 | 0 | 0 | 0 | 4 |
| Tr's >10%Waste | 1 | 6 | 7 | 0 | 0 | 0 | 0 | 0 | 14 |
| Tr's after HU's | 7 | 17 | 21 | 21 | 15 | 0 | 0 | 0 | 81 |

**Figure II.4. Draw Control Record for Monday November, 1996**

# CODE OF PRACTICE FOR ALL SUPPORT AT PREMIER MINE

## FOREWORD

The purpose of the support code at Premier Mine is to set out clearly and unambiguously the support that must be installed in different rock types in different mining areas to ensure the safety of personnel operating in the area. Support must ensure that production is not adversely affected by rock instability. Support must be cost effective. Support standards are drawn up in terms of generally accepted support practices, experience gained on the mine as well as on support recommendations drawn up for the mine by Steffen, Roberstson and Kirsten (Rigorously Determined Support Characteristics and Support Design Method for Tunnels Subject to Squeezing Conditions H.A.D. Kirsten 1992) and Itasca (Numerical Modelling of BA5 Block Undercut Sequence, Premier Mine S.D. McKinnon 1992).

It should be noted that the support practices set out in this Code are the **minimum** required to ensure the safety of personnel and uninterrupted production. Details of the support code for different rock types and mining areas are set out in Appendix 1. Special situations may require support to be designed for a particular area. This will be undertaken as and when situations arise.

## ISSUING AUTHORITY

Premier Diamond Mine

_______________
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M.Sc. Geol.; D.I.C.

March 1994

## LIST OF CONTENTS

## CHAPTER 1

# 1. INTRODUCTION

Support philosophy at Premier Mine is based on identifying different geological domains in the mining area. A geotechnical assessment of each domain is then undertaken so that a rock mass rating can be given to the rock within the domain. The rating provides a good indication of how the rock will behave during mining operations. Laboratory testwork together with the rock mass rating is used to determine the rock mass strength. Numerical modelling of the rock mass in terms of the accepted mining plan is then undertaken to determine the stresses that will be induced in the rock mass and whether this level of stress will result in failure. Further parameters in determining the support system are expected mode of failure, tunnel size and the length of time that the tunnel is expected to remain open. Quality control in the form of pull tests on rockbolts and cable anchors together with thickness and strength testing of shotcrete is carried out on a regular basis.

It should be noted that support requirements are not for specific areas as on the gold mines but rather for different rock types and the stresses that will be induced in these rocks as a result of the mining process.

# CHAPTER 2

## MECHANISED CAVE MINING METHOD

### INTRODUCTION

The mining method used at Premier mine is termed panel retreat caving. In this method an undercut level is developed to allow the oreblock to be mined to be undercut by drilling and blasting. When a large enough area has been undercut continuous caving initiates and the caved ore is drawn through drawbells on the extraction level 15 metres below the undercut level.

Major problems associated with this mining method are the high stress levels that develop ahead of the advancing undercut as the undercut increases in size and the coarse size of fragments that report to the drawpoints. These have to be broken by secondary drilling and blasting. Support on the extraction level must sustain the effects of the high abutment stresses, continuous secondary blasting and lhd impacts as the ore is loaded in the drawpoints.

The layout of the cave is diagrammatically illustrated in Figure 1.

### THE EXTRACTION LEVEL

The extraction level is developed in ore that is relatively weak or cave mining would not be considered as a mining option. As much as 50% of the rock on the level is extracted to create the drawbells, production tunnels and crosscuts needed for mining. The level is subjected to high and variable stress changes as drawbells are developed, the undercut is run over the level and ore is drawn from the drawpoints. Damaged rock is subjected to further erosion by mining activities such as lhd impacts and secondary blasting. Support must ensure the stability of excavations in this harsh environment and that production is maintained at the required tempo.

This demands careful planning of the layout and support system on the level. The layout of the extraction level in the BA5 is set out in Figure 2.

The intention of the extraction level is twofold:

- To allow the efficient collection of ore from the base of the block that is being mined and transport this ore to a system of ore passes for further treatment.

- In most mechanised caves the extraction level is used as the first step in breaking the ore down to a size that can be accommodated in the rest of the extraction process.

This means that drawbells on the extraction level must be spaced in such a way that all the overlying ore is efficiently extracted. Ore must then be transported to the passes using optimum-sized lhds with as short a tramming distance as possible. Constraints are imposed by fragmentation and a need to maintain the structural integrity of excavations needed for mining on the extraction level.

Experience and material flow theory suggest that the maximum practical drawpoint spacing is of the order of 15 metres. The layout is designed to allow access to these drawpoints using lhds. The rock on the extraction level will be subjected to stress changes associated with tunnel and drawbell development as well as the undercut being run overhead and the subsequent mining of ore.

A decision must be made as to whether post, advance or pre-undercutting will be implemented in terms of the magnitude of anticipated stress changes and rock quality on the extraction level. The level of stress change associated with the undercut being run overhead is 2 to 3 times the virgin stress but this can be increased and stress effects aggravated by the geometry of the undercut, a slow rate of undercut advance and remnant pillars as a result of incomplete undercutting.

Ventilation, roadway construction, pass design and the stability of surrounding service excavations are all important aspects in the design of the extraction level.

## SUPPORT ON THE EXTRACTION LEVEL

Support on the extraction level must be designed to ensure excavation stability during tunnel and drawbell development and as the undercut is run overhead. Thereafter support must withstand erosion by mining activities.

The objective of a support system is to provide sufficient confinement to limit the growth of the fracture zone around the excavation so that displacements remain within stable limits during initial development and the subsequent stress changes that characterise cave mining. The fracture zone must be protected by the support system from erosion by mining activities. If stress levels and rock mass strength are such that collapse of the excavation is probable this should be known in advance. This demands a knowledge of rock mass behaviour during these stress changes and of the characteristics of the support system that is installed.

In **hypabyssal kimberlite** small stress changes of up to 2 MPa are felt up to 30 metres ahead of the abutment zone. This level of stress changes is sufficient to force shear movement along joints and lower cohesion and friction angle on joint planes. Fallout of unpinned and keyblocks under the influence of gravity can be a problem. Effective support can be installed by ensuring that rockbolts are installed to pin blocks and that the bolt and resin can accept the weight imposed on them.

As the abutment passes overhead induced stresses force movement along joints and fractures and create induced fractures to a depth of 2 to 3 metres in the well supported major and minor apices. Induced fracturing of the rock is not equally distributed around the excavation and longer tendons at strategic places such as at the point of maximum curvature of the tunnel are sufficient to create an adequately reinforced ring around the tunnel. Interbolt support consists of tendon straps, mesh and shotcrete to protect the steel work from lhds and secondary blasting. This level of interbolt support is sufficient to prevent any extensive block fallout. The core of minor and major apices possess considerable residual strength and can maintain the overall stability of the extraction level layout. The rock around excavations to a depth of 2 to 3 metres is extensively fractured and possesses a lower residual strength which must be mobilised by effective confinement using a rigid lining. The weak, blast damaged rock, however, allows stress to deform the excavation, setting up high in-plane thrusts that damage the rigid shotcrete or concrete lining. Brow wear in the damaged rock becomes a problem as does damage to the interbolt support. A programme of support repair is needed to ensure that drawbells can be mined to completion.

**In the tuffisitic kimberlite breccia** small stress changes of 2 MPa or less force shear movement along the persistent and often gouge filled, slickensided joints. This can result in block fallout up to 60 metres from the abutment zone. The field stress, concentrated by tunnel development, can be sufficient to cause dogearing around some tunnels. Support in these conditions is the same as that explained for hypabyssal kimberlite. The timing of support installation is important as blast fractures form immediately behind the advancing tunnel face and result in ravelling of the rock between bolts if effective interbolt support is not quickly installed. Abutment stresses result in the same pattern of rock damage as described for the hypabbysal kimberlite. In the weaker tuffisitic kimberlite breccia, however, the entire rock mass that forms the major and minor apices possess a low residual strength after the undercut has moved over the area. The rock possesses a low cohesion and angle of friction. Induced fractures result in short embedment lengths and ineffective grouted tendon support especially in the blast damaged zone adjacent to the excavation. The grouted tendons are rendered even less effective by the stress increases and decreases which affect the frictional bond strength. The usual failure mode for the steel tendons is rupture at the rock/grout interface. The result is extensive ravelling of the rock between and around the tendons to a considerable depth into the blast damaged zone in excavations in TKB.

Massive support is needed to ensure that the support system is sufficiently strong and flexible to withstand the imposed stresses. Rigid linings are constantly damaged and brow wear is rapid. A programme of constant support repair is required. This usually involves the installation of massive concrete linings. The lower limit for shotcrete or concrete lining is a thickness great enough to cover installed steel support in the form of rockbolts, tendon straps and mesh. It is not practical to determine a theoretical upper limit for lining thickness. Practical constraints such as cost and tunnel size usually limit lining thicknesses to less than 1 metre.

Experience has shown that even the most massive support cannot guarantee that excavations will not be crushed by high stresses if the area is not undercut or if compaction occurs and allows stresses to be re-imposed.

This logic for support design is not applicable in all areas of the cave. Bullnoses and camelbacks, because of their geometry, are subjected to unusually high stress loads and impacts by lhds. These areas require effective lateral constraint and a rigid lining to protect them from erosion by mining activities. In the BA5 cave the drawpoint lining consists of creating a formwork of two layers of flexible mesh and steel tendons tied into the rock mass using 3 metre long, resin grouted rockbolts. The steel is then covered with up to 300mm of shotcrete. The inherent strength of rock is harnessed using 6 metre long 25 ton cables installed in a ring pattern on a 1 metre spacing in the 5 metres of crosscut immediately behind the drawpoint brow. This pattern of support extends through the footwall which was further supported using at least 300mm of concrete and steel rails. Minor apices adjacent to the drawpoints are strengthened by installing fully grouted 10 metre long 25 ton cable anchors through the apices. These cables are installed from the production tunnel over the crosscuts.

## SUPPORT ON THE UNDERCUT LEVEL

The layout of the undercut level in the BA5 mining block is set out in Figure 3.

Tunnels on the undercut level are sacrificial and are kept to the minimum size commensurate with their intended purpose. This purpose will always include drilling and, where an advance undercut or pre-undercut is developed, access for rubber-tyred loading equipment. Support is kept to the minimum needed to ensure the safety of personnel and equipment. Smaller tunnel sizes, a lower extraction ratio and the temporary nature of the undercut tunnels allows considerably less support to be used on the extraction level tunnels. The abutment stresses that develop on the undercut level are at least as high as those monitored on the extraction level but, although drilling and blasting must be carried out in rock that is often severely damaged by abutment stresses, these activities are of a temporary nature and prolonged mining operations are not required in the area of damaged rock.

Rock damage away from the abutment zone is well correlated with rock mass rating. Lowering of joint cohesion in the abutment zone as a result of shearing on joint planes can result in open joints which can allow the ingress of water into the area further lowering cohesion and the rock mass rating.

At Premier competent **hypabyssal kimberlite** is minimally supported with 1,8 metre long 16mm diameter, resin-grouted rockbolts on a 1 metre spacing continued down to grade (spring) line, installed within 5 metres of the development face. **Tuffisitic kimberlite breccia** is similarly supported. As soon as possible thereafter the tunnels in TKB are sprayed with a sealant to prevent decomposition of the rock. Interbolt support in the form of chainlink wiremesh covered with tendon straps is installed down to grade line to control any ravelling that might occur in the hangingwall.

In blocky norite minimal levels of stress change force movement along the well developed, persistent joints and rockbolts, cable anchors and mesh-reinforced shotcrete are required to ensure tunnel stability.

Geological mapping is undertaken of all mining development in order to define the rock types present. Geotechnical mapping using Laubscher's rock mass classification system is undertaken in rock where support problems are anticipated. The geotechnical assessment involves:

Determination of rock mass strength by laboratory testing
Determination of the jointing pattern
Determination of the condition and orientation of joints
Determination of hydrological conditions in the area

This information is used to arrive at a rock mass rating which in turn is correlated with the expected rock mass strength. Laboratory testing includes testing for rock strength in uniaxial compression and triaxial testing at various confining stresses to confirm the rock mass strength and determine the mode of failure.

Modes of failure that occur at Premier are block and keyblock fallout in the norite, brittle failure in the hypabyssal kimberlite and brittle failure together with strain softening in the tuffisitic kimberlite breccia. Some areas of kimberlite and norite exhibit squeezing ground conditions if water is allowed to enter the area. A far greater effort is needed to support squeezing ground than is required to prevent block fallout and brittle failure. It is therefore important to determine the failure mode so that a suitable support system can be designed in terms of cost and effectiveness.

## 5.1. Special Situations

Special situations are encountered at bullnoses, camelbacks and brow areas.

### 5.1.1. Bullnoses and Camelbacks

On the production level bullnoses and camelbacks are strapped with 10 strands of steel rope. The area is then encased in shotcrete or concrete to prevent damage by lhds and secondary blasting.

### 5.1.2. Brow areas

Drawpoint brows are in tension and are affected by large displacements and secondary blasting. Long, fully grouted cable anchors are installed in these areas. Shotcrete, steel arches and concrete lining are all used to reinforce these areas depending on the nature of the problem.

Support must be installed within five metres of any new face development. This support is termed **development support** and is aimed at ensuring the safety of personnel working in the area in the short term. This support is installed to prevent or limit block fallout and brittle failure in the blast damaged zone around the excavation. **Final support** is installed prior to the area being affected by induced stresses associated with the cave mining method. The final support system installed is determined by rock mass strength, expected failure mode and predicted stress levels.

