A PRACTICAL MANUAL ON BLOCK CAVING

by Dr. Dennis Laubscher

for the International Caving Study (1997-2000)

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DISCLAIMER

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The International Block Caving Study (ICS) was conducted by the Julius Kruttschnitt Mineral Research Centre (Brisbane, Australia) and the Itasca Consulting Group (USA) from November 1997 to November 2000 and was sponsored by Northparkes Mines, De Beers, Rio Tinto, PT Freeport Indonesia, Noranda Inc., TVX Gold, CODELCO and Newcrest Mining.

Dr Dennis Laubscher was subcontracted by the ICS to collate his international experiences in cave mining. Dr Laubscher is a recognised International Mining Consultant particularly in mass mining methods.

The Block Cave Manual has been built mainly around and/or influenced by the experiences of Dr Laubscher in countries such as Zimbabwe, South Africa, Chile, North America and the Philippines. Case histories from caving operations such as Shabani and Gaths, Premier Mines, Codelco Divisions (El Teniente, Andina and Salvador), Henderson, and Bell Canada are presented in the manual as well as comments and input from a number of individuals such as Nick Bell, Jarek Jakubec and Glen Heslop who were subcontracted to the task by Laubscher. It should however be noted that the bulk of Laubscher's recorded experience represents and reflects the work conducted by the above mentioned mines.

The Block cave manual effectively:

1. provides a check list of key topics that should be considered and addressed during the design process;
2. provides definitions and basic design rules in each of the key areas, and
3. provides case histories to support a number of the design rules.

Although some minor editing has been done, the document is still to be professionally edited for content and technical substance by the ICS. The document has only been released to the sponsors of the ICS project for reference and, where possible, for comment and feedback. The ICS is at this stage unable to either fully substantiate or endorse a number of the issues discussed in this document.

A final document will be released once the manual has been professionally edited. Some of the material will be included in the ICS Handbook on Caving to be released during 2001. This handbook on block caving mechanics will be compiled and edited by Professor Brown of the University of Queensland.

Gideon Chitombo
Manager, Mining
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(on behalf of ICS)
24 October 2000

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T R Stacey and W D Ortlepp
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FOREWORD

A Practical Manual on Block Caving

This Cave manual is designed to cover most aspects of a block / panel caving operation. The emphasis is on the practical and management aspects to make a massive caving operation successful. Whilst this is not an academic exercise, there are certain academic aspects that have to be addressed, but the coarseness of the operation means that empirical results and interpretations are of great importance. There is a need to make comparisons with mathematical modelling, but no major decisions should be made on modelling results on their own.

The format of the cave manual is such that it provides a design platform which should be used during the exploration of any massive orebody. A massive orebody can be described as any orebody where in plan the RMR divided by the hydraulic radius exceeds 1.5 and has a draw height greater than 50m. The headings of each sub-section of a section form a check list for planning purposes and there is provision for an **assessment** to be made at the end of each section, the assessment can be regarded as a risk analysis throughout the feasibility study. **This is a very important aspect, because, lack of data or frailties in the interpretation must be identified at as early a stage as possible and not left to the final stages**.

Caving is the lowest cost underground mining method, provided that drawpoint spacing, drawpoint size and ore handling facilities are designed to suit the caved material and that the drawpoint horizon can be maintained for the draw life. In the near future, several open pit mines that produce in excess of 50 000 tons per day, will have to examine the feasibility of converting to low cost, large scale underground operations. Several other large scale, low grade underground operations will experience major changes in their mining environments as large dropdowns to deeper extraction horizons are implemented.

These changes demand a more realistic approach to mine planning than has been the case in the past, where existing operations have been projected to increased depths with little consideration of the change in mining environment which might occur. As economics force the consideration of underground mining of large, competent orebodies by low cost methods, the role of cave mining will have to be re-defined. In the past caving has generally only been considered for rock masses that cave and fragment readily.

The ability to define cavability and fragmentation, the availability of large, robust LHD's, a better understanding of draw control requirements, improved drilling equipment for secondary blasting and reliable cost data have shown that competent orebodies with coarse fragmentation can be exploited by cave mining at a much lower cost than with drill and blast methods.

Cave mining refers to all mining operations where the orebody has caved naturally after undercutting and the caved material is recovered through drawpoints. This includes block caving, panel caving, inclined drawpoint caving and front caving (retreating brow cave). Caving is the lowest cost underground mining method provided the drawpoint size and handling facilities are tailored to suit the caved material and the extraction horizon can be maintained for the life of the draw.

During the early 1990’s, daily production from cave mining operations throughout the world is approximately 450 000 tons per day with the following breakdown from different layouts:-

Grizzly	90 000
Slusher	35 000
LHD	325 000
Total	450 000

By comparison South African gold mines produce 350 000 tons per day.

In the near future several mines that currently produce in excess of 50 000 tons per day from open pit mines will have to examine the feasibility of implementing low cost, large scale underground mining methods. Several cave mines that produce high tonnages from underground, are planning to implement dropdowns of 200 metres or more. This will result in a considerable change in their mining environments. These changes will necessitate detailed mine planning rather than simply projecting current mining methods to greater depths.

As more attention is directed to the mining of large, competent orebodies by low cost underground methods, it is necessary to define the role of cave mining. In the past caving has been considered for rock masses that cave and fragment readily. The ability to better assess the cavability and fragmentation of orebodies, the availability of robust LHD's, an understanding of the draw control process, suitable equipment for secondary drilling and blasting and reliable cost data, have shown that competent orebodies, with coarse fragmentation, can be cave mined at a much lower cost than with drill and blast methods. However, once a cave layout has been developed there is little scope to make changes. Aspects that have to be addressed are cavability, fragmentation, draw patterns for different ore types, drawpoint or drawzone spacings, layout design, undercutting sequence and support design. It is common to find that old established mines, which have developed standards during the course of successfully mining the easy tonnage in the upper levels of the orebody, have a resistance to change and do not adjust to the ground control problems which occur as mining proceeds to greater depths, or to the rock type changes.

Mines that have experienced continuous problems are more amenable to adopting new techniques to cope with a changing mining situation. Detailed knowledge of local and regional structural geology, use of an accepted rock mass classification to characterise the rock mass and knowledge of the regional and induced stress environment are prerequisites for good mine planning. It is encouraging to note that these aspects are receiving more and more attention.

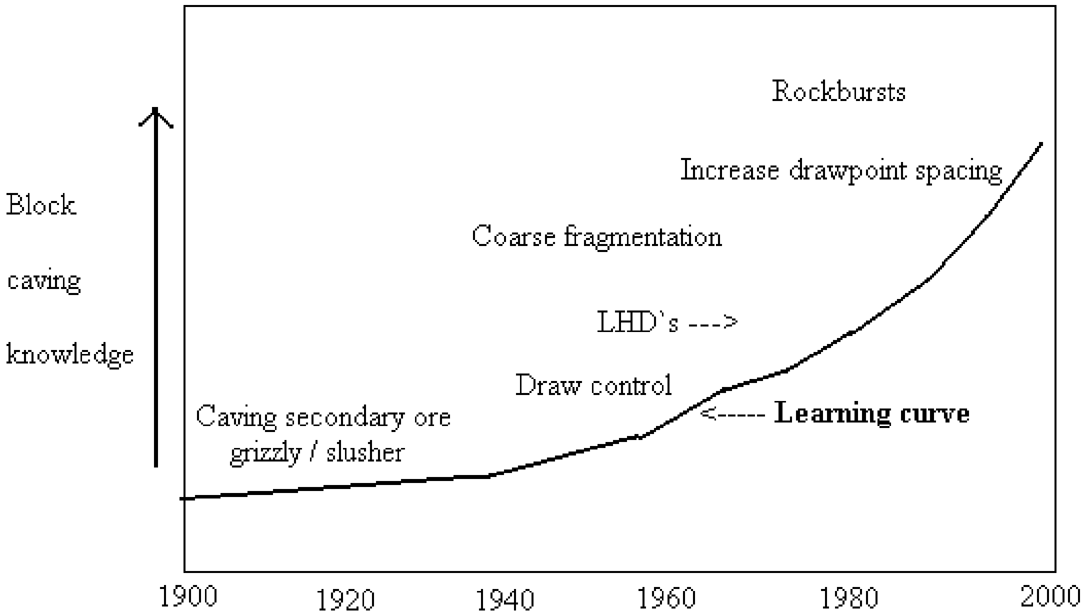


Table 1.0

There is still much to be learnt or discovered in block caving as is shown in table 1.0 above. There is a lot of information available on operating mines, but, the analysis of this data needs to be improved and shared.

Whilst any sound classification system needs to be employed the `Laubscher' rock mass classification system will provide both the rock mass ratings and the rock mass strength as needed for the design of cave mining operations.. The in situ rock mass ratings (IRMR) define the geological environment and the adjusted or mining rock mass ratings (MRMR) consider the effect that the mining operation has on the rock mass. The ratings, details of the mining environment and the way in which this affects the rock mass and the geological interpretation, are used to define:

> Cavability, subsidence angles, failure zones, fragmentation, undercut face shape, cave front orientation, undercutting sequence, overall mining sequence, support design

The MRMR system is the means of communication between planning and production personnel. A classification system must be understood by all involved in the mining operation. It is only by concentrating on one system that this is achieved. There are examples of large mines playing around with different systems and not really achieving at the end of the day. Because of the size of block caving orebodies, there is often a variation in rock mass properties within the orebodies. We have examples where the RMR varies from 30 to 70. It is thus important to define the orebody in zones of similar characteristics.

Table 2.0, at the end of the text summarises the interactive factors that affect a caving operation

Planning a block caving operation is a complex exercise particularly if the orebody has a limited area and there might be caving problems. It is essential that all data is properly plotted and presented on both long and cross sections.

The chance of success is a function of the commitment to laid down standards.

It will be noted throughout this manual that what can be defined as good mining practice is in many cases, not implemented and often done with little effort. The reasons being that it is not convenient or owing to poor planning, the short term targets cannot be met and production calls are allowed to overrule correct long term procedure. Support is often poorly installed even though the correct ingredients are used.

Mathematical modelling has been looked at as the possible solution for solving some of the caving design parameters instead of the empirical approach that is now adopted. There is reference throughout the manual where it is considered that mathematical modelling could be of assistance, however, results to date are not that encouraging. It is for this reason that a section titled ‘ Role of mathematical modelling in block caving design’ is included. The important consideration is the degree of accuracy that can be obtained with the empirical approach and can modelling improve on this. After all the accuracy of the input data required for modelling might not be achieved or the modelling program or computers do not have the capability to model the number of drawpoints in 3-D. Empirical results can be accurate to ± 15%.

Table 2.0

Parameters to be considered before the implementation of cave mining		
Cavability	**Primary Fragmentation**	**Drawpoint/Drawzone Spacing**
Rockmass strength (RMR / MRMR) Rockmass structure-condition geometry In situ stress Induced stress Hydraulic radius of orebody Water	Rockmass strength (RMR /MRMR) Geological structures Joint/fracture spacing & geometry Joint condition ratings Stress or subsidence caving Induced stress	Fragmentation of ore and overlying rock Overburden load and direction Friction angles of caved particles Practical excavation size Stability of host rockmass (MRMR) Induced stress
Draw Heights	**Layout.**	**Rockburst Potential**
Capital Orebody geometry Excavation stability Effect on ore minerals Method of draw	Fragmentation Drawpoint spacing and size Method of draw - gravity or LHD Orientation - structures / joints Ventilation, ore handling, drainage	Regional and induced stresses Variations in rockmass strength/modulus Structures Mining sequence
Sequence	**Undercutting sequence (pre/advance/post)**	**Induced Cave Stresses**
Cavability - poor to good or vice versa Orebody geometry Induced stresses Geological environment Rockburst potential Production requirements Influence on adjacent operations Water inflow	Regional stresses Rockmass strength Rockburst potential Rate of advance Ore requirements Completeness of undercut Shape - lead /lag Height of undercut	Regional stresses Area of undercut Shape of undercut Rate of undercutting Rate of draw
Drilling And Blasting	**Development**	**Excavation Stability**
Rockmass strength Rockmass stability (drillhole closure) Required fragmentation Hole diam., lengths, rigs Patterns and directions Powder factor Swell relief	Layout Sequence Production Drilling and blasting	Rockmass strength (RMR / MRMR) Orientation of structures and joints Regional and induced stresses Rockburst potential Excavation size - orientation and shape Draw point Mining sequence
Support	**Practical excavation size**	**Method of draw**
Excavation stability Rockburst potential Brow stability Timing of support - initial, secondary and production	Excavation stability Induced stress Caving stresses Secondary blasting Equipment size	Fragmentation Practical drawpoint spacing Practical size of excavation Gravity or mechanical loading
Rate of draw	**Drawpoint interaction**	**Draw column stresses**
Fragmentation Method of draw Percentage hangups Secondary breaking / blasting Seismic events Air blasts - drawpoint cover	Drawzone spacing Critical distance across major apex Fragmentation Time frame of working drawpoints	Draw-column height Fragmentation Homogeneity of ore fragmentation Draw control Draw-height interaction Height-to-short axis base ratio Direction of draw
Secondary	**Secondary blasting/breaking**	**Dilution**

fragmentation		
Rock-block shape Draw height Draw rate-time-dependent failure Rock block workability - rock block strength Range in fragmentation size - fines cushioning Draw-control program	Secondary fragmentation Draw method Drawpoint size Gravity grizzly aperture Size of equipment and grizzly spacing Ore handling system - size restrictions	Orebody geometry Mining geometry Fragmentation size distribution Fragmentation range of unpay ore and waste Grade distribution of pay and unpay ore Mineral distribution in ore Drawpoint interaction Secondary breaking Draw control - techniques / predictions Draw markers
Tonnage drawn	**Support repair**	**Ore/grade extraction**
Level interval Shut-off grade Drawpoint spacing Dilution percentage Controls Redistribution	Tonnage drawn Point and column loading Brow wear Floor repair Secondary blasting	Mineral distribution Method of draw Rate of draw Dilution percentage Cut-off grade to Plant Ore losses
	Subsidence	
RMR / MRMR Height of caved column	Minimum and maximum spans Major geological structures	Depth of mining Topography

DESIGN TOPICS

Geological Investigations

GENERAL

The geological investigation provides the regional picture, includes plans and both small and large scale cross and longitudinal sections. The large scale to include surface. The object is to gather data which will be used for mine design. The Geologist / Technician must always consider the end result, this means that the presentation must be relevant to operation. Defining zones of different structural density / pattern is equally as important as lithological changes. Rock mass classification data is collected at this stage. For mining situations the IRMR / MRMR system has proved to be suitable and is described in the MRMR section. The hangingwall is the zone above the orebody and should include all the rock types within the subsidence zone, defined from the lowest point of the orebody. Side dilution will come from this zone. This is the zone which could cave and therefore will not contain installations.

All relevant geological data in the peripheral zone must be plotted on plans and sections and must cover the area surrounding the orebody below the subsidence zone, but will include the failure zones.. Permanent infrastructure will be sited in the peripheral zone beyond the defined failure zone.

ROCK TYPES

This is a detailed description of the rock types in the orebody and peripheral zones with full details of their properties, particularly with respect to the strength of the rock mass and the weathering potential of the different rock types. Includes density. Variations in modulus often result in failure of the competent zones due to stress release. This failure might be violent - strain bursting - as seen in aplite dykes 100m below surface or fracturing of dykes in a talc host rock. It is important that the descriptions are kept simple with emphasis on the mechanical properties.

There should be a detailed description of the rock types in the hangingwall with full details of their properties, particularly with respect to the strength of the rock mass and the weathering potential of the different rock types. As the investigation is concerned with the total propagation of the cave this description must be to surface. Includes density.

INTRUSIVES

Full information on location, strike, dip, size and properties of all intrusives is required. Highlight any characteristics that are different from the host rock types. Intrusives should have their own RMR. Are the contacts sheared or ‘frozen’. The hangingwall might contain sills which could inhibit the propagation of the cave.

MINERALISATION, MINERAL AND GRADE DISTRIBUTION

Does the ore mineralisation have a bearing on the strength of the rock mass in the form of veins? Do the veins have continuity and can they be classed as joints or are they fractures? Is the mineral disseminated? Are the veins weak so that the mineral reports in the fines? Is the mineral in weak zones that will form fines? Is the ore contact sharp or is it gradational? How extensive is the mineralised zone in the peripheral rocks? The grade distribution in the orebody is very important and could be random or in zones. If zoned, then this could influence where mining would start and the subsequent sequence. The block model will show grades for individual blocks, zoning has to be interpreted.

Is the hangingwall mineralised and does the mineralisation have a bearing on the strength of the rock mass in the form of veins? Do the veins have continuity and be classed as joints or are they fractures. Is the mineral disseminated. Are the veins weak so that the mineral reports in the fines. Is the mineral in weak zones that will form fines. This is important as the fines flow faster than coarse rock and therefore the mineral in the dilution zone fines could up-grade the ore.

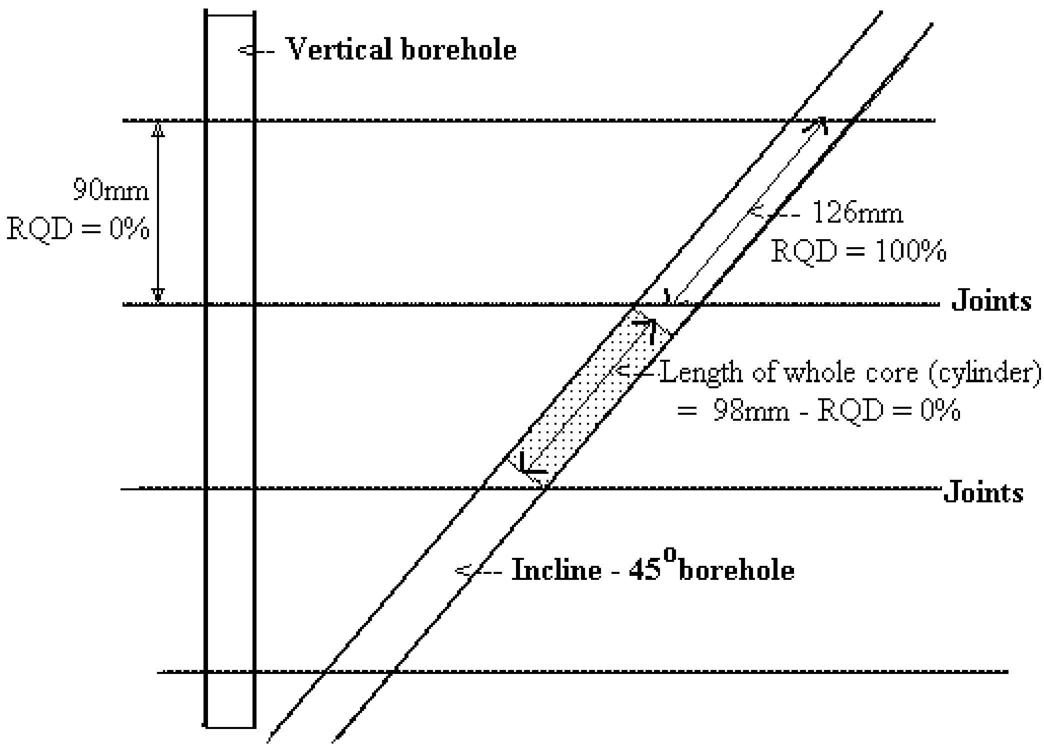
MAJOR STRUCTURES

All major structures in the orebody must be identified. The RMR of the structures to be determined and plotted on plans and sections, and the joint condition rating should be shown in brackets. Major structures influence cavability, cave angles and also the angle of draw zones, particularly, if the structures are shear zones.

All major structures in the hangingwall must be identified. The RMR of the structures to be determined and plotted on plans and sections. Major structures influence cave angles and also the angle of draw zones, particularly if the structures are shear zones.

MINOR STRUCTURES

Joints have sufficient continuity to define rock blocks, whereas fractures do not have sufficient continuity to form rock blocks, but can reduce the rock block strength. Various techniques are used to measure joints and fractures in core and underground mapping. Fracture frequency per metre is a system used extensively, but, joints and fractures must be separated and factors applied according to the core angle of intersection to compensate for sampling bias. Rock Quality Designation - RQD - is a very coarse method of defining competency in broad terms, however, it is very site and borehole angle sensitive and only whole core should be measured.



Difference in RQD for an angled versus a vertical borehole

In the above figure the RQD of the vertical hole is 0% whilst the RQD of the incline hole is 100% if the full length of core is measured. If the length of whole core is measured in the incline hole then the RQD is 0%. There are no specific guidelines on how to distinguish between joints and fractures in core except by appearance, striations on the joint surface, sheared material in the joint and possibly alteration of the wall rock. In some instances, like gypsum filled features it might be necessary to take a 'flyer' and assume that one third are joints

STRUCTURAL ZONES

It is important that any variation in joint/fracture spacing is noted and if possible the orebody should be zoned e.g. well joined = zones with a joint spacing of <1m, medium jointed = joint spacings of 1m to 3m and less jointed = a joint spacing >3m. However, the zoning would be determined by the field evidence and might only be two zones of spacings, for example < 2m and > 2m. These zones are extremely important in cavability assessments and in the fragmentation analyses. The structural zoning must be carried into the hangingwall as it might be found that the orebody contains more structures than the hangingwall and this would influence cavability.

OREBODY - SHAPE, DIMENSIONS, TONNAGE, DIP AND STRIKE

The shape description refers to geometric shapes e.g. pipe, tabular, lenticular and should reflect changes along strike and on dip e.g. a narrowing or bulging and illustrated with relevant plans and sections. The reasons for these changes should be interpreted and presented. The dimensions are those defining an orebody within a specified cut-off grade and are intended to give an idea of the magnitude. Tonnages are shown as the overall tonnage, as well as tonnages between vertical limits and/or as sections along strike so as to reflect any changes in shape and values.

PRESENTATION OF DATA

Hard copy properly drawn longitudinal and cross sections are essential to understanding the orebody. If the correct scale is used there will be space to clearly depict important features.. Reliance on pictures on a computer screen can lead to problems as has been seen on many occasions. 3D solid models of the orebody help in visualising shape and as such, greatly assist in mine planning and explain sequence etc. to the uninitiated. Isopachs of orebody thickness and thickness x grade are also essential. The object is to present the data as clearly as possible since the final decision to mine the deposit does not lie with the geologist.

ACCURACY OF BOREHOLE DATA - ROCK EXPOSURE CORRELATION

A number of block caving operations are being designed solely on borehole information.. This means that the logging of all cores must be done according to a comprehensive system that will provide all the above information as well as the necessary rock mass classification and geotechnical data. There is a tendency to use exploration holes purely for grade and general geological data. Unfortunately this leads to an enormous loss of information. It must be assumed that the drilling program will locate an orebody and therefore, all geotechnical and detailed geological data must be logged. Differences between holes drilled in different directions must be noted as this will indicate a bias. Wherever possible some holes should be drilled on the line of drifts before the drifts are developed so that borehole data can be correlated with rock exposures as soon as the drifts are developed.

The drilling program should be carefully planned with the specific object of gathering geotechnical data at the same time as assay data. Alternate cross sections can be drilled from opposite sides of the orebody. This will ensure that the wall rocks are drilled on both sides of the orebody. It will also reduce the structural sampling bias as the structures that are sub-parallel to the one set of holes will be at a large angle to the core in the opposite set of holes. A series of longitudinal sections should also be drilled from both directions. Because core is subjected to stresses during the drilling operation core can appear to be more fractured than would be the case from underground mapping.

Core Photography

All core must be photographed and presented either as slides or prints. Slides can be projected on to a translucent screen, while the operator can stand behind the screen and map the core at the natural scale.

Drilling techniques

For good reliable data it is essential to have the best core that can be obtained. In good ground, double tube drilling is adequate, but in poor ground triple tube drilling is essential. In poor ground the core should be structurally logged at the drill rig whilst still in the splits and before transfer to the core box. An adequate supply of new or used splits will be needed. In better ground, the core can be transferred to plastic splits (cut from matching size PVC piping). The core should be left in the split and both transferred to the core box, covered with a layer of foam and a secure lid. The boxes should be handled with care. Orientated core will improve the accuracy of the data. On the mechanical side it is possible to obtain good core recovery even in poor ground. The high cost of drilling can only justify better drilling logs, where, the following are recorded :-

- Length of sticks coming out of the core barrel.
- Drilling penetration rates
- Loss of water
- Accurate marking of drillers breaks
- Location of cemented zones.

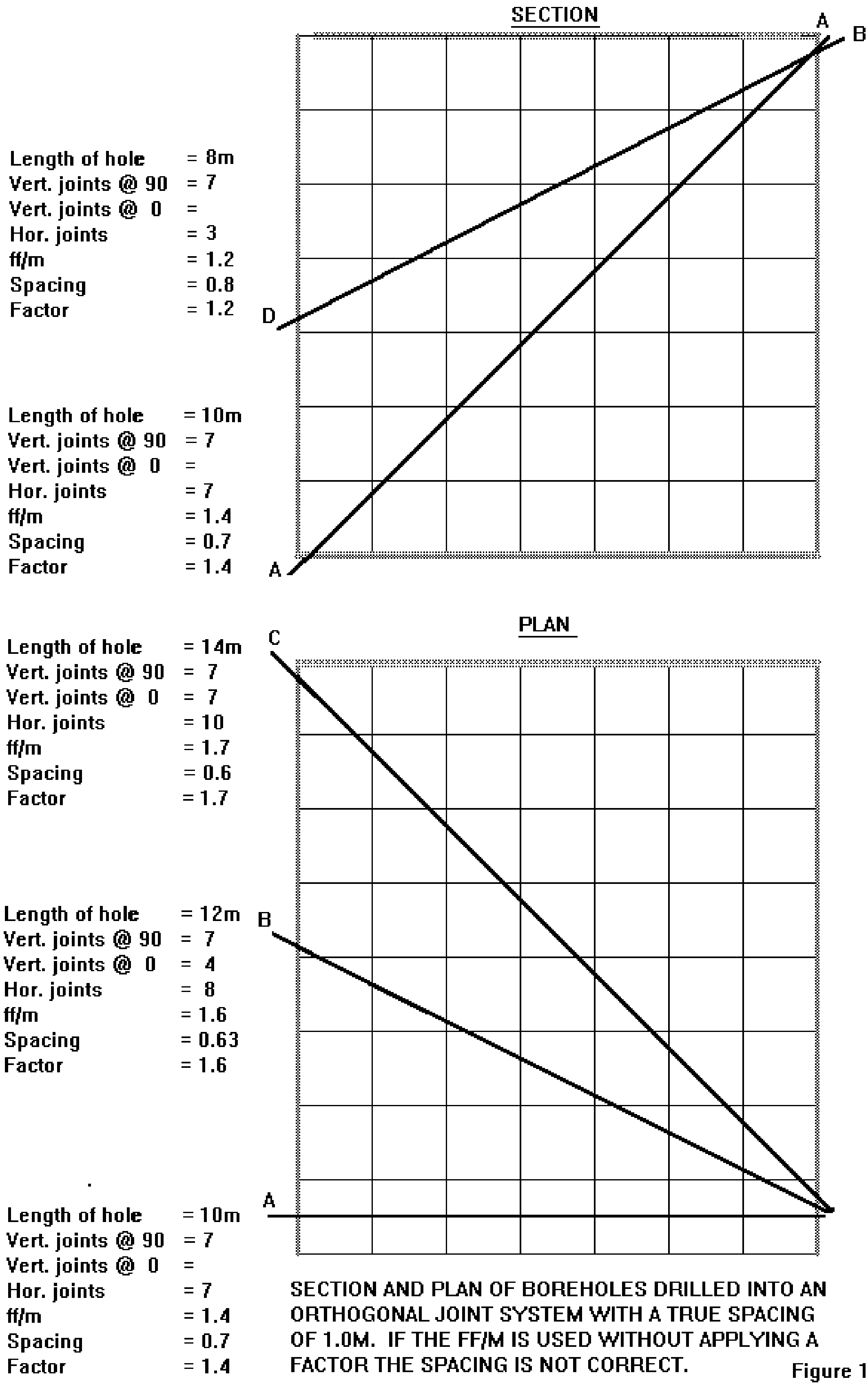
Under no circumstances should “caving” be discarded. Rubble that accumulates at the bottom of the hole when the rods are withdrawn may be due to unstable small fragments from highly fractured zones or they could be fragments produced by borehole break out - which is the fracturing of the sidewall of the hole produced by high in situ stresses.

It seems that drillers are often not aware of the importance of the core and it is up to the mining industry to ensure that the educational aspects are not neglected.

Figure 1 at the end of the text shows the importance of correctly interpreting the core intersections of joints.

ASSESSMENT

It is important that the deposit is viewed as a whole and not as a series of little windows.



SECTION AND PLAN OF BOREHOLES DRILLED INTO AN ORTHOGONAL JOINT SYSTEM WITH A TRUE SPACING OF 1.0M. IF THE FF/M IS USED WITHOUT APPLYING A FACTOR THE SPACING IS NOT CORRECT.

Figure 1

DESIGN TOPIC

Geotechnical Investigations

GENERAL DESCRIPTION

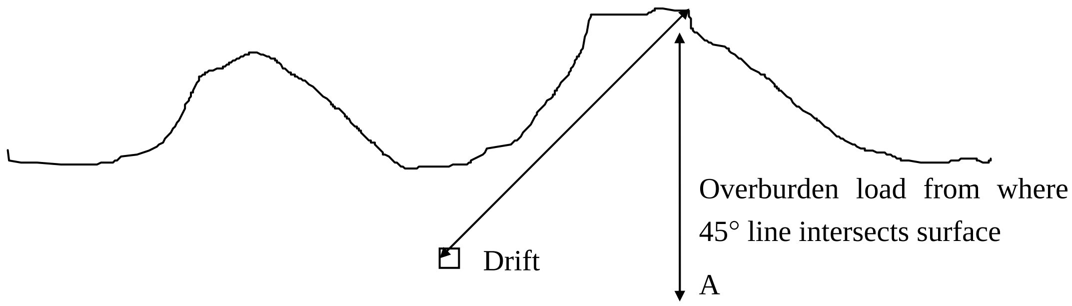
Geotechnical investigations cover all rock mechanics investigations leading up to method selection and during the mining operation, to measure rock mass response and to plan the remedial strategy if that be required. The intention is to ‘convert’ the geological and classification data into the engineering process of designing a block caving operation.

This section must include monitoring programs and here caution must be exercised in the selection of techniques, the KISS principle applies simple and cost effective so that results are readily available. Mathematical modelling recommendations are required.

STRESS MEASURING TECHNIQUES

Regional stresses are usually obtained from in situ stress measuring techniques, such as hydrofracturing which can be done in boreholes at depth or by overcoring techniques if underground access is available. If regional stresses are not available from measurements then they will have to be estimated from data in the district backed by the geological history. It is essential that the values bear a relationship to the geological history. Cognisance should be taken of any core disking or borehole break-outs that occurred during drilling. It should be borne in mind that stress measurements are simply measurements of stress at a point in the rock and that stresses vary from point to point, depending on local structures and rock types. The results of stress measuring programs need to be interpreted and in this it is essential that the accepted stress values bear a relationship to the geological history.

In relatively flat terrain the vertical stress should not exceed the overburden load. In mountainous areas the principal stress can be obtained by the following rule of thumb technique:-



Principal stress in drift equals vertical stress at A

Regional and Local Insitu Stresses (L. Loren)

Determination of insitu stresses is a fundamental part of any geotechnical investigation for a caving mine. Since the ground surface is always stress free, simple static's requires that the vertical stress component at any point be given by the product of the unit weight of the rock and the depth below the ground surface. However, it is impossible to determine the horizontal stress components (and therefore the complete state of stress) without measurement. Lack of knowledge concerning the horizontal stresses comes from poor understanding of the processes involved in geologic evolution at a particular site and inability to quantify how those processes affect insitu stresses. Horizontal stresses may be approximated from regional stress compilations, if available from, for example E. N. Lidner and J.A. Halpern (1978), "In-Situ Stresses in North America: A Compilation," Int. J. Rock Mech. Min. Sci. & Geomech. Abstr., **15, pp.** 183-203. or B. M•ller, M. L. Zoback, K. Fuchs, L. Mastin, S. Gregersen, N. Pavoni, O. Stephansson and C. Ljunggren (1992), "Regional Patterns of Tectonic Stress in Europe," J. Geophys. Res., **97**(B8), pp. 11783-11803. But since local stress conditions often vary considerably from regional conditions, it is usually necessary to measure the local stresses insitu.

Stress measurements provide values at specific points within the rock mass. However, it is often not possible or even desirable to perform insitu stress measurements sufficiently far from underground excavations or topographic features so that the influence of excavation or topography can be neglected. To obtain a general understanding of the stress field, a number of measurements at different sites in the mine area are usually made. Three-dimensional models can then be used to quantify the various forms of induced stress, such as those generated by topography, excavations or material property variations. The following procedure has been used at three mine sites in the Chilean Andes to estimate pre-mining insitu stresses based on insitu measurements.

The total stress field, σ_{tot} , at any point is the sum of initial stress plus any induced stress components, $\sigma_{ind.}$. Initial stress, in turn, is composed of gravitational stress, σ_{grav}, plus an as-yet-undetermined additional horizontal component that will be referred to as a tectonic component, $\sigma_{tec.}$. There are various geologic reasons why this additional horizontal component of stress should be incorporated into the total stress tensor. Equation (1) relates these components:

$$\sigma_{tot} = \sigma_{grav} + \sigma_{tec} + \sigma_{ind} \qquad (1)$$

The induced stress, in turn, is composed of gravitational and tectonic components:

$$\sigma_{ind} = \sigma^{I}_{grav} + \sigma^{I}_{tec} \qquad (2)$$

Substituting equation (2) into (1) produces:

$$\sigma_{tot} = \sigma_{grav} + \sigma^{I}_{grav} + \sigma_{tec} + \sigma^{I}_{tec} \qquad (3)$$

which upon regrouping becomes:

$$(\sigma_{tec} + \sigma^{I}_{tec}) = \sigma_{tot} - (\sigma_{grav} + \sigma^{I}_{grav}) \qquad (4)$$

Terms on the right-hand side of equation (4) are either known (stress measurements are representative of the total stress field) or can be computed using a model with only gravitational loading.

To start the computational process, it should be assumed that the problem geometry (i.e., topography, nearby excavations) is the primary factor generating the induced stress field and that material property variations produce only second order effects. Experience has shown this to be a reasonable assumption.

First, a model is constructed taking into account the topography and excavation geometry. This model is run with gravitational loading only. The resulting stress field at the measurement points accounts for the gravitational component of the induced stress caused by the problem geometry. If the computed and measured vertical components of stress are found to differ by a large amount, then either the model is incorrect(such as incorrect densities), there are other unknown sources of induced stress (such as locked-in stresses from geological processes) or there are significant errors in the measurement. In this situation, it is best to investigate further the reason for the stress anomaly, as confidence in the stress field is a critical design requirement.

The unknown tectonic components can be solved by applying unit normal or shear stress conditions to the model and computing the resultant stress level at all stress measurement points. The correct "far-field" tectonic stress is computed by scaling the unit stress results through a least squares procedure to match the magnitudes of components using equation (4). The total stress field is then specified by the combination of horizontal tectonic stresses as well as the gravitational stresses.

INDUCED STRESS PREDICTION

Numerical modelling is of a great help in providing a picture of the possible induced stresses. If the modelling facilities are not available then stress distribution diagrams as found in many textbooks and common sense will clearly show areas of high stress. This is obviously a back and forward exercise because as more work is done on the planning so the induced stress prediction will be updated.

The definition of high stress is the relationship between rock mass strength and the mining induced stresses which will relate to regional stress, the geological environment and mining geometry. Mining geometry can result in high mining induced stresses due to large leads between faces, or excessive development in abutment stress areas. The main cause of problems in cave mining operations are abutment stresses. The magnitude and damage effects of abutment stresses are well know, but, seem to be accepted as part of a cave mining operation. The damage caused by abutment stresses are extensive on pre developed production levels and drawbells. The advance undercutting technique has been recommended to ensure that there is the minimum amount of development ahead of the cave front.

EFFECT OF MAJOR STRUCTURES

Major structures can have a significant impact on the operation. In some cases they might be beneficial in promoting caving or producing more favourable draw angles and in other cases they can give rise to massive wedge failures, promote unfavourable draw so that there is early dilution entry, influence the direction of drifts and local support requirements. Major structures spaced at regular intervals of 10m, 20m, or 30m could give rise to major blocks in the early stages of the cave and they would report in the drawpoint as oversize or they are known to lie across the minor apexes to give rise to high hangups. These blocks could fail once the cave column had progressed to a at sufficient height so as to impose large enough caving / arching stresses to break these rock block. If the major structures occur as well developed shear zones then the material in the shear will report as fines and move more rapidly through the draw column. These fines will also cushion the large blocks during drawdown and thus reduce the secondary fragmentation.

GEOTHERMAL GRADIENT

A high geothermal gradient and high ambient temperatures could mean the need for refrigeration plants as part of the ventilation system. However, what is important is that the production can come from part of a level for six years and from the whole level in a large orebody for say 15 - 20 years. Whilst the rock temperatures might be high to begin with, there is no increase in development to expose new rock surfaces and as the cave matures the muckpile will cool off and the rising hot air will concentrate in the upper portions of the muckpile until the cave breaks through to surface and there is a release of hot air. This phenomena can be seen in a cave crater on winter mornings with wisps of fog coming from the cave. The question is, is it necessary to spend large sums on refrigeration when the problem might be short term. After all it is not the same as a South African gold mine where fresh rock surfaces are being exposed all the time. Air conditioned cabs or remote loading would overcome a lot of the perceived problems.

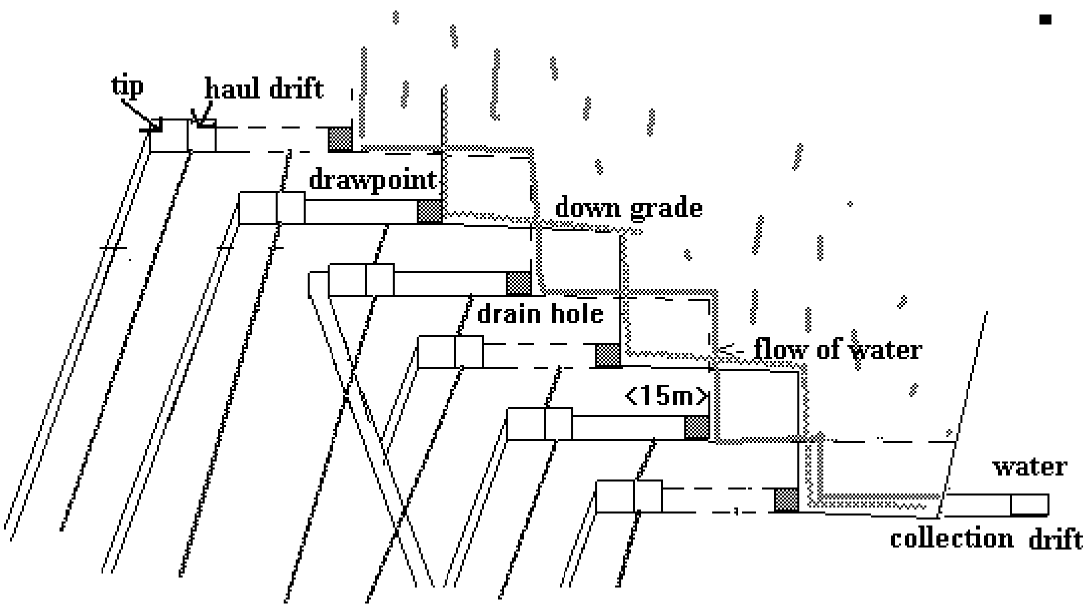
GROUND WATER / SURFACE WATER

Water in a cave is acceptable in minor quantities as damp ore does not generate dust. But water in large quantities can present major problems in the following areas:

- poor working environment
- poor hauling conditions
- risk of mud rush
- washing out of fines
- rapid wear of roadways
- excess equipment wear (tyres and rust)
- support damage (rust)
- problems in orepasses and loading bins

Dewatering programs should be designed and implemented at an early stage.

