

ROCK MECHANICS ASPECTS OF BLOCK CARVING CHRYSOTILE ASBESTOS OREBODIES  
AT SHABANIE MINE, RHODESIA

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ABSTRACT

Five sublevel caving and block caving operations were investigated at Shabanie Mine to establish ground behaviour characteristics, the mechanisms of caving, the conditions required for caving, and the effects of caving on the rock and development openings below a cave in the peripheries of a caved block. This work included the testing and development of a range of instruments to investigate and monitor ground displacement and deformation above the cave back, in the peripheries and below a caved block.

The chrysotile asbestos mined at Shabanie Mine occurs in numerous orebodies located in an ultramafic sill which has been folded and faulted. The fibre occurs as subparallel seams in bodies of partially serpentinised dunite occurring between talc-carbonate zones and carbonated serpentine zones which are developed along the faults. The orebodies are irregular in shape and vary in size and competency. The ore and surrounding rocks are cut by numerous, mostly steep dipping slips (minor faults).

Stress measurements to determine the pre-mining rock stresses were made at six sites at depths of between 200 and 400m, using photo-elastic biaxial strain gauges, and the CSIR triaxial cell was also used at one site. Some of the stresses measured were affected by previous mining as it was preferred to measure the stress in the same rock type that would be mined, rather than in the footwall, out of the influence of previous mining. The results indicated that the lateral stresses exceeded the vertical stresses by up to 50% at the depth of mining. These higher lateral stresses are probably residual tectonic stresses from the period of folding. Repeated measurements along the length of a borehole indicated large variations in stress magnitude and orientation attributed to the effect of variations in rock properties and proximity of structural features.

From observations of small displacements on joints in the peripheries of caved blocks it is concluded that the rock mass as a whole behaves as a quasi-Bingham substance, thereby giving any theoretical stress analysis a path dependency. For the present, the problems of adequate equations

to describe the rock behaviour, sufficiently accurate rock properties have to be solved before stress and displacement analyses be done using a finite element technique.

Two forms of caving were recognised, 'stress caving' and 'mass subsidence caving'. The former occurs when the undercutting results in the formation of a cavity. With sporadic sloughing from the back the caving progresses up to surface. In some cases horizontal tensional cracks have been observed or inferred above the back. The size of area required to be undercut to initiate caving and the rate at which caving proceeds, depends primarily on the competency of the rock in the back and the forces it is subject to. In the relatively high lateral stress-field it is expected that the immediate back is subject to moderate lateral compressive stresses with low or tensile stresses in the vertical direction. Over the central portion of the back the major principal stress is almost horizontal and here joints with low to moderate dips assist caving, while along the peripheries the stress direction changes and where suitably orientated steeper dipping slips may assist caving. Caving occurs as a relatively narrow zone of instability develops in the immediate back as a result of small shear displacements on joints and a form of bed separation occurs in the back. 'Mass subsidence caving' occurs where previous mining has reduced the lateral constraint, taking the form of an orderly subsidence of large columnar blocks of rock. Caving is rapid, requires a relatively small undercut area and results in a low bulking factor.

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Frontispiece: Surface subsidence over Block 6 (left), Block 7 AB (foreground), Blocks 7/1-7/3 (centre), Block 7/2 (right background) and Block 16 (top of hill).

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## CHAPTER ONE

### INTRODUCTION AND LITERATURE SURVEY

#### 1.1 INTRODUCTION

The mining engineer has a responsibility to his company shareholders, his country and to mankind in general not to squander that little portion of the earth's ever diminishing mineral resources entrusted to him. His responsibility is to get the maximum amount of mineral out of the deposit, in the most economic way, with the least risk to human life. In attempting to obtain the maximum ore recovery at the least cost, hidden risks may be taken in the planning stage, which could later result in trouble and the loss of ore, and which by a more cautious approach in planning could have resulted ultimately in a higher mineral recovery. It is only with highly developed rock mechanics techniques and expertise that these hidden risks can be recognised and evaluated.

In the mining of large massive orebodies, open pit mining is the most frequently used mining method because it is a low-cost high productivity method in which ore and waste can be kept separated with relative ease. But with increasing depth the cost of overburden stripping rises steeply, and so for the deeper orebodies, other mining methods become economically attractive. Further, there are also a number of orebodies where open pit mining has been precluded by climatic or topographic conditions. The alternative mining methods include open stoping, cut and fill, sublevel shrinkage stoping, top slicing, sublevel caving and block caving.

The choice of an alternative mining method to open pit mining depends primarily on the shape and size of the orebody, the competency of the ore and overburden, and the vertical and lateral rock stresses. The latter two determine the size of slope which can be mined with relative safety, whether the ore and/or hanging wall will cave satisfactorily. The choice of mining method also depends on the value of the ore and the cost of treating dilution. If the costs of hauling, hoisting, milling and treating the ore are high, a mining method which keeps the amount of dilution down to an acceptable level is required,

and these savings may offset the higher cost of a more selective mining method.

In highly fractured and jointed orebodies, top slicing or block caving may be the only choice, while in very competent ore, the only choice is open stoping or cut and fill mining. Most orebodies, however, fall in between these two extremes. Shabanie Mine is one of the few mines which has used several of these mining methods. Shabanie Mine started with open pit, went onto cut and fill stoping, then block caving, sublevel shrinkage and sublevel open stoping. This mine is one of the largest chrysotile asbestos mines in the southern hemisphere and is one of two principal mines of the African Associated Mines group. The other mine is Gath's Mine, which mines three distinct asbestos deposits several kilometres apart, and uses similar mining methods to Shabanie. The ground conditions encountered on these two mines varies from very poor which caves easily under suitable conditions to moderately competent, which given suitable conditions, will permit large open stopes to be mined.

It has been long recognised that the first block or panel in a block caving operation is the most difficult to cave. Unlike most block cave mines, Shabanie has numerous orebodies and this presents an unusual opportunity to study the mechanisms of caving in the initial operation.

Therefore when the writer was appointed to the Rock Mechanics Research Unit on Shabanie Mine he was given a unique opportunity to study caving methods of mining. The reasons for establishing this research unit were, of course, not altruistic. In some of the early attempts at block and sublevel caving at Shabanie mine the hangingwall failed to cave, while in other early block caving attempts, air blasts were experienced when sudden collapses of the hangingwall occurred. It was clearly evident that criteria for deciding whether an area would cave or not, or whether potential air blast conditions would develop was needed.

In addition to these two problems, a series of costly and extensive collapses occurred on both Gath's and Shabanie Mines in the early 1960's. These collapses illustrated the need for a deeper understanding of the causes, and a need for a method of predicting where such damage can be expected, so that precautionary measures could be taken. This was of particular importance on Shabanie where the next blocks to come

into production would be at depths 50 to 100% greater than had been experienced to date.

#### 1.2 OBJECTS AND JUSTIFICATION FOR THE INVESTIGATIONS

When the research was started, the next block scheduled to come into production on Shabanie Mine was selected for intensive investigation with the object outlined below. The objects of this work can be defined as:

- (a) To develop and test instruments and techniques for monitoring ground displacement and subsidence which would be best suited to the conditions pertaining on Shabanie Mine, and suited to being read by relatively unskilled observers. The purpose of these instruments was to determine the extent of a caved zone, to detect pre-caving ground deformation for prediction of caving, to record occurrence of and course of caving.
- (b) To investigate the mechanics of caving and the conditions required for caving. It was evident from the experience gained on Shabanie Mine and from published accounts of block caving operations that the size of the area to be undercut, the amount of boundary weakening required to initiate caving and the degree of fragmentation of the caved ore are related to the competency of the ore, the undercutting procedure, the geological structures in the back and the stresses in the back.
- (c) To develop and test instruments or techniques for monitoring the effects of stress changes on shafts, crosscuts, drives or other workings in the peripheries of, or below caved blocks. These instruments were needed to supplement visual observations in the analysis and interpretation of damage to these workings and it was hoped that some would be able to provide an early warning of damage so that precautions could be taken.
- (d) To investigate the mechanics and causes of damage to shafts, crosscuts, drives and other underground workings in both the peripheries of cave blocks and in the ground below. In common with many other mines practicing cave-mining methods, this mine had suffered a series of collapses of workings in the peripheries as well as below the cave blocks which affected production and were expensive to support. It was believed that, if the causes and mechanisms of such damage

could be understood, much of it could be avoided or minimised and support could be made more effective.

This dissertation describes the investigations carried out in the first fully instrumented and observed block, supplemented by the results of several additional investigations in blocks mined subsequently. The results of these investigations have been analysed and many improvements in mining techniques derived from them.

### 1.3 LITERATURE SURVEY

As over the years a considerable body of literature has been built up on basic rock mechanics research and on the principles of rock mechanics it was considered that a review of the literature available on caving methods would provide a broad base for the proposed investigations. This literature was found to be fairly limited and could be divided into three classes:-

- (a) The broad descriptions of the mining methods and techniques practiced at various block caving operations. These are valuable where, through previous experience, the methods of undercutting, degree of boundary weakening, or methods of working have had to be modified.
- (b) The papers which describe the results of monitoring programmes or other observations made on block caving operations, and have attempted to analyse them. Very few of these papers are available, but are extremely valuable from a rock mechanics point of view.
- (c) The papers which attempt a theoretical analysis of an hypothetical cave block, assuming a caving mechanism.

#### 1.31 DESCRIPTIONS OF CAVING OPERATIONS IN THE LITERATURE

The earliest references to block caving date back to E.F. Brown's 1898 description of the original block caving operation at the Pewabic Mine on the Menominee Range in Michigan U.S.A. where 'strong' iron ore was mined, using the earth's forces to crush the ore to a size suitable for handling (quoted by Bucky 1945 P 6,7). In this operation a block of ore 60 to 75m by 60m wide and up to 30m in height was prepared by cutting two narrow parallel vertical open stops on opposite ends block undercutting it.

This resulted in a block which was free on the top, bottom and ends.

The forces acting on the block were due to its own weight, and the side thrust between the hanging and footwalls. After a period of 6 to 8 months, when it was believed the ore was sufficiently crushed, stilled drifts were driven in at 7.6m centres to the end of the caved ore where the ore was then shovelled into cars until waste appeared. When this occurred one or two sets of timbers were blasted down and drawing resumed.

Since then, the method has been applied to a large number of ore deposits, varying in competency from extremely weak to moderately strong, and over the years several variations in the techniques of block caving have appeared, such as in the spacing and methods of working the draw-points, support techniques, the method of undercutting a block and in the size of the blocks and amount of boundary weakening.

The example of a block caving mine quoted frequently in text books is the Miami Mine in Arizona where a very large deposit is being mined. The ore is thoroughly fractured, and occurs as copper bearing pyrite and chalcopyrite in a schist which varies from hard and silicified to soft and kaolinised. The first caving procedure used was to undercut a panel the entire 150 to 200m width of the orebody and 45m wide. This, however, led to damage to the extraction openings and therefore to reduce the load on the grizzly level the size of subsequent blocks were reduced to 45 x 90m and later to 45 x 45m. Still later the block sizes reverted to 45 x 90m. Ore pillars (Pillar blocks) 15m wide, were left between the cave blocks to minimise dilution and were subsequently reclaimed. The ore was drawn through finger raises at 5.1m centres put up from the grizzly level to intersect the undercut. Undercutting was done by fan drilling a 4.5m high slice from undercut drifts developed from the finger raises, and blasted. Ore passing from the two fingers to each grizzly was gravitated down long transfer raises to a haulage level below (Bucky 1945 P 9 - 17). Boundary weakening drifts were originally developed on all blocks but the practice was later discontinued.

Although the Miami Mine is one of the best known examples of block caving, it is one of the few block caving mines where pillars of ore were left between the cave blocks. Most block caving operations are carried out in extensive orebodies where one block cave is next to

a previous caved block, in a 'panel retreat' where the ore was drawn so as to maintain the ore-waste contact as an inclined plane.

A study of the other available literature of block caving operations has shown that almost without exception the orebodies mined have been so extensive that several cave blocks could be mined next to each other on the same level or general elevation. In most cases getting the initial block to cave has not presented any real problem as mining was started when boundary cut-off stopes or boundary weakening drifts were still considered necessary. As caving of second and subsequent blocks generally occurred with greater ease than the original, the need for the boundary weakening drifts or boundary cut-off stopes was brought into question and usually discontinued. In two recent examples, Rio bianco and Urad Mines, the initial blocks laid out failed to cave, and expensive programmes to assist caving had to be carried out (Kendrick (1970) and Carpenter and Woolfe (1972)).

The characteristics of ore in which the various block caving mines are operating vary tremendously, probably more than many writers on the subject of block caving are aware, from very soft friable ores, through sticky, clayey partially decomposed rock to extremely competent rock with few joint planes such as the limestones which were mined at the Crestmore Mine in California. The nature of the ore has to some extent influenced the choice of drawpoint spacing, but this appears to be still very much a matter of preference. Two extreme examples are the Sunrise Mine, mining an orebody up to 100m thick, described as mainly a weak red earthy hematite and limonite with some hard, strong specular hematite inclusions. At this mine the distances, centre to centre of the drawpoint finger raises varied from less than 4m at right angles to the grizzly drift to 6m parallel to it (Bucky 1945 P 56). By contrast, at Grace Mine the drawpoint spacing is 15m on strike and 18m down dip in an orebody of 130m vertical thickness. Here the ore is described as consisting of magnetite with scattered lenses of limestone and a few veins of quartz. The ore, while typically moderately friable and easily broken when struck with a hammer, can range from very crumbly to very hard (Anon 1973)).

When an orebody is relatively competent, requiring ring-drilling and blasting, sublevel caving is a frequently preferred mining method. It is essential to sublevel caving, that the hanging wall caves readily

and follows the draw-down of the ore. As most orebodies where this method has been applied are very extensive, the problems of initiating the caving of the hangingwall appear to be rare and consequently in practically all descriptions of sublevel caving operations it is assumed that no problem in caving of the hangingwall exists, and attention is concentrated on the layout and ring design, based on theoretical patterns of draw. Among these are papers by : Cox (1967), Mason, Cokayne and Aaro (1973) and Just (1972).

### 1.32 GENERAL PRACTICAL OBSERVATIONS OF CAVING

At the symposium on block caving organised by the American Institute of Mining, Metallurgical and Petroleum Engineers (AIME) in February 1941, five questions relating to block caving were discussed by an invited panel of leading block-caving mining personalities (Bucky 1942). The questions were:

1.321 How does one determine whether an orebody will block cave? - The following comments were offered: any orebody with the right size, shape, attitude and physical properties will cave; the size should be extensive to permit an undercut large enough to initiate caving; it should have sufficient draw height; the ore should be homogeneous, as weaker ground in the undercut area tends to run out leaving the more competent zones as large blocks; in general the caved ore should not exceed 0,9m in diameter or the largest size which can be handled.

1.322 How does one determine block dimensions and the development work necessary to make an orebody cave? - The unanimous opinion was the size could only be determined by experiment and experience. Larger blocks were required in virgin areas and the size could be reduced by later blocks, but in some ores boundary shrinkage stopes or boundary weakening drifts and corner raises might be needed. Coyote blasts may also be required to loosen joints etc. Smaller blocks were preferred because less maintenance of the underlying excavations was required with them. While it was recognised that the forces exerted by the caved ground on the bottom of the block were not dependent on depth, it was felt, however, that the stresses in the cave back were depth dependent.

1.323 How do the forces act and what are the stresses developed in a block that cause it to block cave? - This question remained unanswered, but points made were: that dynamic and tectonic stresses could affect the stresses in the back, and that a time element was present in the caving process. On the caving mechanism, it was considered that this took the form of raveling or sloughing of the ore from the back, but it was also conceded that this may not occur in soft ores.

1.324 What determines the procedure for removing caved ore? - The drawpoint spacing should be as close as possible, the rate of draw related to the rate of raveling until the block had completely caved to avoid the formation of dangerously large cavities. Once caved, the rate of draw may have to be high to minimise weight on and damage to the underlying extraction openings, and in sticky ores, to prevent consolidation. Rates of draw of 225mm/day to 900mm/day were quoted, with 225mm/day being preferred. No preference was expressed for either the panel retreat or block methods of lowering the ore/waste interface.

1.325 What are the criteria for determining the efficiency of a block caving operation? - The points considered included the cost per ton of ore, rate of production, dilution, loss of ore, cost per unit of ore recovered.

### 1.33 SPECIFIC INVESTIGATIONS OF BLOCK CAVING

The remaining papers describing investigations and observations of block caving operations will be discussed under one of the following headings appropriate to the specific aspects investigated, Rock Mass Quality and Caving, Caving Mechanisms and Surface Features of Caving, Effects of Undercutting on Drawpoint Level.

1.331 Rock Mass Quality and Caving - Very few studies have been made on the quality of the rock required for caving, and to the writer's knowledge the only two were done at Clonca, the first by King (1946) and the second by McMahon and Kendrick (1969). King developed a system of rock classification based on geological structural features, mineralisation, silicification, sericitization and decomposition. The rocks were classified into four classes varying from one extreme. Class One being very strong, hard and competent to Class Four represented by very weak, soft incompetent rock such as the broken and crushed rock

along fractures and faults. Class One was considered uncavable, with very large spans being probable, and if it did cave, the ore would break into blocks too large for the slusher system. The moderately strong Class Two rock was caved successfully but stopes exceeding 60m x 60m were required to initiate caving in a block removed from other caved blocks. Caved, Class Two ground was large with little tendency to pipe or funnel. The moderately weak Class Three rock was steadily cavable and the rock broke up into a size (up to 3m) that was easy to handle through drawpoints and in the slusher drifts. Large drifts in moderately weak Class Three ground would require timbering.

Class Four (very weak) rock would not support large openings and drifting or raising through this ground required complete close timbering. Unsupported openings over 7.5m in span would fail or cave.

This classification system does not appear to have been used again, possibly because of the amount of work involved in determining the classification, or because the classification was too broad to decide borderline problems.

Deere et al., (1967) proposed the use of a 'Rock Quality Designation' (RQD) as an index of rock mass quality. The RQD is a modified core recovery technique in which only lengths of sound core exceeding 100mm in length is counted. This was followed by McMahon's (1968) proposals for another index of rock mass quality, the 'Joint Breakage' index. This index is defined as the percentage of the total area of any exposure surface that is composed of joint faces. Different indices would be obtained from different methods of excavation, for example, it would be higher for conventional blasting than for smoothwall blasting, and even higher in natural exposures. Therefore the mode of formation of the exposure had to be specified.

McMahon and Kendrick (1969) attempted a correlation between various indices of rock mass quality and an independently derived 'cavability number'. In order to obtain the 'cavability number' six experienced mine operators were asked individually to rank the areas they had worked in the Climax and Urad Mines in order from one to ten. One indicated excellent and ten indicated very poor caving characteristics. This showed a close correlation to the secondary blasting efficiencies. The cavability number is therefore a measure of the

quality of the caved product resulting from a certain mechanism. The comparison of the cavability number with the various indices of rock mass quality showed no relation to the Schmidt hammer hardness or to the Joint Breakage Index, but a close relation to the RQD. They concluded that caving behaviour is much more influenced by variations in degree of faulting and shearing than by the relatively minor variations in rock hardness and jointing characteristics measured in the study areas.

1.332 Caving mechanisms and surface features of caving - Observations of caving action have in general been very limited and in almost all cases observations were made in cave blocks which failed to cave satisfactorily. The reasons for this are simply that no monitoring was required for ore which caved freely nor for those block caves which were provided with boundary cut off stopes because the harder ore was not expected to cave easily. One of the few successful caving operations observed was Miami where Fletcher (1960) reported caving action as a sloughing from the back of a cavity within the vertical limits of the undercut areas. In one stope he observed the caved zone to narrow down above the undercut at an angle of  $83^{\circ}$  from the horizontal. In almost all the unsuccessful caving operations such as Jennifer Borate Mine, Kern Co. California (Obert and Long 1962) Urad, Colorado (Kendrick 1970) and Rio Blanco, Chile, (Carpenter and Woolfe 1972), the undercutting and drawing left an extensive horizontal cavity from which occasional sloughing from the back occurred.

There have been several descriptions of the surface subsidence zones developed from caving operations. Typically the subsidence zone is described in terms of the 'angle of draw', 'angle of subsidence' and 'angle of break'. The term 'angle of draw' comes from coal mining practice and has been interpreted in different ways, some writers measure it from the horizontal and others from the vertical; some use it to describe the line joining the limit of mining to the limit of surface subsidence; to others it is the line joining the limit of mining to the limit of lateral displacement on surface. Johnson and Soule (1963) defined the angle of break as the angle (measured from the horizontal) of inclination of the line joining the nearest underground workings with the outer limit of surface fracturing, which are usually tensional frac-

tures resulting from lateral surface movement rather than subsidence movement. The angle of subsidence is the angle (measured from the horizontal) of inclination of a line drawn to connect the nearest underground workings with the outer limit of surface subsidence movement.

At San Manuel, Johnson and Soule described the subsidence zone over the extensive 260m by 880m undercut area at the south orebody as a central core which subsided, and was surrounded by a series of scarps, which decreased in height with distance from the central area, finally grading into tension cracks formed from lateral displacement. The angles of subsidence and break were lower on one side than the other. The difference between the angles of subsidence was larger than the angle of break giving a wider zone of tension cracking on the side with the steeper angle of subsidence.

No significant zone of tensional cracking was noted by Obert and Long at the Jennifer Borate Mine, but the development and failure of the tensional zone on the hill slope above the subsidence zone resulting from caving at Climax Molybdenum Mine has been described by Vanderwilt (1949). This zone of tensional cracking was up to 150m wide and within this zone a series of blocks 15 to 30m wide, 30 to 45m high and 30 to 150m long were formed between the cracks. These blocks were free to tip toward the cave zone at the top but at depth were held fast. The movement of the blocks, in tipping forward, produced additional fractures until the blocks were reduced to weak piles of rock which moved down the slope as a rock slump, rock slide or combination of the two. In a rock slump, blocks rotate backwards at the top and forward at the base. In rock slide the broken material moves as a mass or individual boulders roll on fracture planes parallel to the surface. At San Manuel, Johnson and Soule (1963) noted similar forward tilting with the dropping of pie-shaped wedges into the opening cracks. Heslop (1974) explained the surface subsidence zone at Havelock Mine in Swaziland as the tilting of large slabs of hangingwall rocks towards mining operations with a wedge shaped piece subsiding into the opening final crack. This explanation of the subsidence zone was necessary both at Climax and at Havelock because underground development openings below the subsidence zone showed no evidence of a deep lying shear plane on which the observed

surface failures could take place. At Miami an experiment to leach copper from the tensional zone failed because the tension cracks did not link with the underground workings. However, in a recent series of papers Hoek has attempted to analyse the history of hanging wall failures at the steeply dipping Granjesborg Mine by invoking shear failure on a plane dipping towards the orebody. At this mine the angle of break has shown a gradual lowering with increasing depth of mining from  $80^{\circ}$  at a depth of 140m to  $60^{\circ}$  at a depth of 300m (Hoek 1974).

1.333 Effects of undercutting on the drawpoint level below an undercut -

Merril and Johnson (1964) studied the strain and displacement created by undercutting in block caving in the concrete support work on the slusher and grizzly horizons at San Manuel and Climax Mines. This work showed, as expected, a sharp reduction in vertical loads as the undercut passed over the instrument stations, with an increase in lateral loads. At a station below the centre of the block, after completion of the undercut generally increased lateral and vertical loads were imposed on the concrete, while to the edge of the block, the undercutting resulted in merely a reduction in load. In the periphery increased vertical loads were recorded, but surprisingly, no increased loads were noted on the grizzly horizon as undercutting progressed.

Helliwell and McKorie (1962) at the Errington Underground Mine at Steep Rock placed some load cells beneath a top slicing operation. The results demonstrated that the stresses beneath a caved block were not proportional to the depth and density of the overlying caved ground but were close to that predicted from Jamison's formula for the average stress on the base of a non-flowing bin. At Steep Rock the yields and deformation of yielding arch sets or scum drifts in a block caving operation were also monitored. The results showed the greatest yields in the middle of the centre scum drift. The maximum yields were up to 300% of the other general deformation levels. These indicate that the centre of a caved block carries a higher proportion of the weight of broken material than the peripheries. However, it is considered unlikely that the centre carries the very high proportion indicated by the sand filled bin experiment quoted by Woodruff (1962). It is considered that the high proportion could have been due to the way the bin was loaded,

giving a dense, less compressible central core.

#### 1.34 THEORETICAL ANALYSES

The problem of calculating the stress in the back of a block cave undercut, considered insoluble at the 1941 AIME symposium, came very close to being solved when Sadowsky and Sternberg (1949) developed the mathematical formulae for calculating the stresses around an ellipsoidal cavity. Together with the mathematical formulae for calculating the stresses, they presented a series of graphs of the stress concentration factors along two of the three axes for various principal stress ratios and ellipsoid shapes. Unfortunately the stress concentrations along the third axis were not computed and consequently the stresses in the back could not be readily derived for a practicing rock mechanics or mining engineer. This study was followed by a similar one by Terzaghi and Richart (1952), in which stresses were calculated around prolate and oblate spheroids. Since these papers were published, the finite element technique has become very popular. This is a versatile computer technique which allows an accurate description of excavation shape and body forces, rock properties, elastic or other rheologic laws of ground response to be modelled (Jaeger and Cook 1969). While this technique is theoretically capable of modelling three dimensional problems, it would require such a long computational time on a large computer that it is not considered worth the extra effort. The three dimensional stress analysis programme developed by the South African Chamber of Mines recently promises to fill this gap. At this stage the programme is based on the assumption that the rock behaves as a homogeneous, linearly elastic material which restricts its use (Wagner 1975).

On a simpler, two dimensional basis for analyses, Geldard and Udd (1962) calculated and presented graphically the stress concentrations on the surface of elliptical openings of various shapes from a circle to a crack and with various stress conditions and orientations. This paper was used as the basis for Morrison and Geldard's (1964) paper on the stresses induced by block caving, from which they conclude "that the old, and somewhat obvious practical explanation is still correct; 'it breaks up under its own weight'. The contribution of the field stress is to assist or retard the process by loosening or tightening the

undercut rock mass. The best conditions pertain with the lowest horizontal stress".

Jenike and Leser (1962) adopted a different approach to the problem by regarding the caved and uncaved ore as a continuum, and analysing the caving action as a wave front moving upwards as a function of time and rock properties. While this approach may have some application in the very weakest of caving ores, it is not applicable to most operations where a cavity beneath the back is either observed or inferred.

On the experimental side, most work has been done on the mechanics of drawing the ore in a block caving operation in order to maximise ore recovery and minimise waste incursion. One of the most significant of these descriptions is McNicholas, et al (1946) done at Climax Mine. The experiments were done in a glass fronted box filled with 0.6% MoS containing mill tailings and sand representing the waste capping. From this work various parameters of block caving operations were worked out.

#### 1.35 SUMMARY AND CONCLUSIONS

The most significant point to emerge from the literature survey of cave mining practice was the wide variety of ground conditions to which the method has been applied. In many cases these ground conditions have been inadequately described to enable anything but broad comparisons to be drawn.

While it is clear from the survey that increasing the size of the undercut enhances the chances of initiating a cave; it is also clear that increasing the size of the undercut increases the chances of severe "weight" on the extraction openings. How big is big enough? This is usually determined by experiment, extending the undercut until a cave is obtained, which is not ideal nor always practical. The problem has been recognised and attempts made to solve it by the use of a rock classification system. Unfortunately no standard classification system has been adopted yet and still insufficient data are available for analysis.

A more fundamental approach involving the observation and analysis of the mechanisms of caving appears necessary as a variety of mechanisms as might be expected from the variety of ground conditions. However, to date, insufficient observations have been made to draw any conclusions.



Fig 1,1 Simplified geological map of Rhodesia showing location of Shabani

Some of the theoretical analyses have therefore been based on limited observations of caving under certain ground and stress conditions, and are consequently not universally applicable.

The research described in this dissertation was undertaken because of the need in the asbestos mines of the African Associated Mines group to establish the size of undercut required to initiate caving while not making the area too large which would lead to "weight" problems. Also it was necessary to be able to recognise with certainty which of the smaller orebodies were not extensive enough for caving by merely extending the undercut to the orebody limits, and where other cave induction methods would be needed, or where caving was simply not practical.

There was also a clear need to understand the behaviour of the rock mass as a whole so that effects on the peripheries of creating such a large volume of broken rock, as block caving does, can be established. The literature descriptions indicated the existence of a zone of near-surface effects while damage at depth occurred in certain cases, some of these occurrences appeared to be related to geological features, and no pattern emerged from the literature survey.

#### 1.4 LOCATION AND BRIEF HISTORY OF SHABANIE MINE

Shabanie Mine is immediately south of Shabani township and is approximately 200km east of Bulawayo on the Bulawayo-Fort Victoria road (Fig 1.1). The general altitude is 960m.

The chrysotile asbestos occurs as seams with zones of complete serpentinitisation in a partially serpentinitised ultramafic sill. There are at least 40 orebodies, which have been worked over a strike length of 5km. These bodies are separated from one another along strike and down dip, by dykes and zones of talc carbonate rocks developed along major wrench and thrust faults. The first record of asbestos in this part of the world was in 1906 when some samples were sent to the Bulawayo museum for examination, but there is some doubt that the samples came from the Shabani deposits and may have come from one of the Belingwe deposits. The first claims staked in the area were the Shabanie claims, staked by Mr M. Kerr on behalf of the Rhodesian and General Asbestos Corporation in 1915. In 1916 and 1917 the Birthday and Wil Desperandum

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claims, among others, were stated. Turner and Newall formed the African Asbestos Mining Company in 1919 and took over the Nil Desperandum claims. The Rhodesian and General Asbestos Corporation acquired the Birthday claims, plant and workings from Willoughby's Consolidated Company in 1924 or 1925 and within three years embarked upon an expansion programme. In about 1930 Turner and Newall bought the Rhodesian and General Asbestos Corporation and thus acquired control of Shabanie, Gaths' and King Mines. This brought all the workings in the Shabanie deposit under one management.

As indicated earlier, initially open pit mining was used, and with technical improvements it has been used almost continuously ever since, although its relative importance in tonnage produced has declined over the years.

In the 1920's shrinkage stoping was started in the Nil Desperandum Mine, and this evolved into an overhand cut and fill stoping method. In the late 1930's and early 1940's the method was employed on an increasingly large scale on all three sections of the mine and became the main tonnage producer in the mid 1940's. However, with increasing sizes of the stopes a number of collapses occurred, and this led to the eventual abandoning of this method.

In the early forties, a form of block caving was attempted successfully in one of the orebodies where a large section of the cut and fill stopes had collapsed. After another two major collapses the cut and fill mining method was abandoned and the decision was taken in 1948 to change over to block caving. By this time all the major orebodies down to 235 level on the Shabanie, 295 level on the Birthday and 12 (265m) level on the Nil sections had fairly extensive cut and fill stopes. For the following twenty years the block caving operations were carried out in these blocks. In some cases the stopes have assisted in initiating caving while in others they have presented layout and ground control problems.

Sublevel caving was introduced in 1958. This method was initially known as the "Kiruna Method" and was for a period known on the mine as "sublevel loading". Lately the method has been applied in several mining blocks, but there is some doubt as to its efficiency.

Several large open stopes have developed on the mine mostly as a result of the failure of the hangingwall to cave in block or sublevel caving operations. In a few instances, however, where the orebody was considered too small for caving of the hangingwall, the ore has been mined as a shrinkage open stope.

The mining methods are described in full detail in a further chapter.

CHAPTER TWO

THE GEOLOGY OF THE DEPOSIT

2.1 INTRODUCTION

A full description of the geology of Shabanie Mine is included in this dissertation as it is considered necessary to the understanding of the observations described in this work. It will also enable the reader to draw clearer comparisons with other caving operations. To facilitate the description and understanding of the geology of this deposit, which is relatively complex, a brief description of the geological history is included.

In the following descriptions of the geology and structural geology of the various rock types, the petrological descriptions and the interpretations of geological origin have been based on the works of Keep (1928) and Laubscher (1963).

The chrysotile asbestos occurs at Shabani as sub-parallel seams (Fig 2.1) in at least 40 discreet orebodies which are located in a 15km long, partly serpentinitised and talcified ultramafic sill. The sill was intruded into the pre-Bulawayan basement gneisses about 1 000m below the then developing depository of the Belingwe schist belt. During intrusion some differentiation took place giving a thin zone of less basic rocks near the upper contact and subsequently the sill was folded, faulted and altered by carbon-dioxide bearing hydrothermal solutions which led to the development of the talc-carbonate rocks, serpentinitisation and fibre growth. Fig 2.3 is a geological map and section through the area, and Fig 2.4 illustrates the relationship of the orebodies to the surrounding rocks.

Laubscher (1963, P 5) has given the geological sequence as follows:-

Quartz Veins	}	Period of lateral compression acid intrusion and hydrothermal activity
Aplites, Pegmatites		
Granites		
Shear zones in the gneiss		
Talc zones in the ultramafic .		

Diabase dykes

Actinolite Feldspar	}	Shabani ultramafic
Actinolite		
Pyroxenite		
Harzburgite		
Peridotite		
Serpentinised Peridotite		
Serpentinised Dunite		
Silky fibre zones		
Brittle fibre zones		
Talc carbonate rocks		
Greenstones	}	Bulawayan System
Sheffield Ultrabasic		
Sediments		
Gneiss		Basement

## 2.2 THE BASEMENT GNEISS

The gneiss surrounds the ultramafic sill and underlies the Bulawayan system. The gneiss typically shows a wide range of textures and the mineralogical composition varies from granitic to tonalitic. The underground exposures of these rocks are confined to the main vertical shaft development on all the main levels.

## 2.3 THE BULAWAYAN SYSTEM

This system occurs to the south of the mine, and consists of a 150m thick succession of grits, banded ironstones, slates, limestones, conglomerates and quartzites. This is followed by the Sheffield ultrabasic body and this in turn by greenstones. The succession has been folded to give a north west - south east strike and an 80° south-west dip in this area.



Fig 2,1 Sub parallel fibre seams

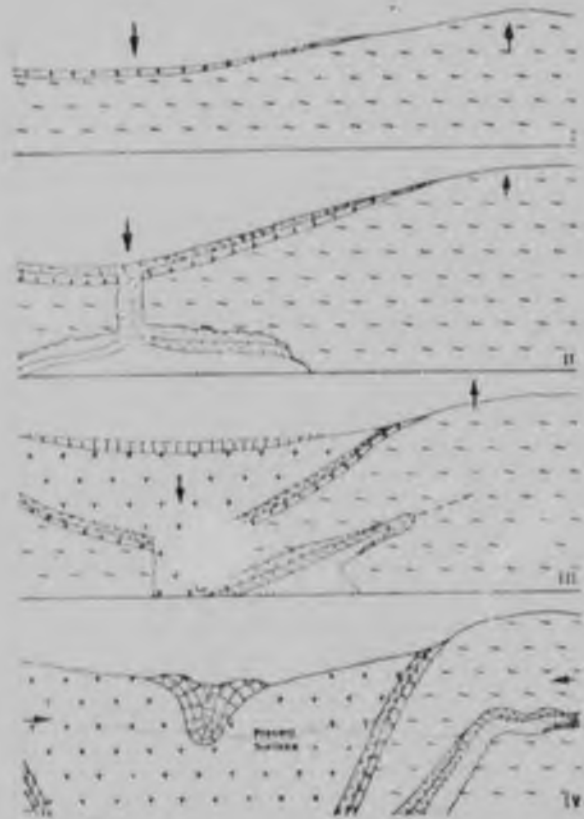


Fig 2,2 Development of the Belingwe Schist Belt (after Laubscher)



Fig 2,3 Geological Map of Shabani Area

#### 2.4 THE SHABANI ULTRAMAFIC SILL

The ultramafic sill has a strike length of 15km, a thickness of at least 1 500m and a 60° south-west dip. The body was magmatically differentiated into a thick dunite zone, overlain by thinner zones of peridotite, harzburgite, pyroxenite and gabbro. The pyroxenite was altered by automagmatic water to talc and actinolite rock and the gabbro to actinolite feldspar rock (Laubscher 1963 P 28;30).

After emplacement, the sill was intruded by diabase dykes and then subject to a high lateral pressure which folded the sill and faulted it. The faulting took the form of thrust faults with associated shearing parallel to the lower contact of the sill, and thrust and wrench faults cutting through the main body of the sill. These faults acted as channelways for the earlier hydrothermal solutions which altered the dunite to serpentine in which the silky asbestos fibre formed. The later hydrothermal solutions were carbon-dioxide bearing and altered the rock in the vicinity of the channelways to talc schist and talc-carbonate rock. In at least 40 of these compartments formed by the faults and associated talc zones, the serpentinisation process went sufficiently far to develop payable concentrations of silky chrysotile fibre.

As the hydrothermal solutions penetrated the sill, the zones of talcification grew in width, altering previously serpentinised dunite firstly to an intermediate stage carbonated serpentine and finally to a talc-carbonate rock. The carbonation firstly made the chrysotile fibre seams brittle, finally the seams were completely altered to talc, remaining recognisable as altered fibre seams. These zones of carbonated serpentine formed between the talcified zones associated with the footwall and faults and the serpentine and silky fibre of the interior (Fig 2,4 and 2,5).

The general relationships of the various orebodies is illustrated in Fig 2,6.

#### 2.41 THE FOOTWALL TALC ROCKS

The footwall talc rocks consist dominantly of talc-magnesite rock, with zones of talc schist and, in some areas, graphitic schists.

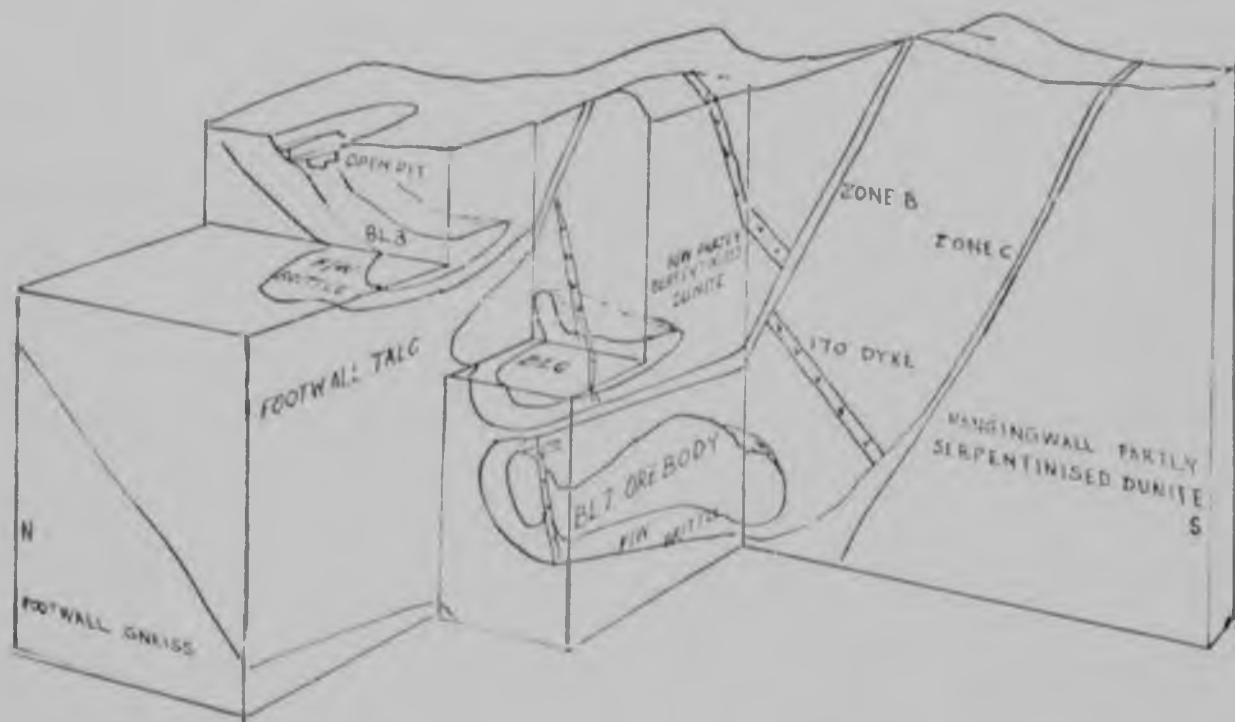


Fig 2,4 Block diagram



Fig 2,5 Cross section through Blocks 3,6 and 7 orebodies

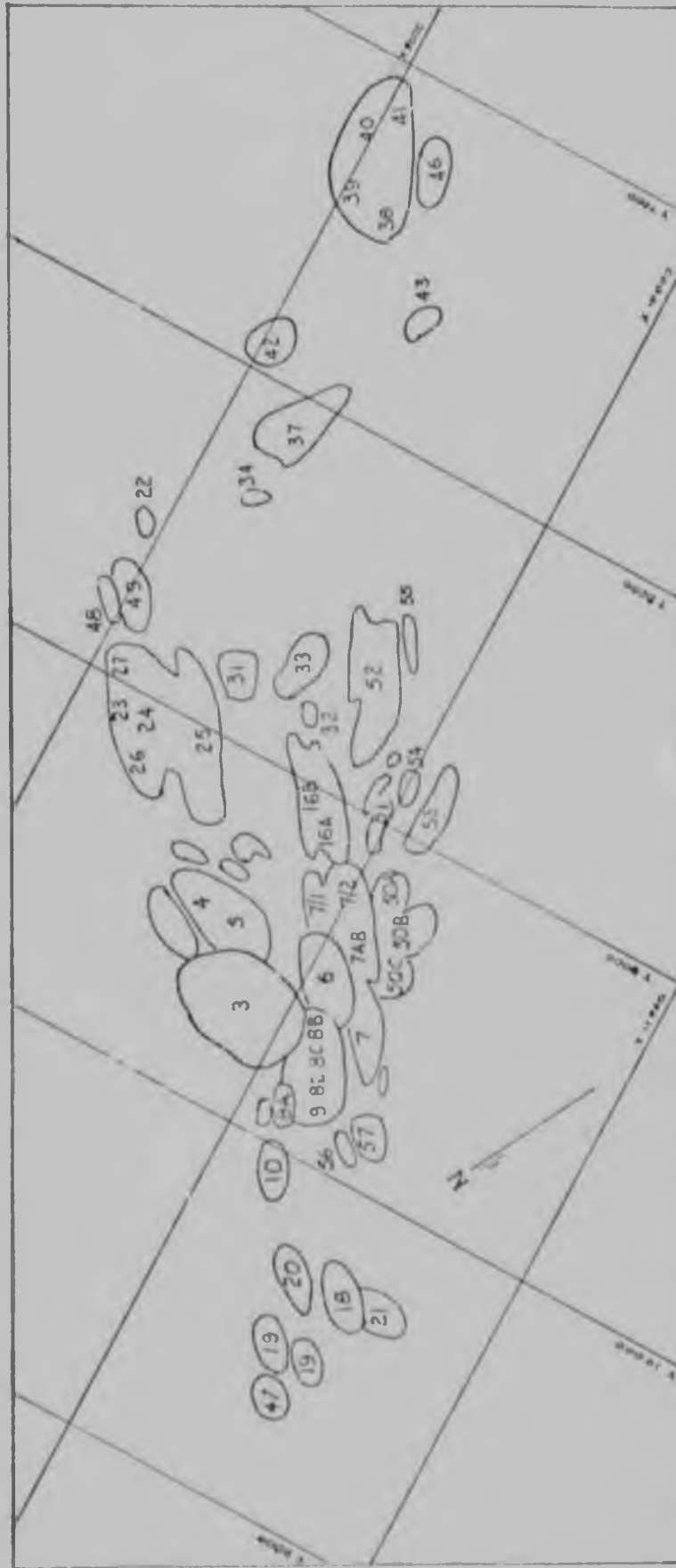


Fig 2,6 Plan showing the relative positions of the orebodies



fact that the Shabanie cores were soaked in water prior to testing, the Shabanie tests gave considerably lower strengths.

#### 2.42 THE BRITTLE FIBRE ZONES

The brittle fibre zones occur between the talc rocks of both the footwall and transgressive talc zones and the silky fibre bodies, as shown in Fig 2,4. These zones represent an intermediate stage in the formation of talc magnesite rocks from serpentinitised dunite by carbon-dioxide metasomatism. The fibre has a lower flexural strength due to the replacement of the outer layers of chrysotile of the fibres by talc and magnesite. The host rock consists of talc, magnesite, serpentine and occasionally olivine (Laubscher 1963 P 19;20).

Contact with the silky fibre bodies is gradational, and this zone is commonly fractured. Minor faults or slips are common in the brittle zones, and are usually cemented by carbonates, making them stronger than the slips that occur in the orebodies. Major slips with vugs are occasionally found. The brittle fibre zones are usually competent (Class 1 to 3) but at the contact with the silky fibre bodies the rock can be highly fractured and very incompetent.

The brittle fibre zone is not normally mined for its fibre content, brittle fibre being regarded as a containment is a soft, silky fibre product. These zones are favoured for siting haulages and other access development workings, because of the generally superior competency of these zones.

Appendix 2 gives some of the properties of these rocks. A very high variation in strength is apparent, and the higher values are probably due to a high degree of carbonation and possibly some silicification while the lower strength specimens have lower degrees of carbonation. These tests were carried out on the mine, after soaking in water for one week, in order to eliminate the variations in strength due to varying water content.

#### 2.43 THE SILKY FIBRE BODIES AND OREBODIES

The silky fibre bodies vary in size and shape, within the lateral limits imposed by the talc-magnesite and adjacent brittle fibre zones

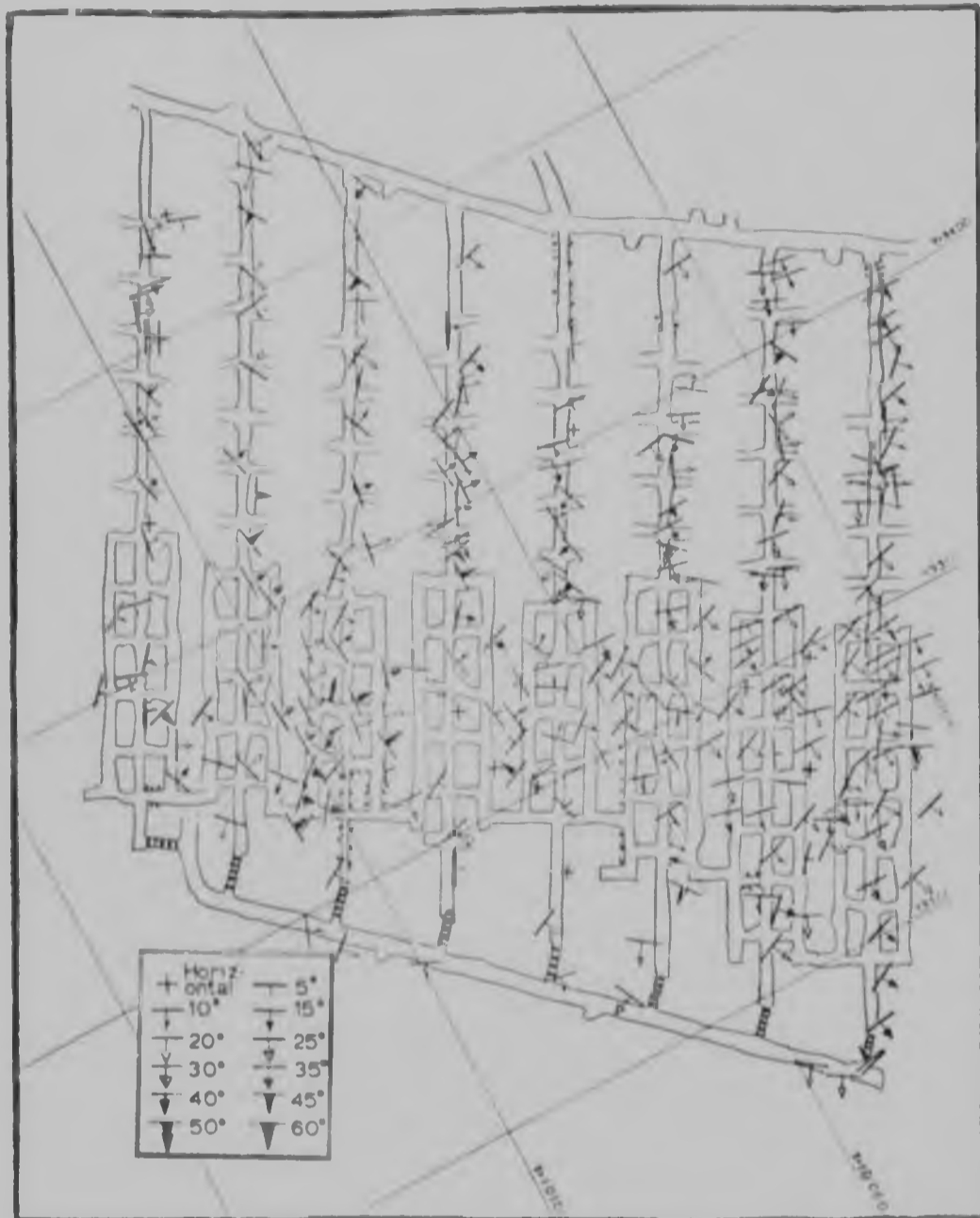


Fig 2,7 Plan of 265 level Block 7AB, showing variation in orientation of fibre seams

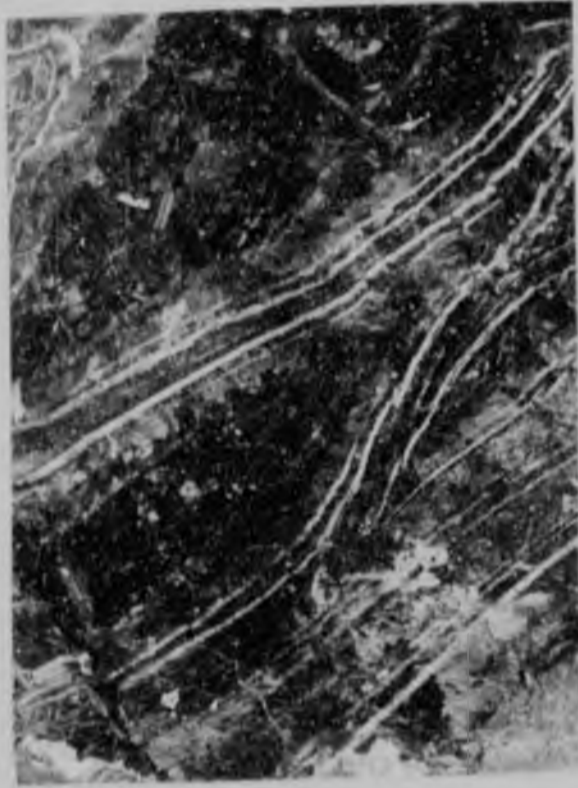
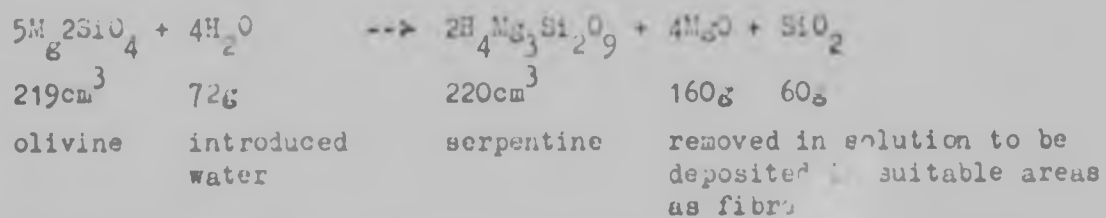


Fig 2,6 Conjugate Ribbon Fibre Seams



Fig 2,9 Double Fibre Seam

developed along major faults and dykes. The fibre length and quantity varies not only from locality to locality within the bodies, but also from body to body, and, depending upon the fibre length and quantity, portions of these bodies may be payable. In general the footwall portions of the fibre bodies and those adjacent to the talc-magnesite zones are richer than the more remote parts of the bodies, therefore the orebodies generally have the hangingwall contact defined by an economic limit, while the footwall and lateral contacts are defined by the brittle/silky fibre contact. The fibre occurs in sub-parallel seams in the partially serpentinised dunite (Fig 2,1). The seams are developed within zones of complete serpentinisation, which are generally narrow in the upper portions of the fibre bodies, forming a small proportion of the total volume of rock. Closer to the footwall or talc-magnesite zones, the fibre seams become more frequent and/or larger and the serpentine bands become wider, until in certain areas, no partially serpentinised dunite remains. Similar serpentinisation zones are formed in the rock adjacent to the numerous slips which traverse the dunite. In the serpentinisation process, water reacts with olivine in the dunite, which results in an increase in volume or the removal of some material. Some of the removed material was deposited in the fractures to form the fibre seams. Laubscher (1963 P 23) considers that here the following reaction took place:



The fibre seams are sub-parallel to each other and vary in dip, generally between  $0^\circ$  and  $90^\circ$ . Fig 2,7 is a plan showing the considerable variations in strike and dip of the fibre seams in Block 7AB. The nature of the fibre seams varies from fibre body to fibre body and within fibre bodies. The most common types of fibre seams are the ribbon seams which generally occur towards the footwall and the resultant rock is moderately competent, and the single or double fibre seams (Fig 2,8 and 2,9) which are usually larger and make the rock slightly

less competent, although this is not necessarily due to the type of fibre seams alone, but may also be due to the fracturing or slips which may be associated with them.

The fibre bodies are cut by many slips (minor faults). The spacing of these slips varies, from less than a metre in the zones of high slip density to several metres in the low slip density areas. They occur in up to five preferred orientations in an area, and most exhibit striations in several directions which are commonly parallel to other slip orientations. Branching of the slips is common and bends also occur but less frequently. Shearing is frequently associated with the larger slips. The amount of displacement is usually difficult to determine; most probably have displacements of less than 1m. The shear strength of the slips is largely determined by the filling materials of which massive, columnar or platy picrolite is almost ubiquitous. Cross-fibre seams are fairly frequently developed in the slip planes. Magnesite and magnetite are common filling materials and may occur in slips with or without crossfibre development. Soft, clay-like talcose gouge is an important, if less common, filling.

The shear strength of slips is also affected by the number of parting planes, and the straightness of the slip. Gouge filled slips tend to be the straightest and are therefore usually the weakest. Slips filled with platy picrolite and/or with associated shearing have intermediate shear strength. The larger slips usually fall into one of these classes, while the minor slips tend to have the harder filling materials and are more irregular. There are, however, some irregular very incompetent blocky zones in which all the numerous, closely spaced, minor slips occur in a multitude of directions and have a weak platy picrolite filling.

Two dominant types of fracturing occur. In the footwall the rock along the contact with the carbonated serpentine of the brittle fibre zone is fractured and frequently these fractures are filled with secondary minerals. In the hangingwall areas where the serpentine zones associated with slips and fibre seams are widely separated, the serpentine is usually fractured with S-shaped fractures (Fig 2,9). These fractures are usually filled with a magnesite paste.

Some of the physical properties of the rocks of this zone are given in Appendix 3.

#### 2.44 THE HANGINGWALL PARTIALLY SERPENTINISED DUNITE

Overlying the orebodies the hangingwall dunite is partially serpentinised, the fibre seams and slips have the same general attitudes and properties but are more widely spaced than in the orebodies and the zones of serpentinisation associated with them are narrower. In the upper portion of the sill the fibre seams are almost totally absent. At the surface the dunite is weathered to a depth of 20 to 30m to a soft material with sub-horizontal magnesite veinlets or to birbirite which is harder. Underground exposures may weather within a few months.

#### 2.5 DIABASE DYKES

A group of diabase dykes are intruded into the ultramafic sill, which post-date the Bulawayan deposition but pre-date the shearing and faulting, and the fibre formation in the Shabanie ultramafic.

Due to hydrothermal alteration the composition varies, with the dominant minerals in the unaltered rock, augitic pyroxene and labradorite, being altered in varying amounts to actinolite and zoisite. In the ultramafic where the contacts have been sheared and also in the footwall talc-carbonate zone, the dykes have been altered to talc-carbonate rock (Laubscher 1963 p 35). The dykes are up to 10m thick and have a general NE-SW strike and dip of 45° E. Typically they have a blocky nature and the competency varies from Class 3A to 4B.

#### 2.6 THE YOUNGER GRANITE AND RELATED APLITES AND PEGMATITES

A younger alkaline granite body occurs to the north-west of the ultramafic body, and is considered by Laubscher to be the source of the rich solutions from which aplites and pegmatites formed in the gneiss, and in the ultramafic sill a coarse hybrid rock of orthopyroxene and feldspar with margins of serpentine and iddingsite, or in areas of lower temperature, a fine grained rock consisting of quartz, feldspar, biotite and muscovite with margins of serpentine and sometimes iddingsite. In the ultramafic these hybrid rocks have a general east-west strike and are nearly vertical. The thickness varies; the majority are less than one metre thick, but occasional exposures are up to four metres thick. The fine-grained hybrid rocks weather rapidly, and are generally closely fractured, constituting incompetent zones. The coarser grained variety is less fractured and less susceptible to weathering and so is usually competent.

## 2.7 THE STRUCTURAL FEATURES

While the various structural features have been discussed in the relevant sections above from geological and rock-strength points of view, but nothing has been said of the overall structural picture.

The greatest amount is known of structures in the central portion of the ultramafic sill, where the mine is situated. In this area the major structures consist of north east - south west striking thrust faults, north-south striking wrench faults and east-west striking wrench faults. In the central portion of the mine the thrust faults are best developed and most have a 60 to 70° north-west dip. The footwall shearing is also a thrust fault with the same strike and a southward dip of 45°. On the eastern side of the mine the dominant major structural features are north-south striking wrench faults while on the west side the dominant structural features are ENE - WSW striking wrench faults. There are, therefore, four orientations of major structural features; NE - SW strike dipping steeply NW, NE - SW strike dipping shallowly SE, N - S and ENE - WSW with steep dips. The major slips, minor slips and shear zones do not exhibit the same spatial distribution as the major structures but generally occur in sympathetic directions. These structures indicate that the major principal stress at the time of the formation of these structures was N24°E and had a shallow dip (Laubscher, 1963 P 44). The majority of the fibre seam orientations do not fit into any of these four orientation groups. The orientation of the fibre seams may be related to the minimum stress direction at the time of formation. The fractures associated with the fibre seams have not been studied in detail, but in some areas their orientation is clearly related to the deduced major principal stress direction.

CHAPTER THREEMINING METHODS3.1 INTRODUCTION

The mining methods used on Shabanie Mine in the past have an influence on the present mining methods in the following aspects:-

- (a) The present lay mining methods have evolved from their predecessors as a consequence of the problems with them and changing economic conditions.
- (b) The extent and location of the earlier mining operations has limited the choice of mining methods, imposed physical limitations on some new layouts and limited the choice in the sequence of mining adjacent blocks.
- (c) The mining methods and ground movements associated with them have a marked effect on the ground behaviour in current mining areas in the close vicinity of previously mined areas.

The main mining methods employed to date are open pit mining, shrinkage and cut and fill stoping, block caving (including panel retreat caving), sublevel caving and sublevel open stoping. The areas mined to date employing these methods is shown in Fig 3,1.

3.2 OPEN PIT

Initially, virtually all the outcropping orebodies were mined as small open pits. With improvements in technique, this mining method has been used almost continuously to the present day. The early operations were very labour intensive, the blast holes were hand-drilled by "hammer boys" and hand cobbling or sorting of the fibre from the host rock was done in the open pits. In 1927 jack hammers were tried and proved successful after some problems with blockages by fibre of the water channels in the drill steels. In the later open cast operations hand cobbling was abandoned. In 1964/65 pre-splitting was first used extensively to provide tidy final benches (Fig 3,2).

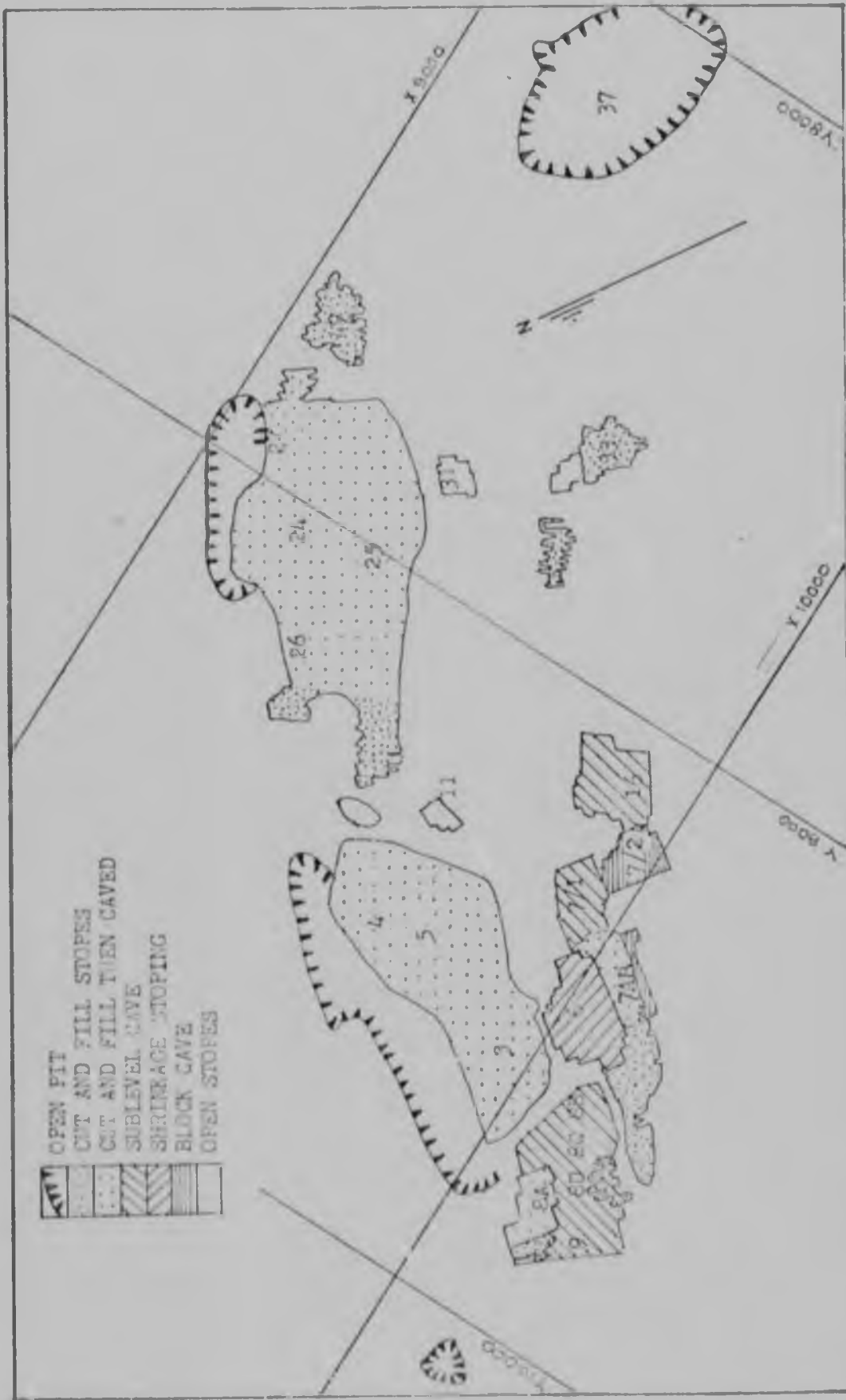


Fig 3.1 Plan showing relative location of mined areas

### 3.3 CUT AND FILL STOPING

The cut and fill stoping method evolved from experimental shrinkage stopes started in 1926/27 in the "F" lode on what was then the Nil Desperandum Mine. These initial shrinkage stopes were 24m wide, and ran from foot to hanging wall at right angles to the strike of the orebodies. Pillars 6m wide were left between the stopes for subsequent removal. This technique was next applied to the Nil "B" lode, where the hand sorting of the fibre bearing rock from waste was done in the stopes. The sorted fibre bearing rock was passed down cribbed orepasses to the tramming level below, while the waste rock was left in the stopes. The waste was supplemented by mill discards to make up the bulk.