# CHAPTER 3

## CODE OF PRACTICE FOR STABILITY OF EXCAVATIONS

### SUPPORT LOGIC FOR DIFFERENT MINING AREAS

| ROCK TYPE | MINING AREA | | |
|---|---|---|---|
| | ROCK AFFECTED BY STRESS CHANGES AROUND CAVE | | ROCK NOT AFFECTED BY CAVE MINING |
| | UNDERCUT LEVEL | PRODUCTION LEVEL | |
| Hypabyssal Kimberlite | Support designed to accommodate large stress changes. Brittle failure expected. Short term excavations | Support designed to accommodate large stress and displacement changes. Brittle failure and stress softening expected | Not applicable |
| Tuffisitic Kimberlite | Support designed to accommodate large stress and displacement changes. Brittle failure and strain softening expected. Short term excavations | Support designed to accommodate large stress and displacement changes. Brittle failure, strain softening and squeezing rock conditions expected | Support designed to accommodate large stress and displacement changes. Brittle failure, strain softening and squeezing rock conditions expected |
| Blocky norite | Support designed to accommodate large stress and displacement changes. Brittle failure, strain softening and squeezing rock conditions expected | Support designed to accommodate large stress and displacement changes. Brittle failure, strain softening and squeezing rock conditions expected | Support designed to accommodate large stress and displacement changes. Brittle failure, strain softening and squeezing rock conditions expected |
| Norite | Support designed to prevent block fallout and brittle failure | Support designed to prevent block fallout and brittle failure | Support designed to prevent block fallout and brittle failure |
| Gabbro and felsite | Support designed to prevent block fallout | Support designed to prevent block fallout | Support designed to prevent block fallout |

### SUPPORT IN UNDERCUT TUNNELS IN KIMBERLITE

| AREA | TKB SUPPORT | HYPABYSSAL SUPPORT |
|---|---|---|
| DRILLING TUNNELS | **DEVELOPMENT:** 1,8 metre long 16 mm resin-grouted gewi-bars on a 1 metre staggered spacing down to grade<br>**FINAL:** Cement-grouted 1,8 m rockbolts on a 0,9 m spacing with mesh continued down to footwall<br>Tendon straps on a 1 metre horizontal spacing. Shotcrete (at least 50 mm) up to a height of 2 metres to cover steel. | 1,8 metre long 16 mm grouted gewi-bars on a 1 metre staggered spacing |
| FOOTWALLS | Support only for special conditions | Support only for special conditions |
| INTERSECTIONS | 3 metre long gewi-bars to be used in place of 1,8 metre rockbolts | 3 metre long gewi-bars to be used in place of 1,8 metre rockbolts |
| BULLNOSES | *Normal tunnel support.* Bullnoses to be strapped with steel rope and shotcreted in when problems arise | *Normal tunnel support.* Bullnoses to be strapped with steel rope and shotcreted in when problems arise |

### SUPPORT CODE OF PRACTICE

## SUPPORT IN KIMBERLITE TUNNELS ON THE EXTRACTION LEVEL

| AREA | TKB SUPPORT | HYPABYSSAL SUPPORT |
|---|---|---|
| PRODUCTION TUNNEL | **DEVELOPMENT:** 1,8 metre long 16 mm resin-grouted gewi-bars on a 1 metre staggered spacing down to grade. **FINAL:** Cement-grouted 1,8 m rockbolts on a 0,9 m spacing with mesh continued down to footwall. Tendon straps on a 1 metre horizontal spacing. Sidewalls covered by 100mm shotcrete to a height of 2 metres. 2 rows of 3 metre long rockbolts in sidewalls: one at tunnel corner and 1 between footwall and grade line. Ten metre long cables to secure drawpoints above the brow area. | 1,8 metre gewi-bars on a 0,8 metre spacing to footwall. Diamond mesh covered by tendon straps on a 1 metre horizontal spacing. Sidewalls covered by 100mm shotcrete to a height of 2 metres. Ten-metre long cables to secure drawpoints over the brow area. |
| DRAWPOINT CROSSCUTS | 1,8 metre gewi-bars on a 0,8 metre spacing. Diamond mesh throughout. 2 layers of mesh and tendon straps covered by shotcrete in drawpoint area. Two rows of cable anchors on a 1 metre spacing starting 1 metre behind drawpoint brow. Two rows of three metre long rockbolts: one at tunnel corner and 1 halfway between footwall and grade line. Rail reinforced concreted footwalls. | 1,8 metre gewi-bars on a 0,8 metre spacing. Diamond mesh throughout. 2 layers of mesh and tendon straps covered by shotcrete in drawpoint area. Two rows of cable anchors on a 1 metre spacing starting 1 metre behind drawpoint brow. Two rows of three metre long rockbolts: one at tunnel corner and 1 halfway between footwall and grade line. Rail reinforced concreted footwalls. |
| BULLNOSES AND CAMELBACKS | Steel rope straps at 300 mm spacing around bullnoses and camelbacks. Can be shotcreted or concreted closed. | Steel rope straps at 300 mm spacing around bullnoses and camelbacks. Can be shotcreted or concreted closed. |
| FOOTWALLS | To be supported only in special situations. | To be supported only in special situations |

## SUPPORT IN NORITE TUNNELS

| AREA | BLOCKY NORITE | COMPETENT NORITE |
|---|---|---|
| TUNNELS | **DEVELOPMENT:** 1,8 metre long 16 mm grouted gewi-bars on a 1 metre staggered spacing down to grade. **FINAL:** Rockbolts on a 0,9 metre spacing with mesh continued down to grade line. Tendon straps on a 1 metre horizontal spacing. Shotcrete up to a height of 2 metres to cover steel. | 1,8 metre long 16 mm grouted gewi-bars on a 1 metre staggered spacing down to grade line |
| FOOTWALLS | To be supported only in special situations | To be supported only in special situations |
| INTERSECTIONS | 3 metre long gewi-bars to be used in place of 1,8 metre rockbolts | 3 metre long gewi-bars to be used in place of 1,8 metre rockbolts |
| BULLNOSES | *Normal tunnel support* Bullnoses to be strapped with steel rope and shotcreted in when problems arise | *Normal tunnel support* Bullnoses to be strapped with steel rope and shotcreted in when problems arise |

The concept of support resistance as applied in the gold mines is not directly applicable to support design in excavations in kimberlite as no accurate way of calculating the support resistance of grouted steel tendons has yet been developed. The support resistance recommended in the situations usually encountered in a mine such as Premier to ensure the safety of personnel and equipment is of the order of 50 kN/m$^2$. In practice considerably higher support pressures are required to ensure that excavations remain stable for their intended life.

Extensive underground monitoring has been undertaken to determine the support requirements needed to maintain the stability of excavations. Monitoring data from sonic probe extensometers has been analysed to determine the actual zone of failure in tunnels in kimberlite. Four zones were defined in terms of the sonic probe data and the installed support system. On the extraction level, in TKB steel tendon support consisted of 1,8 metre long 12 ton rockbolts with an effective embedment length of 1,6 metres, installed on a 1 metre spacing and 25 ton cable anchors with an effective embedment length of 5,5 metres installed with a spacing of 1 metre. A grouted rock anchorage may fail in one or more of the following modes:

1. failure within the rock mass
2. failure of the rock/grout bond
3. failure of the grout/tendon bond
4. failure of the steel tendon or anchor head

Observation showed that rockbolts and cable anchors very seldom failed and that failure of the rock mass usually occurred only after water or unusually high stresses had affected the kimberlite. Failure at the rock/grout bond is the most common failure mode followed by failure at the grout/tendon bond. In order to define some form of index for the effectiveness of the installed tendon support a uniform bond distribution is assumed and the pull out capacity ($T_f$) can be calculated from the equation (Littlejohn, 1992):

$$T_f = \pi D L \tau_{ult}$$

where $\tau_{ult}$ = ultimate bond at the rock/grout interface
$L$ = length of fixed anchor
$D$ = diameter of fixed anchor

This gave a rock grout/rock bond strength of 800 kN/m for hypabyssal kimberlite and 500 kN/m for TKB for grouted rebar and 600 kN/m and 380 kN/m for cable anchors in the respective rock types. Similar values were obtained for the grout tendon interface. It has been shown that bond strength is at least a partial function of confining stress (Kaiser, 1992) and it can be expected that bond strength deeper into the rock mass will be greater than near the surface of excavations where low confining stresses can reduce bond strength by between 30% and 60%. These effects were ignored as values were only used as indices for comparison purposes.

**Zone 1** is closest to the excavation sidewall and extends from the first magnet to the second magnet and is supported with shotcrete, 1,8 metre long rockbolts on a 1 metre spacing and cable anchors on a 1 or 2 metre spacing and extended, on average, 2,5 metres into the rock. Numerical modelling showed that the confining stress at a depth of 1,25 metres into the sidewall is 5 MPa whilst reinforcement, in terms of steel tendon density (rockbolts and cable anchors), is calculated at 252 kN/m² for TKB and 400 kN/m² for hypabyssal kimberlite.

DE BEERS CONSOLIDATED MINES LIMITED - PREMIER MINE

# MINING DEPARTMENTAL PROCEDURE

| | | | | | |
|---|---|---|---|---|---|
| DRAFTED BY | : | Mining Superintendent | PROCEDURE NO. | : | M-DPR-04.7 |
| DISTRIBUTION | : | Supervisory Personnel | REVISION NO. | : | Nil |
| DATE DRAFTED | : | 1 April 1996 | DATE OF ISSUE | | |
| | | | RELATED POLICY | : | Nil |

SUPPORT

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## CHANGES TO SUPPORT SPECIFICATIONS

1. INTRODUCTION

Section 34 of the Minerals Act, Act No. 50 of 1991 states that, if required by the Regional Director concerned, the mine shall draft and apply Codes of Practice in connection with Safety and Health.

In 1993, the Regional Director called for a Code of Practice for support for Premier Diamond Mine; the Code of Practice was finalised and accepted by the Regional Director in mid 1994. As a Code of Practice, this document has the same legal implications as any regulation framed under the Minerals Act.

The Code of Practice provides detailed specifications of support to be used in the different types of rock and in different types of excavation. The detailed specifications given annually for the excavations to be supported during the year must comply with the Code of Practice.

2. RESPONSIBILITIES

It is the responsibility of each legal appointee to ensure that the support specified for a particular excavation complies with the Code of Practice and that the support actually installed is, as a minimum, in accordance with Code of Practice.

3. CHANGES TO SUPPORT SPECIFICATIONS

A change in support specification can be initiated by:

- mining officials (miners, shiftbosses, mine overseers, etc.)
- official union representation through recognised channels
- safety representative acting through their line supervisors
- mine planning personnel
- rock mechanic personnel

3.1 Additional support

The Code of Practice sets out the minimum support requirement. If in the opinion of the responsible person, additional support is required due to bad ground conditions, excessive stress etc, this can be implemented at the discretion of the miner and shiftboss in consultation with the Mine Overseer.

3.2 A reduction in support specification can be initiated by:

3.2.1 A request for a reduction in support specification can be initiated by verbal or written communication to the rock mechanics/geology section or the Mine Superintendent or both.

3.2.2 After considering the request, any change in support specification must be approved, in writing, by the Resident Geologist (or his official representative), the Mining Superintendent or both. Written approval must include a detailed description of the area in which the support specification is to be changed and details of the new support specification and should take the form a memorandum signed by the Resident Geologist or Mining Superintendent (See Appendix I). If necessary a map defining the area can be appended.

3.2.3 Each Mine Overseer must have a copy of the support code of practice and a map of the various levels showing the support specifications for new development within his area of responsibility during the year. The annual development and support plan held by the planning department showing support specifications in terms of the annual mining plan will remain the official document. The survey department keeps maps on a level by level basis showing details of the installed support. Copies of approved changes to support specifications will be given, in writing, to the affected mine overseer, the planning department and the survey department.

P.A.J. HEAP
MINING SUPERINTENDENT

ITR/mdp/A MDPR04.7
1 April 1996

4. If rock conditions are poor during sliping operations rockbolts may be needed to ensure the safety of personnel. These must be installed as soon as practical after the area of pass has been sliped. The pass must be lined with concrete thereafter.

## 3. LARGE EXCAVATIONS

Large excavations such as those required to house workshops and crushers chambers must usually be sited in norite. Such excavations must be sited in competent rock where they will not be affected by or affect other nearby excavations. Such excavations are usually needed for a considerable length of time (several years) and support must be designed to ensure the safety of personnel and equipment in these excavations. Support design is aimed at preventing block fallout. Support must consist of 3 metre long rockbolts on a 1 metre staggered spacing and steel rope anchors on a 2 metre spacing. support must be continued down to the grade line. Rockbolts and anchors must have faceplates. If deemed necessary by the rock mechanic or planning engineer the area must be additionally supported with wire mesh and shotcrete.

K.C. Owen
General Manager
De Beers Consolidated Mines (Pty) Ltd
Premier Mine

APPENDIX I
M-DPR-04.7

**DE BEERS CONSOLIDATED MINES LIMITED - PREMIER MINE**

# MEMORANDUM

TO : MINE OVERSEER SECTION ....................................

FROM : SUPPORT COMMITTEE

SUBJECT : **APPROVED CHANGE TO SUPPORT SPECIFICATION AS DEFINED IN THE CODE OF PRACTICE**

DATE : ........................................................................

| AREA WHERE SUPPORT MODIFICATION IS TO BE IMPLEMENTED: | |
|---|---|
| ORIGINAL SUPPORT SPECIFICATION | MODIFICATION OF SUPPORT SPECIFICATION |
| | |
| REASON FOR CHANGE IN SPECIFICATION | |
| COST OF ORIGINAL SUPPORT SPECIFICATION<br>............. per linear metre | COST OF MODIFIED SUPPORT SPECIFICATION<br>............. per linear metre |

**SIGNATURES OF SUPPORT COMMITTEE** ........................................................................
**MINING SUPERINTENDENT**

........................................................................
**RESIDENT GEOLOGIST**

**REFERENCE NO:** ........................................................................

*A COPY OF THIS DOCUMENT TO BE SENT TO THE PLANNING DEPARTMENT*

*PLEASE NOTE THAT THIS DOCUMENT MUST BE FILED ALONG WITH YOUR SUPPORT CODE OF PRACTICE*

Mesh reinforced shotcrete provides a support pressure of 1 MPa for both rock types.