The Incline Drawpoint layout provides an effective means of removing water from the system at an early stage as the bulk of the water moving down the footwall cave boundary is removed on the upper two levels. The Incline layout can be modified so that the undercut section of the drawpoint is down grade allowing the bulk of the water to flow down to the lowest level, which can be set up as a water collection level.



The same principles can be applied to the horizontal layout, particularly the drawpoints on the flanks if the water inflow is concentrated along the block boundaries.

Water balance calculations (inflow / mine dewatering) should be completed on a regular basis to indicate potential water accumulation and subsequent catastrophic discharge.

DEFINE AREAS THAT NEED DETAILED INVESTIGATION

Areas that require detailed investigation need to be defined at an early stage in the investigation. The early stage could be the conceptual study when mining methods are being considered and the deposit is being extensively drilled.

MONITORING

The monitoring program will be developed as the layout and mining areas are defined. It is worthwhile stating that simple monitoring devices are still effective and can be read during routine underground tours. As the stress levels increase, monitoring of seismic activity becomes more and more important in regulating cave front advance and the rate of caving to minimise seismic events. The number of monitoring devices should be kept to a minimum so as to ensure that results are properly interpreted. Details of monitoring systems are described in the relevant sections.

DESIGN TOPIC

Rock Mass Classification
IRMR/MRMR

This paper discusses updated Rock Mass Classification by Dr.D.H. Laubscher and J. Jakubec

INTRODUCTION

The competency and engineering properties of jointed rock masses can vary greatly. There is a demonstrated need for a systematic numerical system of describing rock masses. Such a system is used:
for communication between geologists, engineers and operating personnel to provide the basis for comparison of rock behaviour from project to project and over time, and to quantify experience for the development of empirical relationships with rock mass properties and for guidelines for method selection, cavability, stability, support design etc.

The history of the MRMR system needs to be recorded. In 1973 D. H. Laubscher met with Z.T. Bieniawski to discuss the rock mass classification system that Bieniawski was developing - RMR - for geotechnical investigations of civil engineering projects and to overcome communication problems (Bieniawski 1973). His approach was better than the system being developed in Zimbabwe by Heslop and Laubscher at that time (Heslop 1973). However, Laubscher decided that a lot more flexibility was required for the different mining situations and used the RMR **concept** for in-situ ratings and brought in adjustments for mining situations, thus the MRMR system was developed (Laubscher 1975) (Laubscher and Taylor 1976) (Laubscher 1990). Over the years changes have been made to the value of the ratings as the relative importance of the items became apparent. For some time there has been concern that the role of fractures/ veins and cemented joints were not properly included. The techniques that have been developed to cater for these items have been included here.

In order to avoid confusion with the Bieniawski RMR, the term IRMR is now used to indicate the rating of the *in situ* rock mass.

The overall objectives of this paper are (1) to show how the MRMR classification system can be applied to jointed rocks and (2) to indicate the changes made to the system over the years. Figure 1 is a flow sheet to assist the reader in following the different parts of the system.

DEFINITIONS

The competency of jointed rock is heavily dependent on the nature, orientation and continuity of the discontinuities in the rock mass. Figure 2 is a diagrammatic presentation of the different structural features.

Faults and shear zones - Major features, large scale continuity and frequently very weak zones. Must be classified separately.

Open joints - An easily identified structural discontinuity that defines a rock block.

Cemented joints - A structural feature that has continuity with the walls cemented with minerals of different cementing strength. In high stress environments cemented joints can impact on the strength of the rock mass, therefore, the frequency and hardness of the cementing material must be recorded.

Fractures and veins - Low continuities and occur within a rock block. The hardness number defines the fill material and open fractures have a hardness of 1.

Mapping and core logging - It is essential in scan line mapping to log the continuities of structures and to distinguish between fractures and joints and other structural defects. In drill core logging the geologist should attempt to classify the defects being logged. It should be noted that joint may have several partings in close proximity, these will behave as a single joint and should be logged as such.

INTACT ROCK STRENGTH (IRS) TO ROCK BLOCK STRENGTH (RBS)

Intact rock strength

The unconfined compressive strength - UCS, is the value derived from testing cores and is the value is assigned to the intact rock strength - IRS. The intact rock specimen may be homogeneous or have intercalation's of weaker material, in which case the procedure shown in Figure 3 should be adopted. Care must be taken in determining this value as often the cores that are selected represent the stronger material in the rock mass. To help the reader in this regard an example is presented. As shown in Figure 3, the UCS values for the strong and weak rock are 100 MPa and 20 MPa respectively. It is estimated that of the total, 45% is made up of weak rock. Using figure 3 one locates this value on the Y axis, moves horizontally to the curve representing the strength of the weak rock, and then drops down to the horizontal axis. In this case the appropriate "corrected" IRS is 37 MPa.

Rock Block Strength (RBS)

To obtain the rock block strength (RBS) from the "corrected" IRS, various factors are applied dependent upon whether the rock blocks are homogeneous or contain fractures and/or veins.

Homogeneous Rock Blocks
If the rock block does not contain fractures or veins then the rock block strength - RBS - is the IRS value reduced to 80% to cater for small to large specimen effect. Thus RBS = 0.8 x "corrected" IRS.

Rock blocks with fractures and veins - Fractures and veins reduce the strength of the rock block in terms of the number and frictional properties of the features (see figure 2). The Moh's hardness number is used to define the frictional properties of the vein and fracture filling. The standard hardness table is used, since values greater than 5 are not likely to be significant. Open fractures / veins would be given a value of 1.The vein and fracture filling must be weaker than the host rock:

Index	1 = Talc, Molybd.	2 = Gypsum, Chlorite	3 = Calcite, Anhydrite	4 = Fluorite, Chalcopy.	5 = Apatite
Inverse	1.0 0.5	0.33	0.25	0.2	

The procedure is to take the inverse of the hardness index and multiply that by fracture / vein frequency per meter, so as to arrive at a number which reflects the relative weakness between different rock masses. This number can then be used in Figure 4 to determine the percentage adjustment to the IRS value.

To obtain the RBS, the corrected IRS is adjusted by the size factor of 80% and then by the fracture/vein frequency and hardness adjustment i.e.

RBS = IRS x 0.8 x Fracture/vein adjustment (F/V) = MPa.

To illustrate this consider the following example:

IRS = 100 MPa
gypsum veins: Moh's hardness = 2,
ff/m = 8.0

The product of the inverse hardness and the fracture frequency is

Inverse of hardness index x fracture frequency = 0.5 x 8 = 4.0

Using Figure 4, one finds that the adjustment is 0.75. Therefore

RBS = 100 x 0.8 x .75 = 60 MPa.

The rating for the Rock Block Strength (RBS) can be read from Figure 5. The slope of the curve is steeper for the lower RBS values as small changes are significant.
In this case it is seen that the RBS rating is

RBS = 17.5

JOINTING

Open Joint Spacing

In previous papers, one had the option of using the RQD and joint spacing or ff/m. However, the fracture/vein frequency and their condition is part of the rock block strength calculation and therefore cannot be counted twice. It is for this reason that the joint spacing rating is reduced to 35 and refers only to open joints. Whilst there are situations where there are more than three joint sets, for simplicity they should be reduced to three sets. The chart in fig. 6 is slightly different to the previous ratings chart in that the ratings for the one and two sets are proportionately higher.

Cemented Joints

The cemented joints will influence the strength of the rock mass when the strength of the cement is less than the strength of the host rock. If the cemented joints form a distinct set then the rating for the open joints is adjusted down according to Figure 7.

For example, if the rating for two open joints at 0.5m spacing was 23, an additional cemented joint with a spacing of 0.85m would have an adjustment of 90%, so that the final rating would be 21, equivalent to a three joint set with an average spacing of 0.65m. The slope of the curve is increased to cater for the significant influence of the closer joint spacing. Failure can often occur at the cemented joint contact under high stress conditions or with poor blasting

JOINT CONDITION

Single joints

The IRMR system is revised to cater for cemented joints and to have water as a mining adjustment, however the joint condition rating remains at 40, but, the joint condition adjustments have been changed to those given in Table 1.

Table 1 - Joint condition adjustments

A.	Large scale joint expression	Adjustment % of 40
	Wavy - multi directional	100
	Wavy - uni - directional	95
	Curved	90
	Straight, Slight undulation	85
B	Small scale joint expression (200mm x 200mm)	
	Rough stepped / irregular	95
	Smooth stepped	90
	Slickensided stepped	85
	Rough undulating	80
	Smooth undulating	75
	Slickensided undulating	70
	Rough planar	65
	Smooth planar	60
	Polished	55
C	Joint wall alteration weaker than sidewall and filling	75
D	Gouge	
	thickness < amplitudes	60
	thickness > amplitudes	30
E	Cemented / filled joints - cement weaker than wall rock. The percentage in the column is the adjustment to obtain the cemented filled joint condition rating	

Hardness	Adjustment
5	95%
4	90%
3	85%
2	80%
1	75%

Multiple joints

Average joint condition ratings are required for IRMR values, however, a significant variation in joint condition ratings could be due to trying to force dissimilar areas into one rating. **It is preferable to use the classification system to show variations in the rock mass as this zoning could influence planning decisions**. A weighted average of joint condition ratings can give the wrong result particularly if the rating of one set is high. For example, a single joint set with 3 joints/m has a joint spacing rating of 22 and a joint condition rating of 20 = 42.

If this set is combined with another set with a joint condition rating of 38 and 7 joints /m then the weighted average of the joint conditions is 3 x 20 + 7 x 38 / 10 = 33. The joint spacing rating for 10 joints (two sets) is 13. Combining the joint condition and joint spacing ratings, one gets a total (combined) rating of 46. This is too high when compared with the 42 for one joint set. The addition of 7 joints must weaken the rock mass. Various procedures were tried to obtain a realistic average joint condition and it was found that the chart in Figure 8 gave the best results by using the highest and lowest ratings. Therefore, if the diagram in Figure 8 is used to average the joint condition ratings this results in 25 (JC) plus 13(JS) = 38 a more likely result when compared with 42 for one joint set.

ROCK MASS VALUES

In situ Rock Mass Rating

The *in situ* rock mass rating is defined as

IRMR = RBS rating plus Overall Joint rating - see Figure 1

Rock Mass Strength

The rock mass strength (RMS) in MPa is derived from the RBS - MPa after provision has been made for the effect of the overall joint rating, because, the strength of the rock mass must recognize the role of the joint spacing and the joint condition. This is shown in the flow sheet in Figure 1 and diagrammatically in Figure 9. The formula is based on the overall joint rating as a percentage of 75 times the RBS in MPa.

RMS = RBS MPa x Overall Joint Rating / 75

For example , assume that

Overall joint rating = 47
RBS value = 30 MPa

then

RMS = 30 x 47/75 = 19 MPa

MRMR ADJUSTMENT PROCEDURE

Introduction

The IRMR rating is multiplied by an adjustment factor to give the MRMR or Mining Rock Mass Rating. The adjustment procedure has been described in previous papers, where it was stated that the adjustment should not exceed two classes, but, what was not made clear is that that one adjustment can supersede another and that the total adjustment is not likely to be a multiplication of all the adjustments. For example, a bad blasting adjustment would apply in a low stress area but in a high stress area the damage from the stresses would exceed that of the blasting and the only adjustment would be the mining induced stress. The MRMR for a cavability assessment would not have blasting as an adjustment, nor would it have weathering as an adjustment unless the weathering effects were so rapid so as to exceed the rate of cave propagation as a result of the structural and stress effects. The joint orientation and mining induced stress adjustments tend to compliment each other. The object of the adjustments is for the geologist, rock mechanics engineer and planning engineer to adjust the IRMR so that the MRMR is a realistic number reflecting the rock mass strength for that mining situation. Whilst expert systems are useful the wide variety of features that have to be recognized in mine planning requires a degree of flexibility in assessing the situation. The complete dedication to computer generated results has lead to some major errors in the past, one must not remove the human thought process. It has been found that there is a better appreciation of the planning process and operation when personnel have to think in terms of adjustments.

Weathering

Certain rock types weather readily and this must be taken into consideration in terms of life and size of opening and the support design. In the case of fast weathering kimberlites, for example is it necessary to seal the rock surface. The weathering adjustment refers to the anticipated change in rock mass strength as the exposed surfaces and joint fillings are altered by the weathering process, it does not refer to the existing weathered state of the rock as that would be catered for by the IRS and then the RBS. The two items that are affected by weathering are the rock block strength - RBS and the joint condition - JC. The RBS is affected by weathering of fractures and veins and penetrative weathering of the intact rock. Borehole cores give a good indication of the weathering process, but, the results are conservative as the surface area of the core is high with respect to the volume of core. The weathering adjustment factors given in Table 2 cover known situations.

Table 2- Adjustments for weathering

Description	Potential weathering and % adjustments				
	6 months	1 year	2 years	3 years	4 + years
Fresh	100	100	100	100	100
Slightly	88	90	92	94	96
Moderately	82	84	86	88	90
Highly	70	72	74	76	78
Completely	54	56	58	60	62
Residual soil	30	32	34	36	38

Joint Orientation Adjustment

The shape, size and orientation of the excavation will influence the behaviour of the rock mass in terms rock block stability. The attitude of the joints with respect to the vertical axis of the block, the frictional properties of the joints and whether the bases of rock blocks are exposed have a considerable influence on stability and the RMR value must be adjusted accordingly. The magnitude of the adjustment is a function of the number of joints that dip away from the vertical and their frictional properties. Obviously a block with joints that dip at 60 ° is more likely to fail than one where the joints dip at 80°. Also the joint adjustment cannot be looked at in isolation as a low angle joint is liable to shear failure whereas the steep angle joint could be clamped. A computer program could be developed to cater for the variety of situations, but, would only be valid if there were sufficient checks along the way. The joint orientation adjustments in Table 3 has now been changed so as to reflect the influence of low friction surfaces as defined by the joint condition rating.

Table 3 - Joint adjustment factors

Number of joints defining the block	Number of faces inclined from vertical	Orientation % adjustments for ranges in joint condition		
		0- 15	**16 - 30**	**31 - 40**
3	3	70	80	95
	2	80	90	95
4	4	70	80	90
	3	75	80	95
	2	85	90	95
5	5	70	75	80
	4	75	80	85
	3	80	85	90
	2	85	90	95
	1	90	95	

The adjustment for the orientation of shear zones at an angle to the development is :-

0 - 15° = 76%,
16 - 45° = 84%,
46 - 75° = 92%.

Advance of ends in the direction of dip is preferable to against the dip as it is easier to support blocks with joints dipping in the direction of advance. An adjustment of 90% should be made when the advance is into the dip of a set (s) of joints.

Mining induced stresses.

Mining induced stresses are the redistribution of field or regional stresses as a result of the geometry and orientation of the excavations. The orientation, magnitude and ratio of the field stresses should be known either from stress measurements and /or stress analyses. If sufficiently high the maximum principle stress can cause spalling, the crushing of pillars, the deformation and plastic flow of soft zones and result in cave propagation. The deformation of soft zones leads to failure of hard zones at low stress levels. A compressive stress at a large angle to structures will increase the stability of the rock mass and inhibit caving and have an adjustment of 120%; this was the situation in a caving operation where the back was stable and caving only occurred when adjacent mining removed the high horizontal stress. Stresses at a low angle will result in shear failure and have an adjustment of 70%. The adjustment for high stresses that cause rock failure can be as low as 60%. A classic example of this was on a mine where the IRMR was 60 in the low stress area, but, the same rock mass in a high stress area was classified as having a IRMR of 40. The 40 is not the IRMR but the MRMR and the adjustment in this case is 40/60 = 67%.

The following factors should be considered in assessing the mining induced stresses: (a) drift induced stresses; (b) interaction of closely spaced drifts; (c) location of drifts / tunnels close to large stopes/excavations; (d) abutment stresses, particularly with respect to the direction of advance and orientation of field stresses - an undercut advancing towards the maximum stress ensures good caving but creates high abutment stresses and *vice versa;* (e) uplift as the undercut advances;(f) column loading from caved ground caused by poor draw control; (g) removal of restraint to sidewalls and apexes;(h) increase in mining area and changes in geometry; (i) massive wedge failures; (j)influence of structures not exposed in the excavation but creating the probability of high toe stresses or failures in the back and (k) presence of intrusives which might retain high stresses or shed stress into surrounding more competent rock. The total adjustment is from 60% to 120%

Blasting

Blasting creates new fractures and opens up existing fractures/joints generally decreasing the rock mass strength. Boring is considered to be the 100% standard in terms of the quality of the wall rock, but, experience on several mines has shown that whilst the rock mass might be stable at the face deterioration occurs ± 25m back and this is a stress relief adjustment. Good blasting can have the effect to allow for some stress relief thereby improving the stability. The adjustments given in Table 4 are recommended:-

Table 4 - Blasting adjustment factors

Technique	**Adjustment (%)**
Boring	100
Smooth wall blasting	97
Good conventional blasting	94
Poor blasting	80

Water/ice adjustment

Water will generally reduce the strength of the rock mass by reducing the RBS and friction across structures and reducing effective stress. The adjustment factors for water are given in Table 5.

Table 5 - Water adjustment factors

Moist	Moderate pressure - 1 - 5 MPa 25 - 125 l/m	High pressure - > 5 MPa > 125 l/m
95 - 90%	90 - 80%	80 - 70%

In the presence of ice in the permafrost areas the rock mass could be strengthened. This will depend on the amount of ice and on the temperature of the ice. Because of creep behaviour of ice the strength usually decreases with time. Adjustments will range from 100% to 120%

DESIGN RATINGS AND STRENGTHS

Design Rating MRMR

The mining rock mass rating (MRMR) is used for design. Thus the IRMR value as adjusted for weathering, orientation, induced stresses, blasting and water.

MRMR = IRMR x adjustment factors

Design Rock Mass Strength

The design rock mass strength (DRMS) is the RMS reduced by the same adjustment factor relating the IRMR to MRMR. In the case where the

RMS	=	40 MPa
IRMR	=	50
MRMR	=	40

then the design rock mass strength would be

DRMS = RMS x MRMR/IRMR = 40/50 x 40 = 32 MPa.

PRESENTATION

The IRMR data should be plotted on plans and sections. The range of 0 - 100 covers all variations in jointed rock masses from very poor to very good. The classification is divided in to five classes of 20 rating with A and B sub-divisions of 10 points. A colour scheme is used to denote the classes on plan with full colour for the A sub-division and cross-hatched for the B sub-division. The colours are:

Class 1	Class 2	Class 3	Class 4	Class 5
100 – 81	80 - 61	60 - 41	40 - 21	20 - 0
Blue	Green	Yellow	Red	Brown

It is essential that classification data is made available at an early stage so that the correct decisions can be made on mining method, layout and support design.

It must be stressed that every attempt should be made to zone the rock mass, averaging a large range in ratings leads to planning and eventually production problems.

SCALE EFFECTS

Scale is a very important factor in considering the behaviour of the rock mass particularly when examining mass mining methods. For example, the stability/cavability of a deposit cannot be based on the MRMR from a drift assessment alone, as, widely spaced major structures play a significant role. That is, structures at 10m spacing would have a marginal effect on the overall IRMR value obtained from drift mapping, but, have a large influence on the cavability of an orebody by providing planes along displacements can occur. The MRMR value gives a hydraulic radius (HR) for assuring cavability This figure should be adjusted for the influence of major structures by using the following procedure to obtain an 'influence' number. The various factors that contribute to the 'weakness' of a major structure have been ranked as follows :

Rankings

A - **Dip** : 0º - 20º = 6, 21º - 40º = 4, 31º - 40º = 2, 41º - 60º = 1, > 61º = 0
B - **Spacing**: 0 - 9m = 6, 10 - 15m = 4, 16 - 21m = 3, 22 - 27m = 1 > 27m = 0
C - **Joint Condition**: 0-10 = 6, 10 - 15 = 4, 15 - 20 = 2, 20 - 25 = 1, > 25 = 0
D - **Stress / structure orientation**: 0º - 20º = 7, 21º - 30º = 9, 31º - 40º = 6, 41º - 50º = 3, 51º - 60º = 2, 61º - 70º = 1, > 71º = 0
E - **Distance of major structures from undercut boundaries** : 0 - 9m = 12; 10 - 20m = 8; 21 - 30m = 2; > 31m = 0.
F - **Stress values - Sigma 1 as % of RMS**: > 100% = 14, 80% - 99% = 12, 60 - 79% = 8, 40- 59% = 4, 20 - 39% = 2, < 20% = 0

The rankings are plotted in Table 7. - the highest likely ranking from the three sets is in the order of 100.

Table 7 - Form for determining the major structure influence number

Major Structures	A	B	C	D	E	F	Total
Set 1							
Set 2							
Set 3							
Total							

The total of the rankings when plotted in Figure 10 will indicate whether the HR is acceptable or whether it should be adjusted up or down.

If there are other features, such as internal silicified zones, that might contribute to stability then a deduction should be made. The magnitude of the deduction should be 15% to 40% of the hydraulic radius.

In the case of pit slopes the MRMR of the bench and the overall slope will vary as shown in Figure 11. The MRMR of the zone in which the benches are cut could be affected by joint orientation, blasting, induced stresses and even weathering. The overall slope angle would be based mainly on the IRMR / RMS values with the MRMR adjustments based on induced stresses as a result of depth and shape of the pit. The presence of major structures could be the significant factor.

PRACTICAL APPLICATIONS

The details of the practical applications can be found in the paper " Planning Mass Mining Operations" (Laubscher 1993). A summary of the applications are: (a) support design (Laubscher 1984); (b) cavability diagrams; (c) stability of open stopes; (d) pillar design; (e) determining cavability; (f) extent of cave and failure zones; (g) caving fragmentation; (h) mining sequence; (i) potential massive wedge failure.

CONCLUSIONS

The object of this paper is to show the changes that have been made to the original MRMR classification system. It must be stressed that where the system is properly applied the results are good. Unfortunately, rock masses do not conform to an ideal pattern and therefore a certain amount of judgment/interpretation is required. A classification system can give the guidelines, but the geologist/engineer must interpret the finer details as has been indicated. It is important that the rock mass be divided into zones in which there is not a great range in ratings and this can only be done if the geologist has an overall understanding of the rock mass.

As it is not possible to precisely define every mining situation, the engineer must use his judgment in arriving at the adjustment percentage. Where the system has been properly applied it has proved to be successful in planning and as a communications tool. However, what has been found on several mines is that the geology departments dabble in different systems and at the end of the day are masters of none. The numbers produced are incorrect because the personnel do not have a feel for the rock mass. The modern tendency, unfortunately, is to have programs that do the thinking for the operator.

ACKNOWLEDGMENTS

A classification system can only develop if it is used and tuned to cater for the many different situations that are encountered in designing mining operations. The writers wish to acknowledge the constructive suggestions from those people who are using the MRMR system and in particular the constructive suggestions from T.G.Heslop.

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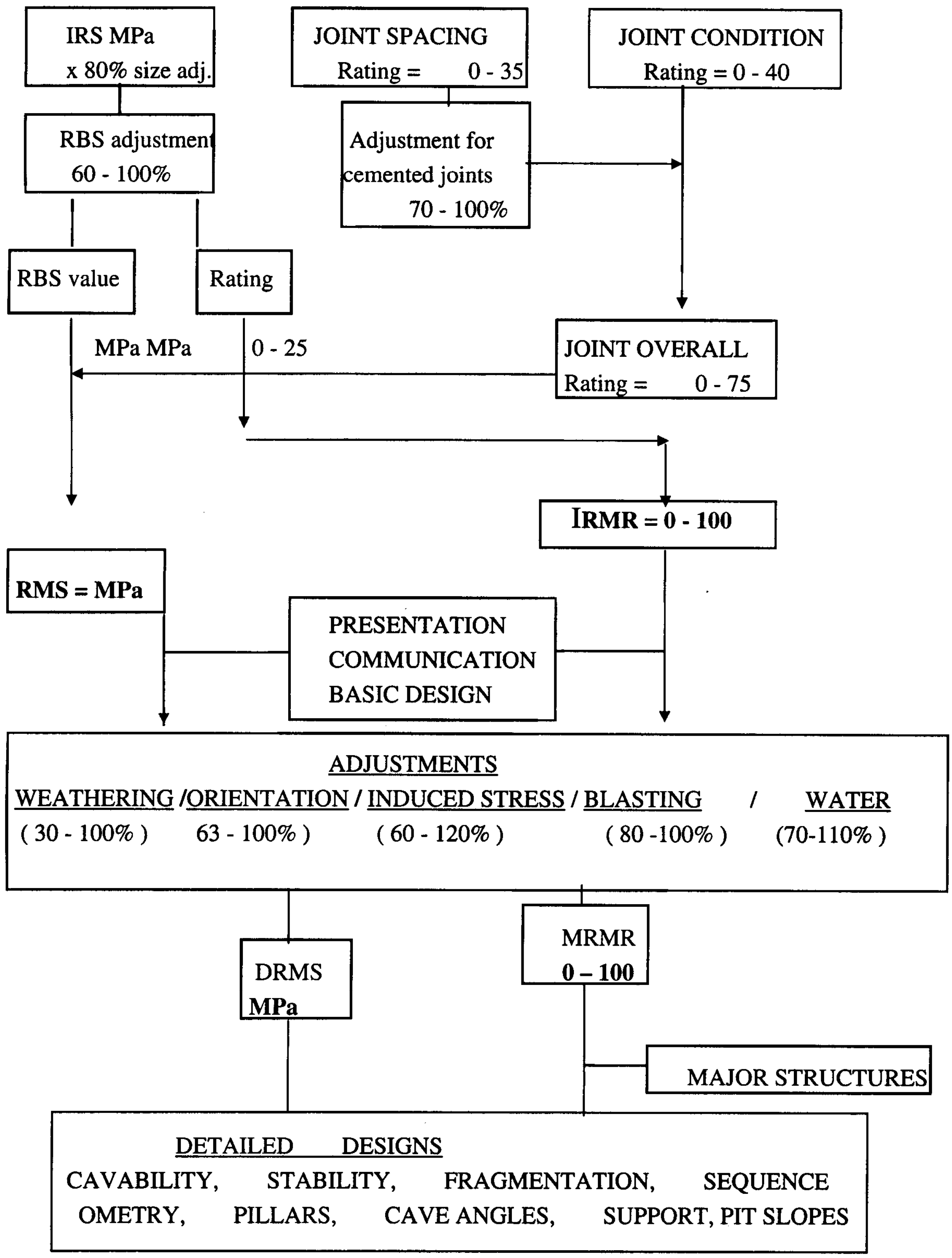
Figure 1 - Flow sheet of the MRMR procedure with recent modifications



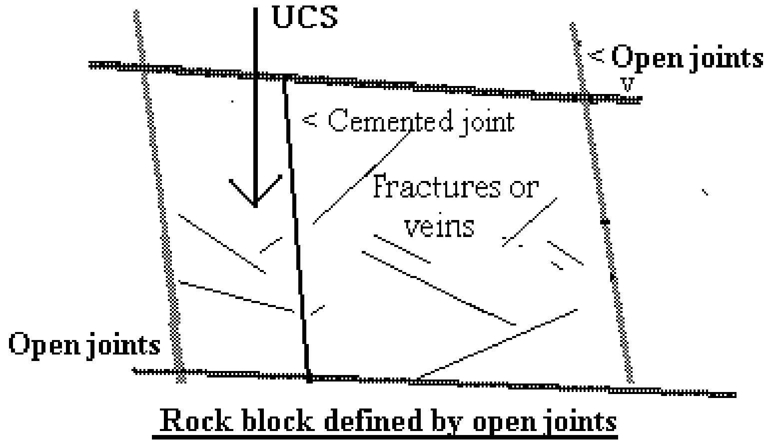


Figure 2. Definition of structural terms

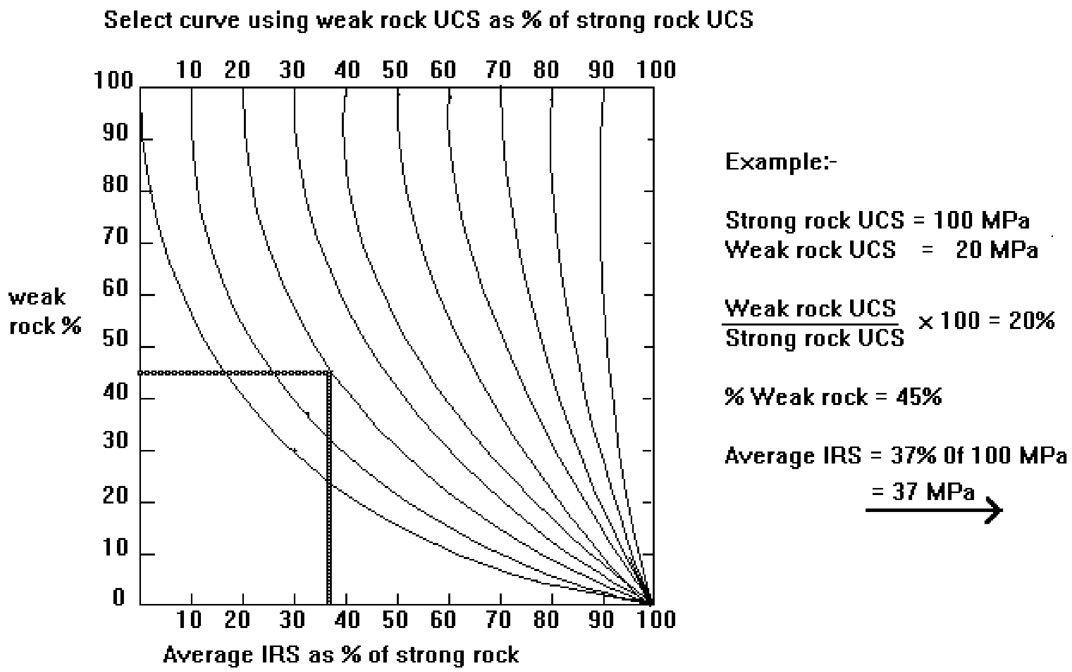


Figure 3. Nomogram for determining the "corrected" IRS value

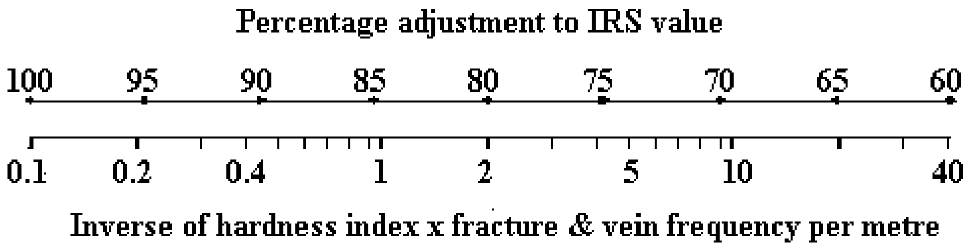


Figure 4. Nomogram for relating the IRS adjustment factor to the hardness index and fracture / vein frequency.

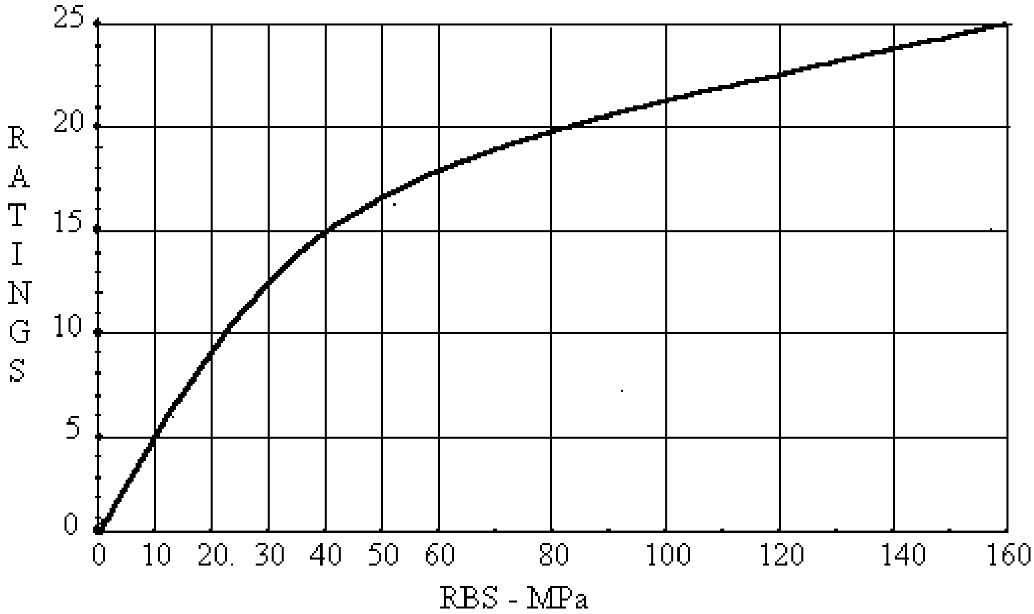


Figure 5. Rock block strength rating as a function of rock block strength.

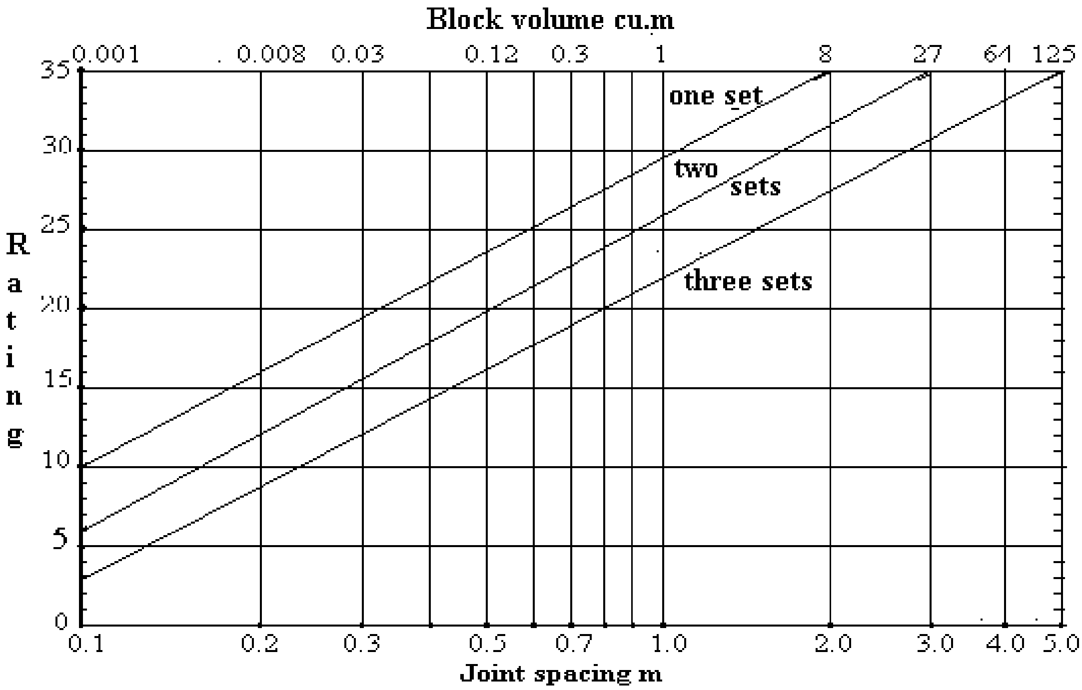


Figure 6. Joint spacing ratings

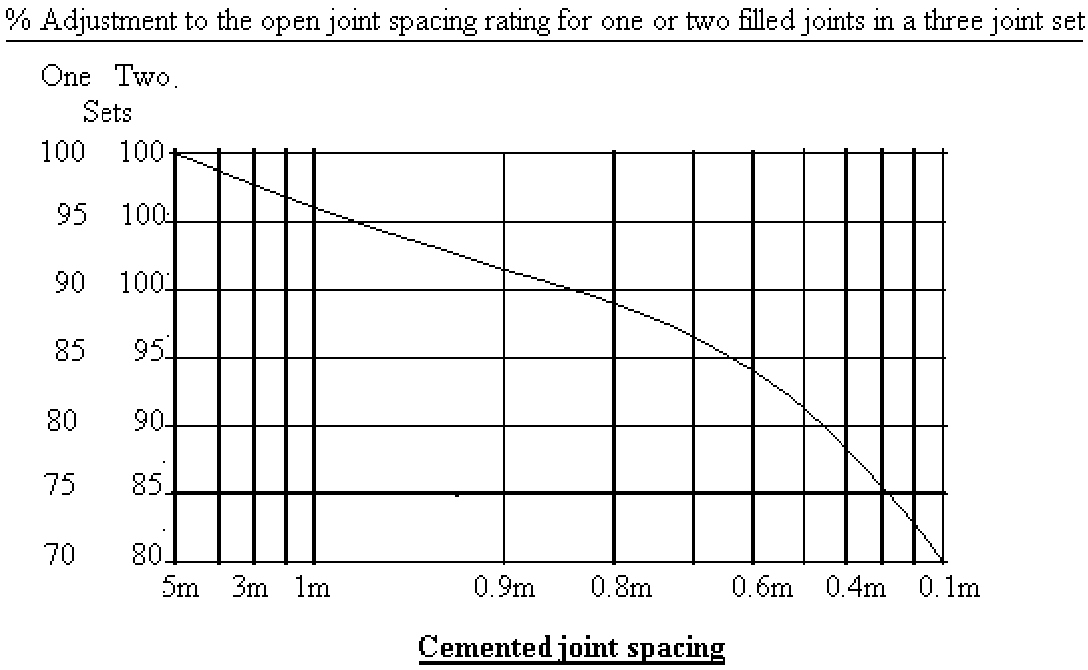


Figure 7. Adjustment factor for cemented joints

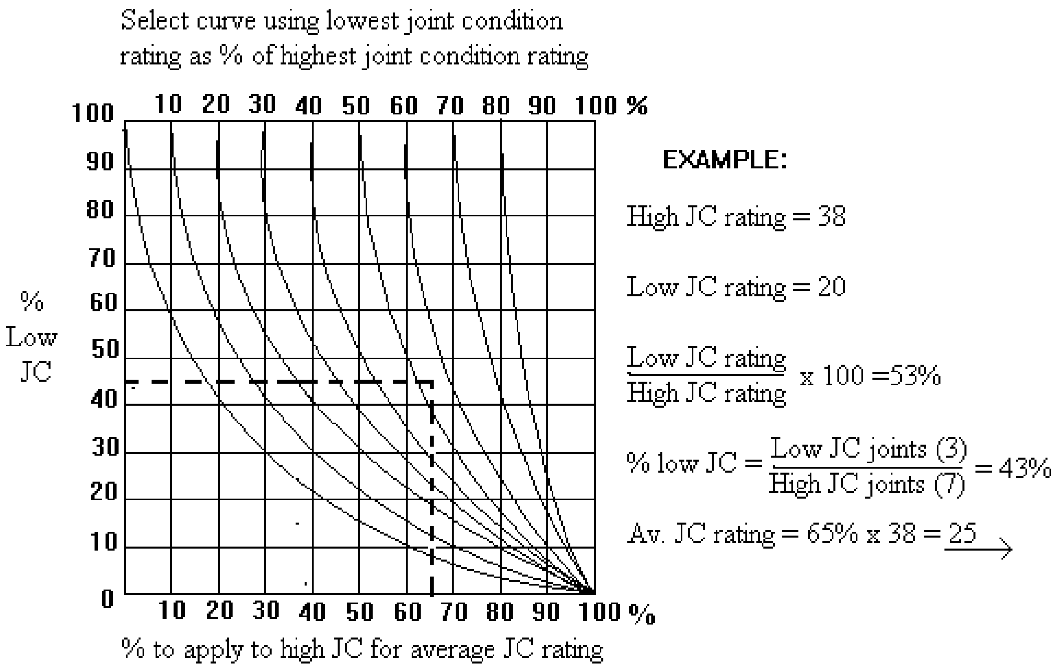


Figure 8. Chart for averaging the joint condition for multiple joint sets.