As the "B" lode stopes approached lowest benches in the open pit severe cracking of the pillars occurred and large rockfalls took place in the stopes. One of the measures adopted to help stabilise the ground was the building of 6m wide waste rock pillars down the centres of each of the stopes, reducing the spans from 24m to 9m.

When cut and fill stoping was introduced to the 170 and Birthday sections in 1937 the normal stope widths were approximately 9m and the pillar widths about 4.5m. Within five years of the introduction of this method, it had become the main mining method. Stopes were started in all the main orebodies down to 295 level (approximately 295m depth). Fig 3.3 is a block diagram showing the mining method, and Fig 3.4 depicts one of these stopes.

"Pillar reclamation", in reality a form of pillar robbing, was practiced almost from the introduction of this mining method. This pillar reclamation together with elastic closure due to the lateral extension of the stopes led to spalling and cracking of these pillars. Hand built waste rock pillars were widely used to replace the natural pillars. As these built pillars were more compressible than the natural pillars and were built in the stopes where a large amount of the elastic closure had already taken place, these pillars did not crack to the same degree and were therefore believed to be far stronger than the natural pillars. This belief led to the systematic replacement of almost all natural rock pillars with hand built pillars in the production stopes, and in 1948/49 to actually building the pillars prior to the start of



Fig 3,2 Pre-split faces

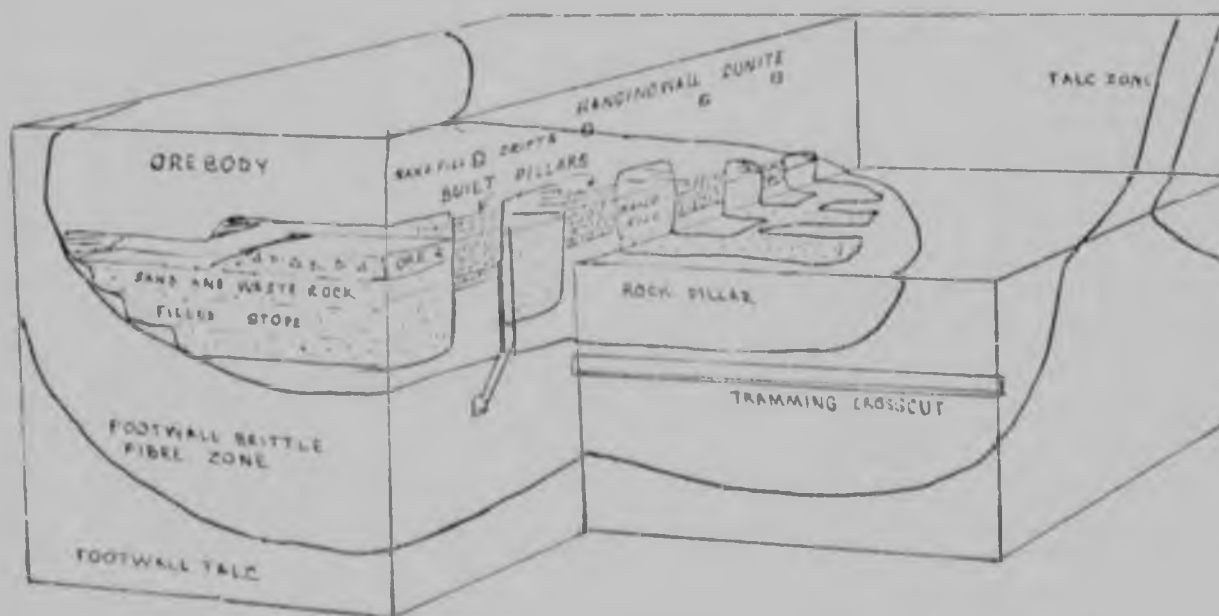


Fig 3,3 block diagram of cut and fill stopes



Fig 3,4 Cut and fill stopes

stopping operations in what is now known as Block 33. In 1945 the 205 level east stopes (now Block 5) and in 1948 the 170 level west stopes (now Block 25) collapsed.

#### 3.4 BLOCK CAVING

Block caving was started on an experimental basis below the area of 110 level west stopes which collapsed in 1941, in an attempt to recover the ore left above the collapsed stopes. Some difficulties were experienced in inducing the subsided hanging wall to break up and draw freely, but by extending the area of the cave block and by local drilling and blasting the block was made productive. A similar method was applied to recover the ore above the Block 5 stopes after they collapsed in 1945, and when the Block 25 stopes collapsed, it became apparent that block caving was a more economical mining method which required less labour. It was decided therefore to change over entirely to block caving. As the cut and fill stopes had a low tonnage output, more stopes were brought into production in several orebodies to supply the required tonnage and consequently almost all block cave production has come from areas converted from cut and fill stoping.

The design of the drawpoints, the pillars over the grizzly drifts and spacing of the drawpoints has changed over the years. The methods of undercutting have largely been dictated by the previous stoping operations. With the exception of some of the earliest grizzly drifts, long-hole techniques have been preferred for cutting the drawpoint cones or troughs, and also undercutting the ore where no previous cut and fill stoping had effectively done this. The thickness of the zone longhole drilled and blasted to create the undercut has varied with time and according to the ore fragmentation expected. In a recent block the undercut consisted of two sublevels, known as the first and second undercuts, from which a 16m thick zone of ore was longhole drilled and blasted to create the undercut. A further sublevel known as the overcut level, was developed between the second undercut and grizzly levels. This level was needed to enable the drawpoint troughs to be cut under void conditions with light charging to minimise blasting damage to the ground surrounding the grizzly drift. When the troughs were complete the overcut level above them was blasted.

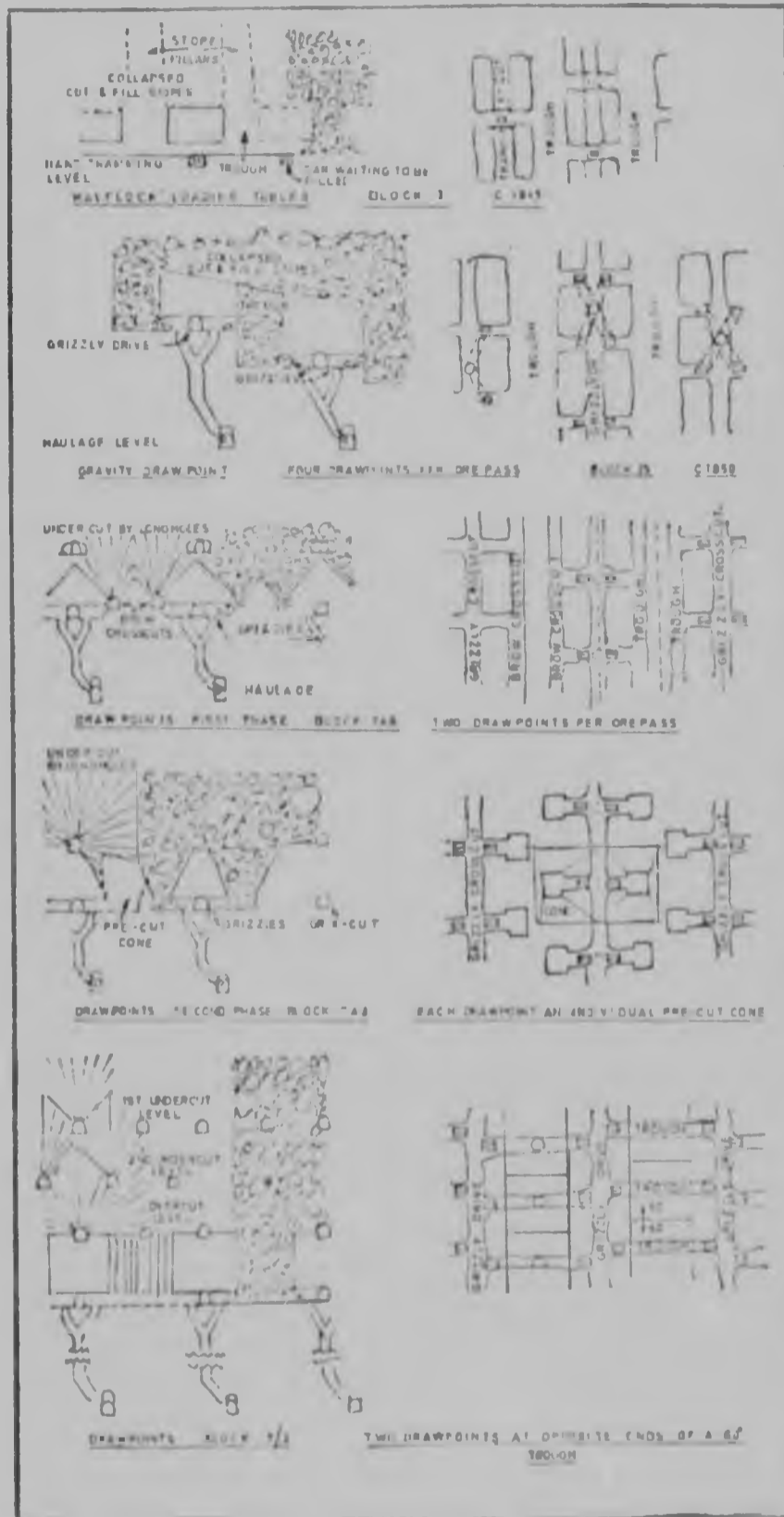


FIG 3.5 Crown Pillar design since 1945

The main changes to the grizzly drift and grizzly levels over the years are:

- (a) the replacement of "Havelock loading tables" by grizzlies at each drawpoint with an orepass to a tramming level below. Up to four drawpoints used a common orepass.
- (b) The drawpoint spacing has been reduced from approximately 10m to 7,5m.
- (c) The shape of the drawpoint and the pillar over the grizzly drift (the "crown pillars") has changed and Fig 3,5 illustrates some of these changes.

In some blocks considerable sections of the grizzly drifts collapsed due to several causes, such as inadequate crown pillar design, poor fragmentation, slow, irregular draw etc. Problems in the initiation of caving are still being experienced in some orebodies where the ore did not cave when the planned undercut was completed.

### 3.5 SUBLEVEL CAVING

Sublevel caving, known until recently as "sublevel loading" on this mine is similar to the method employed in Kiruna in Sweden and other mines, that the sublevel drifts are relatively narrower, and the ring patterns were until recently "old fashioned" by European standards, (with 45° side holes instead of 70°). First applied in 1958, sublevel caving has now become one of the main mining methods. Fig 3,6 illustrates the method. Both hand lashing and loaders have been used for lashing, but to-day lashing is entirely mechanical. As the method has proved to give a low extraction of ore and a high dilution, investigations are in hand to improve this aspect. In a few areas some damage has been experienced to the sublevel caving drifts, but on the whole this method has proved to be relatively trouble-free.

### 3.6 SUBLEVEL ORE STOPING

After the difficulty experienced in inducing the drawpoints below the collapsed portion of Block 3 stopes to flow properly, an experimental block cave was laid in Block 49 with provision for shrinking the ore. The ore was shrunk during 1945 but when drawn the back failed to cave, and the cavity so formed has remained open ever since.

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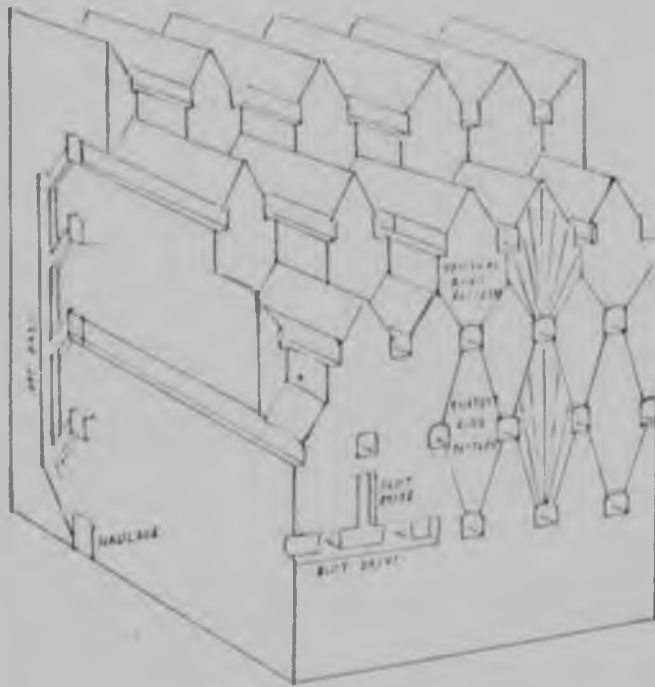


Fig 3,6 Block diagram of sublevel caving

Since 1945, several others failed to cave and in at least two instances, Blocks 11A and 46, the blocks were laid out as an open stope. Block 46 was mined on a sublevel caving principle, and from a stability of the back point of view, was considered to have been successful, but less successful from an ore recovery point of view as blasting threw some of the ore to the back of the stope, and was lost. Block 11A was therefore laid and with drawpoints beneath it, and has been mined successfully.

### 3.7 CONCLUSION

From the foregoing it can be seen that several mining methods have been successfully applied to the Shabarie orebodies; and that all but two are still practiced to-day. These two are the early shrinkage stoping and the cut and fill stoping which evolved from it. They were replaced by block caving because of the collapses which occurred when the area covered by cut and fill stopes became excessive, and because of the

low output per manshift.

The mining methods still practiced include: open pit mining - which is the first choice of mining methods for the near surface ore - body, block caving, sublevel caving, sublevel shrinkage and sublevel open stoping. While sublevel caving can be applied to a relatively wide range of ore competencies and orebody sizes, the ore recoveries achieved on Shabanie Mine are not regarded as satisfactory. The choice of which of the remaining mining methods can or should be used is based firstly on rock mechanics considerations, namely, will the ore cave satisfactorily with reasonable fragmentation which does not require excessive secondary blasting and will the hangingwall cave with an undercut area which does not exceed the plan size of the orebody? If the caving of the ore produces satisfactory fragmentation and the hangingwall will cave, then block caving is chosen. If the ore fragmentation is not satisfactory but the hangingwall will cave then the ore has to be prebroken. Finally, if the hangingwall will not cave then the ore has to be open stoped.

CHAPTER FOURROCK CLASSIFICATION AND INSTRUMENTAL  
TECHNIQUES USED IN THIS INVESTIGATION4.1 INTRODUCTION

While it has been recognised for a long time that there is a definite need for a more accurate way of describing rocks for Rock Mechanics, it is only recently that a satisfactory classification system has been developed. None of the rock classification systems which have been proposed from time to time appeared to have application to the rocks encountered on Shabanie Mine or to be justified from the point of view of the amount of work required to determine the rock class merely for descriptive purposes. The classification system that has been adopted was based on Bieniawski's "Engineering Classification of Jointed Rock Masses" which has been substantially modified by Laubscher (1974).

Although this is a very recently developed technique, it has been possible in some instances to classify portions of the rock in the backs and peripheries of caved blocks which are now inaccessible. This has been possible because, since 1965 practically all borehole cores drilled for geological and rock mechanics purposes have been photographed on 35mm colour slides, with approximately 7 to 8m of core per slide. Further, the borehole cores drilled for rock mechanics purposes in Blocks 7AB, 7/1 and 7/3, and 16 logged in detail by the writer. Underground exposures, now inaccessible, have been related to areas of similar rock type, structures and fibre value which have been classified.

A wide range of instruments have been employed to monitor the effects that the mining operations have on the surrounding rock mass, and on the extraction and other development openings below and in the peripheries of the mining operation. As indicated in the first chapter, part of this investigation was the testing and development of instrumental techniques suited to monitoring the displacements and deformation of the jointed rock mass encountered on Shabanie Mine. While a few of the

instrumental techniques used are accepted procedures, most of the techniques were developed or adapted to suit the ground conditions encountered on this mine, by the writer or by members of the Rock Mechanics Department under his direction. The techniques have in most cases proved very successful and have made it possible to obtain a far more accurate and extensive picture of ground behaviour. Wherever possible the design of the instruments have been kept as simple as possible for reliability and also to keep them easy to read. This has been done at the expense of some accuracy in most cases, but is considered to be justified. Experience acquired on early installations has been used to improve later installations of some instruments.

As much of this dissertation is based on the results obtained from these instruments and as comparisons are made of ground behaviour from one area to another full descriptions of the rock classification system and instrumental techniques used are given in this chapter.

#### 4.2 GEOMECHANICS CLASSIFICATION OF ROCK MASSES

In an investigation by the writer in 1972 into the relationship of damage to the development workings and the sublevel shrinkage faces at Havelock Mine, Swaziland, it became apparent that the location and severity of damage could not be simply related to the location with respect to the sublevel shrinkage faces alone. The size of excavation, the ground competency, and the presence and orientation of weak slips also had to be considered. Therefore, a point system was devised to assess the relative importance of each in a given situation which in addition could give an indication of the type of support which would be required (Heslop 1972).

The "Engineering Classification of Jointed Rock Masses" proposed by Bieniawski in February 1973 (Bieniawski 1973a) was similar in approach to that proposed by the writer but more detailed. In Bieniawski's original proposals eight parameters were to be assessed individually into one of six classes and a rating assigned to each. The overall classification would be determined by the aggregate ratings of the eight parameters. The eight parameters were:

- (1) The rock quality designation (RQD) as defined by Deere (1967)

- (2) Weathering and alteration
- (3) Intact rock strength
- (4) Spacing of joints or bedding
- (5) Strike and dip orientations
- (6) Separation of joints
- (7) Continuity of joints
- (8) Ground water inflow

In a subsequent paper, the number of classes were reduced from six to five (Bieniawski 1973b).

The geomechanics classification system adopted on Shabanie Mine was developed from these proposals by Laubscher in collaboration with the writer and the other members of the Rock Mechanics Department and the Geological Departments of Shabanie and Gathis Mines. During this period Laubscher and the writer had discussions with Bieniawski on several aspects of the classification system. As a result of these discussions Bieniawski had modified his original classification to some extent, reducing the eight classification parameters to six, still subdivided into five classes. This reduction was achieved by dropping the "weathering" parameter as this aspect was adequately covered by the reduced RQD, intact rock strength and altered condition of joints, and by combining the 'joint separation' and 'joint continuity' into a 'joint condition' parameter. The greatest difference between the system adopted for Shabanie and Gathis Mines and Bieniawski's (1974) proposals is that Bieniawski has retained a 'strike and dip orientation of joints' parameter, while it is preferred on Shabanie Mine to apply a series of adjustments to an in situ classification to arrive at an 'adjusted classification' for each specific application. These adjustments take into consideration the strike and dip of the joints in relation to the excavation, the weathering potential, the effects of the regional stress field and stress changes the structure will experience.

This classification system has, like Bieniawski's, five classes covering all variations in ground conditions from extremely poor to extremely good. The class is defined by a points rating assessed from five basic, readily measurable parameters of a rock, the rock quality designation, intact rock strength, spacing of joints, the condition of

joints and the presence of ground water. The points rating reflects the relative importance of each of the parameters (See Table 4,1). The full rating scale is 100 and each class has a range of 20 points. The classes may be subdivided into upper or lower subclasses, for example Class 3A, having a rating of between 50 and 60. If it is desired, a rock may be described as Class 3 (55), the figure in parenthesis being the rating.

It should be noted that for the purposes of rock classification, Bieniawski's definition of the term 'joint' has been adopted with one modification. A joint is any discontinuity which may be technically a fault, slip, bedding plane, fibre seam, or joint which extends from one joint to another, or in an underground exposure has a length exceeding the width of the drift. In a borehole core, a joint is any discontinuity which can be clearly seen to be a fault, slip, bedding plane, fibre seam greater than 15mm in width or a joint which by the presence of some alteration, greater than 10mm in width, can be assumed to extend from one joint to another. This definition specifically excludes minor natural fractures such as those shown in Fig 2,9, and in this aspect differs from Bieniawski's definition which includes all features however minor they might be. It is considered that the RQD adequately caters for these features.

#### 4.21 THE FIVE PARAMETERS

4.21.1 Rock Quality Designation:- The rock quality designation (RQD) as proposed by Deere et al (1967), is a modified core recovery procedure. The RQD is determined in the same way as conventional core recovery determination modified to incorporate only those pieces of hard, sound core which are greater than 100mm in length. Thus closely spaced shearing, fracturing, or bedding planes are reflected in a lower RQD. Weathering may also result in a lowering of the RQD as it is usually accompanied by an increase in friability of the rock and in the number of fracture planes. Weathering is also catered for in an adjustment (The Potential Weathering Adjustment) which is applied to the insitu classification of the rock.

Bieniawski (1974) recommends that the boreholes should be drilled with N-size double tube corebarrel with a non rotating inner tube, to ensure that a high standard of drilling is obtained. On Shabwade Mine, however, the standard core size is BXL, and is drilled with double tube

TABLE 4.1 QUANTITATIVE CLASSIFICATION OF JOINTED ROCK MASSING

Class	1		2		3		4		5	
	A	B	A	B	A	B	A	B	A	B
Fracture	100-81		80-61		60-41		40-21		20-0	
Jointing	Very good		Good		Fair		Poor		Very poor	
1	100-91	10-76	70-59	65-50	55-46	45-36	35-26	25-16	15-6	5-0
	20	10	15	13	11	9	7	5	3	0
2	300	199-181	180-161	160-141	140-121	120-101	100-81	80-61	60-41	40-21
	75	14	13	12	11	9	7	4	2	0
REFER FIG. 4.2										
3	30	20	15	12	10	8	6	4	2	0
	Joint wall rock	Very hard	Hard	Hard	Hard	Hard	Hard	Hard	Hard	Hard
4	Joint surface	Very hard	Hard	Hard	Hard	Hard	Hard	Hard	Hard	Hard
	Joint surface	Very hard	Hard	Hard	Hard	Hard	Hard	Hard	Hard	Hard
5	Water	or completely dry								
	Rating	10	7	4	0					

corebarrel. A high standard of drilling is required and obtained for evaluation purposes as the fibre is usually the first to be lost by poor drilling. Fig 4,1 illustrates the principles of the RQD measurement.

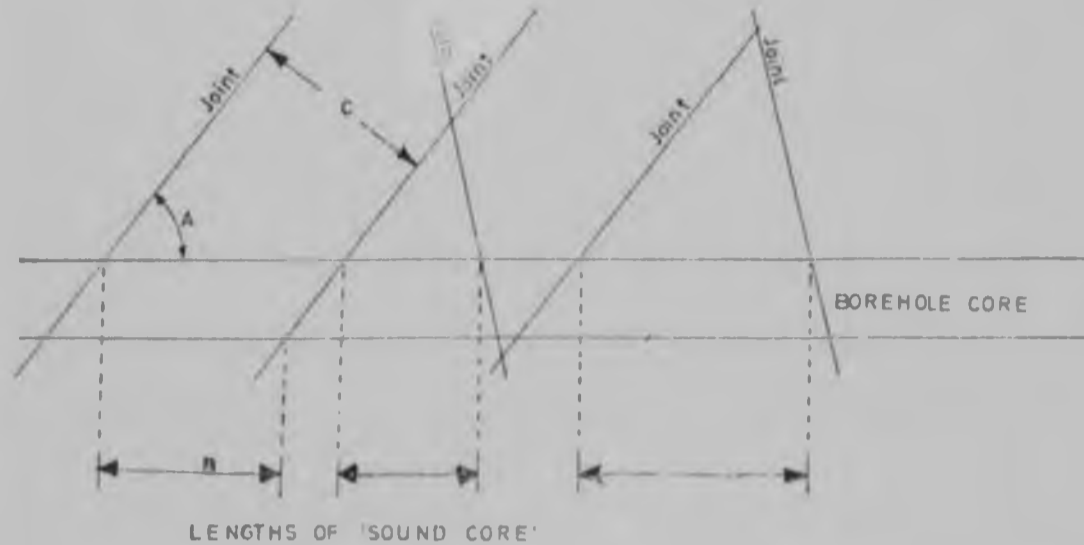


Fig 4,1 Illustration of definition of 'sound core' for RQD purposes

In a rock with a regularly spaced subparallel joint system, the RQD is dependent on the direction the borehole is drilled in with respect to the joint orientation. But by defining 'sound core' as that length of core which has not been cut by a joint, the effect of orientation can be ignored for all angles of intersection greater than  $30^\circ$ . Table 4,2 illustrates this point.

To classify rock from an underground exposure it is necessary either to estimate the RQD or to adapt the RQD measuring procedure to rock surfaces. This has been done by (i) defining a sampling line on the rock surface by means of a tape suspended along the surface and (ii) by using a dummy 100mm length of BXL core to simulate the core measuring technique.

In assessing the RQD from a surface exposure experience with borehole determination of this parameter is needed. Care must be taken to avoid blasting fractures and bedding or shear planes which would not break in a borehole core.

TABLE 4,2  
EFFECT OF BOREHOLE ORIENTATION WITH RESPECT TO THE JOINT SYSTEM ON RQD

Angle between the core axis and joint (A) (See Fig 4,1)	Fracture spacing along axis (B)	True spacing for 100mm sound core (C)	Remarks
90	100	100	} Acceptable
80	107	105	
70	115	108	
60	122	106	
50	133	102	
40	147	95	
30	168	86	} Increase length of sound core to 230mm
20	208	72	
10	320	56	

4.212 Intact rock strength:- The intact rock strength (IRS) is the average unconfined compressive strength of the rock between fractures. A series of compression tests were carried out for various rock types encountered on Shabanie Mine, and the results tabulated (Table 4,3). This table was referred to in determining the IRS parameter in rock classifications done to date, however, it should be noted that an error in calculation made some of the average compressive strengths quoted too high, but the error was in general large enough to affect the ratings.

Recently Taylor (1975) has conducted further tests of intact rock strength (Table 4,4). He has suggested that in cases where the strength of the rock can be visually assessed at higher or lower than the mean, the rating equivalent to a strength one standard deviation higher or lower than the mean should be used.

4.213 Joint spacing:- As defined above in 4.2 a joint includes any fault, slip, bedding plane, fibre seam or natural joint which extends from one intersecting joint to another, or has a length exceeding the width of the excavation.

TABLE 4,3  
INTACT ROCK STRENGTH PARAMETER

ROCK TYPE	IRS (MPa)	RATING
Orebody	55	4 - 6
Orebody hangingwall (Partially serpentised)	69	6 - 8
Immediate hangingwall dunite	103	8 - 9
Hangingwall dunite	155	13
Weathered dunite	69	6 - 8
Black fibre zone	96	8
Brittle fibre zone	172	12 - 13
Talc schist	31	3 - 5
Talc carbonate rock	47 - 69	4 - 7
Diabase dyke	206 - ?	15
Peridotite - partly serpentised	155	12
Actinolite - feldspar rock	207	15
Gneiss	274	15

TABLE 4,4  
REVISED INTACT ROCK STRENGTH PARAMETER

ROCK TYPE	RANGE	IRS (MPa)		NO OF TESTS	IRS RATING		
		MEAN	STD DEVIATION		MEAN	+ 1 STD DEV	- 1 STD DEV
Dunites	58-152	96,7	16,3	43	8	9	7
Orebody serpentine Barren	21-81	47	16,3	12			
Fibre-bearing	18-109	47,8	40,0	6	4	7	3
Carbonated dunite and serpentine							
Barren	72-506	263,1	99,9	22	13-15	15	9
Fibre-bearing	58-176	111,6	51,1	7	9	12	7
Soft talc-carbonate rocks							
Non schistose	13-105	34,8	19,4	38	3	7	2
Schistose	7-47	22,4	7,3	19	2	3	2

The joint spacing parameter is best assessed in underground exposures, and the rating is determined from the minimum, the intermediate and maximum spacing of joints, as in most areas there are at least three sets of joints which intersect each other at nearly right angles. Where there are more than three joint sets, only the three closest spaced sets are considered in determining the rating. The chart in Fig 4,2 is used to determine the joint spacing rating by:

- (a) using the minimum joint spacing as abscissa and intermediate joint spacing as ordinate, a point is obtained in the top left half of the chart.
- (b) A line is drawn from this point parallel to the inclined zone division lines to intersect the diagonal line.
- (c) This intersection defines the ordinate and the maximum spacing the abscissa for another point in the lower right hand side of the chart.
- (d) The joint spacing rating is given by the figure in the zone in which this point falls.

For the odd areas where there are essentially only one or two joint sets, the rating is obtained using the top left half of the chart only.

Determining the joint spacing from borehole cores is not easy. Here a joint is any polished or striated natural surface; or a joint with an alteration zone, or filling, or zone of sheared material exceeding 10mm in width; or, a fibre seam exceeding 15mm in width.

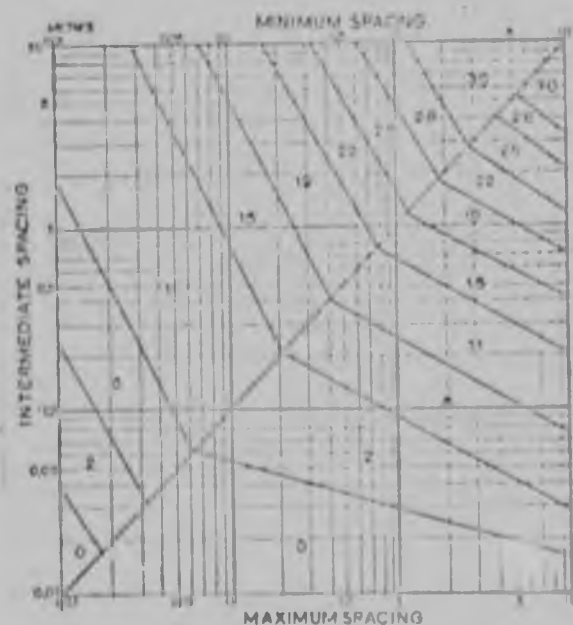


Fig 4,2 Joint spacing parameter rating chart (See text for interpretation)

One and a half metre lengths of core are sampled and the rating determined from the number of joints occurring in each 1.5m section and the orientation of the joints. Unless there is some evidence of a two joint system, three sets of joints are assumed. The ratings are also estimated from the chart in Fig 4.2.

4.214 Condition of joints:- The condition of the joint surfaces in a rock mass play a significant part in the quality of the rock mass. The surfaces may be rough, smooth, or polished. Also the joints may have shearing associated with them, or they may contain sheared or brecciated material or gouge, which reduces the shear strength of the joint, or the joint may contain a cementing material such as magnesite or calcite which increases the shear strength of the joint. Further, the wall rock may have been altered to some degree, the common forms of alteration being the development of chlorite on joints in quartzites and granite gneisses, the development of serpentinisation of joints in dunite, or talc in serpentinite, or weathering in most rock types. The hardness of the joint surface with no alteration is determined by the hardness of the rock mass related to the end members of the intact rock strength scale. Alteration usually results in a reduction in the rating for slips, but in some cases, such as where carbonates have been deposited in the country rock adjacent to a slip, the alteration may increase the rating.

To obtain the ratings in multiple joint systems the weighted average rating of each joint is taken.

4.215 Ground water:- The effect of ground water under pressure, is to reduce the effective stresses on joints. With high water inflow rates, erosion of joint filling material may occur. The former is important in open pit or other slope stability analyses and in underground workings the latter may be important if the area has not been drained by deeper workings.

#### 4.22 ADJUSTMENTS

For the prediction of the stability of and support required for underground headings, or for predicting the cavability of an orebody, etc, adjustments must be made to the insitu classification. The adjustments take into account the effects of the regional stress, and/or possible changes in induced stress, and/or the effects of potential

weathering of the rock over the life of the excavation and the influence on the stability of the excavation of the strike and dips of the dominant structural features.

At this stage, the adjustments to be made to an insitu classification, are at best an educated guess, it is hoped with time to build up a background of case histories which will provide a basis for estimating the adjustments required. It is considered that these adjustments are more realistic in assessing support requirements than Bisniawski's 'stand up time' guide lines.

#### 4.3 SURVEY MONITORING TECHNIQUES

Surveying techniques has played an important part in the monitoring and investigation into the ground deformation and displacement in the peripheries of, and below caved blocks. Surveying techniques were used because, with the base stations located in stable ground, they provided data on the size of the zone of influence of a cave mining operation, and total ground displacements. In the latter aspect surveying techniques are superior to instruments used to monitor displacement or deformation, which usually only provide relative displacements.

Survey and levelling traverses were extensively used in the investigation of ground displacement around Blocks 7AB, Blocks 7/1 - 7/3 and in the initial stages of Block 10. In addition triangulation was used to investigate the behaviour two substantial ribs of solid ground left between caving operations on either side.

##### 4.31 TRIANGULATION OF SURFACE BEACONS

The purpose of the surface beacons located on the 3/6 "rib pillar" (See Fig 2,5) and later in the 5/7 "rib pillar" was to investigate possibly bodily movements of the pillars of solid rock lying between two caved areas. In the case of the 3/6 "rib pillar", tilting of the surface of the pillar was also monitored.

Three beacons were established on the 3/6 "rib pillar" and these were triangulated and trigonometrically levelled from two concrete

theodolite stands located on stable ground to the north and west of the beacons. Surveying was done using a single second theodolite. The beacons consisted of a circular black target mounted on a 3m length of 50mm black piping in a 1,5 diameter by 1,8m deep concrete foundation. For tilting measurements the centre beacon had an additional target mounted 3m above the lower target. To minimise sway in the wind guy wires were fixed to the beacons.

On the 5/7 "rib pillar", four similar beacons were erected except that the foundations used were smaller, and white screens were erected behind the beacons to improve visibility. The number of observations used in each survey were reduced to speed up observation.

The accuracy achieved on the 5/7 "rib pillar" beacons was possibly 25% better than the guaranteed accuracies given below:

- (1) Guaranteed positional accuracy (ep)

$$ep = 7.3 \times 10^{-6} L \operatorname{Cosec} C$$

where: L = length of base = 996,034m

C = Apical angle =  $89^{\circ}37'$

$$ep = \pm 7,3\text{mm}$$

- (11) Guaranteed Vertical Accuracy (ev)

$$ev = \pm 22 \times 10^{-6} \cdot S$$

where S = mean length of sight = 707,294m

$$ev = \pm 15,5\text{mm}$$

- (111) Accuracy of tilt of signal pole = 5 minutes of arc

In the 5/6 "rib pillar" the accuracies achieved were not as good as in the 3/6 pillar because of the reduced number of observations.

The time required to make the observations and calculate the results was about 24 days. The only problems encountered were of a minor nature, such as the initial settlement of the foundations after the first heavy rains, and the flexibility of the centre 6m beacon in moderate winds.

#### 4.32 SURFACE SURVEY TRAVERSES

The purpose of the surface survey traverses was to investigate the subsidence and lateral displacement of the ground above and in the peripheries of potential caved areas. The lines of survey pegs

were established on the surface, extending from beyond the anticipated limit of ground displacement to above the block cave.

Two traverses were used to investigate the surface ground in the peripheries of Block 6. These traverses were again used in Block 7AB. Later, over the eastern side of Blocks 7/1 and 7/3 another survey traverse was laid out to monitor ground subsidence and displacement.

These lines on survey pegs were established at a large angle to the boundary of the block and extended from well beyond the anticipated limit of ground movement.

The survey pegs consisted of 15mm round iron rod, 450mm long, with a 3mm diameter axial hole drilled in one end. The rod is cast in a concrete block 450mm in diameter so that about 150mm protrudes from the surface. The concrete block is founded at a depth of 200-300mm below the surface.

The traverses were surveyed at intervals using a single second theodolite, and distance measurements were done using a steel tape. No corrections were made for tape temperature variations or catenary effects, the former being the larger effect and the most difficult to correct for.

The accuracy achieved was estimated to be:-

Angular error in seconds =  $\pm 7 \sqrt{\text{Number of angular measurements}}$

Linear error in metres =  $0,0115 \sqrt{\text{Number of distance measurements}}$

Errors in latitude and departure depend upon the individual

traverse.

#### 4.33 UNDERGROUND SURVEY TRAVERSES

Underground survey traverses were used to determine the nature and extent of ground displacement in the peripheries of Block 7AB. These traverses subsequently extend to cover the displacements induced by the mining of Blocks 7/1 and 7/3.

The pegs used for the underground survey traverses were constructed from 1,8m lengths of 19mm drill steel which had at one end a number of weld drops on it and a 50mm diameter collar approximately 300mm from that end. This end of the steel was grouted into a 1,8m

long, 50mm diameter hole drilled into the solid rock above the back of the drift. Welded to the lower end of the drill steel was a 19mm x 38mm long chain link with a 1,6mm diameter hole drilled in it. A 25mm diameter pipe placed over the peg during grouting ensured that the peg was free from the sidewall and was not affected by spalling from the back of the drift (Fig 4,3).



Fig 4,3 Underground Survey Peg

The positions of the survey pegs were fixed by traversing from a base station assumed to be in stable ground, using a single second theodolite, and targets which were positioned vertically below the 1,6mm diameter hole by optical plumbing. Distance was measured from target to target by a steel tape suspended in catenary from two supports jacked between the hangingwall and footwall of the drift.

The optical plumbing proved very time consuming and so from 1968 plumb bobs were used instead, with relatively little loss in accuracy. Elevation differences obtained from levelling were used for reducing the slope distance to horizontal distances.

The accuracy achieved in Block 7AB was:-

$$\text{Angular error in seconds} = 6,9 \sqrt{\text{Number of Angular measurements}}$$

$$\text{Linear error in metres} = 0,00096 \sqrt{\text{Number of measurements}}$$

less than 50m

$$+ 0,00131 \sqrt{\text{Number of measurements}}$$

exceeding 50m

Errors in latitude and departure depend on the individual traverse.

The largest problem encountered was tape stretch which necessitated re-calibration of the tape at intervals from a standard tape, which in time also became stretched. This may not have affected the accuracy of Block 7AB but it did affect the accuracy of the surveys in Block 7/1 and 7/3.

The pegs used for the survey traverses were levelled using an automatic level set up halfway between levelling pegs on pre-marked locations. Reflective levelling staves graduated in 0.01ft suspended from the survey peg chain links.

Subsequent to the mining of Blocks 7/1 and 7/3 when the change over to the metric system came, a specially manufactured reflective staff graduated in 1cm units was used in conjunction with a parallel plate micrometer.

The accuracy achieved in levelling was  $0.56 \sqrt{\text{Number of set ups}}$  (mm) before the introduction of the parallel plate micrometer. No estimate of the accuracy using the micrometer is available.

While no major problems were encountered with this technique, surveying of electrified haulages was inconvenient as the work had to be done on Sundays when the power could be switched off.

Survey traverses were discontinued for a variety of reasons such as the difficulty in ensuring that the base pegs were in stable ground with the limited access available, the ability of borehole extensometers and remote displacement meters to provide similar information, the uncertainties introduced by tape stretch.

#### 4.4 INSTRUMENTAL TECHNIQUES FOR MEASURING DEFORMATION AND DISPLACEMENT

The instrumental techniques have been subdivided into three groups, those that measure deformation or displacement around the major openings are dealt with in this section, while those used to measure deformation in and around development openings, and other stress and deformation monitoring or measuring techniques are in subsequent sections.

With the exception of the commercially obtainable instruments, modifications and improvements of some of the locally designed instruments has continued over the years. In the design of these instruments, the philosophy has been to keep the instruments as simple as

practical. This policy has resulted in some loss of accuracy but this is considered to be justified, as more frequent readings can be taken over a wider area.

Some of these instruments are now preferred to survey techniques in around deformation monitoring, because they are simple to read and can be read more frequently, allowing trends to be recognised early.

The readings obtained from the instruments in practice were not as accurate as theoretically possible, because malpractices and short-cuts crept into the reading methods. Although some of the modifications introduced were designed to curb these malpractices, even with these other malpractices crept in. The most common and difficult to eradicate malpractice occurred in instruments requiring repeated measurement to obtain an average, where readings were selected or the instrument slightly adjusted to produce results which displayed a high degree of agreement between them. If less honest, these results not only looked better, but they also made calculation of averages easier. An extreme example of this practice is given in the results of closure-meter No 7/9 during a stable period (See Table 4,5). It can be seen from these results that the six readings taken on each day make a high degree of agreement with a low standard deviation. Contrary to what might be expected if the readings were completely impartially done, there was a larger variation in the weekly means than in the individual results on any one day and this is borne out by the higher standard error of the means.

From the point of view of analysing the results afterwards, the decreased accuracy is of little consequence, but it is essential to the recognition of acceleration trends for practical short term ground behaviour prediction. Therefore, in some instruments repetitive readings were replaced by four readings taken from four different bench marks, where possible involving reading in opposite directions.

TABLE 4,5

TABLE OF READINGS, MEANS AND STANDARD DEVIATIONS OF CLOSURE METER NO 7/9

Date	1st Reading	2nd Reading	3rd Reading	4th Reading	5th Reading	6th Reading	Mean	Std Dev of the 6 readings
16.2.74	26,0	26,1	26,0	25,9	26,1	26,1	26,03	0,08
25.2.74	26,1	26,4	26,3	26,3	26,0	26,2	26,22	0,14
4.3.74	26,9	26,7	26,8	26,8	26,6	26,8	26,77	0,10
11.3.74	26,0	25,6	25,7	26,3	25,8	26,3	25,95	0,30
18.3.74	26,0	26,2	26,6	25,9	26,1	26,0	26,13	0,10
25.3.74	26,0	25,8	25,7	25,8	26,0	25,8	25,85	0,12
1.4.74	25,6	25,7	25,8	25,7	25,6	25,6	25,67	0,08
8.4.74	26,4	26,0	26,1	26,2	26,3	26,2	26,20	0,14
15.4.74	27,7	27,7	27,6	27,6	27,7	27,8	27,68	0,06
22.4.74	26,3	25,5	26,0	26,2	26,1	25,8	25,98	0,29
29.4.74	26,3	26,2	26,4	26,1	26,1	26,2	26,22	0,14
Mean				26,24				
Std deviation of Means				0,56				

## 4.41 REMOTE DISPLACEMENT METERS

The remote displacement meter is used for the same purpose as levelling, that is the monitoring of subsidence. They have the advantage that they may be located in areas directly above an undercut which would become dangerous for levelling. In addition, by installing them in boreholes otherwise inaccessible areas may be monitored.

This instrument is essentially very simple, consisting of 12,7mm plastic hose, with one end fixed to a suitable competent portion of sidewall in an area where it is desired to monitor elevation changes, and the other end with a glass tube attached, fixed to the sidewall in a stable area. The hose is filled with water and caused to overflow at the remote or "open" end, and the resulting overflow water level is measured in the glass tube at the measuring end (See Fig 4,4). The lower end of the glass tube was fitted with a small plastic disc with a 0,9mm diameter hole which acts as a restrictor to slow the rate of flow of water down during topping up operations. Without this restrictor, it was found that the water in the hose gathered so much momentum during the topping up, that the water continued to overflow for a while

after the water level in the glass tube had subsided below level of the "open" end.

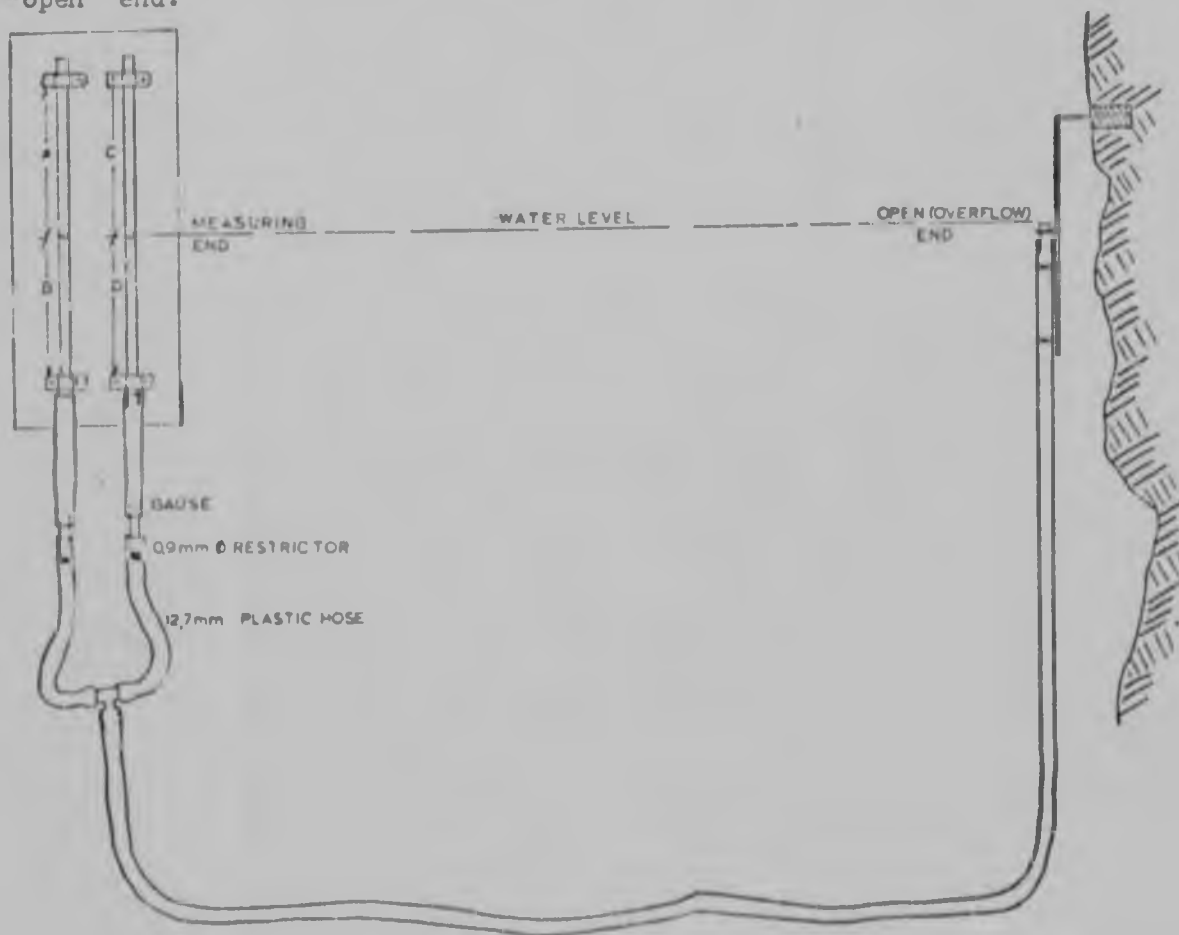


Fig 4.4 General arrangement of Remote displacement meter

The "open" ends were either secured to the sidewall by wire grouted into the sidewall by epoxy resin putty, or in boreholes by a wire clamp. One or more glass tubes would be attached to a wooden board attached to the sidewall at the measuring point. The hose would be filled by syphoning clear water through them.

The main modifications introduced over the years were:-

- 1) The glass tubes were replaced by clear plastic hose in Block 7AB, but these became discoloured with time and glass tubes were reverted to in subsequent installations.

- 2) For Blocks 7/1 and 7/3, and Block 16 the restrictor was placed in a hoseender in the hose some distance below the measuring board to obviate the need to remove the glass tube to clean the restrictor.
- 3) The wire clamps were left off two Block 7/2 borehole remote displacement meters, but were re-introduced when it became evident that without them the hoses slowly slid out of the hole.
- 4) Two glass tubes were attached to each meter each with its own restrictor, and connected to the hose by means of a T-piece. This was done to provide a check on the readings and which would show up blockages in the restrictor. At the same time the method of reading these instruments was modified. (See below) Introduced for Block 7/2.
- 5) The open ends were fixed to the sidewall by means of a 6,3mm round iron L-piece grouted into the sidewall in Block 7/2.
- 6) Instead of siphoning, initial filling (not topping up) was done by means of a bottom discharge drum attached to the water supply, the purpose of the drum being to remove air bubbles from the water supply.
- 7) Iron hoseenders were replaced with plastic because some of the longer term installations were rusting and a 200 mesh screen was placed over the restrictor to prevent the ingress of dirt into the restrictor.
- 8) The wooden glass tube mounting boards were replaced with metal, and fitted with fixed scales.

The reading procedure is relatively simple, using a plastic wash bottle all the remote displacement meters measured at one site are topped up with water. Within two minutes all excess should have overflowed at the open ends, and the water level in the glass tubes should be stationary. The water levels are initially read two minutes after topping up. Readings were in the earlier installations made by measuring the distance between the top saddle holding the glass tube and the bottom of the meniscus, using a scale in millimetres and on the

later installations from scales fixed next to the glass tubes. Three to five readings were taken and the results meaned.

The second glass tube was introduced and the reading system changed to provide a check that the restrictor was not partially blocked or had an air bubble trapped below it and to eliminate repetitive readings. The method of reading was to fill the first tube, and read from the top saddle down to the meniscus, (b) refill the first tube and read from the bottom saddle up to the meniscus, (c) fill the second tube and read from the top down, (d) refill the second tube and read from the bottom upwards. The four readings were numerically different and provided cross checks on each other. In practice, there was a tendency to cut out some of the refillings of the tubes and as a result these checks would show fine agreement, but the readings could still be faulty. This meant that the additional work involved in checking the results was frequently a waste of time.

The fixed scale was introduced after it was recognised that the extra work involved in checking the results was not worth the effort.

The readings obtained from remote displacement meters can be affected by changes in the difference in air pressure between the two ends. These effects are measurable if the measuring and open ends are in different ventilation compartments (in which case they tend to be erratic), or are in a drift along which air is flowing at a fast rate, exceeding 0.7m/sec. along a 100m long remote displacement meter. In general pressure corrections are either too small to warrant correction or too erratic to make reliable corrections.

No corrections were applied for temperature variations, as usually these affected the whole length of the hose. This does not, however, apply to boreholes, where the portion of hose in the borehole would not be subject to the variations in temperature that the portion in the development openings might be subject to. The temperatures experienced underground varied by less than 3°C from summer to winter and the maximum head of water for a meter located in a borehole was about 6m. For a 3°C change in temperature effect on the water level can be calculated from the densities of water at various temperatures, and would be about 4mm. In general, the temperature variations were considerably

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No corrections were applied for temperature variations, as usually these affected the whole length of the nose. This does not, however, apply to boreholes, where the portion of nose in the borehole would not be subject to the variations in temperature that the portion in the development openings might be subject to. The temperatures experienced underground varied by less than 3°C from summer to winter and the maximum head of water for a meter located in a borehole was about 6m. For a 3°C change in temperature effect on the water level can be calculated from the densities of water at various temperatures, and would be about 4mm. In general, the temperature variations were considerably

less than this. Readings of remote displacement meters in a stable area and measured over three years and not compensated for temperature had a variance of less than 0,9mm which indicates that the instrument can be quite accurate even without correction for temperature and air pressure.

#### 4.42 BOREHOLE WIRE EXTENSOMETERS

Borehole wire extensometers can be used for measuring strain, dilation of fractures or joints, lateral movement on joints and subsidence. Unlike most borehole wire extensometers used elsewhere the clamps used on Shabanie Mine are simple and cheap, and the reading and tensioning arrangements also simple but not as accurate as the more elaborate installations used elsewhere.



Fig 4,5 Boyle's rubber sleeve bridging clamp used as a borehole clamp

The first clamps used on the mine were Boyle's Bros rubber sleeve bridging plugs. Designed for securing deflection wedges in boreholes, these have served as inexpensive and adequate borehole clamps. These consisted of a soft neoprene rubber sleeve with a hard rubber cone at one end. This end also had a slightly smaller internal bore. When used as a borehole clamp, a four-hole wire spacer was put into this end, and a washer to which the wire could be secured was attached to the hard cone (Fig 4,5). The rubber sleeve had a diameter 2,5mm larger than that of the borehole, so that when the clamp is pushed into the borehole using an installation tool which is grooved to accommodate the wires of preceding clamps, and which pushes on the hard rubber shoulder inside the clamp, stretching the sleeve longitudinally and reducing it in diameter. When in position and the attached wire is under tension, the clamp is put in longitudinal compression causing an increase in diameter which jams the clamp in position. Each clamp with its wire attached was pushed into position by means of the installation tool attached to a string of either light 19mm diameter conduit rods for

shallow holes or to drill rods for the deeper holes.

The main difficulty experienced was in the installation of more than one clamp in a hole, where a wire from one of the previously installed clamps could easily become caught while pushing a clamp into position. This usually resulted in the wire becoming caught between the clamp and borehole walls, and jamming the clamp in the hole. The jammed clamp could only be removed by drilling and then the whole installation would have to be started from scratch.

The wire initially used was 22 or 24 swg high tensile stainless steel, and passed over a 100mm diameter pulley and fastened to a 45N weights. The pulleys had graphite impregnated bronze bushes 6,3mm in length and 9,5mm in diameter, fitted onto a bright steel shaft 9,4mm in diameter. The measurements were done by means of a vernier caliper between an aluminium reference bar and brass clamps clamped on each wire (Fig 4,6).



Fig 4,6 Pulley arrangement for borehole wire extensometer

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Fig 4,6 Pulley arrangement for borehole wire extensometer

The main modifications introduced with time were:-

- (1) 19 swg Nichrome V 20/80 wire replaced the stainless steel wire as this had a relatively low yield point and could be straightened, when installed in the borehole by straining. This was introduced in October 1970 in boreholes in Block 7/1, Block 16 and one borehole in Block 7/2.
- (2) Aluminium channel section was introduced to provide four different measurements to prevent repetitive reading errors. The arrangement is illustrated in Fig 4,7.

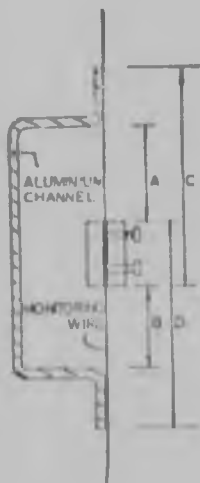


Fig 4,7 Aluminium Channel  
Section reading  
arrangement



Fig 4,8 Metal Borehole Clamp

- (3) Metal borehole clamps were introduced to replace the rubber sleeves as a few rubber sleeve clamps had failed to grip in over-size boreholes, and the metal clamps were potentially simpler to install with the promise that more clamps could be put into a hole. The clamps were constructed from galvanised piping as illustrated in Fig 4,8, were locked in position by jamming a roller into the wedge of the clamp.

Four methods of installation were tried:-

- (a) All the wires and clamps for a borehole were assembled on surface and then lowered down the borehole simultaneously. For lowering, the rollers were held in the clamps by light split cotter pins. Each roller was attached to a common wire which extended from the borehole collar to a weight below the bottom clamp. When in position, starting at the bottom clamp, the monitoring wire of each clamp was given a sharp jerk to pull the cotter pin out and lock the clamp in position. It was not immediately apparent that the monitoring wires had become severely intertwined, with the result that some of the clamps were not satisfactorily locked in position, and when ground displacement started, the displacement of one clamp was transmitted to all the wires. An attempt was made to keep the wires separate by installing spacers at between 3 and 6m intervals, but these twisted out of alignment and jammed the clamp wires instead of keeping them separate and free.

Eight clamps were installed in each of two boreholes in Block 16 and in four holes in Block 7/2 using this method. In two more holes in block 7/2, four clamps were installed. Stainless steel wire was used for the installations because of a temporary shortage of Nichrome. These installations were not regarded as successful and where possible the clamps were removed and replaced. Only two boreholes could be re-equipped in Block 7/2 and only the upper portions of the two Block 16 holes.

- (b) In the first attempt to replace the above procedure each clamp had its own weight and the roller was kept in position by a split cotter pin. The clamps were lowered individually but as the pins came out early in the only installation tried, the method was not regarded as being suitable.
- (c) In the second attempt rollers were kept in position in the clamp by freezing the clamps and rollers in blocks of ice. The clamps had ice-free channelways for wires from the preceding clamps. Four clamps were lowered into position

individually and when the ice melted the clamp locked into position and the Nichrome wire tensioned to remove kinks before the next clamp was lowered. This was used in SldM 100, Block 7/2, and was completely successful.

- (d) The method now preferred is to use rods to push the clamps into position with the rollers kept in position by means of split cotter pins which are individually attached to a heavy gauge wire. When in position the cotter pin is pulled out by the heavy gauge wire retracted from the borehole. The clamp is then locked in position and the rods withdrawn. The rods have pins attached to them so that if necessary the clamps can be rotated so that the clamp lies on top of the roller and can grip satisfactorily. This technique has been used with complete success.
- (4) By attaching remote displacement meters to the weights on borehole extensometers, the instruments could be installed in potentially dangerous areas and read in a safe area. Introduced October 1972 for DD 987 in Block 7/2.
- (5) As a result of investigations by D.G.F. Hedley (1970) and after experiments on the mine, it was concluded that these instruments could be made more sensitive to ground movements by keeping the wires under minimum tensions and increasing the weights at the time of reading. A 4,6N weight is kept on each wire and this is increased to 50N when a reading is made. The tension is increased on each wire in turn and a reading taken. Several readings are made. A modification to the pulley wheels introduced at the same time, was the mounting of the wheels on a rotating 50mm long shaft with reduced diameter bearing surfaces at the ends to reduce the frictional force on the pulley. Also introduced at the same time was a fixed scale measuring arrangement (Fig 4,9). Some of the Block 7/2 installations were modified to incorporate these improvements.

The accuracy of these instruments was dependent on several factors such as the frictional forces involved in the pulleys, and between the borehole sidewall and wires. It is estimated that ground movements of up to 10mm could occur without being registered on the early installations. With the repeated tensioning technique and other improvements the accuracy (of this technique) is now estimated to be of the order of 2mm.

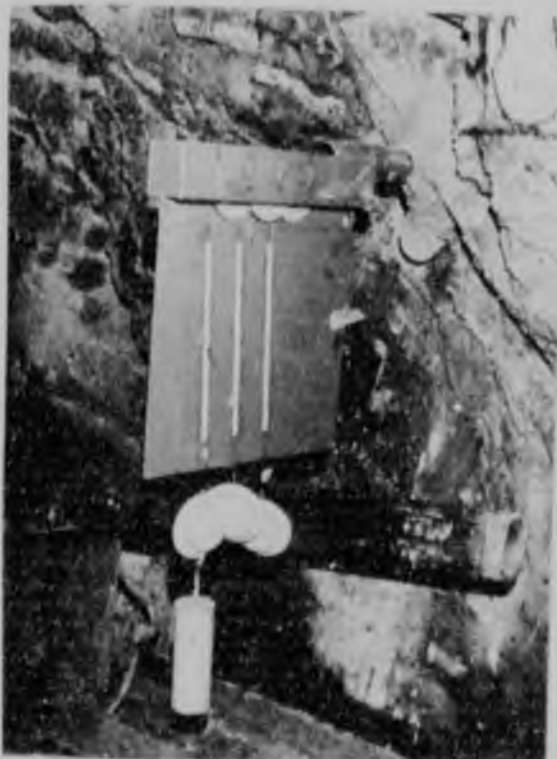


Fig 4.9 Fixed scale reading arrangement for borehole wire extensometer

#### 4.43 WIRE EXTENSOMETERS IN DEVELOPMENT HEADINGS

The function of wire extensometers in underground drifts is the same as that of borehole wire extensometer, that is to monitor strain or displacement on slips, generally in a horizontal direction.

The wire extensometer consists of a wire anchored to a pin in the rock at one end, with the other end over a pulley system at which measurements of ground movement may be made. The difference lies in the wire extensometers which are installed in development openings and not boreholes, and the anchors are 19mm drill steels grouted into holes

drilled into the sidewall of the heading. These instruments have only been used in the peripheries of Block 16, and the pulleys were all of the same type, with impregnated bronze bushes fitted to bright steel shafts, and the measurements were from horizontal length of aluminium channel section.

A recent development of the extensometer is the use of a pulley system similar to that in the borehole wire extensometer, with an extended pointer attached to the pulley wheel for greater precision readings. No installations of this type are discussed in this dissertation.

#### 4.5 INSTRUMENTAL TECHNIQUES FOR MEASURING DEFORMATION IN AND AROUND DEVELOPMENT OPENINGS

The three instruments used for monitoring deformation and displacements in and around drifts are basically all mechanical extensometers, which differ in size, design, and to some extent in application. Used for measuring deformation around development openings, or displacements on slips or cracks, the absolute values of the strains or displacements are of little importance, but a great interest lies in the changes in rates of deformation which have proved to be related in time to other events such as the onset of caving etc. Because of repetitive reading errors the tendency has been to drop the use of closuremeters and the Demec extensometer and, where possible, to use dial extensometers.

##### 4.51 CLOSUREMETERS

These instruments were generally used to measure the changes in dimensions of development openings in any direction. They were also used in some cases to measure the rates of displacement of major slips.

The closuremeter is a portable extensometer which fits between two permanently installed bench marks to take a reading. The extensometer consists of three concentric aluminium tubes with a steel ball bearing mounted on each end. The instrument is adjustable with one foot and one inch adjustments. A spring-loaded section takes up smaller adjustments which are measured by means of vernier or dial calipers (See Fig 4,10). The permanent bench marks were 900mm long, drill steels especially finished ends grouted into the far ends of two jack-hammer

holes drilled opposite each other into the roof, floor or sidewalls of development opening so that the holes were in as nearly as possible a straight line. A Portland Cement grout was used, to grout the drill steel into the end of the hole. It is established that the grout covered about a third of the length of the drill steel.