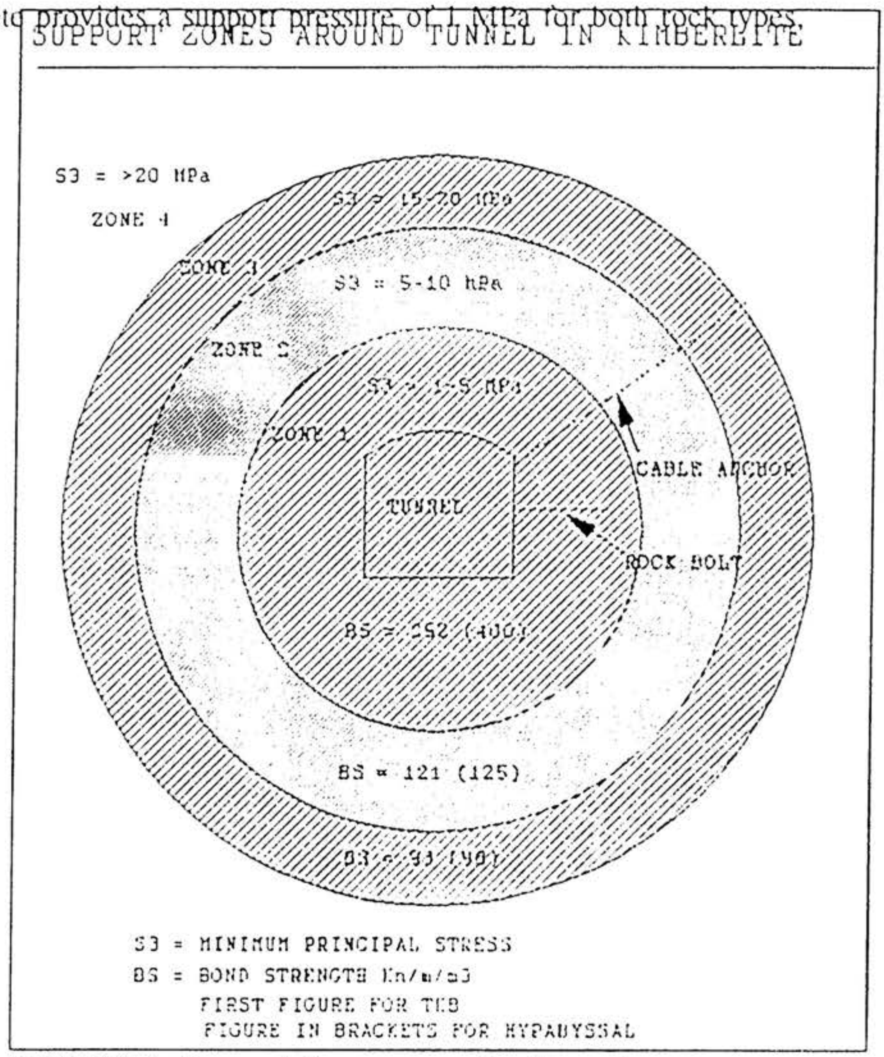



FIGURE 4 Support Zones around a Tunnel

**Zone 2** extends from the second to the third magnet and is supported with cable anchors on a 3 metre spacing radially and (2 metres along the length of the tunnel) and extends from 2,5 to 4 metres into the rock. Modelled confining stress at the centre of this zone is 10 - 15 MPa and tendon density is 121 kN/m$^2$ for TKB and 128 kN/m$^2$ for hypabyssal kimberlite.

**Zone 3** extends from the third to the fourth magnet at an average depth of 4 to 5,5 metres into the rock and is supported with cables on a 4 metre spacing radially. Modelled average confining stress for the zone is 15-20 MPa and tendon density 93 kN/m$^2$ for TKB and 98 kN/m$^2$ for hypabyssal kimberlite.

**Zone 4** extended from the fourth magnet to the reference magnet at an average depth of 5,5 to 7 metres into the rock mass and is not supported. The modelled confining stress is 25 MPa and reinforcement 0.

The above calculations show that, as a result of the greater rock strength of the hypabyssal kimberlite the hypabyssal kimberlite is more effectively supported than the TKB. The calculation also suggests that support in zone 1 is twice as effective as in zone 2 and 3 for TKB and at least three times as effective for hypabyssal kimberlite. Even if the effect of confining stress at depth is taken into account tendon support in zone 1 remains more effective than in zone 2 and 3, on this simplified, comparative basis. The defined zones, minimum principal stresses calculated for these zone and bond strengths relative to the installed rockbolts and cable anchors are illustrated in Figure 4.

# CHAPTER 4

# SPECIAL AREAS

## 1. ROCKBURSTS AND SEISMICITY

The mine has never experienced any rockbursts or seismicity

## 2. SPECIAL SITUATIONS IN NORITE

### 2.1. TUNNELS IN BLOCKY NORITE NEAR CAVE AND AFFECTED BY WATER

Experience has taught that major tunnel stability problems can be expected in wet, blocky norite near a cave area. The area must be well drained and if possible mined in such a way that the area is subjected to a minimum of stress change. The density of development in the area both in the vertical and horizontal plane should be as low as possible i.e. the minimum number of excavations should be sited in such an area.

The rock should be supported with 1,8 metre long, fully grouted rockbolts down to footwall level during development. Six metre long, fully grouted, steel rope anchors on a 1 metre spacing should be installed down to footwall. Thereafter the excavation should be lined with at least 300 mm of concrete (minimum strength 30 Mpa after 28 days). The area must be monitored. If large displacements are still recorded the rock must be further supported with a pattern of tendon straps on a 1 metre spacing installed on 1,8 metre long rockbolts. The steel should be shotcreted closed.

### 2. SUPPORT IN PASSES SITED IN NORITE

The stability of the pass is very dependant on the competence of the norite in which the pass is sited.

1. In competent norite where no stress changes are expected no pass support is needed. This includes most air passes.

2. In competent norite near excavations which can lead to stress changes concrete lining is needed for 20 metres above and below intersecting excavations.

3. The blocky nature of the norite in some areas creates major pass stability problems. In such areas the most effective way of supporting new passes is to drill a raisebore of the minimum practical size (1,8 or 1,2 metre diameter) at the pass position. The pass is then sliped to the required dimension of 2, 3 or four metres, starting at the top of the pass. After each sliping round the pass must be lined with at least of 300 mm of cast concrete (minimum strength 35 MPa after 28 days).

DE BEERS CONSOLIDATED MINES LIMITED - PREMIER MINE

# NOTE FOR THE RECORD

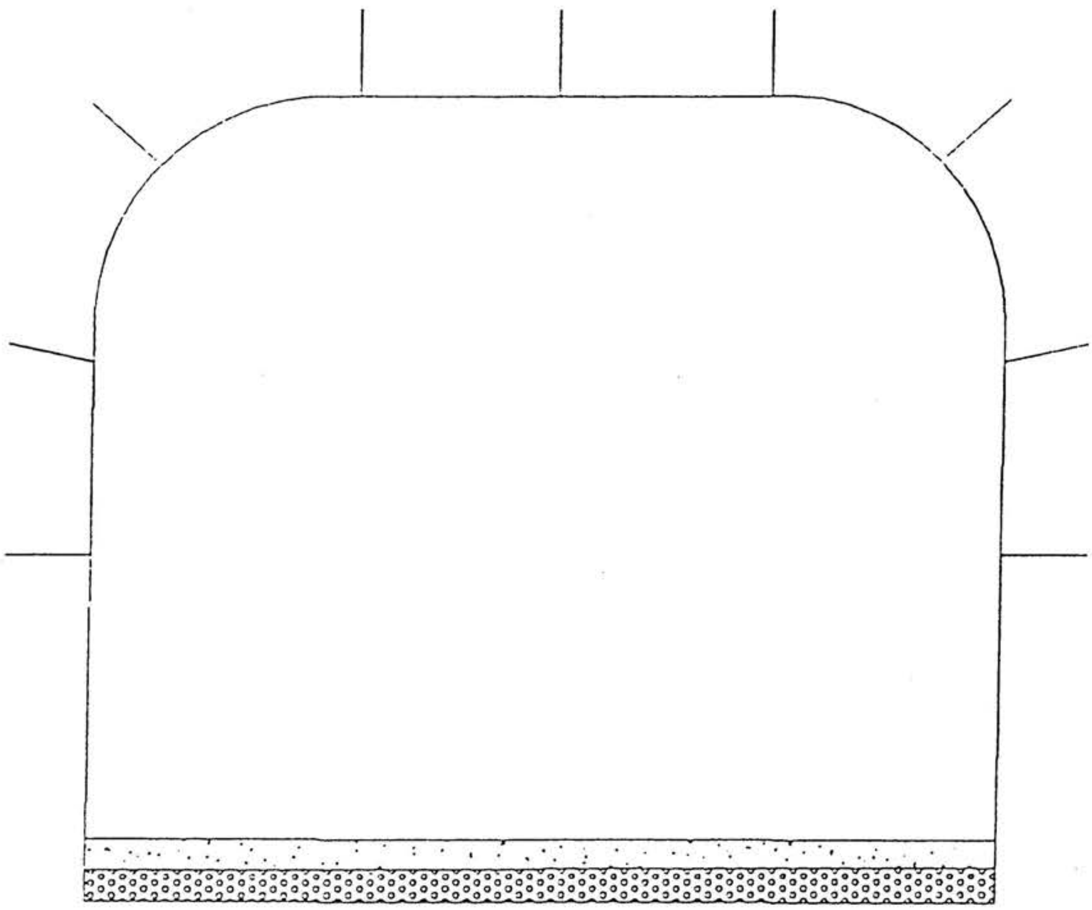
The following diagrams represent examples of guidelines for support type installation underground.

They are specifically for use by the underground tunnel support personnel for ease of installation.

It must be noted that all tunnel support types and/or specifications will be subject to review at any time by the Premier Mine Support Committee, and any changes documented.

GF/mdp/c.WT\note.sup

UNDER-CUT TUNNEL IN HYPABYSSAL
TYPE 1



LEGEND

COMPACT AGGREGATE

100mm CONCRETE

1.8m GEWIS

# DESCRIPTION OF TUNNEL SUPPORT TYPES USED AT PREMIER MINE

| SUPPORT TYPE | DESCRIPTION | COMMENTS |
|---|---|---|
| TYPE 1 | UNDERCUT TUNNELS IN HYPABYSSAL KIMBERLITE | |
| TYPE 2 | UNDERCUT TUNNELS IN TKB | |
| TYPE 3 | PRODUCTION TUNNELS IN HYPABYSSAL | |
| TYPE 4 | PRODUCTION DRAWPOINTS IN HYPABYSSAL: PRE-UNDERCUT AREAS | |
| TYPE 5 | PRODUCTION TUNNELS IN TKB | |
| TYPE 6 | PRODUCTION DRAWPOINTS IN TKB: PRE-UNDERCUT AREAS | |
| TYPE 7 | SUPPORT IN BLOCKY NORITE | |
| TYPE 8 | BLOCKY NORITE +6m ANCHORS/1m SPACING | |
| TYPE 9 | 500mm CONCRETE ARCHES | Superseded by Type 14 |
| TYPE 10 | BLOCKY NORITE+6m ANCHORS+CONCRETE ARCHES | No longer required |
| TYPE 11 | MESH+S/W SHOTCRETE+F/W CONCRETE | No longer required |
| TYPE 12 | MESH+S/W SHOTCRETE | |
| TYPE 13 | MESH+STRAPS+S/W SHOTCRETE+F/W CONCRETE | No longer required |
| TYPE 14 | 500mm CONCRETE ARCHES+ F/W CONCRETE | |
| TYPE 15 | 3m GEWIS+6m ANCHORS+MESH+S/W SHOTCRETE+F/W CONCRETE | |
| TYPE 16 | 1,8m GEWIS+F/W CONCRETE | |
| TYPE 17 | BB1E UNDERCUT TUNNEL IN TKB | |
| TYPE 18 | BB1E PRODUCTION DRAWPOINT [3,6mx3,5mx5m ARCHES] | |
| TYPE 19 | UNDERCUT TUNNELS IN TKB & HYPABYSSAL | |

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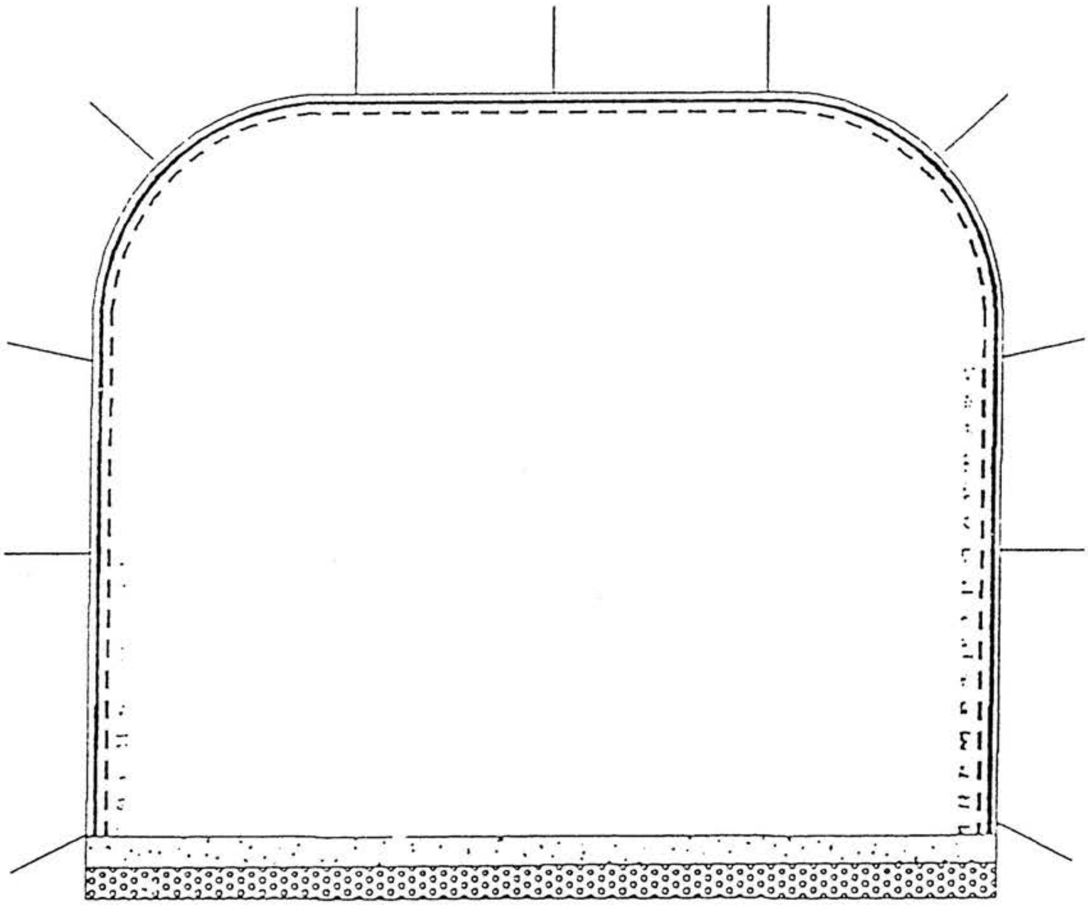
page:Summary
09-Apr-96