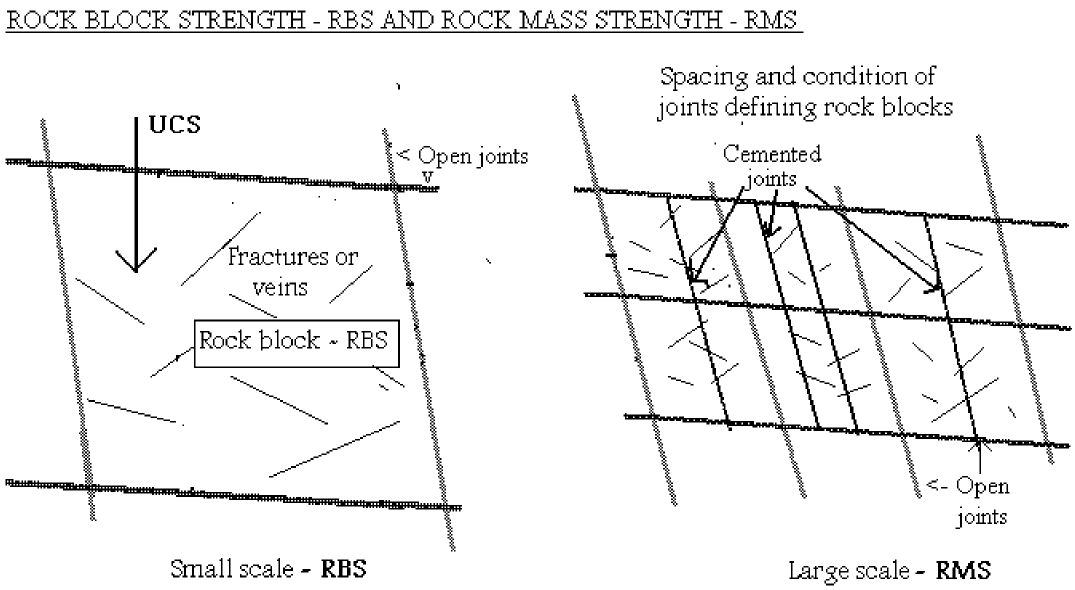


Figure 9. Diagrammatic representation of small scale RBS and large scale RMS

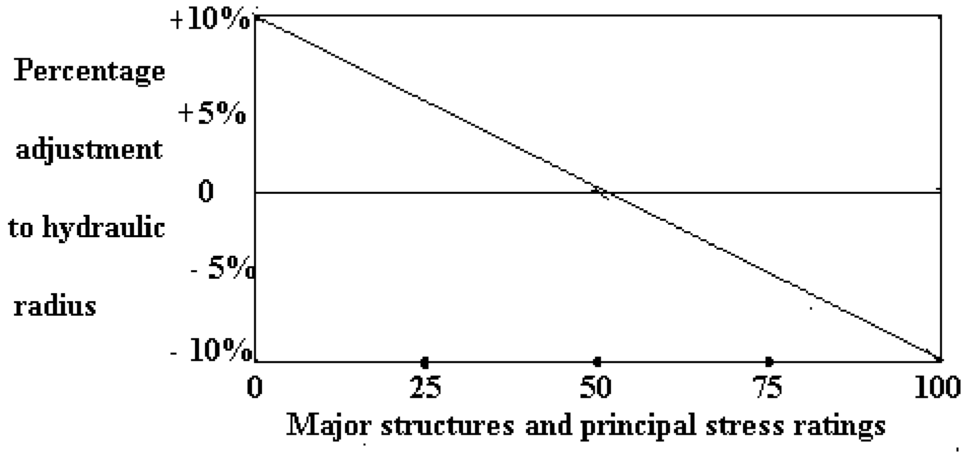


Figure 10. Adjustment factors for the effect of major structures on cavability

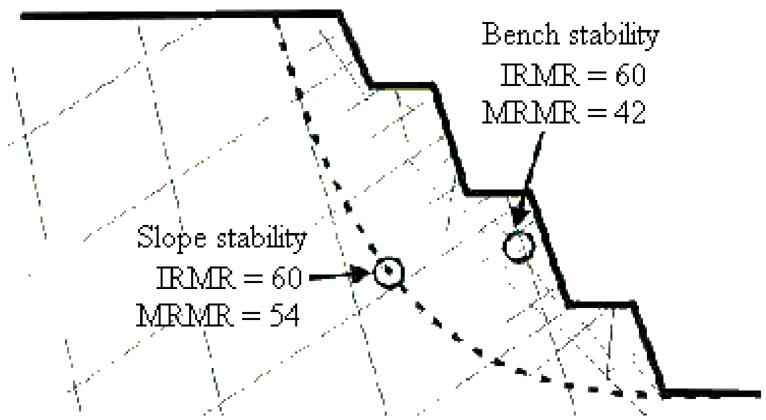


Figure 11. The difference in MRMR values used for bench and slope design.

DESIGN TOPIC

Mining Limits

GENERAL DESCRIPTION

The mining limits are a function of grade, tonnage and potential draw angle above a drawpoint. A sound mining limit is required at an early stage so that the study can proceed on a sound basis of outline and draw heights. There might be a certain amount of to and fro before the limit is finally decided as certain items become apparent with detailed analyses. For example, the draw analysis will produce drawpoint grades for different draw scenarios and this could influence the mining limits.

Extra drawpoints which can carry their development cost and be mined at a profit, but, do not contribute to the overall capital cost might be put in to increase the hydraulic radius to promote caving through a more competent zone or in tight corners which might create overhangs.

GRADE DISTRIBUTION

A record and discussion of the grade distribution will result in the correct decisions being made on the where the mining limits could be.

CUT-OFF GRADE

The cut-off grade is an economic limit which is the calculated value based on all the costs to develop and maintain a block cave and will define the mining limits. Once the viable mining limits have been defined, extra drawpoints might developed where they only have to carry development, support and maintenance costs, provided the average grade meets the planned economic returns...

OREBODY SHAPE - DIP AND STRIKE - DIMENSIONS

This section is to record the data from the geological investigation section to define the orebody shape - is it a pipe, tabular, lenticular or does it have an irregular shape with variations along strike and down dip? It is necessary to highlight any aspects which could influence the mining limits. Dimensions and variations in dimensions are required.

POTENTIAL ORE COLUMN HEIGHTS

Different ore column heights can be examined in terms of the orebody shape and estimates of dilution. These will only firm up when level intervals are finalised and the economics become clear. A start must be made to set the basis for the fragmentation analyses.

Comments From N J W Bell On Shabanie And King Mine Procedure

Planning

It is believed that the following should be put on all plans and sections during the planning phase:

- The outline of the area/s above the 'all in pay limit' **Ore**.
- The outline of area/s which lie between the 'all in pay limit' and the 'working cost pay limit' or marginal material.
- The outline of the area/s above 'all in pay limit' plus 20%. (To ensure high value ground is catered for or deliberately excluded.)
- The insitu RMRs should be shown and in particular major structures and areas of very poor ground such as:

  The footwall talc shear at Shabanie.
  At Gaths, King Section the south slip in the main orebody and the west central footwall shear in west central.

These can often not show as class 5 areas, as they may only be narrow zones. However they can have the potential for "self mining" and special support considerations may be required. The driller's records indicating loss/gain of water, loss of core, or the need to commentate to progress should be recorded on the sections/plans and if continuity is shown could well be significant.

Similarly zones of very good ground e.g. the cherts at Gaths Mine, King Section should also be indicated, together with pods of Class 1 and 2 rock as these can also be of great significance and lead to different effects of stress etc.

All the above are dependent on the range of RMRs at particular mine sites.

- The main structural direction, conditions and dips must also be clearly understood, and ways best to do this is the construction of a +\- 0.5m cube of kaylite painted and on which the main structures (dips and strikes) are clearly marked. These are continuous planes round the block. Their condition being clearly indicated so that the significance can be fully appreciated when development, brows etc are being planned and orientated.

- Topography of the area also can have a major role in the direction of the stress and the direction and flow of material once caving has commenced.

- The size of the equipment that it is planned to use and therefore the tunnel sizes, radius of curvature that need to be mined to accommodate this equipment, with full allowance for the support required.

- Facts to be considered in the planning phase;

  Accesses
  Drainage
  Ventilation – Intake and Return
  Ore handling (Density and Bulking Factors)
  Hauling
  Hoisting
  Logistics for materials / men / machinery (Storage areas and routes)
  Services – air / water / sanitation / power (indicating sub-stations, distribution points, electrical LHD plug points, etc.).
  Sand and stone distribution
  Batching bays / plants for concrete / shotcrete
  Escape routes (2nd egress)

ASSESSMENT

The assessment covers the status of the definition of the mining limits. Bearing in mind that a block cave layout lends itself to overdraw in the final stages, therefore higher grade zones in the hangingwall must be show as these could warrant overdraw before the block is abandoned.

DESIGN TOPIC

Cavability

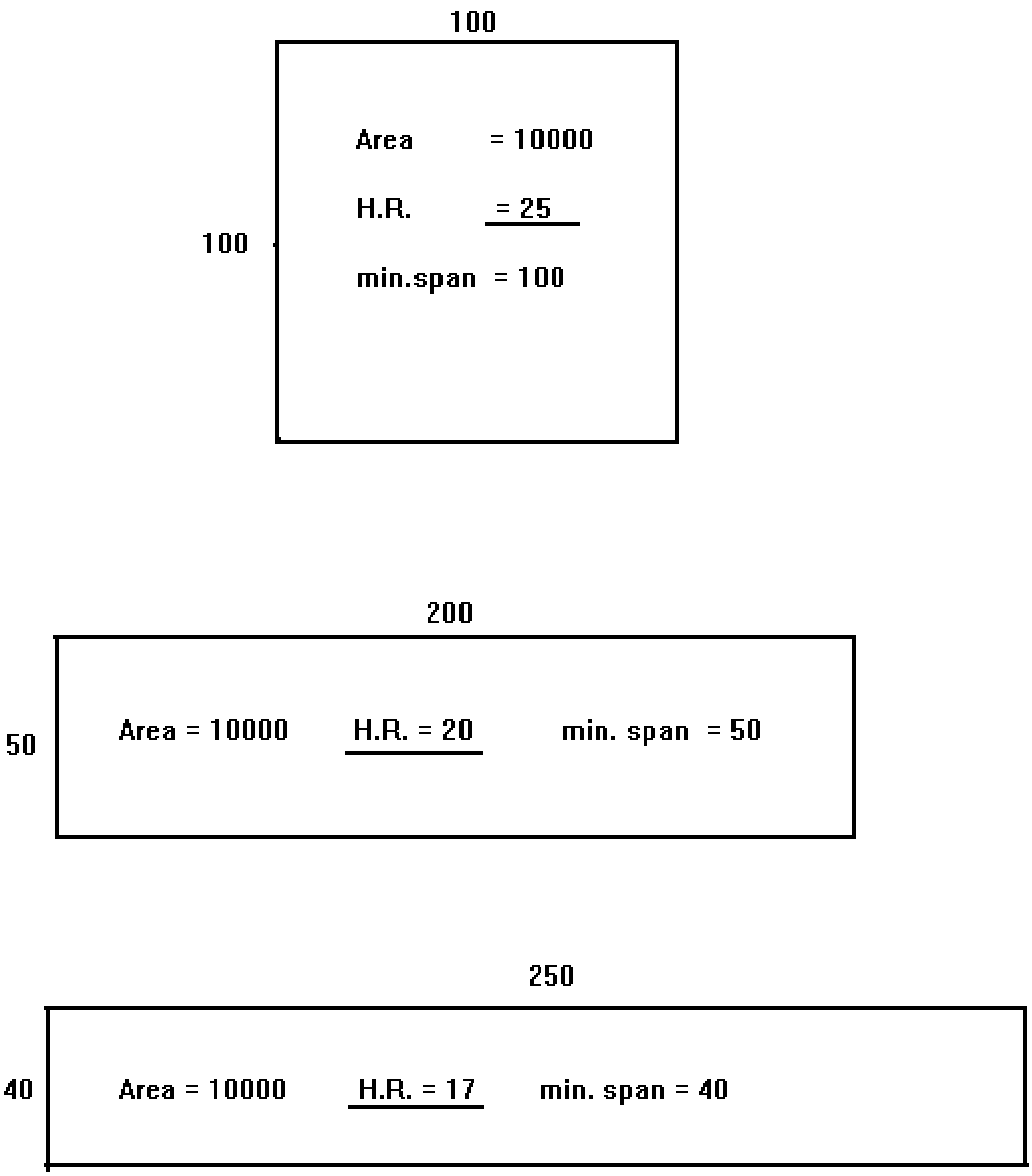
GENERAL

Cavability is usually not a major problem on most caving operations because the orebodies are so large that the hydraulic radius of the footprint greatly exceeds the caving hydraulic radius. In these cases it is usually only a question of how large an area is required to meet the initial production requirements. The bulk of the tonnage that is mined from block caving mines comes from the lateral extension of the cave. Where the hydraulic radius of the orebody is limited then more precision is required in deciding on the cavability, guidelines are provided to place the deposit in the right ‘ball park’, but, the final decision must be based on a close examination of the following factors:-.

- Rockmass strength - IRMR / MRMR
- Relevant major structures
- Regional stress
- Induced stress
- Water
- Location of adjacent mining operations
- Scale of adjacent mining operations - heavy blasting
- IRMR / MRMR of orebody and hangingwall
- Stress effects - shear failure, tension or clamping
- Cave propagation - vertical or lateral extension of the cave.
- Geometry of area under draw
- Hydraulic radius of orebody
- Hydraulic radius to propagate caving
- Minimum span
- Direction of advance of cave front and shape
- Numerical modelling
- Monitoring
- Predicted rate of caving - intermittent or continuous - influence on rate of caving
- Consolidation
- Chimney caves

HYDRAULIC RADIUS

There will be continued reference to hydraulic radius - HR - in this section and therefore a description is in order. The hydraulic radius is a term used in hydraulics and is a number derived by dividing the area by the perimeter. The hydraulic radius required to ensure propagation of the cave refers to the unsupported area of the cave back, that is, there is space into which caved material can move. No pillars can be left and caved material must be removed. The hydraulic radius very neatly brings the minimum span into play as can be seen in the following diagram which shows how the hydraulic radius will vary for the same area if the minimum span is decreased :-



Minimum Span

The minimum span is the critical dimension in promoting caving and as can be seen in the above diagrams the hydraulic radius caters for it even though the areas are the same. In cases where the hydraulic radius of the orebody is borderline and the ratio of maximum span to minimum span is high, then the a small increase in the minimum span will have a significant influence on the hydraulic radius for example, an area of 40m x 250m has a H.R. = 17, increasing the minimum span by 10m to 50m then the H.R. = 21 and caving would be ensured.

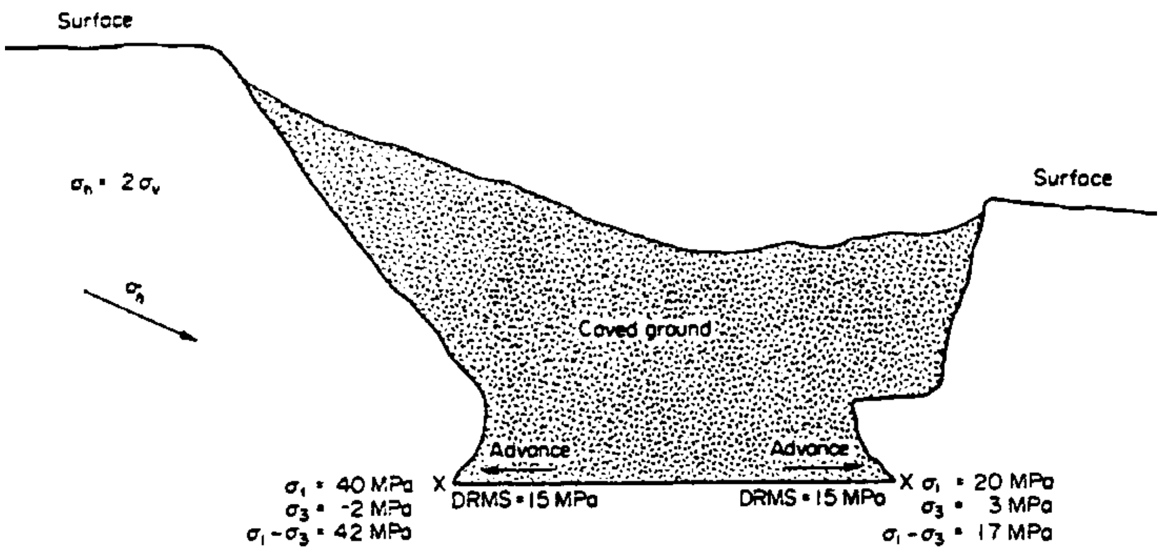
FACTORS INFLUENCING CAVABILITY

Structural Domains

Structural domains must be clearly defined as changes in density or orientation of structures can lead to caving problems in small orebodies and a significant variation in fragmentation.

Regional Stresses

The magnitude and orientation of the regional stress plays a significant role in caving. Undercutting towards the principal stress will improve the cavability and fragmentation, but, could cause squeezing damage or rockbursts. Developing away from the principal stress is advisable in the case of weak ground - an example from Western Section - King Mine.



Induced Stresses In The Cave Back

It is important that the stresses in the cave back are calculated for different heights so that these can be related to changes (if any) in the rock mass or the geometry as the caving progresses. It is on record that caving has ceased as a result of stress or rock mass changes or a change in the geometry.

The induced stress is a function of the orientation of the cave front, shape of cave back, variation in rock types and proximity to previously mined areas. The stresses in the cave back can be modified to an extent by the shape of the cave front. Numerical modelling can be a useful tool that helps to determine the stress pattern associated with several, possible mining sequences. .High horizontal stresses acting on vertical joints will inhibit caving

The stresses in the cave back can be modified to a certain extent by the shape of the cave front, in this respect numerical modelling can be a useful tool. A concave shape to the undercut provides better control of major structures and generally a stronger undercutting
environment.

The magnitude of the principal stress should be related to the RMS (rock mass strength). Once the drawpoints are commissioned then the principal stress in the cave back becomes a higher induced stress and any principal stresses that are more than half the RMS will play a significant role in the caving. All the features that are observed on the level such as squeezing in weaker ground with strain bursts and stress spalling in more competent zones will occur in the undercut back. In fact more so because there is freedom of movement and gravity plays a significant role.

IRMR/MRMR Of Orebody And Hangingwall

The IRMR of the orebody and hangingwall rock mass must be recorded on sections for the anticipated height of caving. This data will show if there are changes in the rock mass, major structures must be allocated IRMR values. This data is also required for fragmentation calculations. When the IRMR has been adjusted to MRMR it will be possible to identify zones where there might be problems in cave propagation. In those orebodies with a range of ratings it is the continuity of the lower ratings that will determine the size of the undercut. Any abnormal features that might impact on the cavability should be noted e.g. a prominent competent zone whose geometry has not been appreciated in the averaging of the RMR such as the silicified core at Northparkes. A feature such as this could result in an increase in the HR.

Major Structures

Major structures have to have sufficient continuity so that they will influence the cavability of the ore. In the chrysotile asbestos mines, shear zones are the major components in initiating the cave. The orientation of the structures is important as vertical structures are not as important as dipping structures. The orientation and dip can influence the direction of undercutting.

Structures

Flat dipping structures angled from 0º to 45º are the most significant structures as both shear and gravity failure can occur. The location of the structure(s) must be noted with respect to the undercut boundaries as a regular distribution is preferable to a concentration of joints / structures in the centre of the undercut area, which could lead to a chimney cave and overhangs along the edges.

Water

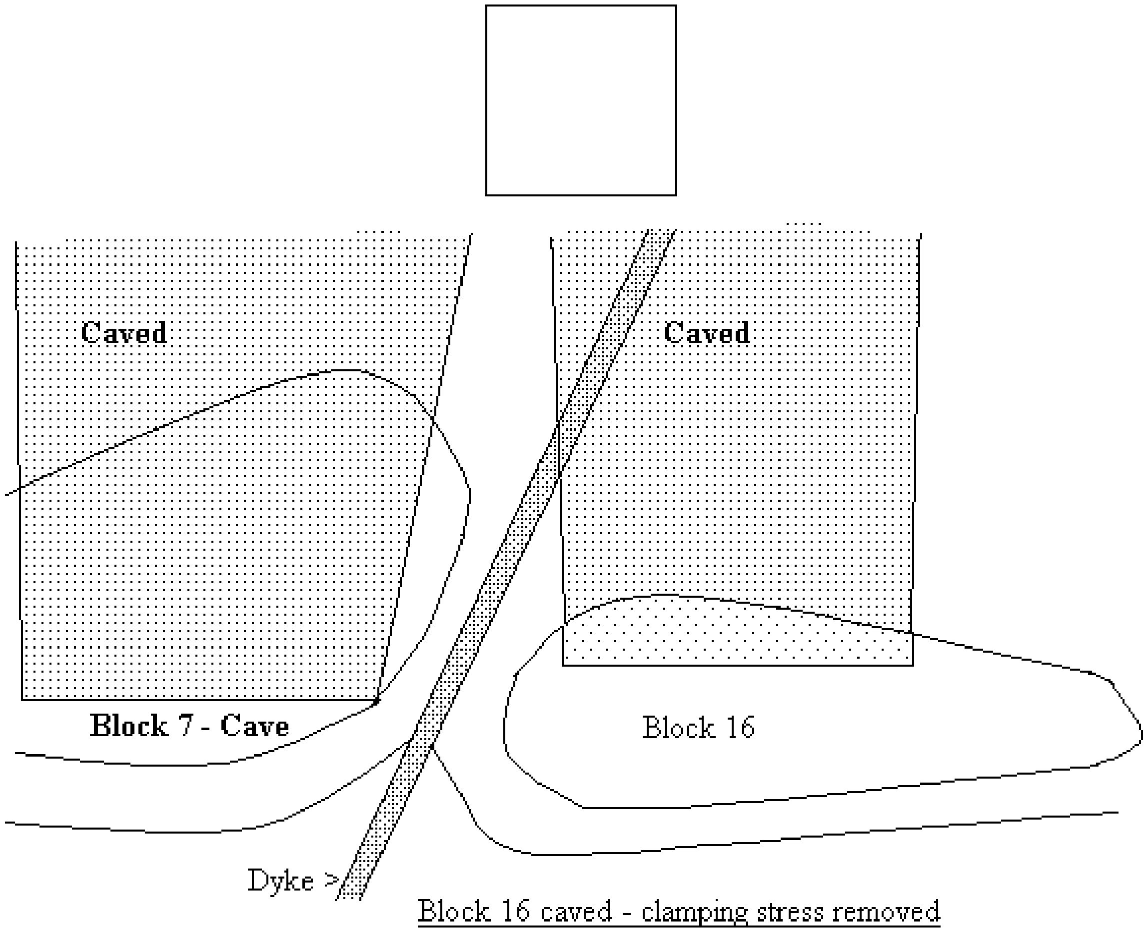
Water in the potential cave zone can assist the cave by reducing friction on joints or with the effects of increased pore water pressure. The source of the water can be ground water or water introduced during the rainy season. At Shabanie Mine, the monitoring of Block 6 cave showed that the stress caving increased after heavy rainfall.

Cave Propagation - Vertical Or Lateral Extension

At the start of a block caving operation the cave will propagate vertically, subsequent mining from the initial block will result in a lateral extension of the caved area..

Vertical Extension (Stress) Caving

Vertical extension caving was originally referred to as stress caving. It occurs in virgin cave blocks when the stresses in the cave back exceed the rock mass strength. Caving may stop when a stable arch develops in the cave back. The undercut must be increased in size or boundary weakening must be undertaken to induce further caving. The following figure illustrates the effect of removing the lateral restraint from block 16 at Shabanie Mine. Block 16 had a hydraulic radius of 28 with an MRMR of 64 and a stable, arched back. The adjacent block, no. 7, was caved and resulted in the MRMR in block 16 being reduced to 56, at which stage caving occurred.



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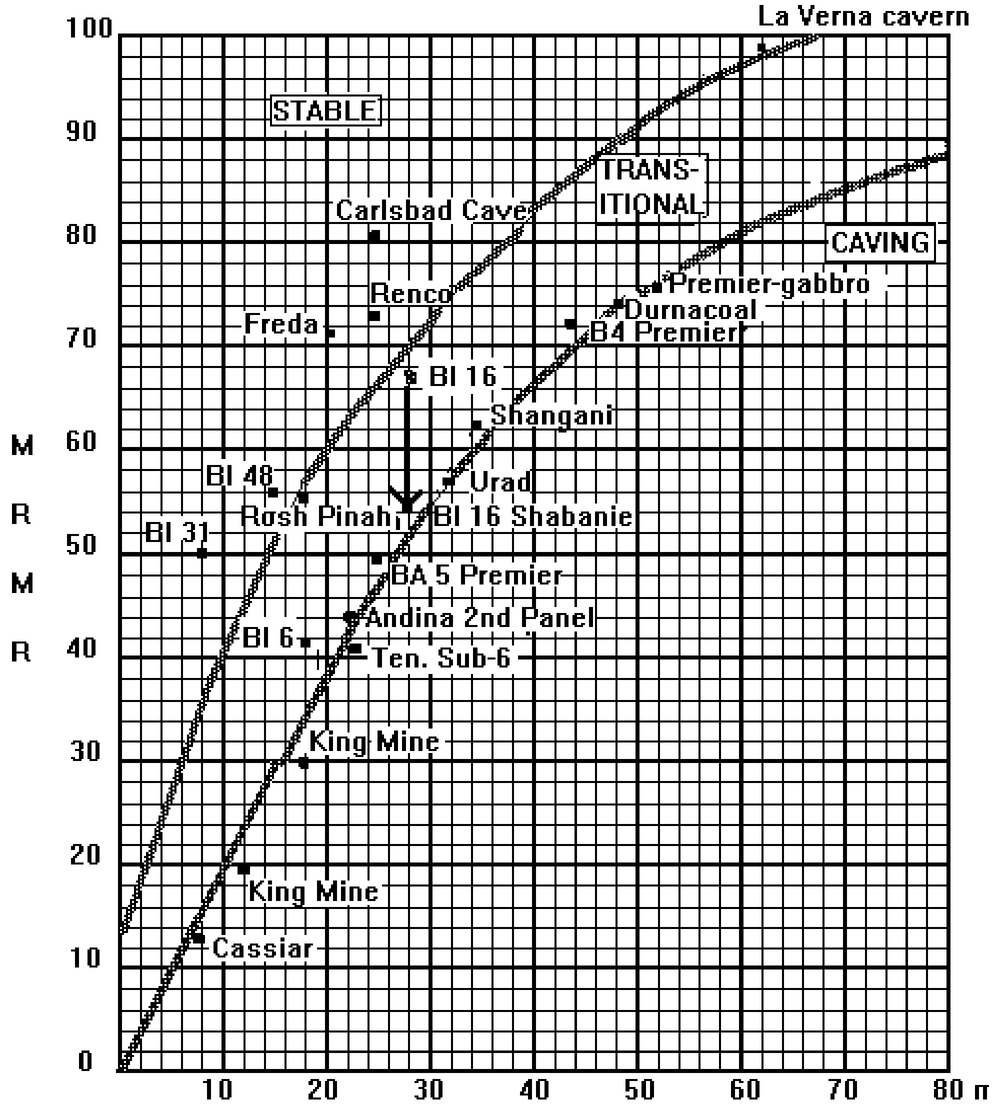
The following figure shows the hydraulic radius of world-wide caving and stable situations based on the average MRMR.



HYDRAULIC RADIUS (METRES) = AREA / PERIMETER

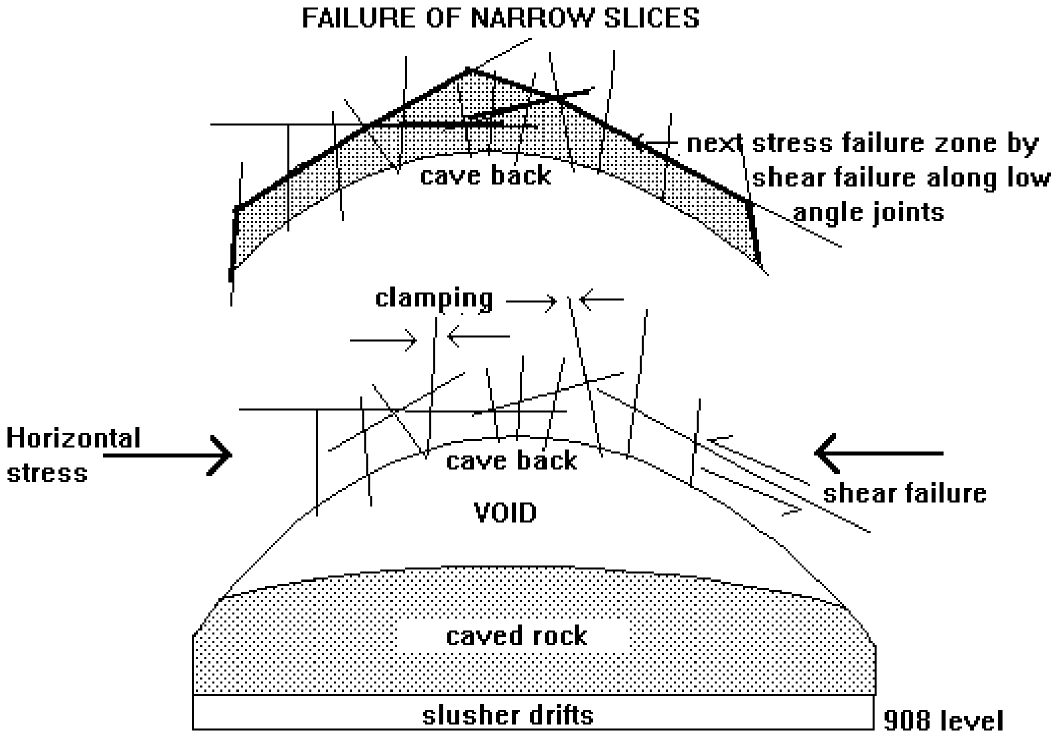
STABILITY DIAGRAM - STABLE: Will form stable back to stopes.

TRANSITIONAL: Supportable in upper band and intermittent caving/arching in lower band and could stabilise with small change to MRMR.

CAVING: Progressive caving of back or sides of current cave areas.

CAVABILITY, function of:-	Rock mass strength]	
	Geological structure]	
	In situ stress]	
	Water]	= MRMR
	Induced stress]	
	Excavation geometry]	

The recent experiences at Northparkes showed that in a low stress environment high horizontal stresses can have a major influence by clamping the unfavourably orientated steep dipping structures and effectively changing the MRMR by 120%. In large orebodies clamping is not a serious problem because as the undercut area is increased in size either the back goes into tension or more favourable structures are exposed and caving occurs

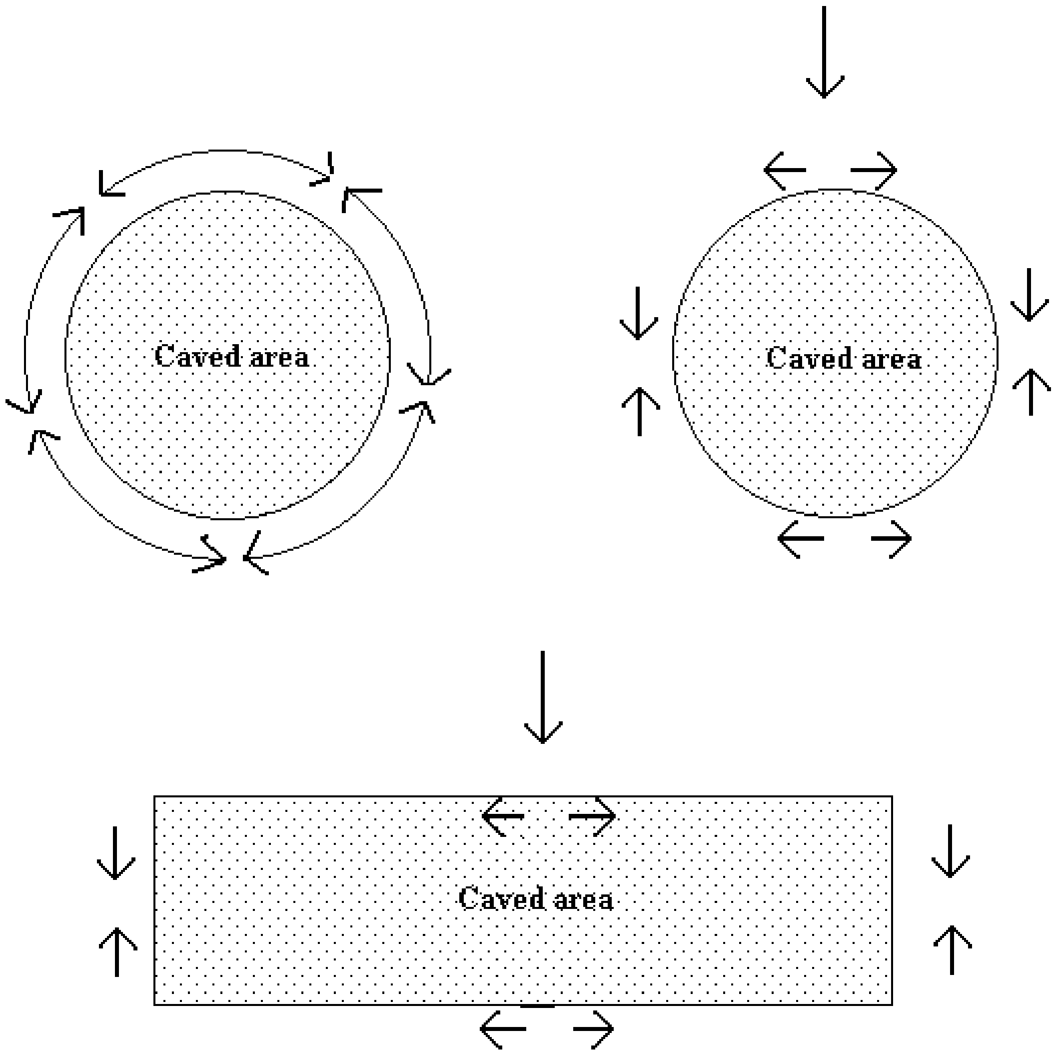


Lateral Extension (Subsidence) Caving

Lateral extension or subsidence caving as it was previously described, occurs when adjacent mining has removed the lateral restraint on the block being caved. This can result in rapid propagation of the cave with limited bulking.. Lateral extension caving occurs when the cave face is advanced from an active cave owing to the removal of a lateral stress and results in caving occurring with a lower hydraulic radius. There can be a rapid propagation of the cave with massive wedge failures if a well developed relaxation zone has formed ahead of the cave front.. In the case of panel caving stress differences and the structural pattern in the advancing cave face will determine the fragmentation. Depth, orebody dimensions and the scale of the operation will have a major influence on material behaviour. A wide orebody with a high draw height will have a slow rate of advance compared to a narrow orebody with a low draw height. This means that in the first case the rock mass will be subjected to induced stresses for a longer period.

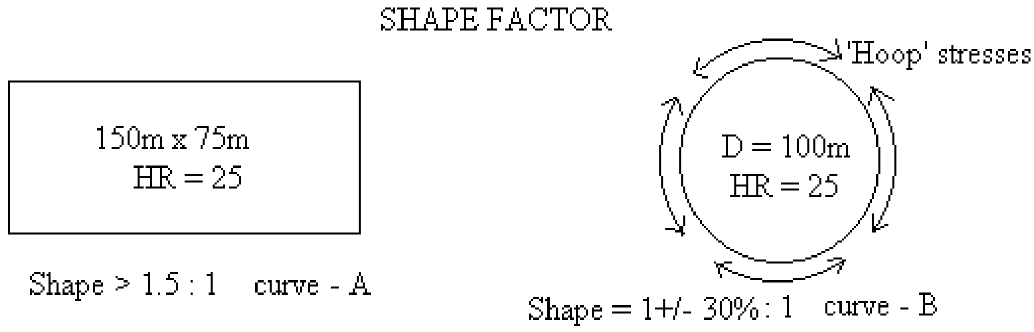
Geometry Of Proposed Cave

The hydraulic radius recognises variation in geometry particularly with respect to minimum span and will give the highest HR for a circle. However, a circle also has ‘hoop’ stresses with uniform stresses and confining stresses on two sides where there are high horizontal stresses and this results in a more stable shape..

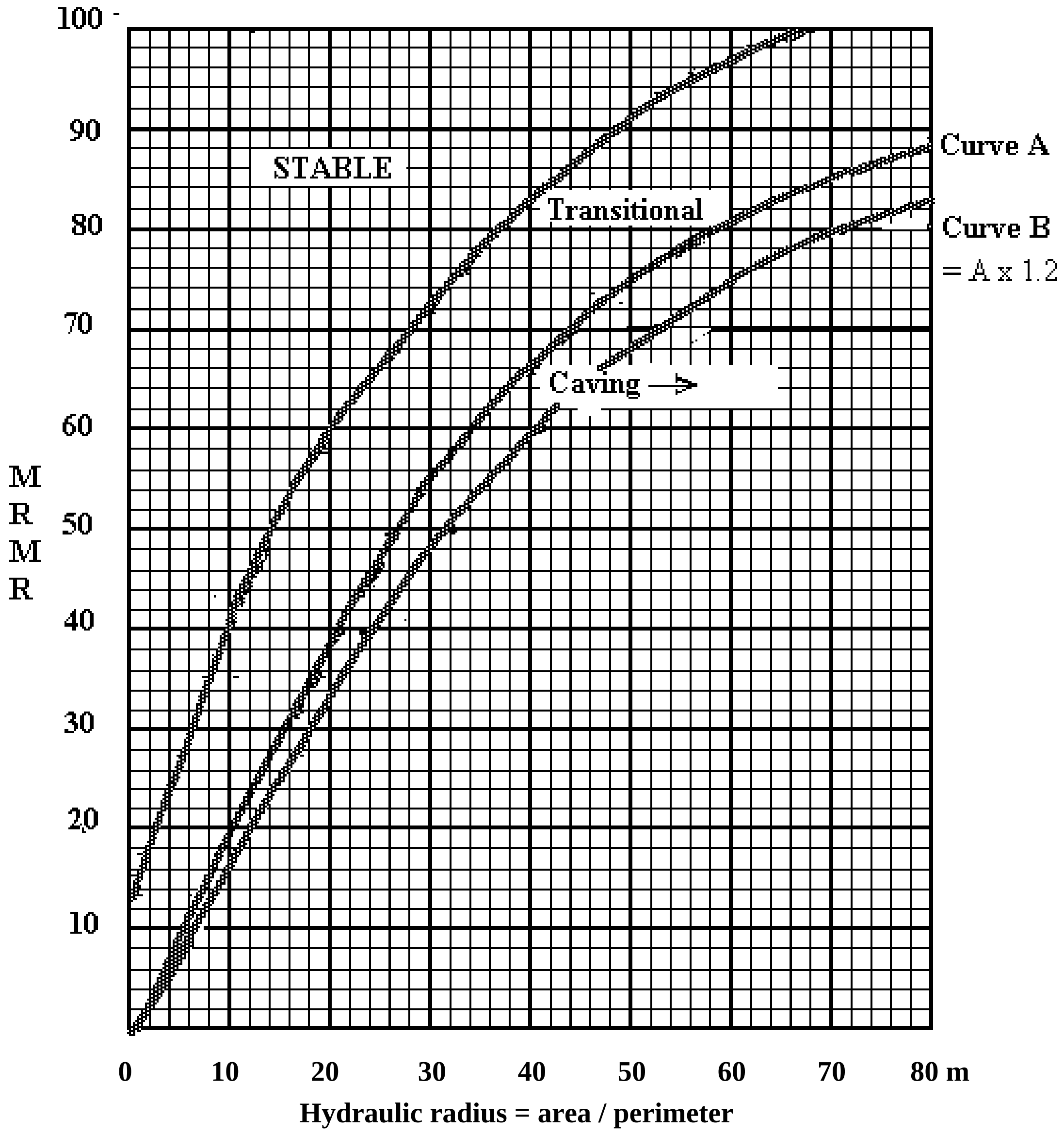


Whilst the MRMR should recognise the induced stresses, an equi-dimensional shape will be more stable than a rectangular shape owing to the 'hoop' stresses, particularly in a horizontal stress environment Based on the experiences at Northparkes, in the Kimberlite pipes and the lack of caving in certain equi - dimensional blocks at Shabanie mine then the empirical cavability diagram of MRMR vs. HR should make provision for a shape factor which will recognise the overhangs that can form in the corners. The ratio for an equi-dimensional shape is 1 ± 30% to 1.