The closuremeter is inserted between the drill steel ends and the measurements taken of the course graduations, and by means of a vernier or dial caliper on spring loaded fine adjustable section. The readings are taken in three positions around the closuremeter at  $120^{\circ}$  to each other, and then another three with the extensometer taken out and re-inserted in the opposite direction. The standard error of the weekly means of six readings usually is better than 0,6mm.

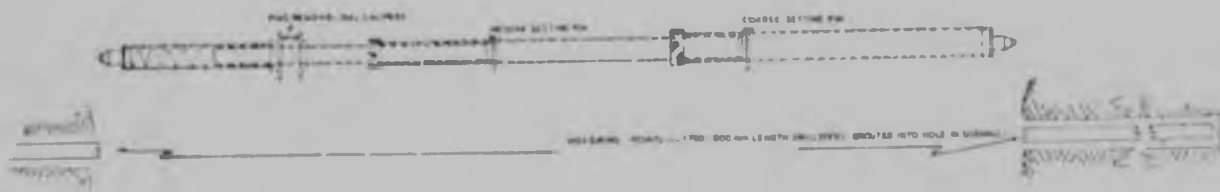


Fig 4.10 Closuremeter

#### 4.52 DEMEC EXTENSOMETERS

The Demec extensometer, a commercially available mechanical strain gauge, was used in some applications as a strain gauge, but it was mainly used in monitoring displacement on slips and openings up of cracks.

The extensometer consists of a dial gauge mounted on an invar bar, with a fixed sharp steel point at one end and at the other a movable arm, pivoted on a knife edge, which has a sharp steel point on one end and a flattened surface on the other. The dial gauge plunger

bears on this surface (Fig 4,11). The instrument is used to measure the distance between two stainless steel discs mounted with epoxy resin glue or putty on the object to be monitored. These discs have punch marks into which the sharp steel points fit. The discs have to be mounted exactly 203,2mm apart, in an arrangement which was varied according to the purpose it was used for. Strain measurements were not attempted on rock surfaces underground, as it was considered that the instrument was not sufficiently sensitive. It was, however, used as a strain gauge in monitoring the loads on timber setts, and also in determining rock properties from borehole cores.

In monitoring displacements on cracks, the arrangement was initially simply three points mounted in an isosceles triangle spanning the crack (Fig 4,12). The results were plotted to form a trace of the movement of the apex with respect to the other two points. It was found that for the most reliable readings the discs should all be mounted in the same plane. Two or three readings in each direction on each leg were taken and the results for each leg averaged. This gave a standard error of 7 divisions (70 macro strains) or 0,014mm.

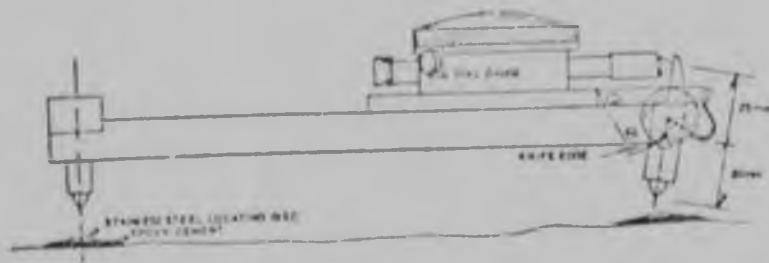


Fig 4,11 Delec extensometer

Later an arrangement of the discs to form a  $45^\circ$  rosette replaced the isosceles triangle arrangement (Fig 4,12). The  $90^\circ$  apical angle was introduced to facilitate plotting, and the  $45^\circ$  measurement to act as a check. In general this arrangement was very successful. It was

introduced over a period between December 1969 and August 1970.

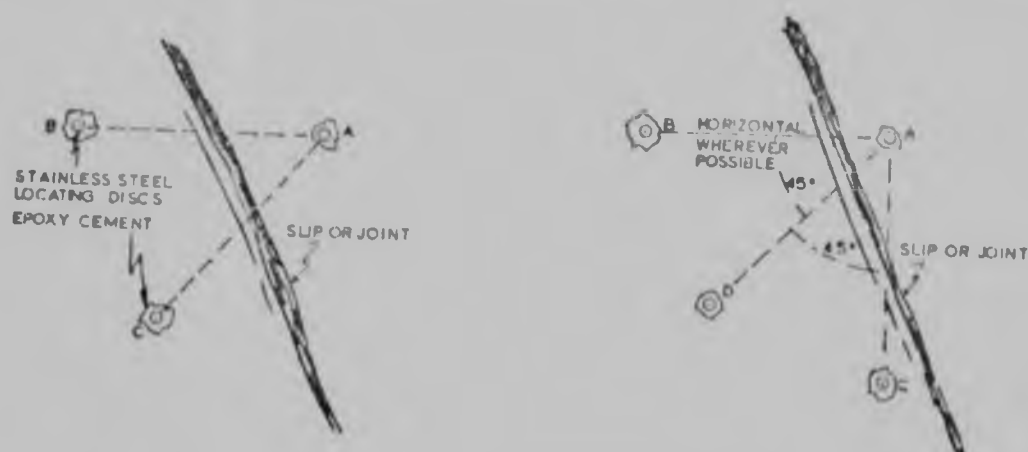


Fig 4,12 Demeo extensometer - two and three point reading arrangements

The measuring points were extremely cheap and easy to install and consequently the number of installations grew until there were too many to read and analyse properly. The selection of the correct slip or crack and correct position on them for monitoring proved to be an important but difficult aspect. A high proportion of the measuring arrangements spanned the wrong portion of the slip as an adjacent, often less likely looking joint displayed all the movement, or were mounted on a portion of side wall which became dislodged by the movement, or were incorrectly positioned to measure the displacement on the slip. These problems were magnified when in the second phase of Block 7AB and in Block 7/1-7/3 attempts were made to select slips for monitoring prior to mining; so that a complete record of the displacements could be obtained, and very few records were obtained from these.

Primarily to facilitate measurements in awkward places, dial calipers fitted with sharp points were used instead of the Demeo extensometer for a period from June 1971, for all rosette arrays, but the accuracy was very disappointing and the technique was discontinued in January 1972.

#### 4.53 DIAL EXTENSOMETERS

The dial extensometer's prime use is for monitoring strain or displacement on slips or shears.

It consists of two lengths of square section tubing, one fitting inside the other. Both are mounted by suitable bolts on the rock surface and both have sections cut away where two rubber pads can be mounted. A wire pointer is placed between the pads and in the event of movement, rolls between the pads (Fig 4,13). The device is provided with a graduated dial, and a "Perspex" face. Readings are taken simply by noting the position of the pointer.

These instruments are theoretically capable of being read to 0,01mm over a length of 1m, or 10 micro strains but this accuracy is probably not achieved in practice because of the flexibility in the mounting of the instruments. An accuracy of 0,05 is probably attainable.

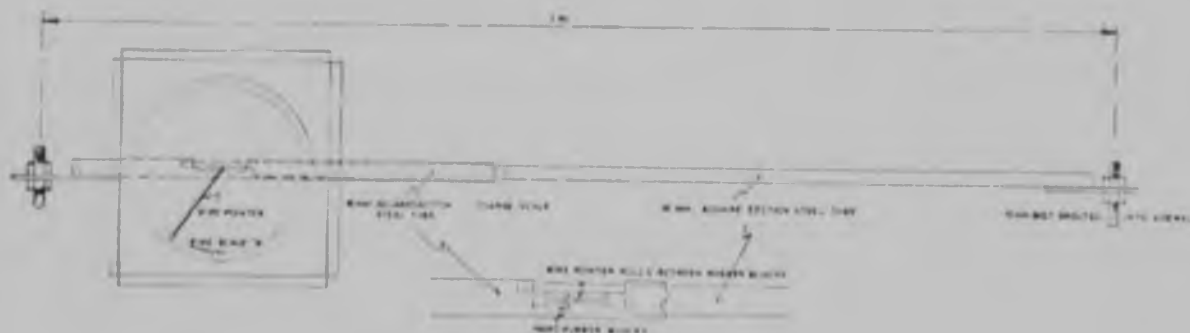


Fig 4,13 Dial Extensometer

#### 4.6 MISCELLANEOUS INSTRUMENTAL TECHNIQUES

Included in this section are a number of instruments designed to monitor specific items or aspects.

##### 4.61 BOLT TENSION METERS

For monitoring the tension on expansion shell rock bolts commercially available photo-elastic rock bolt dynamometers (or bolt tension

meters), made by Horstman Ltd, Bath, England were used. The instrument consists of a steel sleeve 50,8mm long and 57,1mm in external diameter, with a wall thickness of 12,7mm. The bolt passes through two holes drilled diametrically opposite each other allowing the sleeve to be diametrically loaded (Fig 4,14). When the dynamometer is loaded, a photo-elastic pattern is developed in the glass disc mounted in the end of the dynamometer which can be seen when the instrument is lighted and viewed through a circular polariser. The bolt tension meters are installed with a cup washer and spherical seat between the meter and the rock surface, and another spherical seat washer between the meter and the nut on the rock bolt to minimise eccentric loading conditions.

The usual practice in reading these instruments was to view and light the meters through a simple hand viewer (circular polariser) and compare the pattern obtained with a set of standard patterns (Fig 4,15).

The results of compression testing thirty meters were as follows:

Number of Fringes	Mean Load
1	23kN <sup>+7kN</sup>
2	39kN <sup>+7kN</sup>
3	52kN <sup>+8kN</sup>
4	63kN <sup>+9kN</sup>

Repeated testing of one meter showed that the testing procedure was accurate to within 5kN.

Further tests of a meter loaded without a thick washer between the meter and the rock bolt nut, showed that the loads required to produce the same fringe orders were approximately 1,1 times the load required when a thick washer was used.

Also the meters have proved to be fairly sensitive to eccentric loading conditions. It is estimated that the instruments will under mining conditions have an accuracy of  $\pm 6kN$ . (99% Confidence limits).

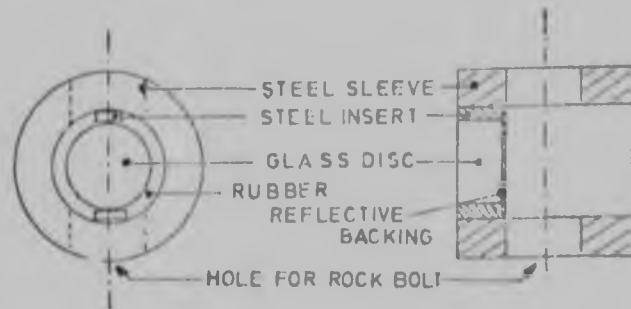


Fig 4,14 Bolt tension meter

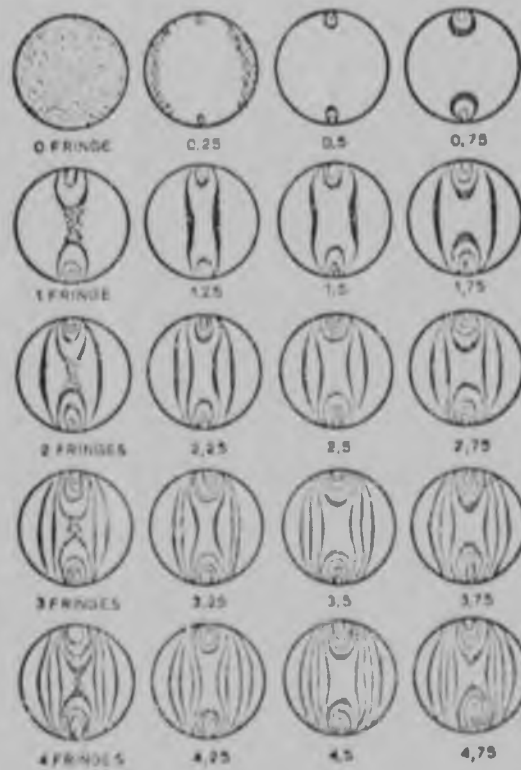


Fig 4,15 Standard fringe patterns for reading bolt tension meters

#### 4.62 STRESS-METERS

Horstman photo-elastic BB 150 glass plug stress meters were used in an attempt to measure charges in stress. The meter consists of a cylinder of optical glass with an axial hole, together with a light source, polariser and quarter wave plate which is grouted into an EX borchole using a carborundum filled epoxy resin cement (Fig 4,16).

before installing the meter, the borehole cores were examined to determine the most suitable ground, and having decided this, the hole is filled with required spacers. The resin is introduced into the hole in a metal container with a wooden plunger, followed by the stress-meter. Using an installation tool the stress-meter was gradually pushed backwards, forcing the plunger to displace the cement and squeeze the cement around the stress-meter.



Fig 4.16 Section through a photo-elastic stressmeter installation

The readings were done by a Rock Mechanics Technician on a weekly basis using a "Precision Hand Viewer" (a compensating polariscope). Interpretation of the photo-elastic pattern developed in these stressmeters proved to be difficult in most cases as seldom were the stresses high enough to take the pattern out of the first order greys, and seldom were patterns symmetrical and recognisable. The technique adopted, therefore, was to sketch the pattern seen and to record the changes in them. Under these circumstances, the directions of the major principal stress could generally not be determined with confidence, and similarly the ratios of the major principal stresses could in most cases not be determined, either by comparison with the standard photographs as suggested by Roberts et al (1964) or by measuring the separation of the zero points as suggested by Baran (1965). The major principal stress could therefore only be determined on about two of the stressmeters installed.

#### 4.63 CAVING INDICATORS

The caving indicator is a simple device used to determine the position of a cave back from a borehole drilled to intersect it. This

was originally a 304,8mm long lead-filled 25mm pipe with a hole drilled diametrically through it halfway along its length. A short length of copper wire was threaded through this hole and tied to a length of heavy gauge fencing wire. The device was lowered down the borehole and when it entered the cavity it was pulled up against the back. When a fall occurred, the copper wire would break and fencing wire could then be retrieved and another indicator attached and again lowered into the cavity. The wire would be measured each time to give a record of the position of the back (Fig 4,17).



Fig 4,17 Caving Indicator

The first modification was to increase the length of the weight, to 600mm to prevent the device being caught inside the borehole and with difficulty in getting lead, 25mm round iron was used instead. Finally it was decided not to leave the indicator wire down the hole, but to lower it down, measure the depth of the back, and then break the copper wire and withdraw the wire, to minimise the risk of the wire being wedged in the hole and breaking somewhere near the top which would make subsequent installations impossible. A light steel cable is now preferred to the fencing wire.

A high degree of accuracy is not required for the study of the rate of caving. It is estimated that the accuracy instrument is better than 0,5% of the depth. The main source of inaccuracy lay in the elastic stretch in the wire which tended to retain some of its coil and kinks.

#### 4.64 LOAD CELLS

Three load cells were designed and installed, and later a further two load cells were installed, with the object of measuring loads imposed on crown pillars by caved ground under draw. The load cells were constructed from three 203,2 x 101,6mm rolled steel joists (RSJ) set parallel to each other at 203,2mm centres and 101,6mm deep in a concrete foundation, and pinned to the crown pillars with old drill steels. The cells were covered with a 600mm wide, 12,7mm thick steel coverplate bolted on only at the ends. Electrical resistance strain gauges were cemented onto one side only of the webs of the RSJ's at 203,2mm intervals in the initial three load cells. Philips PR 9244/04 was used for cementing the strain gauges on and they were insulated with Philips PR 9244/05 water proofing compound. Copper wire leads of 18 swg were lead through boreholes in the crown pillar to the grizzly drive where the readings were made. The load cells are illustrated in Figs 4,18 and 4,19. The readings were made with a Huggenberger Tepic strain indicator.



Fig 4,18 Section through a load cell

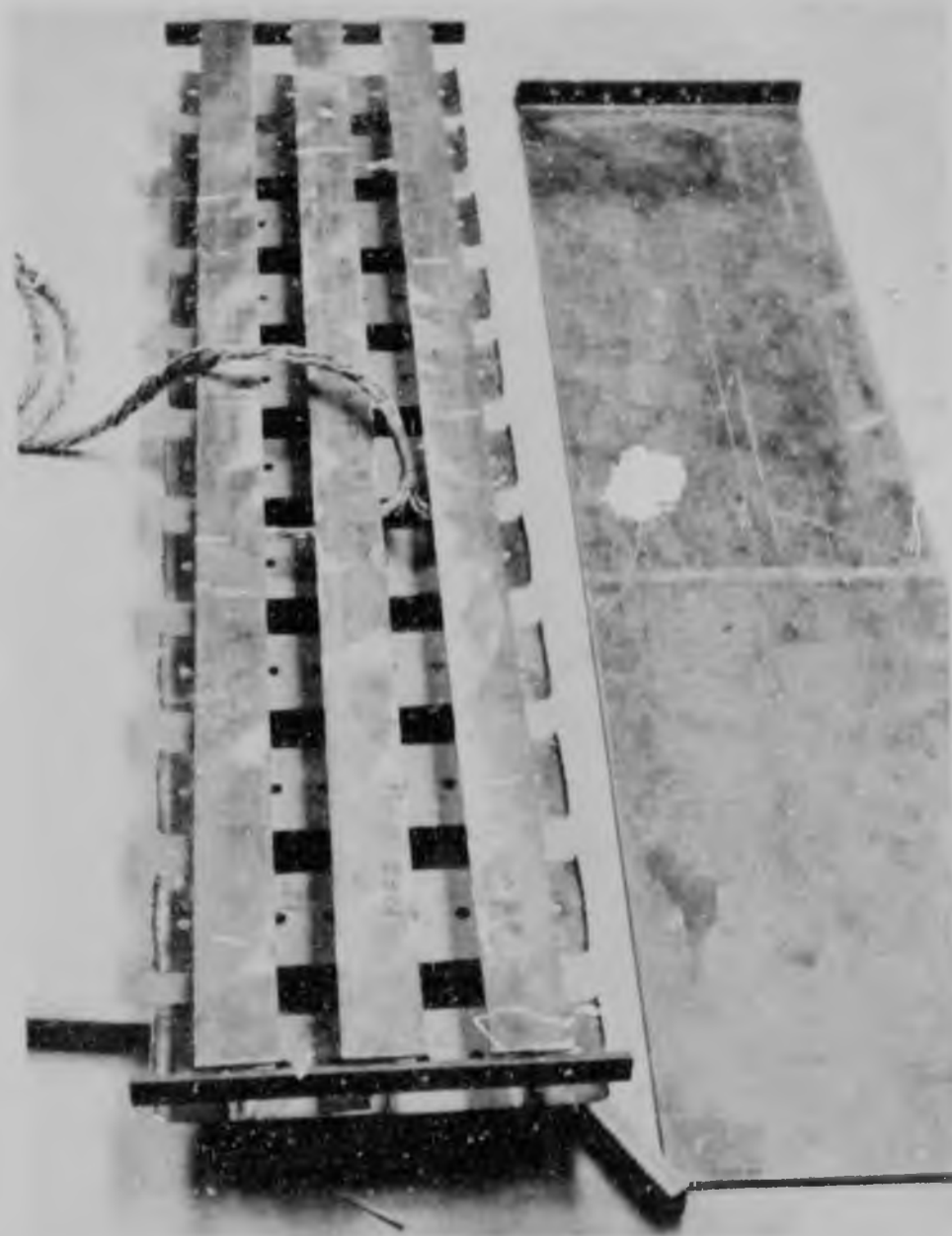


Fig 4,1, Loadcell

Two cells, 1,2m long were installed on the top of the crown pillar and one 2,4m long was installed on the side of the crown pillar above 7/9 W drawpoint in block 7A3.

The arrangement of strain gauges on only one side of the RSJ's proved to be inadequate in the initial load cells. It had been assumed that buckling of the RSJ webs could not take place without relative

movements of the RSJ and the cover plate which would be restrained by friction between the two. This deficiency was rectified in the two load cells installed subsequently, over 5/3E drawpoint. Other minor modifications used here included the use of Philips PR 9248 waterproofing compound and Hottinger Baldwin waterproofing compound, and the setting of the load cells tops flush with the surrounding concrete. Switch boxes were also installed to facilitate readings, but this was only introduced towards the end of the life of these load cells.

The calculation of the loads on the first load cells without making assumptions proved impossible due to the inadequate numbers of strain gauges. A model load cell was built and tested under various combinations of loads, and it was concluded that an estimate of the loads could only be obtained if it was assumed that a limited number of point loads were imposed on the load cells. The assumed positions of the point loads, are illustrated in Fig 4,20. From the calibration tests it was shown that point loads acting in these positions were related to the strain readings by the following empirical formulae:

$$\begin{aligned}
 L1 &= + 2,25 S1 && \text{Where } L1 \text{ to } L7 \text{ and } S1 \text{ to } S3 \text{ are} \\
 L3 &= - 4,5 S1 \text{ or } - 4,5 S2 && \text{as illustrated in Fig 4,20} \\
 L5 &= - 4,5 S2 \text{ or } - 4,5 S3 \\
 L7 &= + 2,25 S3
 \end{aligned}$$

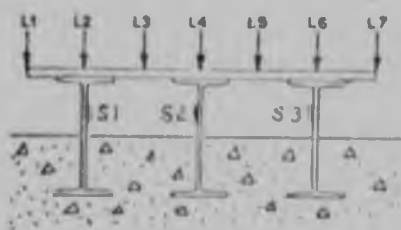


Fig 4,20 Assumed position of loads on a load cell

The second set of load cells did not provide any usable results because a breakdown in either the insulation of the gauges or in the gauge cement which resulted in a meaningless drift in the gauge readings.

#### 4.65 ARCH LOAD INDICATORS

To monitor the loads imposed on TH yielding arches an arch load indicating device was developed. The design requirements were that the gauges could be read without special equipment and could withstand blasting shock, fumes and moisture. These simple devices were constructed of 304,8mm long hacksaw blades attached to the sides of the arch by means of tin-smiths' rivets cemented into holes drilled into the arch. The blades were set with a 3mm bow in them and any changes in strain in the arch was reflected by a change in the amount of this bow. This was magnified by wire pointers made on a jig to ensure uniformity. The pointers rested against a stainless steel bearing plate and the readings were taken on a scale attached as in Figs 4,21 and 4,22. The whole assembly was protected from blasting damage by 151,4 x 76,2mm channel iron cover boxes (Fig 4,23). Two indicators were placed on the straight leg portion of the arch, one on either side on the neutral axis. No modifications have been introduced to date.

The devices were calibrated by fixing two devices to a short length of arch and putting it into a compression testing machine. In this machine it was shown that the readings were unaffected by buckling in one direction, and taking the average readings of the two indicators buckling in the other direction could be accounted for.

The accuracy of the indicators is estimated to be:  $\pm 5kN + 5\%$  of the scale reading. The arch load indicator is a difficult device to install, requiring a high degree of accuracy. It is very light and flimsy but requires a heavy cover box to protect it from secondary blasting damage. The mounting arrangements for the boxes proved to be inadequate, allowing the box to become displaced with secondary blasting, and imposing some bending force on the arch. Further the scales proved difficult to read, and in the humid mine atmosphere, the scales, which were painted onto stainless steel brackets, softened and became easy to wipe off during cleaning. Many instruments were rendered useless in this way.

#### 4.66 PHOTO-ELASTIC DISCS STRESS MEASUREMENT TECHNIQUE

This technique was used for the determination of absolute stresses in rock by sticking a biaxial photo-elastic disc strain gauge to the

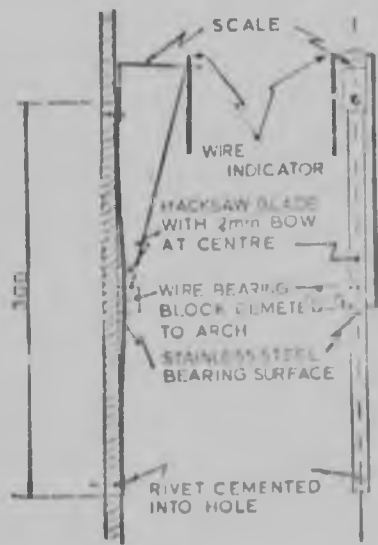


Fig 4,21 Arch load indicator - sectional and front view

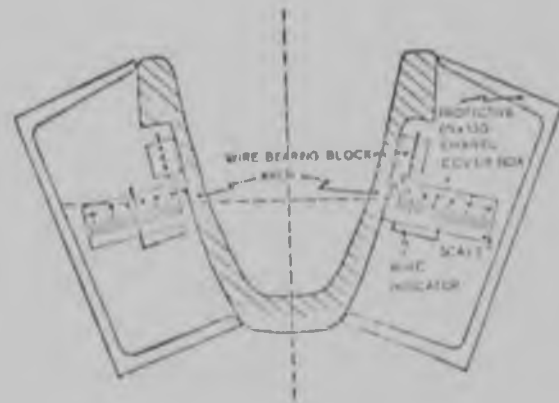


Fig 4,22 Arch load indicator - top view

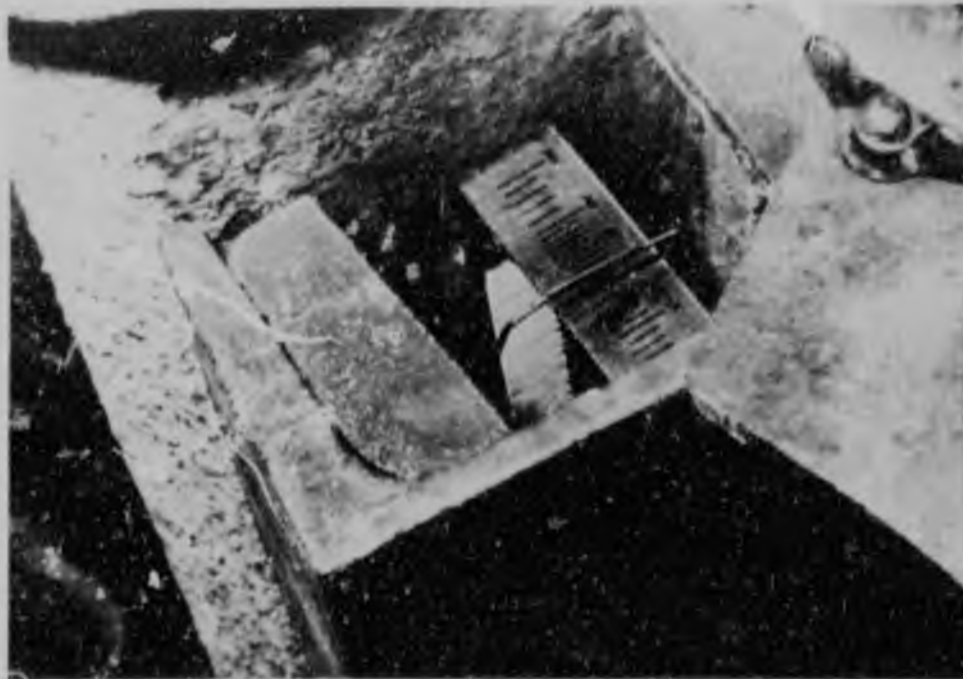


Fig 4,23 Arch load indicator

flattened end of a borehole and over coring it. The disc, made by Horstman's, is made of birefringent plastic 3,18mm thick and is 44,5mm in diameter with a 7,6mm central hole (Fig 4,24). The central portion of the gauge is backed by a reflective coating and a patch of non-stick plastic. The outer is free of reflective coating and plastic and can be bonded to the rock. Changes in strain in the rock are transmitted by this bond and concentrated around the central hole. When viewed in polarised light a fringe pattern can be observed and using a precision hand viewer (circular crossed reflection polariscope) the fringe order can be measured fairly accurately. It has been shown that the fringe order is directly proportional to the major principal strain and that the spacing of the isotropic points depends on the ratio of the major to minor principal strains.

To measure the stress in the rock the gauges were used to measure the strains developed by over coring the flattened end of a NX borehole (Fig 4,24). As there is a difference in the co-efficients of thermal expansion of the disc and the rock, large errors can occur with variations in temperature. Great care must therefore be taken to read the discs at the same temperature as at the back of the hole. For this purpose, the temperature of the drilling water is read as well as the temperature of the air in the borehole. On withdrawal, the overcored disc is read immediately if the ambient temperature and that in the borehole are sufficiently close. If not the piece of core is replaced in the borehole and left until a stable temperature is attained. After the readings has been obtained, the disc was usually warmed and this provided a useful check on the principal stresses ratio.

Where the complete state of stress was desired, three, preferably mutually perpendicular, boreholes were drilled. Approximately ten measurements were made in each hole, starting at 0,9m from the collar. Measurements done close to the tunnel sidewall had to be adjusted for the stress concentration caused by the tunnel, but were virtually unaffected by the stress acting in the direction of the borehole.

The gauges were calibrated by Cook (1968), by sticking one onto

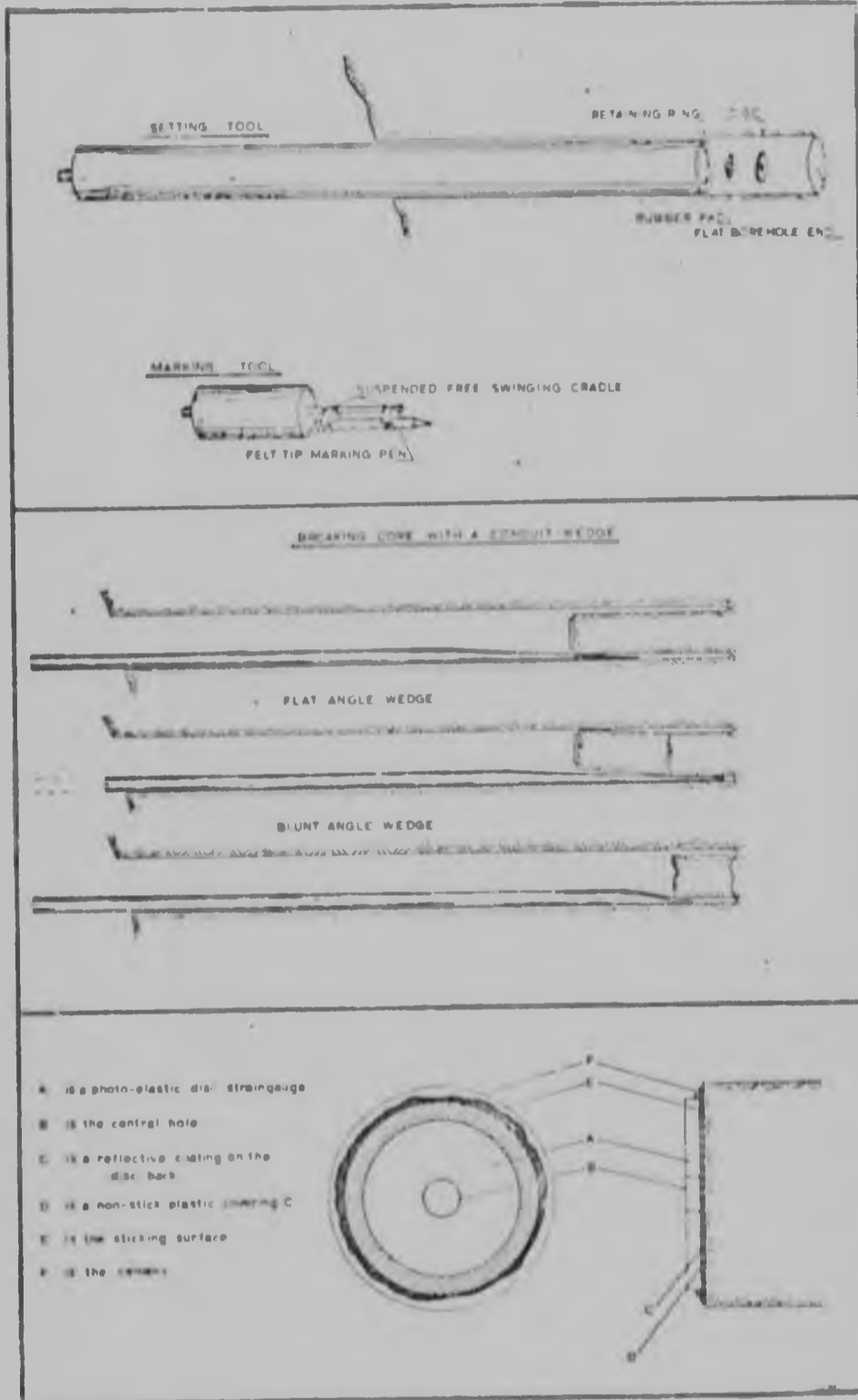


Fig 4,24 Photo-elastic disc for stress measurement

each side of a calibration cross with an electrical resistance stress gauge rosette stuck onto the one side. With this device it was possible to impose both tensile and compressive strains and to measure them accurately. It was in addition, a valuable aid to interpreting fringe patterns. The gauge sensitivity factor so obtained of  $450 \times 10^{-6}$ , was in good agreement with published values. Cook claimed that an accuracy of 0,05 fringes could be realistically obtained. This corresponds to 22,5 micro strains. The accuracy of this technique as applied to rocks on this mine is discussed more fully in the following chapter.

#### 4.7 DISCUSSION

The system of rock classification outlined in this chapter has been accepted by both the technical personnel and operating officials on both Shabanie and Gaths Mines. The classification system is being used not only for descriptive purposes but with adjustments to the insitu classification to cater for specific situations, it is now being used to decide the type and amount of support that a development heading may require.

In general the results of the survey traverses and surface beacon triangulations can be said to have contributed towards the understanding of ground behaviour. The surface beacons indicated the Block 3/6 "rib pillar" was tilting and breaking up at a time when it was found that this mass of rock would remain intact and by tilting en masse shearing off at the back it would destroy the workings beneath it. The survey traverses in particular showed the extent of the zone of influence of a block cave as well as indicating the magnitudes and directions of displacement within it. As indicated earlier, survey traverses have been replaced by instruments to a larger degree. The main reasons for this was that the zone of influence was so large that stable base stations could only be found far into the footwall, and therefore the traverses had to be long and consequently less accurate. Further, the frequency at which surveys could be done was limited by the amount of work involved and in practice were done at best at monthly intervals and at worst three monthly or longer. Instruments on the other hand could be read weekly with minimal amount of work, providing a more accurate rate of

displacement curve on which short term predictions could be based.

The two instruments used for large scale deformation monitoring, the remote displacement meter and extensometer have proved to be extremely valuable tools, and with the several refinements the accuracy and reliability of the instruments has been greatly improved. These two instruments have largely replaced surveying traverses.

The closuremeter was successfully used for monitoring the deformation of grizzly drifts as the instrument sites were overcut, and were also used to monitor the displacement on slips. The Demec extensometer was used almost entirely for monitoring the displacement on slips opening of cracks etc, and for a period the results were satisfactory. But with the increase in number of installations which increased the number of incorrectly placed installations and also contributed to an increase in the number of reading errors, and the general slowing in rate of displacement the results became less satisfactory. The results obtained from the six month trial period is the use of dial calipers in place of the Demec extensometer were not satisfactory. This initiated the development of the dial extensometer to replace the Demec extensometer. The dial extensometer was made longer and therefore capable of spanning several subparallel joints, permanently installed and therefore the results less subjective. To date the dial extensometer has proved an extremely satisfactory tool.

Among the miscellaneous techniques used, the simpler techniques were the most successful. Perhaps the simplest is the caving indicator and is regarded as being a satisfactory device. The bolt tension meters are fine for measuring the tensions on rock bolts, but do not respond to stress changes prior to damage becoming visible, and cannot be used as early warning devices. The glass plug stress meters are in general not sensitive enough and are difficult to read. There is a definite need for a stress monitoring device and work is in hand developing a borehole deformation gauge for this purpose. As a stress measuring device the photo-elastic disc is well suited to our variable stress-field being cheap and quick to install and allowing many readings to be taken. It is however difficult to read at low stress levels and would improve by increased sensitivity. The first load cells used for

monitoring to loads imposed on the crown pillars had an inadequate number of strain gauges, and in the later load cells the strain gauge insulation broke down, but some interpretation could be made of results from the initial installations. The arch load indicator was developed for use on a grizzly horizon, and has been fairly successful, but it is a difficult device to install, and further work is required to increase its ability to withstand secondary blasting and the humid atmosphere.

In conclusion it may be stated that the techniques used for rock classification and rock deformation monitoring are practical and adequate. Further work is, however, needed on devices for monitoring stress in both the rock mass and in support such as the yielding arches.

CHAPTER FIVESTRESS MEASUREMENTS5.1 INTRODUCTION

As Shabanie Mine is situated in ancient highly folded rocks, similar to rocks elsewhere in the world which have exhibited high lateral stresses, a programme of virgin rock stress measurements was undertaken. The first measurements were made by the South African Council for Scientific and Industrial Research (CSIR) under contract at three sites on the mine using the "doorstopper". Although this programme was dogged by problems with strain gauge adhesion, sufficient overcorrections were obtained which indicated a high lateral stress, which prompted further stress measurements by the Shabanie Mine staff, and the CSIR to return to try out their new triaxial cell at the site which had previously indicated the high lateral stress.

The first CSIR measurements were carried out at sites as remote from the mining operations as possible so that the measurements were not affected by the stresses induced by the mining operations. However, it was realised that the geological structures and rock properties could influence the stresses and therefore it was decided to site all future measurements as close as possible to the orebodies, even at the expense of entering the zone of influence of the mining operations.

The purpose of the stress measurement programme was to establish the stress levels in the rock before mining started. While it was realised it was not possible to completely avoid the stresses induced by mining operations it was decided to accept this error. Later, two small specific stress measurements were undertaken to measure the change in stress associated with the mining operations.

5.2 STRESS MEASUREMENTS - METHODS AND RESULTS

Stress measurements using the CSIR "doorstopper" were attempted by the CSIR under contract in February and July 1966. The three sites selected were all on 29<sup>th</sup> level, one near the vertical shaft, one in 23W

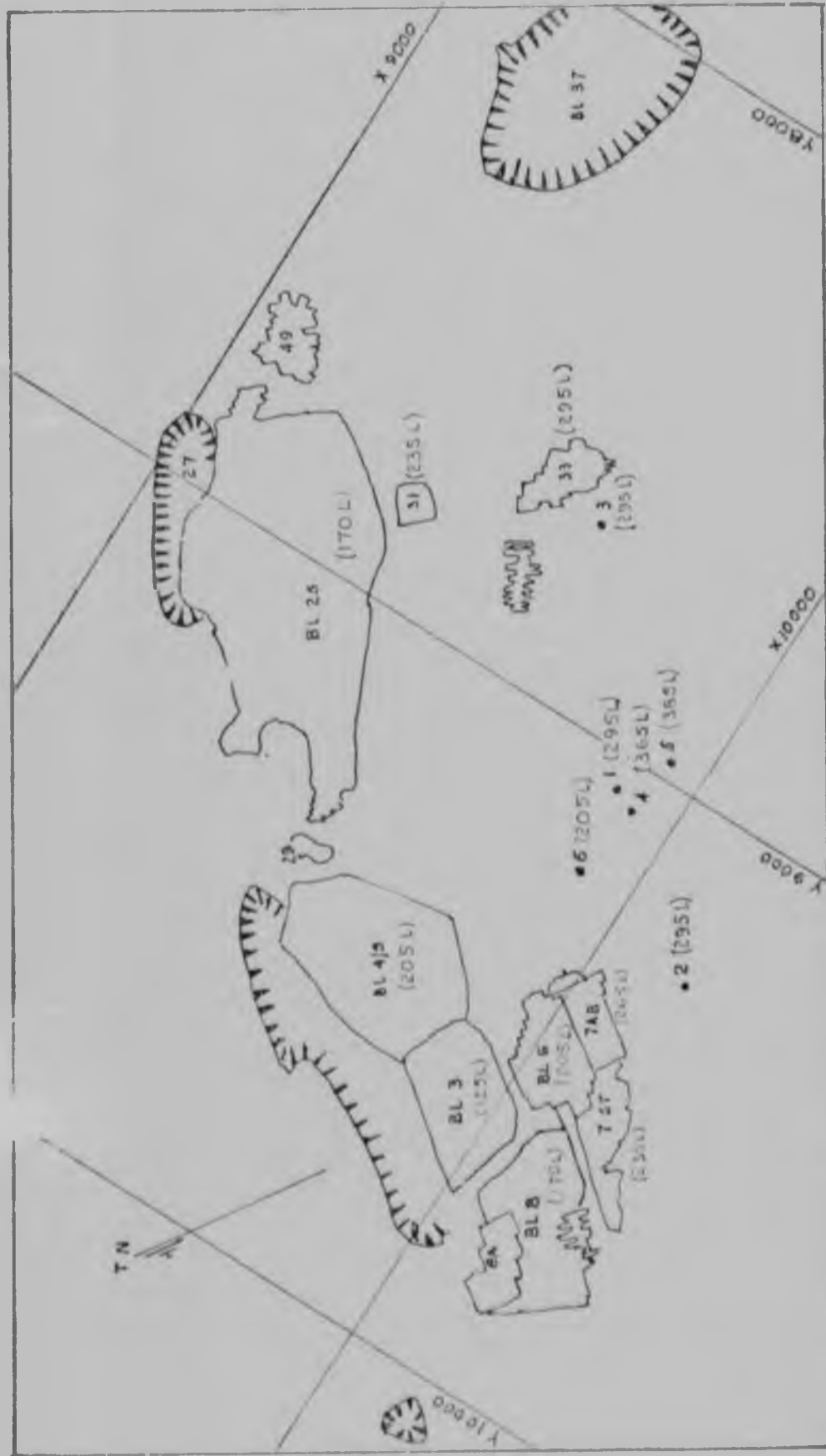


Fig 5.1 Plan showing stress measurement sites in relation to mined area

crosscut Block 16, and one in 6/7 crosscut Block 50. At each site three boreholes were drilled in different directions, and about 27 overcorings were attempted. Of these only six were successful, one in each of the boreholes at the first two sites. The high number of failures was due to cementing the cells on partings of the rock and to poor adhesion of the strain gauge cement to the damp rock. The results based on so few readings were not considered reliable but they indicated a high lateral stress and as mentioned earlier, this spurred on further attempts at stress measurements.

#### 5.21 THE TRIAXIAL STRAIN CELL MEASUREMENTS

As part of the test programme of the triaxial strain cell the CSIR did five stress measurements in a borehole drilled from the southern end of 23W crosscut on 295 level Block 16 in January 1968. This was the site where the highest lateral stress was obtained from the "doorstopper" tests. The triaxial strain cell technique has been described by Leeman (1969) and by van Heerden (1968). Briefly, the technique involves drilling a small Ex (37,6mm diameter) hole coaxially with, and from the end of a NXC (92mm hole and 68,5mm core diameter) hole. At a position in the Ex hole (selected from the core of that hole) the triaxial strain cell is installed. This strain cell has three electrical resistance strain gauge rosettes mounted in rubber and located in a metal body. When the body is in position the strain gauges are pushed against the sides of the Ex hole by compressed air fed to the body. When the cement is set, the pressure is released and the initial strain measurements taken. The cell is then overcored by deepening the NXC corehole and after retrieving the rock cylinder containing the cell, the strains are measured again. The poor cement adhesion experienced previously was overcome by the use of a sealant (Union Carbide Silane bonding agent A 174). The calculation of the principal stresses has been described in detail by Leeman (1969) and van Heerden (1968). The results of the five measurements made at this site are given in Table 5,1 and the principal stress directions are illustrated stereographically in Fig 5,2 and are discussed in 5.31.

#### 5.22 PHOTO-ELASTIC DISC MEASUREMENTS

Stress measurements using photo-elastic discs were started by Cook

TABLE 5,1

The results of the CSIR triaxial strain cell stress measurements at site No 1 23W crosscut, 295 level. (After van Heerden 1968 P 14). Virgin rock stress measurements.

Measurement Number	Depth Metres	Principal Stresses MPa	Direction Mine Bearing	Dip
1	4,6	$\sigma_1 = 19,2$ $\sigma_2 = 16,0$ $\sigma_3 = 8,9$	262° 160° 43°	-32° -19° -53° <sup>c</sup>
2	6,5	$\sigma_1 = 19,5$ $\sigma_2 = 15,3$ $\sigma_3 = 7,0$	264° 162° 48°	-30° -18° -54°
3	7,3	$\sigma_1 = 14,5$ $\sigma_2 = 8,7$ $\sigma_3 = 9,0$	203° 308° 79°	-25° -27° -51°
4	8,2	$\sigma_1 = 18,9$ $\sigma_2 = 12,9$ $\sigma_3 = 9,0$	164° 256° 67°	-5° -33° -57°
5	9,2	$\sigma_1 = 21,0$ $\sigma_2 = 15,3$ $\sigma_3 = 7,6$	325° 232° 64°	-6° -20° -70°
Mean		$\sigma_1 = 16,5$ $\sigma_2 = 15,5$ $\sigma_3 = 7,9$	180° 276° 64°	-14° -29° -57°

of the Rock Mechanics Unit in August 1967. The first results were very poor but improved as a suitable technique was developed. In December of that year a satisfactory set of results were obtained from the 23W cross-cut sites where, the following month, the CSIR triaxial strain cell measurements were done.

This technique requires three boreholes as with the "doorstopper". The photo-elastic discs are stuck on the flattened end of the NX borehole and overcored (Fig 4,24). The photo-elastic disc technique has been described by Hawkes and Moxon (1965) and with the modifications used here, was outlined in the preceding chapter. The first overcorings in each hole are done at about one metre of the collar, where still in the zone of influence of the crosscut, the stresses in the plane of the flattened end are influenced by the development, but the stresses in the direction of the borehole are very small. From these first measurements in the horizontal holes an estimate of the stress acting in the direction of the vertical hole are obtained and similarly estimates of the stresses acting in the directions of the other holes are obtained. Approximately ten overcorings were made in each hole, and a weighted average of the readings were taken. From these averages the principal stresses were calculated. The comparatively large number of overcorings from each hole are required, because the variation in Young's modulus of the dunite and serpentine zones gives a rapid and marked variation in stress along the length of the boreholes, as shown in Table 5,3.

### 5.3 DESCRIPTION OF SITES AND RESULTS

The results of the CSIR triaxial strain cell measurements at Site 1 are given in Table 5,1, and the results of the photo-elastic disc measurements at the first five sites is given in Table 5,2. The location of the stress measurement site is shown in Fig 5,1 and the orientation of the principal stresses from the triaxial cell measurements are shown stereographically in Fig 5,2 and the photo-elastic discs in Fig 5,3.

The triaxial cell measurements were made in an horizontal borehole drilled to a depth of 10m southwards from the southern end of a crosscut on 295 level Block 16. The measurements were made in the hangingwall partially serpentinised dunite, 50 to 60m south of the Block 16 orebody and 15 to 25m north of Zone C talc zone.

TABLE 5,2

Principal stresses at various sites on Shabanie Mine. (Modified after Cook 1969)

Site No.	Level	Principal Stresses MPa	Direction Mine Bearing	Dip
1 (A) (Triaxial Cell)	295	$\sigma_1 = 16,5$ $\sigma_2 = 15,5$ $\sigma_3 = 8,0$	$100^\circ$ $276^\circ$ $64^\circ$	$-14^\circ$ $-29^\circ$ $-57^\circ$
1 (B) (Discs)	295	$\sigma_1 = 14,5$ $\sigma_2 = 14,0$ $\sigma_3 = 5,0$	$224^\circ$ $131^\circ$ $334^\circ$	$-10^\circ$ $-24^\circ$ $-64^\circ$
2	295	$\sigma_1 = 17,5$ $\sigma_2 = 13,0$ $\sigma_3 = 9,0$	$129^\circ$ $29^\circ$ $258^\circ$	$-22^\circ$ $-23^\circ$ $-57^\circ$
3	295	$\sigma_1 = 18,5$ $\sigma_2 = 15,5$ $\sigma_3 = 10,0$	$285^\circ$ $193^\circ$ $102^\circ$	$-58^\circ$ $-1^\circ$ $-32^\circ$
4	365	$\sigma_1 = 19,5$ $\sigma_2 = 11,5$ $\sigma_3 = 8,0$	$124^\circ$ $32^\circ$ $258^\circ$	$-11^\circ$ $-12^\circ$ $-73^\circ$
5	385	$\sigma_1 = 24,5$ $\sigma_2 = 16,0$ $\sigma_3 = 9,5$	$295^\circ$ $44^\circ$ $194^\circ$	$-15^\circ$ $-52^\circ$ $-35^\circ$
6	205	$\sigma_1 = 13,1$ $\sigma_2 = 9,7$ $\sigma_3 = 3,1$	$104^\circ$ $232^\circ$ $340^\circ$	$-47^\circ$ $-30^\circ$ $-27^\circ$

These results confirmed the indication from the "doorstopper" readings that the lateral stresses exceeded the vertical stresses substantially. However, no great difference between the lateral stresses in the crosscut and drive directions was indicated by these measurements. In comparing the principal stresses obtained from the five measurements, it is interesting to note that there was a close agreement in the orientation of the minor principal stress, dipping at between  $50^{\circ}$  and  $70^{\circ}$  north west, and in magnitude varying from 7 to 9 MPa. The major and intermediate principal stresses varied considerably in direction within a plane dipping approximately  $25^{\circ}$  south east. The magnitudes of these stresses varied from 8,7 to 21 MPa with no strong directional predominance. However, a very close agreement in direction and magnitude was displayed in the intermediate and major principal stresses for the measurements at depths of 4,6 and 6,2m. The measurements at 8,2 and 9,2m displayed a fairly close agreement too, but compared with the first two measurements the major and intermediate stresses were transposed. For the third reading at 7,3m the major and intermediate stress had intermediate orientation. Van Heerden (1966) attributed these variations in orientation and magnitude to the stress concentration and the end of the crosscut but as the first measurement was made at a depth of twice the width of the crosscut and as the magnitude of the minor principal stress did not decrease significantly with borehole depth, it is unlikely to be the only reason for the rapid variation in direction of the intermediate and major principal stresses.

The photo-elastic disc stress measurements made at this site gave slightly lower stress magnitudes and in orientation displayed closest agreement with the first two triaxial cell results, the difference being up to  $40^{\circ}$ .

Stress measurement site No 2 was on 295 level in the hangingwall dunite about 20m south of Zone C talc zone and to the south of Block 7AB. The results may have been influenced to some extent by the mining in Block 7AB, but this is considered to be relatively small and the stress magnitudes were similar to those measured at Site 1. Site 3 was located on 295 level, between 40 and 50m away from the old Block 33 cut and fill stopes. This is the only site where the vertical stresses exceeded the horizontal and this is attributed to the higher vertical stress

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induced in the abutments of the stopes.

Site 4 on 365 level was almost vertically below Site 1 on 295 level. The site was located in the footwall brittle fibre zone, with Zone C immediately to the south and the 1/O dyke 30 to 50m to the west. At this site measurements were in only two holes, a horizontal and a vertical, and therefore a shear stress had to be assumed to derive an estimate of the principal stresses. No record was kept of the basis for this assumption.

Some of the measurements at Site 5 were made within a talc zone associated with a wrench fault, while the remainder were made in the adjacent brittle fibre zone. Both these sites are considered to have been outside any significant zone of influence of mining, but it is probable that the local geology has affected the stresses.

The measurements at Site 6 were made at a later date, just after mining had started in Block 16 but before it had progressed very far. It was hoped to obtain a comparison of the stresses before and after Block 16 had caved. The results are not considered to have been affected by the mining in Block 16.

At all the sites, with the exception of Site 3, the vertical stress was exceeded by the lateral stresses in the crosscut and drive directions. In general the stresses in the crosscut direction ( $146^{\circ}$  mine bearing) were slightly larger than the stresses in the drive direction ( $56^{\circ}$  mine bearing).

#### 5.4 DISCUSSION

##### 5.41 ACCURACY

The most important factor affecting the accuracy of a stress measurement is the degree to which the stresses are representative of the stresses in the rock mass in the area. The size of this area must be related to the size of the excavations which will be made. In Tables 5,1 and 5,2 quite marked variations in the magnitude and directions of the principal stresses are shown. These variations are attributed primarily to the variation in the modulus of elasticity between the diorite and serpentinite and proximity of slips and fibre seams. It is clear from these variations that a more realistic estimate of the stresses would be obtained from a

large number of strain readings, and that a high degree of accuracy in the individual strain readings is less important.

TABLE 5,3  
STRESSES MEASURED IN ONE HOLE IN DRIVE DIRECTION AT THE STRESS MEASUREMENT  
SITE ON 385 LEVEL  
(Readings compensated for tunnel stress concentrations)

Reading No.	Depth(m)	$\sigma_x$ (MPa)	$\sigma_z$ (MPa)	$\tau_{zx}$ (MPa)
1	0,8	22,7	15,1	3,8
2	1,2	17,5	13,8	5,2
3	1,4	15,8	13,8	2,4
4	3,3	6,5	8,4	4,8
5	3,6	3,4	6,2	4,2
6	4,1	18,6	18,6	5,5
7	5,9	7,2	7,2	2,8
8	6,2	11,4	0,7	9,2
9	6,7	14,5	15,8	4,8

The CSIR triaxial strain cell gives a high degree of accuracy in the measurements of the stresses at a point, but no study has yet been published on determinations on the accuracy of these measurements in less than ideally homogeneous, isotropic and continuous materials. To obtain a realistic estimate of the stresses in the area such as found on Shabanie Mine numerous measurements would have to be made.

Cook (1969b) claims that in the photo-elastic disc strain measurements, using a precision hand viewer (circular crossed reflection polariscope), readings can be made to an accuracy of 0,05 fringes ( $22,5 \times 10^{-6}$  strains) if the temperature is within  $1^\circ\text{C}$  of the temperature at the end of the borehole. The major principal strain is given by the fringe order, and the minor principal strain by the separation of the isotropic points. The principal strain ratio is checked by warming the disc and reading the fringe order and the separation of the isotropic points again. The heating of the disc changes both the minor and major principal stresses by the same amount thereby changing the principal stress ratios.