# UNDER-CUT TUNNEL IN TKB
# TYPE 2



## LEGEND

MESH

TENDON STRAPS

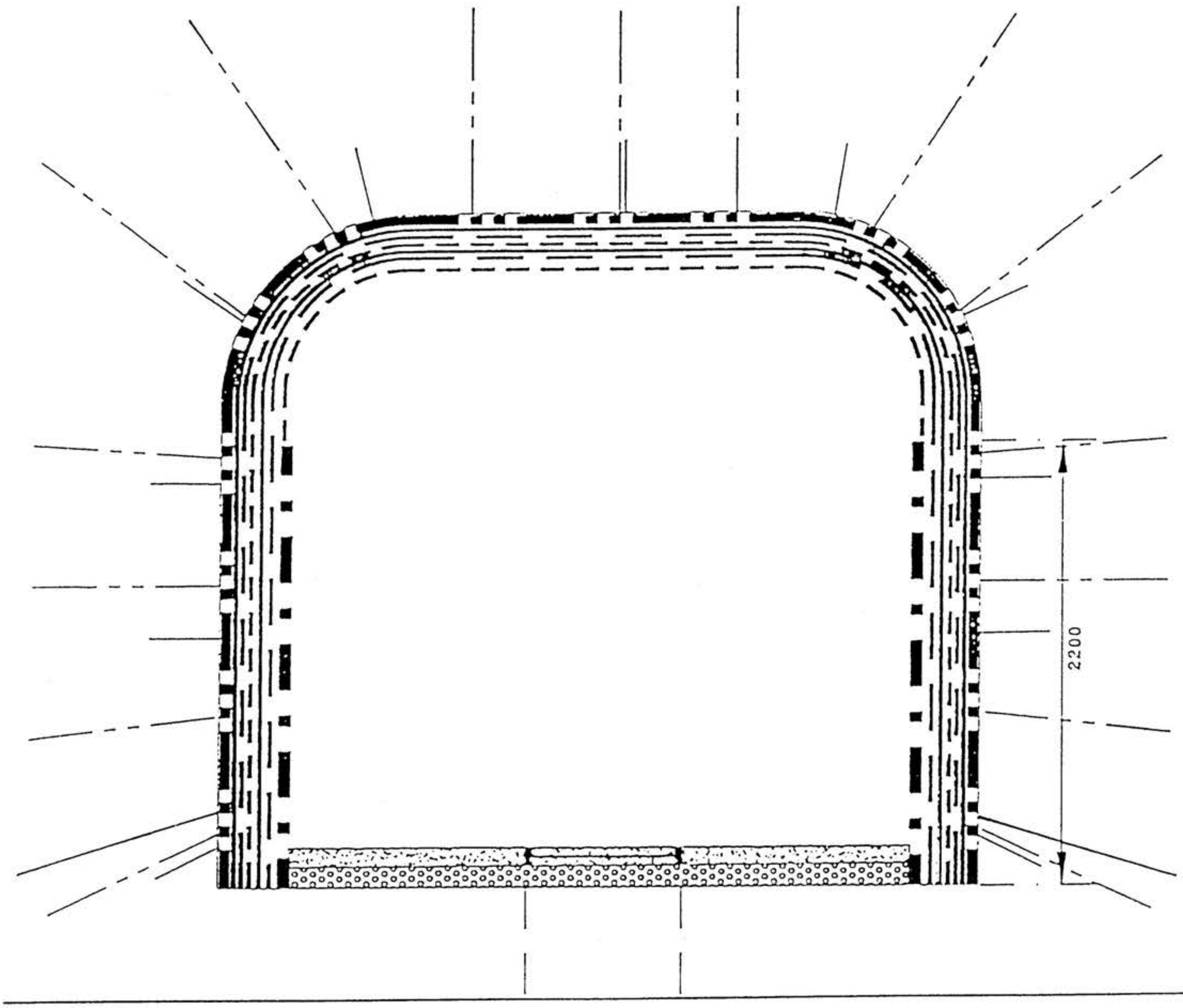
80mm GUNNITE

COMPACT AGGREGATE

100mm CONCRETE

1.8m GEWIS

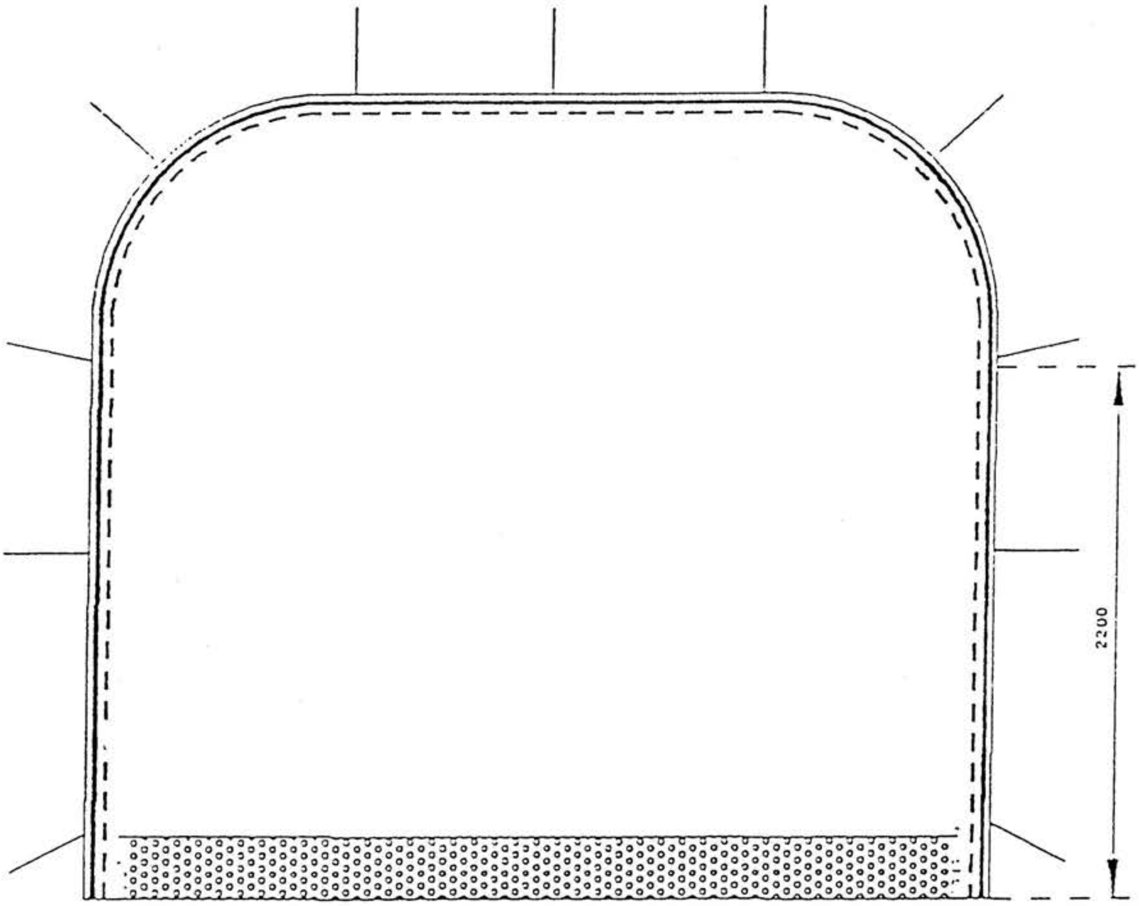
# PRODUCTION DRAWPOINT IN HYPABYSSAL
# TYPE 4




## LEGEND

| | |
|---|---|
| 20mm GUNNITE | 100mm TOPPING |
| MESH | 300mm CONCRETE |
| TENDON STRAPS | 1.8m GEWIS |
| 50mm GUNNITE | 3m ANCHORS |
| 120mm GUNNITE | 6.0m ANCHORS |

# PRODUCTION TUNNEL IN HYPABYSSAL
# TYPE 3




## LEGEND

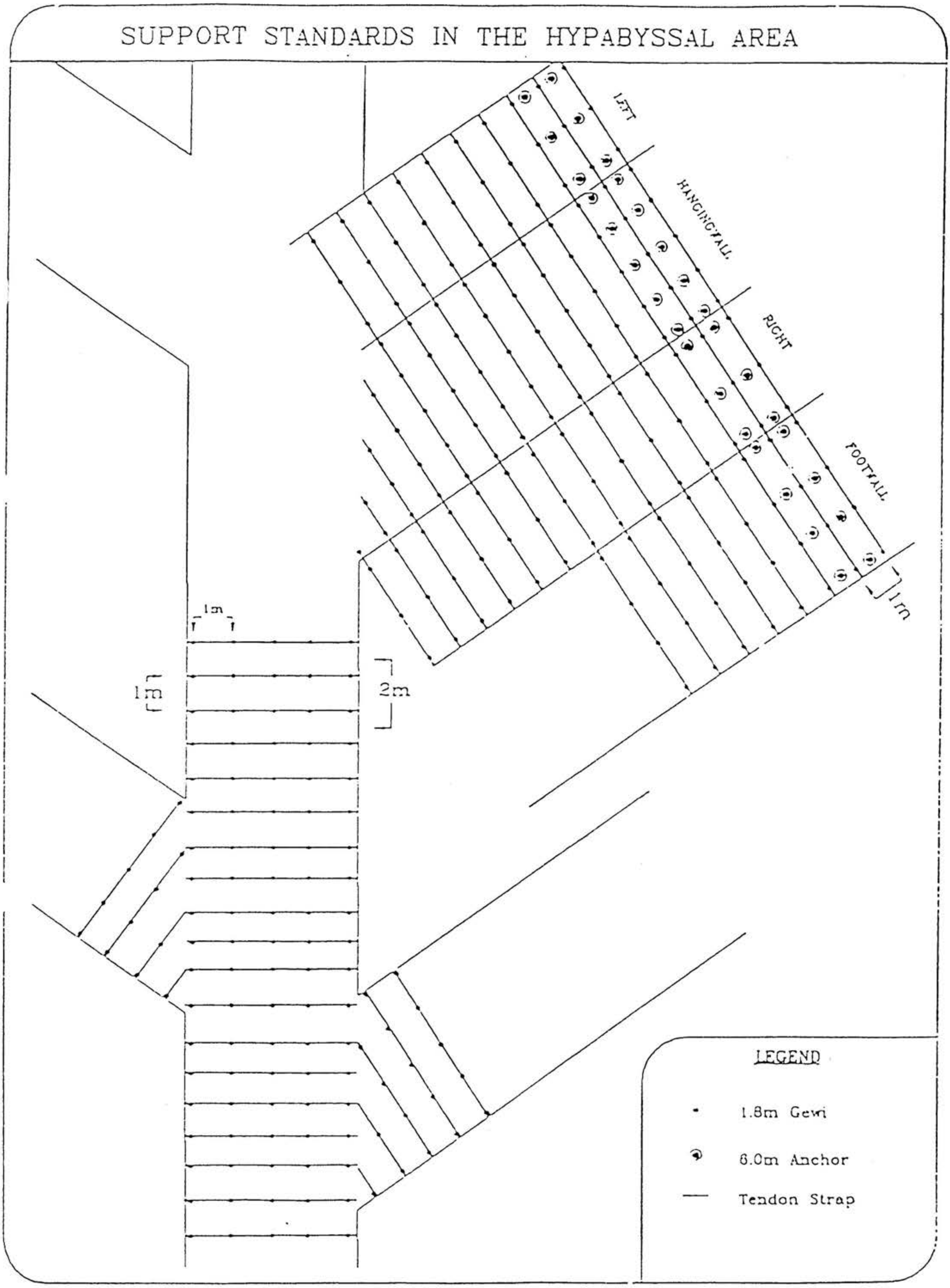
MESH

TENDON STRAPS

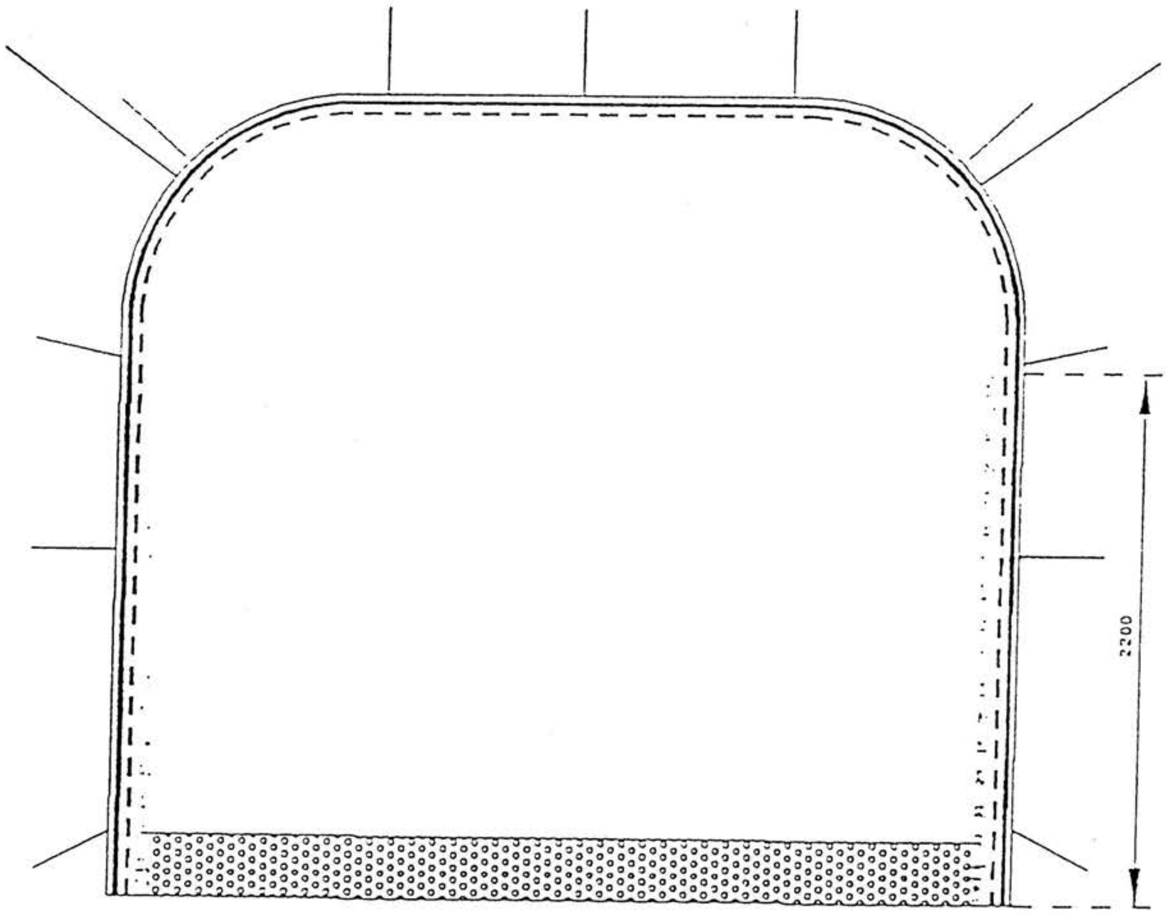
100mm GUNNITE

500mm CONCRETE

1.8m GEWIS


SUPPORT STANDARDS IN THE HYPABYSSAL AREA
LEFT
HANGINGWALL
RIGHT
FOOTWALL
1m
1m
1m
2m
LEGEND
1.8m Gewi
6.0m Anchor
Tendon Strap