The stability effect of the circular shape and the 'hoop' stresses can be catered for by increasing the MRMR. However, by having the two curves might make it easier to arrive at hydraulic radius and would, in fact sound an immediate warning.. The 'equidimensional shape factor should be a ratio of 1 ±.30%.



The following diagram shows two curves- curve A for rectangular orebodies and curve B for orebodies that are equi- dimensional.



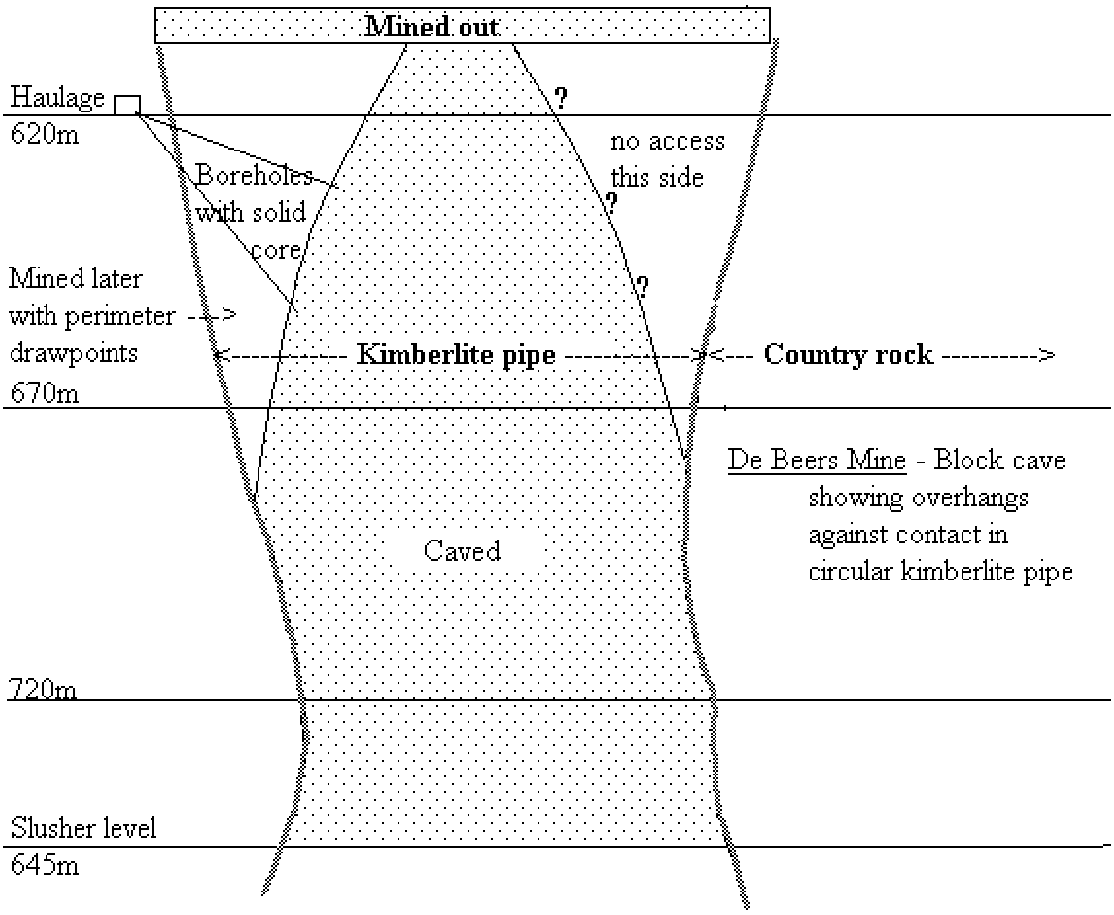
Shallow Depths

At shallow depths the limited mass of the overburden might result in a settling of the jointed rock mass into a stable arch rather than collapse. In the case of Northparkes, the mining of the open pit removed weight from the centre encouraging the formation of a dome.

Overhangs

Overhangs form in structurally unfavourable areas and / or in corners and re-entrants with clamping stresses.. The overhang effectively reduces the hydraulic radius of the cave back as the cave arches into the weaker rockmass. The following section through the De Beers kimberlite pipe in Kimberley shows an arch that broke into the mined out area. The overhang was subsequently mined with perimeter drawpoints.

The following section is through a Kimberlite pipe and shows overhangs against the contact.



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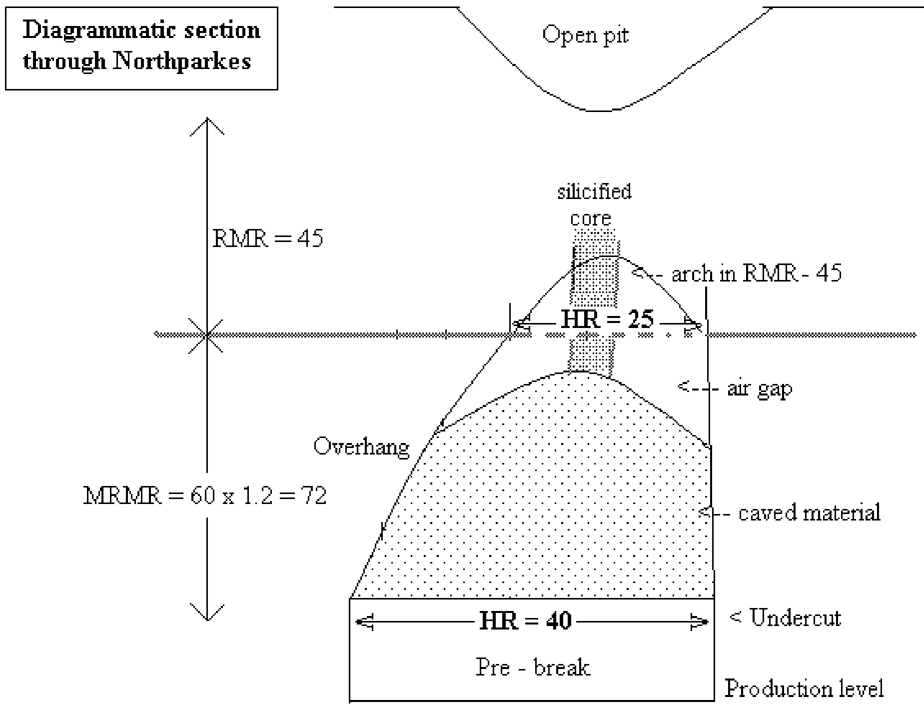
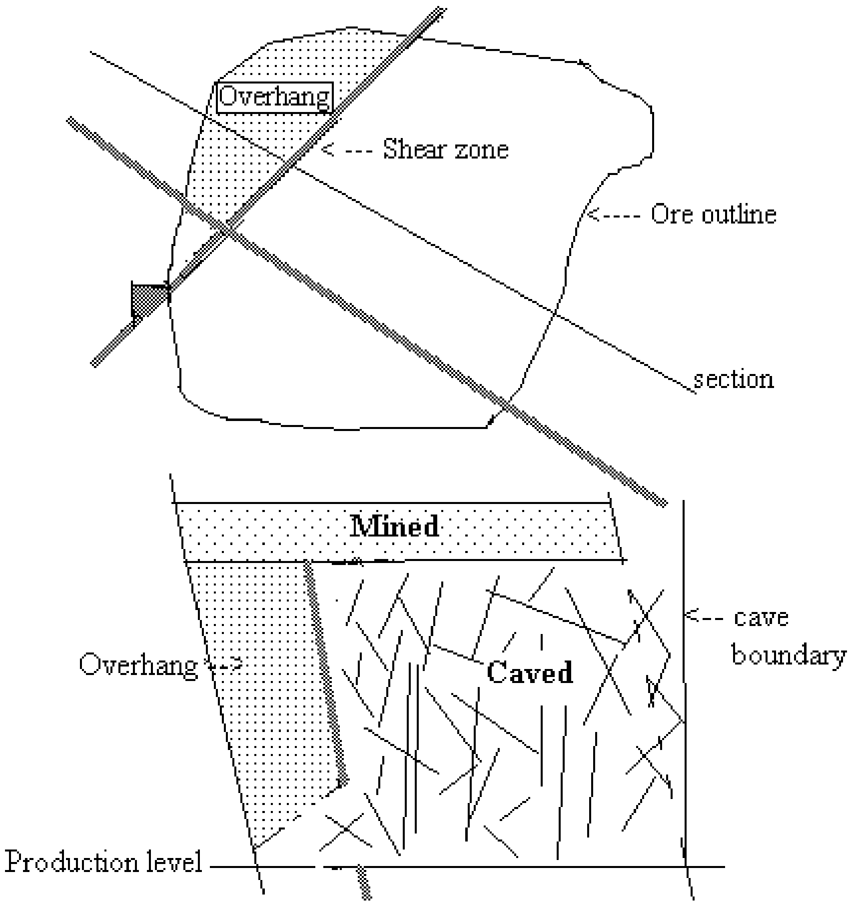
The hydraulic radius on the base cannot be applied to weaker rock higher up if an overhang has formed. In the following section through Northparkes, the MRMR at the undercut could be 60 with an HR of 40, however, an overhang - buttress can mean that the HR at the base of rock with a MRMR of 45 could be

25. The hydraulic radius of 40 on the base cannot be applied to the upper zone (dome) with a MRMR of 45 because that area is not available to it. The following diagram is a section through Northparkes.



There are many examples of permanent overhangs with continued caving to the side owing to a weaker rockmass in that area. In the following diagram in the south-west corner of King mine the more competent corner zone was bounded by major shear zones along which caving occurred, thereby isolating that section of the orebody



Role Of Major Structures

The empirical Laubscher hydraulic radius graph provides the operator with a 'ball park' figure on the cavability of the deposit. The accuracy is a function of the homogeneity of the deposit and the reliability of the input MRMR data. The friction properties / shear strength - of joints and major structures play a very important role in whether an undercut area will cave - these properties are recorded in the Joint Condition section of the IRMR classification and can be related to the angle of friction. The major structures can be the determining factor in assessing cavability, particularly when the MRMR numbers are high.

The MRMR assigned to a deposit does not give sufficient emphasis to the role that major structures play in determining cavability, as they are often included in the drift assessment, for example a narrow fault forming the boundary does not significantly influence the IRMR of the preceding 100m of ore. However, on a mine scale, the spacing, the joint condition and orientation of the major structures with respect to the principal stress and the magnitude of the principal stress are very important factors in modifying the hydraulic radius based on the overall MRMR.

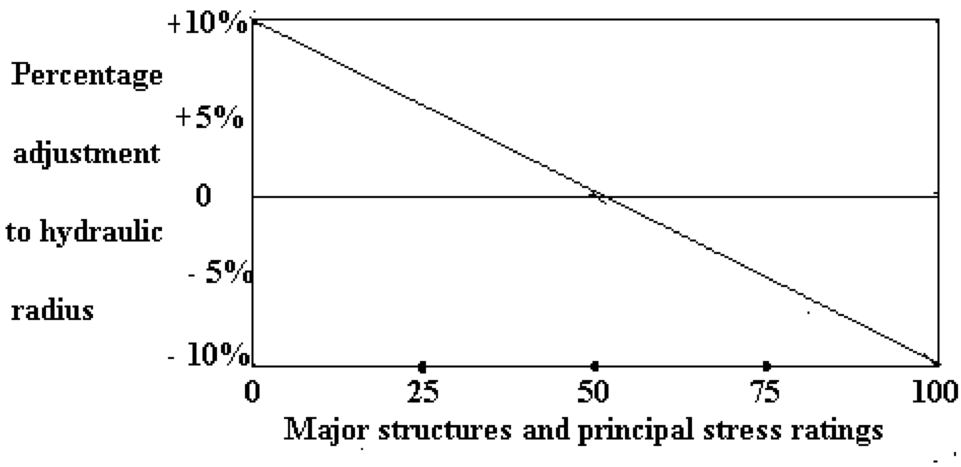
The influence of major structures will be greater in competent orebodies than incompetent orebodies which cave readily. The various factors that contribute to the 'weakness' of a major structure and therefore can influence cavability have been identified and are ranked as follows :-

A - **Dip** : 0º - 20º = 6, 21º - 40º = 4, 31º - 40º = 2, 41º - 60º = 1, > 61º = 0
B - **Spacing**: 0 - 9m = 6, 10 - 15m = 4, 16 - 21m = 3, 22 - 27m = 1 > 27m = 0
C - **Joint Condition**: 0-10 = 6, 10 - 15 = 4, 15 - 20 = 2, 20 - 25 = 1, > 25 = 0
D - **Stress / structure orientation**: 0º - 20º = 7, 21º - 30º = 9, 31º - 40º = 6,
41º - 50º = 3, 51º - 60º = 2, 61º - 70º = 1, > 71º = 0
E - **Distance of major structures from undercut boundaries** : 0 - 9m = 12;
10 - 20m = 8;   21 - 30m = 2; > 31m = 0.
F - **Stress values - Sigma 1 as % of RMS**: > 100% = 14, 80% - 99% = 12,
60 - 79% = 8, 40- 59% = 4, 20 - 39% = 2, < 20% = 0

The rankings are plotted in the following table.. - the highest likely ranking from the three sets is in the order of 100. Form for determining the major structure influence number:-

Major Structures	A	B	C	D	E	F	Total
Set 1							
Set 2							
Set 3							
Total							

The total of the rankings when plotted in this figure will indicate whether the hydraulic radius is acceptable or whether it should be adjusted up or down.



If there are other features, such as internal silicified zones, that might contribute to stability then a deduction should be made, in this case the magnitude of the deduction should be from 15% to 40% of the hydraulic radius.

Direction Of Undercutting

Whilst it is good mining practice to mine from weak to strong rock in certain caving situation it might not be advisable. This would normally be the case if there were no problems in the cavability of the material. However, where the strong material might be left against a boundary and in an unfavourable stress environment because the other material has caved then the sequence would be to start in the strong material. This would allow the stresses to build up in the strong material and also there would be time for caving to occur. Potential damage to the weaker material is avoided by advance undercutting and proper support

In a specific example at San Manuel, advancing an undercut from weak to strong rock led to caving problems and coarse fragmentation, however when the undercut direction was changed to strong to weak rock, caving did occur and fragmentation improved, the induced stresses had longer time to act on the rock mass. At Havelock Mine advancing a SLC from the weak footwall to the competent hangingwall led to high abutment stress problems in the retreating drifts and the hangingwall access drift. The problem was solved by retreating diagonally across the orebody. Good mining practice means that mining should be from poor ground to good ground.. A good example of undercutting in an unconventional direction is at King Mine. Here in the incline drawpoint layout undercutting is from the lower level upwards - a ‘A’ as opposed to the conventional ‘V’. The reason for this was to have the lower section caving as early as possible to promote the propagation of the cave into the hill. With good control no problems were experienced.

Hydraulic Radius To Propagate The Cave

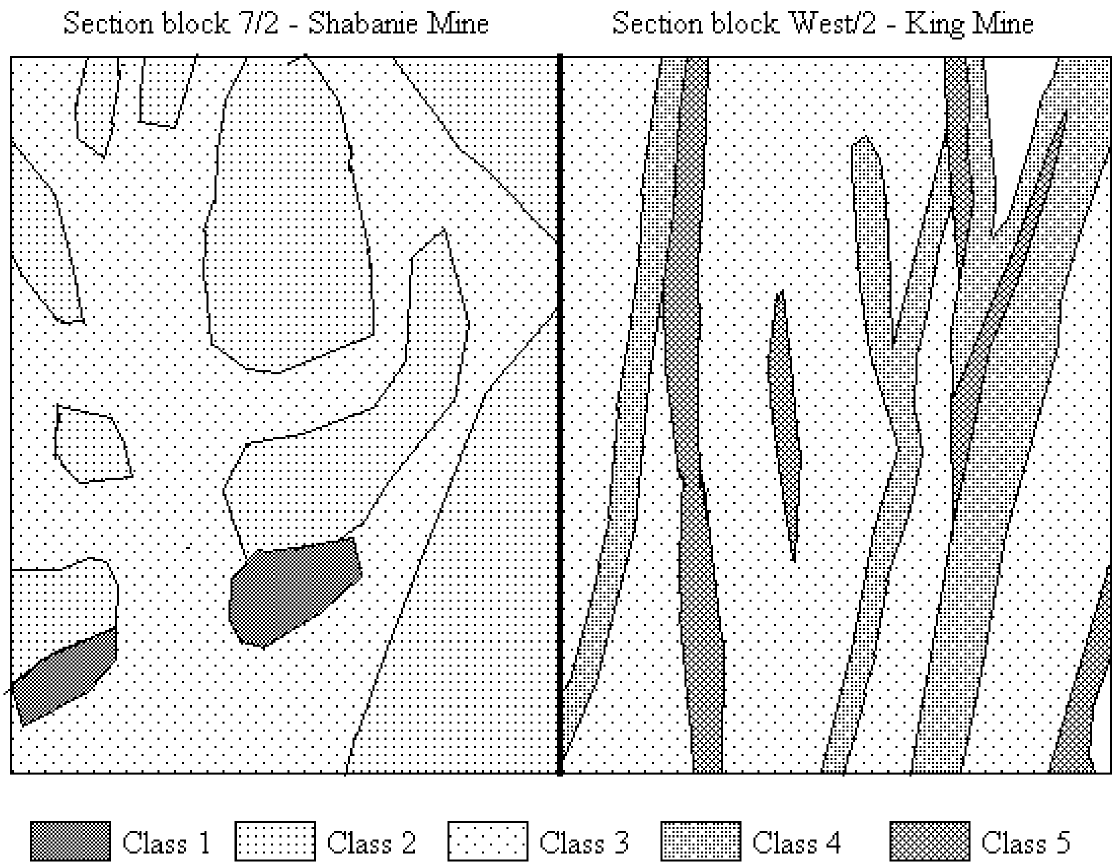
The hydraulic radius to propagate the cave must be based on the highest MRMR zone wherever it may be (the MRMR recognises the stress environment), the higher MRMR might be 100m above the undercut!

Minimum Span

The minimum span was always recognised as an important factor in deciding on cavability. It has been shown that the hydraulic radius recognises the minimum span and area.

Rockmass Strength - Irmr Classes

It is necessary to record for design purposes the range in RMR in the orebody, hangingwall and peripheral zones. The continuity is important. Average values are fine for initial assessments, but, can be misleading if there is large range in RMR and there are large areas of high RMR which could form buttresses for the arch legs of the weaker material or overhangs in the boundary areas. In those orebodies with a range in MRMR ratings, the onset of continuous caving will be based on the lower rating zones if these are continuous in plan and section. This is illustrated in the following figure showing sections through the asbestos deposits of Shabanie and King mines. In the case of Shabanie the pods of class 2 rock are sufficiently large to influence caving and the cavability should be based on the rating of these pods. At King mine the class 5 and 4 zones are continuous and determine the cavability.



GEOMECHANICS CLASSIFICATION DATA

Rate Of Caving

All rock masses will cave. The manner of their caving and the resultant fragmentation size distribution need to be predicted if cave mining is to be successfully implemented. The rate of caving can be slowed by controlling the draw as the cave can only propagate if there is space into which the rock can move. The rate of caving can be increased by advancing the undercut more rapidly but problems can arise if this allows an air gap to form over a large area. In this situation, the intersection of major structures, heavy blasting and the influx of water can result in damaging airblasts. Rapid, uncontrolled caving can result in an early influx of waste dilution. Good geotechnical information as well as monitoring of the rate of caving and rock mass damage is needed to fine tune this relationship.

The formula - **RC > RU > RD** means that the rate of undercutting - **RU** - is slower than the rate of caving - **RC** - but, faster than the rate of damage in the undercut drifts - **RD**. In other words pay attention to all aspects of the caving process, for once the process is set in motion the only control is rate of undercutting and rate of draw.

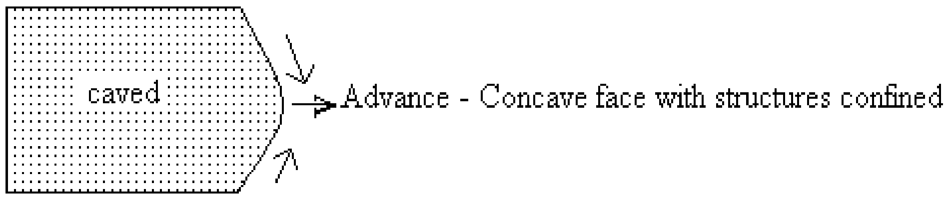
Numerical Modelling

To date mathematical modelling of the cavability of an orebody has not been too successful. maybe the modelling is not capable of coping with the four dimensions, this does not mean that we should not persevere with modelling.

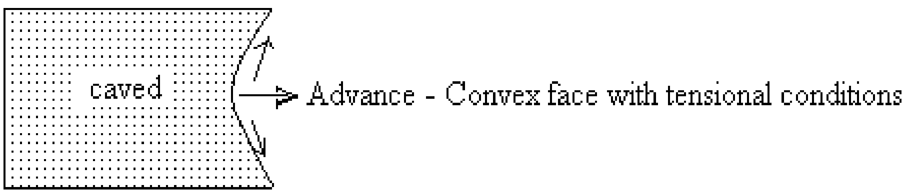
SHAPE OF CAVE FRONT AND DIRECTION OF ADVANCE

Shape

A concave (towards the solid) undercut face, provides better control of major structures, but, will also reduce the cavability.

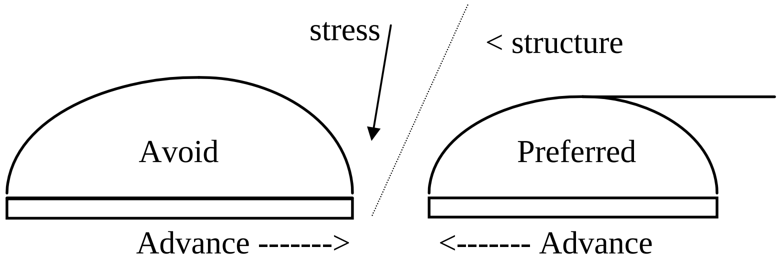


A convex face is less stable both in the cave back and on the undercut and production levels.

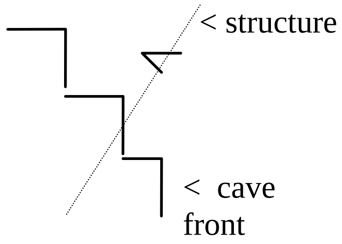


Direction

If possible, the cave front should not be advanced towards structures that could initiate massive wedge failures.



However, if the cave front has to advance towards major structures these should be crossed at as larger angle as possible:



Ratio Of Depth To Hydraulic Radius

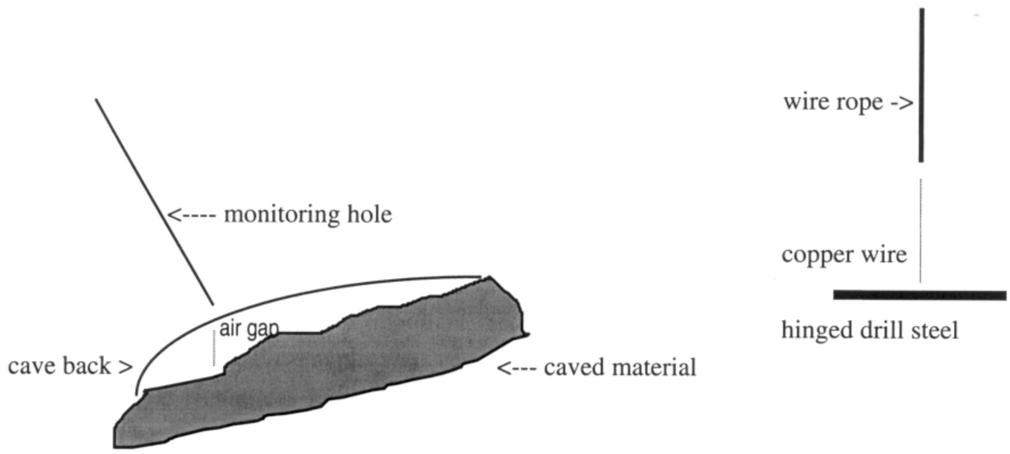
Mine	Ratio of depth to hydraulic radius
Cassiar Mine	40 : 1
Henderson Mine	35 : 1
Shabanie Mine - Bl 58	25 : 1
Shabanie Mine - Bl 52	20 : 1
Northparkes	9 : 1
Andina Mine 2nd panel	7 : 1

MONITORING

Monitoring of a cave is a very important factor and should be considered at an early stage so that boreholes that were used for exploration can later be used for monitoring.
The monitoring techniques range from the very effective simple techniques to the more sophisticated ones:-

Cave Back Open Hole

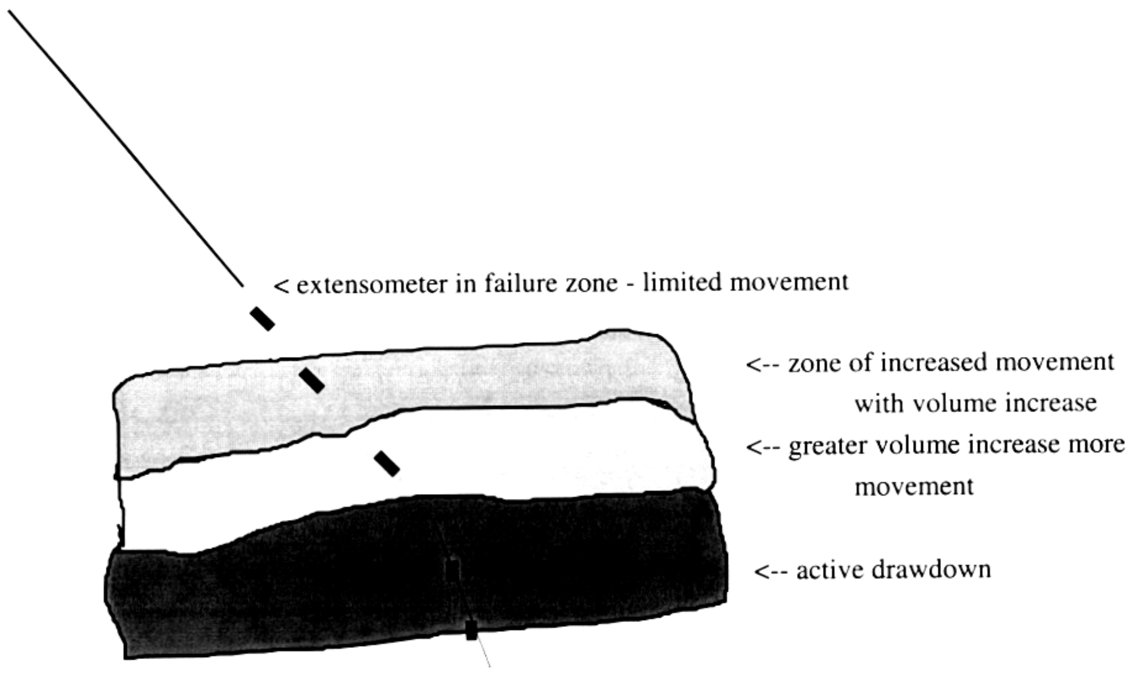
This technique consists of lowering a hinged old drill steel down the hole to hook up with the cave and also to lower the steel to the broken rock mass to determine the height of the air gap. This technique only works if there is a cave back and not a failure zone :-



The drill steel is attached to the wire rope with a length of copper wire so that it can break easily when the back fails. The steel is lowered down the hole and pulled back to the cave back to measure the position of the back, it is then lowered to the broken rock and the height of the air gap established.

Wire Extensometers

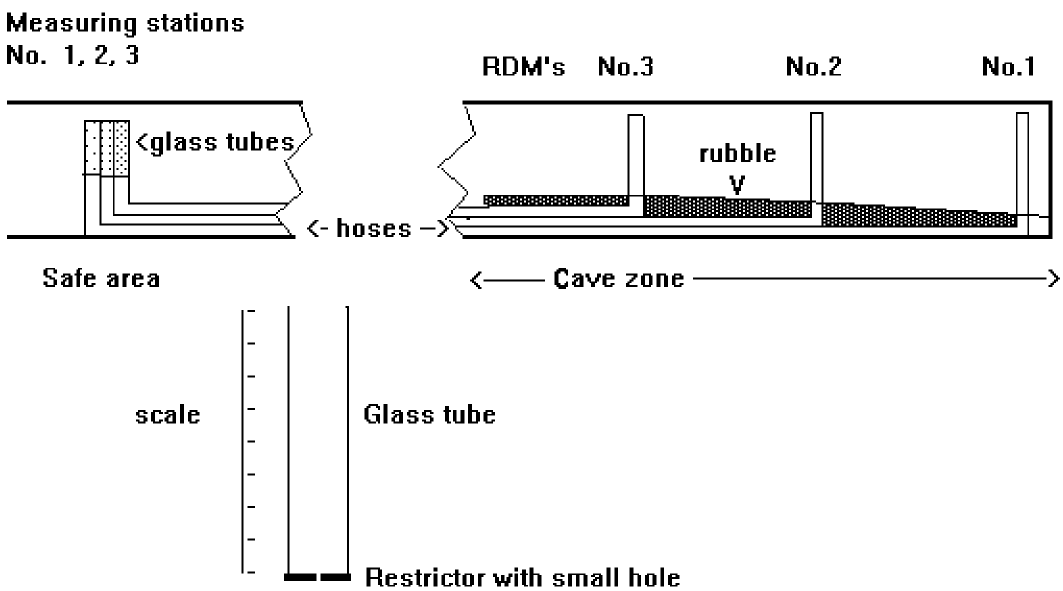
Wire extensometers are very useful devices in that they measure the extent of the failure zone as well as the cave if the cave is in contact with the failure zone. Extra wire can be added so that the extensometers are followed into the cave.



Remote Displacement Meters

Simple remote displacement meters are placed in the crosscut so that subsidence can be recorded after the crosscut is declared unsafe. These consists of lengths of temperature stable hose attached to sidewall in the crosscut with a measuring point in the safe zone. The measuring end of the hose consists of a glass tube fixed to a board with a scale. When the drift subsides the water level in the measuring end drops. Is important that a restrictor is placed in the bottom of the glass tube to stop surges when the system is replenished. The hose is covered with rubble.

REMOTE DISPLACEMENT METERS - RDM's



These devices can also be placed in shallow upholes, the inclination of the holes will depend on the elevation of the measuring site. If a drill hole raise is available then the hole can be steeper or longer.

Tdm - Tmd's

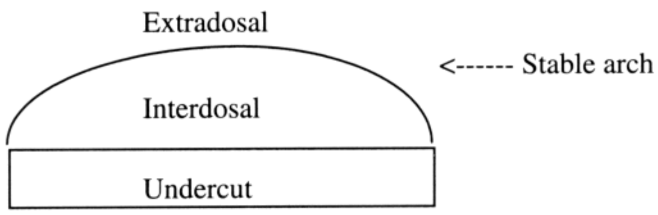
They have limited application as they only record failure or breaks in the cable.

Seismic Monitoring

Monitoring of seismic events can only be justified in high stress areas. In low stress areas the low level of noise does not reveal anything that was not known. In high stress areas rapid cave propagation leads to seismic events and monitoring of the noise in the cave back can lead to better cave management.

Predicted Rate Of Caving, Intermittent Or Continuous

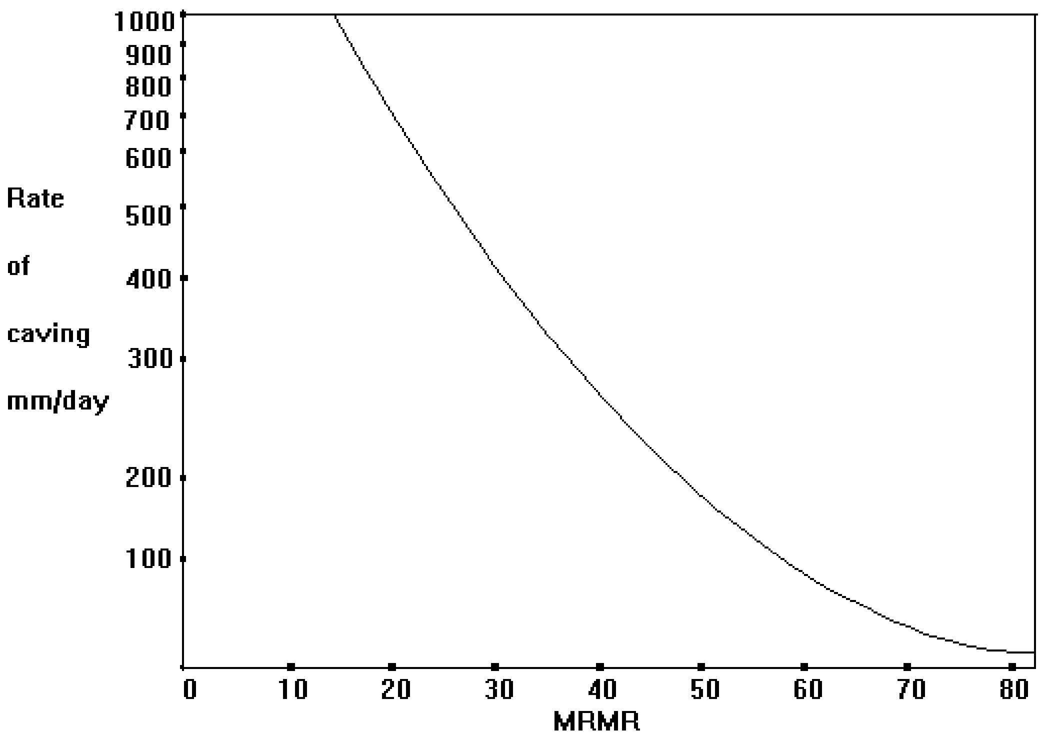
Whilst the propagation of the cave can be monitored it is necessary to predict the rate of caving and any anticipated problems. A distinction must be made between a propagating cave and the development of an arch. The old term interdosal and extradosal zones sum up the situation.



The interdosal zone contains caved material, the arch will only fail if the area is increased or boundary weakening is used either by slotting or blasting of the arch legs. The danger with boundary weakening is that a stable arch can form at the top of the boundary slots, unless, there is a change in the geology or stresses. An arch situation occurred at Premier Mine when material was not drawn on the flanks so as to stop rapid caving of the perimeter weak zone.

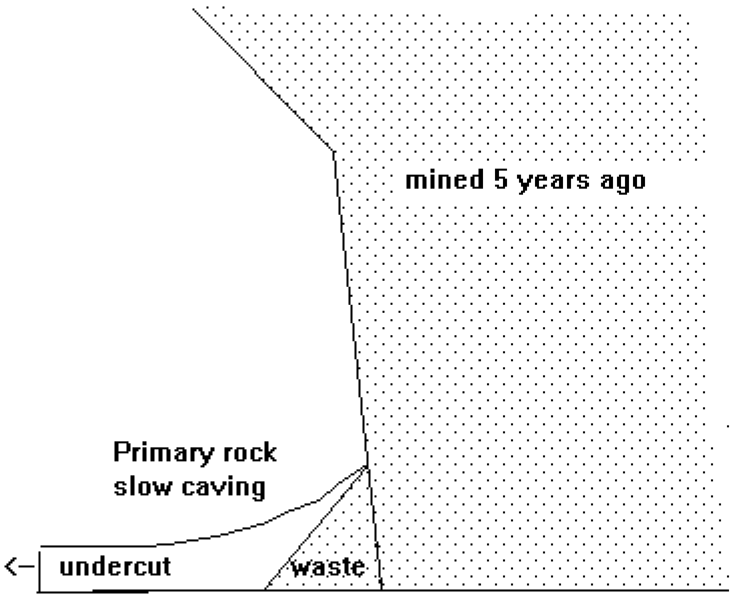
PRODUCTION RATES

Production rates have to be tuned to the rate of caving and should not be exceeded if major problems are to be avoided. The following diagram is an guide to the rate of caving with respect to the MRMR. Production rates should be based on these figures.



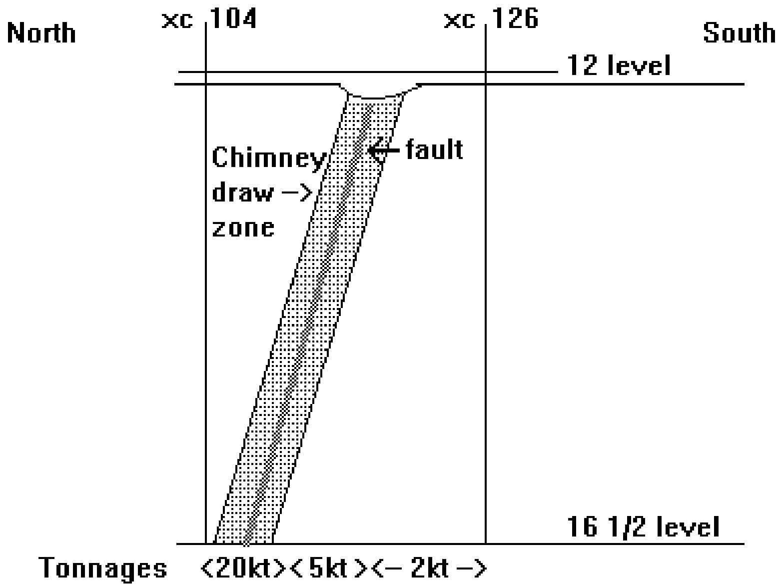
Previously Mined Areas

The effect of slow caving of primary rock -MRMR 60 - against a previously mined areas is to have waste ingress that could cut off the ore unless the draw is controlled to suit the rate of caving:-



CHIMNEY CAVES

Chimney caves occur where there is localised caving as a result of a major structure as recently experienced at Andina mine. The problem was exacerbated by an incorrect high draw on the shear zone and not under the cave as should have been the case.