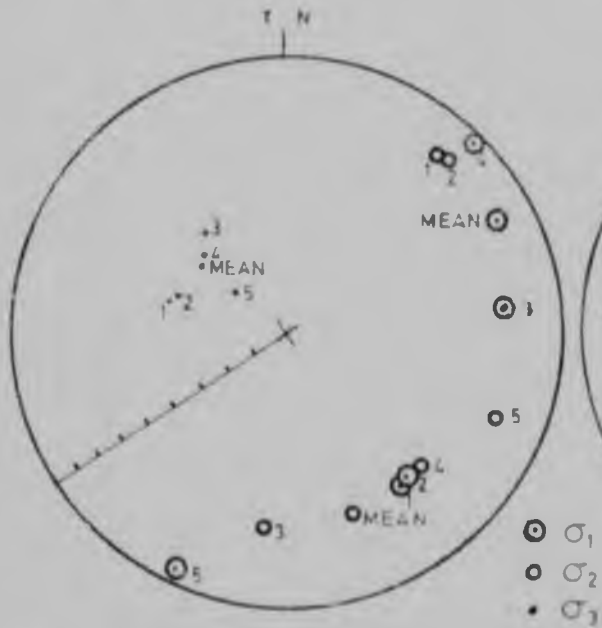


Fig 5,2 Stereographic representation of principal stress directions from triaxial strain cell measurements

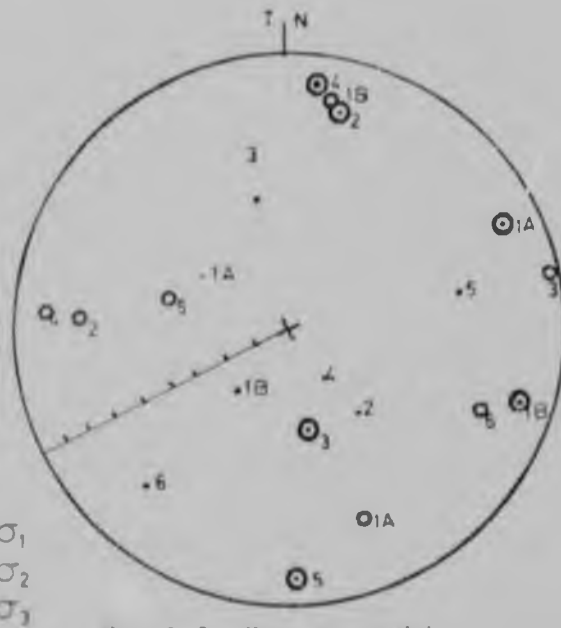


Fig 5,3 Stereographic representation of photo-elastic disc stress measurements

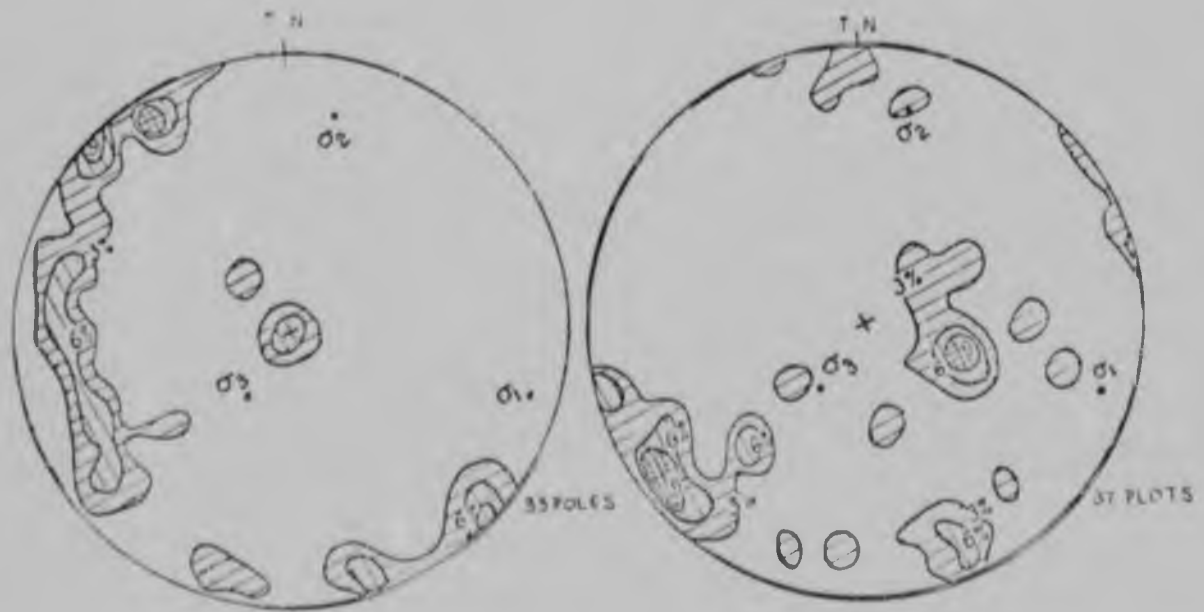


Fig 5,4 Stereograms showing relationship of principal stresses to joint orientation (left) and slickenside directions on joints (right) Site 1, 23 x/c bl. 16 2/3 level

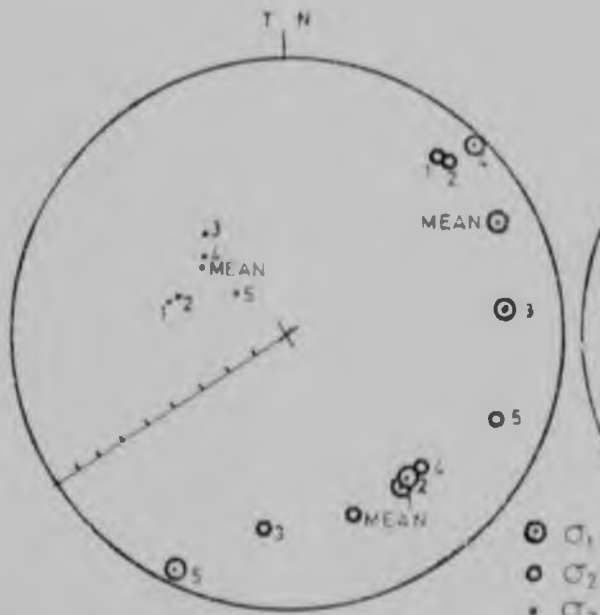


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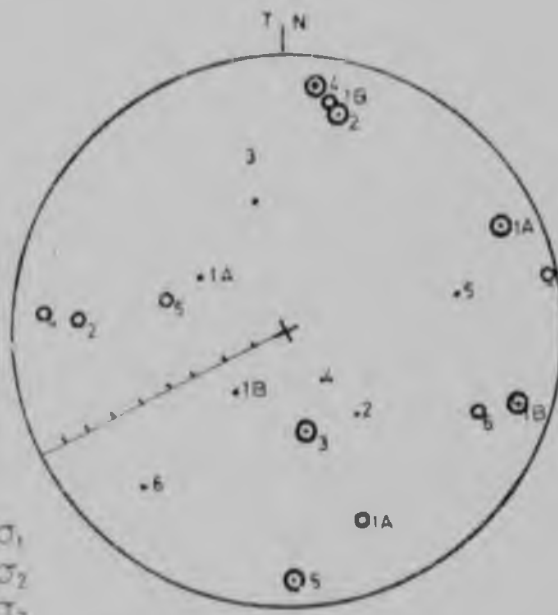


Fig 5,3 Stereographic representation of photo-elastic disc stress measurements

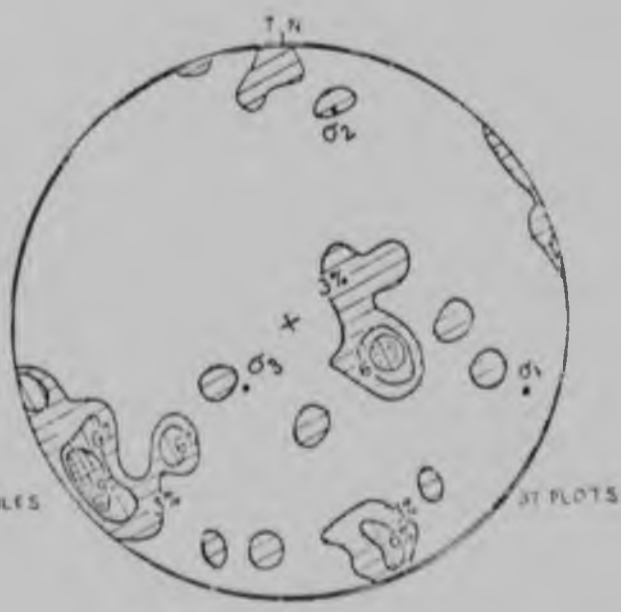


Fig 5,4 Stereograms showing relationship of principal stresses to joint orientation (left) and slickensiding directions on joints (right) Site 1, 23, x/c Bl. 16 2yb level

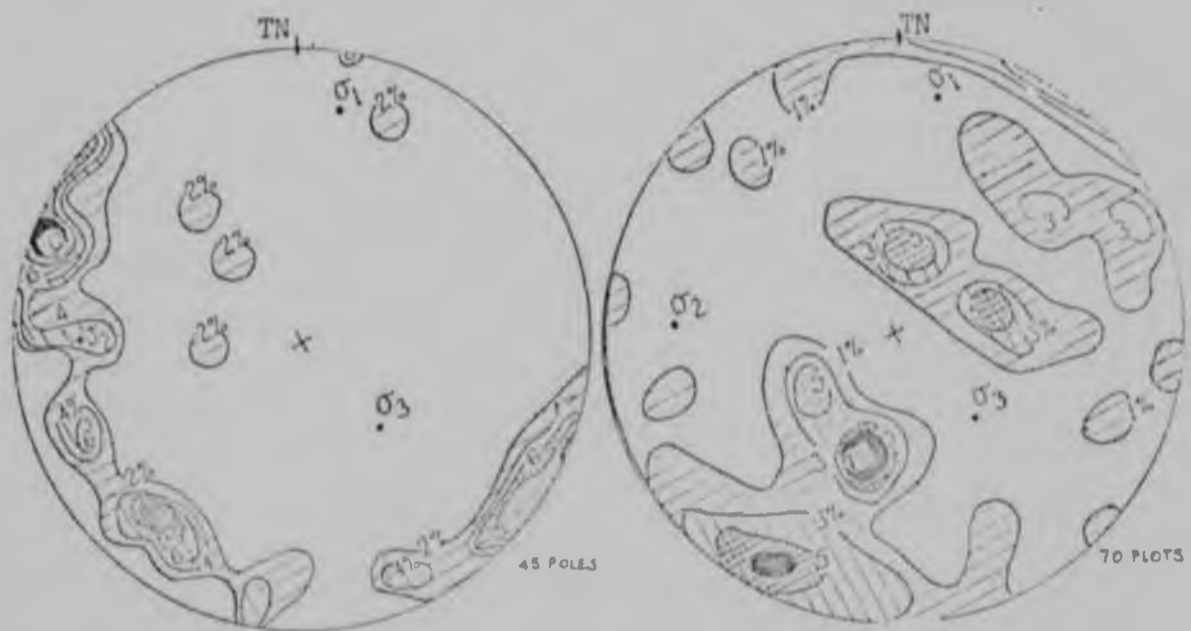


Fig 5.5 Stereograms showing relationship of principal stresses to joint orientation (left) and slickensiding directions on joints (right) Site 2 6/7<sup>th</sup> x/c Bl. 50 295 level



Fig 5.6 Stereograms showing relationship of principal stresses to joint orientation (left) and slickensiding directions on joints (right) Site 3 Main x/c Bl. 33 295 level

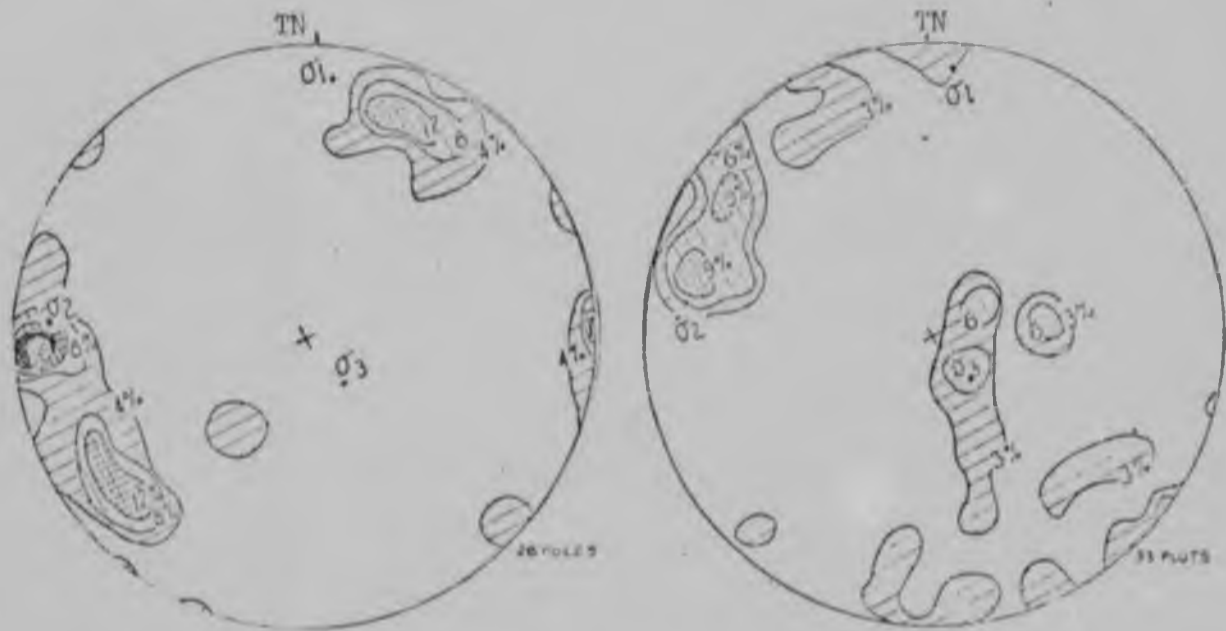


Fig 5.7 Stereograms showing relationship of principal stresses to joint orientation (left) and slickensiding directions on joints (right) Site 4 257 x/c Bl. 16 365 level

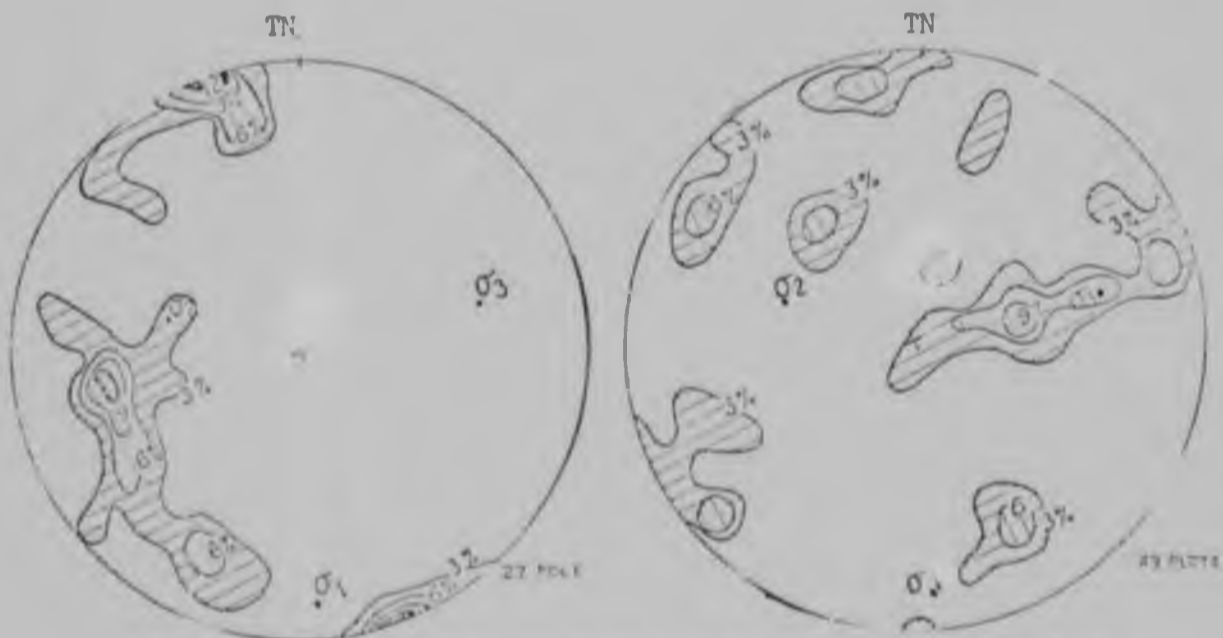


Fig 5.8 Stereograms showing relationship of principal stresses to joint orientation (left) and slickensiding directions on joints (right) Site 5 330W x/c Bl. 51 365 level

#### 5.42 RELATIONSHIP TO LOCAL GEOLOGICAL FEATURES

A detailed structural analyses of each site was undertaken to determine whether principal stress directions bore any relationship to the local structural geology. Stereograms were drawn of the slips, fibre seams and slicken siding directions on the slips and the principal stress directions. These are shown in Figs 5,4 to 5,8.

At some of the sites some relationships could be recognised but these are regarded as coincidental as these relationships do not appear at all sites. It is considered doubtful that any practical information on directions of the virgin rock stresses can be obtained from a study of geological structures.

#### 5.43 REGIONAL STRESS FIELD

From a study of the structural geology of the deposit, Laubscher (1963 P 45) concluded that the direction of compression was N 24° E, (114° mine bearing) in a sub-horizontal plane. It is probable that the moderately high lateral stresses measured are of tectonic origin and may date back to this major deformation period.

The results in Table 5,2 show that the principal stresses vary considerably from site to site, in a similar manner to the local stress variations. Although there is a wide scatter of results, an attempt has been made to relate the vertical, and horizontal stress components in the crosscut and drive directions to depth (Fig 5,9). The regression lines have been calculated using an assumed weighting of 1 to 3 for each result depending of Cook's assessment of the reliability of the result.

The vertical stress components on average are slightly higher than might be expected from the depth and rock density. This may be due to an inherent inaccuracy in the measurement technique, or merely due to the wide scatter of results obtained. The regression lines had low correlation coefficients, and therefore should be used only as an indication of the general stress levels.

#### 5.5 CONCLUSIONS

The results indicate that lateral stress components which exceed the vertical stress component. These components are related to depth by the following relations:

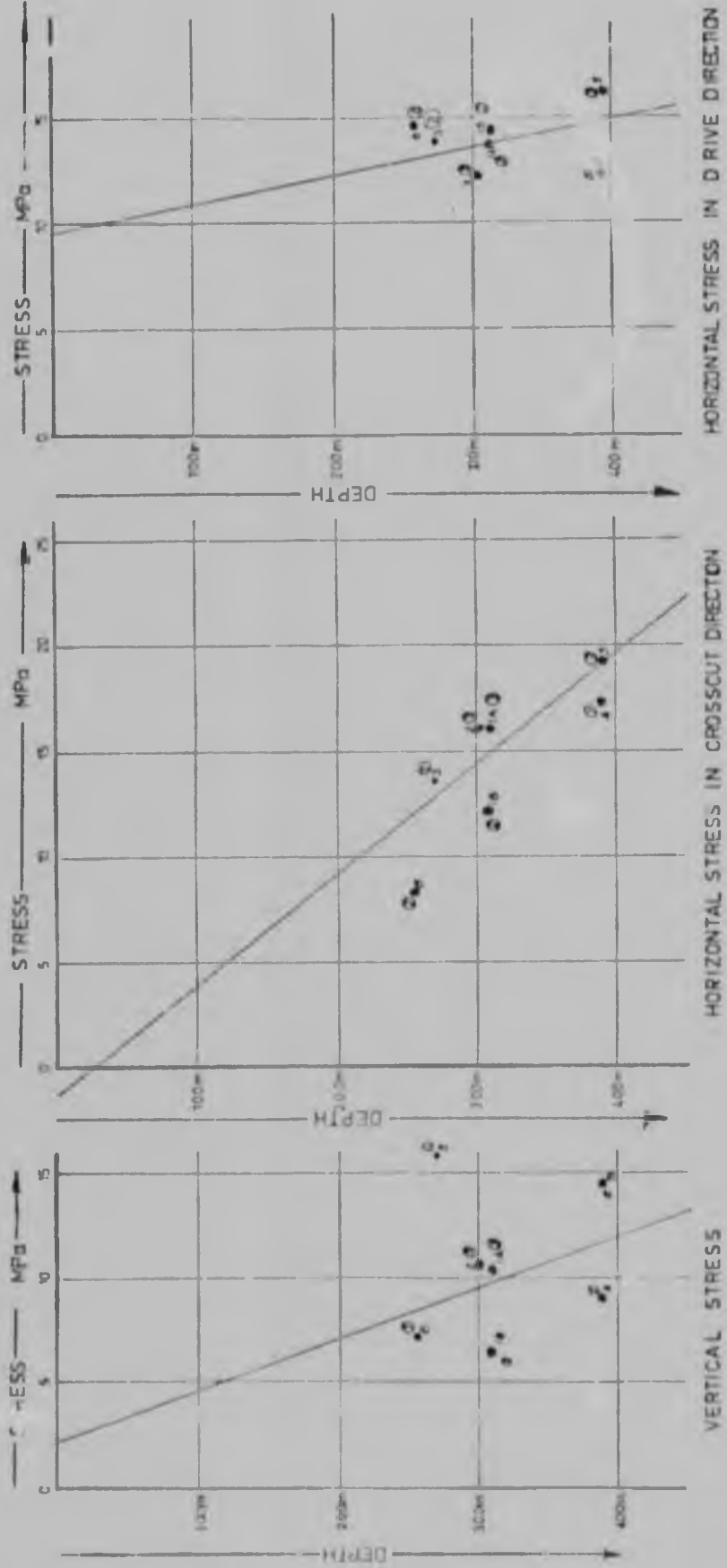


Fig 5.9 Stress-depth relationships

$$\begin{aligned} \sigma_v &= 2.3 + 0.0242xD \text{ MPa} \\ \sigma_h \text{ (crosscut)} &= 1.25 + 0.0525 \times D \text{ MPa} \\ \sigma_h \text{ (drive)} &= 9.5 + 0.0133xD \text{ MPa} \end{aligned}$$

where D = depth in metres

Additional stress measurements are required at greater depths to confirm this relationship.

Variations in stress encountered in the rock are significant. These are attributed to local variations in elastic properties of the rock and the distortions in the stress field caused by the slips and fibre seams.

The triaxial cell is an elegant method but several readings are required to obtain a representative average. The instrument is easy to read, and has a check on gauge adhesion by having three strain readings parallel to the axis of the hole. Only one hole is required.

The photo-elastic discs are cheap and reusable but are more difficult to read and are less accurate. Readings are required from three holes to obtain the complete state of stress. While the technique is less accurate than the triaxial cell this is considered to be unimportant in the highly variable stress environment encountered on Shabanie Mine.

It is considered that the moderately high lateral stresses measured are residual from the tectonic compressive stress which created the major geological structures and folded the ultrabasic sill.

However, no close relationship exists between the measured principal stress orientations and the local geological structures at each stress measurement site, and it is considered that no conclusions on the virgin rock stress orientation or magnitude can be drawn from a study of local geological structures.

CHAPTER SIXDESCRIPTION OF BLOCK 7AB MINING AND GROUND BEHAVIOUR6.1 INTRODUCTION

Block 7AB was the first block on Shabanie Mine to come into production after the initiation of the rock mechanics research programme, which forms the basis of this dissertation. This block was extensively instrumented and all available development workings above, below and in the peripheries were inspected regularly with the object of determining the mechanisms of caving and the minimum requirements for a block to cave, the effects on the peripheries and underlying ground, the extent of the ground affected etc. This block was also used for the field testing and development of the monitoring techniques described in chapter 4.

This block is part of the Block 7 orebody and was initially laid out and developed to provide a reserve source of ore in the event of another block closing prematurely. The name of the block is a contraction of the original subdivisions of the block - Block 7A and block 7B. It was located near the top of the orebody as this was the only accessible portion at that time, and it was sited below Block 6 to ensure 'continuity of cave'. The mining of the block was divided into two phases, the first phase or the southern half, and the second phase or the northern half of the block, which was undercut some months after the first phase. In addition, an effective extension of the undercut was provided by the extensive 235 level cut and fill stopes which had been mined in the upper portion of the orebody.

6.2 GEOLOGY

Block 7AB is located in the upper portion of the Block 7 orebody as illustrated in Fig 2,5 and Fig 6,1. The orebody may be divided into two sections, which roughly coincide with the first and second phases of mining, and in which there is a change in the characteristics and class of the ore. In the northern half, the fibre occurs mainly as conjugate ribbon seams which generally dip towards the south and south east, while in the southern half the fibre occurs as single and double fibre seams, with a general but more variable dip towards the south east (Fig 2,7).

Minor slips occurring subparallel to the fibre seams were more common in the southern half. In general the northern half had fewer slips and aplite-derived hybrid rock intrusions than the southern half (Fig 6,2). The southern portion also contained highly irregular, small zones of blocky very incompetent rock (Class 4) in which the rock was cut into blocks of less than 0,5m maximum dimension by numerous minor platy-picrolite filled slips. These factors are reflected in the geomechanics classification of 265 grizzly level. In the northern half of the block the rock class ranges from 1B to 2B, with a small zone of Class 4 associated with a highly fractured dyke of aplite derived hybrid rock. In the southern half Class 3A predominates, with a smaller area of Class 3B in which the small areas of blocky Class 4 rock are most common, and contribute to the lower competency.



Fig 6,1 Cross-section through Blocks 3, 6 and 7AB

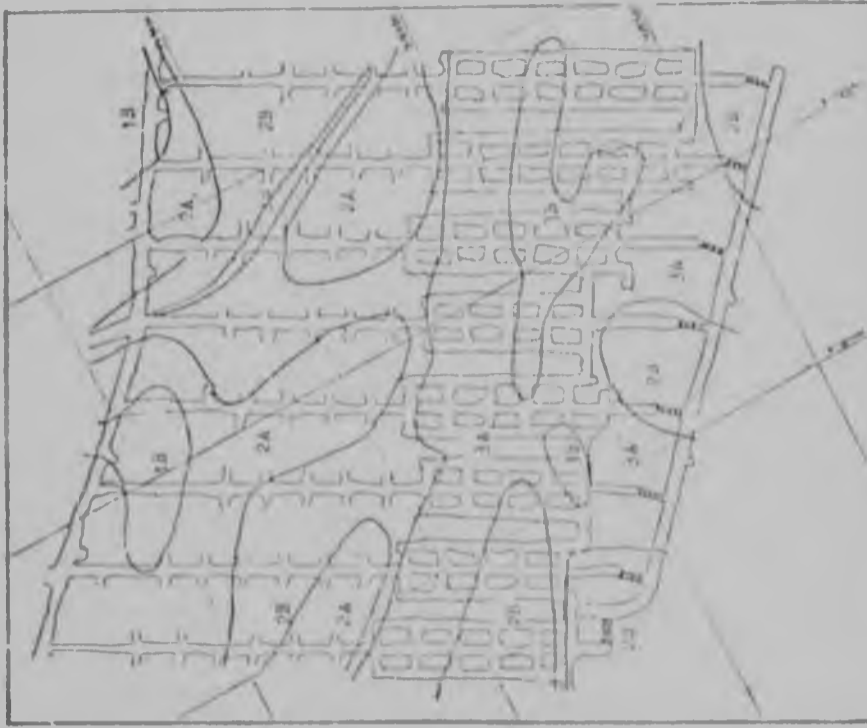


Fig 6.3 265 level Geomechanics classification

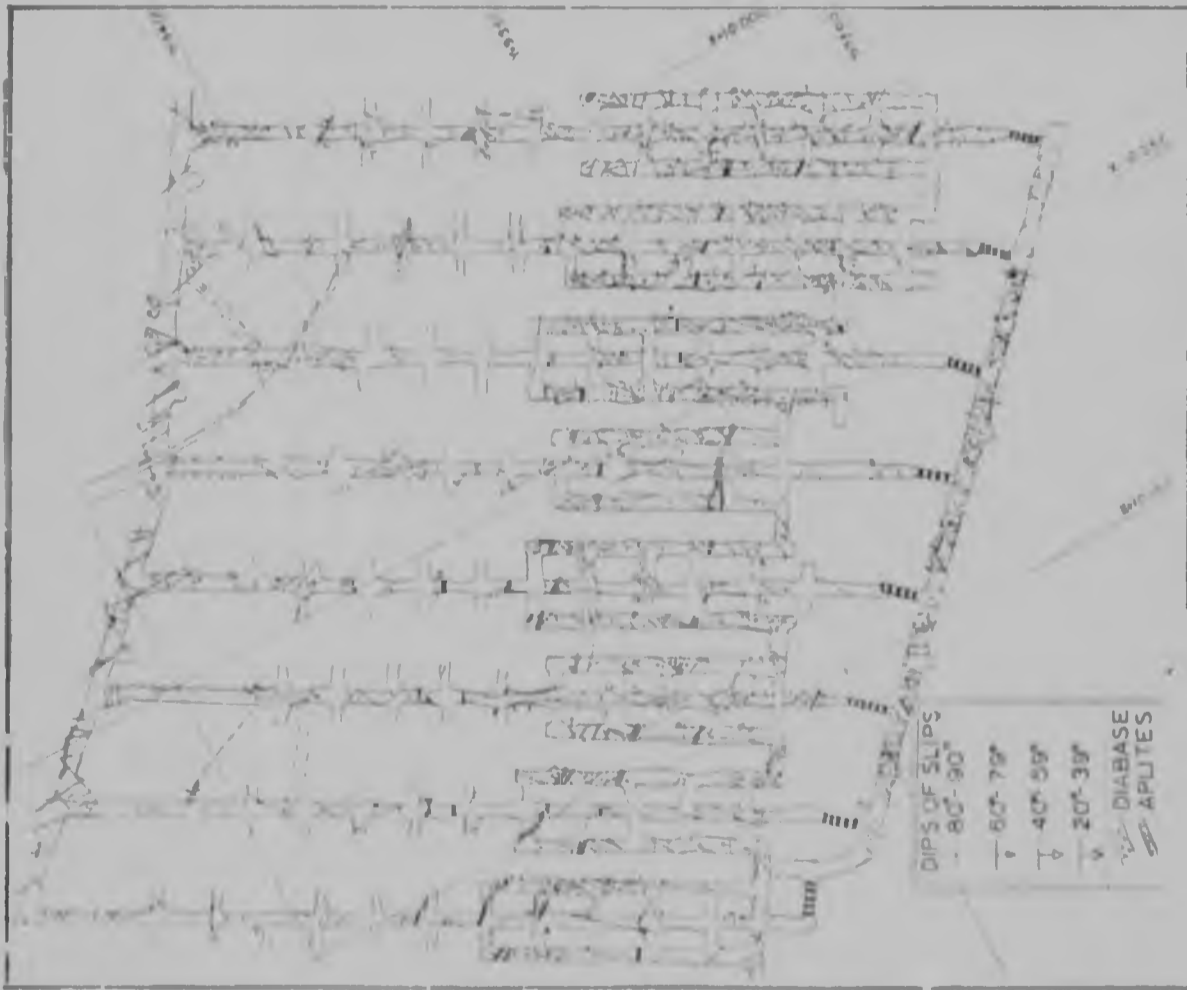


Fig 6.2 265 level geological plan

To the north and above the orebody is a relatively wide zone of carbonated serpentine containing brittle fibre. This zone ranges in class from 1B to the north of the orebody to 3A where more highly fractured above the orebody. In this zone several south dipping shears were developed, and the largest and most important of these is the Travellingway shear-zone, which is an extensive but irregular fracture with coarse slickensiding in the crosscut direction. The irregularities have given rise to vugs with some secondary magnesite deposition. This shear persists into the orebody where it has associated with it a lower class of rock.

Overlying the orebody is Zone B talc zone, and the mined out Block 6 orebody to the north. The talc zone has an irregular central shear-zone and is generally a Class 3B rock. The contacts, however, are highly fractured in places resulting in a Class 4 rock. The hangingwall dunite on either side of Zone B is generally competent being Class 2A and 2B. At the surface there is a 25m thick weathered zone with a 4A classification.

The steeply north dipping Zone A occurs on the north side of Block 6 and is similar to Zone B in competency.

The footwall talc to the north of the orebody is cut by weak gouge-filled shears with an east-west strike and a dip of 35 to 50° south. The competency of this zone is variable, ranging from 3A to 4B depending on the degree of shearing.

### 6.3 MINING

Prior to the instrumentation of the block, the areas that had been mined were - Blocks 3,4,5,6 and the 235 level cut and fill stopes in Block 7. The relative positions of these are shown in Fig 3,1.

This block was undercut in two phases, starting with the southern half known as the first phase between February 1967 and March 1968 and followed by the second phase in the northern half which was mined in October 1968 and July 1969. The first phase undercutting initiated a small zone of caving from the immediate undercut back, together with the old cut and fill stopes created a large effective undercut area and which initiated the main caving period. The date accepted for the caving of the hangingwall of the block was 12th April, 1968. The undercutting of the second phase was required to cave the ore overlying the undercut and underlying the cut and fill stopes.

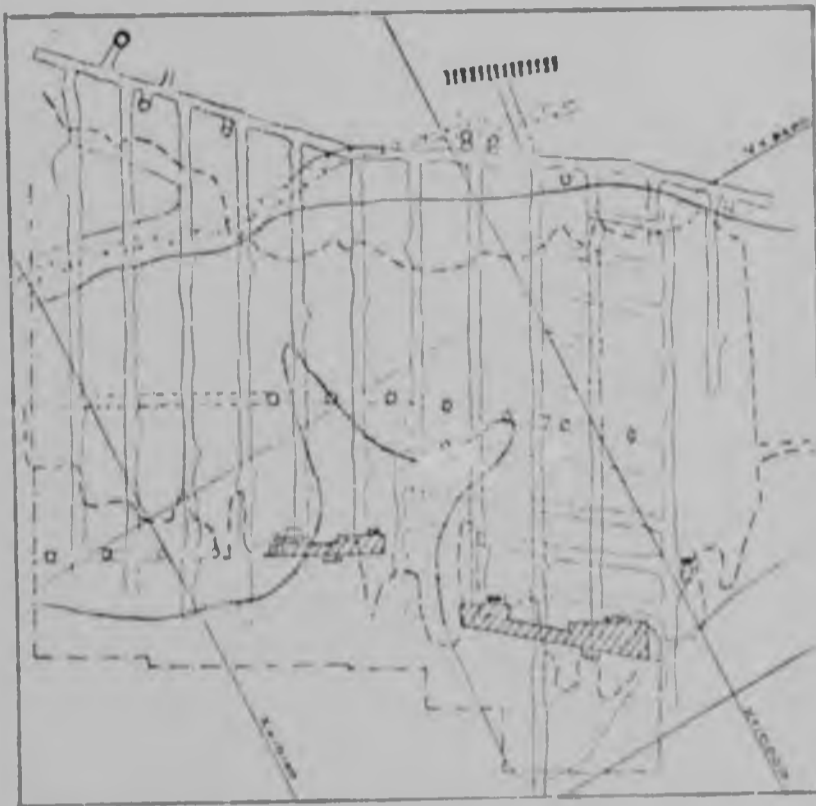


Fig 6,4 245 level Plan showing slots cut and areas shrunk

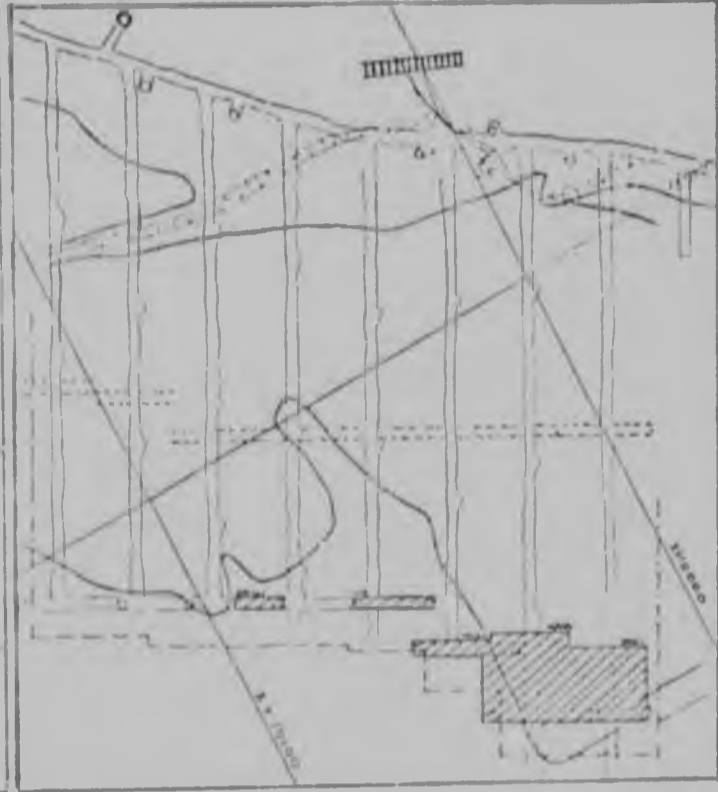


Fig 6,5 255 level Plan showing slots cut and areas shrunk

The idea behind the original layout for the block was to shrink all the ore between the drawpoints and the cut and fill stopes. A start on this was made on 245 and 255 levels on the south western corner of the block, where slots were cut and some shrinkage done (Fig 6, and 6,5). This was abandoned when the shrinkage face advanced to beneath the stopes and excessive quantities of sand filtered through to 245 level. Also, as at that time severe damage was being experienced in both these sublevels, it was no longer considered necessary to shrink the ore and it was thought it would cave easily. The first phase undercutting was therefore reduced to the cutting of 265 level draw troughs, plus a widening of the 255 level crosscuts to complete the undercut (Fig 3,5). Once undercut the drawpoints were not brought into full production, but work proceeded with the preparation to undercut the second phase. This involved developing and cutting of the slots on 245 and 255 levels. The undercutting of the second phase consisted of cutting and emptying pairs of drawpoint cones and then with longhole fans, breaking the ore between 255 and 245 levels into the empty cones (Fig 3,5). The ore between 245 and the stopes was not broken and allowed to cave. When the second phase undercutting was complete, production was resumed in the first phase and continued until depleted. Draw then started on the east side of the second phase and east to west on panel retreat for draw control was practiced.

To prevent the collapse of the stopes of this block when the block was undercut, an attempt to develop a vertical plane of weakness along this boundary was made. This was done by lightly charging and blasting, in an open pit pre-split manner, a series of boreholes drilled upwards and downwards from a crosscut on 205 level, but as shown later, this did not create the desired line of weakness.

#### 6.4 VIRGIN ROCK STRESSES

Using the general stress - depth relationships determined from stress measurements in other parts of the mine given in Chapter 5, it is accepted that prior to any mining in this area, the virgin rock stresses were of the order of:-

205 level  $\sigma_x = 9,0$  MPa  
 $\sigma_y = 12,0$  MPa  
 $\sigma_z = 6,5$  MPa

265 level  $\sigma_x = 12,0$  MPa  
 $\sigma_y = 13,0$  MPa  
 $\sigma_z = 8,5$  MPa

where  $\sigma_x$ ,  $\sigma_y$  and  $\sigma_z$  act in the crosscut, drive and vertical directions respectively.

The mining of blocks 3, 4/5 and 6, and the 235 level stopes redistributed these stresses, which caused small displacements to take place on some major slips. These small displacements in turn caused further redistribution of the stresses. This statement is inferred from observations made on ground behaviour in this block as mining progressed.

It is expected that before mining in Block 7AB started higher lateral stresses existed between block 6 and 235 level stopes, and below these stopes. The lateral stresses around block 6 were reduced in a radial direction, but probably increased tangentially. Also, due to the large span of the 235 level stopes, the abutments of these stopes were fairly highly stressed in a vertical direction.

#### 6.5 INSTRUMENTATION

Prior to the start of the undercutting, the block was instrumented with a variety of instruments on various elevations above, below and in the peripheries of the block. The object of this instrumentation was to investigate the mechanisms of caving, and to determine the conditions required for caving to start and continue to test and develop suitable monitoring techniques, and, to investigate the effects of the caved ground on the underlying rock and extraction workings.

Surveying and levelling traverses were extensively used in the peripheral areas, to monitor subsidence and lateral displacement. Over the back of the block borehole clamps and remote displacement meters were used for this purpose (See 4.3 and 4.4). The location of these instruments is shown in Fig 6,6 to Fig 6,12. The instrument readings were done weekly and the surveys at approximately three monthly intervals.



Fig 6,6 Surface instrumentation and Block 6 subsidence outline

For monitoring local stress changes and local rock deformation the glass plus stressmeters, closuremeters, Horstman bolt tension meters and the Demec extensometer were used. With the exception of the extensometer, these instruments were concentrated in the abutments of the 235 level stopes on 235 level and in the 265 level grizzly level. Measuring points for the Demec extensometer were only established on one level prior to caving, but after the block caved measuring points were established extensively on all major levels and proved to be extremely useful in monitoring the rates of relative displacement on slips and cracks.

An attempt was made to measure the loads imposed on the crown pillars by the caved ground by installing five large load cells on the tops and sides of the crown pillars in two localities (See 4.64).

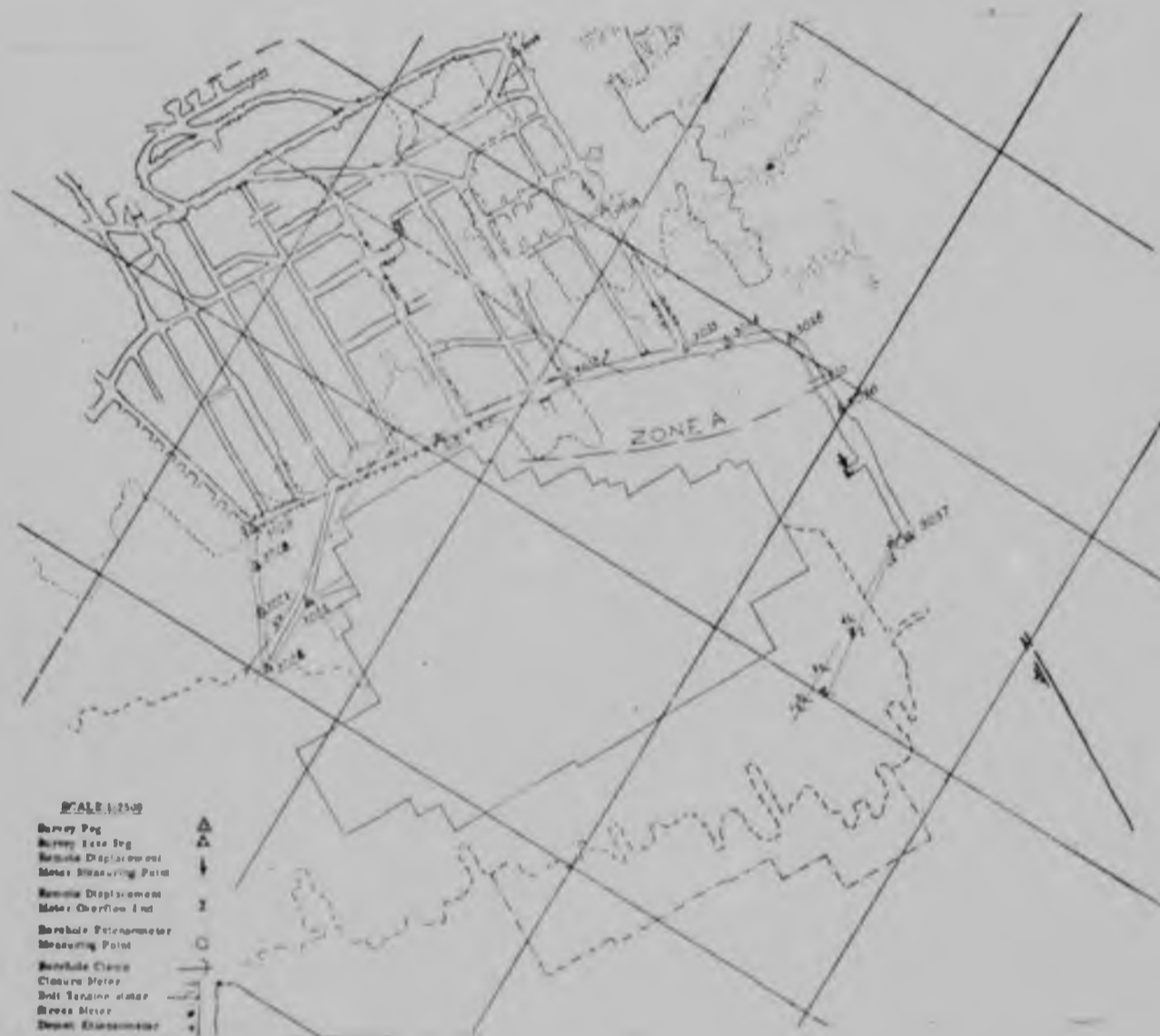


Fig 6,7 125 level Plan showing location of instruments

#### 6.6 VISUAL INSPECTIONS

The 245 and 255 pre-breaking sublevels, the 265 grizzly levels and the 275 trimming level were inspected weekly to observe the caving process and effects on peripheral ground and the change noted in a log book and on plans of the respective levels. The main levels were inspected on a monthly basis for the same purpose. The appearance of new rock falls were noted, and the dilation of cracks monitored by instrumental techniques or by small pats of plaster placed across the open slip or crack. The surface above the block was inspected on a monthly basis, and this was increased to a weekly basis once cracks had appeared on the surface.



Fig 6,6 170 level Plan showing location of instruments



Fig 6,9 205 level Plan showing location of instruments

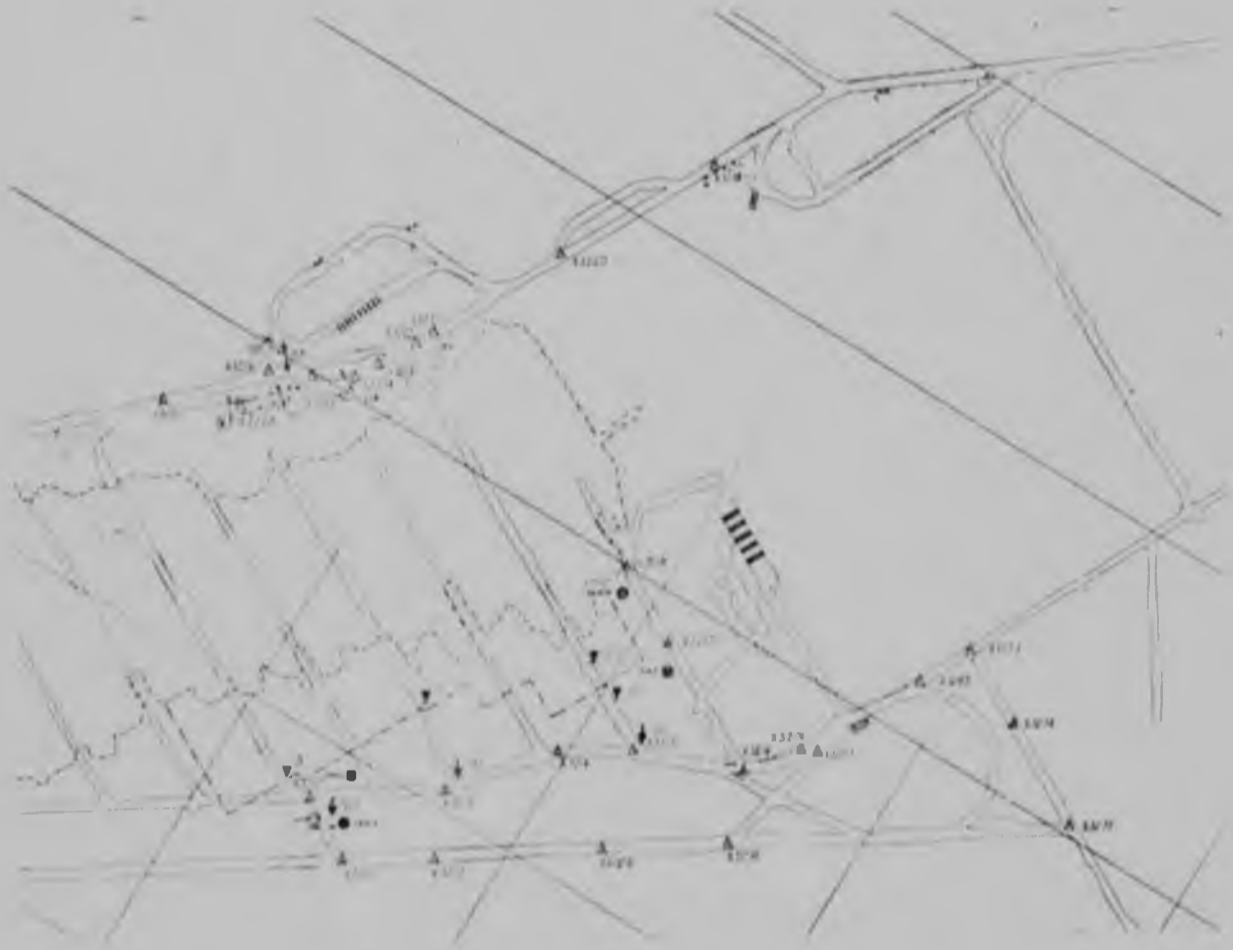


Fig 6,10 235 level Plan showing location of instruments

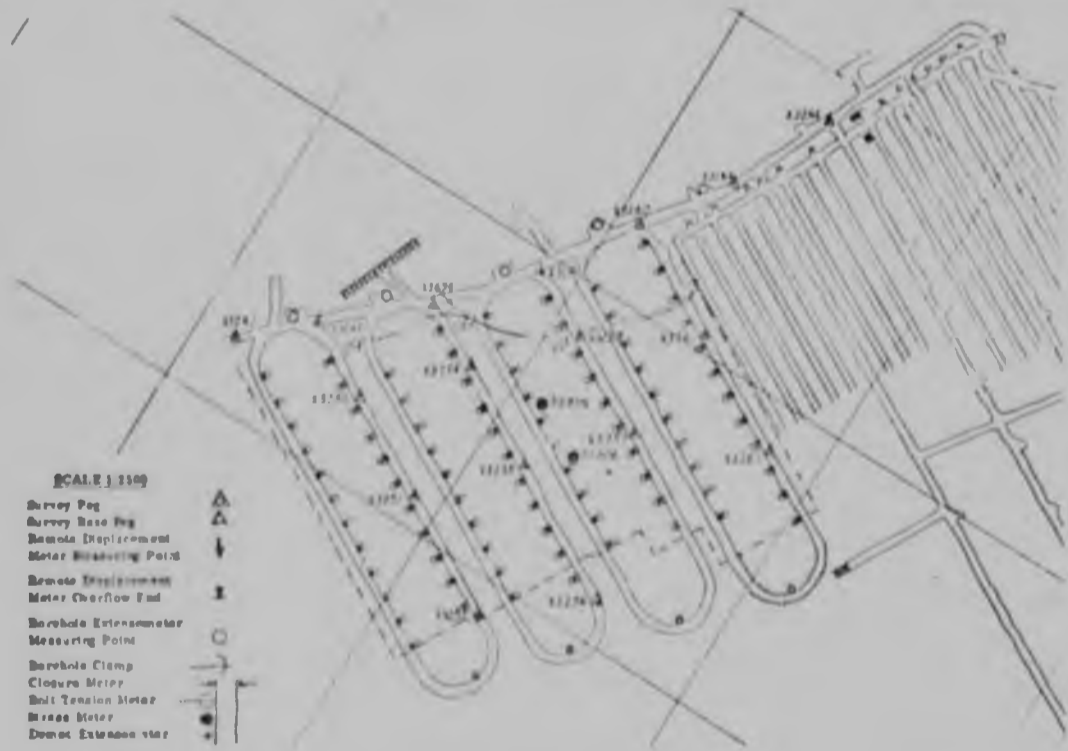


Fig 6,11 275 level Plan showing location of instruments



Fig 6,12 265 level Plan showing location of instruments

#### 6.7 CONDITIONS PRIOR TO UNDERCUTTING

The peripheries of the cave had previously been affected by the caving of Blocks 3, 4/5, 6 and 8. Block 3, lying to the north of the block and above 125 level was the first block to be caved on this mine. The caved area was extensive but relatively shallow. To the west this caved ground ran into the caved ground of Block 8 and to the east the caved ground of Block 4/5. The undercutting in Block 4/5 extended to greater depths, to between 170 and 205 levels. Dilation and shear displacements had occurred and could be observed on slips in the peripheries of these blocks on 120 and 135 levels as well as in 3 shaft.

South of Zone A, overlying the northern side of the block was Block 6. This block had caved through to surface leaving a circular

subsidence zone, flanked by a small overhang on the east side and a much longer overhang on the western side. In the northern and eastern peripheries of this block too, dilation and shear displacements were found on slips on 125, 135 and 170 levels. On 170 level these displacements had necessitated the building concrete pillars to support a wide, previously stable drift. It is expected that similar displacements and dilation of slips had also occurred on the south and western sides, although no workings were available for inspection in this area.

The three survey beacons established on the pillar between Blocks 3 and 6 were shown to be slowly moving northwards towards the mining in Block 8.

The 235 level stopes had not had any visible affect on either the foot or hanging wall development of 235 level. The pillars within the stopes were not inspected by the writer prior to the commencement of undercutting, but it is expected that similar to other stopes elsewhere on the mine that some cracking of the pillars had occurred.

#### 6.8 RESULTS OF MONITORING AND VISUAL OBSERVATIONS

As the investigations covered a period of four years and a wide range of inter related facets of ground behaviour were investigated, the description which follows has been divided into three main sections; firstly investigations of caving processes, secondly the effects of undercutting and caving on the rock in the peripheries and thirdly the effects of undercutting, caving and production on the rock and extraction openings below the undercut. Each of these sections will be subdivided into subsections dealing with specific areas or time periods.

##### 6.81 INVESTIGATION OF CAVING PROCESS

The observation and monitoring results from the back of the undercut to surface are described in this section. This ground may be divided into three zones: from the undercut back to the cut and fill stopes, between the stopes and Block 6, and from Block 6 general grizzly elevation to surface in the southern periphery of Block 6, in which certain characteristics in the rock behaviour have been recognised.

The undercutting was done in two phases, and during the first phase the whole of the sublevels above the undercut were open to inspection,

but shortly after the start of the undercutting of the second phase, 245 level was sealed off for ventilation purposes, preventing observation of the caving process. Also no observations could be made on 255 level, as in this phase it was the undercut level.

#### 6.811 Visual observations on the sublevel immediately above the undercut

The slots on 245 and 255 levels were almost complete on the eastern half of the block, and on the western half slot cutting had barely started when the shrinkage of the first phase was abandoned in favour of drawpoint troughs (Fig 6,4 and Fig 6,5). The weekly observations of the ground on these sublevels as the troughs were crashed revealed initial differences in ground behaviour between the eastern side and western side of these levels. The observations in this area were made over a period of February to May 1968. On the eastern side of 255 level small cavities overlying the drawpoints were frequently observed, and large boulders resting on the crown pillars could be seen at times. On 245 level overlying this area the ground subsided in large intact blocks bounded by near vertical slips or shears in March and April 1968. Little other damage was noted in the crosscuts, which remained open for several meters into the subsidence area.

Over the western half of the block, on 255 level, small cavities overlying the drawpoints were also observed but less frequently than in the east. A large cavity overlying several drawpoints developed on the extreme west side of the block. On 245 level no subsidence similar to that observed on the east occurred and cavities were seldom seen, but large sections of the crosscuts, when undercut, developed severe spalling before collapsing. Even as late as April 4, 1968 it was possible to see up to 18m down an undercut and damaged crosscut. Also two extensive horizontal tensile cracks were observed on 245 level between pairs of crosscuts shortly before they collapsed.

#### 6.812 Observations and results between the 235 level cut and fill slopes and block 1 rizzly

The undercutting of the southern abutment of the cut and fill slopes, first intersected the eastern slopes, leaving a wedge shaped pillar between the slopes and the undercutting operations (Fig 6,13). At an

early stage in the intersection of the undercut and stopes, small, slow vertical displacements were recorded over the stopes by the 205 level remote displacement meters. Inspection of the pillars in the cut and fill stopes showed light cracking of the pillars. As the wedge between the southern limit of the stopes and the undercutting was reduced, the rates of subsidence recorded over the back increased. Sharp increases in the rates of subsidence were recorded on the east side of the block early in March 1968. At the same time there was a smaller increase in the rate of subsidence over the western side of the block (Fig 6,14 and (,1)).

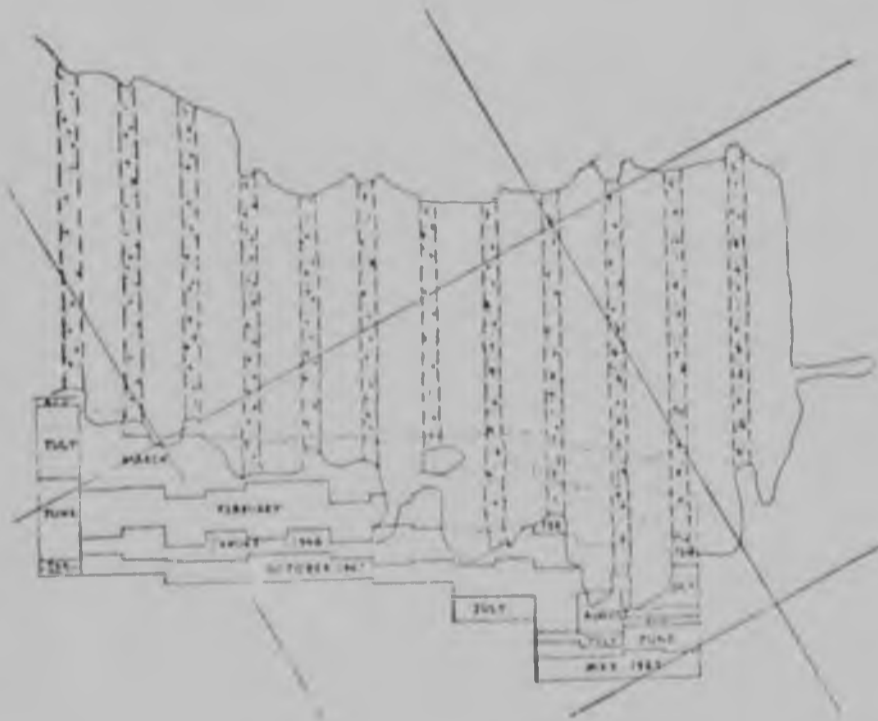


Fig 6,13 235 level stopes and 205 level troughing face positions

On 205 level the increased subsidence rates coincided with the appearance of extensive almost horizontal tensile cracks in the ground overlying the stopes. North of the northern abutment of the stopes, 6

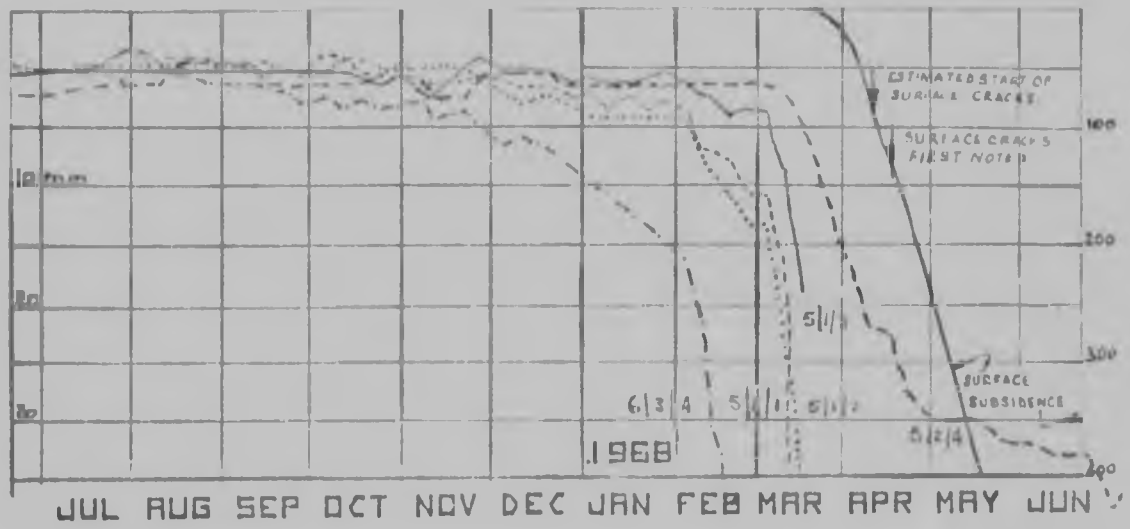


Fig 6,14 Graphs of subsidence - east side of block

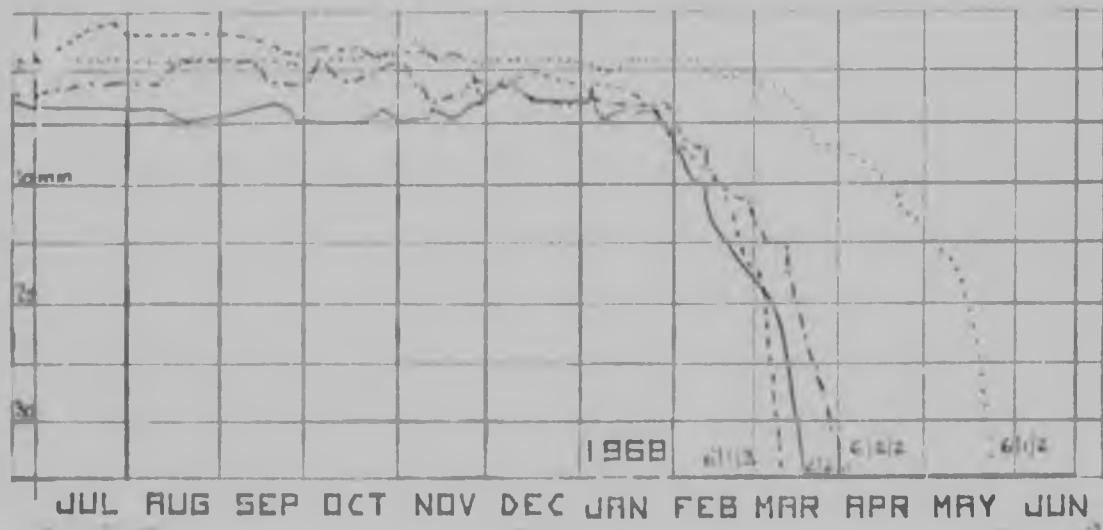


Fig 6,15 Graphs of subsidence - west side of block



Fig 6,16 Subsidence contours over the stopes 205 level

grizzly drive exhibited extensive spalling and cracking which was interpreted as being due to a high lateral stress with spalling from the back with buckling of the concrete floor.

Fig 6.16 is a plan showing the subsidence contours recorded on 12 April 1968, on 205 level. It is estimated that the cracks appeared on surface at this time (See section 6.73).

On the west side of the block the subsidence extended beyond the presplit boundary. The instruments located on either side of the presplit indicated a uniform subsidence gradient, with no discontinuous differential movement on the presplit hence presplitting had proved to be ineffective. An examination of the stopes on the west side of the block showed that the pillars had been severely damaged to 82m beyond the western limit of the block and lightly damaged to 130m west of this limit. In May 1968 horizontal tensile cracks appeared in the 205 level tramming crosscuts west of the presplit.



Fig 6,17 Surface subsidence zone 20th September 1969



Fig 6.18 Block 7AB surface subsidence zone October 1970 Scale 1 : 2500

6.813 Results and observations from the southern periphery of block 6

Shortly after the increase in the subsidence rates recorded underground, cracks appeared on surface. When first observed on April 18, 1966, the cracks were a few centimetres wide, and some had small vertical displacements. The rate at which the cracks subsequently opened indicated that they had appeared within a week of their discovery. The appearance of these cracks was taken to indicate that the hangingwall had caved. The survey of previously established survey pegs was resumed and this indicated an increasing rate of subsidence which closely paralleled that being recorded on 170 level (Fig 6.16).

At the same time the two surface boreholes which were equiped with caving indicators, did not record any falls from the backs of the cut and fill stops, but instead the wires were highly tensioned indicating

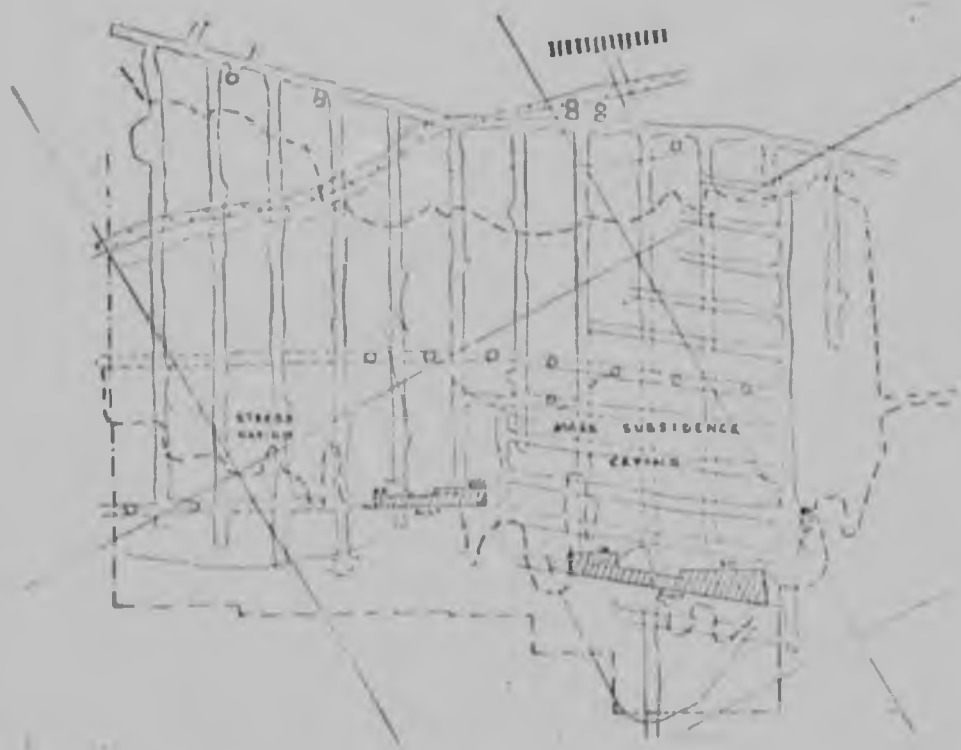


Fig 6,19 255 level Plan showing areas of mass subsidence and stress caving

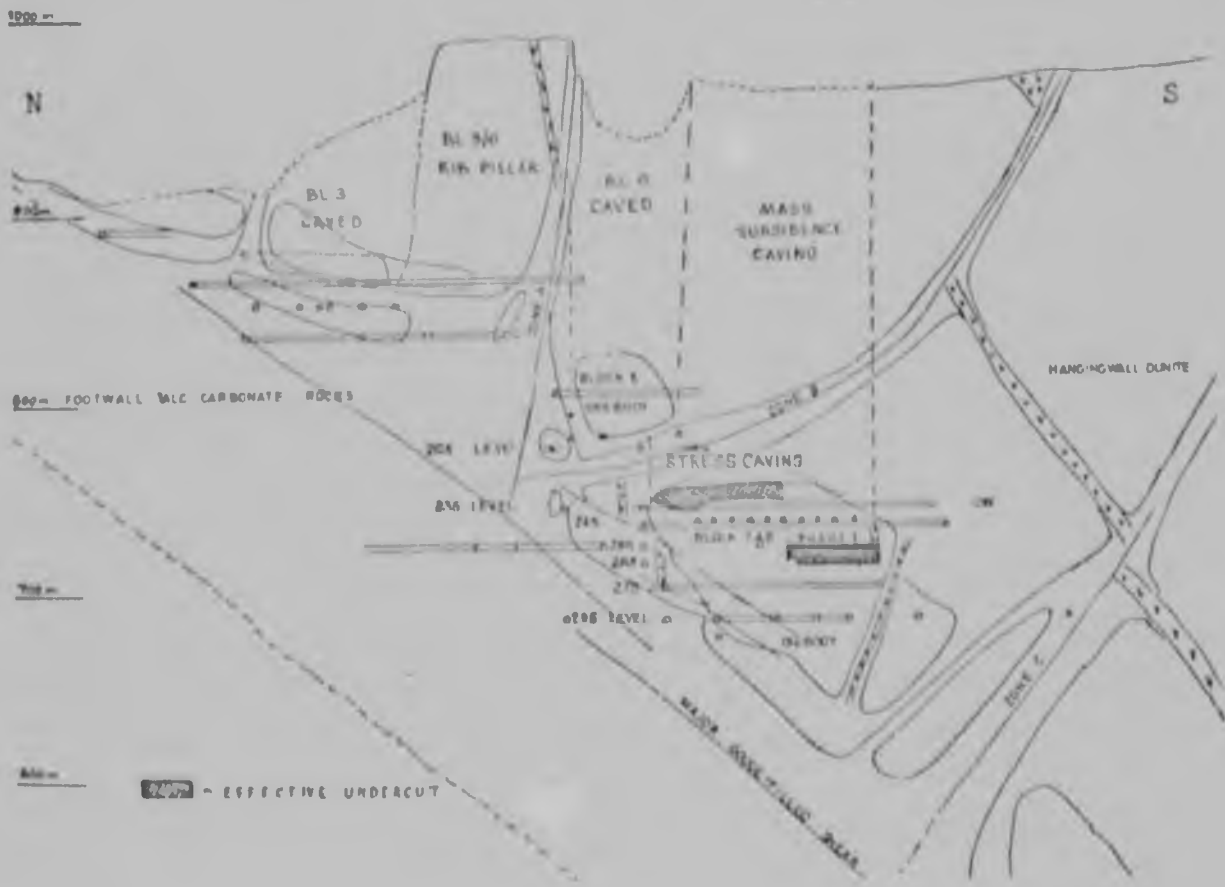


Fig 6,20 North south section showing area of mass subsidence and stress caving

that they had been caught by shear movements on slips and stretched.

With further mining operations subsidence within the surface cracks continued. A survey of the surface subsidence zone conducted on September 20, 1969 showed that the volume increase of the caved ground was 6%. Another survey was done in October 1970 after further draw and this gave a volume increase of 12%.