# PRODUCTION TUNNEL IN TKB AREA
# TYPE 5




## LEGEND

MESH

TENDON STRAPS

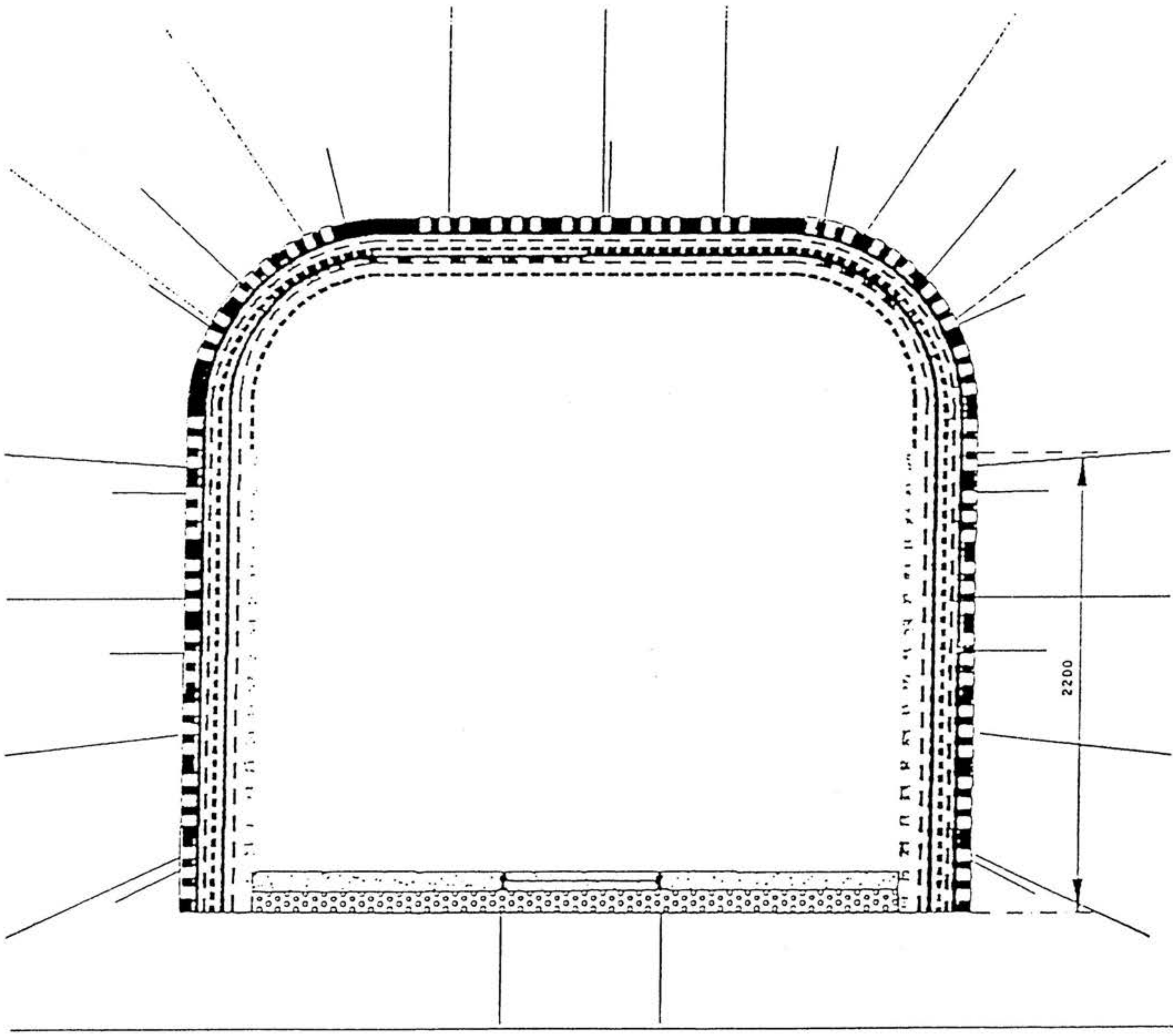
100mm GUNNITE

500mm CONCRETE

1.8m CEWIS

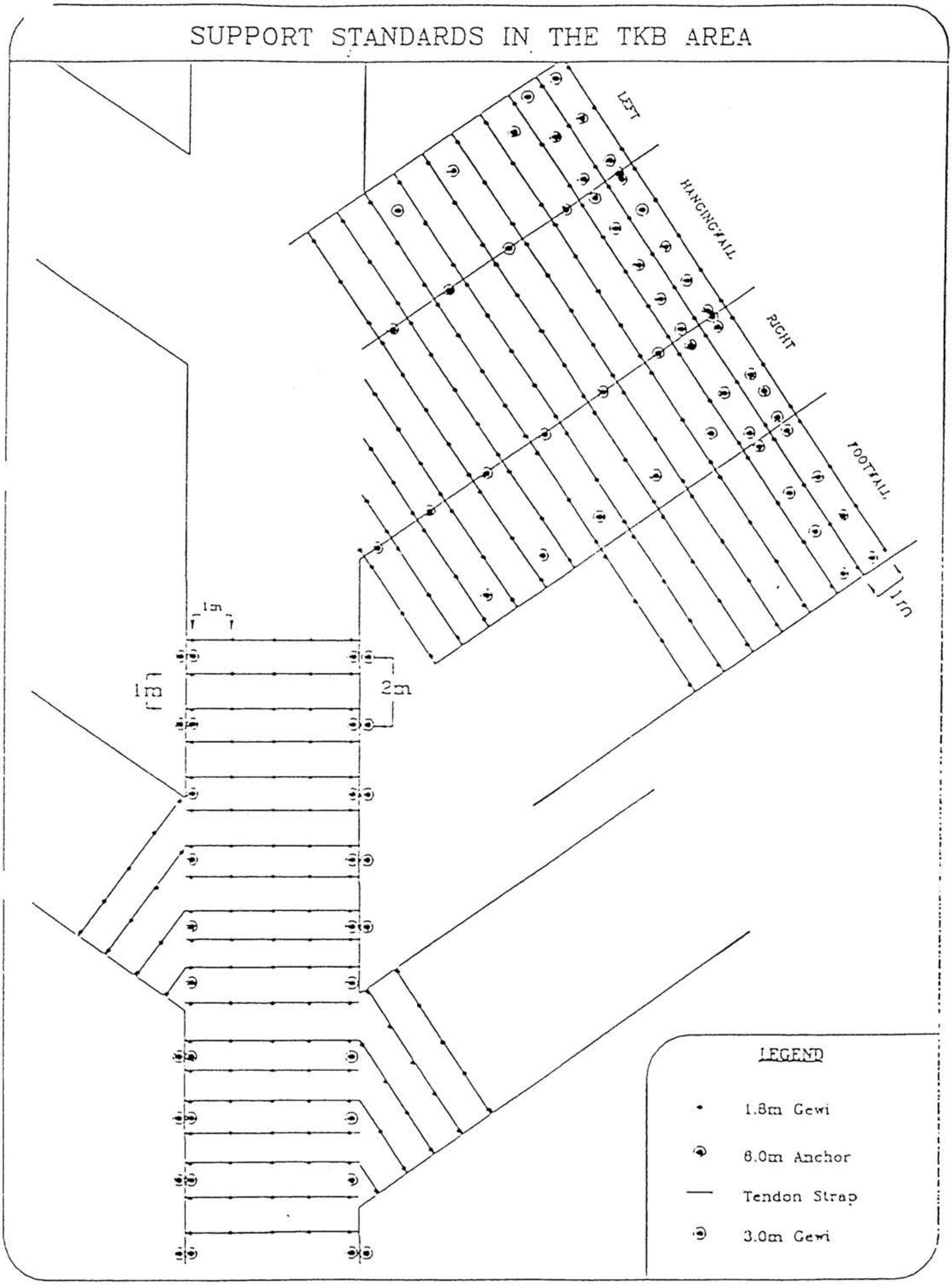
3.0m CEWI

# PRODUCTION DRAWPOINT IN TKB AREA
# TYPE 6

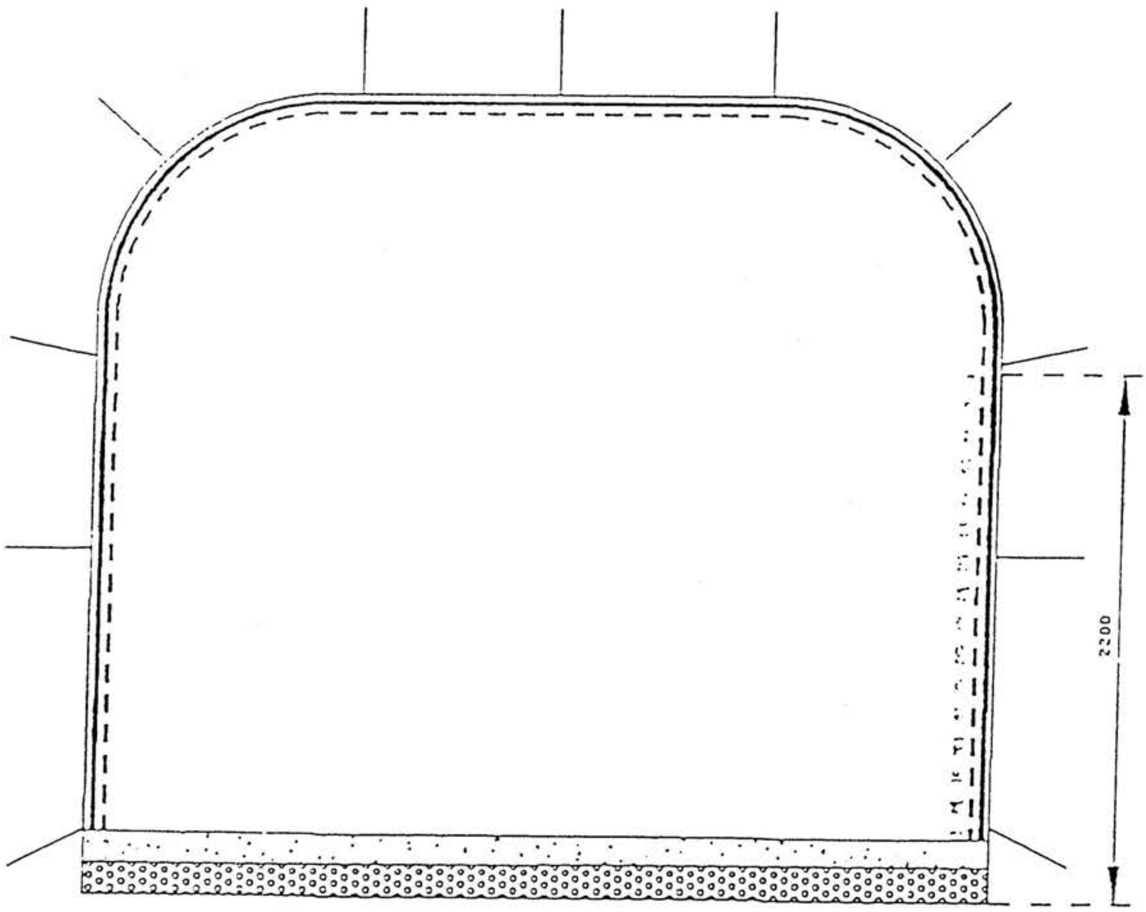



## LEGEND

- 20mm GUNNITE
- MESH
- TENDON STRAPS
- 50mm GUNNITE
- 120mm GUNNITE
- 100mm TOPPING
- 300mm CONCRETE
- 1.8m GEWIS
- 3m ANCHORS
- 6.0m ANCHORS


SUPPORT STANDARDS IN THE TKB AREA
LEFT
HANGINGWALL
RIGHT
FOOTWALL
1m
1m
2m
1m
LEGEND
1.8m Gewi
6.0m Anchor
Tendon Strap
3.0m Gewi