SECTION ON GP - 42

CAVE INDUCTION / BOUNDARY WEAKENING

As mentioned a stable arch can be destroyed by increasing the undercut area, however, when the orebody has a finite area then some other methods have to be employed. Coyote blasts have been tried, but if the arch is very stable then this will not work. Hydraulic fracturing has proved to be successful in weak fractured rock, particularly if the water plus a low friction additive is allowed to penetrate the whole rock mass, however it was not successful in strong rock. Blasting of individual boreholes filled with water after hydraulic fracturing will result in the water being driven into the fractures to cause displacements. The hydraulic fracturing exercise at Northparkes Mine was successful in the fractured zone above the void, but, not successful in the tight overhang corner. This technique needs refinement’

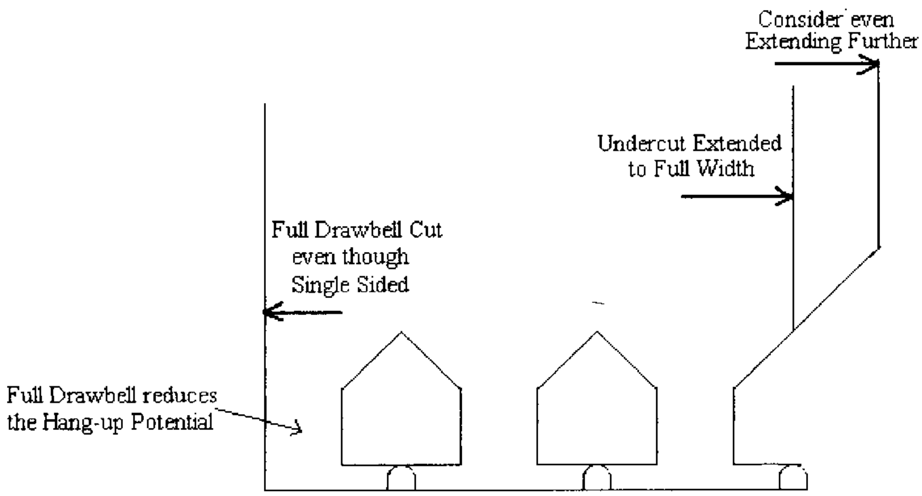
In the past on block cave mines when it was planned to mine the orebody as blocks instead of a panel retreat, the initial blocks sometimes required boundary weakening in the form of blasted slots to remove the confining stresses. Pre-splits have been tried, but where not successful if there were major structures which had a greater influence on the cave. However, if the pre-split plane was orientated at an angle to the confining stress to promote shear failure then they might be more successful.

Comments From N.J.W.Bell - Shabanie / King Mines

For small orebodies and the effects of RMR, MRMR, the hydraulic radius and structures their orientation, condition and location relative to the orebody all play a vital role.

The stresses that are likely to build in the back and on the periphery of the block are controlled by:

- the effects of vertical slots and local structures,
- the quantities of water
- the nature of the rock
- draw control
- The possibility of overhangs in more competent areas on the periphery.
- slow rates of draw from boundary drawpoints owing to friction and abutment support of the rock
- The extensions of the undercutting to take care of this and to make sure there are no tight corners or peculiar shapes. Every where these should be smoothed in order to facilitate a more uniform and consistent draw.



Extention of Undercut Area to Reduce the Overhang Potential

Direction of Undercutting

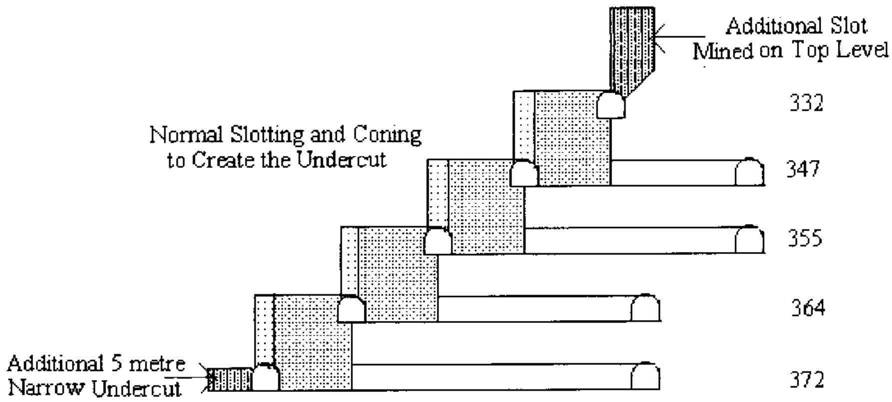
Block 53 at Shabanie was originally undercut on 540 level and pre-broken using COP 6 holes drilled from the horizon above, 480 level. This continued from west to east away from the dyke towards the hanging wall and more competent rock.

Subsequently a Sub Level Cave extended the under cut area to the east on 570 level. This gave difficulties as it was retreated in a drive direction with loads being transmitted down to the X/Cuts 15 metres below. However, once this operation was complete and the breaking was retreated on the diagonal x-cut direction no further problems have been experienced on the levels below. Except where little or no support had been installed and then damage occurred when the broken area was increased above the X/Cuts in question.

Overhangs

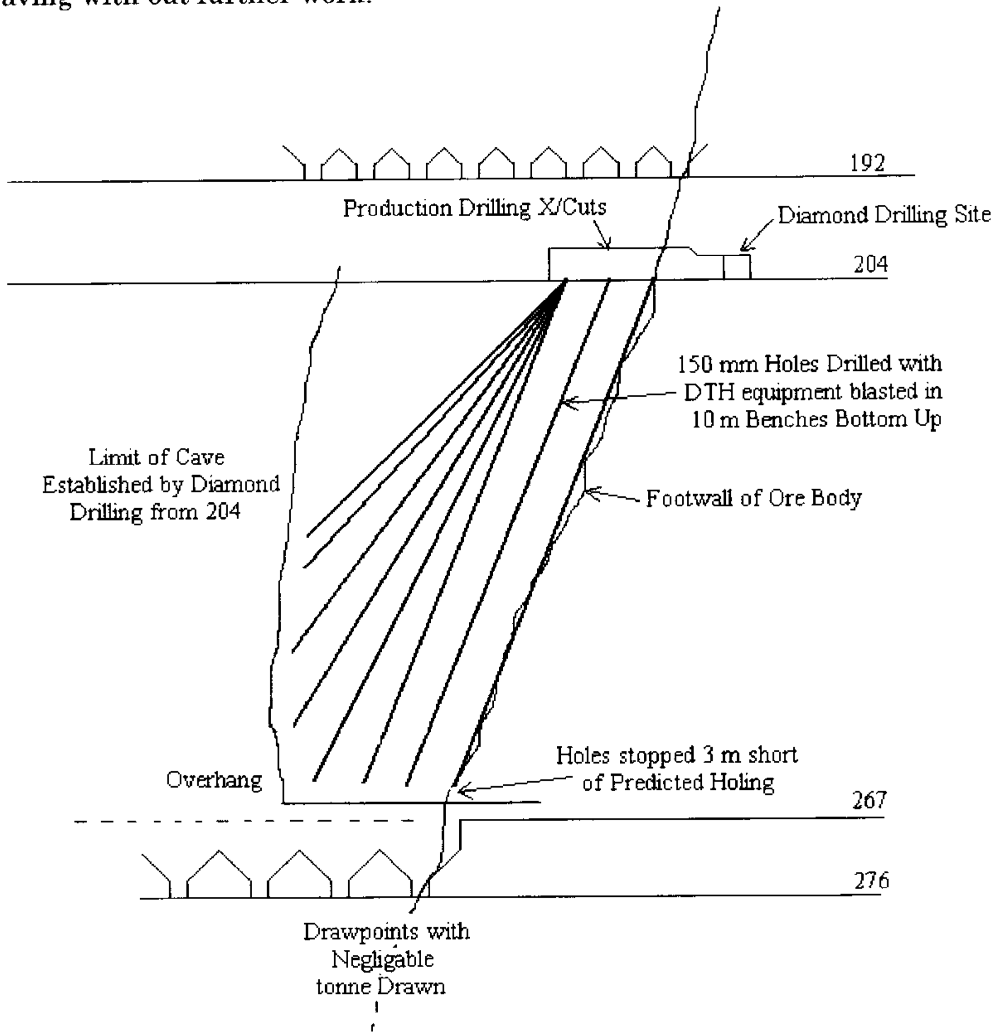
At King in order to reduce the possibility of overhangs on the main ore body an additional breaking slot was mined. This was done as a single slot drive on the level above and directly along the top loading level brow line. This was to try and encourage the cave to go to the vertical and back to the planned limit.

In addition on the lowest level, in order to try and encourage the inclined draw cave, a narrow undercut was taken 5 metres into the side to encourage the break up.



**Gaths - King Section Main over 4
Showing the Additional
Top Slot and Bottom Narrow Undercut**

An overhang did occur in W10/3 that affected and was seen from the drawpoints near the footwall where more competent ground Class 3A/2 occurs. The area was then diamond drilled to determine the full extent. As can be seen this was extensive – development was then mined and 150mm holes were drilled with DTH equipment and then charged in 10m slices from the bottom up. The final remnants around 204 level caving with out further work.



W 10 / 3 Footwall Overhang
and Subsequent Long Hole Drilling and Blasting to Wreck it

Predicting the rate of Caving

The cave effect can be a series of expanding caves in low rock strength areas. There can be an initial cave where the block caves at small hydraulic radiuses; however, you need to expand this area in order to get continuous cave linking vertically affecting the hard pods and to bring the ore down to the drawpoints. There has got to be a careful consideration of draw, because tonnage's available during the undercutting are readily available as is tonnage of weak flowing ground. A larger under cut area leads to more consistent draw and interaction to bring down and break the large hard pods. This is particularly the case in orebodies such as King West Central where there is predominance of weak ground with relatively hard pods.

Bulking Factors

These need to be considered carefully because for some deep and blind orebodies, like Shabanie the caving process will not reach surface. Owing to the material in the cave back bulking and filling the void and stabilize.

What effect does this have on regional stresses and what does it do to future cave blocks down dip from the one currently being mined?

In block 6 at Shabanie the bulking factor was only 20 %.

The bulking factor of loaded ore at Shabanie and Gaths Mine, King Section is from 2.78 insitu to 1.5 in LHD buckets, large granbies to 1.0 in confined spaces e.g. small granbies and skips.

Natural swell on blasting has traditionally been worked on 30% with a minimum for choke blast conditions e.g. narrow under cut of 15%.

The bulking of rock depends on the Primary Fragmentation that is generated and the range of particle sizes. A flat histogram of particle size will have a lower volume increase as the material will 'pack' better than material where the particles are all of a similar size.

DESIGN TOPIC

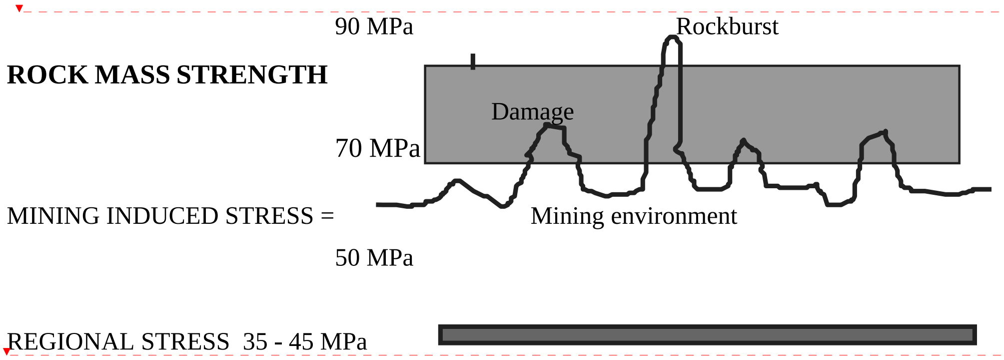
Rock Burst Potential / High Stress Damage

GENERAL DESCRIPTION

Rockbursts have become a major problem on block caving mines in competent rock, where the regional principal stress is > 35 MPa. The possibility of rockbursts occurring must be established so that the undercutting procedure, face orientation, sequence and the support can be designed.

REGIONAL AND INDUCED STRESS

In a high regional stress environment means the induced stresses can easily exceed the rock mass strength as the rock mass respond to the large excavations that are being created by cave mining. So much so that small changes or errors can result in major seismic events and associated rock bursts. This can be shown by the following diagram:-



All mining operations can be designed on the above basis. The foregoing is not the situation at the start of mining , but which may occur as mining progresses and the induced stresses increase .

ROCKMASS STRENGTH

Rockbursts will not occur in weaker rocks that will yield rather than store energy as the induced stress increases. Squeezing ground conditions apply as is the case in many chrysotile asbestos mines at depth.

ROCK TYPES - DIFFERENCES IN MODULUS, BRITTLE OR PLASTIC .

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Differences in modulus between rock masses can lead to failure in the more competent rock under low stress situations and violent / rockburst failure in high stress environments as the weaker rock yields and transfers stress. There are numerous examples of this happening e.g. diorite and andesite contacts. In other areas it could be weaker dykes in strong rock or strong dykes in a weaker rock.

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RATE OF CAVING

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The rate of caving has a significant impact on seismic activity as a rapid draw down places the cave back in a high induced stress situation.. This has been confirmed by experience on Teniente mine. The records show that the increase in seismic activity to the east of Isla, could be related to a high rate of draw in Ten -4 Sector D. It is also suggested by the Geology department that a new fault zone - Falla Portezuelo - forms the eastern boundary of Sector D and Isla. The inference is that the rock mass response in Sector D is transmitted along Portezuelo falla and materialises as seismic events in an area close to a mining operation where the induced stresses are at a higher level, that is, placing the rock mass closer to the critical state. There much evidence on the Asbestos mines that rock mass response to mining operations can be measured in faults up to 200m away on the same level.

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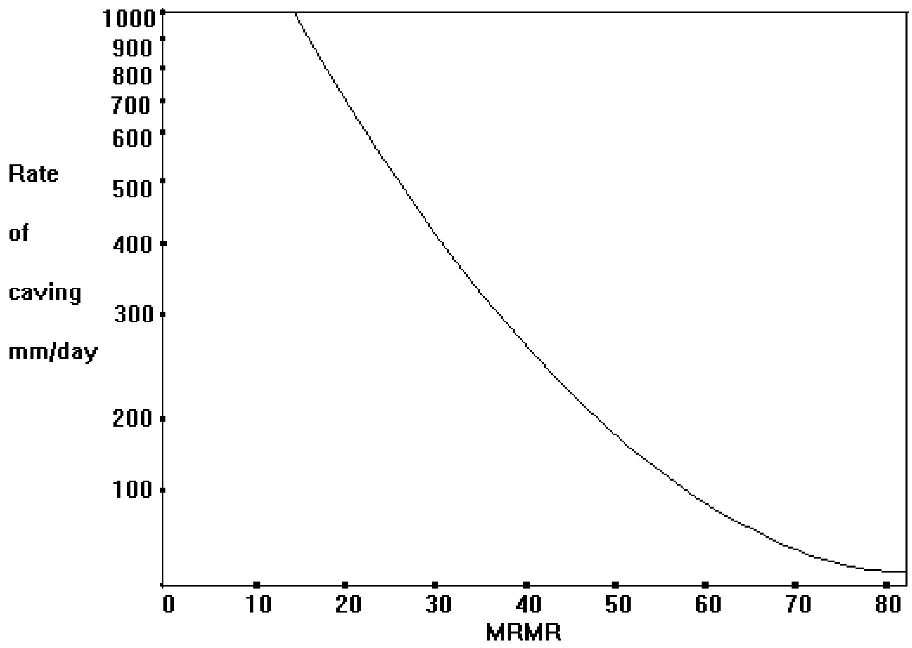


DRAW RATE LESS THAN RATE OF CAVING TO ENSURE THERE IS NO SIESMIC ACTIVITY IN THE CAVE BACK

UNDERCUTTING

It might be advantageous to repeat the principles of advance undercutting, namely to avoid damage to the extraction level by developing the drawpoints and drawbells in destressed ground. The undercutting techniques can vary from narrow `longwall' stopes to SLC operations. The narrow `longwall' stope with no or limited muck removal is favoured for the following reasons:-

- In a high stress environment the narrower the stope the lower the energy release.
- It has been shown on South African gold mines that backfilling of stopes decreases the abutment stresses, thus by mining the narrow undercut under semi-choke conditions the undercut is effectively backfilled until the drawbells are commissioned.
- It has been shown at Teniente that the level of seismic activity is related to the extraction rate or the rate of propagation of the cave or the rate of increase in the size of the excavation. By blasting single rings the undercut is advanced in a controlled fashion and the rock mass can respond in a gradual way.
- The shape and orientation of the cave front need not be the same as the draw face. There might be the situation where the undercut should be advanced at right angles to a contact but the draw area can have a triangular shape so as to use the higher stresses to promote caving and fragmentation.

REMNANTS

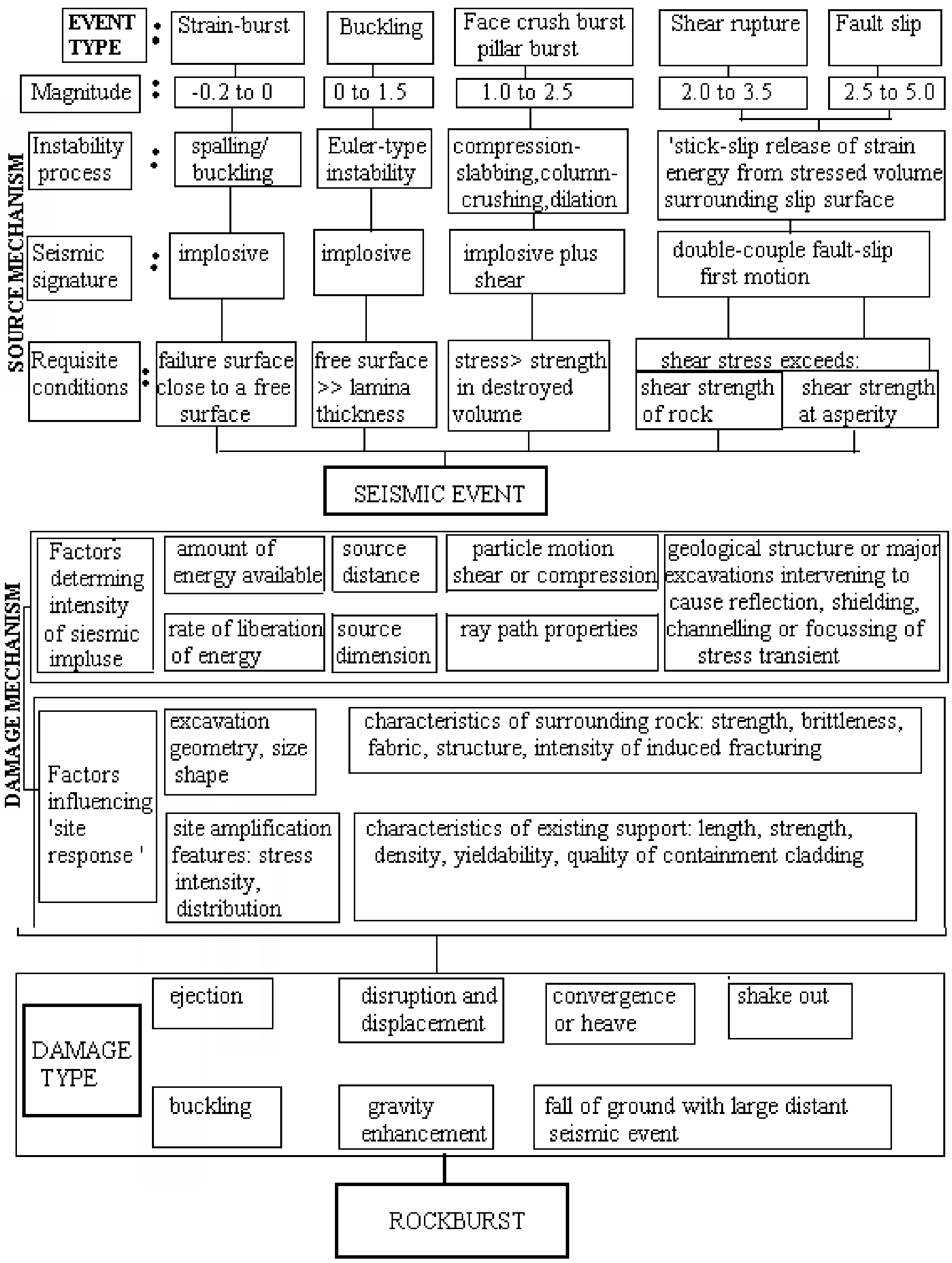
The old axiom that the undercut must be complete before it is advanced is very true. However, there is often a rather sloppy approach to undercutting and some wishful thinking that the undercut is complete. Every precaution must be taken to ensure that there are no pillars left. It has been said ` Do not worry that pillar will crush!!', this might be the end result, but, the damage on the way to the crushing can be appreciable. In high stress areas and with brittle rock pillars can be loci for seismic events, as seen by some events on high stress mines

ROCKBURST CLASSIFICATION

The text and charts in this section have been taken from *Rock Fracture and Rockbursts* by W.D.Ortlepp, Monongraph Series M9 SAIMM 1997. The esential purpose of the book will be adequately served if the reader should understand a rockburst to be, simply... **a seismic event which causes violent and significant damage to the tunnel or hte excavations in the mine.** "A simplified classification system has been proposed by Ortlepp and Stacey (Rockburst mechanisms in tunnels and shafts - *Tunnelling Underground Space Technology.* Vol 9 No. 1, 1994) Although the magnitudes of the smaller events are essentially guesses, it should be noted that the energy range possibly extends over ten orders of magnitude. It would seem most unlikely that one simple mechanism could cover this entire range.

Seismic event	Postulated source mechanism	First motion from seismic records	Richter magnitude M1
Strain-burst	Superficial spalling with violent ejection of frsagments	Usually undetected could be implosive	- 0,2 to 0
Buckling	Outward explusion of large slabs pre-existing parallel to surface of opening	Implosive	0 to 1.5
Face crush / pillar burst	Violent explusion of rock from stope face or pillar sides	Mostly implosive, complex	1.0 to 2.5
Shear rupture	Violent propagation of shear fracture through intact rock	Double - couple shear	2.0 to 3.5
Fault-slip	Violent renewed movement on existing fault or dyke contact	Double - couple shear	2.5 to 5.0

Rockburst Flow - Chart



ROCKBURST SUPPORT

The reader is referred to a paper by Stacey and Ortlepp presented at Massmin 2000 which describes the rockburst support systems.

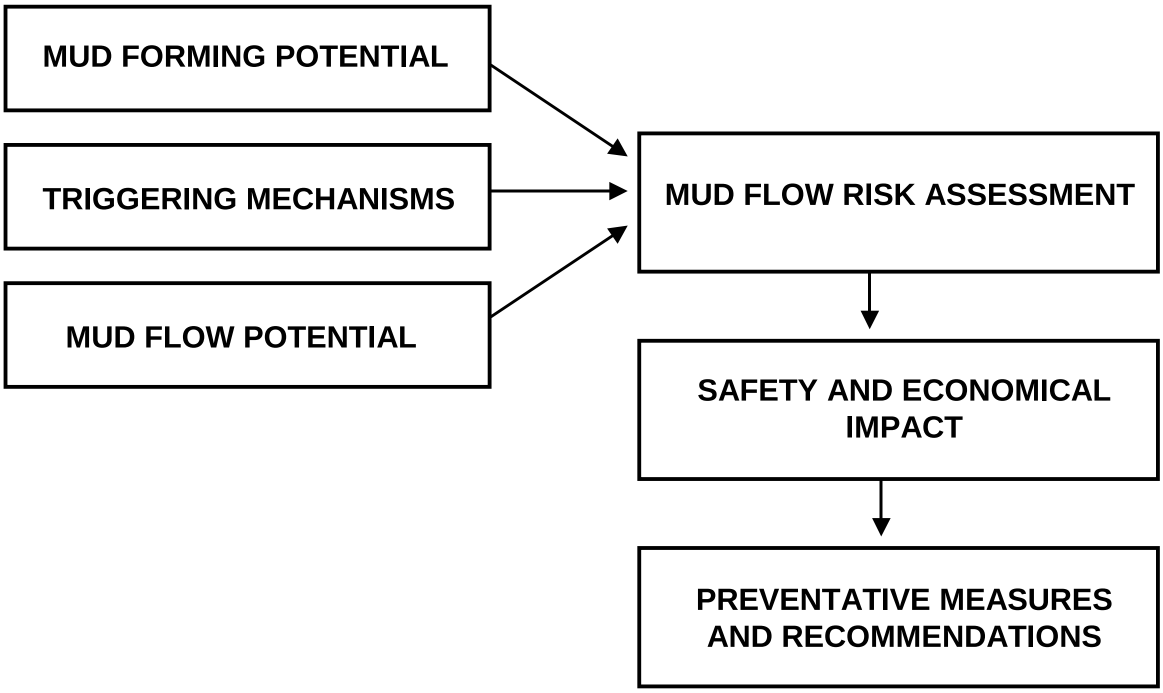
ASSSESSMENT

DESIGN TOPIC

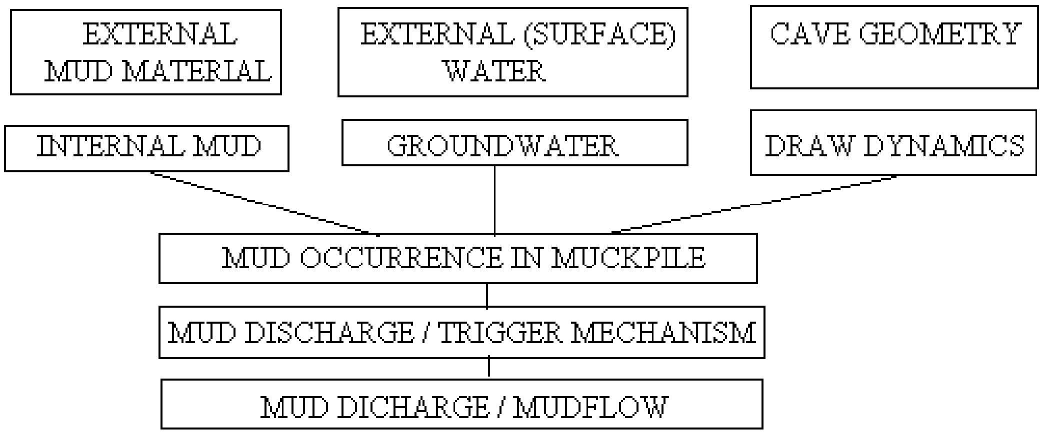
Mud Flow / Mud Push and Water Inflow Potential

Mud flows have occurred in several block caving in totally different environments, for example, the semi - arid area of Kimberley, South Africa and the high rainfall area of the Philippines. There are two basic types of mud sources; external and internal. A distinction must be made between mud flows as a result of caving into a man made hazard such as the tailings at Mufalira or mud concentration in an old pit at a small diamond mine in South Africa and mud occurring naturally as part of the caving process. However in both cases the results could be disastrous – mud entering the underground workings putting men, equipment and production at risk. The mud flow risk assessment should be done for any caving operation and the investigation should include the following steps:

(i) Mud forming potential
(ii) Water
(iii) Triggering mechanisms
(iv) Mud flow potential
(v) Mud flow risk Assessment
(vi) Safety and Economical Impact
(vii) Preventative Measures



A typical flow chart of the investigation is illustrated below;



MUD FORMING AND MUD CUMULATING POTENTIAL

The potential source of mud needs to defined, in high rainfall areas it will be any finely decomposed material at or near surface. In semi-arid areas it is a mud with certain low friction characteristics. When the mud source has been identified the mining operation can be designed to minimize the effects. Any previously mined areas, such as old pits and cut and fill stopes must be viewed with caution. Part of the mud source investigation is:

I. Mine geometry
II. Surface Topography

III. Geology/Lithology

IV. Subsidence Predictions

Cave material properties:

* Weathering susceptibility of the lithological units
* Fragmentation of the caved material
* Fines forming potential of the caved material
* Water absorption and plasticity of the caved material
* Permeability of the caved rock mass
* Viscosity of the various caved materials (mixtures)

Mine Geometry

The overall mine and cave geometry should be assessed. Location of old workings with respect to the potential cave failu**re zone is also very important.**

Surface topography

Surface topography should be investigated in terms of location of potential source of mud (e.g. slime dams, water reservoirs).

Distance from the potential source and pathways on which the mud or water can enter the cave area should be also investigated.

If the surface of the cave is irregular with hills and valleys, then mud can collect in valleys to form mud pools. `Rat holes' have intersected such features and there have been mud rushes documented in these cases. Potential for preferential mud flow should be assessed.

Geology and Lithology

Weathering susceptibility of the individual lithologies intersected by the cave should be investigated. The time of disintegration has to be relative to the duration of the mine operation. In the same way the infill of large-scale structures (e.g. fault gouge) could be a source of mud forming materials. Structural geology is important also in terms of water pathways (surface and groundwater).

Subsidence Predictions

It is important to assess the subsidence zone around the cave in terms of potential intersection of various lithologies, large scale structures, aquifers and sources of mud or water on surface. In many cases the mud prone layers (clay rich soils and sediments) could be striped away from future crater area.

Cave Material Properties

The composition of the mud must be established so as to determine whether it is a likely threat in the situation where pockets of mud can be squeezed out or extruded under pressure.

I. Weathering susceptibility of caved material
II. Fragmentation of the caved material
III. Fines forming potential of the caved material
IV. Water absorption and plasticity of the caved material
V. Permeability of the caved rock mass
VI. Viscosity of the various caved materials (mixtures)

Weathering Susceptibility

Weathering susceptibility of the caved material is critical in terms of generating fines or clays. Typical examples of highly susceptible rocks are Karoo mudstones from South Africa or certain kimberlites.

Fragmentation

Coarse fragmentation will allow for the rapid flow of mud. A range in fragmentation could result in pockets of mud. A uniformly fine fragmentation should result in mud being concentrated and then flowing en mass if there is a chimney cave or isolated draw.

Water absorption and plasticity

Water absorption and plasticity index will be important to assess the physical properties of the caved material. Water absorption will be important for water balance calculation.

Caved Mass Permeability

Permeability of the caved mass will be related to fragmentation and amount of fines and clays. Clay rich materials could cause “plugging” of the muckpile and water or mud accumulation in “pockets”.

Viscosity of mud

Viscosity is a function of water content, but, mud with a low viscosity but the right composition can move rapidly under pressure.

WATER - SURFACE / UNDERGROUND

Water is one of the fundamental parameter for mud flow - a certain amount of water is required to create the mud. It has to be stressed that this amount does not have to be excessive if water could accumulate.

Source of Water

There are two basic sources of water in the cave area:

I. Surface

II. Precipitation (surface runoff
III. “man made” such as mine water reservoirs etc.
IV. Sub-surface
V. Groundwater (aquifers)
VI. Water seepage
VII. Water from flooded underground excavations (e.g. Cassiar)
VIII. Water from hydro-fracturing
IX. Water from unplugged drillholes

In case of surface water it is important to investigate the volume and intensity of precipitation – basic balance between water loss (evaporation, evapotranspiration etc.) and infiltration. Even in the arid area the water entering the cave could be surprisingly high due to the intensity of the rainfall. Another aspect which has to be carefully investigated is preferential flows. For example,the tension cracks forming on the perimeter of the subsidence zone could cause large quantities of the surface water to be directed into cave area without having the chance to evaporate.

In high rainfall areas mud will be moved into the draw column and drawn down if there is a uniform draw or it could be moved into areas where it could concentrate as pools or pockets. In high rainfall areas craters are filled with waste rock to allow rain water to flow over the ‘cave’. Surface drains are an essential part of mud management. On many mines streams and rivers have been diverted, however, in some cases the diversion tunnels are too small for major floods and the cave has been flooded.

Rate of flow through cave column

Rain water will flow through the cave at a certain rate depending on the permeability and the depth. The following figures are from Shabanie and King mines :-

Provided a similar flow pattern is recorded then there are no damming problems and the chances of a mud rush are not likely.

Underground water must be led away from the cave by developing drainage tunnels and drilling de-watering holes. Large quantities of water must be controlled, not only from the mud rush aspect but also to have a clean mine and to reduce wear to roadways and to generally maintain a high level of good housekeeping which **always** leads to higher productivity.

Water Balance

One of the most important tool is a overall water balance. Comparison of the inflow and underground pumping rates could for example indicate water accumulation within the cave area. In case of mud flow susceptible mines the “wet drawpoints are good drawpoints” indicating good draining potential of the muckpile.

TRIGGERING MECHANISMS

The potential triggering mechanisms leading to the mud mobilisation has to be investigated. The following triggering mechanisms could cause the mud flow:

I. Excessive rainfall
II. Collapse of crater wall/subsidence
III. Collapse of arched material within the muck pile
IV. Irregular draw
V. External failures and inrushes (slimes)
VI. Seismic event
VII. mass blasting or hydro-fracturing

MUD FLOW POTENTIAL

Flowability of the materials within the cave area should be investigated. It is important to investigate potential “blends” of the material in the cave as mining progress. Void ratio (based on the fragmentation) and amount and physical properties of the fines are investigated. Flow under pressure or dynamic load (such as air-blast) could be different than “static” flow under gravity. There are known cases of the mud flow (under pressure) of the very stiff mud.

MUD FLOW RISK ASSESSMENT

In the mud flow risk assessment for the particular mining operation a **likelihood of occurrence** and **consequences** of; mud forming potential, triggering mechanisms and mud flowing/discharge has to be assessed and overall qualitative risk assessment of mud flow potential investigated.

SAFETY AND ECONOMICAL ASPECTS

The potential impact of mud flow relative to fatalities and economic loss has to be assess.

I. Safety - loss of lives
II. Economics - loss of reserves
- loss of production
- loss of properties

PREVENTATIVE MEASURES

Mining method / layout / uniformity of draw

It has been established on the block cave mines in Kimberly that as long as there was uniform draw there were no mud pushes. This data is from observations over a mining period of ninety years. Mud pushes occurred with irregular draw which can occur at the end o the life of the block, mining against old blocks with a mud problem or using sub level caving as a method owing to it's irregular draw pattern.

With uniform drawdown over the whole block or continuous moving strip mining there were not mud push problems. A uniform controlled draw results in fewer problems. The layout must be designed to ensure a uniform draw and draw management must be practised.

Draw rate

The draw rate must be commensurate with draw control plans for a uniform draw down. High draw rates lead to irregular draw and often isolated draw.

Classification system

On mines with a mud push potential it is necessary to devise a classification system

Monitoring

Surface appearance, composition of ore zone - mud generating, composition of dilution zone in terms of generating mud, rainfall and underground pumping correlation, underground water sources, flow rates through the cave, variations in discharge within the block - relate to RMR.

Monitoring Guidelines should include (but not to ………) the following:

I. water balance
II. draw management
III. drawpoint monitoring (water, fines, mud, oversize, convergence, operational/production)
IV. mass balance
V. crater wall stability
VI. subsidence
VII. ventilation changes

ASSESSMENT

The factors influencing mud flows/pushes are identified and the chances of a mud flow can be eliminated by spending the right level of money and by having a finger on the pulse of the operation

Comments From N.J.W.Bell - Shabanie / King Mines

Rainfall and Pumping at African Associated Mine's Shabanie and Gaths Mines

Weather

Shabanie Mine at Zvishavane and Gaths Mine, Gaths and King Sections at Mashava are in a low rainfall section of the country of Zimbabwe, where rain tends to only fall between November and April and there are very occasional heavy showers in the middle of winter July/August.

When reviewing the rainfall it is regarded that "rain" would only be classified as being significant when more than 30mm in a month had fallen. The periods covered also included major droughts when very little rain fell on the country as a whole and it became very apparent, when analyzing the data, that the following factors were significant:

i. number of months in which more than 100mm fall
ii. months in which 200mm or more fall and
iii. The length of the actual rainy season.

At all three sites the number of months with 100mm or more went from none to a maximum of four. Rainfall of greater than 200mm in a month normally did not occur and when it does only once per year, however in one year there were two months.

Pumping

Increased pumping in the active caves at King started either in the month of the rainfall or the month after. Possibly there is a two-week delay for the rain to percolate through. However, on the reduction of the higher pumping rate at the end of the season at King where the column of broken rock above the drawpoints is more concentrated, and more like a pipe, pumping on average continued three and half months after the main rains. The shortest delay was a year when the pumping stopped at the same time as the rains ended. The longest was 8 months of higher pumping, which happened to be the year before.

Increased pumping in the active caves at Shabanie started either in the month of the rainfall or the month after. Possibly there is a two-week delay for the rain to percolate through. At Shabanie of the periods looked at, actually showed no increase in pumping whatsoever, despite early normal rainy seasons. Of the years where pumping was higher and continued for 2 to 3 months after the rain ceased, say 2½ months.

At Gaths were the cave is not active and the mine has been shut in the period under review, delay in increased pumping start ranged from 0 to 4 months with an average of some 2½ months delay. The stop ranged from 0 – 7 months after the end of the rain, The average being 4 months.

Consideration

In dry climates with sporadic rain:

i. At Gaths inactive cave there is a delay and the cushioning effect appears to be longer. Obviously is needs to percolate through and is not channeled through the active cave. 2½ months from the rain when their pumping increases and continues for some four months after the rains have stopped.

ii. Where the ore is flatter and therefore you have a mixture of active and inactive caves. At Shabanie there is a slight delay as in the active cave for the rains to come through the delay in the flow through at the end of the season with the cushion effect within the column is less. At Shabanie we have dump material on the inactive cave and this tends to drain the water back to the footwall and out of the cave area.

iii. The active cave at King Section, which is pipe like seems to come through immediately. However, it does last longer and therefore there is less pumping than the actual rainfall that takes place. A cushion effect occurs in the broken ore.

Cyclone Eline

During this event Gaths Mine King Section had pools of standing water in the bottom of the cave area above the drawpoints. Water also ran through the draw column and out of the drawpoints. Both these abnormalities however cleared very quickly once the rain stopped.