The final shape of the surface subsidence zone is shown in Fig 6,18 and this shows that a  $80^{\circ}$  (from the horizontal) overhang developed on the east side of the block. On the south side of the block, the lowest angle of subsidence was  $78^{\circ}$ , and small tension cracks were observed to an angle of  $70^{\circ}$  (the angle of break). The angle of subsidence bears a close correlation to the average dip of the E-W striking slips in this area. The average dip of these slips is  $80^{\circ}$ N. On the west side the subsidence zone embraced the western overhanging portion of Block 6 and extended to 86m W of the drawpoints. Underground the cut and fill stopes had suffered severe damage up to 82m and had collapsed to 43m W of the drawpoints, indicating an angle of subsidence of  $71^{\circ}$  from the drawpoints and  $80^{\circ}$  from the limit of stope collapse. Again the angle of subsidence appears to be determined by the dip of slips on this side.

#### 6.814 Discussion

Caving in this block has occurred in different areas under different conditions. Two sets of conditions, each with its own form of caving and characteristics can be recognised:-

- a) where previous mining has removed or reduced the lateral restraint, such as above 205 level south of Block 6 and between the undercut and the stopes, on the eastern side of the block the caving is characterised by the orderly subsidence of large vertical columnar blocks bounded by steeply dipping slips (Figs 6,19 and 6,20). This form of caving has been called "Mass Subsidence Caving" (Heslop 1969). The low bulking factors recorded in this block are attributed to the large blocks and the orderly form of subsidence.
- b) In the absence of previous mining to reduce the lateral stresses, the form that caving takes is very different; it has

been called "stress caving" and in this block there were two areas where it occurred, in the ground between the stopes and Block 6 and between the undercut and stopes on the western side of the block (Figs 6,19 and 6,20). In both these areas development was under considerable lateral stress as evidenced by damage to development openings such as spalling from the back and heaving of the concrete floors. In addition horizontal tensile cracks were observed over extensive areas, indicating buckling resulting from a high lateral stress and a reduction in vertical stress. It may be postulated that had the rate of draw of the caved area below been higher these cracks may have developed into substantial cavities.

#### 6.82 THE EFFECTS OF UNDERCUTTING AND CAVING ON THE PERIPHERAL AREAS

The observations made in the peripheries of this block have been used as the basis for much of the interpretation of ground behaviour which follows in chapter 8. The effects on the peripheral areas have been divided into three time periods, relating to the undercutting and caving of the block, and are discussed separately below. The effects of the undercutting and caving of the immediate hangingwall of the first phase were observed over a relatively short period and in only two localities. However, when ore overlying the stopes caved the effects were widespread and felt over a long period of time. The undercutting of the second phase mainly effected the immediate footwall.

##### 6.821 The effects of undercutting the first phase

Effects of mining on the peripheral ground were observed almost immediately after mining started in the block, when in October 1967 damage was recorded in the low competency Class 3B blocky ground in the hanging-wall drive on 235 level. This section of the hangingwall drive had been instrumented with closuremeters, stress meters, some borehole clamps and remote displacement meters, and a survey traverse (Fig 6,10). The photo-elastic glass plug stress meters did not reveal any significant stress changes, but the closuremeters showed a N - S dilation of the ground. The survey pegs revealed a similar lateral movements of the peripheral ground towards the caved ground. These displacements were first recorded in January 1968 and again in February, prior to the caving of the ground

overlying the stopes in April 1968. Deterioration in this area became so severe in March 1968 that it was decided to discontinue regular observations and instrument readings. Below this area in the hangingwall drive on 270 level, south of the 265 level grizzly crosscuts, small dilation and shear displacements were noted on several slips. Displacements were recorded on some of these slips from the time of installation of the Demec extensometer monitoring points in November 1967 through 1968 and into 1969. Fig 6,21 shows the displacement recorded on one of the major slips intersected by this drive. This displacement is similar to that recorded on other slips except in the magnitude of the displacement which was larger than most of the other slips monitored. The greatest rates of displacement were recorded in early February 1968, and slower, fairly constant rates were recorded from late February to June with no significant change which could be related to the caving of the ground overlying the stopes in April.

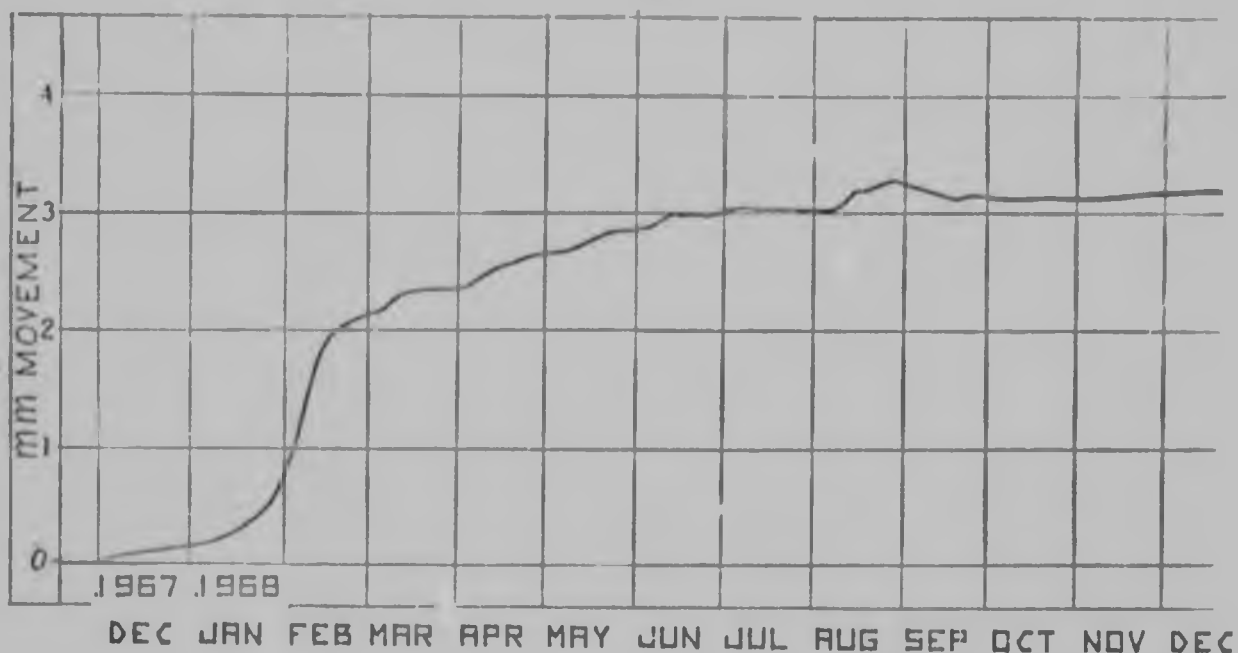


Fig 6,21 Displacement monitored on a major slip on 270 level by Demec extensometer monitoring point No 2

6.822 The effects of the caving of the ore overlying the stopes

The peripheries of the potential cave zone were monitored by survey traverses carried from base pegs in areas considered originally to be

relatively stable in the footwall or on the eastern side of the block. The initial surveys were done in November and December 1966, and repeat surveys were done at two to three month intervals between December 1967 and 1971.

Prior to April 1968 the periodic triangulation of surface beacons on the Block 3/6 rib pillar indicated that the pillar was moving north-westwards towards Block 8 at a rate of 8mm/month. During April the direction and rate of displacement changed abruptly to a south-westward direction towards Block 7 at a rate of 25mm/month. Shortly after this the pillar started to crack and disintegrate. Underground prior to April the survey traverses on 125 level also indicated that the ground in the north-western periphery of the Block 6 caved area was moving at a rate of 2 to 4mm/month north-westwards towards Block 8. As the Block 7AB caved zone developed, further displacements were noted on slips and shears which included some on which movement had occurred previously during the mining of Blocks 6, 4/5 and 3. These displacements could be seen on both sidewalls of the development opening and so represent shearing of the rock mass, and not merely the failure of the ground in the zone of influence of the development. These displacements were not always easy to recognise, but on 125 level the most easily recognised displacements were on steep N-S trending slips on the north eastern periphery of the block. Here, almost all exhibited lateral displacements of a few millimetres to the left. On the eastern side of the block both left-lateral and right-lateral displacements were noted. The greater rates of displacement recorded on the survey traverse were in the period April to May 1968 and the relative peg displacements confirmed the displacements observed on the slips and shears, as the pegs along in the east were displaced towards the north relative to the north-side and base pegs. In the northern and western peripheries dilation and a few lateral displacements were noted on steep dipping E-W striking slips. A rapid displacement towards the developing cave was shown by the survey traverses into this area in April and June 1968.

The displacements observed on the slips and recorded on the survey traverses on 170 level were very similar to those noted on 125 level. In the eastern periphery on 170 level the caving of Block 7AB resulted in similar renewed and new dilation and left lateral displacements on the

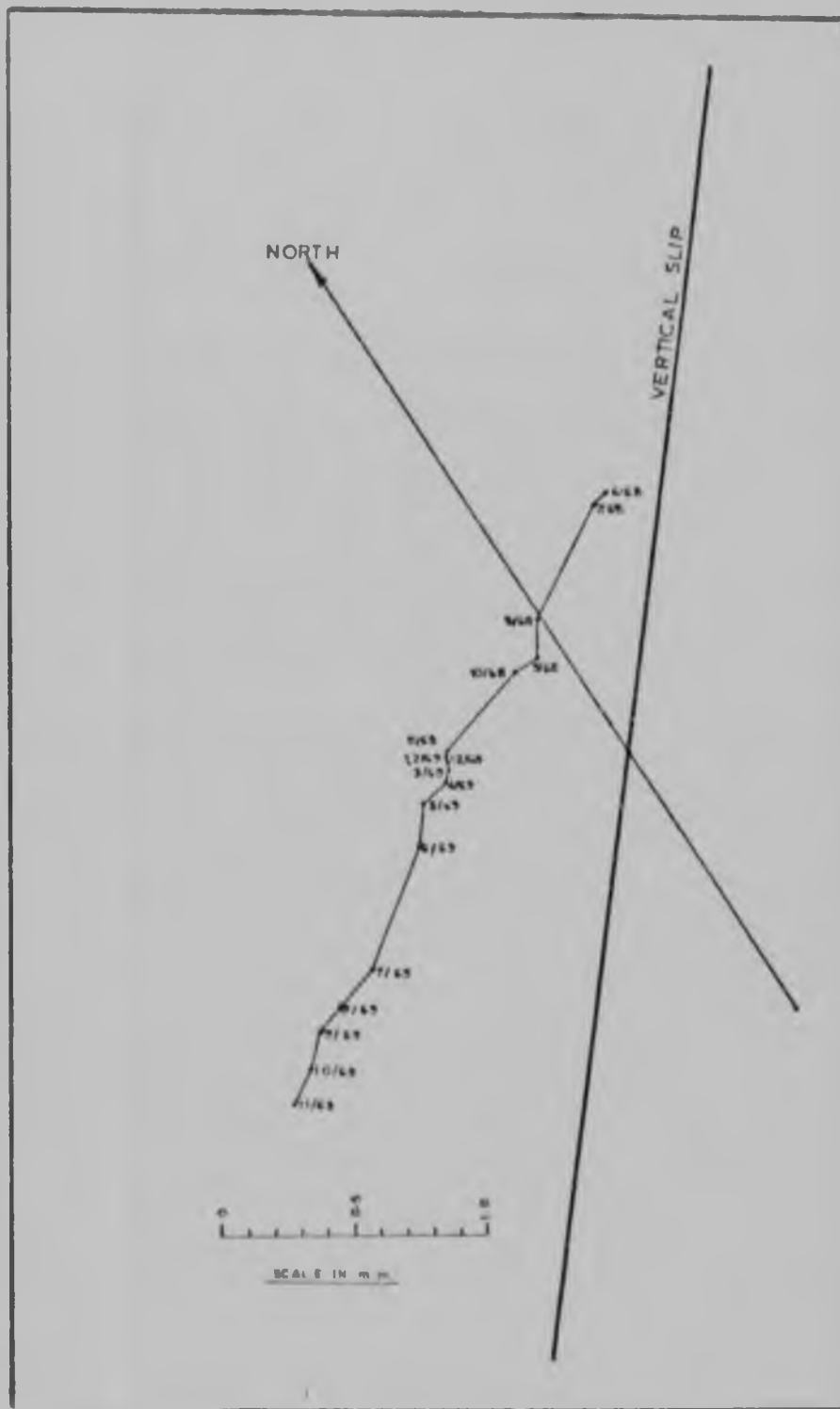


Fig 6,22 Displacement monitored on Demeo extensometer point No 41

N-S trending slips, and on the E-W trending slips both left-lateral and right-lateral displacements. Many of these slips were monitored with the Demec extensometer from May and June 1968. Fig 6,22 illustrates the left lateral displacement on a large vertical NE-SW striking slip in this area. The displacements on these slips were accompanied in this area by severe cracking of the pillars and sidewalls, local opening and shearing of slips and minor rock falls in the vicinity of the No 3 shaft station. To investigate the damage in this area two stress measurements were made in June 1968 (Heslop and Cook 1968) (Fig 6,23). These measurements revealed that the vertical stress at the two sites were less than 1MPa above and below the 6MPa vertical virgin rock stress estimated from the stress-depth relationship given above in 5.5. The stress in the crosscut direction at the two sites were 72% and 56% above the estimated 6,75MPa virgin rock stress, while the stress in the drive direction were 24% and 34% below the estimated 11,5MPa virgin rock stress. It was concluded that this damage was largely due to the reduction in the lateral stress at right angles to the crosscut rather than the increase in stress in the crosscut direction as the former could be expected to have a greater effect on the stresses around the crosscut than the latter. This reduction in stress would affect stability of the wide E-W spans in the vicinity of the shaft station causing the small pillars in the area to carry an increased load and crack.

The footwall drive on 170 level in the northern periphery was extensively concrete-supported and this was locally damaged between April and June 1968. Where the ground was not concrete-supported, regional dilation of E-W striking slips was most commonly seen. The survey traverse showed the greatest rates of displacements in the period May to September 1968. On the east side of the block subsidences of up to 11mm were recorded on Peg X 3045 and the pegs in this area were displaced relative to the base pegs up to 12mm northwards (towards Block 6). No significant lateral displacements were recorded along footwall drive relative to the base pegs (see below).

On 205 level, the pattern of regional dilation and displacement on slips was in general similar to those noted on the levels above. The area in the vicinity of 3 shaft station also suffered damage similar to

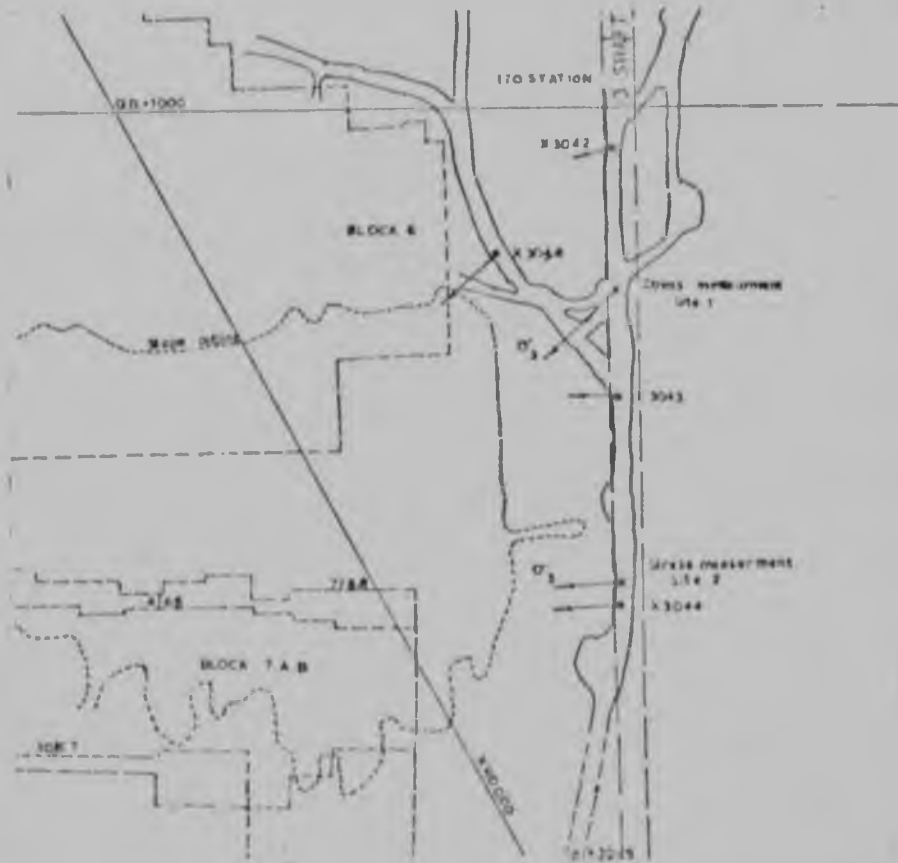


Fig 6,23 Stress measurement sites



Fig 6,24 205 level Survey peg displacements, and displacement on slips

that in the same area on 170 level. However, on the eastern side of the block an interesting normal fault type of displacement was noted on the Zone B shear which dips eastwards at  $35^{\circ}$ . This is taken to represent the tilting of the ground beneath the shear towards the cave block to the west (Fig 6,24). This is an example of displacement on a shear with a dip 'favourable' to stability i.e. dipping away from the cave block, and is similar to others described by Johnson and Soule (1963) and Heslop (1974).

On the northern side of the block small lateral displacements were noted on the Zone A shear, and in two localities this displacement dislodged blocks from the sidewall and back of the footwall haulage. In one of these localities the low Class 4 sheared ground was associated with this shear, and the movement caused several major rockfalls here over a period of time (Fig 6,24). This form of damage has been called 'secondary induced damage'.

On 235 level on the footwall side there were relatively few slips which displayed the type of displacement observed on the levels above. On the eastside of the block in footwall drive, displacement was recorded on an extensive south dipping slip, which appeared to be a local effect.

The survey traverses on 125 and 150 levels both showed no significant displacement to the footwall (north periphery of Block 6) relative to the base pegs, while the surveys of 205 and 235 levels beneath these areas displayed significant southward displacements. It is therefore concluded that none of the base pegs on these four levels were located in completely stable ground. The levelling results from 170 and 205 levels showed anomalous small rises in the footwall indicating the possibility of base peg subsidence. The peg displacements illustrated in Fig 6,24 have been adjusted to compensate for a slight displacement of the base pegs. The surveys were not frequent enough nor sufficiently accurate to permit a more detailed correlative analyses of ground displacements and mining activity as has been done in the next section.

#### 6.823 Effects of undercutting the second phase

In the "second phase" the drawpoint design was changed from troughs to pre-cut cones, and it was intended to shrink 245 and 255 levels. To do this a slot was developed and cut on both levels across the entire length of the block during May and June 1968. The troughs were then extended to the slot position and the first cones were cut in October.

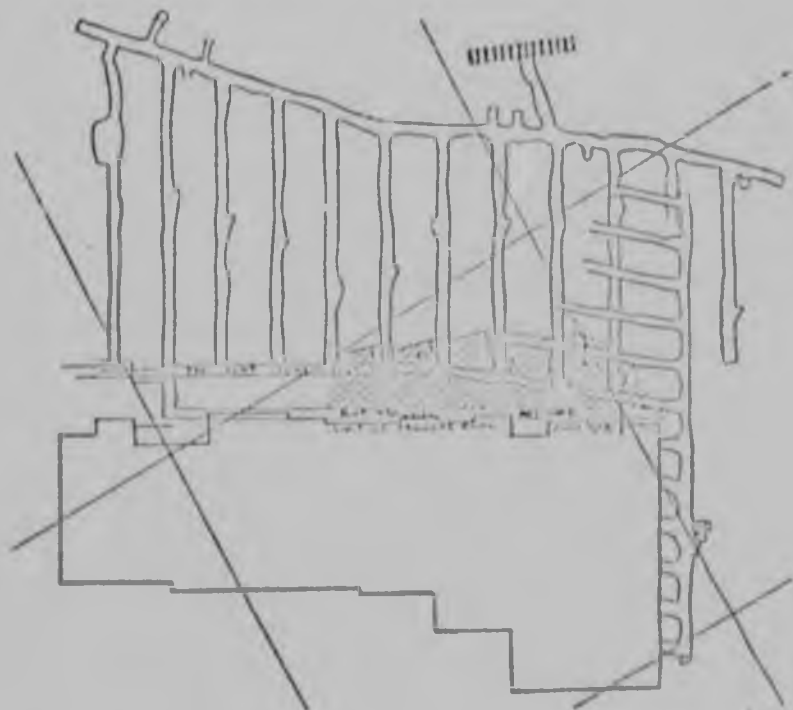


Fig 6,25 Area damaged by gravitational sliding on a south dipping slip  
245 level

In the intervening period, July to October, a 10m wide zone developed to the north of the slots in which there was a general opening up of east-west striking slips, and small horizontal displacements took place on the steep N - S striking slips. Further, sliding occurred on three slips dipping south at about  $45^{\circ}$ , during July, which caused the loss of the shrinkage blastholes and rock falls in 5, 7 and 8 crosscuts on 245 level and 7 and 8 crosscuts on 255 level (Fig 6,25).

In the footwall there is a major shear which cuts through 235-295 level travelling way, and became known as the Travelling way shear (Fig 6,1) and has been monitored for displacement on it by closuremeters from mid 1968 to date, and for a period by the Demec extensometer. The apparently

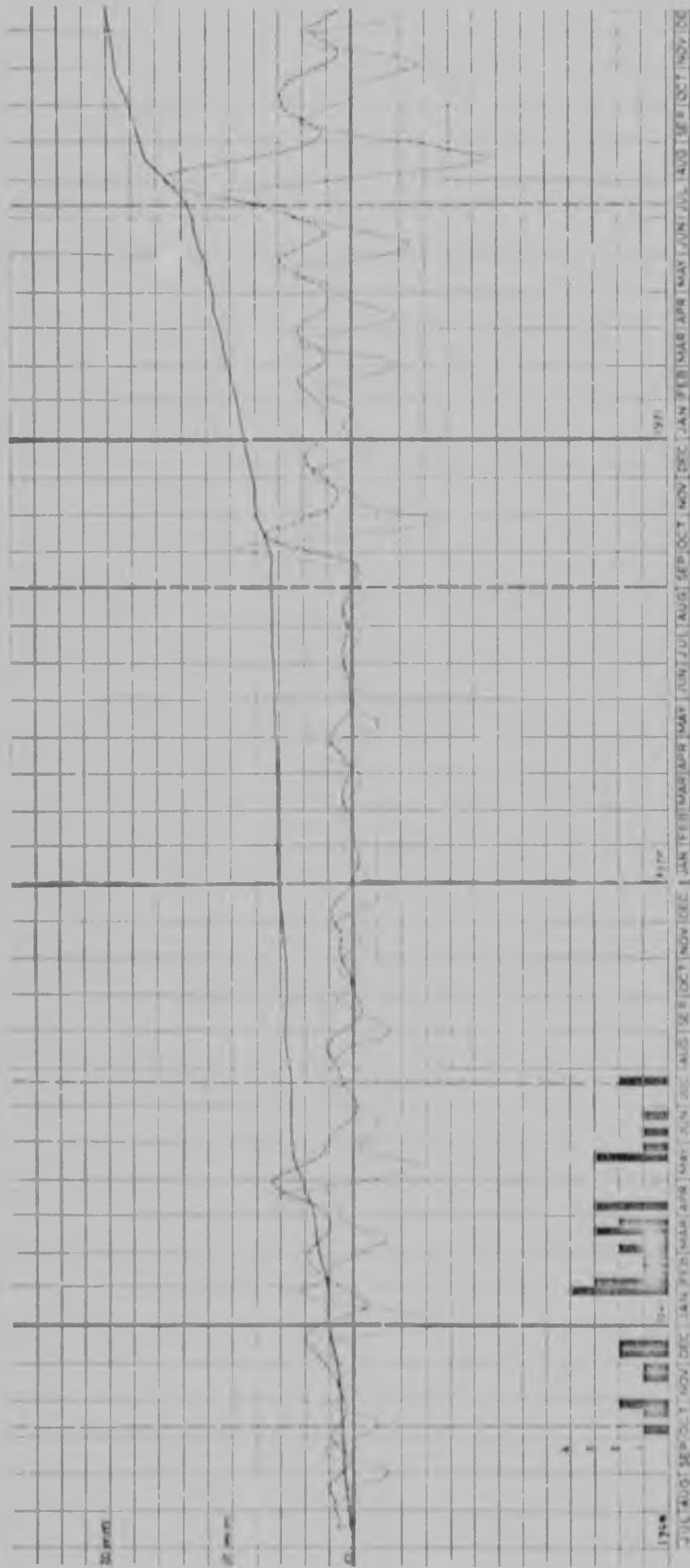


Fig 6,26 Displacement on The Frevelingway Shear related to mining activity.

Solid: 5-week moving average-displacement

Dashed: Rate of displacement based on above

Dotted: Acceleration based on above

Histogram illustrates number of drawpoint cones cut per week in the second phase

erratic movements took place on the shear from the start of monitoring, and these could be correlated with the coning of drawpoints and crashing of the ground overlying in the second phase. When this operation was completed, the rate of displacement on this shear slowed markedly. Fig 6,26 illustrates the total displacement, rate of displacement and acceleration recorded on this shear and a histogram showing the number of drawpoints crashed per week. The acceleration curve and the histogram of the number of drawpoints crashed showed a close similarity which becomes even more striking when it is pointed out that crashing the ground overlying the drawpoints followed a few days after the coning of the drawpoints.

Despite the displacements recorded on the Travelling way shear, the undercutting of the second phase has only a relatively small effect on the peripheral rocks. On surface visible changes continued to occur with more subsidence and the appearance of odd new cracks. The pegs on the south side continued to move slowly, while the 3/6 rib pillar moved towards the block at an increasing rate until the pillar cracked and the surveys were discontinued.

Underground on 120, 170, 205 and 235 levels, small displacements continued to be recorded by the levelling and survey traverses and by the remaining instruments. The Demec extensometer monitoring points recorded slow displacements with some slips showing a stick-slip form of movement.

#### 6.824 Discussion

The caving of the ground overlying the undercut and the old cut and fill stopes had the effect of creating a near vertical bin filled with large columnar slabs, and in forming this, considerable changes in stress took place; stresses normal to the periphery of the bin were reduced and redistributed in the peripheries and below the cave. It is deduced from the cracking and damage observed in the peripheries that the lateral stress in the lateral stress direction were reduced in the normal to the periphery of the bin, and increased in the other direction. The extent to which slips are affected by these changes is evidently dependent on the properties of the slips. Weak gouge filled shears such as those occurring in the main talc zones have displayed displacement considerable distances from the mining. Closer to the cave the stronger, less continuous slips are affected. From the observations of damage at

the time of caving and the survey and Demec extensometer records the maximum rates of displacement were reached on most slips shortly after caving reached surface, and relatively high rates of displacement continued for some weeks before slowing down to a stop over several months.

The removal of the lateral support resulted in the collapse of certain sections of 245 and 255 levels when sliding occurred on weak gouge filled minor slips dipping towards undercutting troughs.

It was expected that there would have been some damage due to abutment loading in the peripheries of the 235 level stopes, but in the northern abutment of the stopes the ground competency was high, estimated to be Class 1B or 2A and despite extensive development openings there was only a little evidence of such damage in the form of minor cracking and spalling. In the southern abutment on 235 level there was no direct evidence of increases in vertical loading, but rather evidence of lateral relaxation from the closuremeter measurements located in the periphery. The rock in this area is estimated to be Class 3A or 3B, in places had previously been supported, and some extensive damage and collapses occurred as undercutting and caving progressed.

#### 6.83 THE EFFECTS OF UNDERCUTTING AND PRODUCTION ON THE GROUND BELOW THE UNDERCUT

The development below the block consisted of the grizzly horizon, 265 level, an intermediate tramming level (275 level) and a main tramming level (295 level). This level was supported initially with rockbolts, each crosscut having a different type of rockbolt, for example, tensioned bolts, grouted and tensioned rockbolts, grouted re-bars etc.

265 level was instrumented with Horstman bolt tension meters, and closuremeters. In addition three load cells were installed on a crown pillar in the first phase and two in the second phase. Three Horstman photo-elastic stress meters were also installed. On 275 level the instrumentation consisted of survey and levelling traverses and two photo-elastic stress meters.

##### 6.831 First phase undercutting

Very shortly after undercutting started damage was evident on the grizzly horizon and in November-December 1967 the undercutting had to be

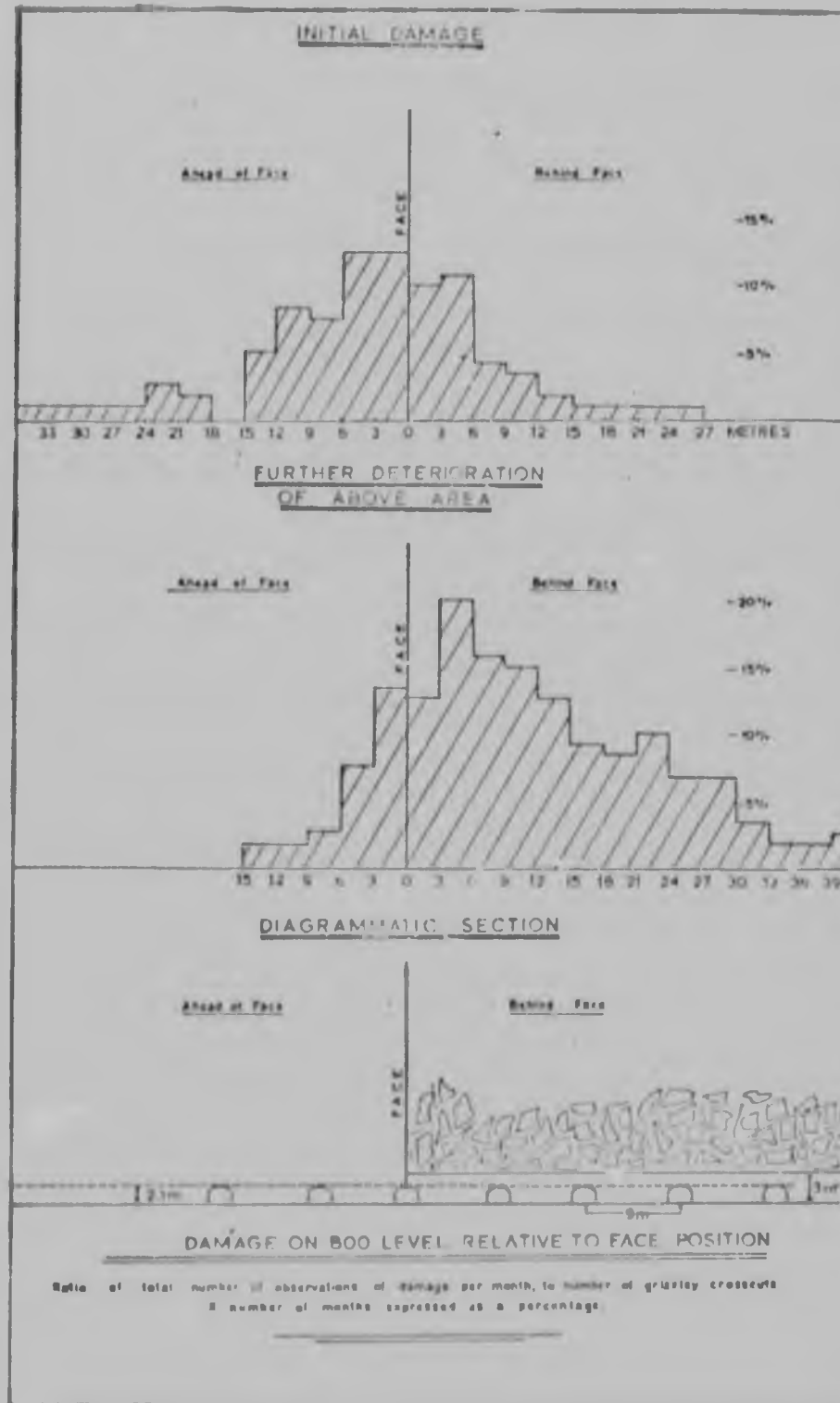


Fig 6.7 Location of damage in grizzly crosscuts relative to face position

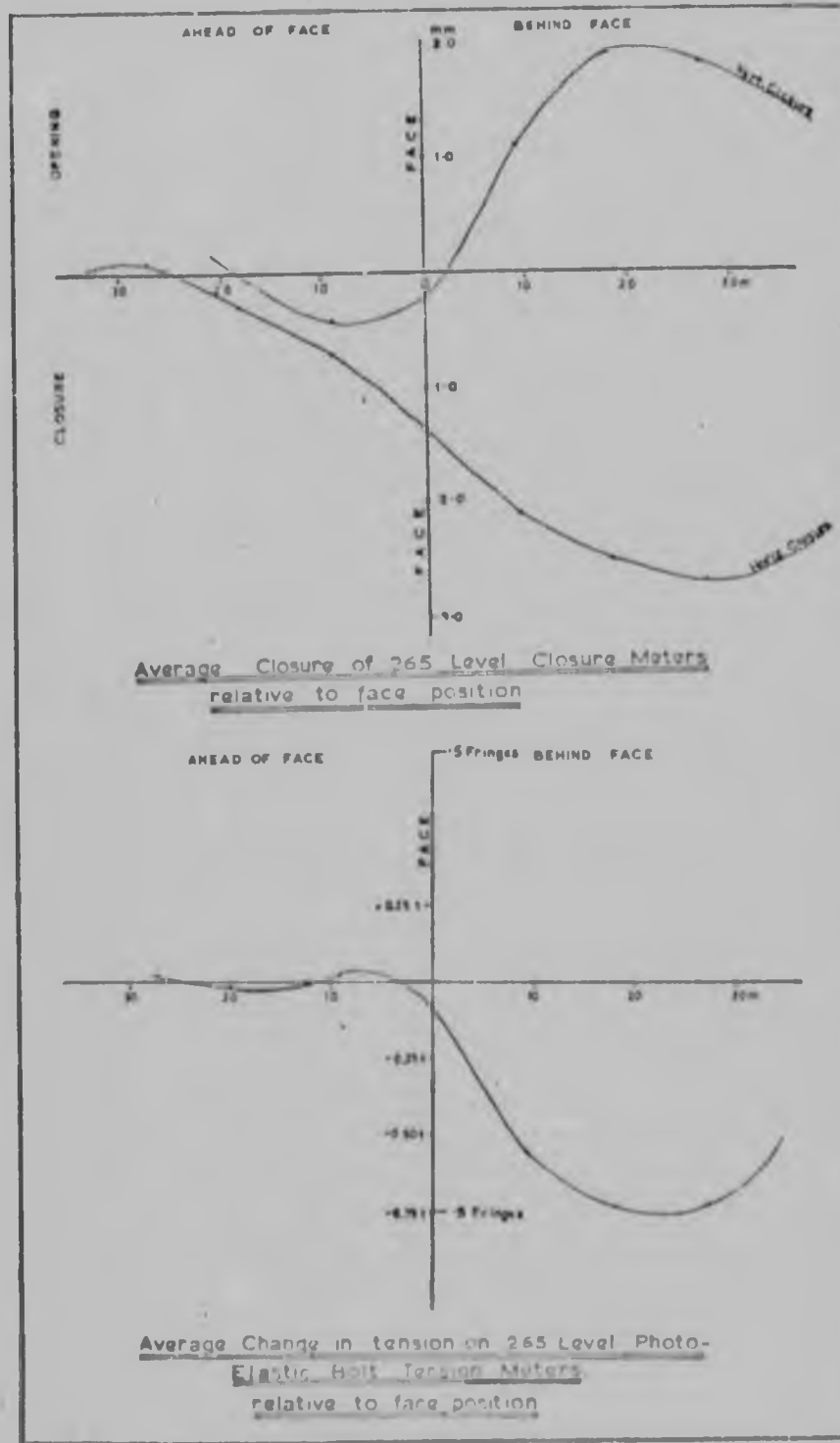


Fig 6,28 Average closure and rock bolt tension recorded in grizzly crosscuts relative to face position

brought to a halt while yielding arches were installed in the less competent zones. This damage was attributed to the very blocky, weak nature of the ground, the generally higher stress environment in the abutment of the 235 level stones and to blasting shock waves. When undercutting was resumed no further serious damage was experienced in any grizzly crosscuts but, minor changes, in the form of light spalling and dislodging of loose blocks and minor cracking, were observed occurring ahead and just behind the advancing undercut face. The closuremeters and bolt tension meters also showed changes which were related to the face position. However, in every case, damage was visible before responses were detected on the instruments. These relationships can be seen in Figs 6,27 and 6,23.

Significant upward movements of 5 to 7mm in the centre of the block were recorded on the survey pegs on 275 level. The surveys showed the southern most pegs in the tramming crosscuts were displaced up to 15mm northwards relative to the northern most pegs on this level during the period October 1967 to February 1968. In general only minor damage was noted on this level during this period.

#### 6.832 Second phase undercutting and coning

The pre-coning of the second phase started slowing in October 1968 with a few drawpoints per month which was later increased to 16 drawpoints per month. This operation was completed in June 1969.

With the advance of the pre-coning operations only light damage occurred just ahead of coning as had been observed in the first phase. The horizontal closuremeters recorded similar lateral extension, whilst the vertical closuremeters (installed only in this phase) showed a vertical contraction (See Fig 6,28). No bolt tension meters were installed in this phase.

On 275 level, the levelling and survey traverses after recording small falls further slow rises of the ground below the rock were recorded with some further displacement of the pegs towards the middle of the block. No other significant changes were recorded.

#### 6.833 Production from the first phase

During the pre-coning and undercutting of the second phase, the drawpoints in the first phase were worked on a rotational basis, each

drawpoint being "issued" (required to be worked) one day per week to prevent consolidation of the ore in the draw columns. But because of the small tonnage called for, not all drawpoints were worked when issued and the result was a very irregular state of draw. The average rate of draw for these drawpoints was less than 25mm per day. In November 1968, a nine metre length of 7 grizzly crosscut was heavily damaged, with opening of vertical cracks which appeared to extend right through the crown pillar. This area was immediately supported and the nearby drawpoints were worked continuously. In March 1969, similar damage occurred in 1, 2 and 3 grizzly crosscuts, which spread and deteriorated in subsequent months (Fig 6,29). In May further collapses occurred in 6 and 7 grizzly crosscuts with further damage and extensions to the damaged areas in the following month. The damaged area was supported with 25kg T.H. yielding arches installed, initially at 0,90m centres and subsequently at 0,45m centres. As soon as possible after the damaged drawpoints had been supported they were brought into full production in an effort to keep the grizzly crosscuts open to complete drawing of the ore, but repairs and replacement of arches were required frequently. At full production the draw rate remained at less than 100mm/day, a relatively slow rate when compared with other block cave mines.

The load cells installed on the top and side of the crown pillar in 7 grizzly crosscut recorded the build up of loads prior to the collapse of the crown pillar.

The calculation of the loads on the load cells without making assumptions proved impossible due to the inadequate number of strain gauges. But if it is assumed that the loads were a limited number of point loads acting on narrow longitudinal zones of the load cell then an estimate of the maximum point loads can be obtained.

The two 1,2m long load cells, Nos 1 and 2, were installed on top of the crown pillar, No 1 being above 7/9 drawpoint and No 2 between 7/9 and 7/10 drawpoints. The cells were set in concrete so that the tops were 101,6mm higher than the surrounding concrete and thus the edges of these cells were highly loaded. The readings largely reflect these loads (Figs 6,30 and 6,31). They show that during the low draw period of December-January 1969 the western edge of No 2 was highly loaded, but no excessive loads were recorded on No 1, possibly due to arching over this cell. Some very high loads were recorded on No 1 during the March - April low draw period, while there was an increase in load during May on No 2. These cells have also shown that the arrangement of rock on the load

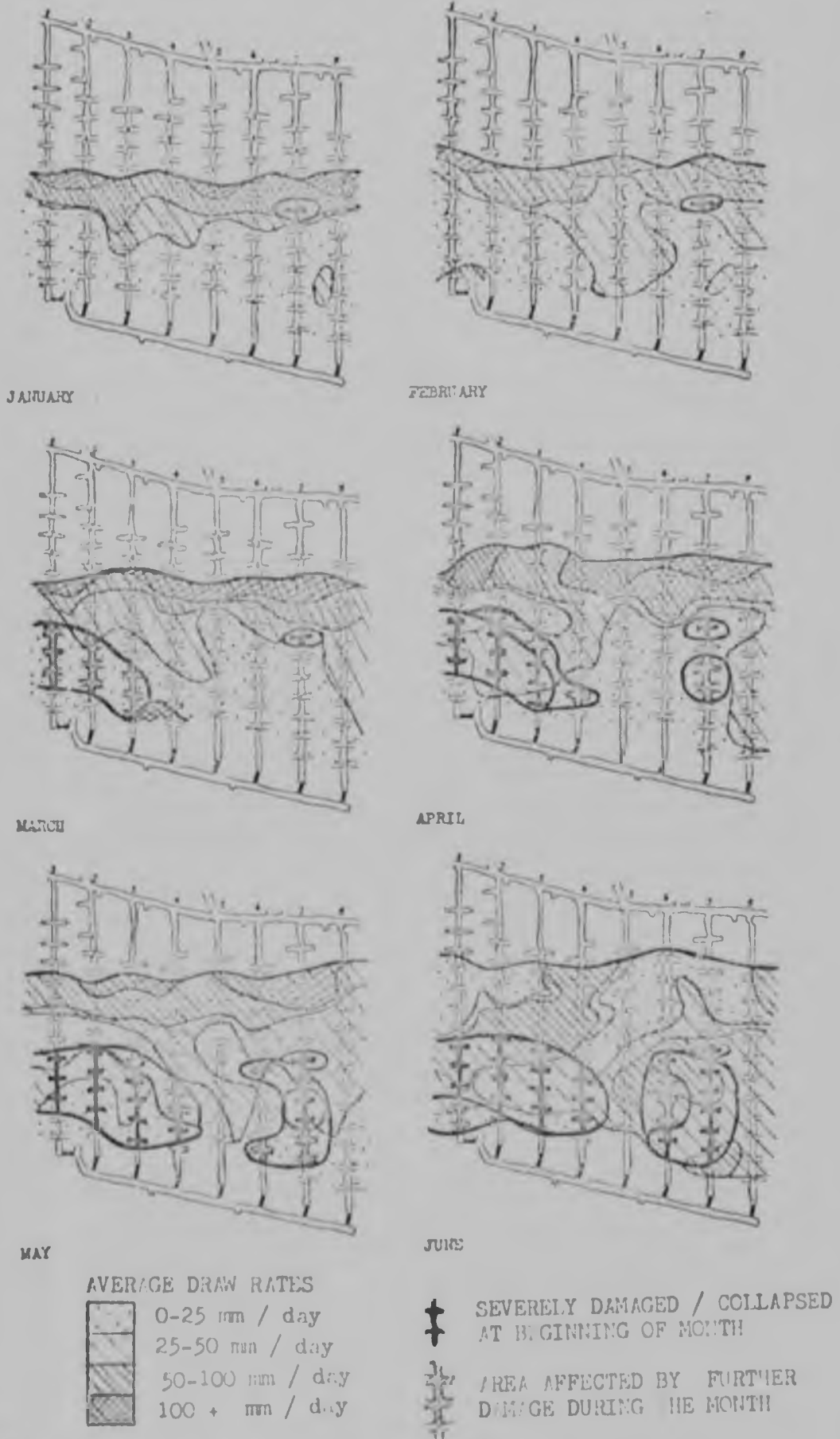


FIG 6,29 Progressive failure of crown pillars on 265 level related to rates of draw

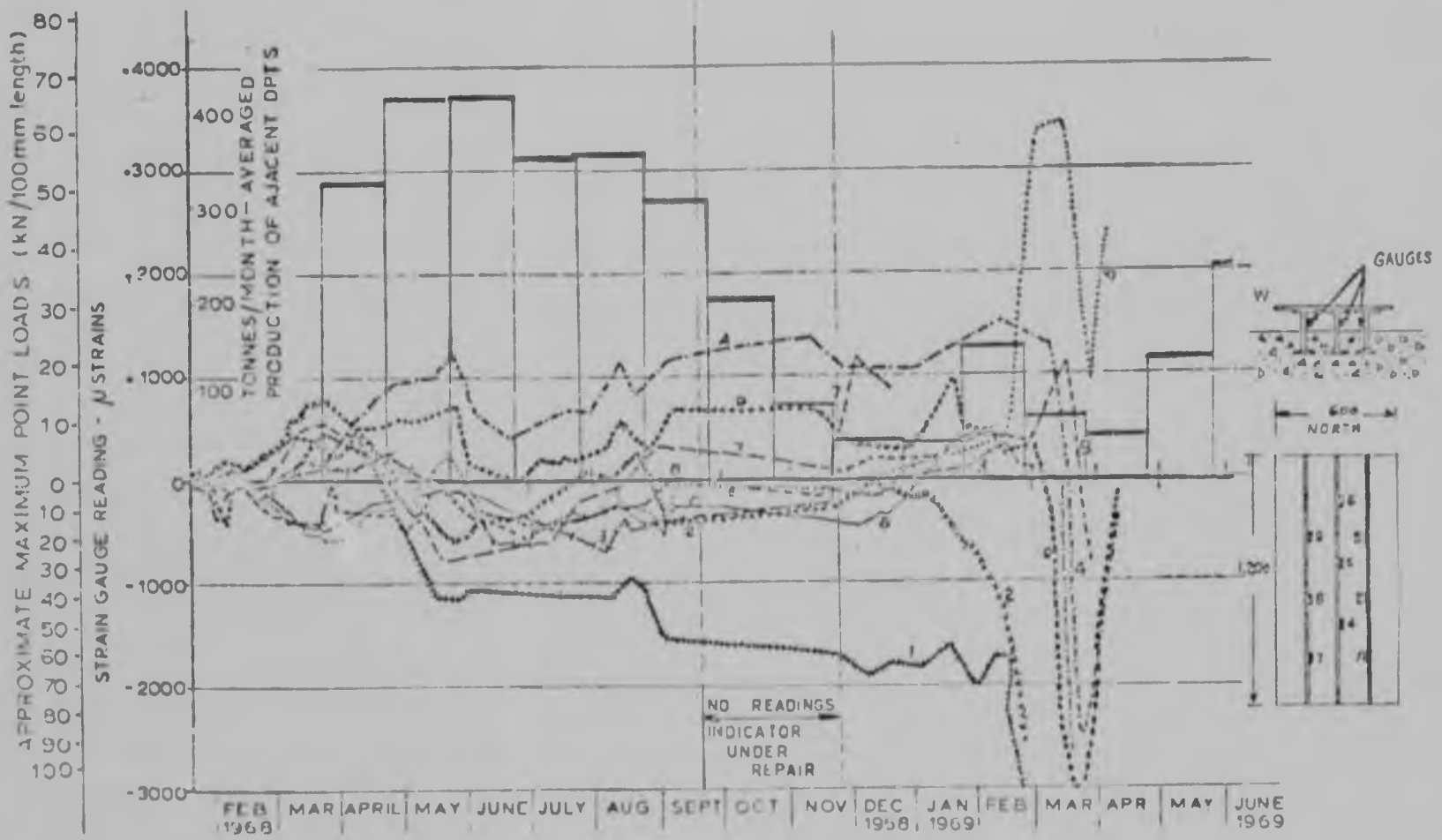


Fig 6,30 Load cell No 1 results monitoring broken ore pressures on crown pillar crest

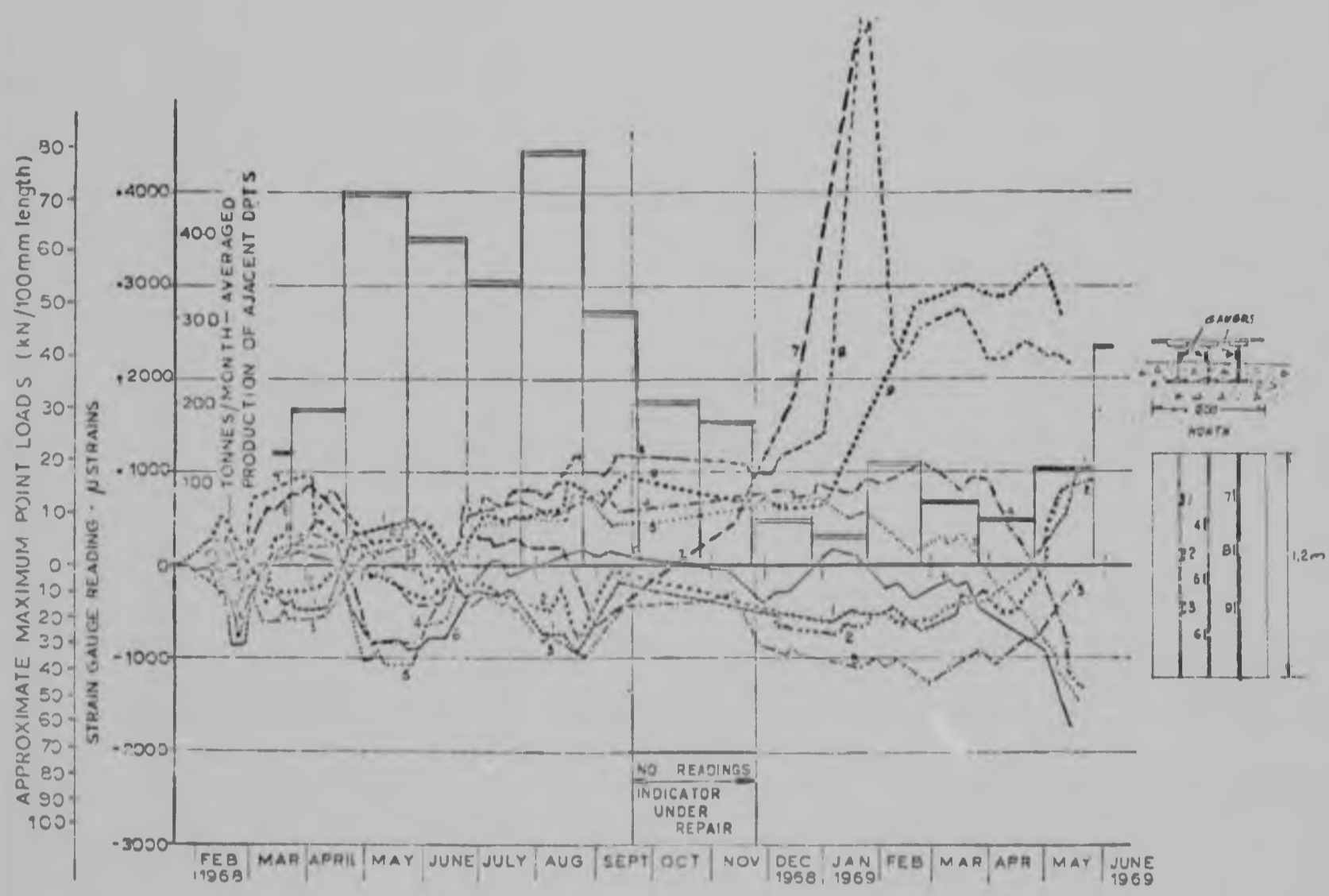


Fig 6,31 Load cell No 2 results monitoring broken ore pressures on crown pillar crest

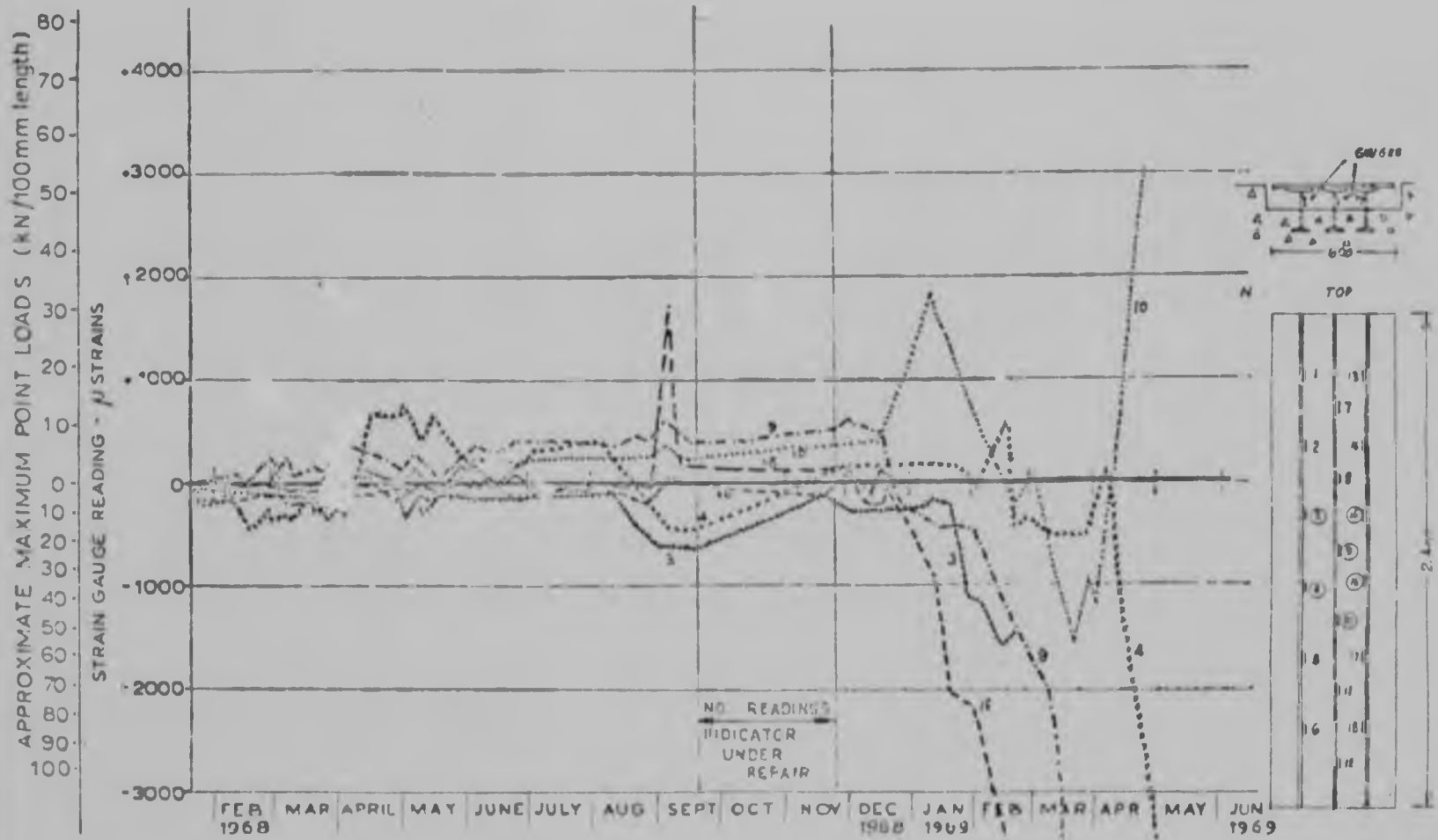


Fig 6,32 Load cell No 3 results monitoring broken ore pressures on side of crown pillar

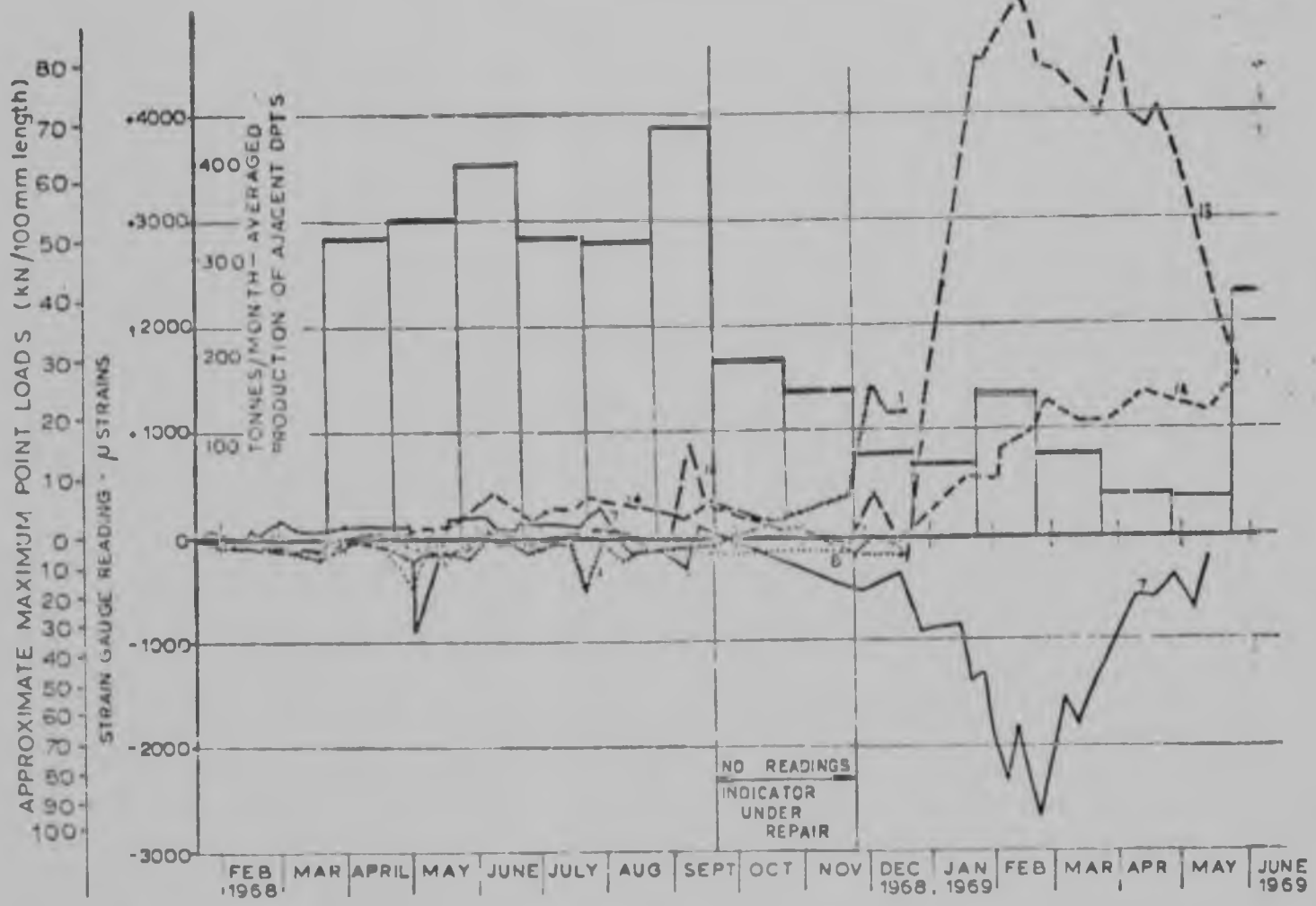


Fig 6.33 Load cell No 3 results monitoring broken ore pressures on side of crown pillar

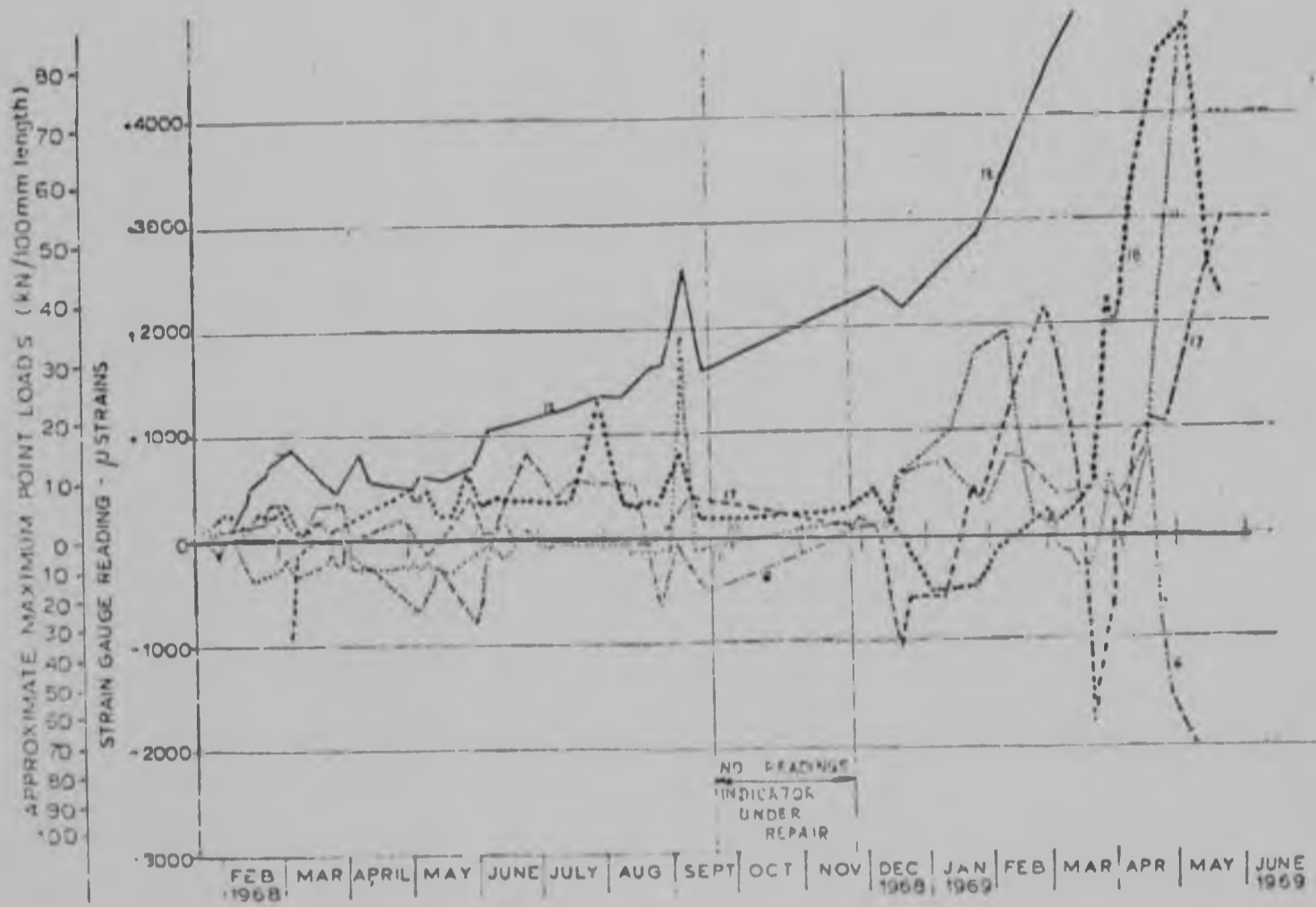


Fig 6,34 Load cell No 3 results monitoring broken ore pressures on side of crown pillar

cells change at wide time intervals, indicating a slow movement of the ground immediately above the crown pillars.