# TUNNEL IN BLOCKY NORITE
# TYPE 7




## LEGEND

MESH

TENDON STRAPS

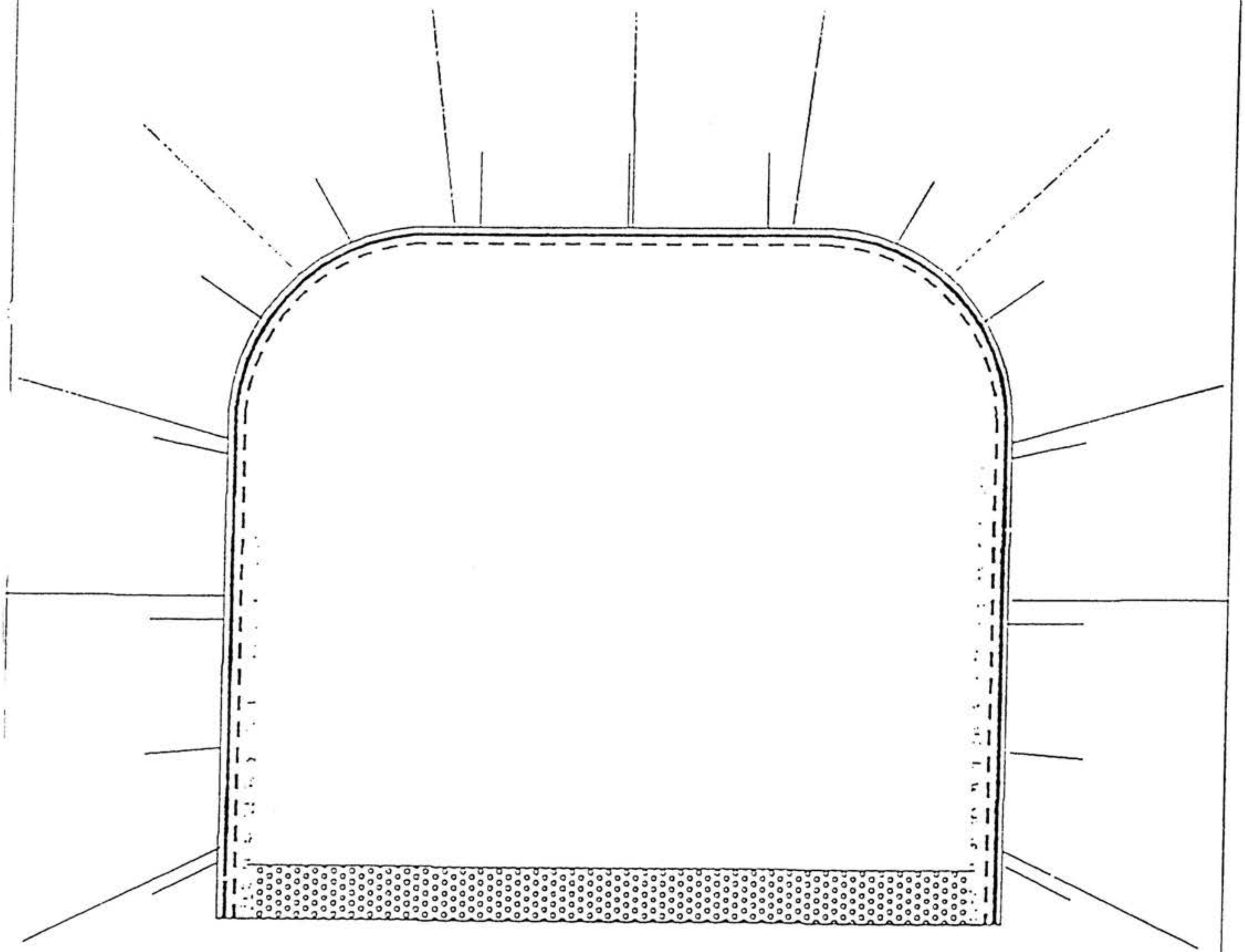
120mm GUNNITE

COMPACT AGGREGATE

150mm CONCRETE

1.8m GEWIS

# BLOCKY NORITE
# TYPE 8



## LEGEND

MESH

TENDON STRAPS

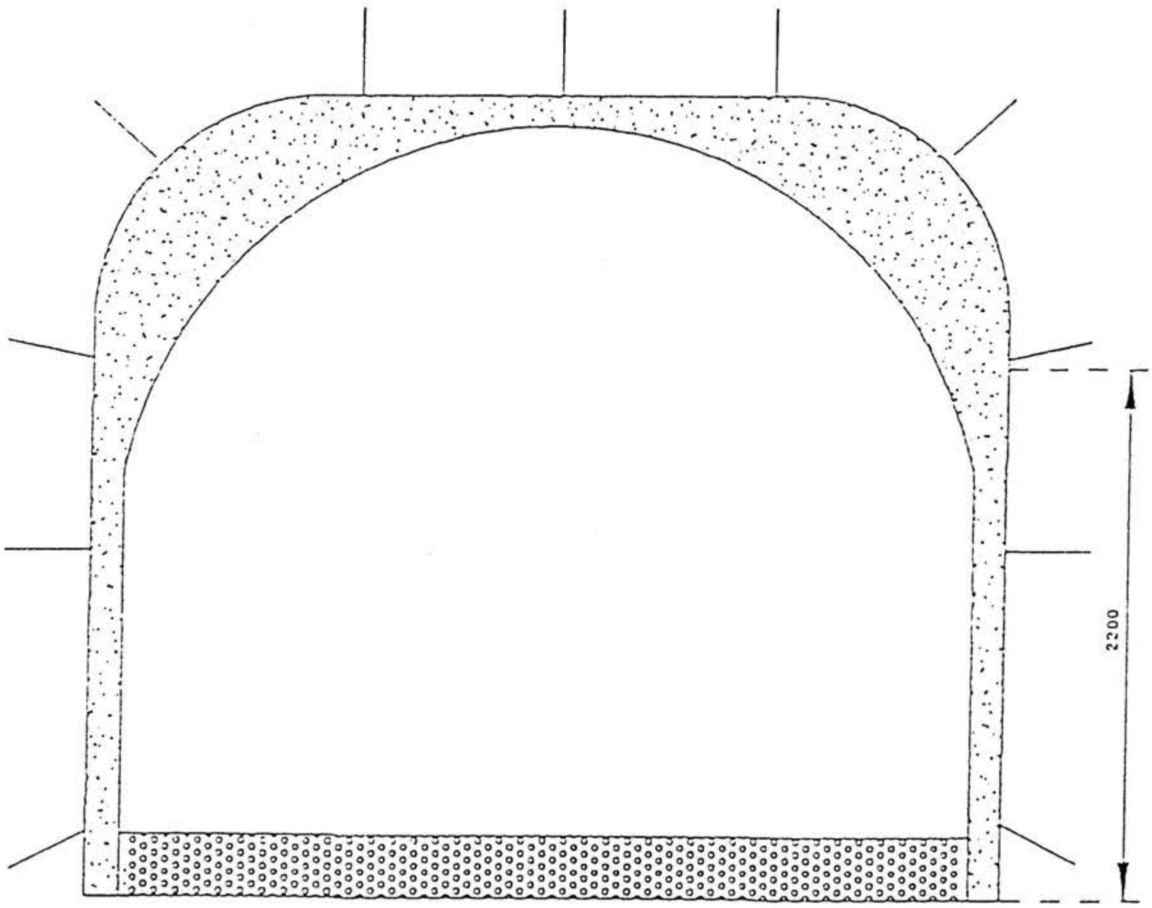
120mm GUNNITE

250mm CONCRETE

1.8m GEWIS

6.0m Anchors

# 500mm CONCRETE ARCH + F/W CONCRETE
# TYPE 14



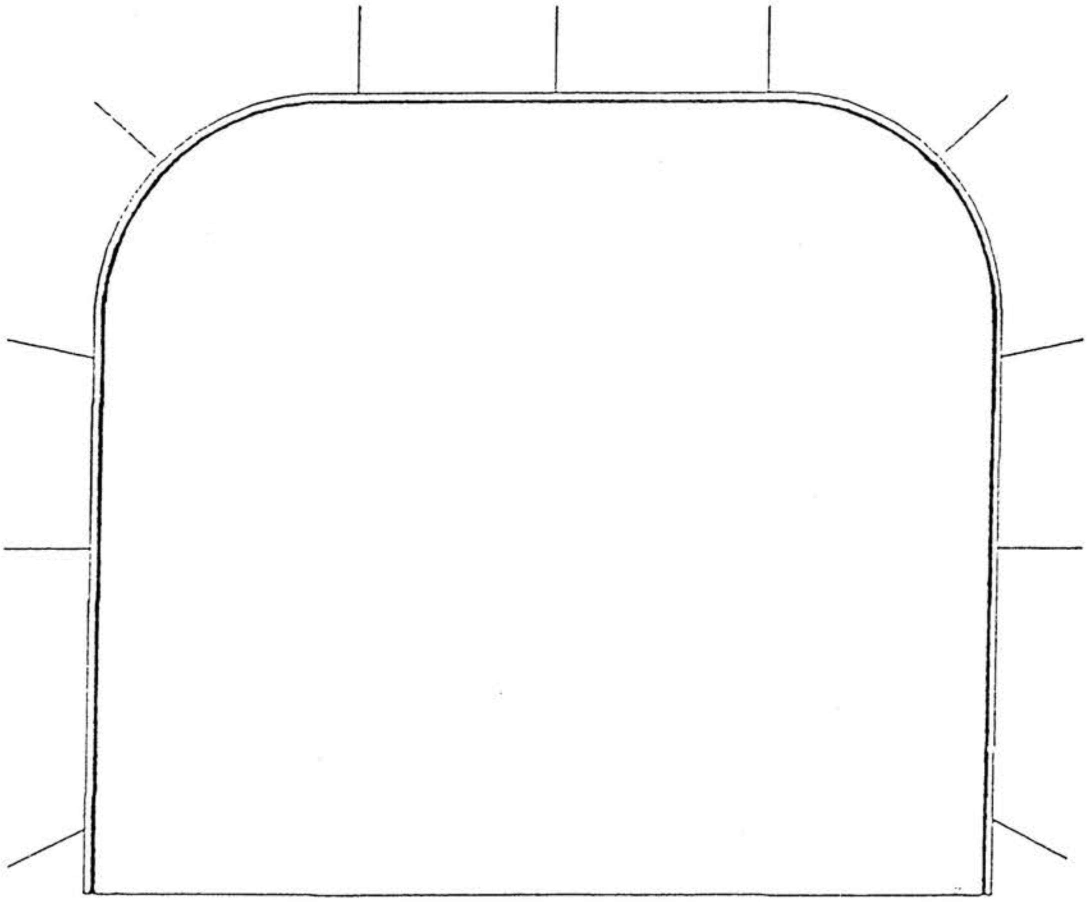

## LEGEND

300mm CONCRETE

0.5m CONCRETE ARCH

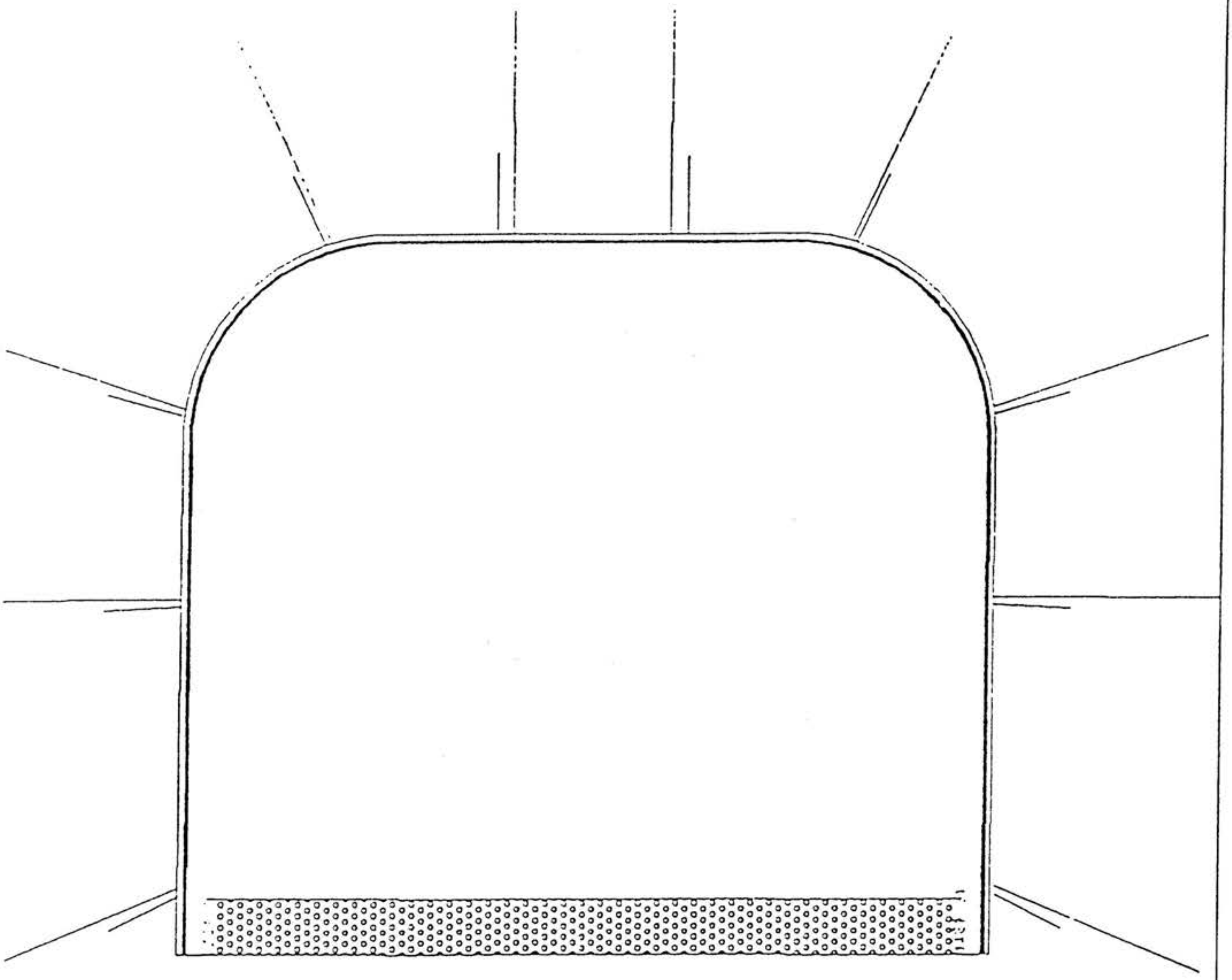
1.8m GEWIS

# MESH AND SIDE-WALL SHOTCRETE
# TYPE 12



## LEGEND

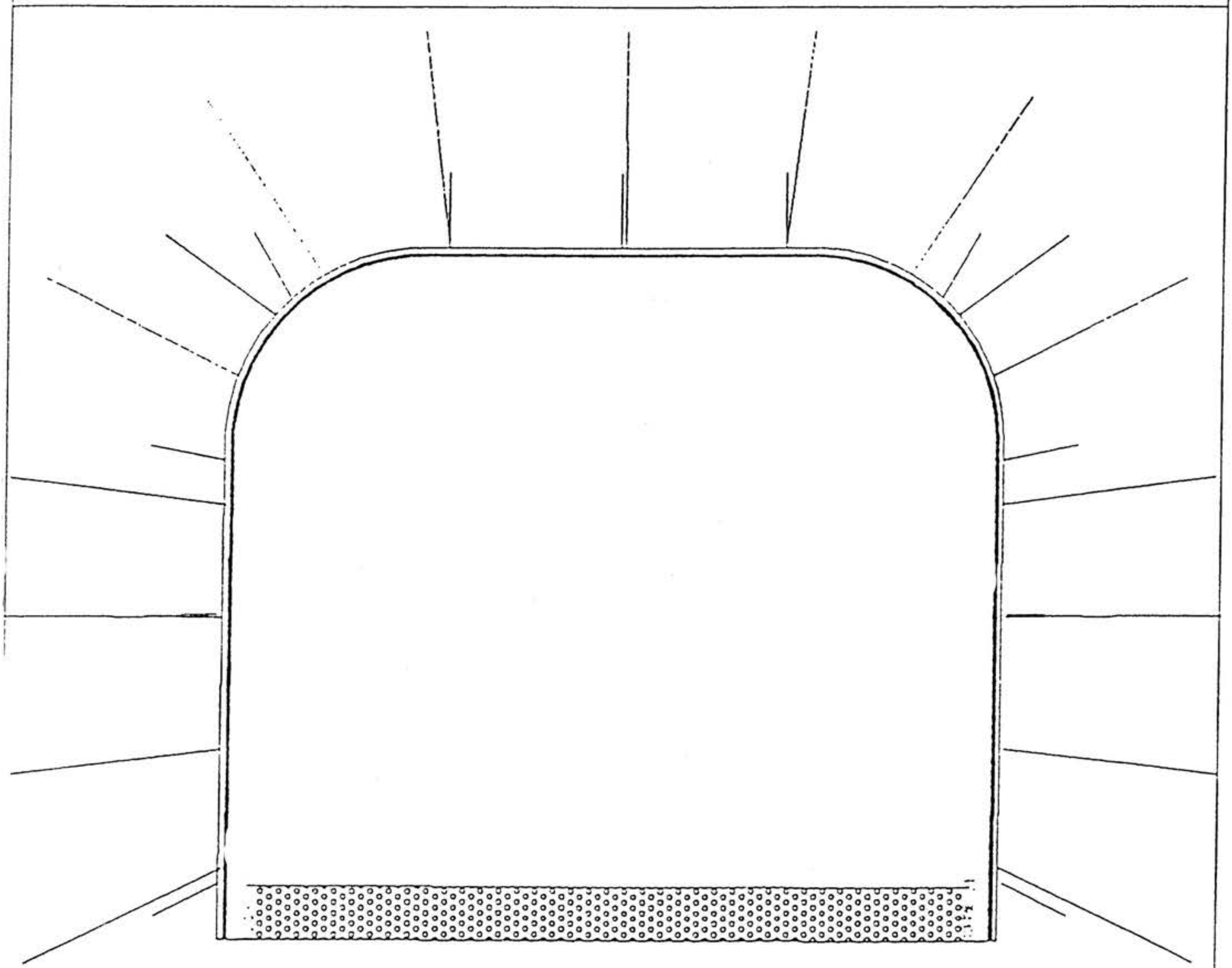
# BB1 E: UNDER-CUT LEVEL
# TYPE 17



## LEGEND

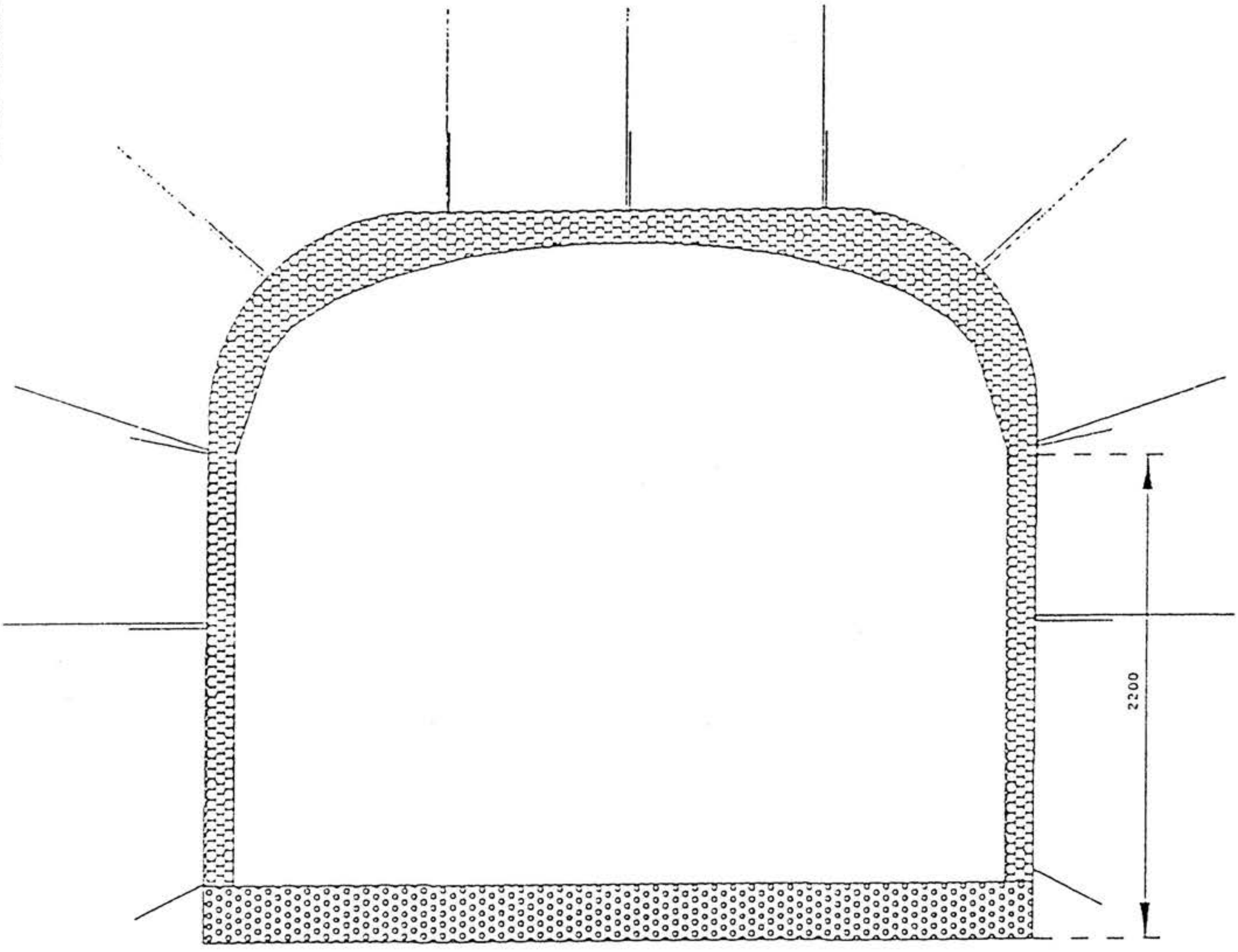
- MESH
- 80mm GUNNITE
- 500mm CONCRETE
- 1.8m GEWIS
- 6.0m Anchors

# TYPE 15



## LEGEND

# BB1E : PRODUCTION DRAWPOINT – 5m ARCH
# TYPE 18




## LEGEND

400mm CONCRETE ARCH

300mm CONCRETE

1.8m GEWIS

6m ANCHORS (2 ROWS)

TYPE 1 (UNDERCUT TUNNELS IN HYPABYSSAL KIMBERLITE)

| DESCRIPTION | AMOUNT | UNIT | UNIT COST | COST |
|---|---|---|---|---|
| 1.8m GEWIS(material) | 9.00 | each | 13.71 | 123.35 |
| RESIN CAPSULES(25x300) | 54.00 | each | 1.17 | 62.99 |
| FOOTWALL CONCRETE(100mm) | 0.90 | m ^ 3 | 216.00 | 194.40 |
| DRILLING CONSUMABLES(1.8m) | 9.00 | each | 8.21 | 73.87 |
| | | | TOTAL | 454.60 |

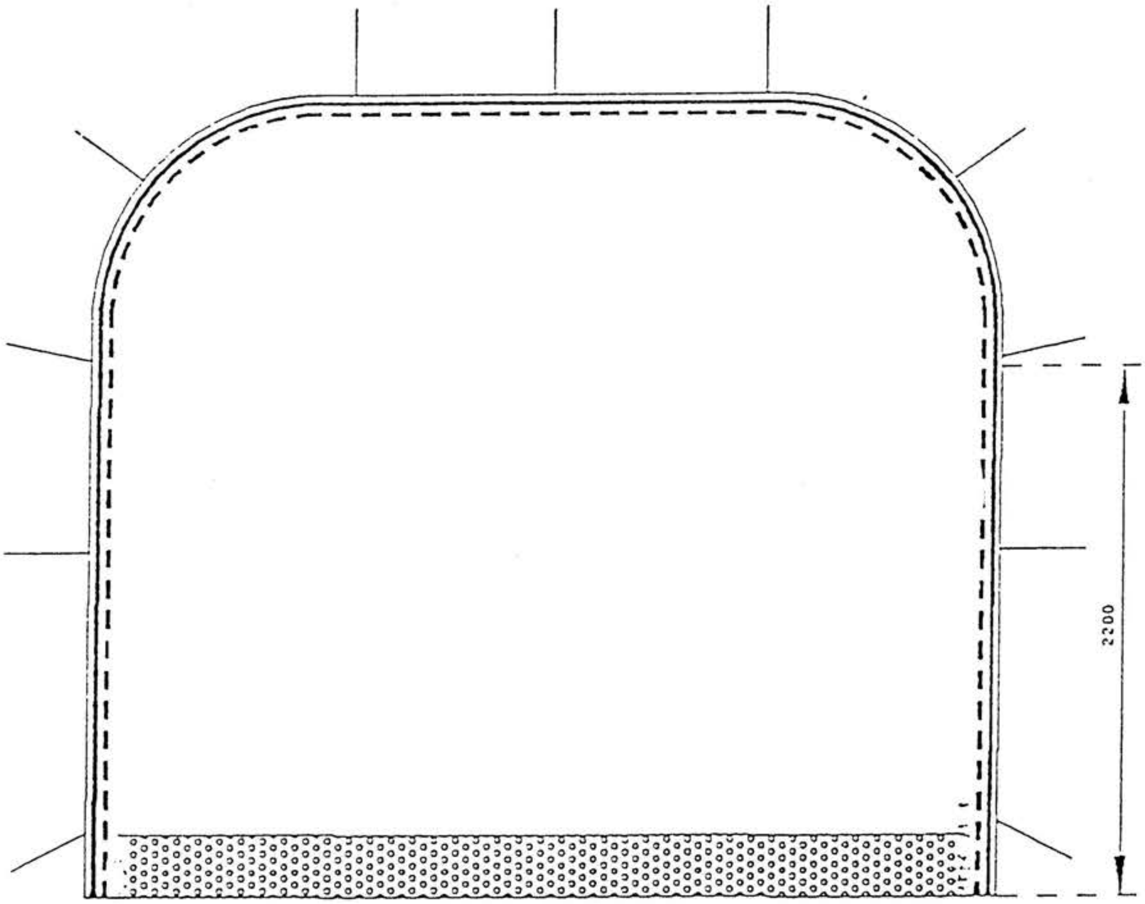
TYPE 2(UNDERCUT TUNNELS IN TKB)

| DESCRIPTION | AMOUNT | UNIT | UNIT COST | COST |
|---|---|---|---|---|
| SEALANT | 9.00 | litre | 5.42 | 48.79 |
| 1.8m GEWIS(material) | 9.00 | each | 13.71 | 123.35 |
| RESIN CAPSULES(25x300) | 54.00 | each | 1.17 | 62.99 |
| WIRE MESH | 9.00 | m ^ 2 | 4.69 | 42.20 |
| TENDON STRAPS | 3.00 | each | 48.06 | 144.18 |
| GUNNITE 80mm | 4.40 | m ^ 2 | 53.05 | 233.43 |
| FOOTWALL CONCRETE(100mm) | 0.30 | m ^ 2 | 216.00 | 64.80 |
| DRILLING CONSUMABLES(1.8m) | 9.00 | each | 8.21 | 73.87 |
| SHOTCRETE CONSUMABLES | 4.40 | m ^ 2 | 5.40 | 23.76 |
| | | | TOTAL | 817.37 |

TYPE 3(PRODUCTION TUNNELS IN HYPABYSSAL)

| DESCRIPTION | AMOUNT | UNIT | UNIT COST | COST |
|---|---|---|---|---|
| 1.8m GEWIS(material) | 11.00 | each | 13.71 | 150.76 |