DESIGN TOPIC

Primary Fragmentation

GENERAL DESCRIPTION

In caving operations, fragmentation has a bearing on:-

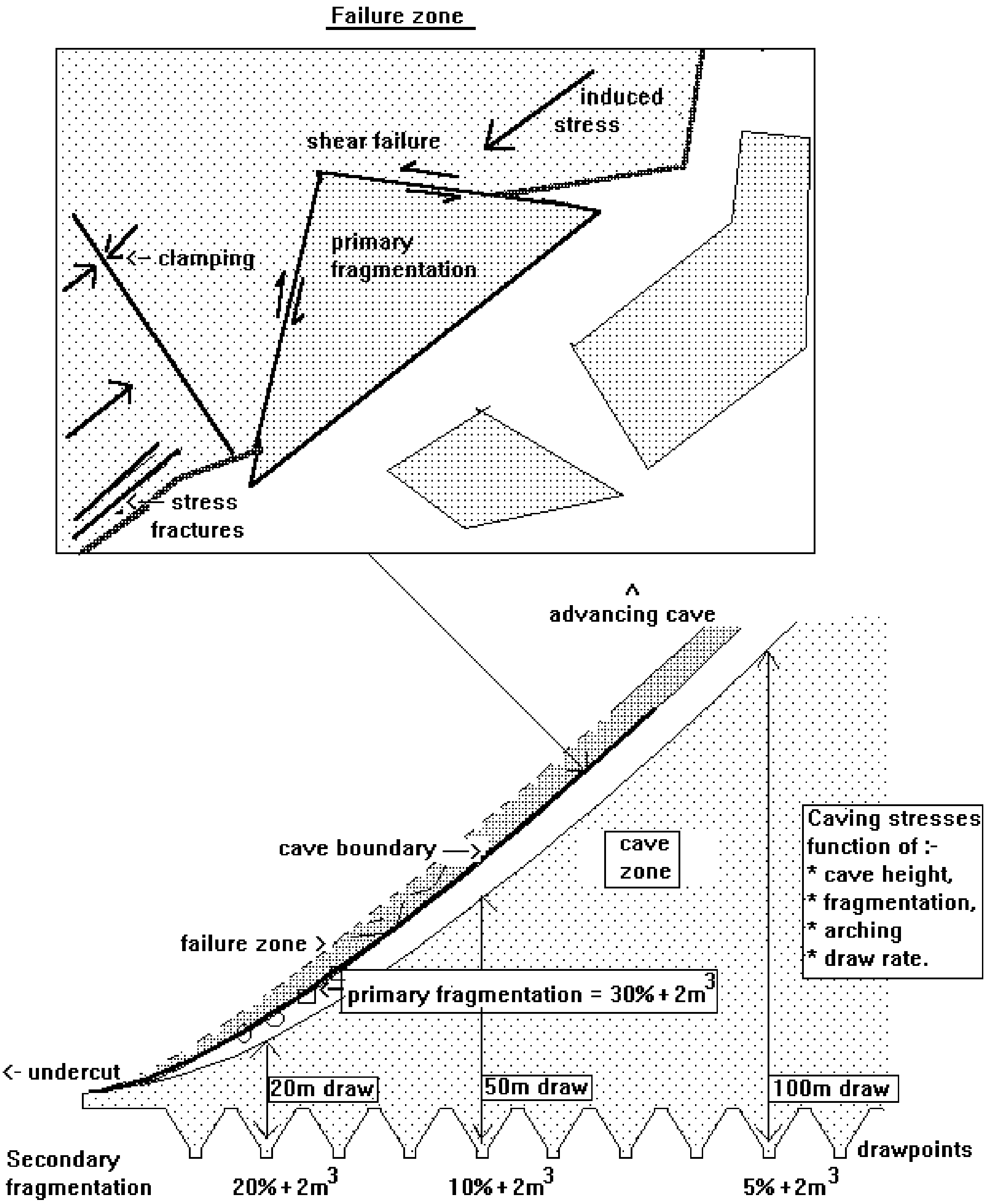
- Drawpoint spacing
- Dilution entry into the draw column
- Draw control
- Drawpoint productivity
- Secondary blasting/breaking costs
- secondary blasting damage

The input data needed for the calculation of the primary fragmentation is :-

- In situ rock mass ratings - IRMR
- Intact rock strength - this must be a realistic value and not selected cores
- Joint spacing as the mean and maximum and minimum
- Average joint dip and direction as well as the range
- Orientation of cave front
- Induced stresses

A computer simulation programme to calculate primary and secondary fragmentation has been developed. **Reference is made to the +2m³ content of the caved material, this is a standard adopted on the basis that a 6yd LHD can handle 2m³ material (provided the rest of the ore handling system can) and most LHD operations use 6yd machines. However, if the plan is to use 8yd machines then the cut-off value becomes 3m³.**

Caving results in **primary fragmentation** which can be defined as the particle distribution that separates from the cave back and enters the draw column.



PRIMARY AND SECONDARY FRAGMENTATION

The orientation of the cave front/back with respect to the joint sets and direction of principal stress can have a significant effect on primary fragmentation.

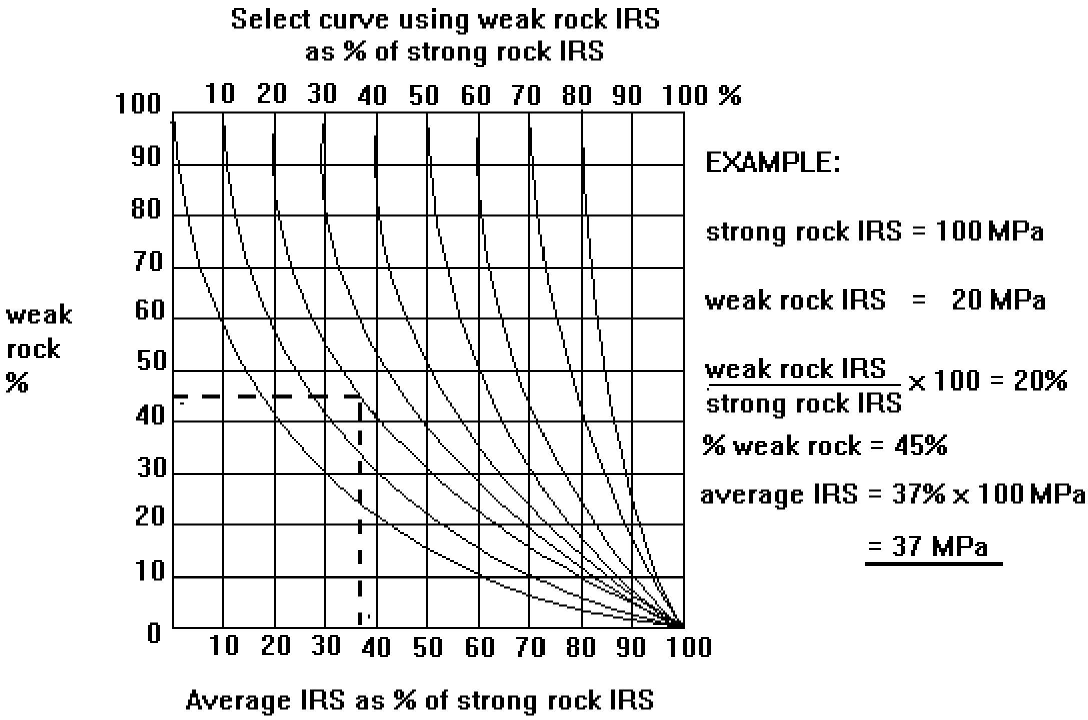
Advancing an undercut towards the principle stress will result in high abutment stresses which will induce caving and improve the primary fragmentation, but could result in damage on the undercut and production levels. Advance undercutting avoids some of those problems.

INPUT DATA FOR A FRAGMENTATION ANALYSIS

Rock mass characteristics - The following points are of paramount importance in successfully completing a fragmentation analysis:-

- The IRMR of the orebody and the hangingwall zone for at least twice the orebody height needs to be known.
- The IRMR needs to be plotted as zones if there is a range greater than 10.
- Low rating zones in a high rating zone can lead to failure of the more competent rock at low stress owing to tensile stresses in the more competent rock.
- The geological data forms the basis for the analysis and must be input with a clear understanding of the objective.
- The IRMR / RMS defines the overall rock mass strength, the IRS provides the data to determine the strength of the potential rock block as defined by the joints with the strength of the rock block influenced by the fractures/veins.
- The joint condition ratings are a measure of the frictional properties of the joint.
- The ratings also apply to the veinlets/fractures as they have an influence on the strength of the rock block.
- Serpentine and asbestos veins in a partially serpentinised dunite decrease the strength of a dunite from 120 MPa to 30 MPa. when that rock mass is subjected to stress.
- The rating adjustment for a soft vein is the same as for wall rock alteration provided the vein has not been used in the average IRS calculation

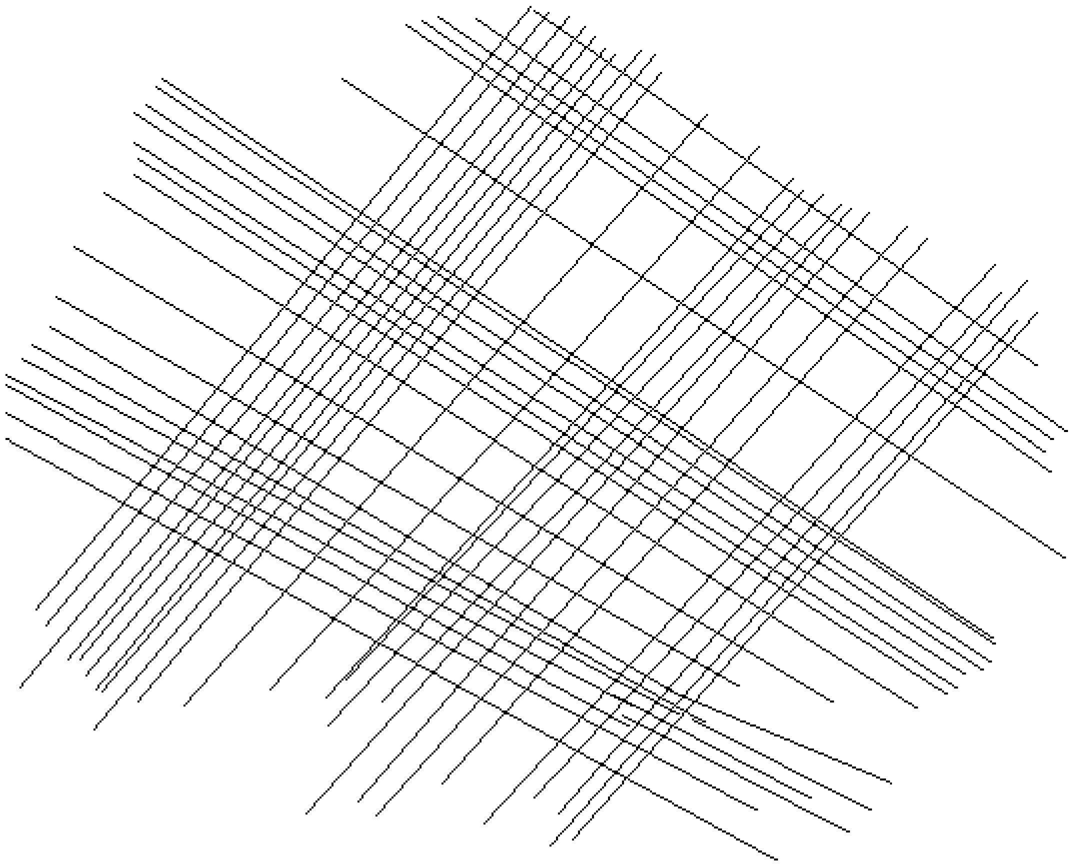
Intact Rock Strength - The IRS must also be an average value for the intact rock and not a value for selected samples representing the strongest core. The following diagram shows the technique to arrive at the average value of the IRS for the unjointed material. Weak rock refers to the weaker elements of the rock block - these sections are usually not tested for intact rock strength.



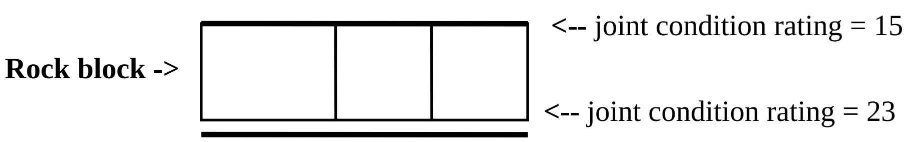
Geological Structures and Zoning of Joints - There must a good record of both major and minor structures. Major structures could define districts and they could influence the primary fragmentation by allowing more rapid subsidence. All the joint data has an important role in the fragmentation analysis, the range determines the acuteness of the corners and as such the amount of corner failure during drawdown. The minimum and maximum spacings are also critical as they specify the end members, for example, an average spacing of 1m and 0.5m minimum with a 1.5m maximum means that the largest block is not likely to exceed 2m3, but, if the maximum were 3m then the largest block could be 24m3 - a significant effect for production efficiencies.

Number And Spacing of Joint/Fracture Sets - This calls for precise structural mapping, something that is often lacking, it puts structural geology into the engineering category and presents a challenge to the geological fraternity.. It should be possible to define joint spacing or ff/m zones in the orebody. By defining the different zones the fragmentation analysis becomes more accurate, by taking an average value for the whole orebody the overall fragmentation assessment could be worse than it really is. For example the overall +2m^3 might = 60%, whereas by assessing two zones the result would be Zone A at 70% of the area = 10% and zone B at 30% = 70% +2m^3 (very coarse). Combined result = 28% +2m^3

It also identifies areas where the coarse fragmentation could call for modifications to the layout or an increase in the secondary drilling equipment. **It is imperative that the orebody is zoned according to the joint spacing.**



Joint Condition Ratings - Joint condition ratings must be assigned to individual joint sets as a set might have lower ratings than the others and therefore would always tend to fail before the others and would form a primary block boundary, whereas with the others it might only be every 2nd or 3rd joint e.g.:-



VERTICAL EXTENSION OR LATERAL EXTENSION CAVING

Effect of stress - The primary fragmentation from stress caving is generally finer than from subsidence caving owing to the failure not only along joints but often of the rock blocks under higher stress. There is stress spalling and fracturing of the rock due to differences in modulus.

Subsidence caving in a low stress environment or in a relaxation zone adjacent to a operating cave can lead to failure along opened widely spaced joints resulting in large blocks. The location of the undercut with respect to surface or previous mining

is required to decide on how primary fragmentation calculations have to be done to recognise changes in the stress environment. A cave front moving upwards towards a sill pillar below a previously mined area should enter a higher stress zone with resultant finer fragmentation.

Cave Front - Dip and Direction - The orientation of the cave face will not change as the cave propagates, but, the dip of the face can vary from a low angle at the start of the caving to the latter stages when the cave has broken through to surface. The induced stress is the strike and dip stress in the a narrow zone behind the cave face. The normal stress is the stress from the caved material acting on the solid cave face, this stress will only be significant , 0.5 to 2 MPa, if the face is vertical or dipping towards the cave. The orientation of the cave face with respect to joints and stresses has a significant bearing on the primary fragmentation. It is important to correctly assess the magnitude of the defined stresses.

The orientation of the cave front/back with respect to the joints can lead to a significant difference in primary fragmentation. The fragmentation results from the BCF program can be used to decide on the best direction for advancing the undercut to achieve optimum caving.

Cave Back - Strike & Dip Stresses - Numerical modelling will provide the right input data provided the engineer has figured out the shape of the cave back. Once again there will be reiterations as the influence of stress and structures is assessed.

Draw Rate - The draw rate is a very important factor in that it must provide space for caving, it must not be too fast to create a large air gap and possible air-blasts, it must not be too fast to create seismic activity. Production must be based on this value and not some ideal figure created by economic factors such as a short term return on investment that ignores long term consequences. There is also the question of time dependant failure - does a slow draw rate mean improved fragmentation??

BCF PROGRAM

The BCF (DOS) program was developed to determine the primary and secondary fragmentation and has proved to be a reliable tool provided the correct data is input.

PRACTICAL CONSIDERATIONS

Stresses change with depth, thus, if a high ore column is going to be drawn then the primary fragmentation for 100m intervals should be determined and the secondary fragmentation for the next 100m based on that primary fragmentation result:-

Secondary (2)

100m ________________ Primary fragmentation (2)

Secondary (1)

________________ Primary fragmentation (1)

0m

Where a cave is started below an open pit or a sill pillar in a high horizontal stress environment, the stresses in the ` pillar' increase as the pillar width decreases and this results in finer primary fragmentation.

SIZE DISTRIBUTION

A high +2m³ result with a steep curve indicates that the bulk of the large rocks are not greater than say 6m³, therefore, will report in the drawpoint and not be a problem in secondary breaking. The oversize distribution becomes a problem with flat curves where there are large blocks even though the %+2m³ is nearly the same.

ASSESSMENT

Primary fragmentation calculations must be conducted for different heights in the column and at stage when data becomes available. These assessments will indicate production problems or planning requirements.

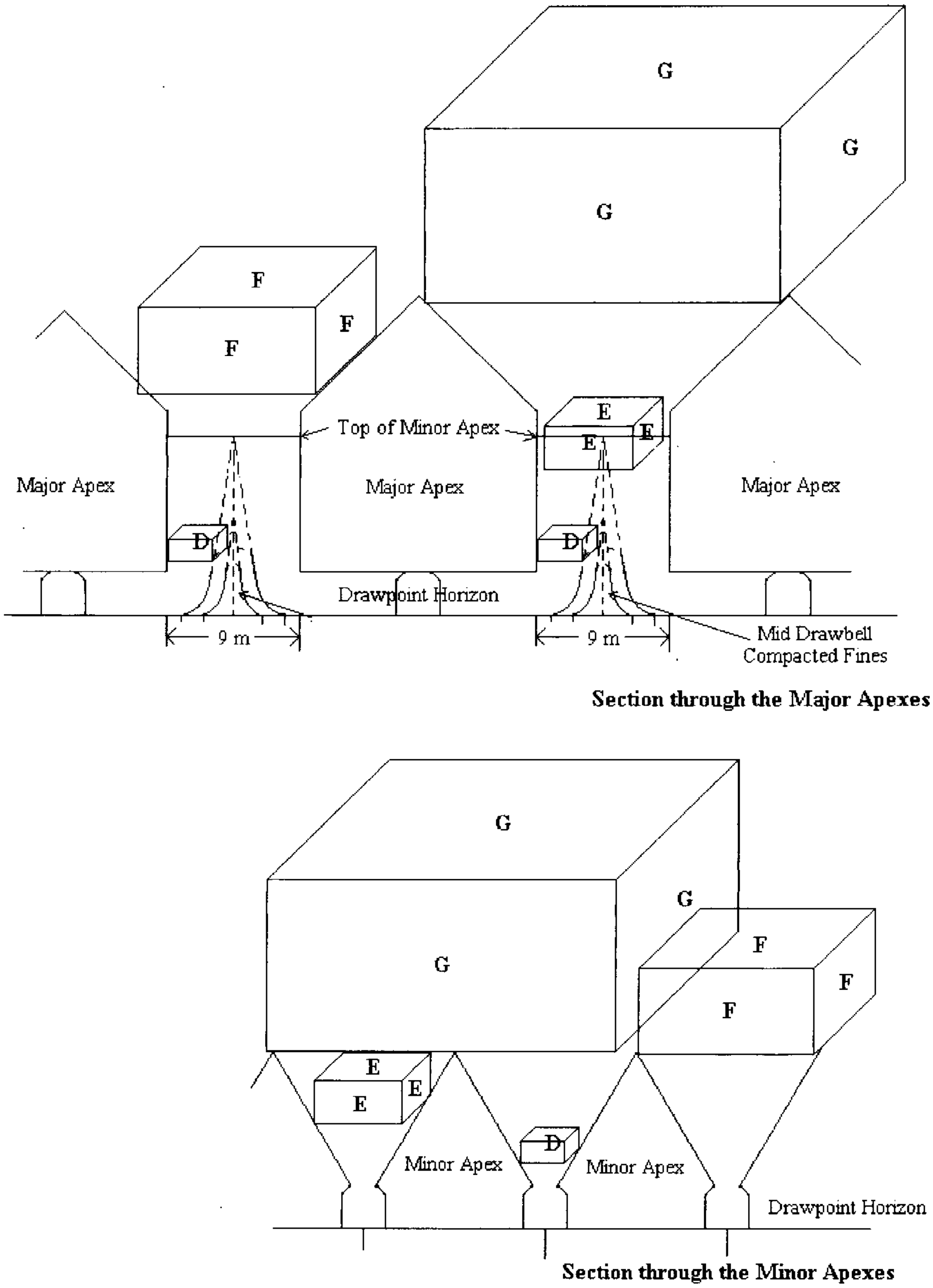
Contribution From N.J.W.Bell - Shabanie Mine

Rock Sizes

These can be defined as per the tabulation below:

Rock Sizes TABLE 1

Rock Fragmentation Size	Remarks / Potential	Length Range metres	Mean Length L metres	Mean Volume L X L/2 X L/2 m^3	Maximum Volume m^3
A	100 % through 1.5m X 0.3 m Grizzley	< .5	0.25	0.004	0.031
B		0.5 to 1.0	0.75	0.11	0.25
C	100 % into LHD Bucket	1.0 to 2.0	1.5	0.8	2
D	Hang-up in Drawpoint Throat	2.0 to 4.0	3	7	16
E	High Hang-up	4.0 to 8.0	6	54	128
F	Drawbell Blocker	8.0 to 16	12	432	1,024
G	Double Drawbell Blocker	> 16	24	3,456	Infinite



Block Sizes and Where They Will Report in the Drawbells

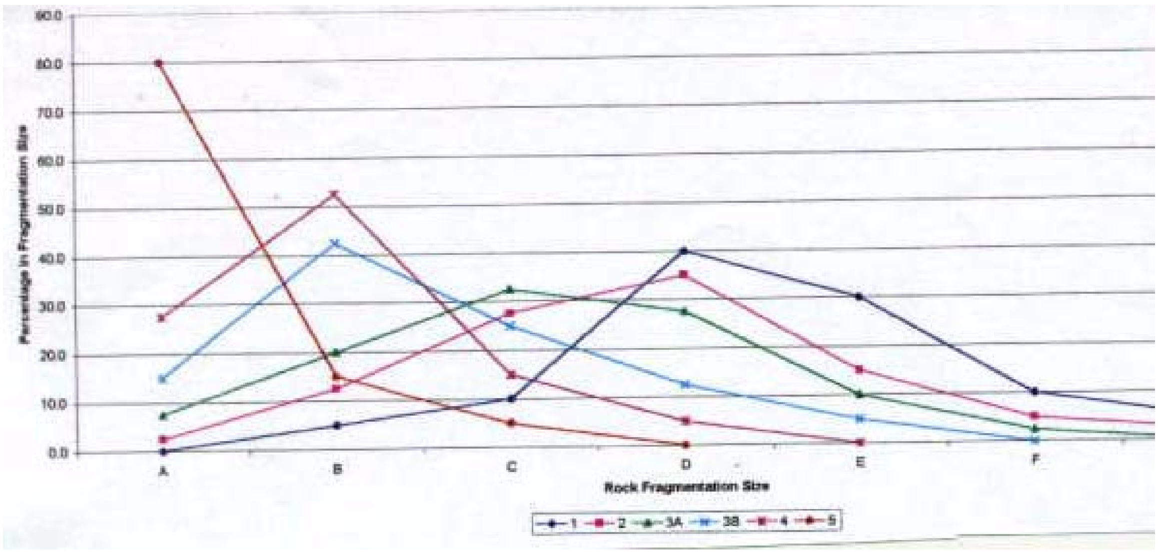
Initial Base Fragmentation - Free Caving

Percentages of rock sizes that occur in naturally free caving ground, with no other effects than just relaxation of the material and natural falling into the void below is shown in the tabulation below and the graph that is derived from it. This sort of distribution at Shabanie and Gaths has worked out reasonably well with regard to the secondary blasting efficiencies that have resulted after consideration of the other factors. It appears to be a good base to start.

The example used throughout is Main/5E at Gaths Mine, King Section.

							TABLE 2
Expected Base Fragmentation from Insitu Classification (RMR)							
Rock Fragmentation Size	Insitu Classification (RMR)						Block/Drawpoint/Zone Calculated Distribution
	1	2	3A	3B	4	5	
A	0.0	2.5	7.5	15.0	27.5	80.0	23.1
B	5.0	12.5	20.0	42.5	52.5	15.0	29.3
C	10.0	27.5	32.5	25.0	15.0	5.0	22.7
D	40.0	35.0	27.5	12.5	5.0	0.0	16.9
E	30.0	15.0	10.0	5.0	0.0		6.3
F	10.0	5.0	2.5	0.0			1.5
G	5.0	2.5	0.0				0.4
stribution by Classification in Block/Drawpoint/Zone							
	0	15	29	22	19	15	
rcentage > 2 m^3	85	58	40	18	5	0	25
Input							
lculated Average Distribution of Rock Sizes							

Distribution of Insitu Fragmentation by RMR Classification



It will be noted that class 3 IRMR has been split into 3A and 3B as they tend to behave differently. 3B ground tends to break and flow and generally behave more like class 4 while 3A ground tends to report as hard pods.

Primary Blasting Fragmentation

Here the initial base fragmentation from free caving is further reduced owing to th effect of blasting. It is assumed in this section that the blasting is efficient, n excessive powder factors and reasonable swell relief.

TABLE 3

Primary Breaking Effects on Fragmentation

Expected Fragmentation by Rock Size after Primary Blasting					
Rock Size before Primary Blasting	**Percentage Rock Size after Primary Blasting** A	B	C	D	Percentage now > 2 m^3
A	100				
B	95	5			
C	85	10	5		
D	70	15	10	5	5
E	55	20	15	10	10
F	40	25	20	15	15
G	25	30	25	20	20

From Block/Drawpoint/Zone Data Fragmentation if Primary Blasted

Rock Size	**Percentages** Insitu from Insitu	after Primary Blasting
A	23.1	86.1
B	29.3	8.0
C	22.7	4.2
D	16.9	1.8
E	6.3	
F	1.5	
G	0.4	
Percentage > 2m^3	25.0	1.8

Stress Caving

The estimated stress effect that the cave back will be subjected to will be dependent on:

- ❖ The range of RMRs within the back. The greater the range the less effect as weaker material will tend to squeeze out before stresses can be built up.

- ❖ The actual direction and interplay of the structures, because again if there are any flat lying structures this will tend to, particularly if they are conducive lead to spaling from the back rather than allowing the stress to build.

Consider if we start with the stress effect as a maximum (100%) and everything then tends to reduce this percentage. The percentages of the rock in the back that RMR is class 5 is doubled, class 4 as is and class 3b halved. Therefore at Gaths King Section Main/5E reduces 100% to 30% see below:

TABLE 4

Potential Stress Build Up in the Cave Back

Class	Insitu Classification Percentage	Factor	Reduction in Stress Levels
5	15	2.0	-30
4	19	1.0	-19
3 B	22	0.5	-11
Total			**-60**
Therefore Remaining Stress Potential			**40**

Now adjusted (reduced) for structural factors.

The structures in this case (Main/5E) had minimal effect as they were near vertical and tended to clamp.

Stress Effects on Fragmentation

TABLE 5

Maximum Stress Caving Effects

Rock Size	Percentage Rock Size after Maximum Stress Effect A	B	C	D	E	F	Percentage now > 2 m^3
A	100						
B	10	90					
C		25	75				
D		10	50	40			40
E			30	60	10		70
F				50	50		100
G					70	30	100

From Block/Drawpoint/Zone Data Fragmentation

Calculated Remaining Stress Effect 40% of maximum

The range of IRMR's in the cave back will determine the estimated stress effect.
The more homogeneous the greater the effect.

Reduction of stress for structures 5 Percentage input

Reduced Stress Effect for Structures 35 % of maximum

Also the structures in the back which might lead to spalling as opposed to stressing.

Rock Size	Percentages			
	Insitu from Insitu	if Maximum Stress Effect	Initial Calculated Fragmentation	Adjusted Fragmentation
A	23.1	26.0	24.2	24.1
B	29.3	33.7	31.0	30.8
C	22.7	27.3	24.5	24.3
D	16.9	11.3	14.7	14.9
E	6.3	1.6	4.4	4.6
F	1.6	0.1	0.9	1.0
G	0.4		0.2	0.2
Percentage >2 m^3	25.0	13.0	20.2	20.8

ASSESSMENT

DESIGN TOPIC

Secondary Fragmentation

GENERAL DESCRIPTION

Secondary fragmentation is the reduction in size of the primary fragmentation particle as it moves down through the draw column. The processes to which particles are subjected determine the fragmentation size distribution that reports to the drawpoints. A well jointed material with a high rock block strength can result in a stable particle shape at a low draw height. A wide range in rock mass ratings will result in wide range in fragment size distribution as compared to the fragment size distribution produced by a rock mass with a narrow range of ratings, as the fine material produced by the former tends to cushion the larger blocks and prevents further attrition of these blocks. This is a common occurrence on chrysotile asbestos block cave mines where the shear zones will have ratings from 10 to 24 and this material will cushion the larger primary fragments with ratings of 50 to 65. A slow rate of draw allows a higher probability of time dependent failure as the caving stresses act on particles in the draw column.
Fragmentation is the major factor that determines drawpoint productivity. The following are required to determine the secondary fragmentation :-

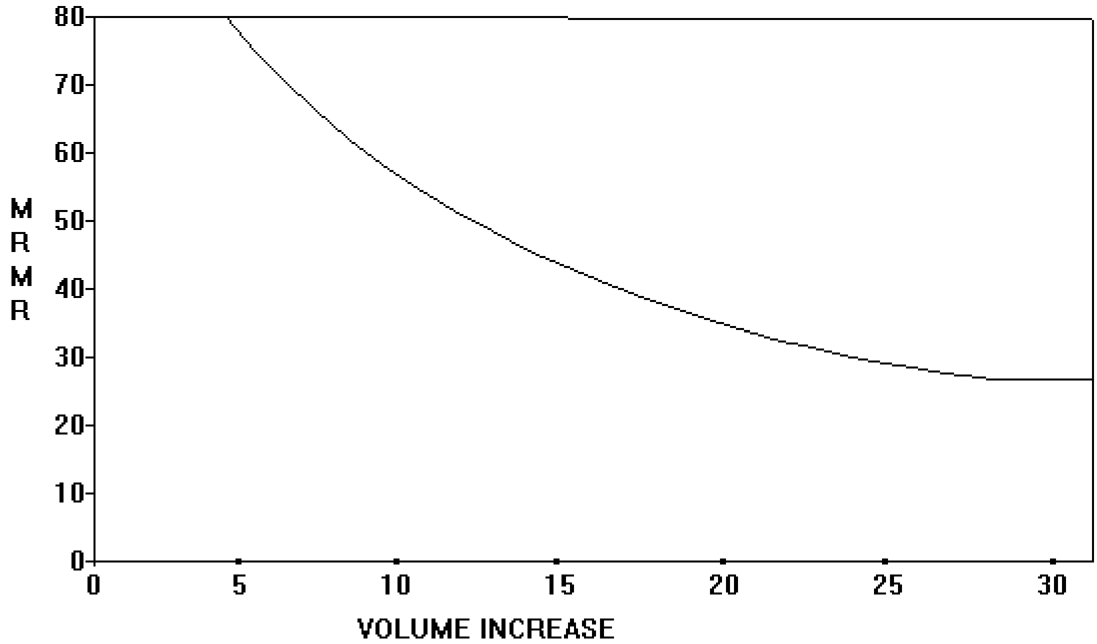
- Draw height - caving stresses
- Draw rate
- Shape of particle
- Workability of the particle
- Draw strategy

PRIMARY FRAGMENTATION

As the secondary fragmentation is calculated from the primary fragmentation it is necessary to record the primary fragmentation data for different elevations. There can be a change in primary fragmentation in the cave column owing to stress differences or changes in rock mass properties. This also applies to the hangingwall zone as finer material could present a dilution problem.

VOLUME INCREASE

The volume increase is a function of the MRMR and primary fragmentation and can range from measured 6% for very coarse material to 30% for fine material. A high volume increase means lower density caved material and also a slower propagation of the cave.

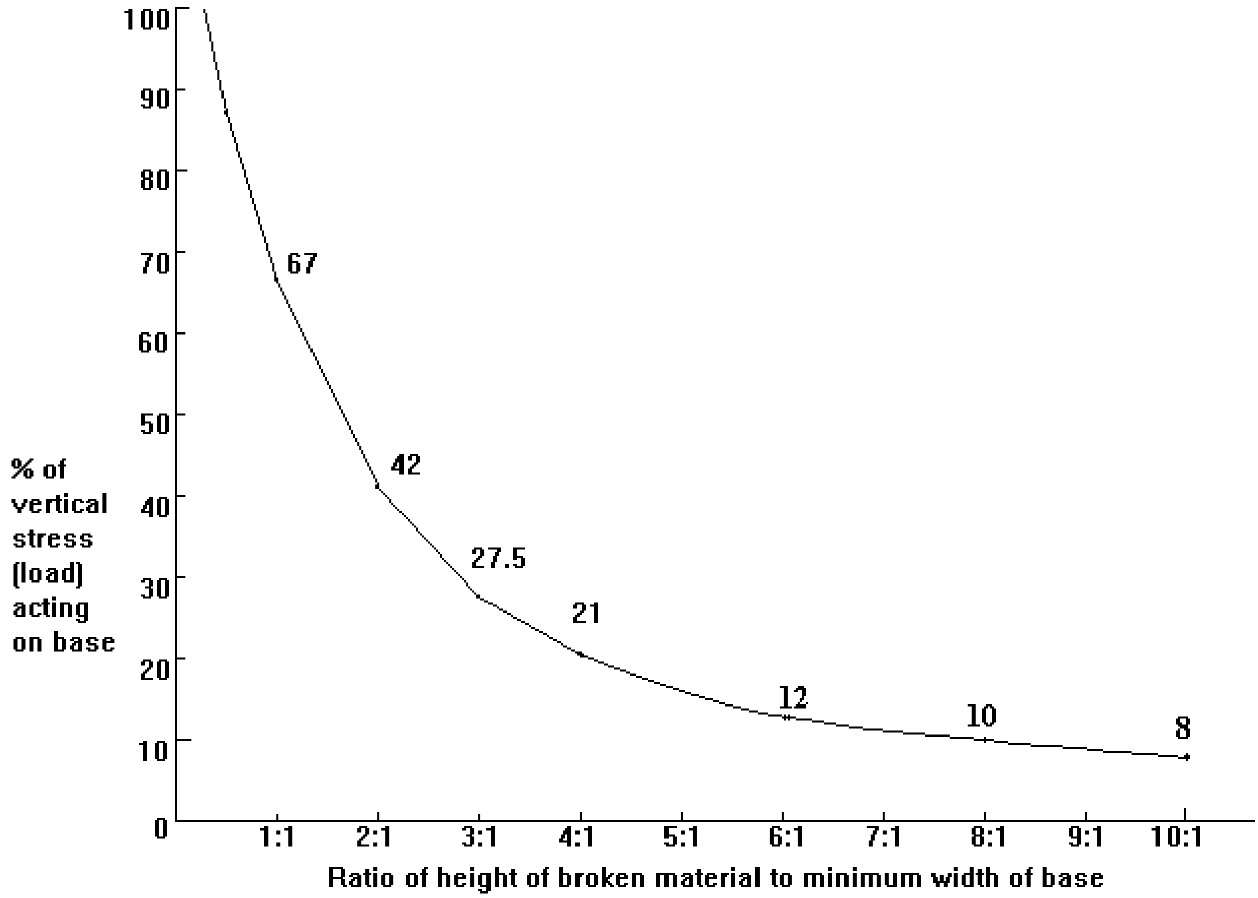


DRAW AND COLUMN HEIGHT

The draw column is the height of the ore and dilution that will be drawn through the drawpoint. Column height is the height of the broken ground to surface overlying the drawpoint and its mass has a significant bearing on the secondary fragmentation process. The BCF program makes provision for the various stages of caving with provision for the input of the volume increase data.

HEIGHT TO BASE RATIO

The height to base ratio gives an idea of the stresses that will act on the base and the sides of the draw column. With high height to base ratios the bulk of the 'weight' is carried by the sides through internal arching. The arching stresses will increase with an increase in the height to base ratio. These principles apply to active and static drawzones, the active drawzone will arch and the stresses will be thrown onto the static column. If a large column is not drawn for some time the loading on its base becomes significant and the underlying pillars are damaged.



The load on the base should be higher in the centre of the mining area than on the sides.

CAVING AND ARCHING STRESSES

The caving stress is the load imposed on stationary particles by the superincumbent caved material. This is likely to be significant if the height of the cave column is appreciable and there is irregular draw.

The drawdown of caved material results in the formation and breakage of arches. The arches serve two purposes namely to move material in the draw when it fails and to break rock blocks. Where there is coarse fragmentation, the collapse of an arch affects a large area and material flows into that space and this leads to lateral migration of material. Large arches will form in a column of coarse fragmentation and will require a large interactive area to break the arch or time for a block in the arch to fail.

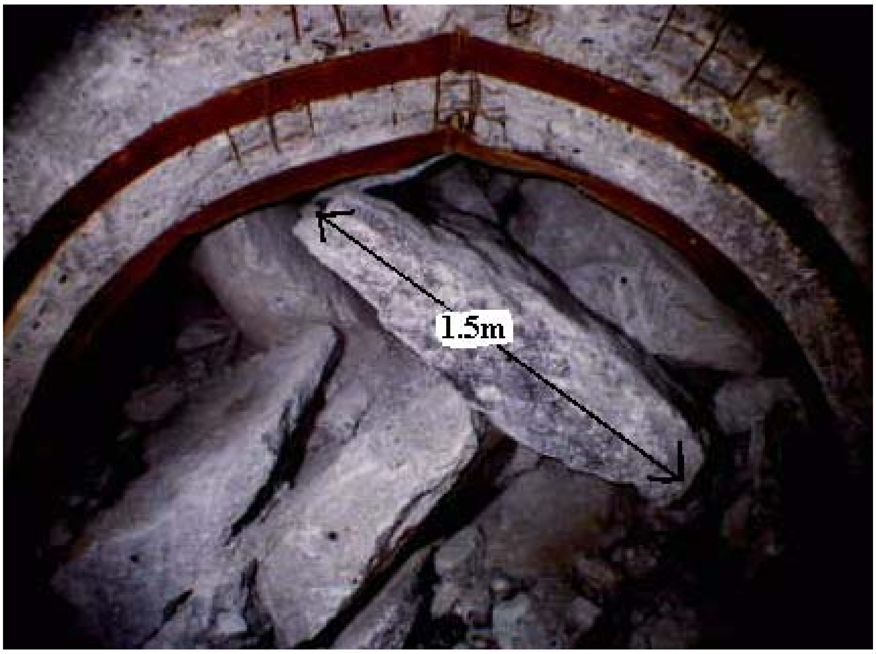
ROCK MASS PROPERTIES

Frictional Properties

Arches in low friction material tend to collapse in a shorter time than with high friction material.

Rock Block Workability - Shape and Strength

The workability of the rock block is a function of shape and strength. An angular block will reduce more readily than one with a cubic shape. Low rock block strength means rapid failure of the rock block in an arch. Dominant parallel structures give rise to slabby blocks, particularly if the rock blocks have a high strength as shown by the breccia from Teniente.



In the fragmentation program it is assumed that 5% fines will be created by the rounding of corners, this assumption is based on an empirical assessment of different shapes.

Impact Breakage

Impact breakage occurs when there is an air gap between the cave back and the rock pile. the degree of breakage will depend on the height of fall, the block shape and its RBS.

DRAW PROCEDURES

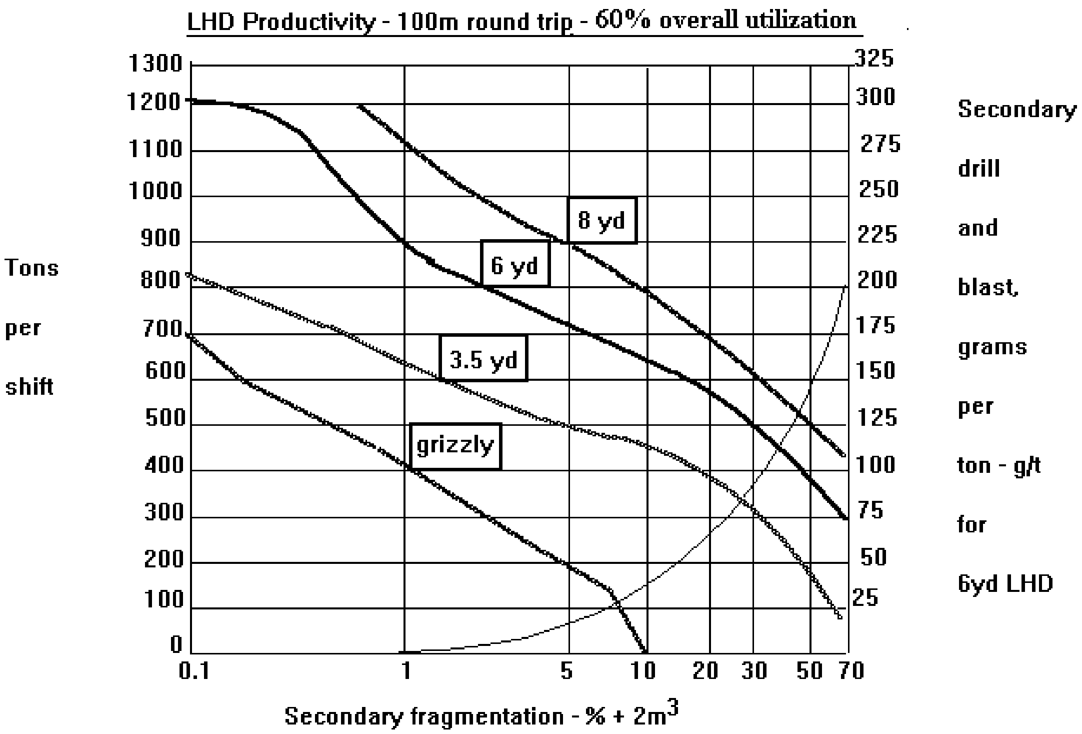
A low draw rate will result in time for the rock blocks to be subjected to the caving and arching stresses - time dependant failure.. This is particularly important in the early stages if good fragmentation is required. Irregular draw is often the result of having zones of well fragmented material available, allowing for high productivity from those drawpoints at the expense of the drawpoints with coarse material. Sound draw control management is required not only to maximise recovery but, also to improve the fragmentation. The operating personnel must understand the consequences of the unfortunately common practices of drawing ‘easy’ ore.