Load cell No 3 which was 2,4m long was installed on the side of 7/9W drawpoint crown pillar. The top was flush with the surrounding concrete and, therefore, the edges were not as highly loaded as the other two cells, and the results are not comparable. The results have shown that large increases in load occurred during the low draw period of December-May and that the greatest arching forces occur between 1,2m and 2,4m below the top of the crown pillar (Figs 6,32; 6,33; and 6,34).

The damage to the crown pillars in this phase is probably the result of a combination of factors:-

- a) the mass subsidence caving of the ore between 235 and 255 levels which gave rise to poor fragmentation. Secondary blasting accounted to 50% of the cost of working this ore.
- b) Inadequate crown pillar design, which was without lateral buttressing pillars, and to the undercutting method which subjected the crown pillars to heavy blasting shock and high stresses which may have destroyed the cohesion of the slips.
- c) Incompetent ground conditions. Two thirds of the drawpoints in the first phase in incompetent blocky ground (Class 4A and 3B) have been severely damaged as compared with one half of the drawpoints situated in more competent ground (Class 3A and 2B).
- d) The practice of only drawing small tonnages from the drawpoints at wide spaced intervals giving rise to uneven draw conditions. Fig 6,29 shows how the areas of low draw rates were subsequently heavily damaged. The load cells installed on the top of and on the sides of the crown pillar above 7/9N drawpoint and between 7/9 and 7/10 drawpoints showed increases in loads during the low draw periods prior to the severe damage to the crown pillars in this area.

#### 6.834 Production from the second phase

After each second phase drawpoint was coned and overcut it was worked at a moderately high rate until July 1969, when all coning and production in this phase was stopped and production was switched to the rapidly deteriorating first phase. Nothing was drawn from these drawpoints until a grizzly crosscut was brought into production in August 1970. Thus for

over a year the southern most line of drawpoints in this phase stood, unworked, adjacent to the continuously working first phase drawpoints. Only slight damage in the form of further displacements on slips and light spalling was observed in this line of drawpoints during this period. Similar observations were made elsewhere in this portion of the block, and the severity of this damage appears to have been uniform over the whole phase, and not more severe along the southern boundary.

The second phase was brought into production again in August 1970. Starting on the east side, with No 8 grizzly crosscut, one crosscut was brought into production at a time, in a panel retreat. During this period some cracking and displacements on slips were noted, but these never approached the severity experienced in the first phase. Even towards the end of the life of the block, when certain drawpoints were stopped, and others around them continued to be working no serious damage was noticed.

The superiority of this area is attributed to the higher rock classification, the better drawpoint design, better undercutting method, and possibly the draw control policy.

#### 6.9 SUMMARY

The instruments gave reasonably good results, which when combined with the visual observations made it possible to deduce the caving mechanisms operating in this block. However, the presence of the 235 level cut and fill stopes was a complicating factor in the analyses of the observations in this block. Therefore, further observations from other blocks are described in the following chapter, and the discussion of these conclusions is deferred to Chapter 8. It is interesting to say that in this block two distinct caving mechanisms were recognised, viz 'mass subsidence caving' in areas where the lateral stress had been reduced in one direction and 'stress caving' where no such lateral stress reduction had occurred.

The results obtained from and the experience gained in the use of the various instruments and techniques was used in deciding the form of instrumentation to be used in subsequent blocks.

In general, the surveying techniques required a considerable amount of skilled observer time, and on most levels the base stations could not be located in stable ground, therefore, less emphasis was placed on these techniques in subsequent blocks. The remote displacement meters and bore-

hole wire extensometers proved to be reliable and easy-to-read, and with refinements to improve accuracy could be used more extensively and in some cases replace surveying techniques.

The glass plug stressmeter proved to be not sensitive enough and subject to very local variations in rock modulus of elasticity and local structures. Also the patterns proved were difficult to interpret. Consequently this meter was not used in any other blocks described, but where stress measurements were required overcored photo-elastic disc was used.

The caving indicator failed to record the caving progress between the stopes and surface. Consequently borehole wire extensometers were preferred in areas where mass subsidence type caving is anticipated. The smaller instruments such as the Demec extensometer and closuremeter proved valuable in analysing the displacement on slips and cracks. However, the results were not comparable with one another; but were meaningful when rates of displacement were compared with the previous record of the instrument. For succeeding blocks, further Demec extensometer reading points were laid out, but with one exception, no more closuremeters were used. Instead the dial extensometer was used in its place to improve the accuracy of the readings.

The long term instability of the electrical resistance strain gauges and/or difficulty in maintaining an effective waterproof insulation, combined with the large number of gauges to be read precluded the load cells from further use.

CHAPTER SEVENOBSERVATIONS FROM OTHER BLOCKS7.1 INTRODUCTION

In addition to Block 7AB described in the previous chapter, the caving of four other blocks have been studied. These blocks provide a far wider range of conditions, and puts this study of the caving process on a firmer basis. While three of the blocks described here were mined after Block 7AB, one, Block 6, was mined some years previously and is included because it is an important example of stress caving where a large cavity developed and was monitored as caving progressed to the surface.

7.2 BLOCK 6

Block 6 was one of the first blocks in which caving was investigated, and the results of this investigation played an important part in the thinking behind the planning of subsequent blocks such as Block 16. Block 6 caved through to surface on March 13, 1963, and certain facets of the caving were investigated by Cathoral (1963). The block lay immediately south of Zone A and block 3, above Zone B (Fig 7,1) and was initially mined by cut and fill stoping. In 1958 grizzly drives were developed below these stopes, and pre-breaking drives above them. The one overlying the drawpoints was shrunk on an east to west retreat and the drawing of the pre-broken ore resulted in the development of a cavity. The shrunk area was 100m east to west and 100m north to south at the time.

In the northern abutment of the cavity displacement was observed on the talc zone, Zone A crosscut on 135 level. Further information on this displacement was obtained from the two specially drilled holes from 125 level (Fig 7,1) which showed that during drilling in April 1962 a vertical displacement was taking place on the zone. Within 3 months the displacement was large enough to close the holes completely.

The cavity resulting from the removal of the pre-broken ore was investigated from three surface boreholes drilled during 1962. Two of which were rigged with caving indicators in September 1962. The depth

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The cavity resulting from the removal of the pre-broken ore was investigated from three surface boreholes drilled during 1962. Two of which were rigged with caving indicators in September 1962. The depth

of the back when intersected by these two boreholes was 45 and 62m. The third hole intersected fractured ground at 47m in the southern abutment of the block (Fig 7,1). When the cavity was intersected, some 70 to 100m vertical height of ground had caved, leaving a cavity of 6 to 13m in height.

Between September and December 1962 no falls were recorded from the back, but at the beginning of December the first of several falls were recorded, and in March 1963 the back reached the surface weathered zone. On March 13, the cave broke the remaining 20m through to surface dropping the surface by approximately 20m. This took place within a day of a major slot blast underground (Fig 7,1). Fig 7,2 illustrates the surface subsidence zone in relation to the mining at the time of caving and the final outline of the block in plan and Fig 7,3 the relation of first phase mining to the surface subsidence zone. The northern side of the subsidence zone was bounded by the major weak shear zone in the centre of Zone A giving an overhang of  $80^{\circ}$ . The other three sides of the subsidence zone was bounded by steep dipping slips. The subsequent westward extensions of the pre-break did not produce any extension of the surface cave outline. However, sloughing from the back of the cave of this western extension was monitored by two additional boreholes rigged with caving indicators (Fig 7,4).

From this block it was concluded that (1) caving took place as a series of sporadic falls from the back of the cavity. These falls could be triggered by blasting shock or 'lubrication' of the joints by water during the rainy season. (2) A minimum span of 45 to 60m was regarded in 1962 as the criterion for caving. Subsequently, the hydraulic radius of this block (area/perimeter) of 19,5m was adopted as the only criterion for caving. (3) The displacements recorded on Zone A in the northern abutment could have had the effect of reducing the lateral stress in the back.

### 7.3 BLOCKS 7/1 AND 7/3

#### 7.3.1 INTRODUCTION

Despite the subdivision into Block 7/1 and Block 7/3 these blocks were planned and worked as a single continuous sublevel caving operation. They are located immediately east of Block 7AB and undercutting started in August 1970 on the south east side of the block, retreating northwards

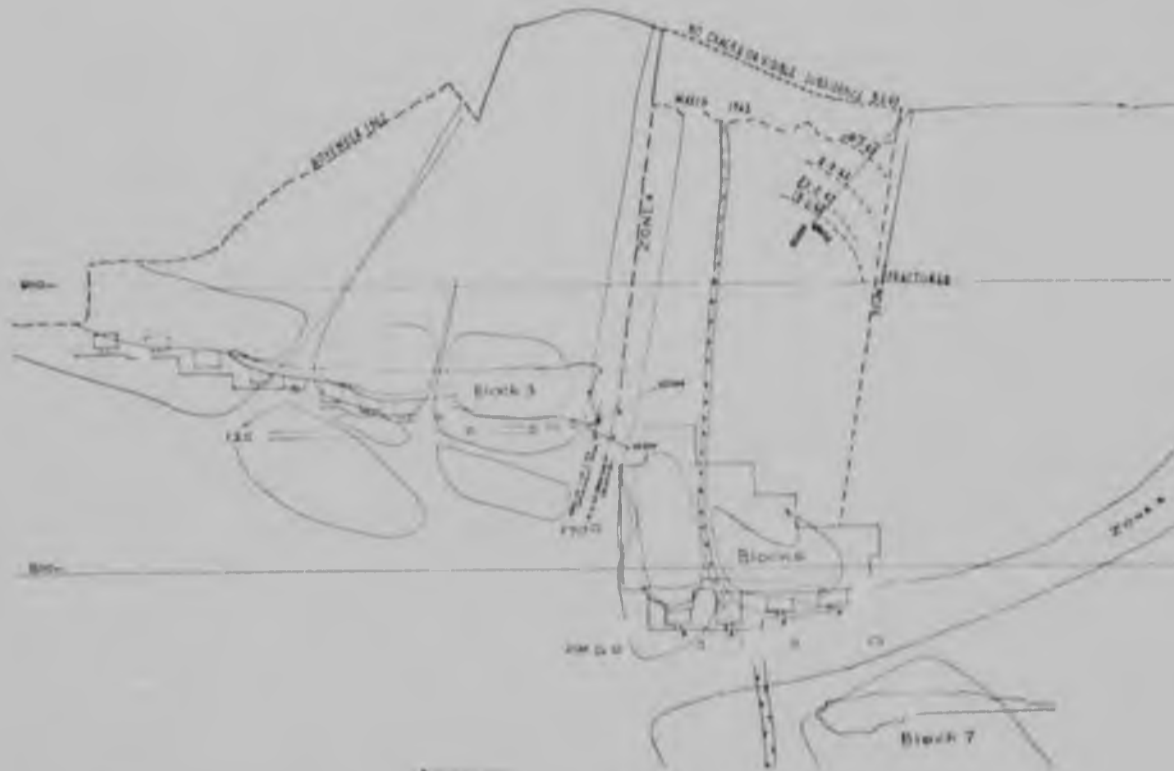


Fig 7.1 North-south section through Blocks 3 and 6



Fig 7.2 Plan of Block 6 surface subsidence zone relative to undercut area

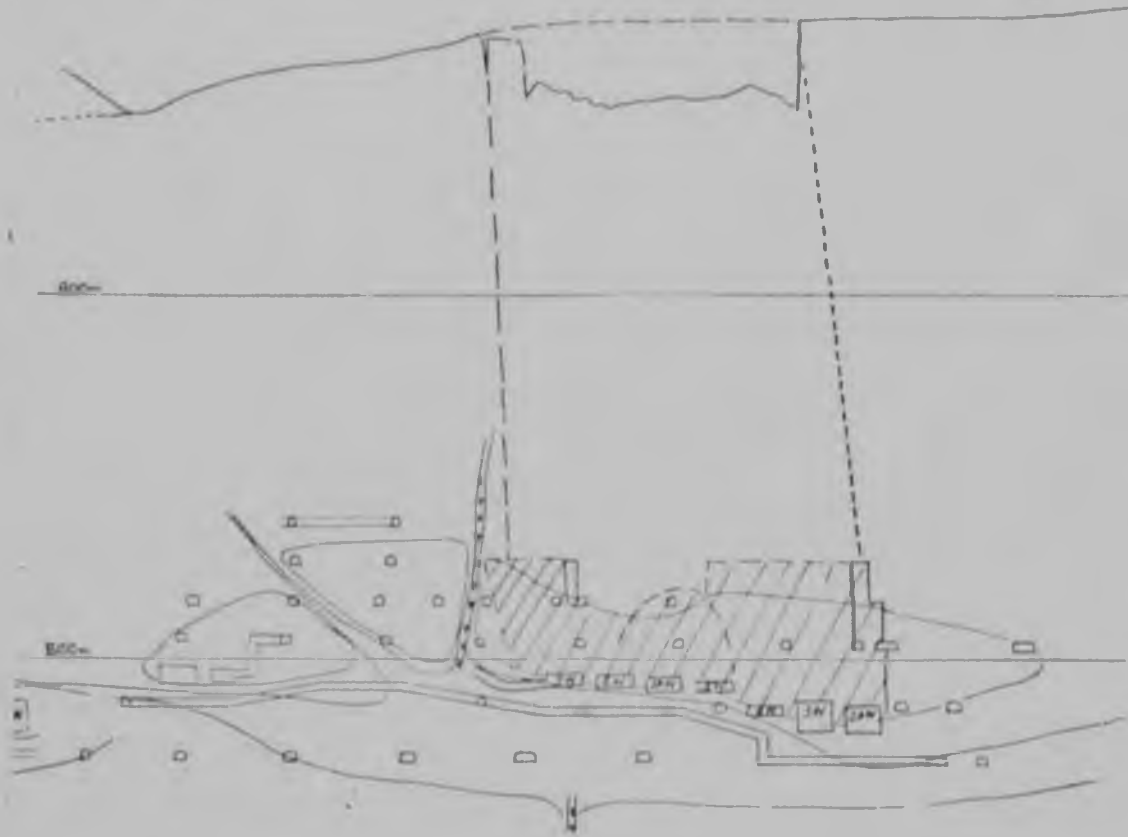


Fig 7,3 East-west section through Block showing relation of mining to surface subsidence zone (first phase)

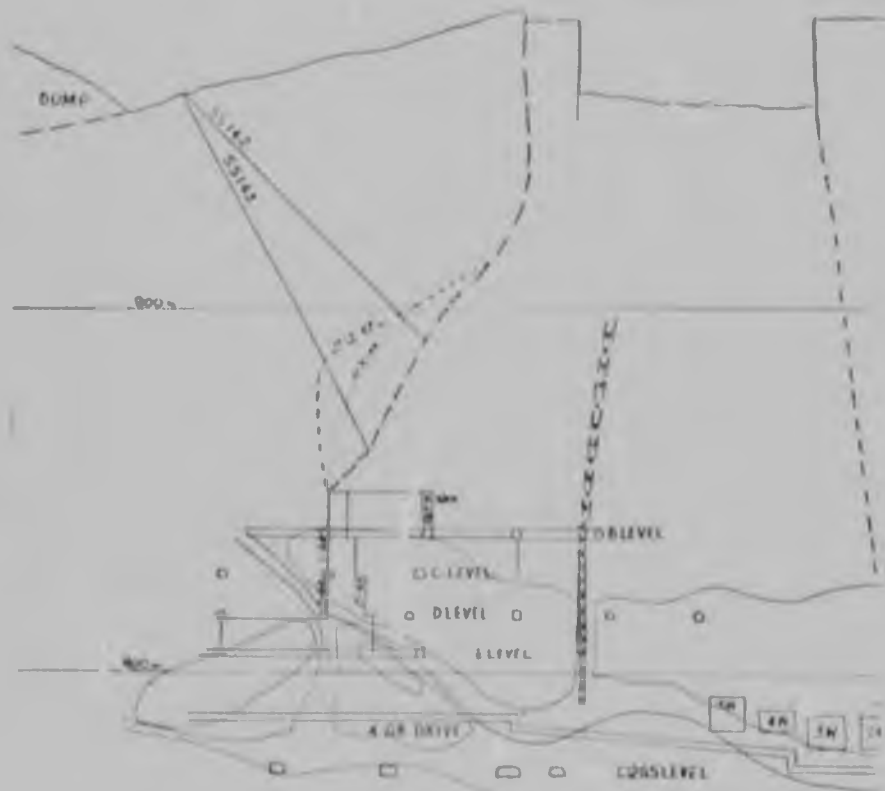


Fig 7,4 Block 6 cave outline with westward extension of shrinkage (second phase)

and westwards towards Block 7AB. It will be shown that a result of this direction of undercutting stress caving occurred in the early stages, and that mass subsidence caving occurred later due to the proximity of Block 7AB. It will also be shown that the caving of this block affected a very large area surrounding the block.

### 7.32 GEOLOGY

The geology of these blocks is very similar to the second phase in Block 7AB; the area is overlain by the shallowly east dipping Zone B talc-carbonate "sill" which merges over the eastern limit with the talc and shearing associated with the 170 dyke (Fig 7.5). This dyke is largely talcified, and talc-gouge filled shears are usually developed on the upper and lower contacts. The dyke is 10 to 20m thick and dips south east at  $50-60^{\circ}$ , its competency varies from Class 2 to Class 4 depending on the degree of shearing. Above this zone and dyke is the normal

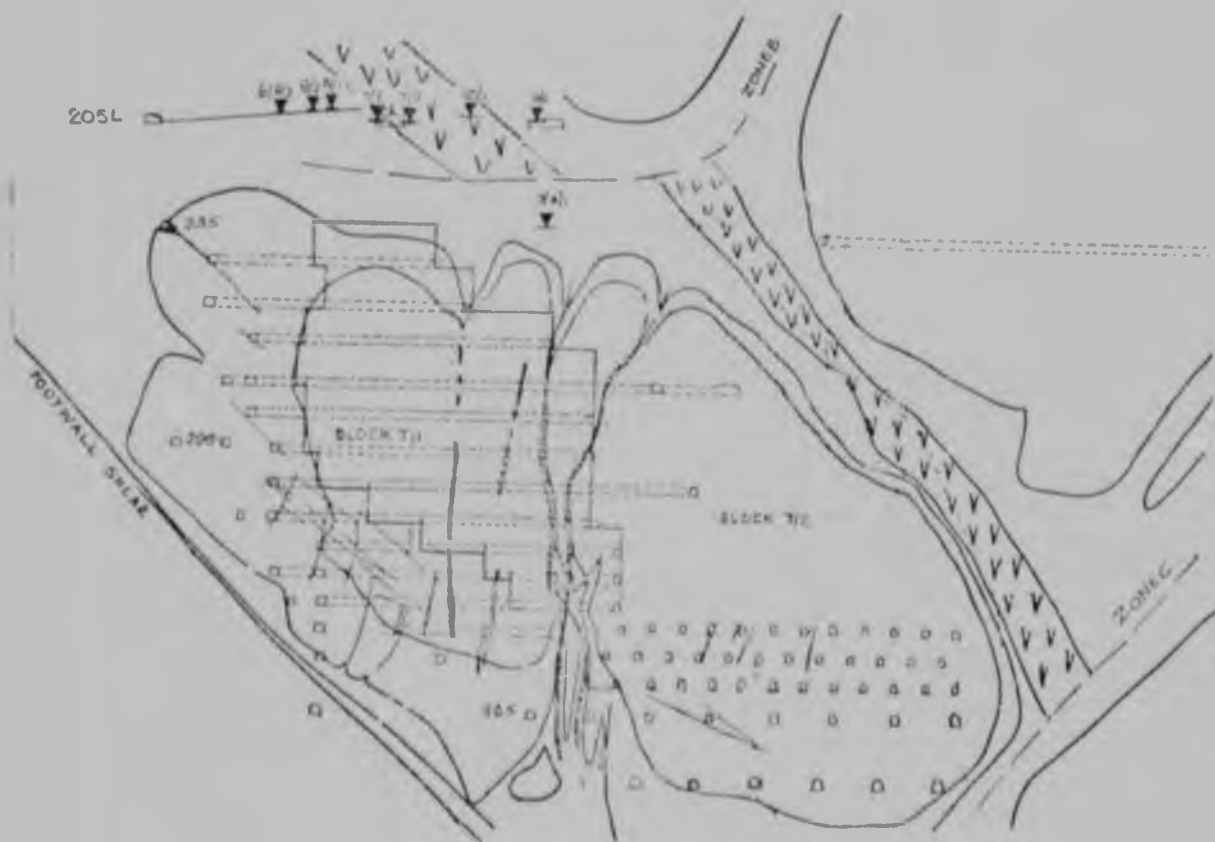


Fig 7.5 North-south section through Blocks 7/1 and 7/2

partially serpentinised hangingwall dunite. On the northern side of the area is the steeply N-dipping Zone A talc zone, and on the south side of Zone B. Below the dyke this zone flattens out to become the Zone B "sill" over the block. The orebody is very similar to the second phase of Block 7AB with a classification (estimated) of 2B with conjugate ribbon fibre seams dipping at 30-45° southwards. There are also south dipping slips, parallel to the fibre seams, some of which contain minor gouge filling. The largest of these is the eastward extension of the Travelling way shear which can be traced from Block 7AB through Block 7/3 to the boundary of Block 7/1.

Along the southern boundary with Block 7/2 are two larger aplite intrusions associated with an east-west shear. The ground associated with this, in the orebody is blocky and incompetent. This zone has acted as a channelway for hydrothermal solutions so that in the footwall the zone has a core of talc-carbonate rock with carbonated serpentine and brittle fibre on either side, and, on the east a broad tongue of carbonated serpentine with brittle fibre penetrating the orebody along this zone (Fig .,5).

### 7.33 PREVIOUS MINING AND GROUND CONDITIONS PRIOR TO MINING

The previously mined areas consisted of Blocks 4/5 above 170 level on the north, Block 3 above 125 level on the north west, Block 7AB and Block 6 on the west side as well as the small remaining portion of the Block 7, 235 level stopes.

An overhang had developed over the eastern side of Block 7AB as shown in Fig 6,17. Some cracks were visible on surface some 40m east of the eastern limit of the Block 7AB subsidence. On 125, 170 and 205 levels small movements on, and dilation of slips had been noted in the eastern periphery of Block 7AB over the back of Block 7/1-7/3. Access to 125 level had been lost because of deteriorating ground conditions north of Block 7AB.

The stresses given in Table 7,1 were measured on 170 level in the periphery of block 7AB, in June 1968 (See 6.822). If it is considered that some further caving and subsidence occurred in subsequent months and that further displacement was monitored on the slips in the area, it is expected that the vertical stress component given in Table 7,1 was

subsequently reduced slightly and similarly the horizontal stress in the drive direction. Some increase in the north - south stress is likely with the increase in the east-west dimension of caved ground.

TABLE 7,1

## 170 LEVEL STRESS MEASUREMENT RESULTS

Ov (vertical)	site 1 5,2 MPa	site 2 6,9 MPa
Oh (crosscut)	11,8 MPa	10,8 MPa
Oh (drive)	8,3 MPa	7,6 MPa

## 7.34 INSTRUMENTATION

The instrumentation in this block was not as comprehensive as in Block 7AB, firstly because of the mining method employed, and secondly due to the lack of access points. The instrumentation consisted of a surface borehole which was equipped with a caving indicator, remote displacement meters installed in the main crosscut on 170 level (Fig 7,6) over the northern side of Block 7/3. Remote displacement meters and borehole wire extensometers were installed in the boreholes drilled from and on 205 level in the northern and eastern side of the blocks and from 235 level remote displacement meters in two boreholes on the southern side of the block (figs 7,6; 7,7 and 7,8). Also the survey and levelling traverse in the footwall of Block 7AB was extended deeper into the footwall. The remaining extensometer points on all levels continued to be read, but the method of reading was not satisfactory and the results therefore can be discounted. (See 4.25)

## 7.35 MINING METHOD

The block is being mined by sublevel caving using crosscuts spaced at 7,5m centres with a south-north retreat. If cavities develop in sub-level caving the ring blast throws the ore to the back of the cavity, out of reach and to avoid this an adequate cover of caved ground is required to keep ore blasted within easy reach of the loading crosscuts. It was therefore important that the block was undercut and that the hangingwall should cave before the block was put on a production footing. The undercutting of the hangingwall was done at the top of the block on the upper three levels, 245, 255 and 265 levels. On each level the undercut was

started on the south and advanced northwards. 255 and 265 levels had to link with the previously undercut portions on 245 and 255 levels.

The undercutting started at the east side on 245 level in August 1970, on 255 level in September and on 265 level in October. Undercutting on the west side along the boundary with Block 7AB lagged behind the rest of the undercutting and had to be speeded up (Fig 7,9).

### 7.36 OBSERVATIONS AND MEASURED RESULTS

#### 7.361 Observations and measurements above the undercut

Small subsidences were first recorded by the 170 level survey traverse, and by the remote displacement meters over the western end of the block in January 1971. In April, more widespread subsidence was recorded, when widening of the surface cracks in the overhanging eastern portion of Block 7AB (Fig 6,18). In May damage was noted in the main crosscut on 170 level, and in June sharp increases in the rates of subsidences over the back of the block were recorded by the remote displacement meters on 170; 205 and 235 levels. The largest subsidences were over the centre of the block and decreased outwards from the centre towards the perimeter and from 235 level upwards to 170 level. Fig 7,10 illustrates the relationship between undercutting and subsidence, of a typical remote displacement meter over the centre of the block on 205 level which shows the increasing rate of subsidence with a constant rate of undercutting. Fig 7,11 illustrates the amounts of subsidence recorded on 205 level in relation to the undercut area.

On July 12, 1971 the cave back broke through to 205 level over a distance of 20m as a large open cavity. On the eastern side of the block the caving broke through to the weak Zone B shear dipping east  $30^{\circ}$  E (Fig 7,12). Displacement had been recorded on this shear zone at the time Block 7AB caved (Fig 6, 4). Over the week-end of July 17 the area of caving extended 12m to the east and possibly through to Block 7AB on the west. The back moved up to between 170 and 205 levels. On 205 and 235 levels, opening up and lateral displacement along slips were noted in the ground in the vicinity of the cave.

When the cave broke through to 205 level on July 12 some of the remote displacement meters were lost, but those which had not been lost on the west side continued to record increasing rates of subsidence after the cave broke through, while those on the eastern side recorded

started on the south and advanced northwards. 255 and 265 levels had to link with the previously undercut portions on 245 and 255 levels.

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Fig 7,6 Block 7/1-7/3 170 level instrumentation



Fig 7,7 Block 7/1-7/3 205 level instrumentation



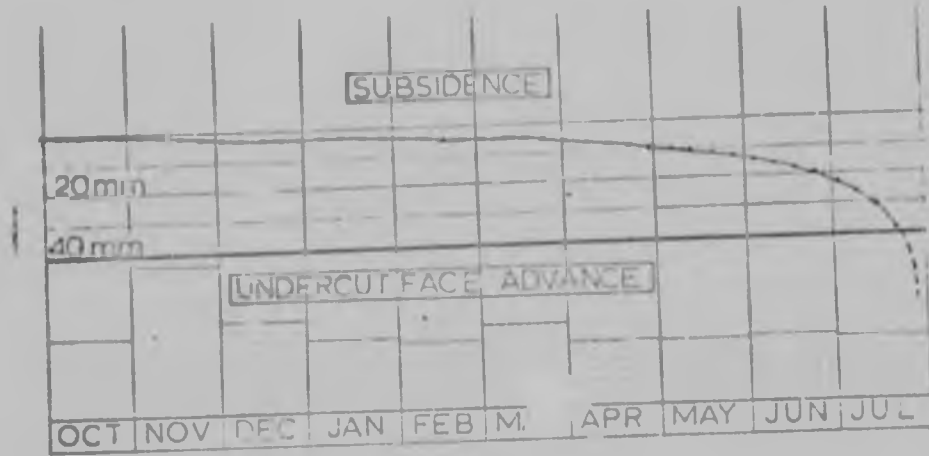


Fig 7,10 Subsidence recorded on remote displacement meter 6/8/2 on 205 level, related to rate of undercut face advance (working months)

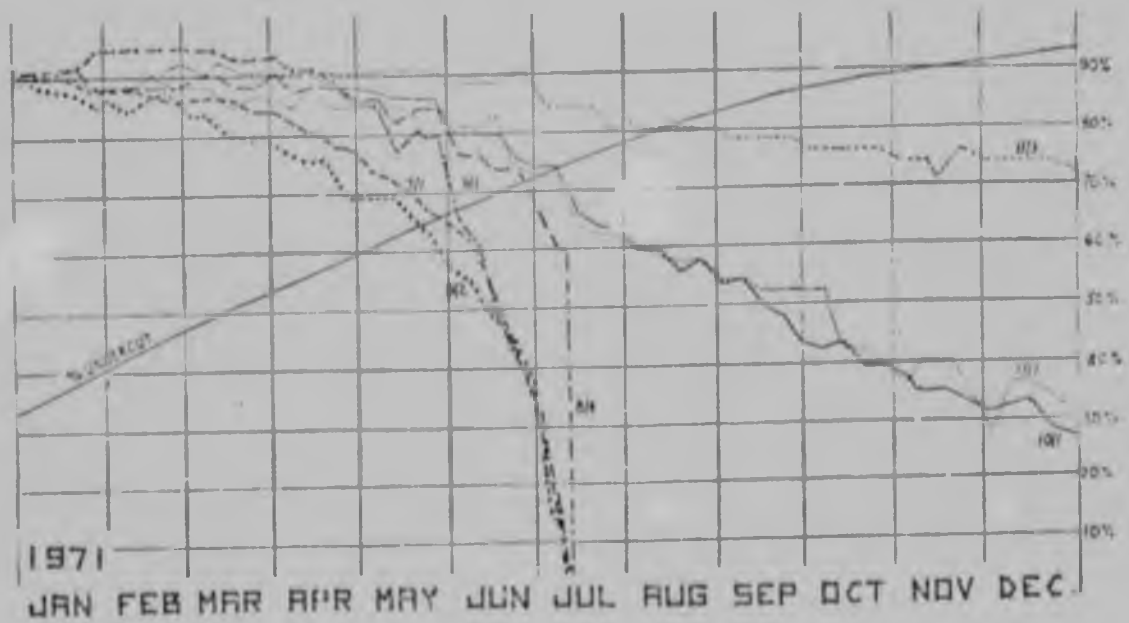


Fig 7,11 Subsidence curves for 205 level remote displacement meters ( calendar months )

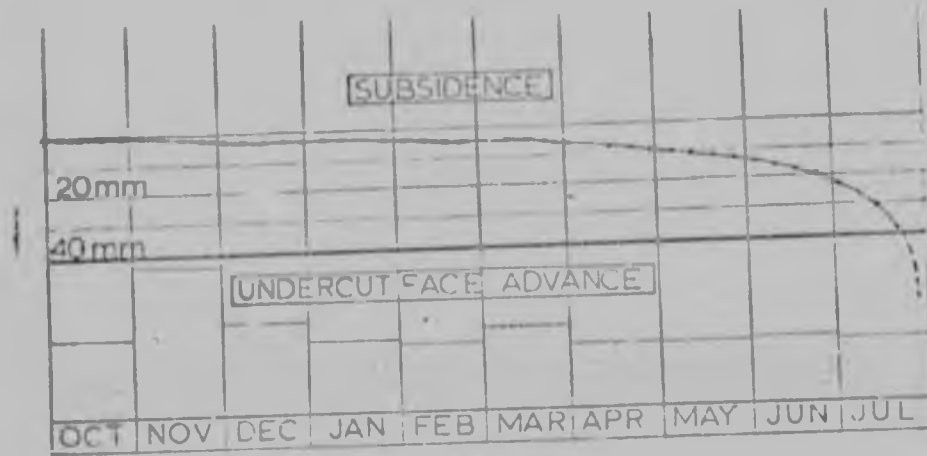


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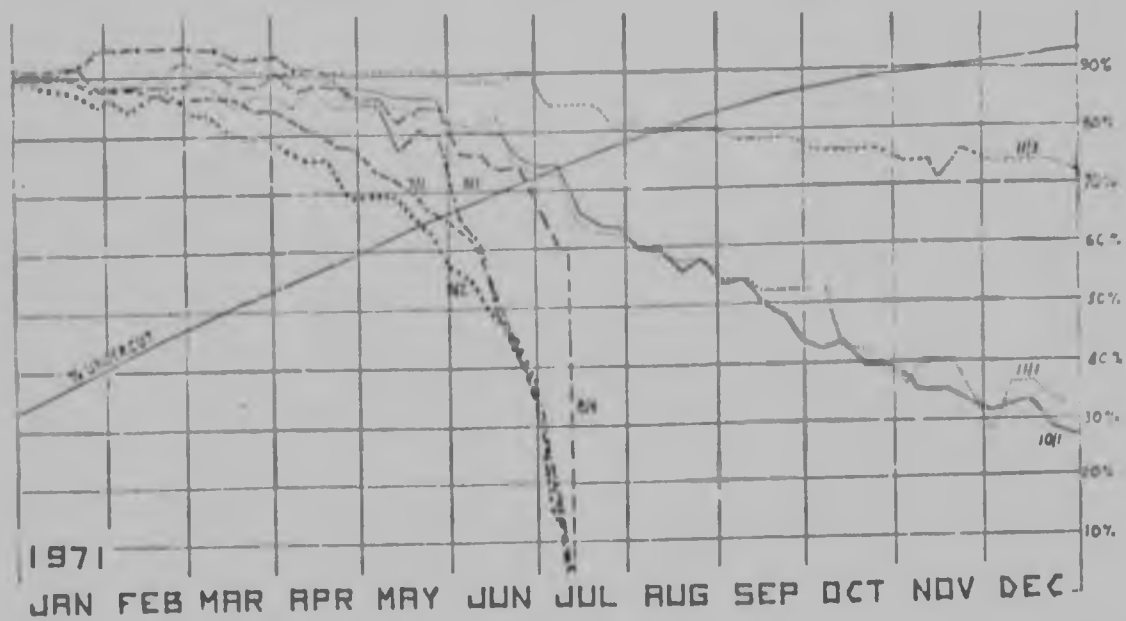


Fig 7,11 Subsidence curves for 205 level remote displacement meters (calendar months)

decreasing rates of subsidence.

When the back of the cave reached 170 level in August, it was characterised by rockfalls in the main crosscut, initially over the middle of the block, and later up to 10m north of the undercut. As far as could be ascertained, there was no large cavity beneath the back, as had been seen on 205 level, and broken material filled the cave. Horizontal tensile cracks also not observed.

In August and September 1971 several new cracks were observed on surface. Most of these trended NE-SW and appeared initially as sporadically developed cracks a few millimeters wide. The cracks initially covered only the extreme western side of the block, giving an east-west overhang of 90m but in March 1972 more cracks were observed midway along the southern side of the block which spread north in July 1972 into the "ribpillar" between Blocks 5 and 7/1. Survey beacons indicated displacement of the ground towards Block 6. These cracks joined up to form a continuous scarp. To the west of this scarp westward tilting of blocks bounded steep slips were observed. Over the centre of the block large subsidences appeared.

This scarp formed approximately halfway across the block leaving an overhang over the eastern side, which was monitored by several remote displacement meters and borehole extensometers on and below 205 level. This instrument recorded a continuous slow subsidence between July 1971 and August 1973 when increased rates of subsidence were recorded as a result of the progress of the Block 7/2 cave. The surface borehole was rendered useless when a fall broke the wire at the top of the hole and pulled it in. Thereafter it was impossible to lower a caving indicator down past the loosely spiralled old wire in the hole. The overhang is illustrated in section in Fig 7.13 and in plan in Fig 7.14.

#### 7.362 Observations from the peripheries of the block

Prior to the cave back reaching 205 level on July 12, 1971 there were no visible effects of the progress of the cave on the peripheral ground. The fall that occurred on that day must have been large enough to cause significant redistribution of stresses, for there was an immediate increase in the rate of displacement recorded on the Travelling way shear (Fig 6,26). Also small rock falls occurred as a result of small movements on Zone A on 205 level and also on 235 level, to the east of the block. Dilation and small lateral displacement or slips were noted on 205 level on the footwall side of Block 7/3 and in the shaft crosscut on

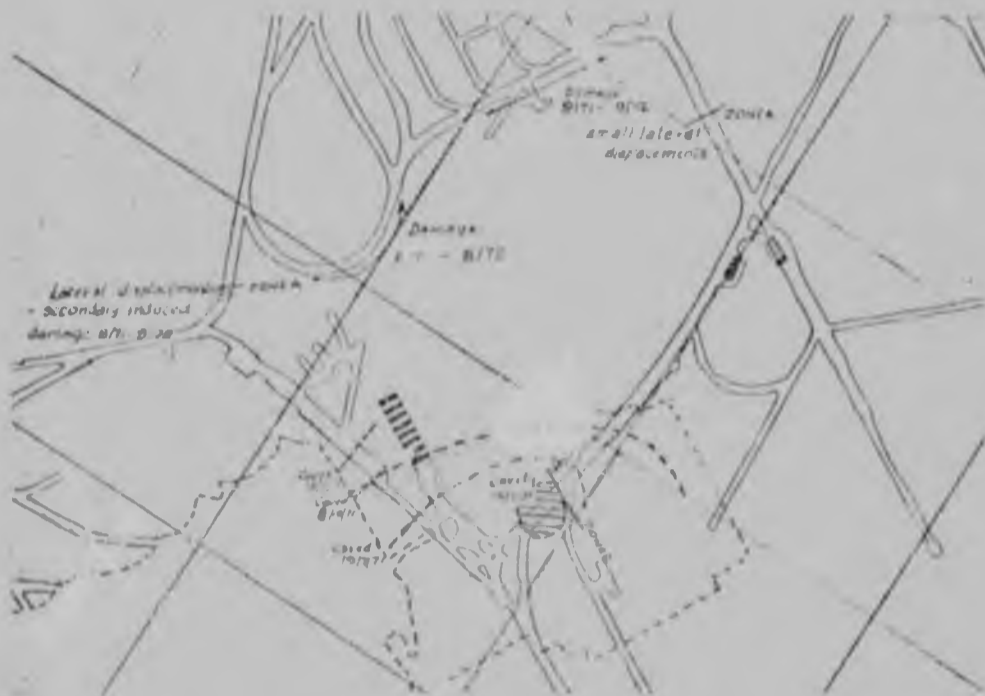


Fig 7.12 Plan of 205 level showing cavity and areas of damage

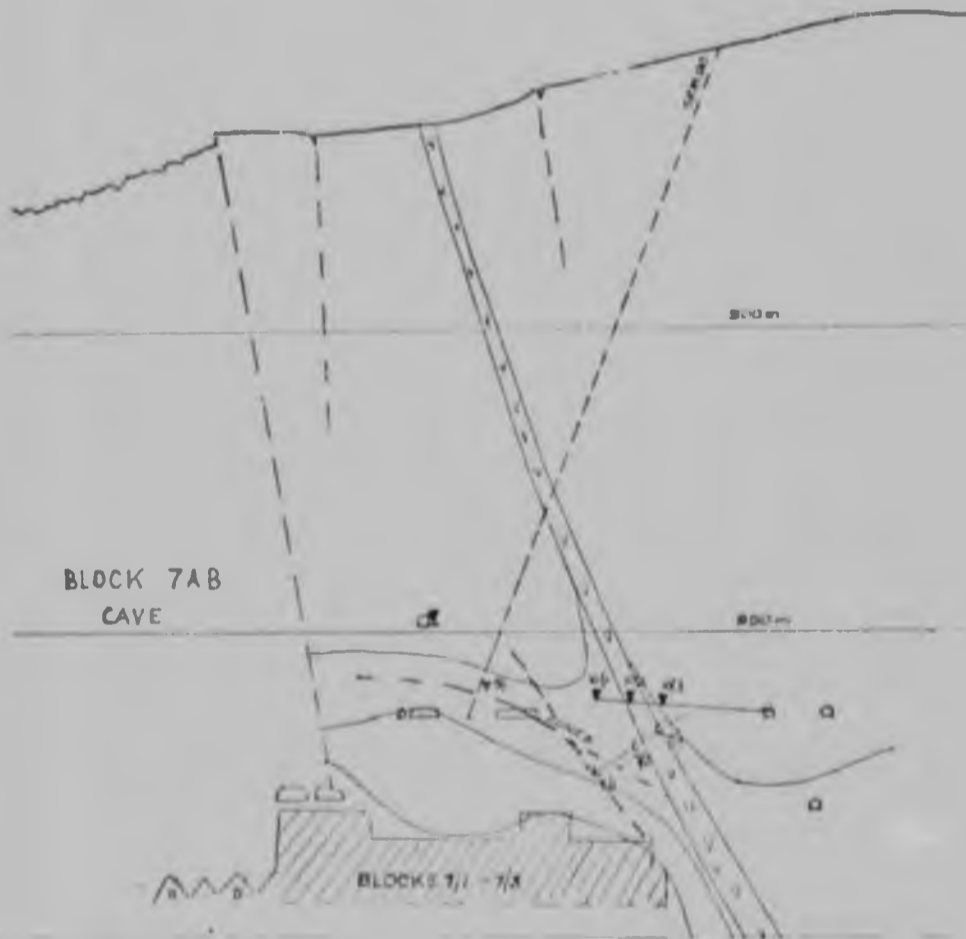


Fig 7.13 East-west section through Block 7/1 and 7/3 scarp

the hanging wall side of 235 level.

In the succeeding months further small sections of 205 level and the main crosscut on 170 level collapsed, and cracking and spalling continued in the periphery of the block for a considerable period (Fig 7,12).



Fig 7,14 Surface subsidence zone Block 7/1 - 7/3

the hanging wall side of 235 level.

In the succeeding months further small sections of 205 level and the main crosscut on 170 level collapsed, and cracking and spalling continued in the periphery of the block for a considerable period (Fig 7,12).



Fig 7,14 Surface subsidence zone Block 7/1 - 7/3

Along the northern periphery, the most significant ground displacements were associated with the Travelling way shear which moved rapidly for a six week period following the caving of the block and then slowly at a decreasing rate for a protracted period of nearly two years. The displacement on this slip resulted in east-west cracks, some damage and spalling in development that intersected it and also some damage in ahead of sublevel caving faces as they approached this shear on 255 level.

#### 7.363 Ground behaviour below the block

Because of the mining method, with its continuously advancing faces and new levels being brought into production, no instruments were installed below the block, and the visual inspections were only made when damage occurred. Only four instances of minor damage were noted.

#### 7.37 DISCUSSION

The caving on 205 level had, in common with the caving in Block 6 and portions of Block 7AB, a cavity which progressed upwards by sporadic falls from the back. Unlike Block 7AB, there was no significant evidence of damage due to a high lateral stress, in the back, the only damage being a rock fall on the 205 level 3 shaft station from which no conclusions were drawn. However, the stress measurements done in 1968 closer to the Block 7AB cave on 170 level indicated an enhanced lateral stress of approximately 12 MPa in a direction parallel to the periphery of Block 7AB, and the stresses normal to the periphery was reduced to about 5 MPa. It may be concluded that in this area the stresses in the back not sufficiently reduced to allow mass subsidence caving nor so high as to cause rock failure and a rapid caving action.

The surface subsidence zone was not spectacular, but consisted of a zone of cracking and orderly subsidence as had been observed over Block 7AB. There was a general tilting of the columnar blocks of rock bounded by slips towards Blocks 6 and 7AB indicating a reduction in stress in that direction.

Two forms of caving are indicated by their distinct characteristics which are similar to the two forms of caving observed in Block 7 AB, known as stress caving and mass subsidence caving.

In the peripheries, the ground behaviour was essentially similar to that observed in Block 7AB, but because the Block 7/1 - 7/3 caved area

was smaller than that of Block 7AB, the effects on the peripheries was also smaller.

#### 7.4 BLOCK 16

##### 7.41 INTRODUCTORY REMARKS

Block 16 was laid out as a sublevel caving operation and was expected to cave when the undercut area had reached a certain size, but when the undercut reached this size no caving occurred and several measures were taken in attempts to induce caving. From the failure of this block to cave as expected and the failure of most of the cave induction attempts, much has been learnt about caving. Discussion of this block is included here because it provides further information on the conditions required for stress caving, the stress caving mechanisms, the limitations to the application of an undercut and criterion for caving, and among other things on the effectiveness of the cave induction measures taken.

Block 16 is located to the east and above the 170 dyke which overlies the east end of Block 7. This is a large block which is subdivided from west to east into Block 16A, Block 16B, Block 16C etc. and has been mined since 1968 by sublevel caving.

##### 7.42 GEOLOGY

Being a sublevel caving block it is not the structural geology of the orebody but of the hangingwall that controls the caving behaviour of the ground overlying the undercut. The hangingwall contact of the block is an economic cut off, with a decreasing number of fibre seams upwards away from the orebody. The rock classification of the hangingwall rocks is given in Fig 7,22 and Fig 7,23. The north side of the orebody is bounded by the Zone B talc zone which strikes E-W and dips steeply northwards at about  $65^{\circ}$ . Zone C runs parallel to Zone B and occurs to the south of the orebody, but is not as close to the orebodies as Zone B (Fig 7,15). To the west of the block is the  $65^{\circ}$  east dipping 170 dyke which separates Block 16A from Block 7 as it passes below Block 16A and above Block 7 (Fig 7,16). Some talc developed along the contacts of the dyke, and two major gouge-filled shears are developed along the contacts. The hangingwall partially serpentinised dunite is cut by a steep dipping slip.



Fig 7.15 Section through Block 16

#### 7.43 CONDITIONS PRIOR TO MINING

This block was in an almost virgin state prior to mining, being situated well away from any previous cave mined blocks, and unlike most other blocks, it did not have any cut and fill stopes. No displacement was known to have occurred on any slips in the back prior to mining, but it is probable that the caving of Block 7AB and later in 1971, Block 7/1 increased the lateral stress in the ground above the block. Stress measurements were done in this block on 295 level during the stress measurements programme prior to mining, and the results are given in Table 5.1

and discussed in Chapter 5.

#### 7.44 MINING METHOD

The block was mined by sublevel caving from drives with a west to east direction to retreat. The top level of the block was 275 level, and on this level just over half the tonnage broken was called for but somewhat less was actually drawn. It was anticipated that when 275 level was complete the block would cave. It did not. It was thought necessary to continue mining, drawing only a limited tonnage leaving a substantial cover of broken ground as a precaution against air blasts, and to increase the undercut area as rapidly as practicable in the hope that the block would cave. The next level was 285 level which slightly widened the 275 level north-south span but not the east-west span as crashing on this level also stopped at the eastern retreat crosscuts, on the boundary of 16A and 16B. Crashing on 295 level extended the undercut by 20m on the south side and continued through from Block 16A into 16B until all the ore on this level was crashed increasing the span and undercut area considerably. The crashing of 305 level increased the north-south span marginally.

#### 7.45 INSTRUMENTATION

The instruments initially planned for this block covered only the north west corner which was expected to cave. Further instruments were installed to monitor the extensions to the undercut and at various times as the undercut was extended in an attempt to induce caving. The initial programme to monitor the back included from 205 level: four boreholes fitted with remote displacement meters, three boreholes fitted with wire extensometers, and one borehole fitted with a caving indicator, and from surface one borehole which was unsatisfactorily equipped with borehole wire extensometers. Wire extensometers and remote displacement meters on 205 level, to monitor deformation of the peripheries.

From 235 level two boreholes were fitted with wire extensometers, to monitor the western periphery, and wire extensometers and remote displacement meters were installed in the available tunnels. On 295 level, one borehole and a crosscut were fitted with wire extensometers.

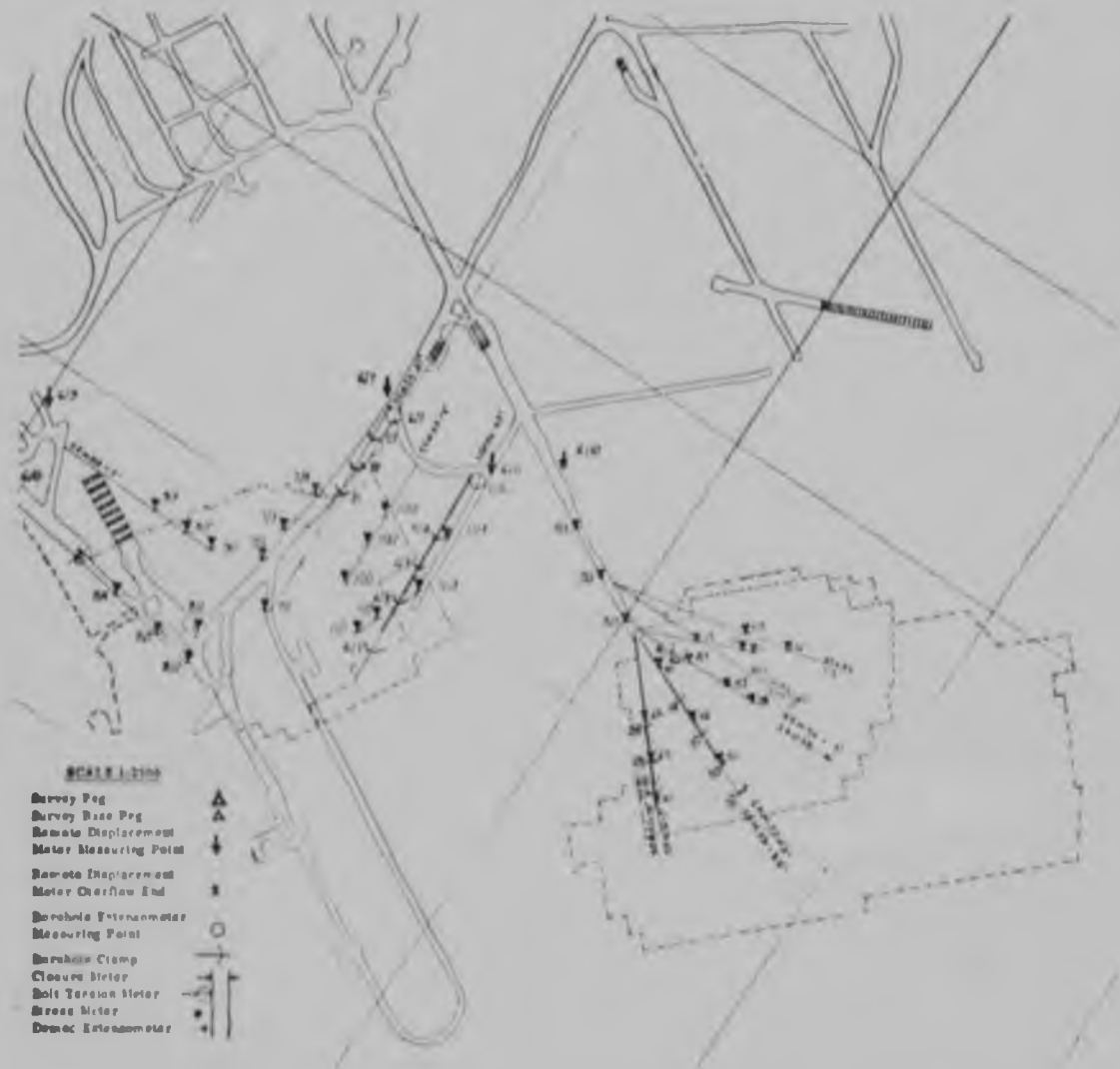


Fig 7,16 Instrumentation 205 level

Also on this level a survey levelling traverse was brought in from the footwall and down the eastern perimeter of the block ( Fig 7,18).

The lower half of one of the 205 level boreholes was also lost. Only half the length of the boreholes drilled for remote displacement meters could be used for the planned remote displacement meters because of the deflection of the boreholes. Otherwise all other instruments were installed

as planned. The location of these and subsequent instruments is shown in Figs 7,16; 7,17; and 7,18.

The subsequent instrumentation included three extra remote displacement meters over the back of the block on 235 level. An extra borehole on 205 level fitted with remote displacement meters and an extra borehole from 205 level drilled to intersect the back and later fitted with borehole clamp wire extensometers. To monitor the effects of extending the undercut into 16B three holes were drilled from 235 level and fitted with remote displacement meters and wire extensometers.



Fig 7,17 Instrumentation 235 level



Fig 7.18 Instrumentation 295 level

Later still another hole was drilled from 235 level over the south western corner of the block and fitted with borehole wire extensometers (Fig 7.17). During late 1973, a borehole was drilled from surface and monitored with a caving indicator.

#### 7.46 GROUND BEHAVIOUR

##### 7.461 Observations above the block

The behaviour of the rock above the block is the most important aspect of this block, because the ground did not cave as expected from

the previous example of Block 6, and undercutting started in this block in January 1969 when approximately half 275 level had been crashed, one clamp indicated an 8m fall from the back as the face passed beneath it. Other very small movements were recorded at that time. No further changes were recorded until January 1970 when 275 level was almost complete and some very small subsidences were again recorded over the back. In March 1970 photographs of the cavity left between the broken ground and solid back were taken from two vertical holes drilled from 235 level (Fig 7,17). The photographs were taken with the camera developed by the CSIR to photograph potential sink holes intersected by drilling in the Carletonville area. Two complete sets were taken from each hole, giving 360° stereopair coverage from which directions and distances could be worked out. The limit of adequate illumination in the photographs was about 5,4m.

The photographs from DD 914 showed that rock falls from the back had occurred on the northern and western and southern sides. A cavity up to 3m high was left between the fallen rock and the back. On the eastern side the blasted rock banked up to the hanging about 3m from the borehole. From DD 916 a cavity was found to the north of the hole, and to the south blasted material was banked up to the hanging about 2m from the camera. Another borehole intersected the back in April, and this too showed broken material up against the back. In June and July after 285 level crashing had progressed three quarters of the way across the block, removing the support the broken material was giving to the back, several falls of up to 6m in height were recorded by caving indicators in these holes. At the same time one borehole wire extensometer clamp moved down 317mm. No subsidences were recorded by any of the other instruments at the time of these falls.

Because a cover of caved ground is needed in a sublevel caving operation, and in this case some protection from airblasts was required, it was considered necessary to try and induce caving. The first attempt at cave inducing was made by drilling two, then four holes from 235 level over the back and pumping water under pressure into these holes. The pressures were initially low but were gradually increased by the periodic addition of fine sludge to water. The pumping started in December 1970 for a few days and on the 6th and 16th of that month, two falls of 0,5 and 6m from the back were recorded. Two more holes were drilled

and pumping in all four was resumed later in December with pumping for 6 hours per day. Pumping was resumed in April, for four days, and in May for three weeks. In June the pressure was increased to approximately 5 MPa and pumping continued every day until July 25. During this period, from December 18 to July 31, no further falls were recorded and no changes were recorded on any of the instruments.

The hydraulic cave induction experiment having failed, an attempt was made to cut a slot on the northern boundary of the block by blasting 26 holes drilled from SA/9E crosscut on 235 level (Fig 7,19). On August 1,

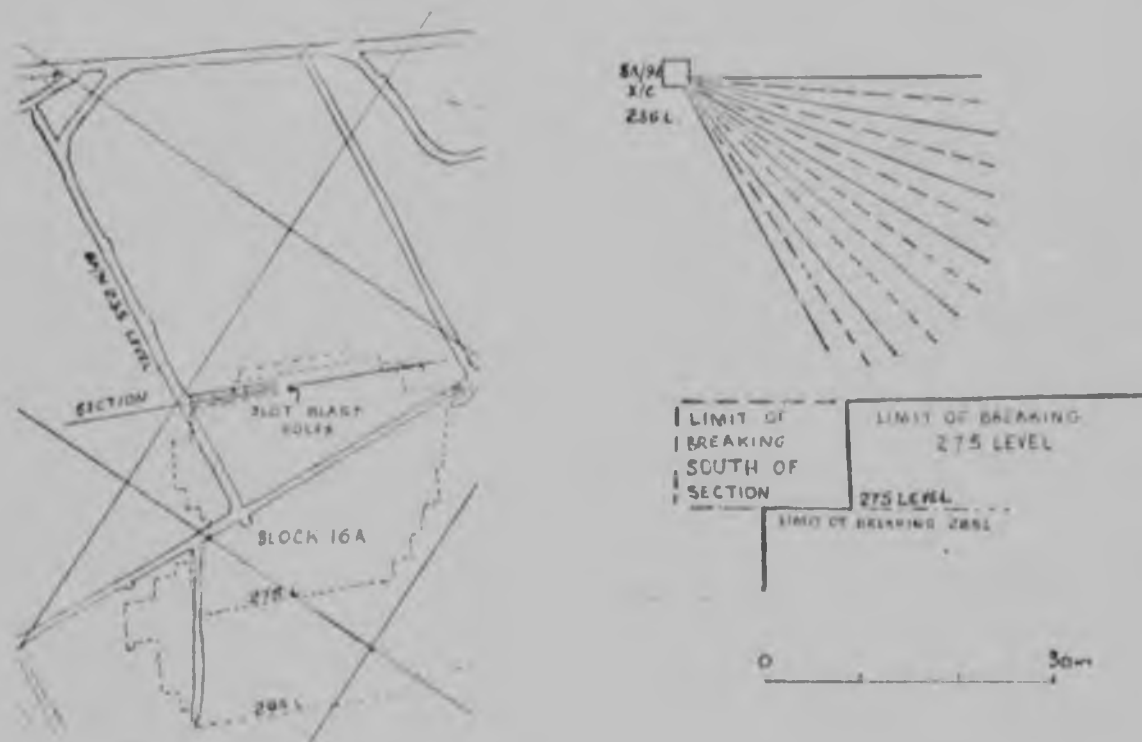


Fig 7,19 Block 16 cave induction blast hole pattern

1971, 24 of the 26 holes, charged with 32 cases of gelignite, were detonated simultaneously. It is doubtful whether the slot was successfully cut, as the instruments did not show any significant changes attributable to this blast. In retrospect the charge was also considered

to be too small to cut the slot. It was therefore decided to increase the undercut area by extending the 295 level crashing into Block 16B. Between August and October four falls of up to 6m were recorded for the back and some small subsidences were recorded on the other instruments. At the end of October, the back position given by three boreholes over a strike distance of 30m was approximately 30m above 275 level. Whether these falls were due to the attempt at slot cutting, or, due to the effects of Block 7/1 caving, or, due to the effect of extending the undercut eastwards into Block 16B, is not clear. It is probable that the slot blast had the least effect because the size of this blast was relatively small and apparently failed to create the desired slot.

Small subsidences and a fall from the back were again recorded in February and on March 17, 1972 a substantial fall occurred, bringing the back to just above the 235 level horizon. On the northern side cavities of 6m and 10,5m were measured and on the western side about 17,5m. The 205 level instruments continued to record subsidence at a slow, but unchanged rate, which did not reflect this fall. In July, subsidences large enough to indicate that caving had reached 235 level were recorded over the central position of 16B on the new eastern remote displacement meters shortly after the 295 level crashing had passed beneath these instruments (Fig 7,20). The gradual increase in the rate of subsidence experienced prior to caving in Blocks 7AB and 7/1 did not occur in this area. The steady rate of subsidence of 0,1mm/day which was recorded over the preceding three months, suddenly accelerated. These instruments were lost in this action. The remaining three remote displacement meters continued to record subsidence until another was lost in November 1972, and from then until September 1973 a relatively slow rate of subsidence was recorded.

Over Block 16A there was a slowing of the rate of subsidence recorded in July and with the exception of two borehole wire extensometers which recorded continuous slow subsidence, no changes were recorded between July 1972 and September 1973. In September 1973 general increased rates of subsidence were recorded, these were due to the progress of the Block 7/2 cave (See Fig 7,33).

Pushing a caving indicator up one of the 205 level holes drilled for remote displacement meters in October 1973 indicated that a large fall

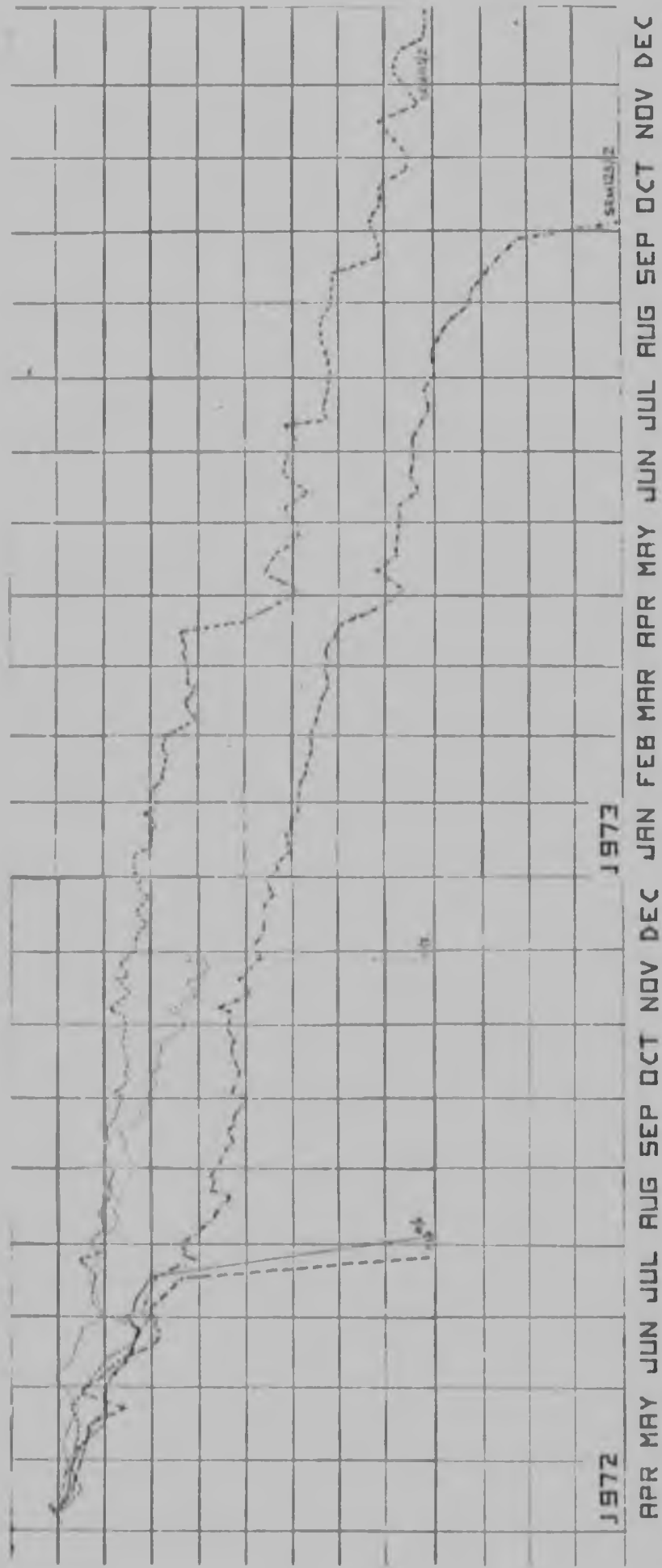


Fig. 7,20 Subsidence recorded by remote displacement meters over Block 16B

had taken place over the centre of the block, leaving a cavity on the 205 level elevation. There is no indication as to when this fall occurred, but it is suspected that it occurred in October 1973 when some sudden damage was noted on 305 level. This cavity progressed by sloughing from the back until it broke through to surface in June 1975.