| DRILLING CONSUMABLES(1.8m) | 11.00 | each | 8.21 | 90.29 |
| RESIN CAPSULES(25x300) | 66.00 | each | 1.17 | 76.98 |
| WIRE MESH | 10.60 | m ^ 2 | 4.69 | 49.70 |
| TENDON STRAPS | 3.00 | each | 48.06 | 144.18 |
| GUNNITE 100mm | 4.40 | m ^ 2 | 66.32 | 291.79 |
| SHOTCRETE CONSUMABLES | 4.40 | m ^ 2 | [illegible] | 23.75 |
| FOOTWALL CONCRETE(500mm) | 1.80 | m ^ 3 | 216.00 | 388.80 |
| | | | TOTAL | 1 216.26 |

# BB1E : PRODUCTION DRAWPOINT – ENTRANCE TYPE18




## LEGEND

MESH

50mm GUNNITE

80mm GUNNITE

300mm CONCRETE

1.8m CEWIS

TYPE 4(PRODUCTION DRAWPOINTS IN HYPABYSSAL)

| DESCRIPTION | AMOUNT | UNIT | UNIT COST | COST |
|---|---|---|---|---|
| 1.8m GEWIS(material) | 143.00 | each | 13.71 | 1 959.84 |
| DRILLING CONSUMABLES(1.8m) | 143.00 | each | 8.21 | 1 173.74 |
| RESIN CAPSULES(25x300) | 858.00 | each | 1.17 | 1 000.77 |
| GUNNITE 20mm | 132.00 | m^2 | 13.26 | 1 750.75 |
| SHOTCRETE CONSUMABLES | 132.00 | m^2 | 5.40 | 712.80 |
| WIRE MESH | 132.00 | m^2 | 4.69 | 618.90 |
| TENDON STRAPS | 52.00 | each | 48.06 | 2 499.12 |
| GUNNITE 50mm | 132.00 | m^2 | 33.16 | 4 376.88 |
| SHOTCRETE CONSUMABLES | 132.00 | m^2 | 5.40 | 712.80 |
| WIRE MESH | 55.00 | m^2 | 4.69 | 257.88 |
| TENDON STRAPS | 34.00 | each | 48.06 | 1 634.04 |
| ROOFBOLT NUTS | 66.00 | each | 2.48 | 163.94 |
| ROOFBOLT WASHERS | 66.00 | each | 2.41 | 158.95 |
| GUNNITE 120mm | 52.80 | m^2 | 79.58 | 4 201.80 |
| SHOTCRETE CONSUMABLES | 52.80 | m^2 | 5.40 | 285.12 |
| GUNNITE 50mm | 31.00 | m^2 | 33.16 | 1 027.90 |
| SHOTCRETE CONSUMABLES | 31.00 | m^2 | 5.40 | 167.40 |
| FOOTWALL CONCRETE (300mm) | 12.96 | m^3 | 216.00 | 2 799.36 |
| RAILS | 24.00 | metre | 89.82 | 2 155.68 |
| ANGLE IRON(60mmx60mmx10mm) | 24.00 | metre | 12.07 | 289.79 |
| 3m GEWIS(material) | 12.00 | each | 15.21 | 182.48 |
| DRILLING CONSUMABLES(3m) | 12.00 | each | 10.80 | 129.60 |
| KIMBACRETE | 6.00 | bag | 27.03 | 162.19 |
| 6m ANCHORS(material) | 30.00 | each | 58.11 | 1 743.44 |
| KIMBACRETE | 30.00 | bag | 27.03 | 810.97 |
| DRILLING CONSUMABLES(6m) | 30.00 | each | 14.91 | 447.44 |
| BULLNOSE | 1.00 | each | 1 244.11 | 1 244.11 |
| CAMELBACK | 1.00 | each | 1 244.11 | 1 244.11 |
| | | | TOTAL | 33 911.81 |