An uniform draw over the whole mining area means little relative movement between rock blocks compared to when zones of interactive draw are drawn on a regular schedule of one shift or one day so that high and low pressure areas are set up to promote differential movement. ‘Rocking the block’ was standard procedure in the past to promote fragmentation.

PRODUCTIVITY AS A FUNCTION OF % + 2M^3 MATERIAL

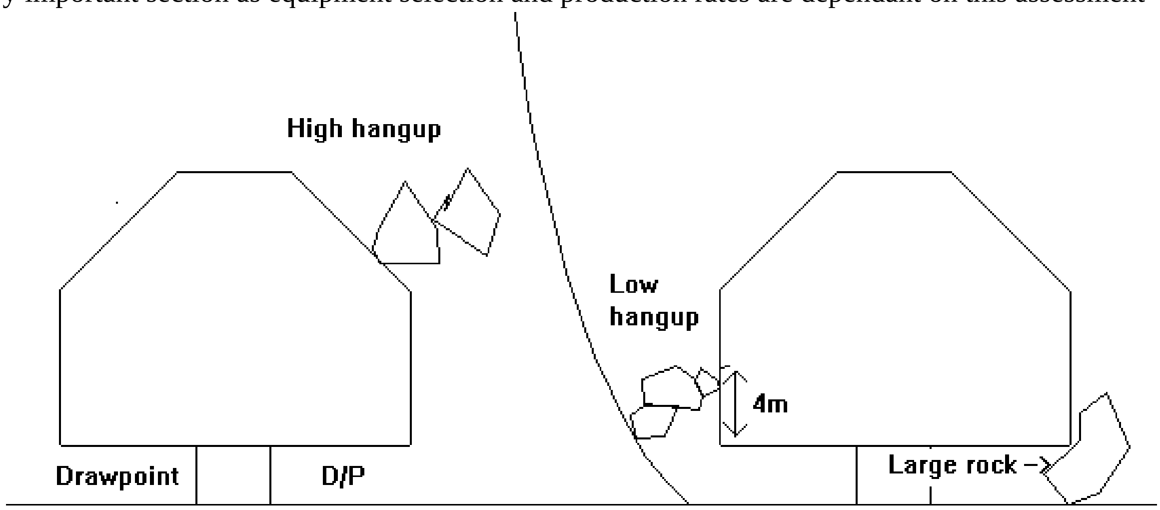
Fragmentation is the major factor that determines drawpoint productivity. High productivity can be expected from 6yd LHDs loading the material from the drawpoint in the following photograph:

Experience has shown that a two cubic metre block is the largest size that can be moved by a 6 yd LHD and still allow an acceptable rate of production to be maintained. In the following graph, the productivity of 3,5yd, 6yd and 8yd LHDs and a grizzly layout are related to the percentage of fragments larger than two cubic metres. Secondary explosive usage is based on the amount of oversize that cannot be handled by a 6yd LHD.



The range in fragmentation for different draw heights must be recorded as the average % + 2m³ and should also reflect the range in particle size. This data will ensure that the productivity for the life of the block can be assessed. The LHD fleet could be reduced in time.

Is important to define hangups and the location of hangups. A hangup is when a rock arch has formed above the brow. A low hangup can occur from 1m to 4m above the brow and a high hangup from 4m to 12m above the brow. Large rocks on the floor in the drawpoint do not constitute hangups. This is a very important section as equipment selection and production rates are dependant on this assessment



FINES CUSHIONING

The ratio of fines to medium / coarse fragmentation needs to be noted as a high percentage of fines will cushion the coarser fragments and reduce the secondary breaking effects. This has been a common experience on some of the asbestos mines with well developed shear zones providing large quantities of fines. The BCF program makes provision for this as input data.

ASSESSMENT

Comments From N.J.W.Bell

Falling Effects

The falling of rocks from the cave back onto other caved rocks must be regarded as significant if the falls are 10m or more. Additional factors must be built in for this additional fragmentary effect. Slabs tend to break and 'rounding' takes place. Slab shape and aspect ratios are important.

Gravity Effects on Fragmentation TABLE 6

Maximum Gravity Caving Effects (Falling 10m or more)

Rock Size	Percentage Rock Size after Maximum Gravity Effect A	B	C	D	E	F	G	Percentage now > 2 m^3
A	100							
B	20	80						
C		25	75					
D		5	20	75				75
E			10	15	75			90
F				10	15	75		100
G					10	15	75	100

From Block/Drawpoint/Zone Data Fragmentation after Stress Effects

Estimated Gravity Effect 10 % of Maximum - Input

The Estimated Gravity Effect will be determined by:

The IRS of the rock block

The chance of a 10m or greater void below the cave back at that time

The cushioning effect of the broken ground

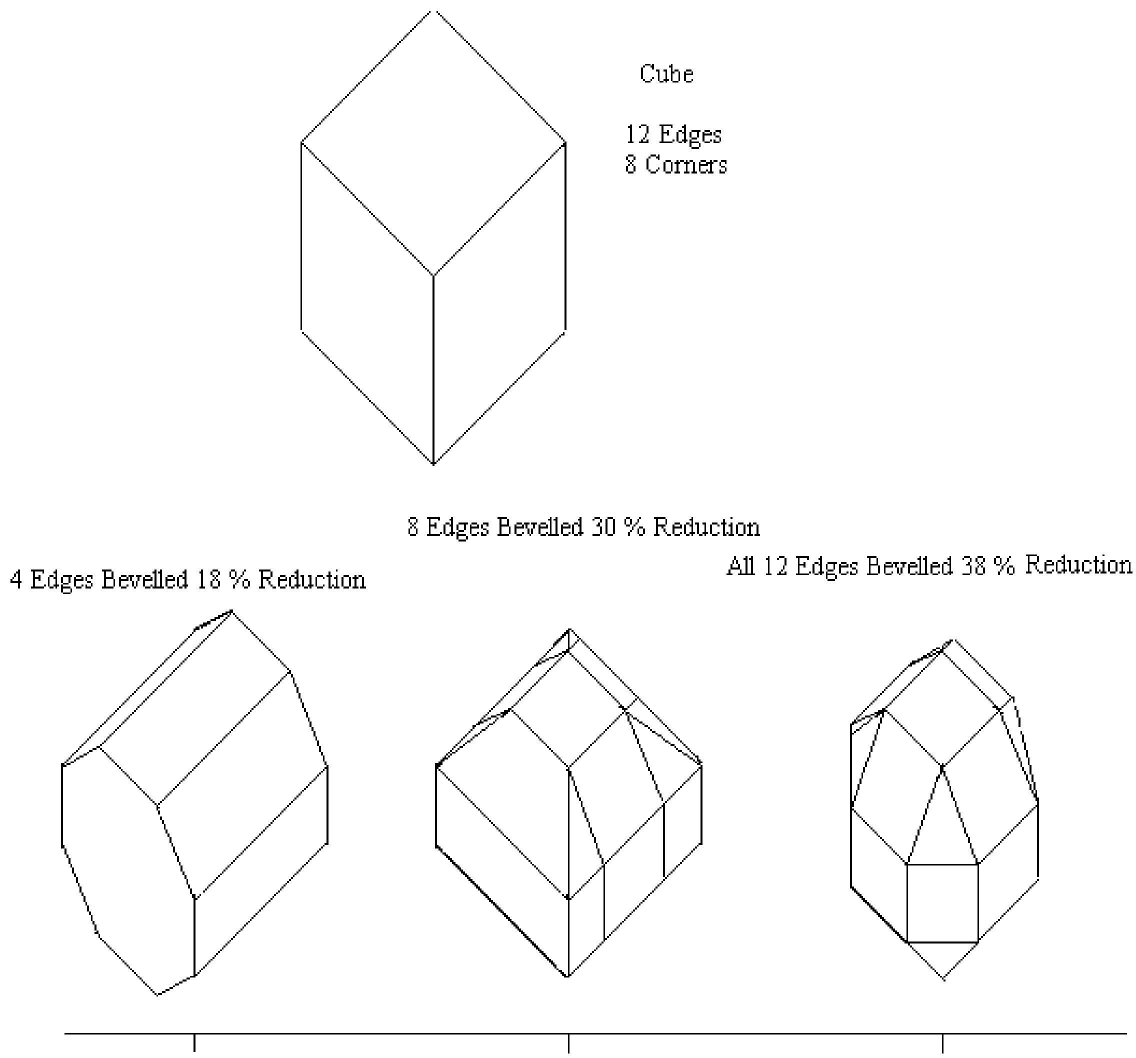
Rock Size	Percentages After stress from stress cave	If maximum gravity effect	Calculated fragmentation
A	24.1	30.3	24.7
B	30.8	31.5	30.9
C	24.3	21.7	24.0
D	14.9	12.0	14.6
E	4.6	3.6	4.5
F	1.0	0.8	1.0
G	0.2	0.2	0.2
Percentage>2m³	20.8	16.6	20.4

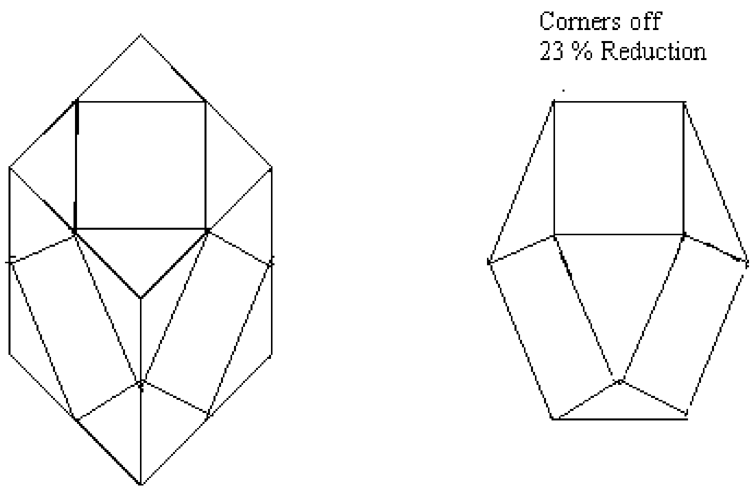
“Rounding” Effects

As the draw progresses there is a ‘rounding’ effect from angular tending towards spheres and is a component of the distance the ore has been drawn through an active column. The rounding also depends on the intact rock strength (toughness) of the rock.

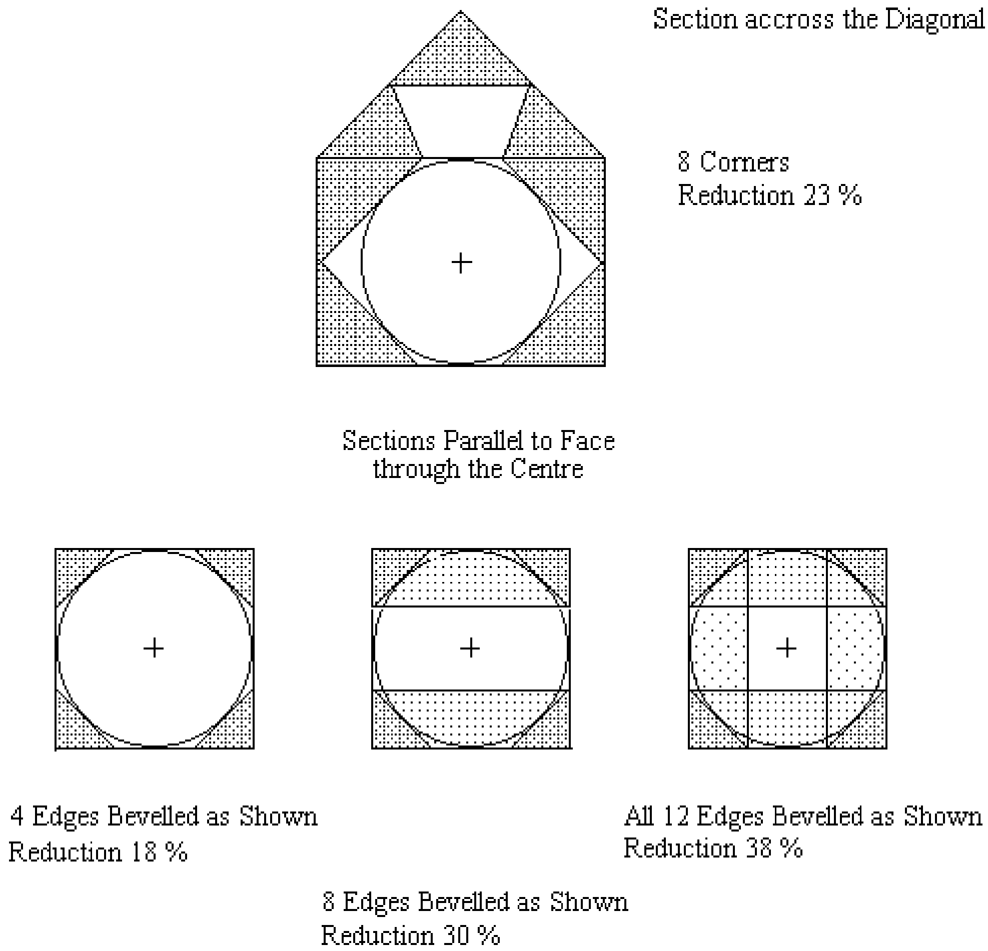
The grinding action within the column owing to the formation and collapse of hang-ups. Minor stresses on the weak points/edges of the blocks tending to “round”. The tendency to do this will depend on the nature of the rock and the nature of the joints etc on which it is broken. The closer it gets to round the less and less effect there is in this action.

“Rounding” maximum is a cube to a sphere where 48% of the material would have been knocked off. However, it is likely that only a 20 to 35 % volume reduction will be the maximum achievable. This will only be achieved in very high draw columns and where soft rocks are involved. In actual fact a 15 % volume reduced rock looks reasonably round and stable. 'Rounding' will be progressive with the distance drawn according to the IRS.





Rounding of Rock Blocks with Draw



Rounding of Rock Blocks with Draw

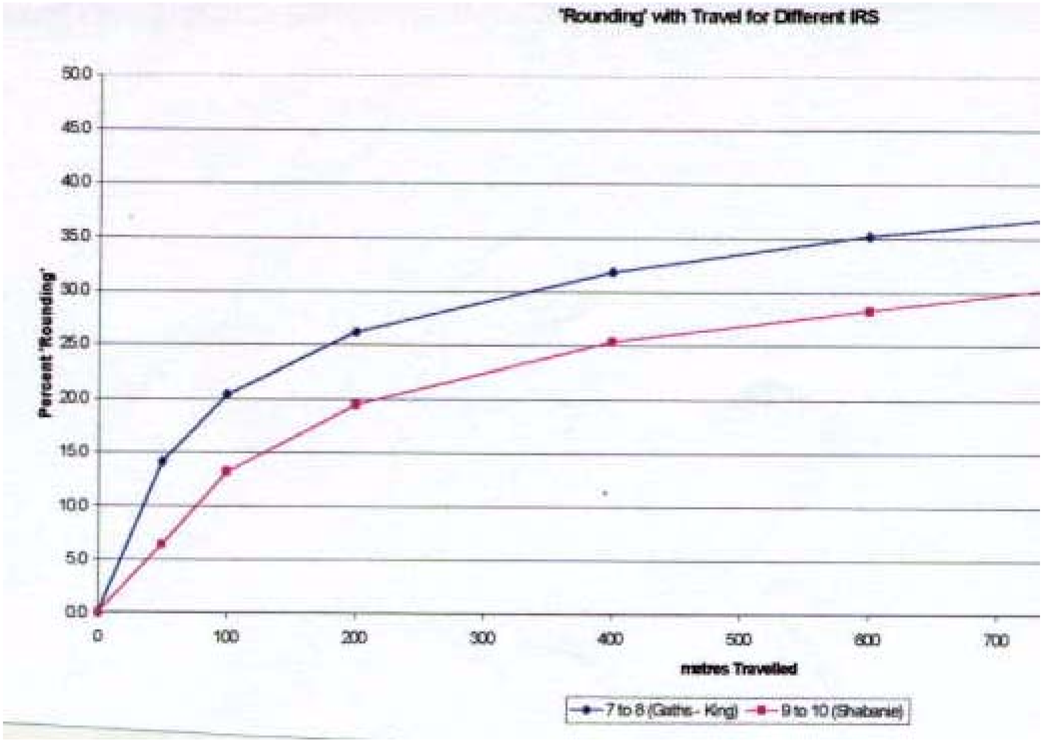
From observations "rounded" rocks report after 50m of draw at Gaths, King section and at 100m at Shabanie. The table and graph below shows this and the anticipated 'rounding' with draw distance and by IRS.

'Rounding' Effects on Fragmentation TABLE 7

The 'Rounding' Effect is a function of distance traveled and IRS
A Sphere Calculates at a 48% reduction in volume from a cube.
At 15 % reduction in volume of a rock tends to be described as 'Rounded'

Rounding Factors are applied to distance traveled - remembering relative speeds.
These are applied in the draw calculations as they progress.

Distance Travelled metres	Percentage Volume Loss per 10m Travelled **7 to 8** (Gaths - King)	**9 to 10** (Shabanie)	Distance Travelled metres	Percentage Cumulative Volume Loss **7 to 8** (Gaths - King)	**9 to 10** (Shabanie)	Remarks **IRS**
0			0	0.0	0.0	
0 to 50	3.0	1.5	50	14.1	6.4	
50 to 100	1.5	1.5	100	20.4	13.2	
100 to 200	0.8	0.8	200	26.2	19.5	
200 to 400	0.4	0.4	400	31.8	25.3	
400 to 600	0.3	0.2	600	35.2	28.2	
600 to 800	0.2	0.2	800	37.4	31.0	
800 to 1000	0.1	0.1	1000	38.3	32.7	



The rounding will produce fragments that are two and three rock sizes below the block from which they came see tabulation below. The ratio of smaller fragments will be approximately half each.

G to E and D
F to D and C
E to C and B
D to B and A
C all to A
B all to A
A will obviously remain in this fraction

It is acknowledged that there will then be further rounding, but this will be taken care of with the onward migration of the particles from that size down, as the draw progresses.

Particles so released now accelerate and join a finer fraction. This is best handled as a migration down the material size at each period of draw, depending on the IRS.

There is a need to account for the mineralisation in the different rock block sizes. If peculiar circumstances exist as for instance with fibre seams where these are less competent and therefore they are likely to migrate and break off earlier and travel further the value must be so adjusted. This would particularly be in the hanging wall at Shabanie.

Handling of the Fragmentation Changes with Circumstances and Time

See also Section 30 Draw Control.

Using the above system to establish the primary fragmentation expected in a column. One can then develop a model for say 10, 25 or 50 metre vertical lifts of what is likely to be the primary fragmentation in the draw column.

Mines should select their own to suit the varying situations (ore heights, location of values, changes in RMR, etc.).

The secondary fragmentation which results from the rounding effects will be dealt with depending on the distance the particles flow, against the relative speeds and the degradation down the scales from the primary fragmentation as predicted.

DESIGN TOPIC

Secondary Breakage

GENERAL

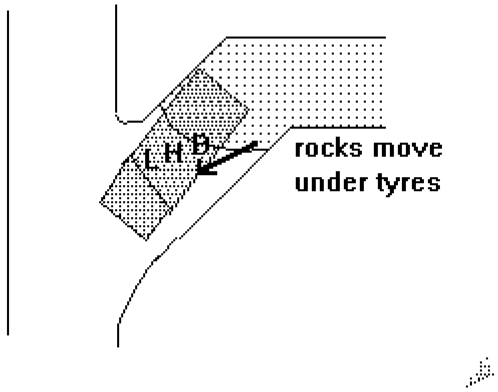
Large rocks reporting in the drawpoints and forming hangups are a major factor when designing the block caving of competent orebodies with coarse fragmentation. In the past, when large rocks occurred as a low percentage of the draw, the technique was to jack hammer large rocks in drawpoints or to place lay-on charges on them. Hangups were bombed or if it was a high hangup they were drilled through the major apex. However, with the increased percentage of oversize with the caving of more competent orebodies, the placing of lay-on charges (bombing) is not acceptable owing to the damage caused to the drawpoint and brow and the high powder consumption. The common, but unacceptable practice of placing the charge between the rock and brow resulted in extensive damage. Jumbo's are used effectively to drill and blast large rocks in drawpoints with low powder consumption as the rock has free faces. A hangup drill was developed in Zimbabwe some 20 years ago to effectively drill and bring down hangups up to 10m in above the brow. More recently a highly sophisticated drill has been developed in South Africa to remotely drill hangups. However, this machine has not operated in a full production capacity so its overall performance is not known. A secondary (mezzanine) drilling level, in the major apex that intersects the corner of the drawbell is recommended as a solution to the hangup problem and to ensure high production rates. Efforts are being directed into developing non explosive techniques to break large rocks in drawpoints, results to date are encouraging.

SECONDARY FRAGMENTATION DATA

The secondary fragmentation data is prepared for increasing draw heights and will show whether there is a progressive decrease in coarse fragmentation or whether the $+2m^3$ material will reach a constant level for the remaining draw life. It is important to establish from the start whether high hangups a temporary problem or are they to be a factor for the life of the operation? An accurate assessment of the secondary fragmentation means that the right equipment can be on site and the production potential of the block can be determined.

DRAW METHOD, EQUIPMENT AND GRIZZLY OPENINGS

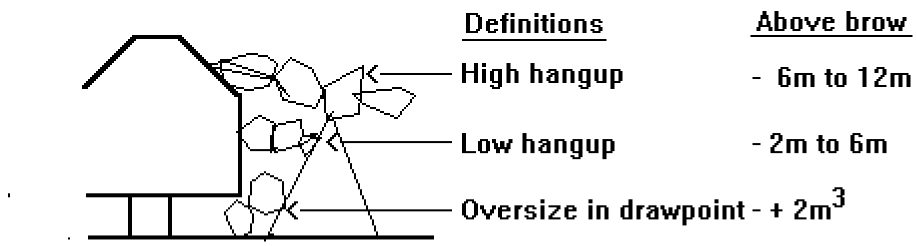
The secondary breaking assessment requires that all factors such as size of grizzly and crusher openings are recognised. Some mines do not have grizzly's on the extraction level orepasses and the material goes directly to a pickhammer operating on a grizzly. In this case the LHD operator must use his discretion. In other cases pickhammers operate on the grizzly's on the extraction level. These various techniques and systems need to be analysed. A LHD layout with large equipment and small grizzly openings on the level means that the full potential of the equipment is not realised. Large drawpoints mean that large rocks can report in the drawpoint and be broken on the ground. However, the experience at Henderson with wide drawpoints, must be taken into consideration, namely that LHD's load on the same line in angled drawpoints and cannot load material on the sides. Rocks outside the loading zone roll into the drawpoint drift and get behind the wheels to cause high tyre wear.



Techniques such as using the LHD bucket to break rocks on the floor of the drawpoint is not a good idea as the floor and LHD can be damaged leading to high maintenance costs.

PERCENTAGE OF HANGUPS

The following diagram shows the percentage distribution of hangups from different mines and can be used as a guide. A more accurate way is to look at the size distribution from the BCF program.



NOTE: The location of a hangup is a function of the size of the drawbell

Distribution of hangups/oversize as a percent of $+2m^3$ tonnage				Nominal Drawpoint Spacing
Mine	% Oversize	% Low hangup	% High hangup	
Bell	60	30	10	15m
King	50	30	20	10m
Salvador	45	35	20	14m
Shabanie	50	35	15	11m
Teniente	60	30	10	16m
Average	53	32	15	

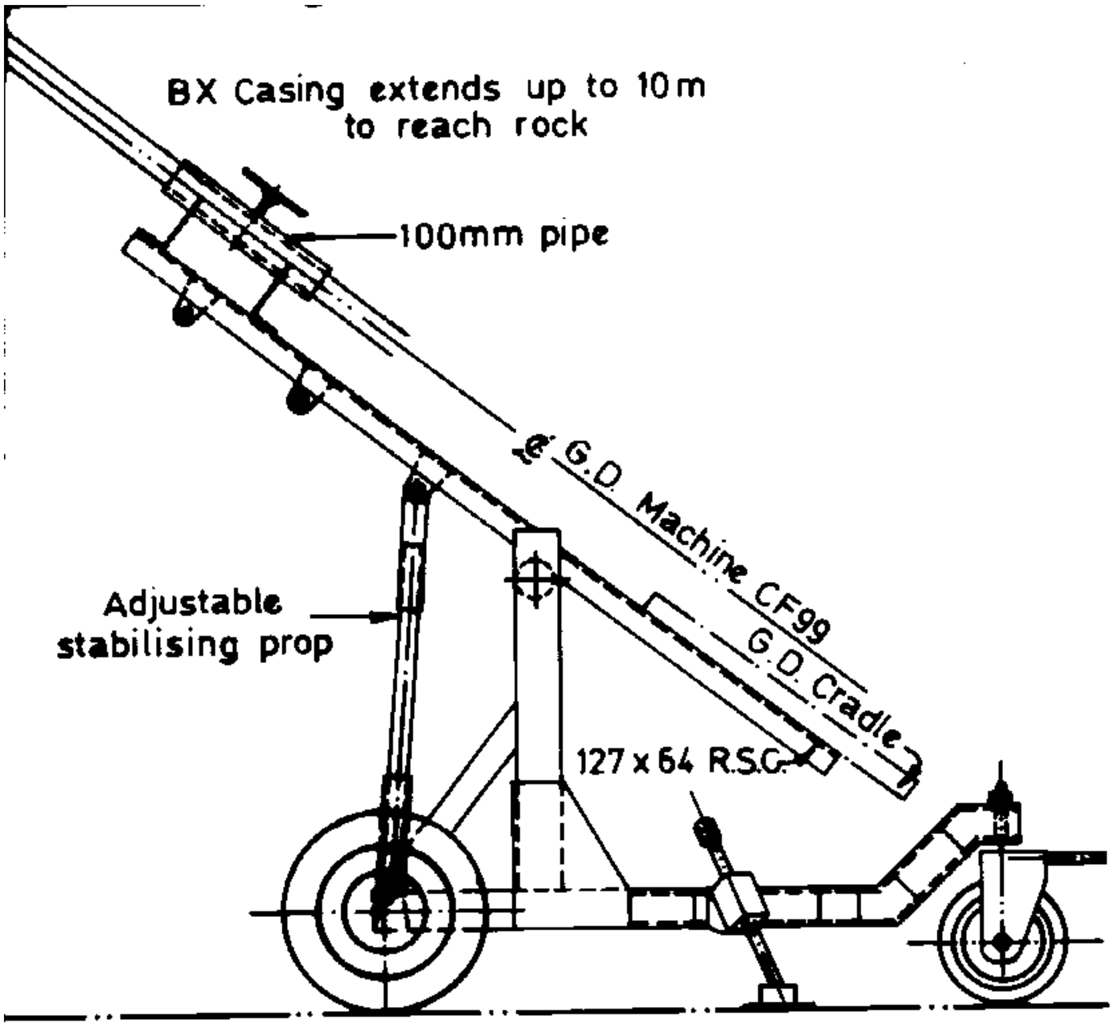
For example, at 10000 tpd, if the $+2m^3$ material is 40% of the available tonnage then 2100 tons would occur as large rocks in the drawpoint and would have to be drilled by a jumbo. Low hangups would account for 1300 tons which would be drilled with a jumbo and a hangup machine. A special high hangup machine would be needed to drill the 600 tons in the high hangups. The number of times a high hangup is likely to occur can be calculated by estimating the tonnage in a normal high hangup and dividing this figure into the total high hangup tonnage as determined from the distribution chart above. For example, estimated hangup at 200 tons, 15% of oversize = 600 tons, therefore number of hangups per day = 3.

In terms of bringing down a hangup, the important question is the stability of the hangup. Hangups form due to arching above the drawpoint and are revealed when the interdosal ground is removed. A certain period of time is required to allow the arch to settle and stabilise before it can be drilled. If there is any doubt about the stability then a lay-on charge or an explosive bag (see details in blasting techniques) is strategically placed to drop the arch. If the hangup remains it can be drilled and blasted. All these factors must be taken into consideration when scheduling production.

DRILLING EQUIPMENT

A jumbo or smaller rig, with limited elevation, can be used to drill large rocks in a drawpoint and to drill low hangups, whose height above the brow is dependant on the location of the legs of the arch or the key block.

High hangups have been successfully drilled with a hangup rig developed on King Mine, Zimbabwe some twenty years ago. The rig is shown below and works on the principle of locating the drill rods in flush jointed diamond drill casing



THE KING RIG

The King Rig has been operating for some twenty years was designed to deal with large rocks and hang ups above the brows of drawpoints, especially in caving blocks when no prior examination of the ground is required once the calculated tonnage from the primary broken ground has been drawn off. The rig is remotely controlled affording maximum safety to the operators which is particularly important when dealing with problems in primary broken ground e.g. early in the drawpoint life or sub level cave operations. The King Rig has also been used to drill pillars left in the undercut and ribs left between drawbells across the minor apex.

KING RIG UNDER THE BROW, DRILLING A HANGUP

When a hang up occurs in a drawpoint which necessitates it being drilled by a King Rig the following procedure is adopted:

1. Eye bolts need to be installed in the drawpoints on each side, one metre above the floor and on the centre line, one metre back from the brow position. Normally done during the construction phase and are only used for drilling high hang ups.

2. Each set of 75mm drill casings to have one casing with an anchor ring welded to it for attaching the supporting chain block.

3. The hang up shall be tested by prodding to ensure that it is secure enough to be drilled and not easily dislodged.

4. The adjacent drawpoints (across the minor apex and in the common drawbell) shall be checked and cleared of all personnel as working these drawpoints may affect the drawpoint to be drilled.

5. The drawpoint approach shall be cleared of all rocks and loose material to allow free access for the rig and the operators.

6. The distance to the hang up is then estimated and sufficient casing put in place to reach it. The rig is then moved forward angling the casing up as it progresses under the brow until it points at the hangup. The chain block is used to assist.

7. Once the rig is in its correct position, it is jacked up off its wheels and the whole rig stabilised with a prop and chains, before drilling commences.

8. Additional casings are then pushed up to the rock where the first hole is to be drilled.

9. Sufficient (there is always a tendency to drill too few) holes are drilled in the rock to ensure that it is broken up efficiently. Spacing of holes shall be at approximately 1,5m intervals and drilled through the rock. A sketch is made as the holes are drilled indicating their location and depth.

10. The rig shall be operated from behind (well back) and to the side of the rig to ensure the safety of the operator in the event that the hang up drops and the rig is moved.

11. When drilling is complete the rig is removed from the site before charging takes place.

12. Charging operations are carried out using 6m lengths of lightweight 50mm aluminium irrigation pipes with a shoe type (supporting) coupling. The pipes are lifted up to the hole to be charged. Charging takes place using conventional rods through the pipes. The charge and timing of each hole, if not instantaneous is decided from the sketch produced during the drilling. Initiation is by detonating cord.

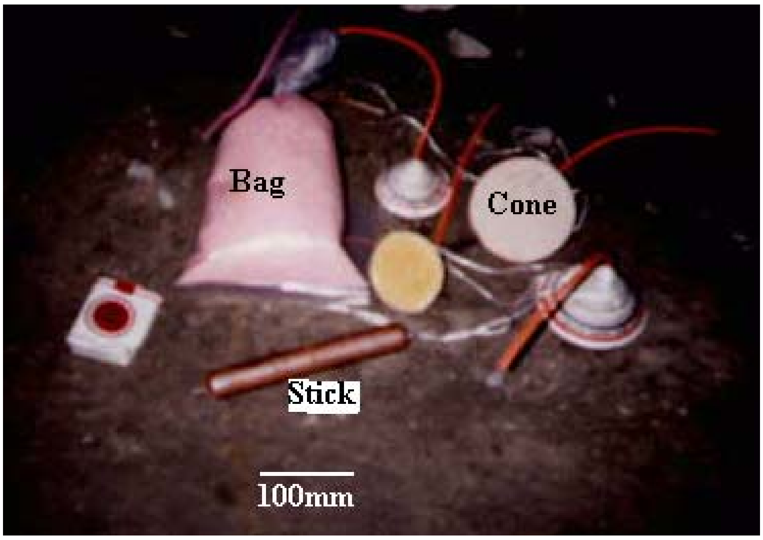
The King rig has had a very successful record, however in spite of its success other block caving mines have not accepted this unit. This is an interesting point which occurs all the time namely the reluctance to accept others success and to always look at the negative aspects.

BLASTING TECHNIQUES

The old system of breaking large rocks with lay-on charges is no longer acceptable owing to the extensive damage to brows and the high cost of the inefficient blasting. Drill and blast is the most efficient technique as the hole can usually be sited to break effectively - after all the rock block has free faces on all sides. On a Canadian Mine, the secondary blasting powder consumption during the grizzly mining stage was 400g/t. The change to LHD mining and to drilling oversize resulted in a reduction to 80g/t. The Chilean mines break large rocks in the drawpoints with very little explosive and figures of 6g/t are quoted for mining primary rock. Large rocks with cemented joints break more readily that homogeneous rock blocks.

OVERSIZE IN A DRAWPOINT - EASY TO DRILL AND FRAGMENT

One of the problems with the use of lay on explosives is overcharging and the common practice of placing the explosive between the brow and the rock. When operators complained about the strength of the rock in a drawpoint, an investigation showed that 22 tons of explosive had been blasted as ‘lay on’ charges in that drawpoint.



Conical charges are effective when properly placed.

Conical charges - made from funnels - placed on large rocks in a blasting bay at Henderson mine

The porous bag technique used effectively on Bell Mine once again has not found favour on other mines. It has proved to be a very useful technique to use in hangups where the stability is in question. Detonating cord is run through an aluminium tube into a stick set in a recess at the end of the tube. A porous bag is placed at the end of the tube so that the stick is inside the bag. The bag is placed between rocks and filled with ANFO. The bag is now jammed between the rocks, the tube is withdrawn and the detonating cord is now available to initiate the blast. As the charge is in intimate contact with the rocks, the breaking is efficient.

Detailed secondary blasting explosive consumption records should be kept so as to be able to distinguish between the different techniques and for varying fragmentation and rock block workability (strength). For the same fragmentation, the powder consumption at Shabanie mine is 30% higher than at King mine owing to the greater RBS of the Shabanie material.

NON EXPLOSIVE TECHNIQUES

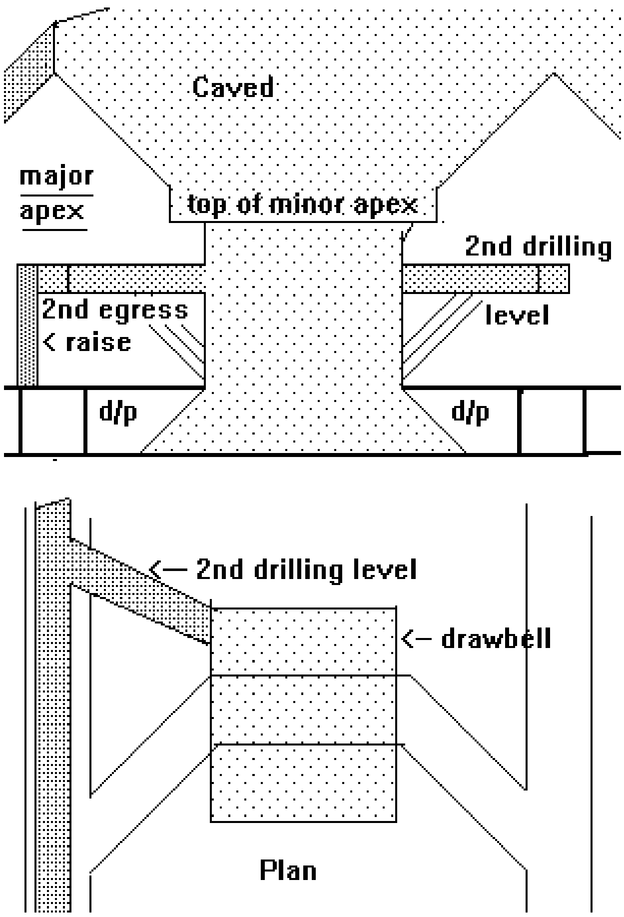
A lot of attention is being paid to non-explosive techniques in terms of safety , ventilation and non interference with production. Comparisons must be made between a well organised blasting system and the proposed non explosive technique. There are merits in the use of explosives especially when dealing with hangups.

Northparkes comments: Non- explosive breaking - very successful with the ‘robust’, i.e. mechanised boulder buster (high pressure water). We were also using the ‘sunburst’ system of expanding gas induced fractures, but we had an accident in which someone nearly lost a hand, so we stopped using that system. The cycle of drilling and breaking a rock is about 20 minutes. This is not critical to us as we only have 15 oversize per day, that is 1 oversize per 800 tons. As a matter of interest we have one hangup per 3000 tons - 4 per day. For the ‘robust’, two cartridges are usually used - one in the breach mechanism to set off the blast and a booster in the hole. A couple of advantages of the non-explosive form of secondary breakage is that it does not affect production as there is no need to clear the area and there is little fly rock and percussion, hence little damage to ground support. Our secondary breakage costs for last financial year were A$0.16 per ton mined.

SECONDARY DRILLING LEVEL

Numerous observations of hung up drawpoints led the writer to the conclusion that a small drift about 5m above the brow would provide an ideal site from which to safely drill and blast hangups. This impression was confirmed at Premier mine when it was possible to stand in an old drift immediately above a drawpoint and watch the rocks move and arch as the LHD loaded in the drawpoint below.

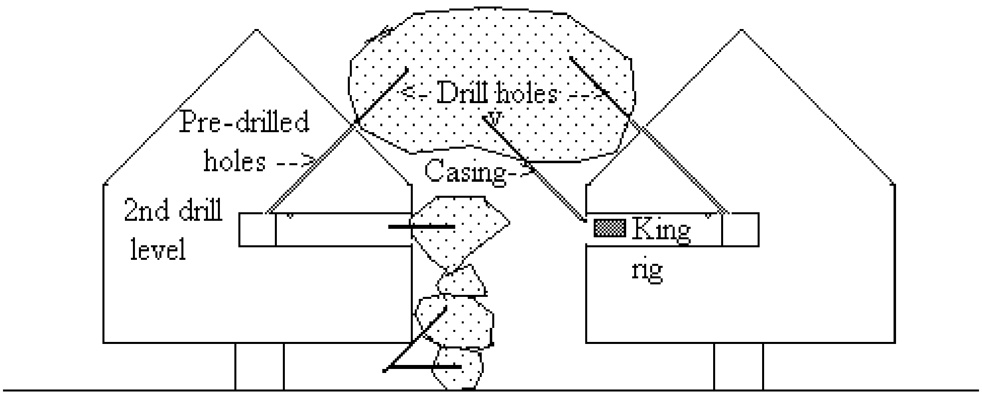
The following layouts would cater for coarse fragmentation and if the ‘King hangup rig’ is used, coarse fragmentation is not insurmountable and becomes a cost and production rate risk. This level is developed behind the advancing undercut, but ahead of the mining of the drawbell. The drift would be developed into the corner of the drawbell - it would not be in the brow over the drawpoint.