In Fig 7,21 an attempt is made to relate the various parameters of this block to events recorded in the back. Figs 7,22 and 7,23 are sections through the back illustrating the stage of mining and events in the back.

#### 7.462 Observations in the peripheries

In general, very little has been noted in the peripheries of the block. Apart from some slight damage reported ahead of the face on 275 level as that level was nearing completion, and some spalling at the pillars left between the retreat crosscut and face on the extreme east end of 295 level, no damage has been noted in the abutments. The lack of evidence of abutment loading on 275 in Block 16A was investigated in April, 1970 by doing a series of stress measurements in the abutment on 275 level in the south eastern corner of the block within 10m of the face. These showed that in the abutment the stresses were approximately the overburden load and it was concluded that the abutment loads had been widely redistributed by small displacements on slips. There is some evidence for this redistribution in the movements which were recorded on the 295 level borehole extensometer located across the Zone B steep major structural feature on the north side of the block. Displacement on this zone has also been recorded in the northern abutment on Block 16B (Fig 7,24).

#### 7.463 Observations from below the block

No instruments were installed below the block, and no regular inspections were made of the development below the block. The mining personnel, have however, kept a watch on the area and reported any significant damage, which included a light spalling below Block 16B undercut area, and the sudden collapse of a structurally weak area on 305 level which could have been due to a substantial fall from the back, or to the load extended by an unbroken portion left on 295 level above.

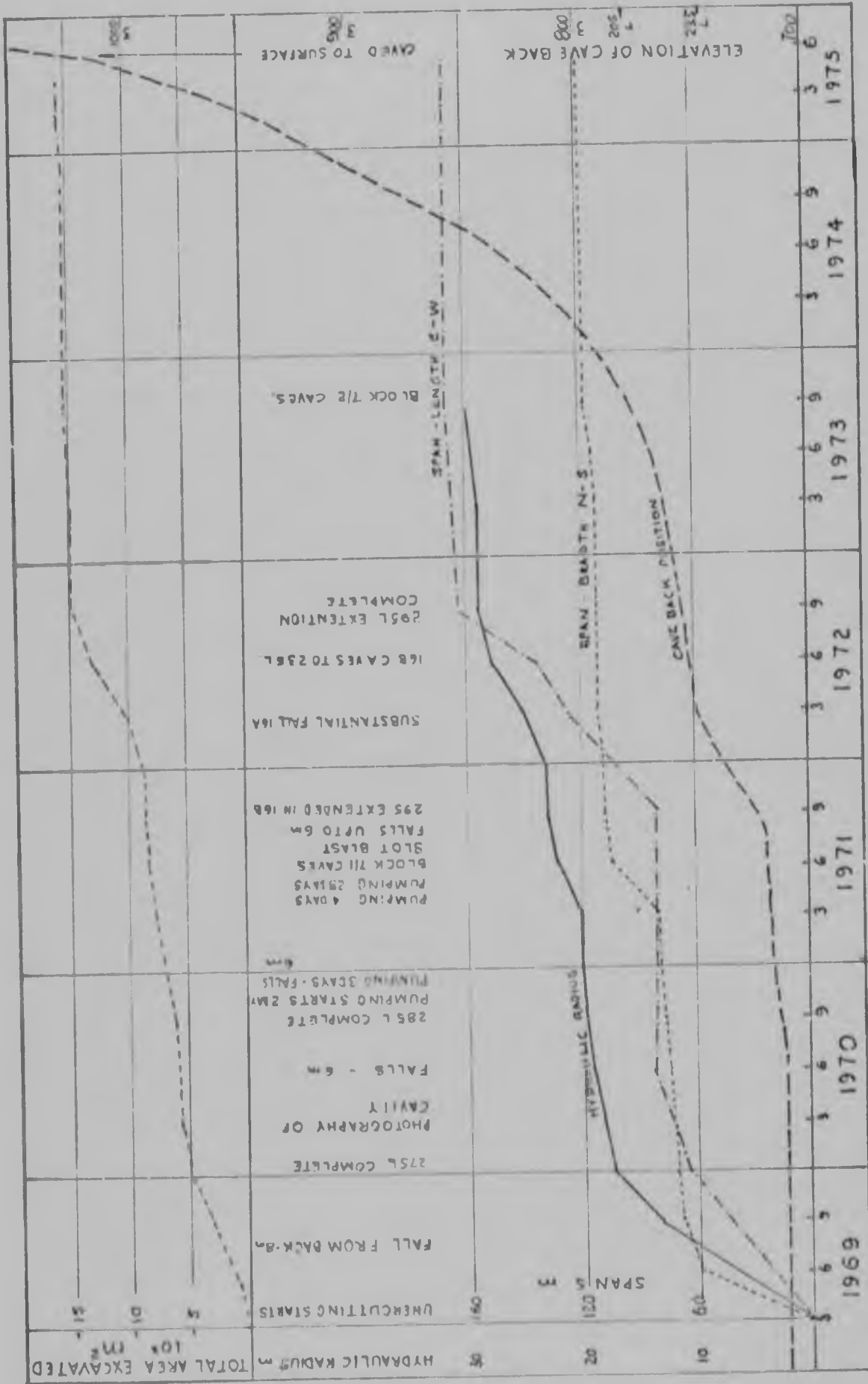


Fig 7,21 relation between caving progress and block parameters

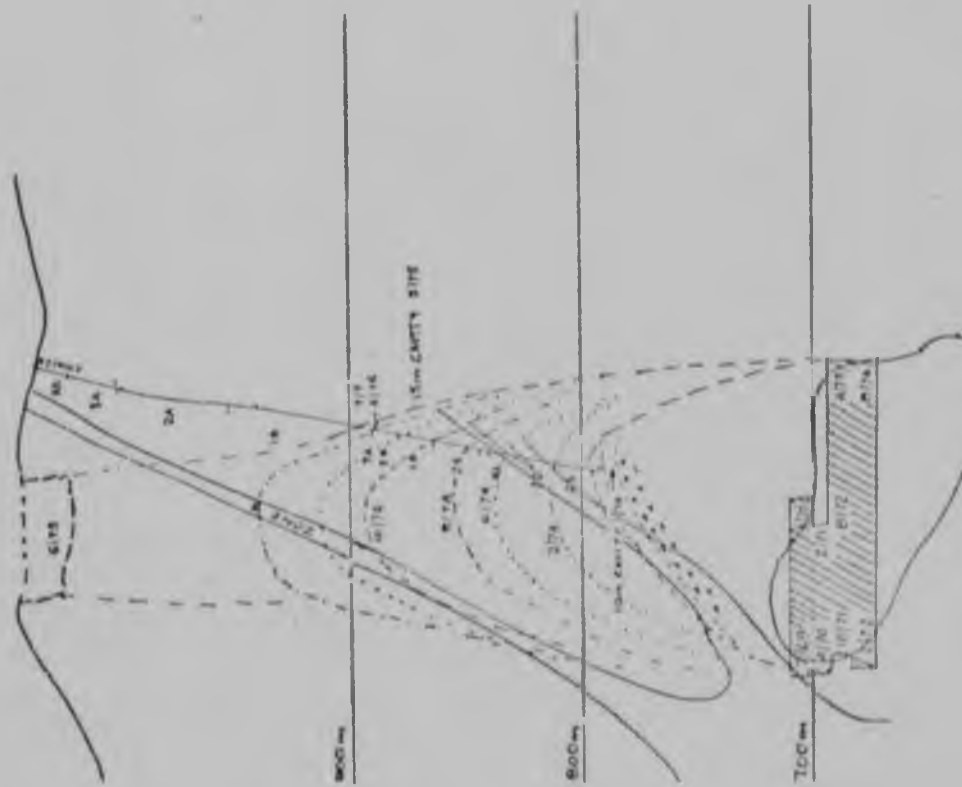


Fig 7,23 Section through surface borehole



Fig 7,22 Section through the 205 level caving indicator boreholes

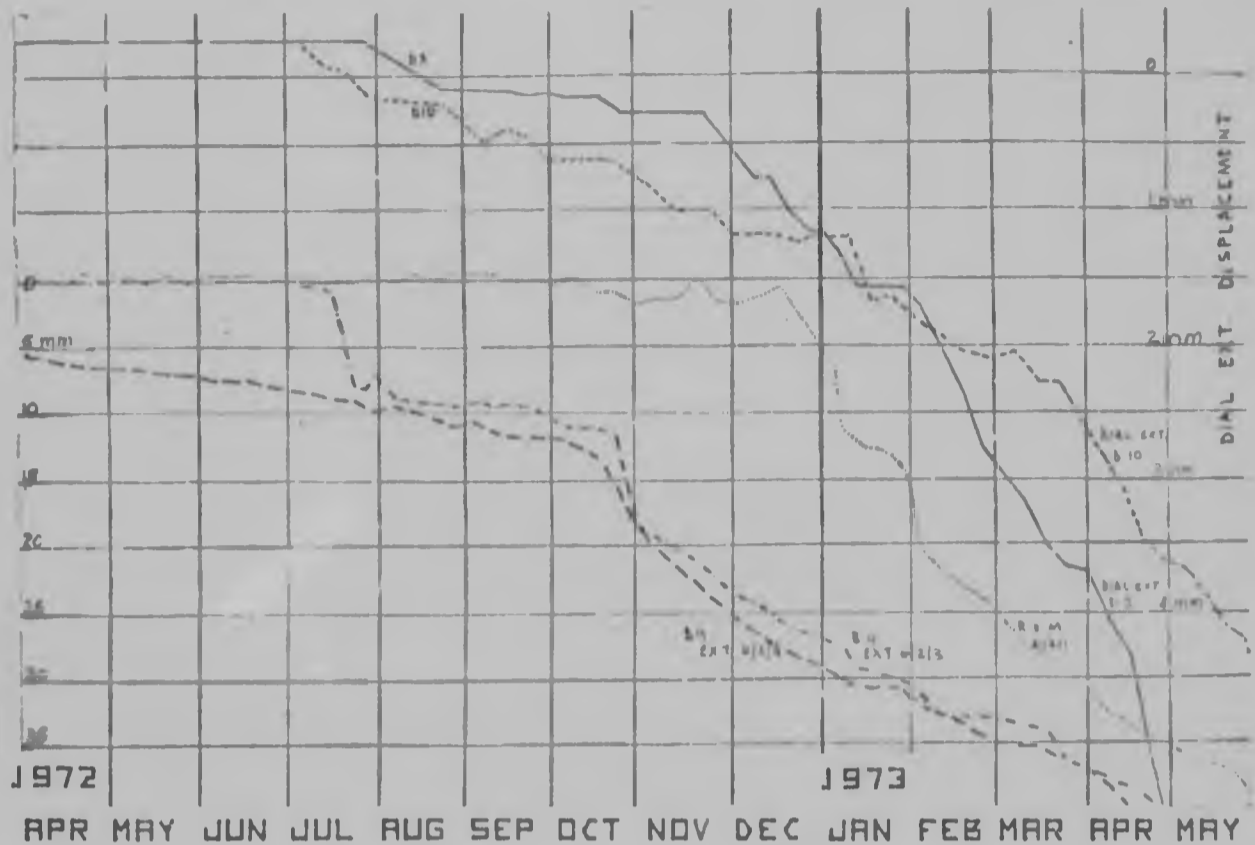


Fig 7.24 Subsidence curve and related displacement recorded on Zone B

#### 7.47 DISCUSSION

The progress of caving in this block was similar to that experienced in Block 6, but in size, this block was considerably larger. The 19.5m hydraulic radius of Block 6 which was used as the basis for deciding how big the undercut area should be to initiate caving, proved to be inadequate. Attempts to induce caving by increasing the water pressure in the back and by attempting to blast a slot on the northern boundary, failed, despite the fact that caving of Block 6 was attributed to increase in ground water in the rainy season and to blasting activity underground. This block differed from Block 6 in its isolation from previously mined areas, its greater depth, and possibly more competent hangingwall ground conditions.

There has always been a fear of an airblast in this block if a sufficiently large slab were to fall from the back. The known two larger falls, and several smaller falls did not, however, produce any airblasts.

## 7.5 BLOCK 7/2

### 7.51 INTRODUCTION

Block 7/2 is included as an example of a block caving operation, which unlike Block 7AB is not complicated by the presence of old cut and fill stopes or an overlying caved block. As will be shown, the course of caving was similar to that in Block 7/1 despite the initial difference in shape of the undercut and geological factors. An interesting aspect of the caving of this block was the effect that this had on the adjacent uncaved Block 16 and on the overhanging portion of Block 7/1.

Block 7/2 lies immediately south of Block 7/1 and to the west of Block 16 which it partly underlies. Undercutting started in April 1972 and was completed in December 1973.

### 7.52 GEOLOGY

The block is on the south east end of the Block 7 orebody, with an arbitrary line dividing this block from Block 7/1. There is, however, a vertical zone of shearing with associated aplites and talcification metasomatism which occurs in Block 7/1 and near the boundary between these blocks (Fig 7,25). As with Block 7/1 the eastern side of this block is overlain by the south-eastward dipping 170 dyke and its associated talc zones at the contacts. This dyke is largely talcified, and weak talc-gouge filled shears are developed on the upper and lower contacts. The western margins of Block 16A overly the dyke and the eastern limit of Block 7/2 (Fig 7,26). To the south of the block is the 65° dipping Zone C talc zone.

The incompetency of the orebody on the grizzly is lower on the northern side when 35-45° south dipping shears with thin gouge fillings are common and there is a higher slip density which reduces the rock class to 4A in places. The southern half of the block is more competent with a fewer, smaller fibre seams and more widely spaced slips, having a 2B and locally a 2A rating. The immediate back of the undercut has a 3A rating, overlain by Class 2 ore (Fig 7,25).

### 7.53 PREVIOUS MINING AND GROUND CONDITIONS PRIOR TO MINING

To the north-west of the block was the previously caved Block 7AB and on the northern side is Block 7/1 and Block 7/3, which had caved

through to surface prior to the commencement of undercutting in this block. The bottom of Block 7AB was on the 600 level elevation, some 65m above the top undercut. As the undercutting progressed northwards, so the mining in Block 7/1-7/3 was deepened. The caved areas of blocks 7AB, 7/1 and 7/3 had resulted in a reduction of lateral stress in the north-south direction and small dilation and lateral movements had been noted on some of the weaker slips, notably on the talc-gouge filled shear on the underside of the 170 dyke. Displacements were noted on 235 level on slips in the periphery of Block 7/1, but there was no evidence of any displacement on slips and only locally light spalling indicated the increase vertical stresses in the peripheries of Block 16 on this level.

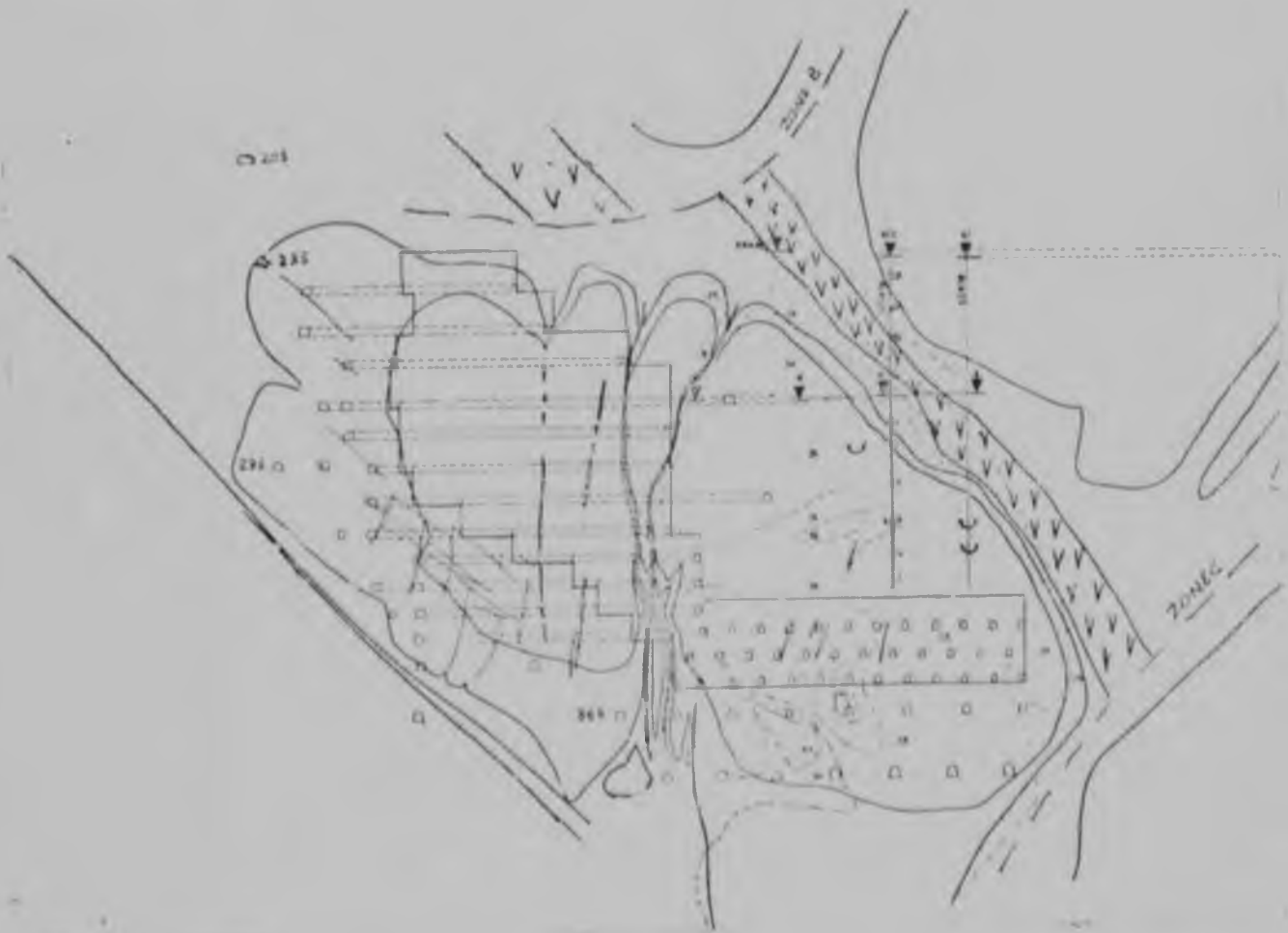


Fig 7.25 N/S section through Blocks 7/1 and 7/2

It is considered that the lateral stresses in the back in the north-south direction had been reduced by the caved ground in Blocks 7/1, 7/3 and 7AB, whilst the east-west stresses may have been increased. In



Fig 7,26 E-W section through Block 7/2 and 16

the peripheries of Block 16A higher vertical stresses were indicated, while it is also probable that the lateral stresses in the zone between the lowest levels of Block 16A and the top of Block 7/2 undercut, were also increased.

#### 7.54 MINING METHOD

As indicated above, block caving was the mining method selected for this block. The grizzly horizon on 365 level was overlain by one overcut horizon located 9m above it and two undercut horizons, at 16,5 and 24m above it (See Fig 7,25). Crashing on the top undercut (345 level) started in the south east corner of the block in April 1972 and progressed northwards towards 7/3, as illustrated in Fig 7,27. The crashing of the second undercut followed closely behind. The overcutting of drawpoints was left until the drawpoints were required for production. This was started in April 1973.

The undercutting progressed at a steady rate from the start of the undercutting until March 1973, when it was decided to stop undercutting for four months until access to Block 7/4 was established, when undercutting could be resumed. Undercutting was resumed in August 1973, and completed in December 1973.

#### 7.55 INSTRUMENTATION

The instrumentation of this block consisted of borehole wire extensometers located in three boreholes drilled from 205 level to monitor the eastern side of the block. The clamps in one of the boreholes were located over the back while the clamps in the other two were in the periphery. While these installations gave some useful information, their performance was not satisfactory. Remote displacement meters were also located in boreholes and development on 235 and 275 levels. Borehole clamps were located in the back and western periphery in boreholes drilled from 235 level, and a caving indicator in a 235 level borehole drilled into the south western corner of the block where a cavity was expected to develop.

The locations of these instruments are shown in Figs 7,25; 7,26; 7,27 and 7,28.

#### 7.56 GROUND BEHAVIOUR

##### 7.561 Ground behaviour above the undercut

The undercutting operation, which progressed from the south east towards the north west resulted in a limited amount of caving forming a dome shaped back over the undercut area. This was indicated by the wire extensometers in the two boreholes drilled from 235 level which were not

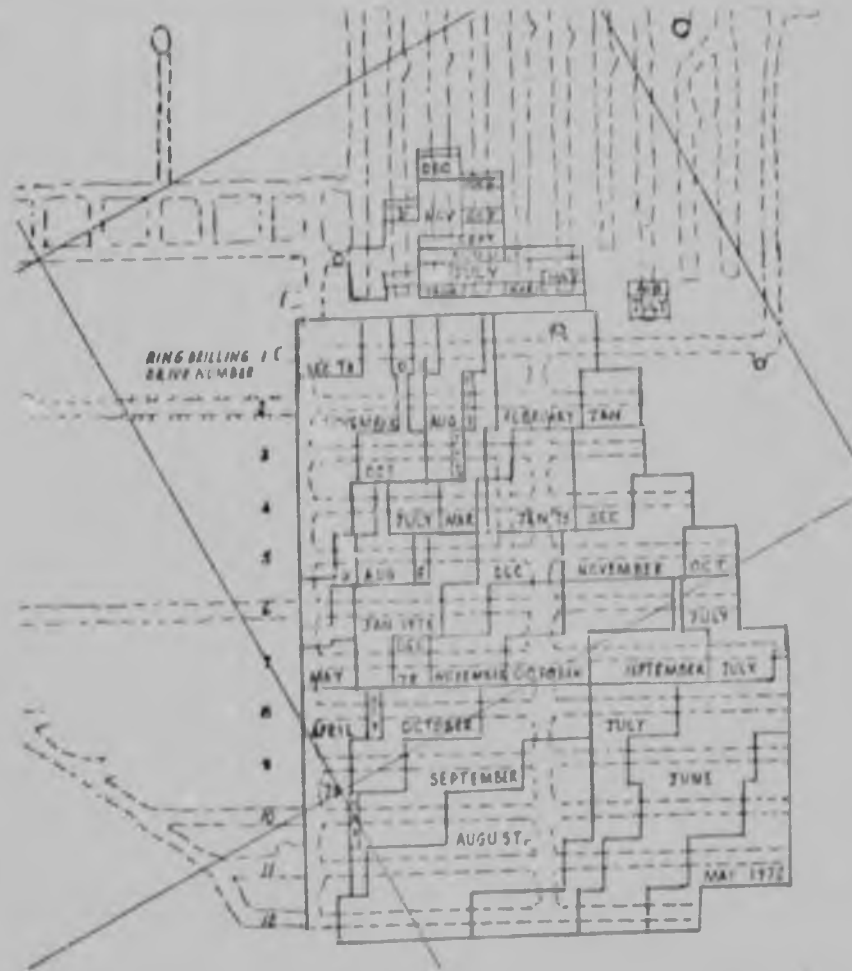


Fig 7,27 Undercut face position Block 7/2



Fig 7,28 235 level instrumentation Block 7/2

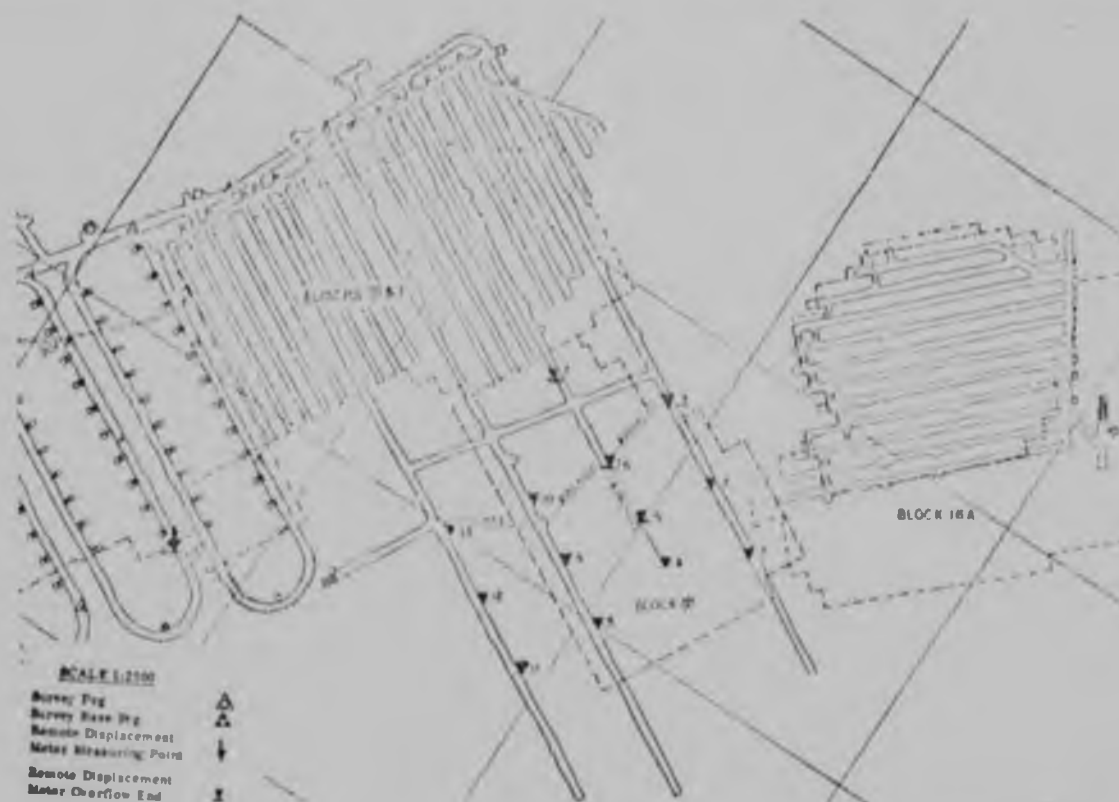


Fig 7,29 275 level instrumentation

satisfactory, and were replaced. These replacement extensometers recorded slow subsidences in January 1973, at distances of 30 to 40m above the back when the undercut face had passed beneath them, a distance of 25m, and the E - W span was 66m. These movements were slow and small and indicated the formation of tensile cracking in the back similar to that observed on 2C5 level in Block 7AB. On the west side however, the formation of cavities was indicated by the wire extensometer in SRM 100 which recorded more rapid subsidences at elevation of 15m and 25m above the undercut. The lowest clamp indicated that a cavity of at least 0,25m had developed slowly enough to be monitored before the clamp wire was broken in January. The next clamp indicated the formation of a similar cavity 10,5m high where intersected by the boreholes. It is remarkable that up to 10,5m of wire was exposed for 18 months without being cut by a rock fall of any form.

SRM 101 drilled from 235 level into the southern half of the block which was rigged with a caving indicator, showed a sporadic westward extension of the cave limits, although a cavity large enough to accept the caving indicator did not develop.

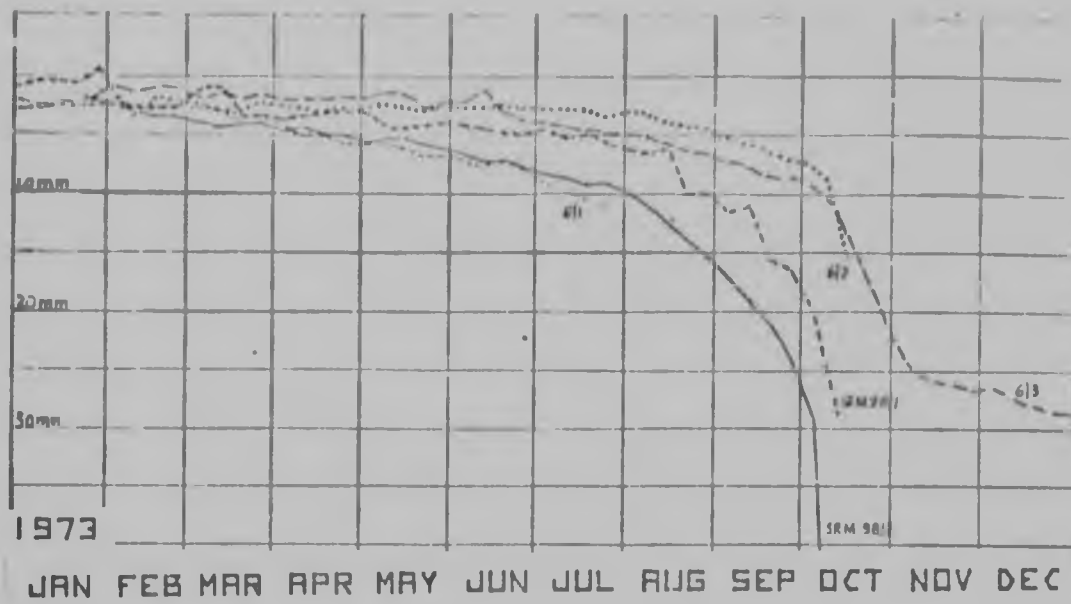


Fig 7,30 Subsidence graphs 235 level Block 7/2

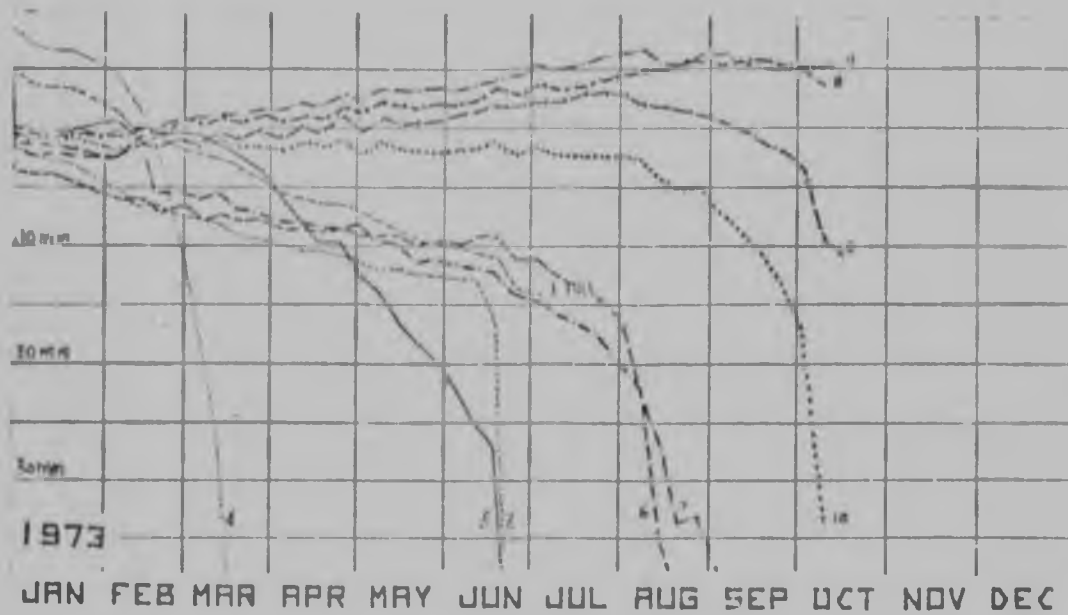


Fig 7,31 Subsidence graphs 275 level Block 7/2

On 275 and 235 levels, general slow subsidences were recorded over the undercut area during 1971. Rapid subsidences on 275 level recorded over the central portion of the block during March, April, May and June 1973, indicating that the back had reached this level (Fig 7,31 and 7,32). Again no sudden falls were recorded, at least not in the initial stages, as the subsidence graphs indicate accelerating subsidence before the instruments were lost. In August and September when the undercutting of the remaining portion of the pillar between Block 7/2 and 7/1 was resumed, increasing rates of subsidence were recorded on the north

side of the block on both 235 and 275 levels. In October, general subsidences were recorded indicating that a major change had taken place. For the first time, the effects of caving were visible on 235 level, where, in the shaft crosscut, dilation of N - S striking slips indicated that in the northern portion of the block next to Block 7/1, the caving was adopting the form that was observed in Block 7AB to the south of block 6.

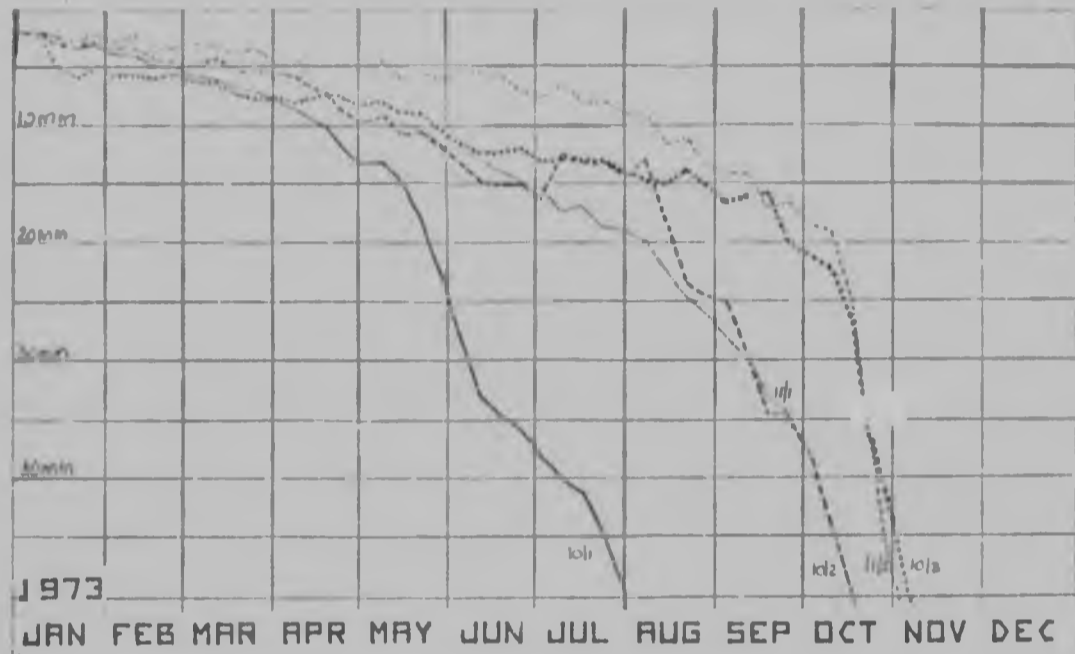


Fig 7.32 Effects of block 7/2 caving on Block 7/1 overhang

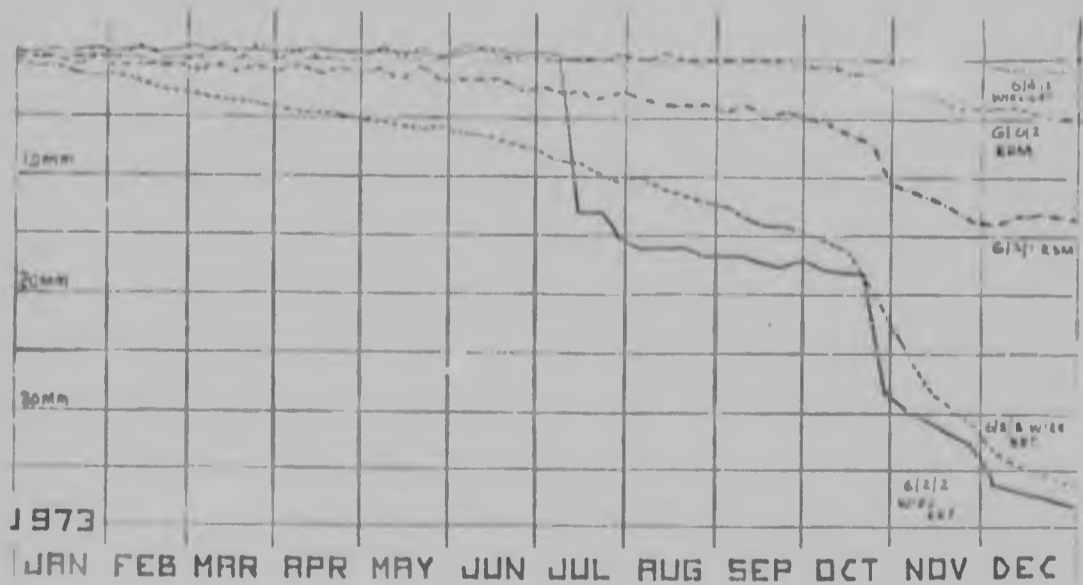


Fig 7.33 Effects of Block 7/2 caving on Block 16 instruments

side of the block on both 235 and 275 levels. In October, general subsidences were recorded indicating that a major change had taken place. For the first time, the effects of caving were visible on 235 level, where, in the shaft crosscut, dilation of N - S striking slips indicated that in the northern portion of the block next to Block 7/1, the caving was adopting the form that was observed in Block 7AB to the south of block 6.

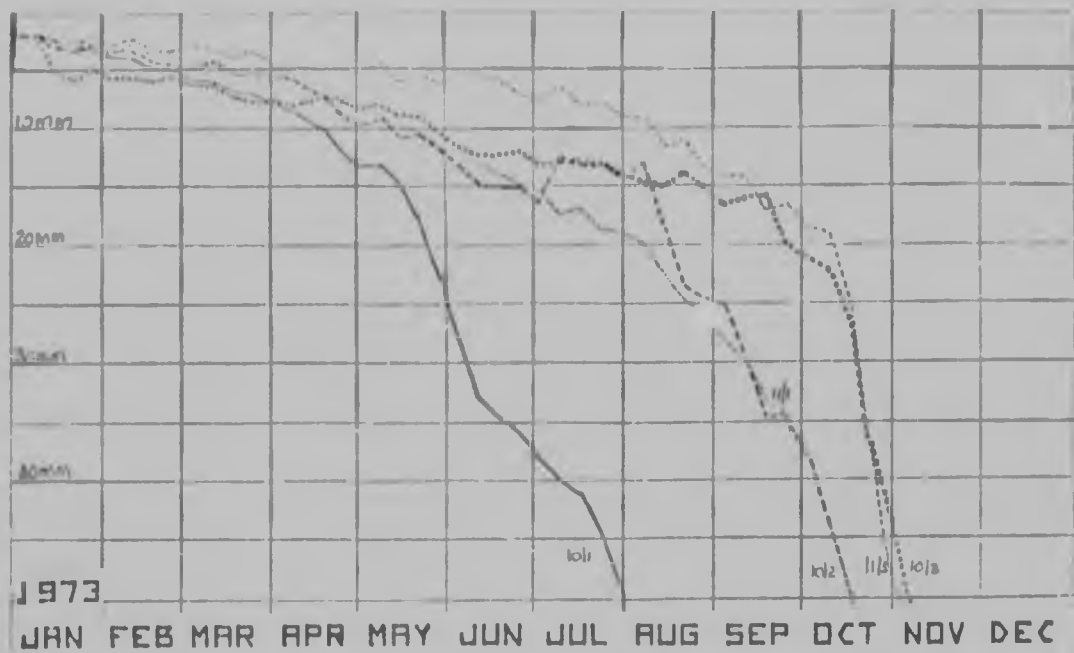


Fig 7.32 Effects of Block 7/2 caving on Block 7/1 overhang

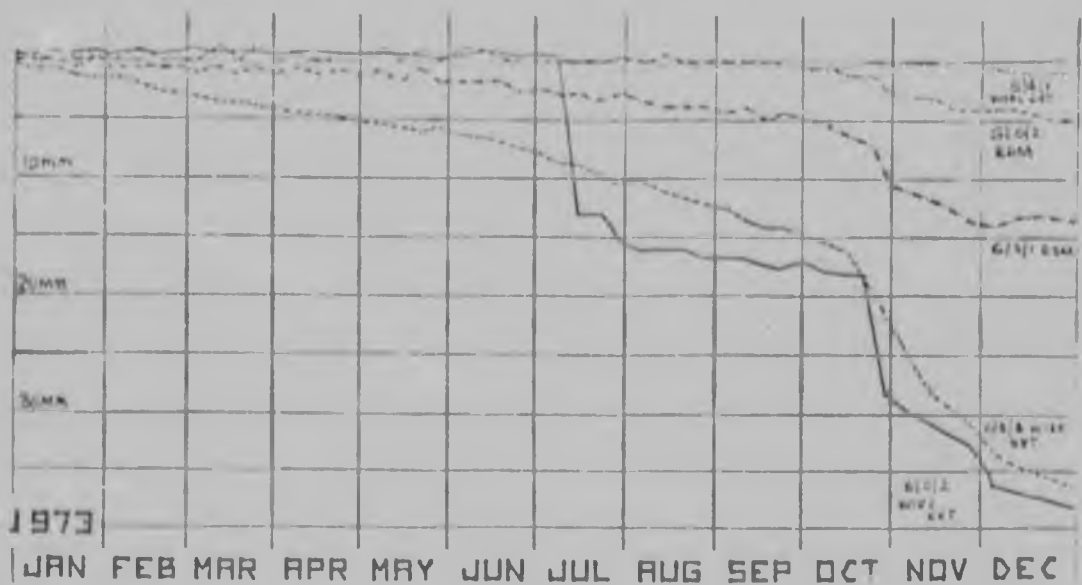


Fig 7.33 Effects of Block 7/2 caving on block 16 instruments

On surface, the cracking observed over the northern portion of the block, initially in September, extended significantly in October 1973 (Fig 7,34).



Fig 7,34 Surface cracking Block 7/2

Subsequently, the surface cracks extended over the Block 16. These cracks did not have any initial vertical displacement but were caused by the relative lateral displacement of the ground on opposite sides of steep dipping slips. This corresponds to an angle of break of  $76^{\circ}$  with respect to Block 7/2 but Block 16 may have influenced this.

#### 7.562 Observations from the periphery

With Block 7/1 on the northern side of the block and block 16 on the eastern side, the caving of Block 7/2 had a significant effect on the stability of the overhang over the eastern end of Block 7/1 and the back of Block 16. These effects will be discussed under separate headings.

Only on the western side of the block could the effects of caving on the peripheries be observed. Here, small dilation and left lateral slip displacement were observed on north/south striking slips on both 235 and 295 levels. The small rises recorded on the 275 level remote displacement meters on the western side of the block Nos 8, 9, 10 and 11 appear to be anomalous and could be due to a faulty reading board mounting.

On the eastern side of the block below Block 16 some movement was noted on a major E-W striking shear and on the shearing associated with the dyke. These may have started before, but were only noted in October, during which month the greatest rate of displacement occurred. Further displacements were recorded at a slow rate for some months afterwards. These displacements are important because they could cause rock falls which would prohibit the use of the bins located in the area, cutting the production from Block 7/1 and 7/3.

#### 7.563 Observations below the undercut

During the undercutting of this block extensive damage was experienced ahead of and below the 345 level undercut face.

Ahead of the face, the damage observed was in the nature of spalling and small local shear at dilation displacements slips in the rock surrounding the development. This was particularly noticeable in the less competent areas, and in the "pillar", between block 7/2 and 7/1 on the northern side of the block where yielding arches had to be installed.

On the lower undercut level (350 level), the area affected was larger than on the upper undercut level, with spalling and small local shear and dilation movement on slips in the rock surrounding the development. This was most noticeable in the less competent ground on the corners of the ring drilling drives and crosscuts. The appearance of the damage was broadly related to the 345 level face position, as it occurred in a broad zone both ahead of and behind the face position. In the more competent

areas, the fractures observed in the development consisted of spalling cracks in the back, and small, compressional displacements on shallow dipping slips, which gave the impression that the cause was a horizontal, not vertical stress. Rock bolting improved conditions sufficiently to allow undercutting to proceed.

On 360 level, damage similar to that observed on 350 level occurred. However, on the western abutment an unfavourable slip orientation necessitated yielding arches. Here, displacement on N-S slips were noted which were more regional than confined to the zone of influence of the development.

The grizzly level, 365 level, was extensively supported by steel arches as a precautionary measure did not show the visible effects of the undercut face advance. In the western abutment, on 350 level, spalling and regional movements were observed in the unsupported areas. The arch load indicators responded to the undercutting with increases and then decreases in the loads imposed on the arch. On the southern side of the block the arch load indicators showed an increase in load throughout the southern-most grizzly drive. This drive, despite being located in good ground suffered considerable spalling, necessitating the completion of arch installation.

#### 7.57 EFFECTS OF CAVING IN BLOCK 7/2 ON BLOCK 7/1

The caving of block 7/2 had an effect on block 7/1-7/3 mainly in the overhang over the eastern end of the block. The boreholes and displacement meters located on 205 level recorded general slow subsidences through 1972 and to August 1973, and then accelerated between August and October, coincidental with the removal of the pillar between 7/2 and 7/1. Several of these instruments were lost during October as the caving of Block 7/2 induced further caving in Block 7/1 overhang (Fig 7,34). The ground on the east side of the block displayed a relief of stress in an E-W direction where the newly caved ground intersected a drive, and could be seen subsiding almost intact on nearly vertical slips.

This also caused a general deterioration in the blocky incompetent, and sheared ground associated with the 170 dyke in the eastern periphery of Block 7/1-7/3.

#### 7.58 EFFECTS ON CAVING IN BLOCK 7/2 ON BLOCK 16

During 1972 and 1973, very slow subsidences were recorded over the back of Block 16 by the 205 level remote displacement meters and borehole

wire extensometers. However, in July 1973 three borehole extensometers on the western side of the block recorded sharply increased subsidence rates possibly caused by some failure in the ground between Blocks 16 and 7/2. During August and September some instruments recorded a gradual increase in subsidence and on October 24 a general speeding up in the rate of subsidence started (Fig 7,33). At that time it was discovered that the cave back had reached the 205 level elevation over the central portion of the block. It was then decided to drill a hole from surface to intersect the back and to monitor the progress of the cave upwards. This borehole was completed in December by which time the cave back had reached the 170 level elevation. From then until June 1975 the back moved upwards at a rate of several metres per month (Fig 7,24). The caving indicator also showed that a cavity existed between the broken ground and the back from time to time. This cavity broke through to surface in June 1975 as a large sinkhole. From this it was deduced that the cave back was again a steep sided dome similar to that deduced for Block 7/4, as the instruments on the north west and south west sides of the block had indicated that a complete failure of the ground lying between block 7/2 and 16 had not occurred, but instead relatively small movements had been recorded indicating some failure or displacement on slips on this zone. The western abutments of the block were still relatively solid on the 205 level elevation while over the centre of the block the cave back was measured above the 170 level elevation.

#### 7.59 SUMMARY

The undercutting of Block 7/2 resulted in initially a dome shaped cave with areas in beneath the back where cavities developed. These cavities over the centre of the block narrow but on the western periphery up to 10,5m high. The undercutting resulted in a zone of damage broadly related to be face position on the second undercut and overcut levels. This damage, however, had the appearance of being caved by a lateral not vertical stress.

The completion of the undercutting along the boundary with Block 7/1 caused mass subsidence type caving on the northern side of the block with a steep overhang over the south side. This also initiated further

caving of the overhang on the eastern side of Block 7/1, and, the caving of the back Block 16. In this block lateral stress relief was indicated by cracks on surface and the caving took the form of an upward propagation of a steep sided dome shaped cave back beneath which a cavity periodically developed.

#### 7.6 DISCUSSION

Some common characteristics and interesting differences can be recognised in the caving of the four blocks described in this chapter.

A substantial area was undercut in each of the blocks before caving started. This 'undercut' was created by the shrinkage operation in Block 6, sublevel caving in blocks 7/1-7/3 and 16 and by the undercutting of the ore for the block caving of 7/2. Consequently, the back was roughly arch or dome shaped (determined by the ore-body shape) in the first three cases, and flat in the latter. The mined areas were partially filled with the blasted ore leaving one or more cavities between the pre-broken ore and the back.

Initially caving took the form of sporadic sloughing from the back of these cavities, or where the broken ore filled the undercut to the back cracking with more or less detachment from the sides and slower subsidence. With sloughing from the back the cavities of Blocks 6 and 16 progressed to surface. This caving mechanism was similar to the stress caving in Block 7AB but in all these cases, the previously mined block on one side influenced the stresses in the back, and probably contributed to the instability of the back. In Blocks 7/1-7/3 and 7/2 the influence of the adjacent caved block became sufficient to cause the mode of caving to change from stress caving to mass subsidence caving between 30 and 60 metres above the initial back elevation. The caving mechanisms are discussed more fully in the next chapter.

The ground in the peripheries of each of these blocks was affected by the caving manifest in the displacements observed on slips and in the damage to drifts in this ground. As discussed more fully in the next chapter the rate of displacement on the slips were normally at a maximum at the time of caving, and subsequently slowed over a considerable period of time. The damage observed in the peripheries took the form of slabbing and falls of rock dislodged or forced from the back or sidewall of develop-

ment openings, also rock falls and spalling attributable to changes in the stress field in the area, with the consequent changes in stress around the excavation and its stability.

## CHAPTER EIGHT

### ANALYSIS OF GROUND BEHAVIOUR OBSERVATIONS

#### 8.1 INTRODUCTION

In this chapter it is hoped to analyse the ground behaviour by evoking theoretical model materials to describe the stress-strain-time relationships which are important in understanding the ground behaviour. It will be shown that the ground on this mine on a large scale behaves as a quasi-Bingham substance and on a small scale may be either elastic or plastic. Later it will be shown how these characteristics effect the practical side of the mining namely, the condition required for caving and the form that the caving will take as well as the stability of development workings.

#### 8.2 MECHANICAL MODELS

The stress-strain-time relationships of geological materials may be represented by one of eight mechanical models given in Jaeger's "Elasticity Fracture and Flow" (1956 P 99 - 105) (Fig 6,1).

- (1) The perfect elastic or "Hookean" substance Represented by a spring, with the extension being instantaneous and proportional to the stress.
- (2) The perfectly viscous fluid or "Newtonian" substance Represented by a dashpot, the rate of strain  $\dot{\epsilon}$  is related linearly to the stress, and the strain is determined by the duration of the stress and strain rate.
- (3) A yield stress  $J_0$  Represented by a friction contact which would only yield if the applied forces exceeded the frictional force ( $J_0$ ) due to static friction.
- (4) The perfectly plastic solid or "St. Venant" substance Represented by a spring and friction contact which has an elastic strain until the yield stress is reached.
- (5) The "Kelvin", "Voigt" or "Elasto-Viscous" substance Represented by a spring and dashpot in parallel in which the stresses on the system are equal to the sum of the stresses on the spring

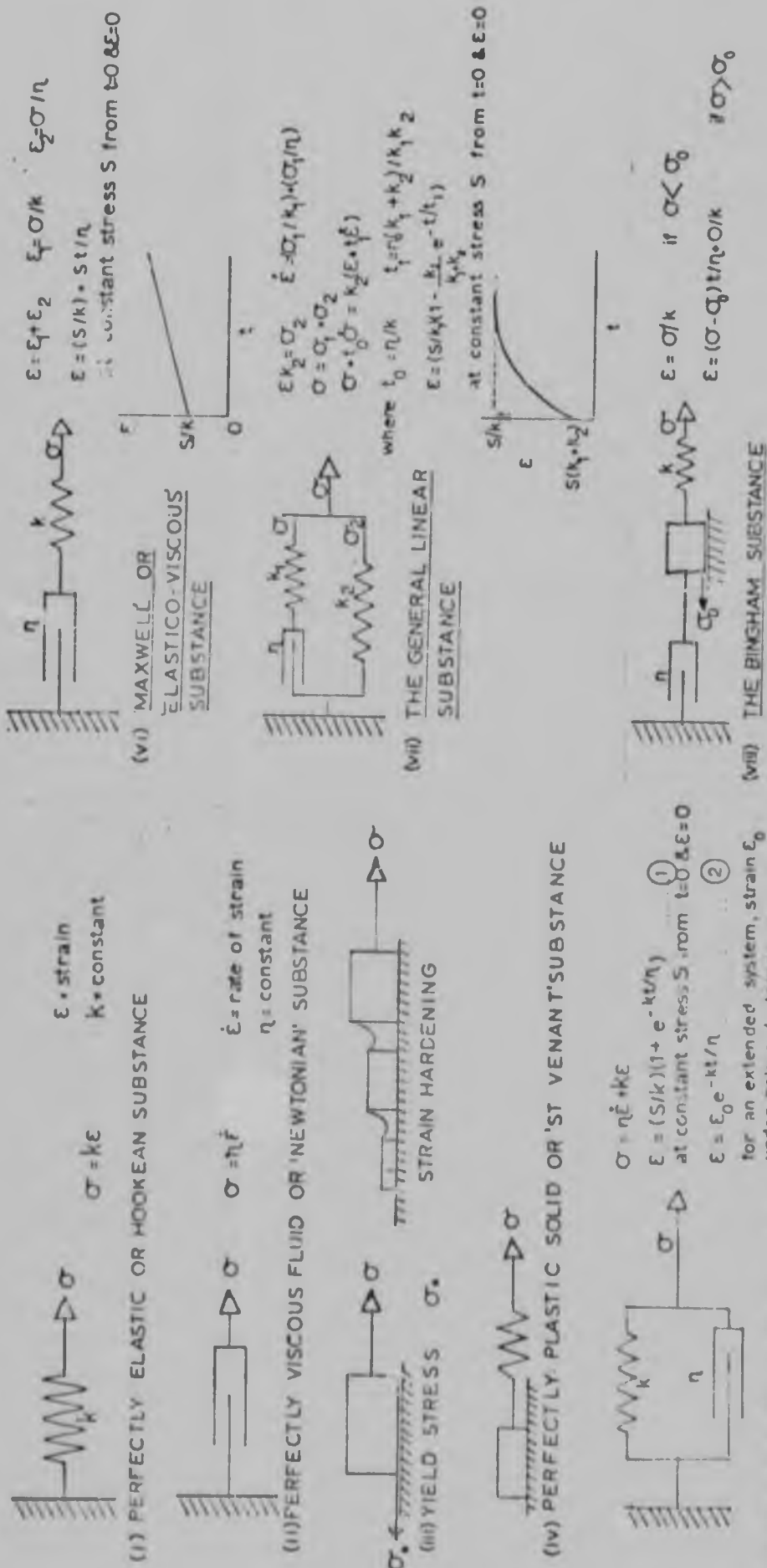


Fig. 8.1 Rheological models

(E) and dashpot ( $\eta$ ). Should the system be subject to a constant load, the ultimate strain would be reached exponentially and if the stress is removed the strain is decreased exponentially to zero strain.

- (6) The "Maxwell" or "Elastico-Viscous" substance Represented by a spring and dashpot in series, this material is like pitch, elastic to instantaneous loads, but flows viscously with sustained low stresses. Under constant stress, there would be an instantaneous elastic strain followed by linearly increasing time-dependant strain.
- (7) General linear substance Represented by a spring in parallel with a Maxwell element, the material under constant load has an instantaneous strain which increases exponentially to a final value determined by spring on its own.
- (8) The "Bingham" substance This is represented by a spring, friction surface and a dashpot in series, the dashpot here restricts the rate of movement once the yield point is passed, unlike the perfectly plastic material which has no limit to the rate of movement. This material behaves elastically for stresses less than the yield point and for greater stresses it deforms with steadily increasing strain. A portion of this strain is not recoverable on release of the deforming stress.

### 8.3 SMALL SCALE DEFORMATION

On a small scale, such as in laboratory specimens, most of the rocks found on the mine behave elastically. The possible exceptions are the footwall talc rocks where the gouge filled shear horizons and weak sheared zones can be expected to give it a viscous nature. The incompetent nature of these rocks, makes core specimens of these rocks for laboratory testing practically unobtainable.

The elastic nature of the dunite, serpentine, carbonated serpentine and talc-carbonate rocks is demonstrated by the instantaneous response to loading and absence of significant permanent deformation after a period of sustained load. These properties are illustrated in the graphs in Fig 8,2.

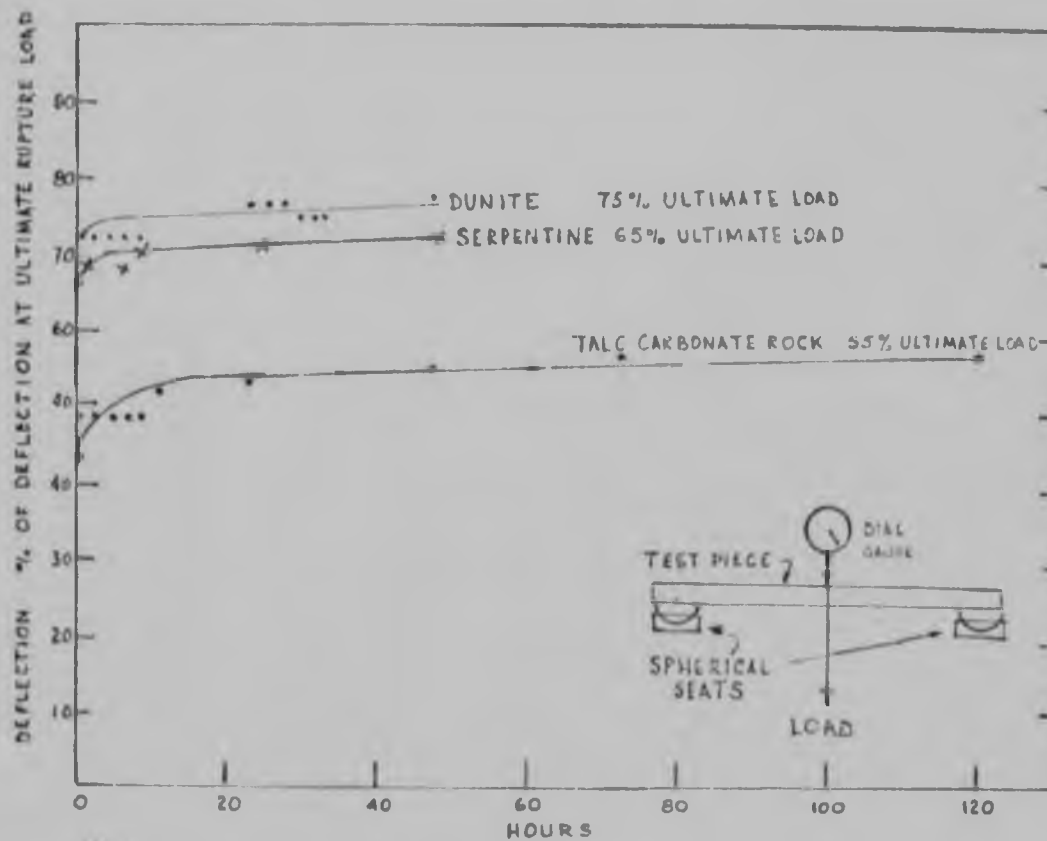


Fig 6.2 Creep test results

#### 8.4 LARGE SCALE DEFORMATION CHARACTERISTICS

When a large volume of rock containing several joints is stressed, the rock may not only deform elastically, but depending upon the directions and magnitudes of the three principal stresses to the orientation and frictional properties of the joints, there may also be a plastic component resulting from small shear displacements on one or more joints.

Small displacements on joints have been observed and recorded in many locations in the peripheries of caved blocks on this mine (Fig 6,23). In other localities, where high stresses were expected, there were no visible effects of the stresses in the form of spalling and other damage and from this it was inferred that the stresses had been reduced and redistributed by small shear adjustments on joints. The lower-than-expected abutment loads in the peripheries of Block 16A was one example (See 7.462).

The rate of displacement measured by the Demec extensometer on slips in the peripheries of Block 7AB were generally very slow and apparently continuous, but there were some which displayed small sporadic displacements. These slow, frequently uniform rates of displacement

indicate a time factor in the deformation characteristics of the ground. This could be either a time dependent response of ground to a change in stress, or to an immediate response of the ground to a gradual change in stress. A gradual change in stress can be brought about by advances in the mining or to the gradual development of a zone of adjustment around a mining operation. As the caving of block 7AB was relatively rapid and in the peripheries was followed by a protracted period displacement on slips, it is considered that the time dependency is primarily a property of the displacement on the slips.

because of the small displacements on the numerous joints and the time-dependent response, the rock mass behaves as a quasi-Bingham substance. The spring provides an instantaneous elastic response, the dashpot a controlled rate of sliding of the friction block when the applied load exceeds the yield point provided by the friction block.

#### 8.41 THE NATURE OF DISPLACEMENTS ON JOINTS

##### 8.411 Type of joints on which displacement occurs

In the peripheries of a caved block, the joints which show displacement first and which exhibit the greatest amounts of displacement are the suitably orientated major structural features such as faults, major slips or shears in the area. These major features are usually planar and may have associated zones of shearing and in many cases a gouge filling. Closer to the caved block the displacement occurs more frequently on minor joints such as slips which are not as smooth or planar.

##### 8.412 Amounts of shear movement and dilation

The monitoring of the direction of displacement on these slips showed that the lateral displacement was almost invariably accompanied by dilation. This could be caused by the "riding up" of irregularities in the slip surfaces on each other during lateral displacement, however, on the more irregular minor slips it is probable that reduction in the stress or change to tensional strain in a direction perpendicular to the slip face is required to overcome the interlocking of the irregularities. The lateral displacement and dilation components of movements recorded on a slip in the periphery of block 7AB is given in Fig 6,22.