TYPE 5(PRODUCTION TUNNELS IN TKB)

| DESCRIPTION | AMOUNT | UNIT | UNIT COST | COST |
|---|---|---|---|---|
| 1.8m GEWIS(material) | 11.00 | each | 13.71 | 150.76 |
| DRILLING CONSUMABLES(1.8m) | 11.00 | each | 8.21 | 90.29 |
| RESIN CAPSULES(25x300) | 66.00 | each | 1.17 | 76.98 |
| SEALANT | 12.00 | litre | 5.42 | 65.06 |
| WIRE MESH | 10.60 | m^2 | 4.69 | 49.70 |
| TENDON STRAPS | 3.00 | each | 48.06 | 144.18 |
| GUNNITE 100mm | 4.40 | m^2 | 66.32 | 291.79 |
| SHOTCRETE CONSUMABLES | 4.40 | m^2 | 5.40 | 23.76 |
| 3.0m GEWIS(material) | 2.00 | each | 15.21 | 30.41 |
| DRILLING CONSUMABLES(3m) | 2.00 | each | 10.80 | 21.60 |
| RESIN CAPSULES(25x300) | 18.00 | each | 1.17 | 21.00 |
| FOOTWALL CONCRETE (500mm) | 1.80 | m^3 | 216.00 | 388.80 |
| | | | TOTAL | 1 354.33 |

TYPE 6(PRODUCTION DRAWPOINTS IN TKB)

| DESCRIPTION | AMOUNT | UNIT | UNIT COST | COST |
|---|---|---|---|---|
| 1.8m GEWIS(material) | 143.00 | each | 13.71 | 1 959.84 |
| DRILLING CONSUMABLES(1.8m) | 143.00 | each | 8.21 | 1 173.74 |
| RESIN CAPSULES(25x300) | 858.00 | each | 1.17 | 1 000.77 |
| SEALANT | 143.00 | litre | 5.42 | 775.29 |
| GUNNITE 20mm | 132.00 | m^2 | 13.26 | 1 750.75 |
| SHOTCRETE CONSUMABLES | 132.00 | m^2 | 5.40 | 712.80 |
| WIRE MESH | 132.00 | m^2 | 4.69 | 618.90 |
| TENDON STRAPS | 52.00 | each | 48.06 | 2 499.12 |
| GUNNITE 50mm | 132.00 | m^2 | 33.16 | 4 376.88 |
| SHOTCRETE CONSUMABLES | 132.00 | m^2 | 5.40 | 712.80 |
| WIRE MESH | 55.00 | m^2 | 4.69 | 257.88 |
| TENDON STRAPS | 34.00 | each | 48.06 | 1 634.04 |
| ROOFBOLT NUTS | 66.00 | each | 2.48 | 163.94 |
| ROOFBOLT WASHERS | 66.00 | each | 2.41 | 158.95 |
| GUNNITE 120mm | 52.80 | m^2 | 79.58 | 4 201.80 |
| SHOTCRETE CONSUMABLES | 52.80 | m^2 | 5.40 | 285.12 |
| 3,0m GEWIS(material) | 24.00 | each | 15.21 | 364.95 |
| DRILLING CONSUMABLES(3m) | 24.00 | each | 10.80 | 259.20 |
| RESIN CAPSULES(25x300) | 216.00 | each | 1.17 | 251.94 |
| GUNNITE 50mm | 31.00 | m^2 | 33.16 | 1 027.90 |
| SHOTCRETE CONSUMABLES | 31.00 | m^2 | 5.40 | 167.40 |
| FOOTWALL CONCRETE (300mm) | 12.96 | m^3 | 216.00 | 2 799.36 |
| RAILS | 24.00 | metre | 89.82 | 2 155.68 |
| ANGLE IRON(60mmx60mmx10mm) | 24.00 | metre | 12.07 | 289.79 |
| 3m GEWIS(material) | 12.00 | each | 15.21 | 182.48 |
| DRILLING CONSUMABLES(3m) | 12.00 | each | 10.80 | 129.60 |
| KIMBACRETE | 6.00 | bag | 27.03 | 162.19 |
| 6m ANCHORS(material) | 30.00 | each | 58.11 | 1 743.44 |
| DRILLING CONSUMABLES(6m) | 30.00 | each | 14.91 | 447.44 |
| KIMBACRETE | 30.00 | bag | 27.03 | 810.97 |
| BULLNOSE | 1.00 | each | 1 244.11 | 1 244.11 |
| CAMELBACK | 1.00 | each | 1 244.11 | 1 244.11 |
| | | | TOTAL | 35 563.20 |

TYPE 7(SUPPORT IN BLOCKY NORITE)

| DESCRIPTION | AMOUNT | UNIT | UNIT COST | COST |
|---|---|---|---|---|
| 1,8m GEWIS(material) | 11.00 | each | 13.71 | 150.76 |
| DRILLING CONSUMABLES(1.8m) | 11.00 | each | 8.21 | 90.29 |
| RESIN CAPSULES(25x300) | 66.00 | each | 1.17 | 76.98 |
| WIRE MESH | 11.00 | m^2 | 4.69 | 51.58 |
| TENDON STRAPS | 3.00 | each | 48.06 | 144.18 |
| GUNNITE 120mm | 4.40 | m^2 | 79.58 | 350.15 |
| SHOTCRETE CONSUMABLES | 4.40 | m^2 | 5.40 | 23.76 |
| FOOTWALL CONCRETE (150mm) | 0.54 | m^3 | 216.00 | 116.64 |
| | | | TOTAL | 1 004.33 |

TYPE 17(BB1E: UNDERCUT LEVEL)

| DESCRIPTION | AMOUNT | UNIT | UNIT COST | COST |
|---|---|---|---|---|
| 1.8m GEWIS(material) | 10.00 | each | 13.71 | 137.05 |
| DRILLING CONSUMABLES(1.8m) | 10.00 | each | 8.21 | 82.08 |
| RESIN CAPSULES(25x300) | 60.00 | each | 1.17 | 69.98 |
| WIRE MESH | 9.00 | m ^ 2 | 4.69 | 42.20 |
| GUNNITE 80mm | 9.00 | m ^ 2 | 53.05 | 477.48 |
| SHOTCRETE CONSUMABLES | 9.00 | m ^ 2 | 5.40 | 48.60 |
| 6m ANCHORS(material) | 10.00 | each | 58.11 | 581.15 |
| DRILLING CONSUMABLES(6m) | 10.00 | each | 14.91 | 149.15 |
| KIMBACRETE | 10.00 | bag | 27.03 | 270.32 |
| FOOTWALL CONCRETE(500mm) | 0.30 | m ^ 3 | 216.00 | 64.80 |
| | | | TOTAL | 1 922.81 |

TYPE 18(BB1E: PRODUCTION DRAWPOINT [3,6mx3,5mx5m ARCHES])

| DESCRIPTION | AMOUNT | UNIT | UNIT COST | COST |
|---|---|---|---|---|
| 1.8m GEWIS(material) | 143.00 | each | 13.71 | 1959.84 |
| DRILLING CONSUMABLES(1.8m) | 143.00 | each | 8.21 | 1173.74 |
| RESIN CAPSULES(25x300) | 858.00 | each | 1.17 | 1000.77 |
| WIRE MESH | 74.20 | m ^ 2 | 4.69 | 347.90 |
| GUNNITE 50mm | 43.40 | m ^ 2 | 33.16 | 1 439.06 |
| SHOTCRETE CONSUMABLES | 43.40 | m ^ 2 | 5.40 | 234.36 |
| GUNNITE 80mm | 30.80 | m ^ 2 | 53.05 | 1 634.03 |
| SHOTCRETE CONSUMABLES | 30.80 | m ^ 2 | 5.40 | 166.32 |
| RAILS | 24.00 | metre | 89.82 | 2 155.68 |
| ANGLE IRON(60mmx60mmx10mm) | 24.00 | metre | 12.07 | 289.79 |
| 3m GEWIS(material) | 12.00 | each | 15.21 | 182.48 |
| DRILLING CONSUMABLES(3m) | 12.00 | each | 10.80 | 129.60 |
| KIMBACRETE | 6.00 | bag | 27.03 | 162.19 |
| 6m ANCHORS(material) | 18.00 | each | 58.11 | 1 046.07 |
| KIMBACRETE | 18.00 | bag | 27.03 | 486.58 |
| DRILLING CONSUMABLES(6m) | 18.00 | each | 14.91 | 268.47 |
| CONCRETE ARCH(400mm) | 24.60 | m ^ 3 | 216.00 | 5 313.60 |
| BULLNOSE | 1.00 | each | 1 244.11 | 1 244.11 |
| BULNOSE WALL | 6.30 | m ^ 3 | 216.00 | 1 360.80 |
| FO[illegible]WALL CONCRETE(300mm) | 12.96 | m ^ 3 | 216.00 | 2 799.36 |
| | | | TOTAL | 23 394.75 |

file name:SUPPORT

Ag.R 7/07.

| DESCRIPTION | AAC STOCKCODE | UNIT | AUG 1993 COST | BASE JAN 1994 |
|---|---|---|---|---|
| GUMPLANK | 15133 0385 32 01 | EACH | 13.51 | 14.59 |
| 1.8m GEWIS | 27345 1250 01 03 | EACH | 12.69 | 13.71 |
| RESIN CAPSULES | 27349 1535 01 01 | EACH | 1.08 | 1.17 |
| WIREMESH(3mx15m) | 76121 3916 03 05 | ROLL | 162.80 | 175.82 |
| TENDON STRAPS(4m) | 27351 1456 01 00 | EACH | 44.50 | 48.06 |
| ROOFBOLT NUTS(20mm) | GRN | EACH | 2.30 | 2.48 |
| KIMBACRETE(BOX) | 71115 0091 67 04 | BOX | 25.03 | 27.03 |
| 6m ANCHOR(40t) | 27349 1231 01 09 | EACH | 72.02 | 77.78 |
| BARREL(40t) | 27349 1233 01 18 | EACH | 9.09 | 9.82 |
| WEDGE SET(40t) | 27349 1234 01 05 | EACH | 8.04 | 8.68 |
| DISTRIBUTION PLATE(40t) | 27349 1237 01 01 | EACH | 7.39 | 7.98 |
| 6m ANCHOR(25t) | 27349 1235 01 00 | EACH | 34.80 | 37.58 |
| BARREL(25t) | 27349 1238 01 07 | EACH | 4.77 | 5.15 |
| WEDGE SET(25t) | 27349 1239 01 02 | EACH | 5.77 | 6.23 |
| DISTRIBUTION PLATE(25t) | 27349 1232 01 04 | EACH | 8.47 | 9.15 |
| CALCIUMCHLORIDE | GRN | KG | 1.37 | 1.48 |
| FOOTWALL TOPPING | 71115 0072 68 08 | 25kg BAG | 6.70 | 7.24 |
| ANGLE IRON(60mmx60mmx10mm) | 11455 1033 32 00 | 6mLENGTH | 67.08 | 72.45 |
| 40KG RAIL(12m) | 38613 8022 32 08 | LENGTH | 998.00 | 1 077.84 |
| KIMBASEAL | 27349 1212 57 00 | 25L DRUM | 125.50 | 135.54 |
| 3m GEWIS | 27345 1255 01 00 | EACH | 14.08 | 15.21 |
| PRE-MIX | GRN | 25kg BAG | 5.95 | 6.43 |
| ROOFBOLT WASHERS(20mm) | 27345 1258 01 07 | EACH | 2.23 | 2.41 |
| CONCRETE | | M^3 | 200.00 | 216.00 |
| DUST SUPPRESSANT | | M^2 | 1.37 | 1.48 |
| DRILLING CONSUMABLES(1.2m) | | EACH | 6.00 | 6.48 |
| DRILLING CONSUMABLES(1.8m) | | EACH | 7.60 | 8.21 |
| DRILLING CONSUMABLES(3m) | | EACH | 10.00 | 10.80 |
| SHOTCRETE CONSUMABLES | | EACH | 5.00 | 5.40 |
| GUNNITE(20mm) | DENSITY OF | M^2 | 12.28 | 13.26 |
| GUNNITE(50mm) | PRE-MIX=2.15 | M^2 | 30.70 | 33.16 |
| GUNNITE(80mm) | REBOUND=20% | M^2 | 49.12 | 53.05 |
| GUNNITE(100mm) | | M^2 | 61.40 | 66.32 |
| GUNNITE(120mm) | | M^2 | 73.68 | 79.58 |
| GUNNITE(150mm) | | M^2 | 92.11 | 99.47 |
| FOOTWALL TOPPING | | M^2 | 26.80 | 28.94 |
| DRILLING CONSUMABLES(6m) | | EACH | 13.81 | 14.91 |
| DOUBLE LOCKING UNIT | 27349 1250 01 02 | EACH | 28.09 | 30.34 |
| CROSBY CLAMPS | 25813 3329 01 07 | EACH | 5.27 | 5.69 |
| BULLNOZE: | | EACH | 1 151.95 | 1 244.11 |
| 20*6m ANCHORS | 696.00 | | | |
| 5*KIMBACRETE | 125.15 | | | |
| 40*CROSBY CLAMPS | 210.80 | | | |
| 20*DRILLING CONSUM(1.2m) | 120.00 | | | |
| CAMELBACK: | | EACH | 1 151.95 | 1 244.11 |
| 20*6m ANCHORS | 696.00 | | | |
| 5*KIMBACRETE | 125.15 | | | |
| 40*CROSBY CLAMPS | 210.80 | | | |
| 20*DRILLING CONSUM(1.2m) | 120.00 | | | |

file name:SUPPORT ESCALATION: 8 % TO JAN-94 