#### 8.413 Irregular displacement rates recorded on some slips

Whilst the displacement recorded on the majority of the slips monitored with the Demec extensometer was apparently continuous and slow, there were a significant proportion of slips which displayed a jerky motion. As these instruments were read once per week, there may have been more slips which illustrated a jerky motion where the period of jerks was less than one week. However, where jerky displacements were recorded on slips, the jerks occurred at long and irregular intervals of time. It is probable that these were the result of the irregular fluctuations in frictional force caused by shearing off the species of irregularities rather than a sudden change in stress. Jaeger and Cooke (1964 P 63) recognise two 'stick-slip' phenomena, one resulting from shearing of irregularities during sliding, and the other phenomenon observed in the sliding on finely ground surfaces of rock. This type of 'stick slip' sliding they have called 'stick slip relaxation oscillations', and ascribe it to a coefficient of static friction that is higher than the dynamic friction. Under a constant overall strain rate, there is a build-up in elastic strain until the stress is sufficient to overcome the static frictional resistance, and then shearing starts and continues until the force drops to the dynamic frictional resistance which brings movement to a halt, until once again the elastic build-up exceeds the static friction resistance.

#### 8.414 Relationship of rate of movement to changes

It is usually very difficult to determine when, exactly, a change in the stress field takes place so that the effects on slips can be related to them. When Block 7AB caved, the slips being monitored were those on 270 level on the south side of the block, and of these most recorded substantial movement in January and February prior to caving with a general slowing down in March and April, and with little or no change in the rate of movement at the time of caving in April. It was clear that the slips in this area were being affected by the undercutting operation and hardly by the break through of the cave to surface. Elsewhere in the peripheries, however, displacement was noted on numerous slips during April and May. Only after this displacement had started were these slips monitored. The gradual slowing in the rates of displacement recorded

showed that after a major change in stress, adjustments in the form of displacement on slips continued for several months.

During the undercutting of the second phase of Block 7AB, displacement was recorded on the Travelling shear (See 6.35). This movement was related to the crushing of drawpoints in this area. The rate of displacement slowed considerably during the production from the first phase, speeded up during the working of the second phase, and substantially on the day Block 7/1-7/3 caved. The displacement on this slip was, in general, very closely related to the passage of mining. The other slips being monitored during this period and in this area did not show this distinct relationship to the progress of mining (Refer Fig (,26)).

#### 8.415 Orientations of slips on which movement occurs

Displacement was recorded on several joint orientations in the peripheries of the Block 7AB and these movements could be broadly categorised into lateral (wrench fault) displacements on steep dipping slips striking at an acute angle to the boundary of the caved area, normal and reverse fault displacements on slips striking fairly parallel to the periphery (Fig 8,3). On slips perpendicular to the boundary of the caved area particularly on the upper levels, such as in 125 level main crosscut, small lateral displacements were observed occasionally. Broadly, the directions of displacement on the joints indicate a reduction in lateral stress perpendicular to the boundary of the caved area, and/or an increase in parallel to the boundary. The vertical displacement on the slips striking parallel to the boundaries are due to the vertical stress, either overburden load only, or also increased temporarily by abutment loads, influenced by a reduced horizontal stress. The slips perpendicular to the boundary, on which lateral displacements were observed on the upper levels, were, it is believed, merely reflecting the displacements of ground below the observed slips, in other words this ground was being carried by the ground moving beneath it.

#### 8.42 THE STRESS MODIFYING EFFECTS OF SHEAR MOVEMENTS

In the preceding two chapters it has been suggested that the stresses in the peripheries had been modified by the observed small displacements on slips and shears. It was also speculated that the stresses measured in the abutment of the first level in the sublevel caving of

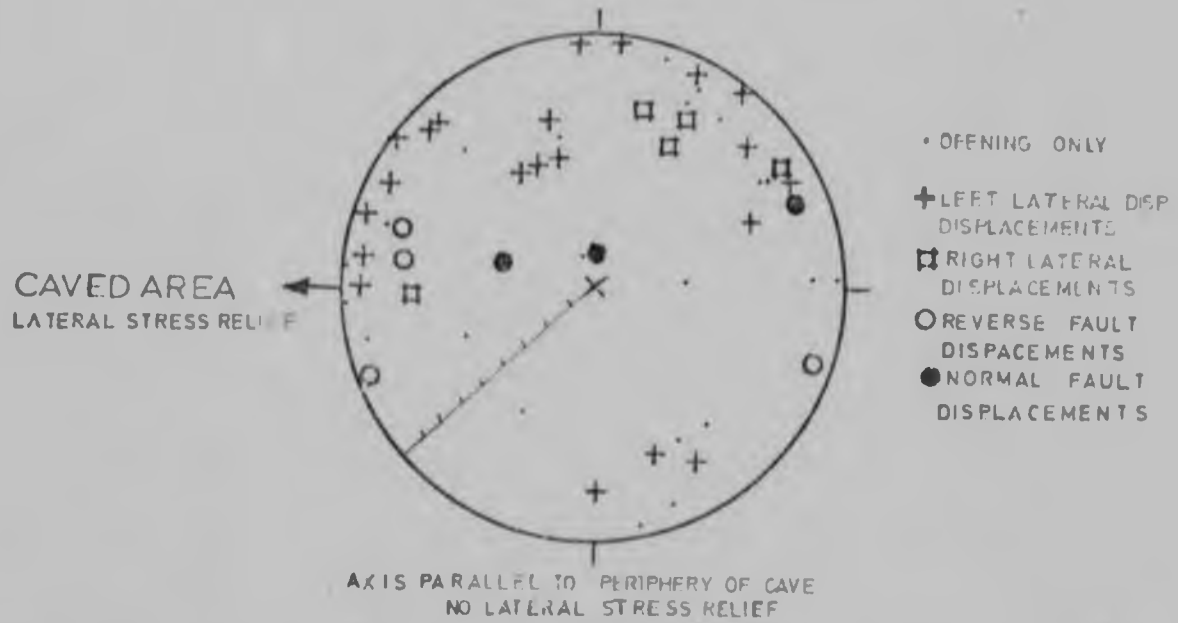


Fig 6,3 Stereographic representation of displacements observed on slips in the peripheries of block 7A3 slip direction relative to centre of caved area

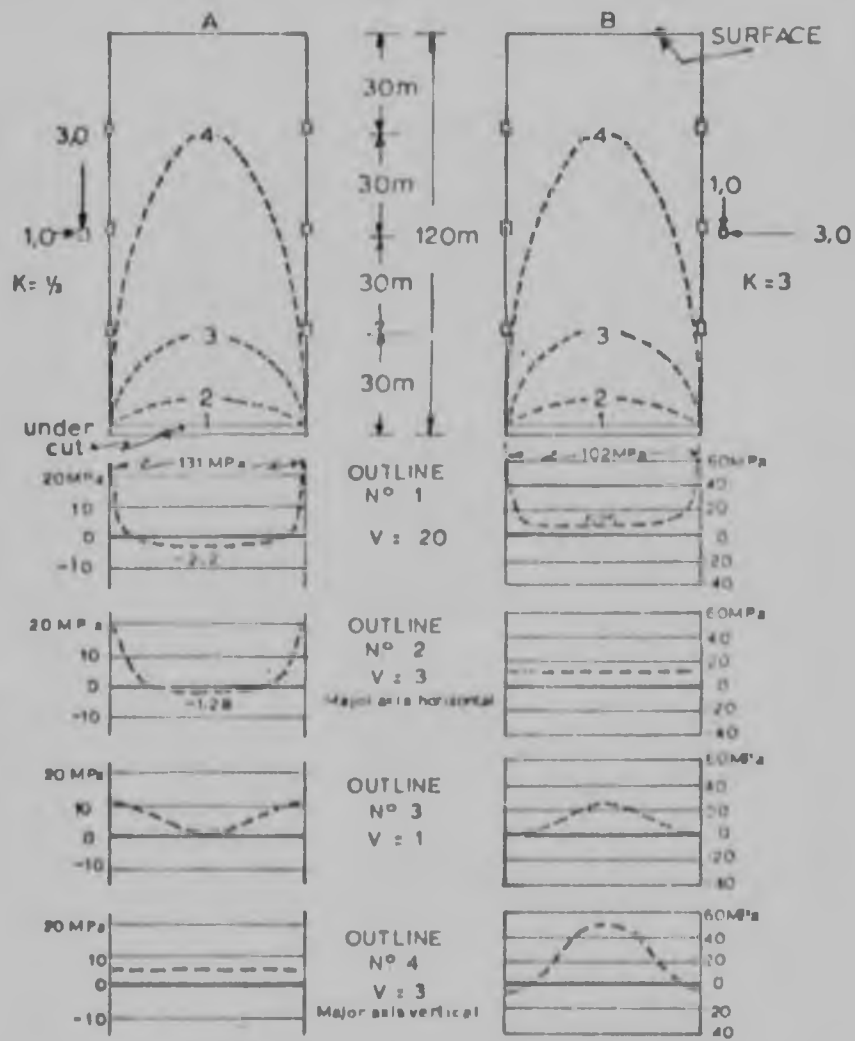


Fig 6,4 The stress changes induced in a block under two different conditions of loading (after Morrison and Geldard 1964)

Block 16A were not as high as had been expected because of small displacements on slips the maximum stresses in the immediate abutment had been reduced and redistributed further away from the face. Using Hookes law for uniaxial compression a first estimate of the effect that the small displacements observed on slips could have on the stresses could be obtained. For example, the elastic deformation of the partially serpentinised dunite with a modulus of elasticity of 50 GPa, under a stress of 1 MPa would amount to 0,02mm per metre.

From this it can be appreciated that the small movements of the order of millimetres observed on the many slips in the peripheries could have a significant effect on the stress field surrounding the block. However, in a quasi-Bingham substance the total deformation is dependent on the magnitude of the deforming stress and the duration of the stress; in the back or periphery of a cave block the magnitude and duration of the deforming stress would depend on the situation. In general the larger the zone in which shear displacement on slips is possible, the larger the total deformation will be and the longer it will take for the principal stress deviations in this zone to decrease to the yield strength of the rock mass and the final state of stress to be reached.

The shear displacements occurring on the major slips or the yielding of the rock mass would tend to reduce the principal stress deviations within the yielding zone and to increase the stress in the areas adjacent to it outside the potential yielding zone. Where, however, the principal stress in the yielding zone is due to gravitational forces, then the yielding will not reduce the principal stress deviations unless bridging occurs which redistributes the weight of the material overlying the yielding zone onto the peripheral mass outside the yielding zone, or the lateral forces resisting yielding, are increased by some process such as consolidation or an event such as the cessation of mining.

The small displacements of the order of millimetres observed on many slips in the peripheries of Block 7AB could have had a significant effect on the stressfield surrounding that block. Fig 6,24 illustrates the slips which had movement noted on them on 205 level in the peripheries of Block 7AB and Block 6, together with the ground movements recorded on the survey pegs. The total movements estimated for slips on the eastern perimeter of the block on 170 level amounted to 6mm in 160m, while the total shortening recorded on the survey traverse was 20mm. The 14mm strain given by the difference between these indicates that the horizontal

stress may have been increased by the order of 4,5 Mpa and this is in close agreement with the difference between measured lateral stresses of 11,8 and 10,9 MPa and the extrapolated virgin rock stress of 6,5 MPa (See 6.3). There was further an adjustment indicated by the 6mm observed yielding which may be equivalent to a stress relief of about 2 MPa.

From this example it can be concluded that when an excavation is made in the type of ground encountered at Shabanie, there is a redistribution of the virgin stress around the block, determined by the shape of the excavation. These stresses would be modified locally by local differences in the elastic properties of the rock, and on a larger scale by differences in the elastic properties of the various major rock types. If in this ground there were also suitably oriented slips, and if the stress conditions are such that the shear stresses exceeded the frictional and cohesive strength of the slip, then displacement would occur on these slips, reducing the stress in the area and increasing it outside the potential failure zone.

In this way the stresses around the major excavations at Shabanie can be expected to be significantly different to those in an elastic medium. The yielding on slips tends to reduce the general level of stress in high stress areas and redistribute them over a wider area and so the high stresses that lead to rockbursts on other mines are unlikely to be reached on this mine.

#### 9.5 CAVING MECHANISMS IN A QUASI-RINGHAY SUBSTANCE

Two distinct situations in which caving can occur have been recognised in the blocks studied. These result in caving mechanisms which are sufficiently different to warrant identifying names, viz "stress caving" and "mass subsidence caving". These two forms of caving may be observed in different areas in a single block, "stress caving" occurs in areas where no previous mining has removed the lateral constraint in one direction and "mass subsidence caving" occurs in areas where the lateral constraint has been significantly reduced. The characteristics and mechanisms of these two forms of caving are discussed below.

##### 8.51 STRESS CAVING

###### 8.511 Principles

In the literature it has long been recognised that more difficulty

is experienced in initiating caving in the first block in an area than in subsequent blocks. The stress caving situation may be equated with the caving of the first block in other mines. However, on Shabanic Mine due primarily to the undercutting sequences adopted it also occurred in subsequent blocks.

Although the rock on this mine is anisotropic because the joints have preferred orientations, it is worth while looking at Morrison and Geldard's (1964) paper on caving in isotropic rock in which they calculated the stresses in the back at various stages of caving (Fig 6,4). In a low lateral stress field, on the completion of the undercut horizontal tension of 0,7 units (one unit being equal to the vertical stress at the depth of the undercut) would exist over much of the back of the block. The effect would be to open up the joint planes to the degree required to induce local and progressive caving. However, with successive falls from over the centre of the block, the back would become more arched and progressively less favourable to caving as the tangential tensile stresses gradually change to compression with increasing height of the arch. This they say could arrest the cave, but consider that with the weakening effects of abutment shearing, fringe drifts and/or abutment stopes, the back shape will vary between outlines 1 and 3 i.e. with tangential tensile stresses in the back. In a high lateral stress field, the tangential tensile stresses disappear, and are replaced by compressive stresses which tend towards tightening rather than loosening the undercut mass. This they conclude militates against free caving and good fragmentation. Immediately after undercutting, the tangential stresses in the back would be slightly compressive, unlikely to support the dead weight of the rock in the back and falls would occur. With these a state of uniform tangential compression is rapidly approached, which could restrain the dead weight and arrest the cave.

In an anisotropic rock, the orientations of the planes of weakness must be considered. Fig 6,5 illustrates the stress directions in the back of Block 16, obtained from a finite element analysis. From this it is clear that in a high lateral stressfield, the required orientations of the stresses for shear failure on the steeply dipping slips occurs only in a small area near the extremities of the undercut. However, shearing on joints with low dips is probable over the greater proportion of the back and if these joints persist above the back, a series of

failures could occur giving an upward progression of a cave. The maintenance of a cavity below the back would assist this process.

If, however, the dominant joint direction is horizontal, the structures in the back would resemble a series of beams or plates which would buckle and sag as illustrated in Fig 8,6. Denkhaus (1964 P 319) points out that with deformation of the abutments the maximum stresses and possible failure area lie more towards the centre of the undercut than it would with rigid abutments. Due to the bending of the lower beam, the span of the second beam would be smaller than the first. As the deflection of each beam depends on its thickness and elastic deformation modulus, it is possible that the lower beam deflects more than the overlying one so that a gap, termed the Weber cavity, occurs between the two beams. If one joins the abutments of all beams one obtains a dome shaped curve (Fig 8,6).

This beam concept is based on the assumption that no bond exists between successive layers so that they can deflect freely. If such a bond exists the two layers behave like one thick beam. If, however, the shear force caused by the deflection loosens the bond, deflection of both beams suddenly increases and this may manifest itself in a burst-like fall of hanging over a large area. With a high lateral stress the increased deflection of the beams can be expected. Parker (1966 P 1192) observed this type of buckling in a high lateral stress field at White Pine Mine, Michigan, where he also noted that the buckling lead to a reduction in the lateral stresses in the beam. This would tend to favour caving in a rock also containing vertical joints. A form of caving similar to this was in fact observed by Fletcher (1960 P 421) at Miami Mine.

The caving process which fits the observed features of stress caving on Shabanie Mine may be summarised as follows:

In the back of an undercut area the rock is subject to stresses which are the result of the redistribution of the pre-existing stresses around the undercut and the weight of the rock in the back. If the joints in the back are favourably orientated with respect to the stress directions and if the shear and/or tensional strength of the joints are exceeded by the stresses imposed on them, some dilation and lateral displacements will occur. These will result in a modification of the stress field and further yielding may ensue. In the high lateral stress field of Shabanie



Fig 8.5 Stress directions in and around Block 16 from a finite element analysis

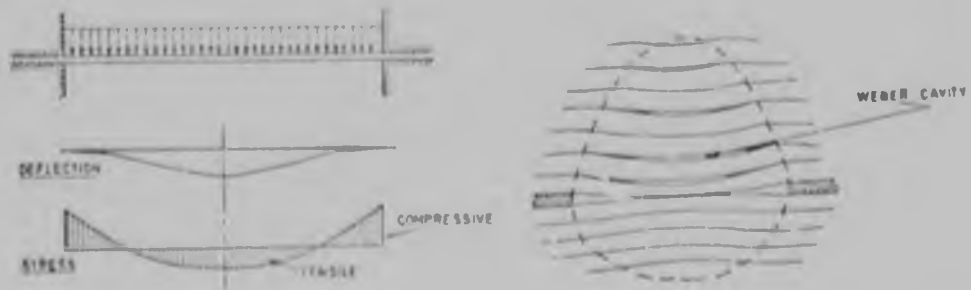


Fig 8.6 Buckling of beams (after Jenkman)

Mine the joints which would have a 'favourable' orientation over a flat back of an undercut are those joints with a dip of up to  $35^{\circ}$ . Joints with steeper dips may have a favourable orientation over a narrow zone along portions of the peripheries. With a back, which is flat and contains almost horizontal joints some bed separation could develop, which with buckling, could result in a reduction in the normal stress and shearing on other more steeply dipping joints. If, however, the back is dome-shaped then shearing may occur on the more steeply dipping slips in the zone along portions of the peripheries of the back. The zone is wider than would be the case with a flat back.

If these processes result in the detachment of blocks of rock from the rock mass, which may continue to subside under gravitational forces alone when the underlying ore is drawn, then the ground may be considered to be caving. A block may be considered to have caved when sufficient ground has entered this state to enable extraction to proceed efficiently and safely. Caving may be considered to be complete when it reaches surfaces.

#### 8.512 Discussion of examples of stress caving on Shabanie Mine

Stress caving occurred over the sublevel shrinkage of the eastern half of Block 6, over the top of the sublevel caving operations in Blocks 7/1-7/3 and 16, in Block 7/2 and in Block 7AB between the cut and fill stopes and Block 6, and over the western half of the first phase undercut area. With the exception of Blocks 7/2 and 7AB the backs of the undercut areas were all arched or dome-shaped.

In all blocks except Block 7AB the visual and instrumental data indicated that the cave back assumed a steep sided dome shape, and that cavities developed between the cave back and caved material. The cave back progressed upwards by sporadic falls. Horizontal tensional cracks were observed in the back in both areas in Block 7AB and, in Block 7/2 where cavities are considered to have formed from the gradual widening of a tensional crack. For that portion of Block 7AB which formed the exception to the general observations above, it is suggested that had the undercut not been the cut and fill stopes in which the pillars slowly crumbled, the observed tensional cracks may have developed into a cavity. Also a dome-shape might have formed if Block 6 had not overlain the area.

In all the blocks studied the majority of the slips were steep-dipping, and in general the only joints which were relatively flat dipping were the fibre seams. However, in the northern half of Block 7/2 a set of  $35^{\circ}$ S dipping slips had been mapped. It is probable that the caving took place by the progressive shear failure on these joints in the peripheral areas and by separation, buckling over the central portion of the back. The major geological features which probably assisted the caving process includes the Zone A shear in the back of Block 6, Zone B in Blocks 7AB and 7/1-7/3, and the shearing associated with the 170 dyke in Blocks 7/2 and 7/1-7/3. In Block 16 the Zone B shear lay to the north of the block and in the initial stages had no effect on the caving. Later the cave back progressed through this zone with no apparent deflection.

The stresses in the backs of all these blocks were probably affected by previous caving in the near vicinity. The one possible exception was Block 16 which started in an isolated area and the caving did not progress very far until the Block 7 caved area was extended towards it by the caving of Block 7/1-7/3 and Block 7/2. It appears that the caving of Block 7/2 adjacent to Block 16 changed the stresses in the back sufficiently for caving to proceed at a fairly constant rate with sloughing from the back of a cavity but, not enough to change the mode of caving to mass subsidence caving.

The rate of propagation of the cave back is relatively slow, but depends very largely on the structures, the class of rock in the back, and the stresses in the back. The rates of caving recorded in Block 16 varied from a few falls per year to a fairly regular 10m/month.

No comparative studies have been made on the fragmentation resulting from this form of caving but it is expected that in the same class of rock, fragmentation with this form of caving would be better than with mass subsidence caving because with sporadic falls from the back of a cavity the blocks become rotated relative to each other which would allow greater arching and attrition in the draw column.

## 8.52 MASS SUBSIDENCE CAVING

### 8.521 Principles

Mass subsidence caving occurs quite readily where the lateral

restraint has been significantly reduced in at least one direction above the undercut. The reduction in lateral stress is usually provided by a common boundary with an adjacent caved block, or in some cases, by a slot cut along one or more boundaries. The lateral stress has to be sufficiently reduced to allow shear failure in a vertical sense on the near vertical slips. When caving occurs it is characterised by the orderly subsidence of large columnar blocks of rock bounded by these nearly vertical slips.

#### 6.522 Discussion of examples

Mass subsidence caving was observed in two areas on Block 7AB, where in the first area the slot cutting and shrinkage on the south side, and in the other the Block 6 cave removed the lateral restraint. This form of caving also occurred in Blocks 7/1-7/3 and in block 7/2 where the mode of caving changed as the caving progressed upwards from stress caving to mass subsidence caving, and the removal of the lateral restraint by the caved zones in block 7AB and Block 7/1-7/3 respectively. The propagation of the cave to surface was rapid in all cases and the surface subsidence took the form of the orderly differential subsidence of blocks of rock bounded by steep dipping slips.

As the caving of these areas proceeded rapidly, utilising only the relatively minor nearly vertical slips, it appears that major structural fractures are not essential to this form of caving, but would obviously help.

Open cavities were observed in only one of these four areas, and as these cavities were of limited extent, it is concluded that cavities will probably only develop at the undercut elevation and would be limited in area by the spacing and orientation of the nearly vertical slips.

Shortly after Block 7AB caved a low bulking factor of only 6% was calculated for that block, and a subsequent survey showed that the factor had increased to 12%. These low bulking factors are probably a direct result of the orderly subsidence of large columnar blocks of rock formed in this sort of caving.

#### 8.53 CAVING CRITERIA

An early idea on caving on this mine was that a new cave block should be sited adjacent to a previously caved block to ensure "continuity

of cave". On this basis it was decided to start the mining in Block 7 adjacent to Block 6. Subsequently, the hydraulic radius of Block 6 was adopted as the caving criterion for Block 16 as it had a similar major structural feature in the northern periphery.

It has since become obvious that the hydraulic radius as a criterion for caving is inadequate. But there is still a clear need to have a simple way of assessing whether an orebody will cave satisfactorily, with a given undercut area, or alternatively guide lines are required for determining the size of the undercut needed to induce caving. In addition an indication of the size the caved material will be, is also desirable.

While it is clear that whether or not a block will cave satisfactorily depends upon the stresses in the back and on the orientations, density and properties of the joints, and in certain circumstances on the intact rock strength, particularly in tension, it is also clear that each of these aspects has to be quantified before an analysis can be made on a block which is on the border line between the caving or non-caving fields. For non-borderline cases the analysis can be very much more rudimentary. In the most elementary of these, aspects such as minimum span, hydraulic radius or area become the most important criteria while the structural geology is assumed to be a constant for a particular mine. This was born out by the assertion by Smith, Liedich and Mosier that one's past experience in a particular orebody was the best guide to the size of an area to be undercut to induce caving (Ducky 1942). More recently, Carpenter and Woolfe (1972) stated that at Rio Blanco, the minimum span was the main factor which determined the cavability, while Obert and Duvall (1966 P 574) quoted area of 40 x 55m in a salt mine with a compressive strength of 3,5 MPa which did not cave, and 43 x 86m in a borate mine with a compression rock strength of 6 to 45 MPa which stood intact for 25 years with heavy spalling only.

Good prospects for a simple approach to determining the cavability lie in the use of the geomechanics classification system. Laubscher (1975) has suggested that a guide to the cavability, fragmentation amount of secondary blasting can be obtained from the insitu classification (See Table 8,1). Also by applying a series of adjustment factors to the insitu classification the size of the area to be undercut can be determined. The adjustments are made for the effects that potential weathering, the field and induced stresses, changes in stress, movement size and joint

orientation, have on the stability of the back. Laubscher suggests the following guide to these adjustments. The adjusted rating is arrived at by multiplying the insitu by the adjustment factors, which are expressed as percentages. For potential weathering a total adjustment to 75% can be made by considering the possible decrease in RQD as the rock weathers giving an adjustment to 92%, the decrease in rock strength if weathering takes place along micro structures decreasing the rating to 96%, and by altering the condition of joint surfaces an adjustment of 82% is possible, giving a total maximum adjustment to  $92\% \times 96\% \times 82\% = 75\%$ .

TABLE 8.1  
 GEOMECHANICS CLASSIFICATION OF ROCK MASSES APPLIED TO CAVABILITY  
 (AFTER LAUBSCHER 1975)

ADJUSTED ROCK CLASS	1	2	3	4	5
Undercut area as Hydraulic Radius	not practical	30m	30 - 20m	20 - 8m	8m
Dia. of equivalent circular area		120m	120 - 80m	80 - 32	32m
INSITU CLASS	1	2	3	4	5
Cavability	Nil	Poor	Fair	Good	Very Good
Fragmentation	-	Large	Medium	Small	Very Small
Secondary blasting	-	High	Variable	Low	Very Low

Similarly the effects of the field and induced stresses can be allowed for: If the joints are kept in compression then the rating may be increased up to 110%; if shearing on joints is possible then the rating decreased to 90% and if the joints open the rating may be decreased to 76%. Similar arguments and adjustments can be applied to changes in the stressfield due to subsequent mining operations. The effects of the size and orientation of the joints may be accounted for by another adjustment to 70%. Laubscher further recommends that the total adjustments be limited to 50%.

These adjustments were derived empirically for various applications of the classification system such as assessment of caving, the design of support for underground excavations or open pit stability. At this stage of development, a pre-requisite to deciding the adjustment factors to be used is a sound practical knowledge of the magnitude and orientations of the stress likely to be developed in the back, both initially and as caving proceeds, and whether these will cause shear or tensional failure of the joints in the back. Although guide lines can be drawn up for assessing these factors in a quasi-Bingham substance in a moderately high lateral stressfield as formed at Shabanie, the system remains a first approximation.

Analysing the cavability of borderline cases therefore requires more sophisticated techniques for determining the stresses in the back and the likely effects on the geological structures in the back.

Theoretically it is possible to analyse the stresses in the back of the block by means of a three dimensional finite element analysis, which would take into consideration the stress modifying affects of small movements on slips, and which could analyse the probability of failure of structures in the back, and modify the shape of the cavity accordingly, and repeat the analysis. This approach appears imminently suitable for determining whether a block will cave, what course the caving will take, as well as effects that can be expected to develop in the peripheries. But a three dimensional analysis such as this would require a very large computer and long computation times because the rock properties, structures and the mining sequence in the area would have to be modelled. This would also involve a considerable amount of data preparation. The approach is therefore considered impractical.

A more practical approach would be to select a suitable section through the block and to analyse the stresses in the back by means of a two dimensional finite element analysis. McMahon and Kendrick (1969) did this for the Urad Mine, under gravitational forces only. On this mine, initial lateral rock stresses would also have to be applied. McMahon and Kendrick point out that the two dimensional analysis assumes plane strain which requires that all the deflections are in the plane studied. The most valid cross-section is generally through the middle of the cavity in a direction normal to the long dimension of the cavity. Errors in the estimation of stress concentration due to the plane strain assumption are less than 6% for a cavity with a length:width ratio of 5 and less than

15% for a cavity with a length:width ratio of 2.

In a two dimensional analysis such as this, the elastic properties of the various rock types could be modelled. The programme could also be designed to test for shear failure on specified joint planes and to calculate the displacements on the joint planes, and then repeating the analysis of stress distribution taking into consideration the displacements so obtained. Alternatively, the programme could provide simply the principal stress magnitudes and orientations and the areas of joint failure, determined manually from these. If extensive zones of failure are indicated, the shape of the cavity could be modified accordingly and re-analysed. This approach was adopted in the finite element analysis of Block 16 (Fig 8,5). However, in this case no failure zone was indicated, except for the narrow zone of vertical tensile stresses at the back and as no further major failure appeared likely in the stressfield presumed for the new shape of cavity, no recalculation of the stresses was done. This particular analysis was not considered to be an accurate reflection of the conditions surrounding the block, and it is considered that the stresses and structures in the immediate back determine the extent of caving in the absence of any major, weak, structural features.

These techniques are based on the assumption that the ground behaves elastically except where tension or shear failure is indicated for joints with particular orientations. There is an element of simplification in this, such as ignoring the effect of slips which do not intersect the plane of the analysis at a large angle, and ignoring the minor, less frequent or less continuous joints. A more realistic stress distribution might be obtained if an elasto-plastic behaviour is assumed. Reyes and Dece (1968) developed a finite element method employing a generalised von Mises criterion which accounts for both internal friction and cohesion. This yield function plots as a cone in principal stress space, and the stress-strain relations could be obtained by regarding this function as a plastic potential. They used this displacement method of finite element analysis with triangular elements and linear displacement functions, with a step by step application of load. At each step increments of displacements which satisfy the equilibrium were determined by a recursive process.

They analysed the stress concentrations around circular openings in an infinite medium for arbitrary value of yield parameters and initial state of stress, and concluded that the results appeared to be "fairly

realistic and reflect the effects of volumetric expansion accompanying yielding as implied by the stress-strain relations used".

In application to the large excavations which constitute a cave undercut, it may prove necessary to introduce the added refinement of modelling the progress of the undercut as well.

#### 8.54 CAVE INDUCTION METHODS

When block 16 failed to cave, ways of inducing the block to cave were attempted without success (See 7.461). The first technique tried was the pumping of water at high pressure into four boreholes drilled over the back for over a month, during which time several pressure drops were noted but no significant falls occurred from the back. Later fans of several holes were charged and blasted in an effort to cut a slot on the northern side into which compressive lateral stresses in the back could be relieved. This too was not successful.

Elsewhere the other methods have been tried with varying degrees of success such as boundary weakening or isolation, longhole drilling and blasting, coyote blasts and extending the minimum span or area.

It is highly probable that in the blocks where cave induction methods worked, the blocks were initially in a state approaching instability and close to caving, while in blocks where they were not successful the measures taken were less adequate or the back was very stable.

If remedial action has to be taken to induce caving, it is considered that what ever action is taken it should be done with a specific object in mind which will achieve a major change in the stability of the back. For instance, extending the undercut to bring a major weak shear into the back will have more effect than merely increasing the minimum span by a corresponding amount.

#### 8.6 BEHAVIOUR OF CAVED GROUND AND ITS EFFECT ON THE GRIZZLY HORIZON

The nature of the caved material and how it behaves under draw can significantly effect the economics of a block caving operation. In general, the initial fragmentation of the ore after caving and the attrition the larger blocks of ore suffer in the draw column determines the amount of secondary blasting required, the frequency and size of hang-ups, the productivity of operations and the width of the draw column. The secondary blasting is not only expensive, it is time consuming and damages the

drawpoint support work and grizzlies. Although poor fragmentation will increase the frequency and size of hang-ups, widening the draw columns and improving the final ore-recovery, these hang-ups impose point loads on the crown pillars which may severely damage them. This damage may be limited by a larger crown pillar, but with a wider drawpoint spacing, there is a greater tonnage per drawpoint requiring a longer time to draw, and a greater amount of repair work from secondary blasting and attrition. Also with a wider drawpoint spacing the rate of draw is reduced, and so for the same production, a larger area has to be drawn. Both of these factors has a bearing on the stresses imposed on the extraction horizon and may contribute to the damage on the grizzly horizon.

Each of these aspects could form a large study of its own, and so the discussion here will be confined as far as possible to the stress effects of the caved ground.

One of the earliest studies pertinent to this question was Janssen's study of the stresses exerted on the sides and base of a non-flowing bin, from which the following formula was derived.

$$\sigma_v = \frac{\gamma Z}{1 + \tan^2 \theta} \left( 1 - \exp\left(-i \tan \theta \frac{Z}{P}\right) \right)$$

Where  $\sigma_v$  is the average unit pressure on the bottom of a caving block

$\gamma$  = density of the ore

$$i = \frac{1 - \sin \phi}{1 + \sin \phi}$$

$\phi$  = angle of internal friction of fractured ore

$Z$  = depths

$\theta$  = angle of friction of ore on surface of the caved ground

From this formula can be shown that only a small proportion of weight the total caved ground is carried by the base, a conclusion which has been verified during model tests done by the Rock Mechanics department on this mine. For a given depth, the larger the hydraulic radius of an undercut, the higher the average stress on the base, and conversely for a given hydraulic radius, the greater the depth the smaller the proportion of the total weight the material in the bin is carried by the base.

Lucas and Verner (quoted by Woodruff 1962 P 519) have attempted to investigate the stress distribution across the base of the bin, by filling a bin with sand and measuring the stress distribution on the base. The

result showed a peak in the stress in the centre of the bin equivalent to twice the average stress on the base. The limited model experiments done on Shabanie Mine showed that under draw, these peak stresses may occur anywhere in the undercut area and do not necessarily occur in the centre. It is concluded that the peak stress found by Lucas and Verner was due to method of loading the model, of which no details are given by Woodruff. In the Shabanie model experiments the peak stresses occurred in various positions related to the rates of draw from the drawpoints.

It has been the experience on several mines that the larger the area under draw, the higher the stress on the grizzly level as indicated by the Janssen's formula above.

At Kimberley at the Bultfontein Mine the practice is to work a narrow, four-drawpoint-wide strip across the pipe, which is scanned backwards and forwards by closing one line of drawpoints at the rear and bringing in another at the front. This strip mining is done basically for draw control purposes, but has had beneficial effects on the stresses experienced on the bottom, which have decreased the damage previously experienced.

In Block 7AB first phase, severe damage was experienced in the low-draw rate areas. However, in the second phase and in Block 7/2 no severe damage was experienced, despite irregular rates of draw. In both these instances, however, it is probable that the improved drawpoint design has enabled the grizzly level to withstand these forces.

In practice an effort is made to maintain an even rate of draw over the block to minimise the ingress of dilution, which also has beneficial effects on the pressures on the grizzly horizon. But where there is a lot of coarse material in the draw columns leading to frequent hang-ups, with the concomitant point loads on the crown pillars, an even rate of draw is difficult to maintain. By producing from more drawpoints, the average rate of production per drawpoint can be brought down to a rate comparable to that achieved in the most frequently hung-up drawpoints. If the object is primarily to prevent excessive loading on the crown pillars, this measure may be self-defeating, as Janssen's formula shows that increasing the area under draw increases the average stress on the base.

To summarise, the grizzly horizon is subject to damage in the first instance by the abutment loads imposed in the undercutting stage, and to

blasting damage in the cone-cutting and overcutting stages. The effects of abutment loading have been reduced in block 7/2 by keeping the undercut well ahead of the overcutting and cutting the transverse drawpoint troughs (cone-cutting), that is by minimising the amount of rock removed in the abutment areas. An attempt to minimise blasting damage is made by pre-coning the drawpoints so that blasting charges can be kept light with void conditions in the cones. In production, the grizzly horizon is subject to varying loads imposed by the broken ground, which can be concentrated into point loads by large blocks of rock or hang-ups. The average stress imposed on the grizzly crown pillars can be minimised by keeping the area under draw as small as possible, maintaining an even rate of draw, and bringing down hang-ups as soon as possible. These are not always achieved in practice, and therefore the crown pillars must be designed for maximum strength and minimum stress, and supported adequately. The support policy and design is determined by the class of rock in the first instance, as, if it is considered that the ground is good enough for an adequately supported drive to last the life of the block, then the support work should be active, reinforcing the rock, whereas if the rock is poor, passive support which can be quickly and easily repaired should be considered.

The current ideas on crown pillar and drawpoint design are embodied in the block 7/2 design, shown in Fig 3.5. In this design, transverse troughs have been chosen so that lateral buttressing pillars are left between the drawpoints. The crown pillars were also kept as large as practicable for maximum strength and the sides kept vertical to minimise the chances of arching in the drawpoints which could impose a high lateral stress on the crown pillars. It is still too early to draw conclusions on this design, but it has been noted that in at least one drawpoint some considerable attrition of the crown pillar has taken place and the largest and most difficult to break-up hang-ups occur above the tops of the crown pillars.

#### 8.7 FORMS OF DAMAGE TO DEVELOPMENT OPENINGS

Four forms of damage to development openings have been recognised. The damage can in each form be attributed to stress changes accompanying a major mining operation, other than the advance of the development in itself. The four forms are:

- (a) High stress damage This form of damage occurs in high stress areas, such as in the abutments of an undercut where little or no stress reducing movements could occur on joints, because there were insufficient joints with suitable orientations and sufficiently low shear strengths. The shape and size of the development openings in this zone affects the ultimate stress concentrations and severity of damage. The severity of the damage is also greatly influenced by the rock competency. Higher stress concentrations are more probable in the smaller stress concentration zones, such as in the abutment of a narrow undercut, or beneath a narrow vee notch where there is a lower probability of suitably oriented joints for stress relief.

This damage takes the form of local movement on joints and spalling in a zone surrounding the development. The best examples of this type of damage were in the abutments of Block 7/2 on 345 level and on 350 level.

- (b) Reduced lateral stress damage Development done in a high lateral stressfield can be stable with considerable spans. Similarly complex horizontal layouts involving small pillars could also be stable. The creation of a large caved area in the vicinity could reduce these stabilising lateral forces and change the stressfield around the excavations, putting the back of wide excavations into tension and increasing the compressive stress in the sidewalls of the pillars and development.

Examples include the 3 shaft station layouts on 170 and 205 levels which experienced heavy damage when block 7A3 caved.

- (c) Secondary induced damage This is a common form damage to development. It occurs where shearing on a slip dislodges and forces off pieces of rock from the sidewall or back into an underground working.

It is considered that the movement would have occurred on the slip regardless of whether the development intersected it or not. If it was intersected by a drift and the ground in the proximity of the joint were competent, damage could be minimal, but if this ground were incompetent, the small movements could upset the stability of a marginally stable low competency zone. Given no extension to the major mining effort, the rate of movement would decrease with time and eventually stop, and the amount of damage induced would decrease similarly.

This form of damage can occur relatively remotely from the major excavation as the general stressfield need only be enough to cause the movement on a major joint which usually have a lower frictional resistance, while the stressfield may not be enough to cause damage to the drift alone.

An example of this form of damage was on 205 level where movement on Zone A caused several rock falls over a protracted period of time.

- (d) Gravity sliding damage A pre-requisite of this form of damage is a major, weak joint dipping towards the mining operation. If the shear strength of the joint is sufficiently low, and there is not sufficient lateral restraint, the superincumbent rock may start to slide towards the mining operation. Although the redistributed regional stress may have had a role in initiating this movement, it will continue to slide under its own weight, until there is an increase in lateral stress which would slow down and eventually bring this movement to a halt. This could occur some months after the cessation of drawing down dip.

Because movement is caused by gravitational forces, the duration of the force is unlimited, and the displacement can only be stopped by building up the lateral resistance by stopping the draw down dip. The amount and severity of the damage to development openings located in the moving mass is potentially unlimited. It is possible, however, for drifts in competent ground in the moving mass to survive some moderately large movements without severe damage, while others in less competent ground may be severely damaged by small displacements on the shear.

Examples of this type of damage occurred in Block 7AB on 245 and 255 levels at the start of undercutting of the second phase. The worst example was, however, in Blocks 4 and 5 where a 60m wide zone slid at least 100mm on a gouge-filled fault in the immediate footwall. Damage to workings above this zone was very severe, leading to continuous replacements of the T.H. yielding arches. Sliding continued for two months after the cessation of mining.

Recognition of the form of damage would help in deciding what type and amount of support work would be most effective. In general for high stress damage, active (rock reinforcing) support is considered to be the most effective. For reduced lateral stress damage active support may be effective, but where the lateral stress is so reduced that the steeply dipping joints are opened up, passive support techniques will be more effective.

If the rock affected by secondary induced damage has reinforcing potential, active support can be used provided that the reinforcing bolts do not cross the major joint. Support will not stop displacement on the major joint, it should therefore reinforce the rock on either side of the joint, while allowing the displacement on the joint to continue. If the damage is due to gravitational sliding, active support may be effective for a while but ultimately the disruption of the ground will be so bad that frequent replacement of passive support may be required.

CHAPTER 9SUMMARY AND CONCLUSIONS9.1 GENERAL

In this study of the ground behaviour associated with block and sub-level caving mining operations, the factor which has emerged as having the greatest effect on the caving process and the ground reaction to changes in stress induced by undercutting or caving, is the high probability of shearing on joints in the rock mass. In the periphery of a caved block shearing has been observed on the weaker, more continuous joints at intervals as close as three to six metres. The shearing displacements give the rock mass as a whole a plastic yielding property, which determines the course of caving of undercut areas, the stability of workings in the peripheries of caved areas and influences the selection of monitoring devices and analysis techniques.

The relatively high lateral stress field is also an important factor determining the response of the rock to undercutting or caving. The effect of these two factors have been left in almost all sections of this investigation. The conclusions drawn from each facet of the investigation are discussed separately below.

9.2 GROUND BEHAVIOUR CHARACTERISTICS

The behaviour of the rock mass may be described as a quasi-Bingham substance, represented by a spring, friction block and dashpot in series. Under stress, the rock mass deforms elastically, and with relative shear movements on the suitably orientated joints, non-recoverable deformation occurs in the rock mass. Observations of shear displacement indicate that shearing occurs first on those suitably orientated structures with the lowest shear strengths. At higher stress deviations, more joints with higher shear strengths are affected. The spacing of joints on which shearing occurs varies, but is generally in the range of 3 to 30m. In general, the shearing occurs at a slow rate, and the magnitudes of the shear displacements are dependent on the magnitudes of the stress deviations and the duration of the stresses. In the periphery of Block 7AB

the observed shear displacements accounted for 25% of the total measured deformation.

It is concluded that these movements could significantly affect the stress distribution in the rock around a block cave undercut or in the peripheries of a caved zone.

### 9.3 VIRGIN ROCK STRESSES

The rock stress measurements made at six sites on levels ranging in depth from 200 to 400m indicated that the lateral stresses exceed the vertical stresses. These higher lateral stresses are considered to be residual tectonic stresses from the period of lateral compression which resulted in the folding and faulting of the ultrabasic sill. Regression lines of the vertical and lateral stress components of the six results indicate that the measured vertical stresses are slightly higher than those expected from the rock density, that the lateral stress components parallel to the strike of the ultrabasic sill increases gradually with depth and at right angles to the strike the lateral shears component increases significantly with depth. The correlation coefficients for these regression analyses were poor and therefore further measurements should be made at greater depths to check these relationships.

It was noted that a marked variation in principal stress orientations and magnitudes existed between successive measurements made in each borehole. This variation is attributed to the proximity of major joints such as slips or fibre seams, and to the differences in the modulus of elasticity of the serpentine and dunite. It is concluded, therefore, that several measurements had to be conducted in each borehole to obtain a meaningful average.

There is a broad agreement in the orientation of the major principal stress at right angles to the strike of the sill, as deduced by Laubscher, for the tectonic stresses which resulted in the folding and faulting of the sill, and the generally higher lateral stress component in this direction indicated by these measurements. However, a detailed study of the geological structures at each site failed to reveal any relation between the structures and the measured principal stress orientations. It is concluded therefore that no useful guide to present-day stress orientation could be obtained from an analysis of structural geological features.

#### 9.4 INSTRUMENTATION

In ground which is cut by numerous joints and behaves as a quasi-Bingham material, the techniques of instruments which have proved most successful for monitoring deformation and displacement are those which extend over several metres such as the survey traverses, remote displacement meters and both forms of wire extensometers, which monitor several joints. The instruments which monitor displacement or deformation over shorter distances have proved to be useful for monitoring relative displacements on specific joints, but, in monitoring deformation generally, there are too many local variations in ground behaviour to derive at any meaningful absolute results from a few instruments. Therefore enough instruments must be installed to obtain sufficient results which have statistical significance. The same difficulties are encountered with rock stress monitoring devices such as the photo-elastic stressmeter, or, support load monitoring devices such as the photo-elastic bolt tension meter or arch load indicator. Individual instrument results should be analysed by reference to the previous record of the instrument, and instrument comparisons should be made on a qualitative and time basis. On this basis some predictions on the date of caving or duration and severity of further damage can be made.

While satisfactory deformation monitoring techniques have been developed, considerable further work requires to be done on support monitoring techniques especially for grizzly horizons where instruments would be subject to heavy secondary blasting damage and to a humid and corrosive atmosphere. Also further work is required to develop suitable techniques for monitoring the loads imposed on the crown pillars by the broken ground which have not been monitored satisfactorily to date.

#### 9.5 GEOMECHANICS CLASSIFICATION

The geomechanics rock classification system has been adopted for descriptive purposes by production and service department personnel. In this dissertation it has been used largely for this purpose. With the application of assessed adjustments to account for potential weathering, in situ stress and changes in stress, joint orientation and blasting effects, the system could be applied to designing the support requirements for an underground excavation, and a support guide based on the in situ and adjusted classification ratings has been proposed by Laubscher (1975).

Similarly, the classification system has potential application in assessing the cavability of a block and the drawpoint rock size. At the present stage of development, the guide to cavability is still too crude to provide anything but a first guide to the potential for caving a block, as the rock classification is not sensitive enough. Further work is required to formulate rules and guidelines for determining the adjustments to be made, and further work will also have to be done on improving the classification sensitivity; there may be some advantage in changing the relative importance ratings of the various parameters, which are currently assumed to have universal application to problems ranging from tunnel stability to pit slope angles. There also should be an investigation into the possible advantages in defining the rock class by three ratings instead of one, as proposed by Barton et al (1974), which define the three main rock parameters vis: the block size, the interblock shear strength and the active stresses (in the soil mechanics sense). Further work should also be done on the application of the system to problems on other mines.

#### 9.6 CAVING

Two forms of caving have been recognised:- mass subsidence caving and stress caving; the former occurs where the lateral stress has been sufficiently reduced to allow shearing on the steeply dipping joints. The lateral stress reduction may be due to an adjacent previously caved block or a boundary isolation pit, and the caving takes the form of the orderly subsidence of large columnar blocks bounded by steeply dipping slips; the bulking factor is low and rate of propagation of caving to surface very rapid. The latter form of caving is characterised by cavities beneath the back and the caving action is largely a sloughing from a relatively narrow zone of instability in the back. The stability of this zone is determined by the presence of suitably orientated joints with respect to the stresses in the back. These stresses are considered to be compressive due to the regional lateral stresses, and also because the back develops into an arch or dome shape in which compressive stresses are expected. However, tensile stresses in the vertical direction over the centre of the undercut may contribute to the instability of this zone. The propagation of the cave back to surface is sporadic and usually slow.

The factors which determine the cavability of the ground are the competency of the rock mass; the orientation and shear strength of the

joints in the back relative to the stresses in the back. In general, the lateral stresses tend to inhibit caving, but there remains a zone in the back which may become unstable and cave with even small changes in lateral stress such as those brought about by mining of an adjacent block. With the lateral restraint reduced over a substantial area of the back, the caving will take the form of mass subsidence caving.

The geomechanics classification with adjustments can provide a first guide to the potential cavability of the ground. As indicated above, the system requires some improvements in sensitivity and the rules for adjustments defined. However, even with these it is still considered that in borderline cases a more sophisticated approach is required. For this the most promising technique is a finite element analysis which used an elastic-plastic yield condition and can model the heterogeneity and anisotropy of the rock in strength and deformation, and can cater for the path dependency of the problem. The path dependency arises from both the mining sequence and permanent deformation. For this technique several problems have to be solved, which include establishing the correct constitutive equation and the determination of the material properties, applied loads and in situ stresses to the required accuracy to obtain reliable predictions (Pariseau 1970).

#### 9.7 EFFECTS OF CAVING ON SURROUNDING ROCK

The periphery of a caved zone is subject to a reduction in the lateral stressfield in one direction, and an increase in the other direction parallel to the periphery. This results in the elastic deformation of the rock mass, plus permanent deformation resulting from shearing on joints under suitable stress conditions. On the surface, the angle of subsidence as defined by the limits of the subsidence zone related to the underground mining varies between  $78^\circ$  and an overhanging  $60^\circ$ , and these are determined by the orientation of the joints. The minimum angle of break, measured to the last surface cracks from the limit of mining, was  $70^\circ$ .

Four forms of damage to development openings have been recognised; high stress damage noted in a few areas such as ahead of an undercut face, reduced lateral stress damage occurring in the peripheries of a caved area where the reduction in the lateral stress due to caving reduces the stability of wide openings, secondary induced damage localised to the

vicinity of a major shear and caused by displacements on that shear, and gravity sliding damage caused to workings located within a block of ore overlying a weak shear on which it is sliding towards the mining operation.

After the appropriate adjustments have been made to the geomechanics classification, Laubscher's support guide could be used to determine the support requirements of workings subject to high stress damage or reduced lateral stress damage. With respect to secondary induced damage, the adjusted classification may help decide the support requirements, but not with any certainty as the irregularities may induce further damage as displacement continues. Situations where gravitational sliding is likely to occur are best recognised beforehand and measures taken to avoid the sliding occurring by mining sequence modifications or leaving buttressing pillars if such is warranted.

Beneath a cave block the crown pillars and grizzly horizon are subject to vertical loading due to the remaining weight of caved material not carried by the sides, and due to high point loads imposed by arching in the broken ground. In general, the pressure on crown pillars is related to by the rate of draw, the drawpoint spacing and frequency of hangups. In the design of the grizzly elevation, the layout should provide as much lateral support to the crown pillar as possible. The drawpoint spacing is usually determined by experience with the fragmentation encountered with the ore. The widest drawpoint spacing consistent with an adequate ore recovery will give the lowest overall stress on the grizzly horizon for a given rate of draw.

## APPENDIX 1

PHYSICAL PROPERTIES OF TALC CARBONATE ROCK AND  
TALC SCHIST FROM THE FOOTWALL AND TALC ZONES

TALC CARBONATE ROCK - CSIR TESTS				
NO.	Density t/m <sup>3</sup>	Compressive strength $\sigma_c$ MPa	Young's Modulus E GPa	Remarks
1023 1	2,752	234,4		Silicified
2	2,699	265,6		"
3	2,907	53,0		
4	2,736	68,3		
5	2,930	74,4		
6	2,954	124,1	49,6	Carbonated
7	2,712	74,2		
8	2,917	56,4		
9	2,728	324,0		Silicified
10	2,915	40,5		
Mean	2,83	131,5	49,6	
Std Dev	0,11	103,4		
10 311	2,775	66,1		
12	2,899	45,2		
13	2,959	70,0		
14	2,955	56,0		
15	2,847	70,2		
16	2,952	85,2		
17	2,868	59,1		
18	2,904	80,7		
19	2,860	75,3		
20	2,858	65,3	51,7	
Mean	2,91	67,3	51,7	
Std Dev	0,017	11,9		

## APPENDIX 1 CONT.

PHYSICAL PROPERTIES OF TALC CARBONATE ROCK AND  
TALC SCHIST FROM THE FOOT-WALL AND TALC ZONES

TALC CARBONATE ROCK - SHABANIE MINE TESTS			
No.	$\sigma_t$ MPa	$\sigma_c$ MPa	REMARKS
U 412 64		68,2	Mildly schistose
72		27,9	" " *
73		55,2	" " *
60		44,2	" " *
81		24,1	" " *
84		8,0	Fine grained schist *
86		27,9	Mildly schistose *
86		41,7	" " *
86		23,3	" " *
86		42,6	" " *
102		49,8	Fairly coarse grained *
102		8,0	" " " *
102		32,3	Mildly schistose
102		15,7	" " *
103		33,5	" " *
103		24,0	" " *
103		17,7	" " *
105		16,0	" " *
107		21,7	" " *
108		8,0	" " *
SRM 126	204,5	6,0	29,2
	208	7,1	34,8
	207,5	4,5	28,6
Mean	5,9	29,6	
Std Deviation	1,3	15,4	

NOTE

Compression tests done on manually-operated model 420 Farnell Compression test machine using spherical seat platens. Specimens soaked in water for one week. Tensile strengths determined by Brazillian test.

\* Failure on plane of weakness

## APPENDIX 1 CONT.

PHYSICAL PROPERTIES OF TALC CARBONATE ROCK AND  
TALC SCHIST FROM THE FOOTWALL AND TALC ZONES

TALC CARBONATE ROCK - SHABINIE MINE TESTS				
No.		$\sigma_t$ MPa	$\sigma_c$ MPa	REMARKS
SRM 125	40m		17,6	Coarse grained
	52,4		70,0	Medium grained
	48,5		49,0	" "
	23,5		18,7	" "
	24,0		12,9	" "
	22,0		16,7	" "
	23,0		21,0	" "
	13,2		36,2	Medium grained-
	1,9		25,7	soft talc
	13,6		25,7	" "
	14,3		26,9	" "
	10,5		56,0	" "
	4,9		11,6	" "
	15,8		25,7	" "
	2,6		107,5	Carbonated
SRM 126	220,5	18,2	173,0	"
	200,0	9,7	37,3	"
	197,0	10,1	58,1	"
Mean		12,7	44,0	
Std deviation		4,6	40,3	

NOTE

Compression tests done on manually-operated model 420 Farnell Compression test machine using spherical seat platt ns. Specimens soaked in water for one week. Tensile strengths determined by brazilian test.

## APPENDIX 2

PHYSICAL PROPERTIES OF CARBONATED SERPENTINE  
FROM THE BRITTLE FAILURE ZONE

SOLE AND DEPTH	TENSILE STRENGTH $\sigma_t$ MPa	COMPRESSIVE STRENGTH $\sigma_c$ MPa	REMARKS
SRM 126 212,0 m	10,1	218,5	
177,5	10,4	156,5	
226	9,3	176,3	
218	10,9	269,0	
224	12,5	102,8	
SRM 412 90 m		264,0	* Almost barren
90 m		367,0	
90 m		576	
92 m		279,6	*
93 m		323,5	
93 m		235,0	*
96 m		469,0	
96 m		524,0	
97 m		176,6	*
91 m		162,0	*
Mean	10,64	288,3	
Std Deviation	1,19	141,0	
* Failure on plane of weakness			

APPENDIX 3

PHYSICAL PROPERTIES OF PARTIALLY SERPENTINISED  
DUNITE FROM FIBRE BODY AND HANGINGWALL

CSIR TEST RESULTS				
HOLE NO. X DEPTH	YOUNG'S MODULUS E GPa	POISSON'S RATIO	$\sigma$ MPa	REMARKS
LD 781	73,8	0,296	149,9	Partly serpen- tinised dunite
DD 781	73,8	0,333	161,3	
DD 780	93,7	0,306	166,6	
DD 780	90,3	0,213	164,1	
DD 779	73,8	0,267	159,3	
DD 779	74,5	0,261	169,6	
DD 804	51,4	0,27	105,5	Partly serpen- tinised dunite
SRM 2	43,6	0,29	144,1	
SRM 16	55,4	0,32	135,8	
SRM 17	81,7	0,31	153,1	
SRM 18	74,1	0,31	159,8	
SRM 33	67,6	0,29	106,9	
SRM 34	79,3	0,31	169,6	
SRM 47	87,6	0,27	177,2	
SRM 46	84,8	0,26	146,2	
Mean	73,7	0,29	151,4	
STD Deviation	14,3	0,03	21,0	

## APPENDIX 3 CONT.

PRACTICAL PROPERTIES OF PARTIALLY SERPENTINISED  
DUNITE FROM FIBRE BODY AND HANGINGWALL

SHABANIE MINE TEST RESULTS			
HOLE NO. X DEPTH	COMPRESSIVE STRENGTH $\sigma_c$ MPa	TENSILE STRENGTH $\sigma_t$ MPa	REMARKS
SER: 126 136,5	86,5		HW Dunite Coarse Grained
136,5	87,9	5,9	
247,5	103,0	4,5	
121,5	95,9	6,0	
172,5	95,9	13,4	
103,5	83,5	6,7	
75,0 <sub>1</sub>	85,7	-	Med. Grained HW Dunite
75,0 <sub>2</sub>	87,9	9,5	
85,5	87,2	-	
64,5	90,5	4,4	Partially Weathered Dunite
54,0	72,6	1,0	Partially Weathered Dunite
37,0	40,6	-	Mod. Weathered Dunite
46,5	36,6	-	NXM
42,0	139,5	7,8	
26,0	65,1	-	
13,0	65,1	-	Sirberite
Mean	83,3	6,7	
Std Deviation	24,5	3,3	

GLOSSARY OF LOCAL TERMS

- Airblast** The escape of a large volume of air at very high velocities through narrow exits due to the collapse of a large undercut area.
- Aplite** In the serpentinites, a hybrid rock resulting from reaction between a silica rich solution and silica deficient serpentine, resulting in grained orthopyroxene and feldspar rocks with margins of serpentinite and iddingsite. Quartz, biotite and muscovite are present in some localities.
- Back** The roof or hanging of a drift or a stope. (See also cave back)
- Block** An orebody or a portion of an orebody.
- bolt Tension Meter** A device attached to rockbolts used for measuring the stress on the bolt. (See 4.61)
- borehole Clamp** A device which is clamped into a borehole to which a wire is attached for the purpose of measuring the movement of the ground in which the clamp is located.
- Brittle Fibre** Springy chrysotile fibre which may be broken after less than four flexes.
- Cave Back** The under surface of the solid rock overlying the caved material in a block cave mining operation.
- Caving** A method of mining in which the orebody or hangingwall is undercut and required to collapse or fall in, in at least sufficient quantity to allow the extraction of the ore to proceed efficiently and safely.
- Caving Indicator** A device for locating the back of a cave consisting of a centrally pivoted weight lowered on a wire down a borehole. (See 4.63)
- Closuremeter** A mechanical device for measuring the distance between two pins set either in the sidewalls or in the hangingwall and footwall of development. (See 4.51)

Crashing	The procedure of longhole drilling and blasting.
Crown Pillar	The solid rock left overlying a grizzly drift beneath the overcut and between the drawpoint cones or troughs.
Development	Used in the sense of Nelson's Development Drivages the shafts, tunnels, crosscuts, raises and winzes to prove or render accessible the ore to be extracted (Nelson, 1964).
Drawpoint	A heading which intersects caved ore and is used for removal of ore by gravity flow. Ore coming from a drawpoint may pass directly to a box on a haulage via a grizzly or may be mechanically loaded.
Drawpoint Cone	The cone or transverse trough cut below ore pre-broken ore to which a drawpoint is linked.
Drawpoint Trough	A trough parallel to a grizzly drift cut below caved or prebroken ore to which several drawpoints may be linked the length of the trough.
Extensometer Points	An arrangement of small stainless steel discs used for measurement of strain or displacement by means of a portable mechanical strain gauge.
Fibre Seam	A layer of chrysotile asbestos fibres closely packed parallel to each other and at almost right angles to the plane of the layer or seam.
Grizzly	A horizontal bar placed across an orepass to prevent oversize particles from entering the orepass.
Grizzly Drift	A drift with drawpoints located on one or both sides and equipped with grizzlies and orepasses for the purpose of drawing off prebroken or caved ore.
Hand Cobbing	The removal of large fibre seams from waste rock by hand.
Harsh Fibre	Springy chrysotile fibre which may be broken after between four and twelve flexes.
Hydraulic Radius	The area divided by the perimeter; a convenient way of relating size and shape and used as a factor in determining the caving potential of a mining area.

Joint	For the purpose of rock classification any joint, slip, fault, bedding plane or weak fibre seam continuous from one joint to another or at least 3m in length. (See 4.2)
Load Cell	Devices used to measure the loads imposed on the Block 7AB crown pillars by the ore. (See 4.64)
Main Levels	Haulage levels 30m or 90m intervals linked to the main shafts.
Orebody	An economic deposit of chrysotile asbestos, defined by an economic cut off (usually in the hangingwall) and/or by the brittle/silky contact (usually in the footwall). (See 2.43)
Overcut Level	The level overlying the grizzly level and below the undercut level, used for overcutting the drawpoints.
Photo-elastic Disc	A clear plastic disc which stuck on the flattened end of a borehole and overcored can be used as a strain gauge to measure rock stress. (See 4.66)
Picrolite	A non-fibrous variety of serpentine, apple green in colour, and usually occurring in slips, it is usually columnar, or it may be platy. (See 2.43)
Pre-Coning	The cutting of drawpoint cones or troughs before overcutting the drawpoints.
Presplit	A line of holes charged and blasted before the approach of the major mining effort to limit explosive damage or to create an artificial line of weakness.
Remote Displacement Meter	A device for measuring subsidence constructed from a water filled hose fitted with a glass tube at one end. (See 4.41)
Retreat	The working by longhole drilling and blasting towards the points of access. The direction in which this activity progresses.
Rib Pillar	The large relatively narrow and high strip of uncaved ground left between two caved blocks, usually left uncaved because of the presence of a talc zone or dyke.

Ring	The fan of holes drilled from a drive or crosscut to be charged and blasted for the purpose of either sublevel caving, shrinking an orebody or undercutting a block.
Ring Drilling Drive	The drive or crosscut in which ring drilling is done.
Serpentine	The process of alteration from a dunite to a serpentine involving the addition of water and silica and the removal of magnesium.
Shrink	The breaking of ore by longhole blasting withdrawing only enough ore to have room for the expansion that occurs when the next portion of ore is blasted.
Silky Fibre	Soft, flexible chrysotile fibre with a high tensile strength.
Slips	Minor faults of small, frequently indeterminate throw. (See 2.41)
Slot	Vertical excavation at the end of a ring drilling drive to provide room for expansion in longhole blasting and a fall to break to.
Spalling	The splitting of rock parallel to the surface of a drift due to either weathering or to stress.
Stressmeters	A rigid device for measuring the stress in rock. (See 4.62)
Sublevels	Levels developed between the main levels for whatever purpose, not necessarily linked to the main shafts.
Sublevel Caving	A mining method involving the drilling and blasting of ore and the extraction of that ore by lashing in a series of parallel drifts off set with respect to each other on each sublevel.
Talc-Carbonate Rock	Talc-magnesite rock, consisting of talc and magnesite.
Talc Schist	A schist in which talc, associated with carbonates and occasionally quartz, is the dominant schistose material.
Undercut	The level(s) on which a cave block is undercut. The undercut level(s) overlying the overcut and grizzly levels.

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