Whilst the secondary level concept has been recommended to mines with a coarse fragmentation problem it has not been accepted by the industry even as an experiment.

It is extremely difficult to understand this 'tunnel vision' where people prefer to continue with an inefficient and costly system. One of the arguments is that it will weaken the pillar (major apex), is fallacious as the weakest part of the major apex is on the extraction level where 50% of the rock has been removed.

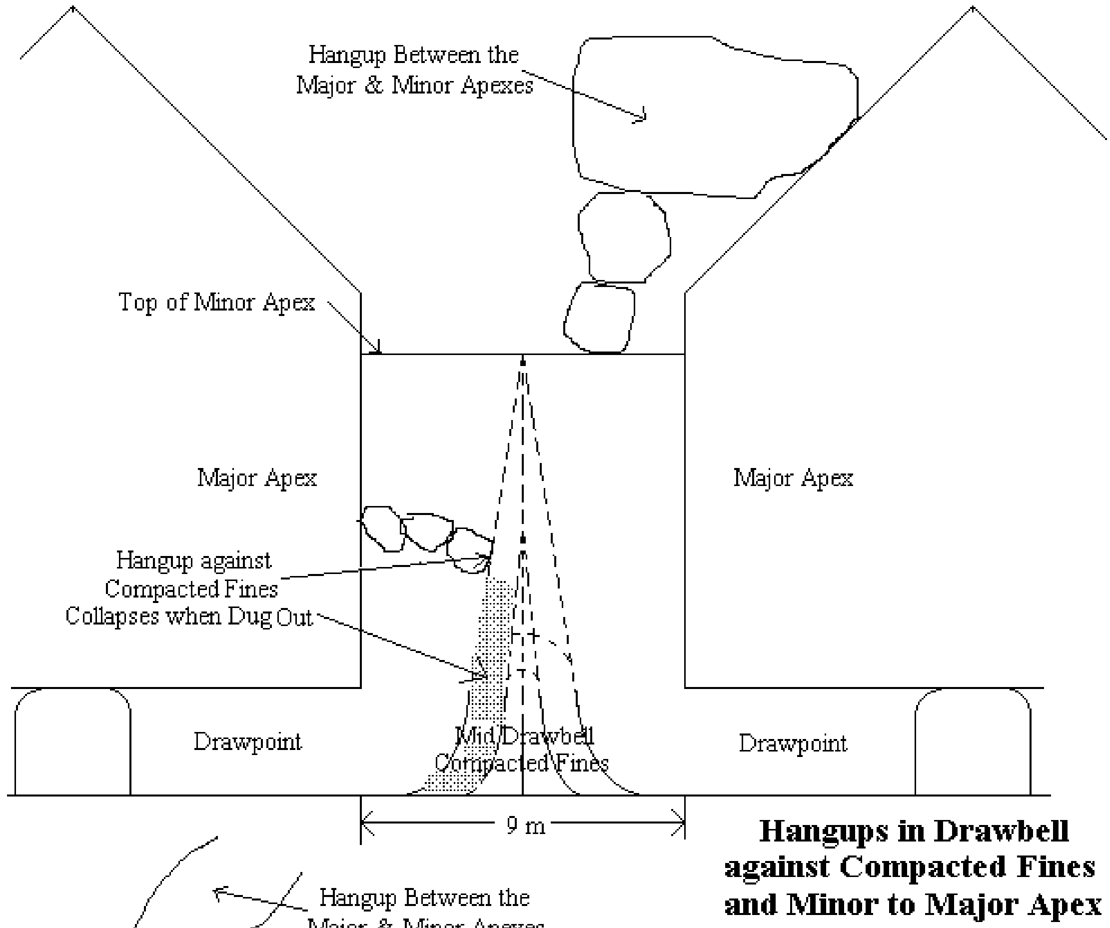
For very high large hangups pre-drilled holes from this secondary drilling level seems to be the way to go. The use of casing, between the end of the pre-drilled hole and the rock for blasting, to drill through and subsequently to charge the holes, is essential.

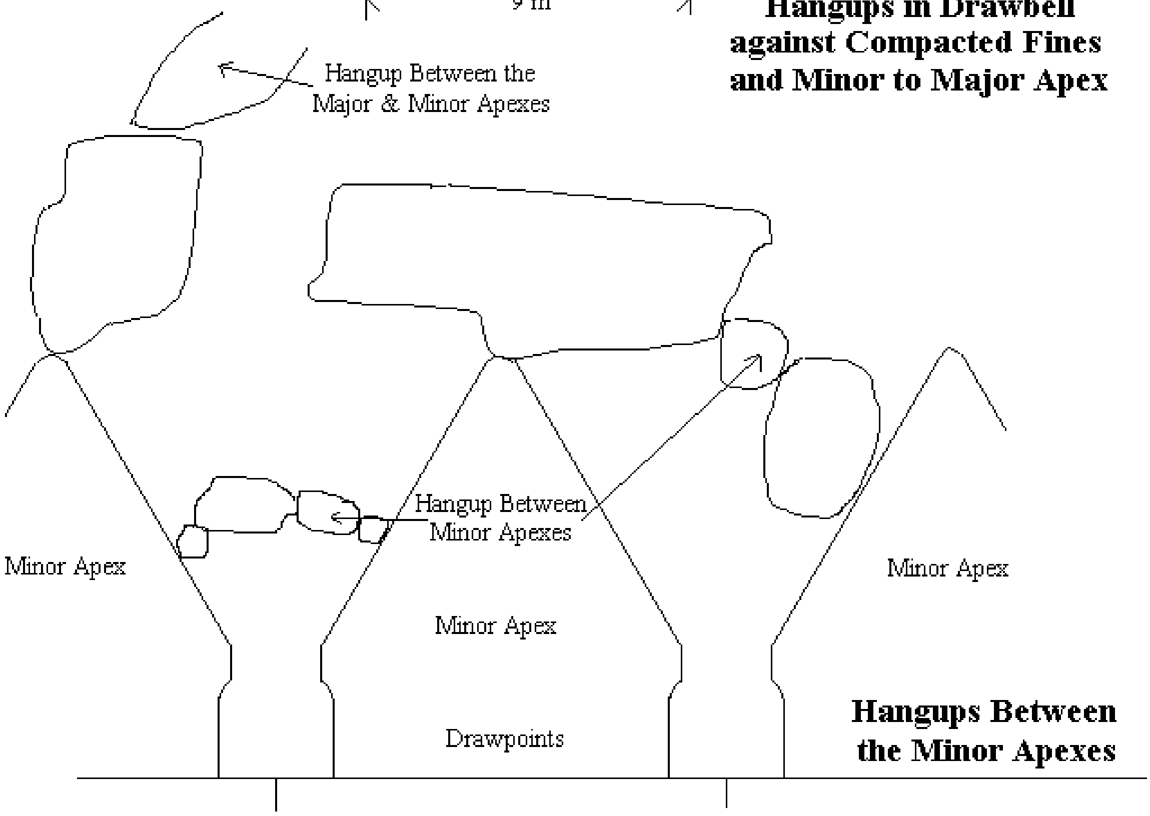


The secondary drilling level becomes an exhaust airway after the blast, thus clearing the production drift of gasses in a very short time.

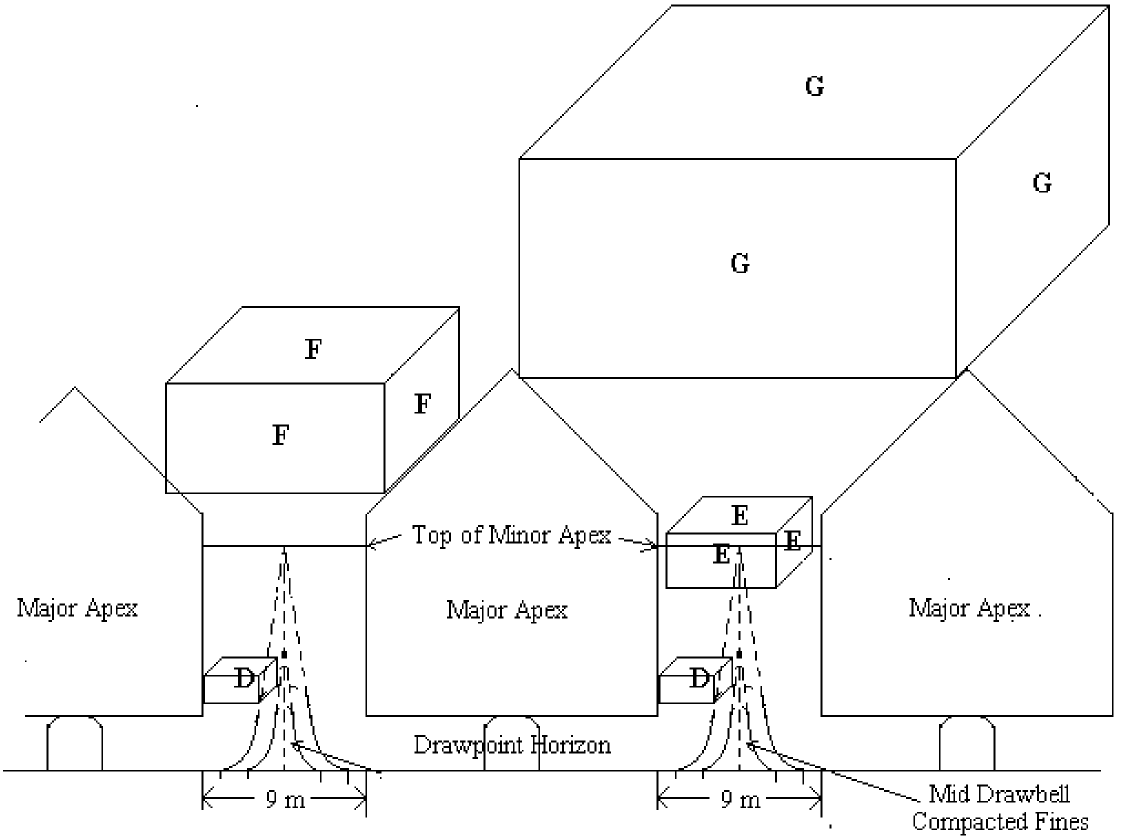
CONTRIBUTION FROM N.J.W.BELL

The following diagrams are based on experience in the Asbestos mines in Zimbabwe and illustrate the location of large rock blocks

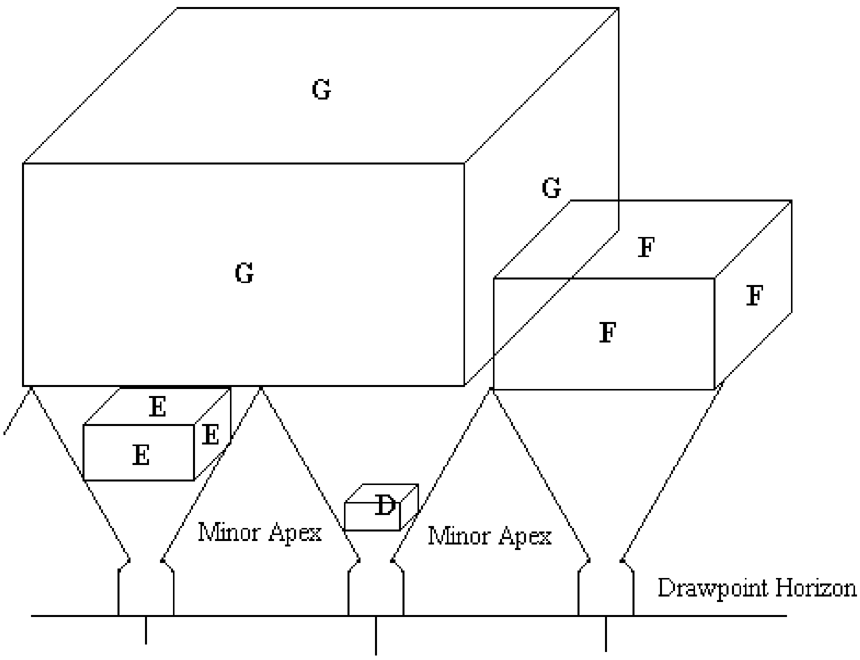




Hangups in Drawpoints



Section through the Major Apexes



Section through the Minor Apexes

DESIGN TOPIC

Drawpoint Spacing

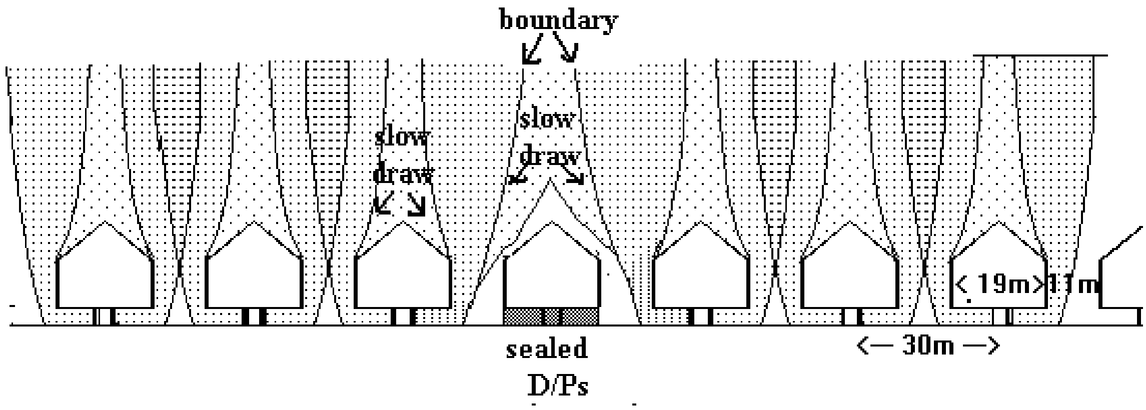
GENERAL DESCRIPTION

Drawpoint spacing is one of the most important and controversial items in cave mining. Often mine planners will only consider spacings that have been or are in use regardless of the mining environment or the rock mass. The object of this investigation is to present sound design parameters backed by underground experience. There are good reasons to increase drawpoint spacing so as to improve the strength of the extraction level, larger and longer drawpoints for larger LHD's and to reduce the amount of development, provided there are sufficient drawpoints. All this is only permissible if an economically viable quantity of ore is recovered at an acceptable dilution content. The major technical consideration is the flow of ore into drawbells, set at the optimum spacing for the interaction of ore of that fragmentation and friction. **There are other important practical considerations, namely, number of drawpoints required for production, the availability of drawpoints, the effect of closing drawpoints when there is a wide spacing, changes in fragmentation with draw and the fragmentation of the waste material.**

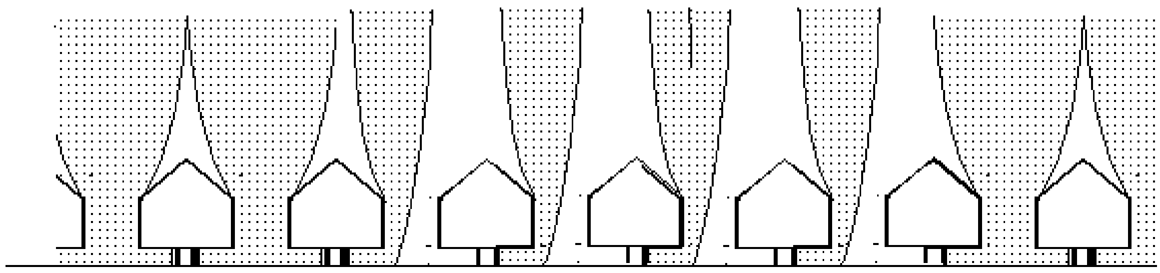
DESIGN PARAMETERS

General

The recommended design parameters are discussed in the following sections and are based on 3-D sand model tests, fragmentation, marker experiments and empirical data from underground observations. Hopefully the PFC program will be of use to solve some of the unknowns. Some operators have bitten the bullet and increased spacings beyond that recommended by the DHL fragmentation design curve. In these cases ore recovery, dilution entry and dilution percentage need to be monitored carefully. The final proposed changes to be made at Henderson mine are significant - the area of influence was first increased from the original of 148m² to 190m² = 28% and not that significant, but the plan is now to increase from 190m² to 300m² = 160% and this is significant. What is more important is that the spacing of the drawzones over the major apex has gone from 17m to 18m to 20m. This is a major change and will be watched with interest as it leaves no room for closed drawpoints and poor draw. The argument that Henderson have used is that in areas where they have sealed drawpoints the ore recovery has been acceptable. This can be misleading in that if a length of a production drift is sealed off and there is good draw on either side then boundary conditions exist and the bulk of the ore overlying the sealed section will be drawn.



This argument does not apply if every second drawpoint were sealed off, as then the draw would be isolated.



Fragmentation data

There is a definite relationship between fragmentation and the isolated drawzone diameter and therefore drawpoint spacing. The fragmentation data must be presented in a form so that it can be used to determine the isolated drawzone diameter. There will be a difference in isolated drawzone with a large range in fragmentation even for the same % $+2m^3$.

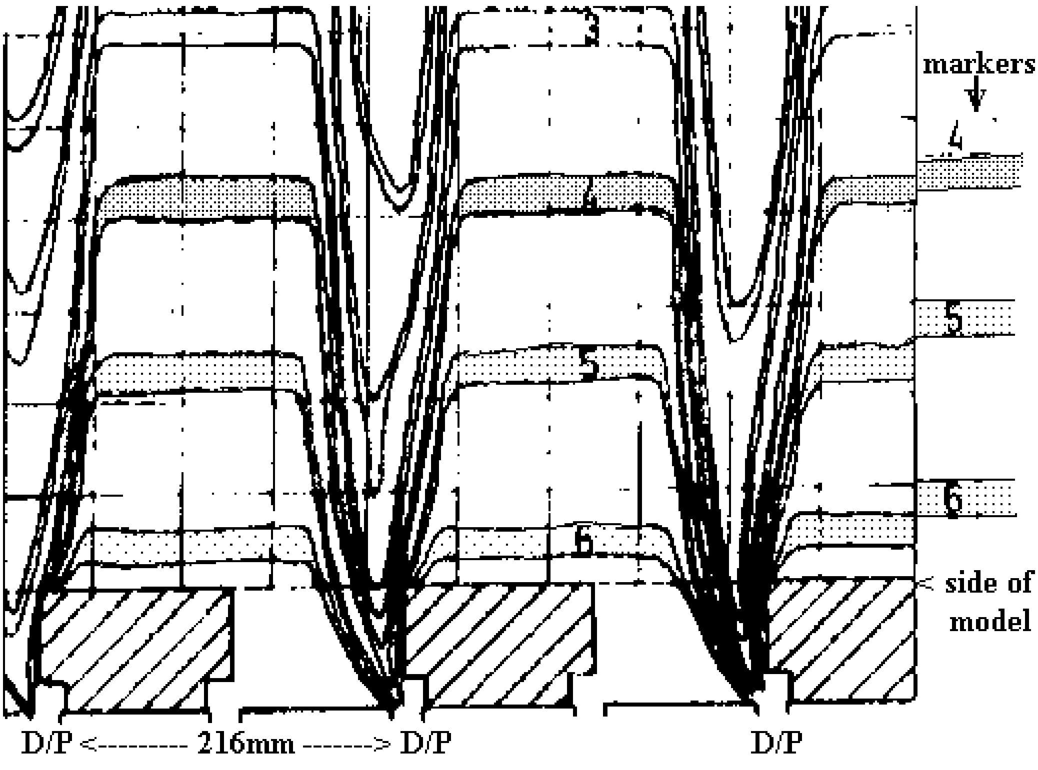
Arching - role of PFC

Arching is a fundamental part of caving and is more pronounced the larger the fragmentation. The formation and breaking of large arches is the mechanism for ground to move over large areas. Arches with a 30m span have been observed underground. This is an area where the PFC program might be able to assist in determining the three dimensional results of arch failure.

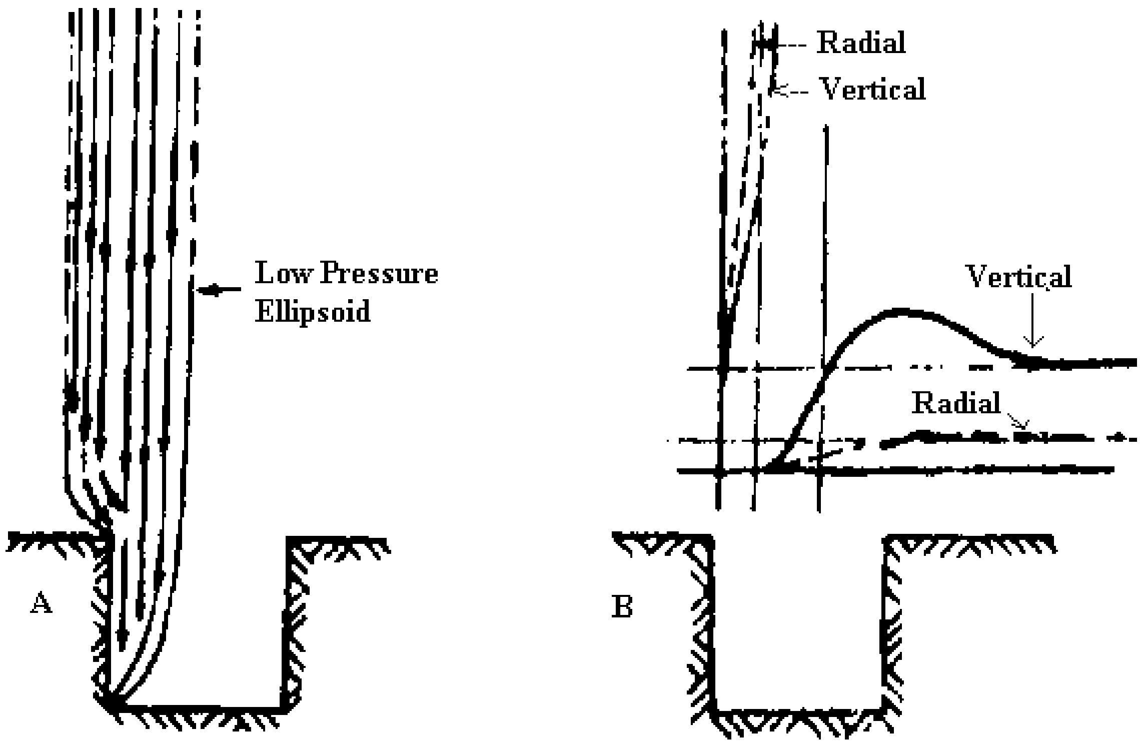
MODEL STUDIES

The following diagrams are sections through a three dimensional sand model with 50 drawpoints at a spacing of 108mm. The model with a height of 2400mm high and a base of 760mm x 760mm, was developed at Shabanie Mine in the late 1970's to determine loading on the base of a cave, the ratio of load distribution between sides and base, and, eventually to carry out draw control experiments for block caving and sub level caving methods. The model at the time was the only three dimensional model in use. Experiments were carried out varying the spacing of the drawpoints and are described by Marano(1980) and Heslop & Laubscher(1981).

Experiment 1 - In the following diagram the drawpoints were located at 216mm and at that spacing there was no interaction between the drawpoints and isolated drawzones developed. The diameter of the drawzone was measured at 108mm.



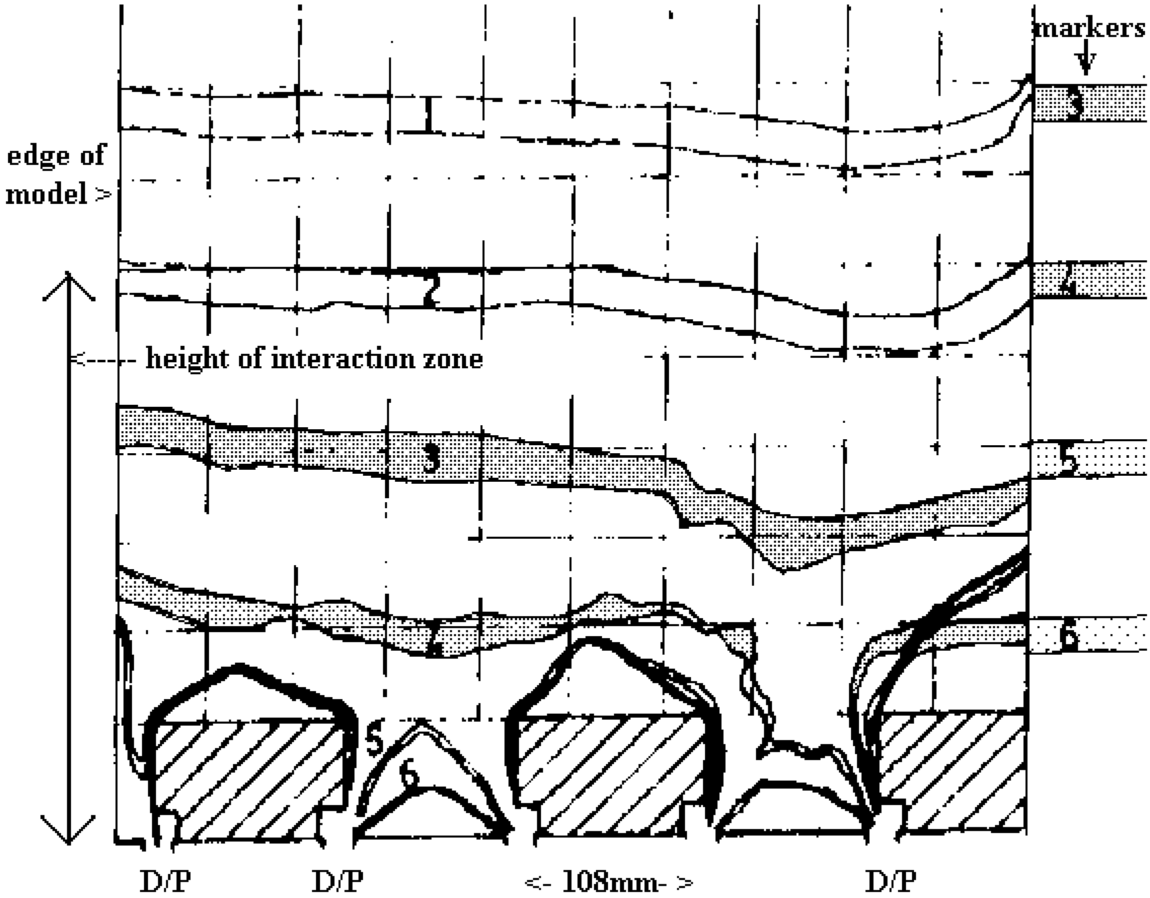
The model tests have shown that when an isolated drawpoint is worked an ellipsoidal zone of low density material develops, in which the vertical and lateral stresses are reduced by drawdown. With draw, this zone extends through to surface. The draw column rapidly widened out to a finite diameter a short distance above the major apex and remained at this diameter to surface. Frictional forces exerted by the subsiding draw column on the static surrounding material increase the vertical stresses in the adjacent static ground while the radial stresses are reduced, but not sufficiently to induce lateral material movement in this zone.



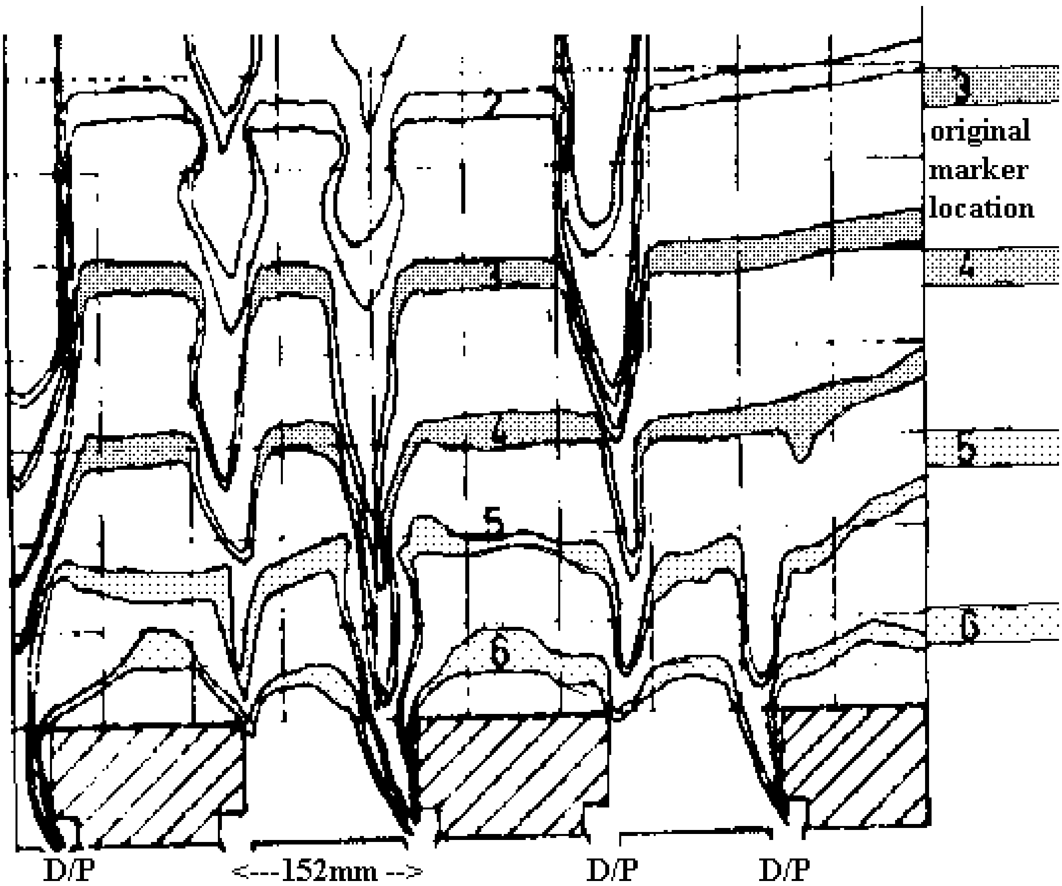
Flow lines and inferred stress around an isolated working drawpoint

Experiment 2 - In the next experiment all drawpoints were drawn and this resulted in a uniform drawdown of the material a certain distance above the drawpoints. This gave rise to the concept of drawpoint interaction and the height of the interaction zone (HIZ). It is evident that there was low angle movement of material over the major apex and that markers 6 and 5 had been drawn out indicating that nearly all the material above the base of marker 6 had been drawn. The conclusions were:-

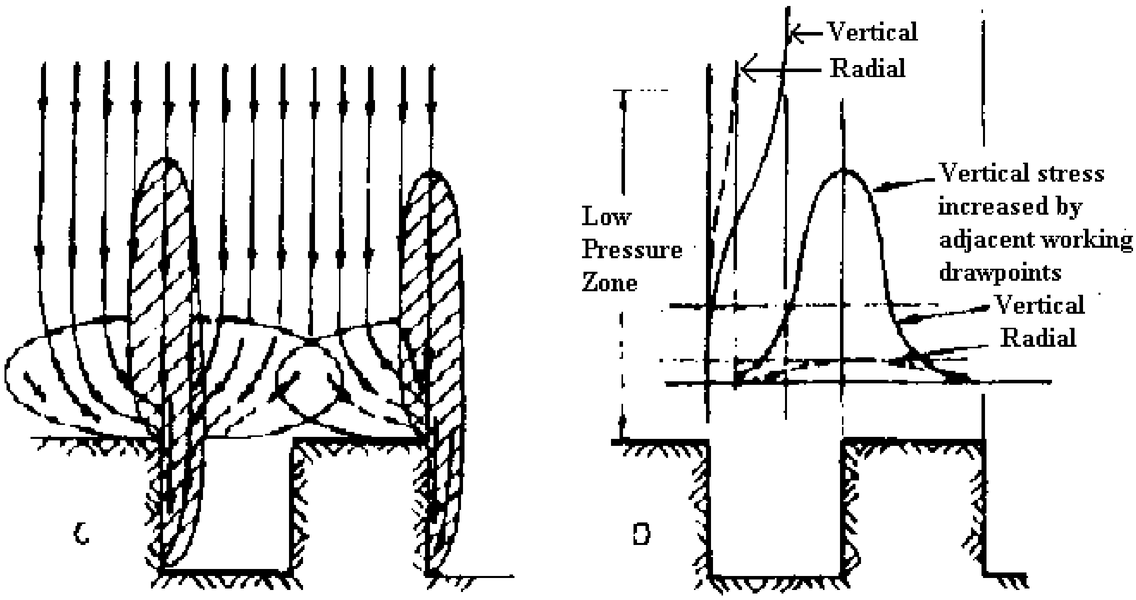
- Good interaction between drawpoints and low angles of draw over major apex
- Limited mixing in the draw column and low dilution



Experiment 3 - In the next experiment the drawpoint spacing was increased to 152mm with a fairly uniform draw from the drawpoints. The horizontal limit of the interaction zone was no longer clearly defined owing to a high degree of mixing, but, overall a drawdown of material between the drawpoints with markers 6 and 5 showing a distinct thinning over the major apex.



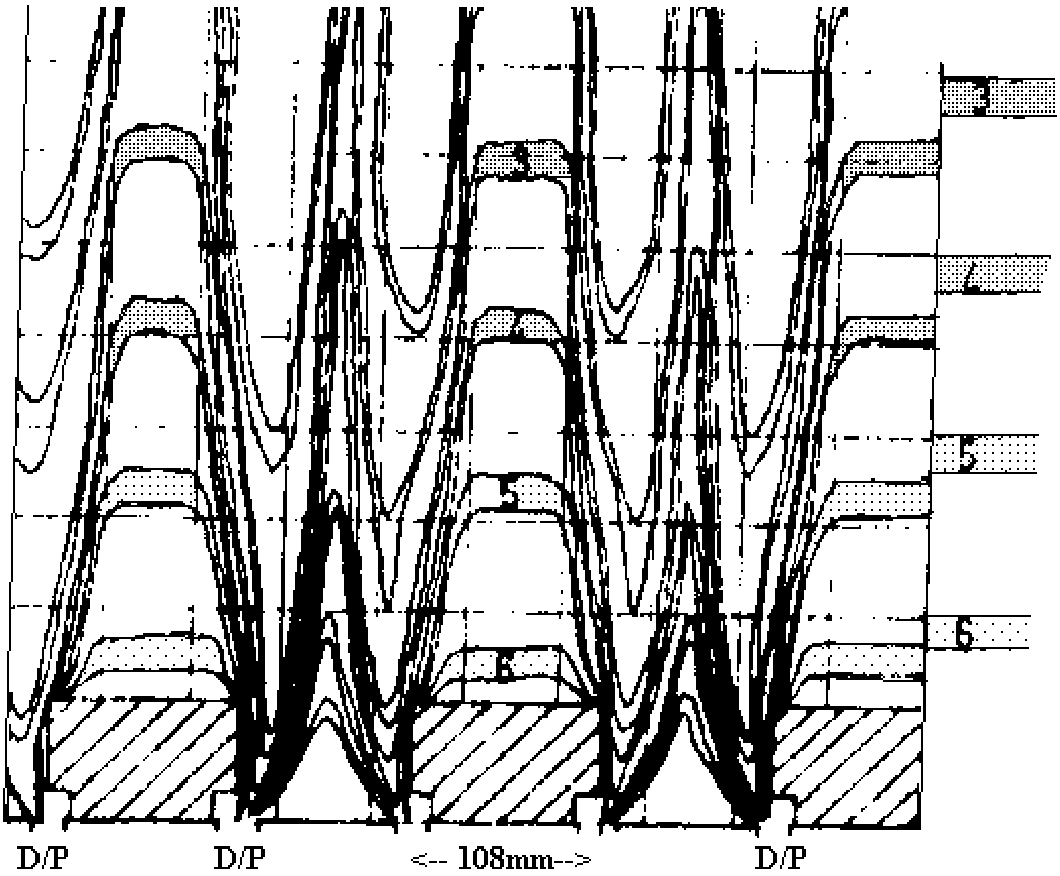
Lateral stresses in the induced stress zone surrounding a draw column is possible if the vertical pressure is further increased by working adjacent drawpoints.



Flow lines and inferred stresses between adjacent working drawpoints

Experiment 4 - The following diagram shows the situation if drawpoints are spaced at 108mm but are drawn in isolation i.e. that is a time period between adjacent drawpoints.

It can be seen that a large percentage of material is left in situ, particularly over the major apex. This is the situation that occurred on Andina Mine where drawpoints were spaced at 8m but the draw on a daily basis was excessive from the drawpoints and there was no interaction as the orepasses could be filled with 800 to 1000 tons in a shift.



There is still a need to continue with three dimensional sand model tests and to tie these results in with the numerical modelling done with the PFC.

Two dimensional models are of no use in block caving analyses. They are used to model sub level caving situations. Whilst 2-D models show different flow characteristics for a variation in the material used e.g. sand, magnetite, lead shot and by varying the packing density from compact to loose. These confirm what is to be expected in practice.

UNDERGROUND MARKER EXPERIMENTS

Behaviour of markers - Markers have been placed above production levels on many mines. The shape and density of the marker should relate to the material being drawn, old tires are often used, but these do not resemble cave material. Concrete blocks with number encased in the concrete were used at King Mine. The movement of these markers does not always conform with the theoretical interpretations. A lot of the marker experiments have been done on mines where there is an irregular draw and poor draw control. What is clear is that markers can move appreciable distances down channelways and in these cases this information should not influence decisions on the spacing of drawpoints. At King Mine, markers confirmed that the angle of draw was towards higher ground. Movement of markers placed over the major apex could be used to confirm draw patterns.

"A paper by J. Alvial in Massmin 92 describes the Teniente experience with the recovery of markers of old tyres. As the tyres were not filled with concrete they were not an ideal marker, but, the results did show the lateral movement of material, particularly with varying draw. These conclusions were made by Alvial :-

- In general the markers only provide slip information. The trajectory will depend on how the draw is handled, the rate of draw, the characteristics of the broken ore, such as moisture and size, the draw (zone) spacing, and structural - geological constraints such as faults and lithological contacts that define preferential directions of flow.

- The horizontal displacement of the markers range between 2 and 42m with an average of 14.5m.

- The vertical slip angle ranged from 60° to 88°, with an average of 80°

- The small number of recovered markers (NOTE : He does not state the number that were placed) enables general opinions to be formed in regard to the internal gravity flow. In general, only trends can be observed: the tyre markers flow is orientated towards areas of high draw. Figure 14 in the paper, shows two groups of markers that identify areas of high draw rate: markers 5, 6, 11 and 12 are orientated in the same direction - north to south, when related to markers 3 and 4 a high rate zone is identified. On the other hand markers 8, 9, 10, 13, 14, 15 and 16 are orientated in a circular shape, establishing a type of whirlpool around an area of high draw."

Marker programs are very useful, but, markers should resemble the material being drawn and should be placed in strategic surveyed locations. Markers that are not recovered must also be recorded in the analysis. Marker experiments are only of use if they are done with a sound draw control program. No conclusions can be drawn in areas where there is haphazard draw as occurs on many mines.

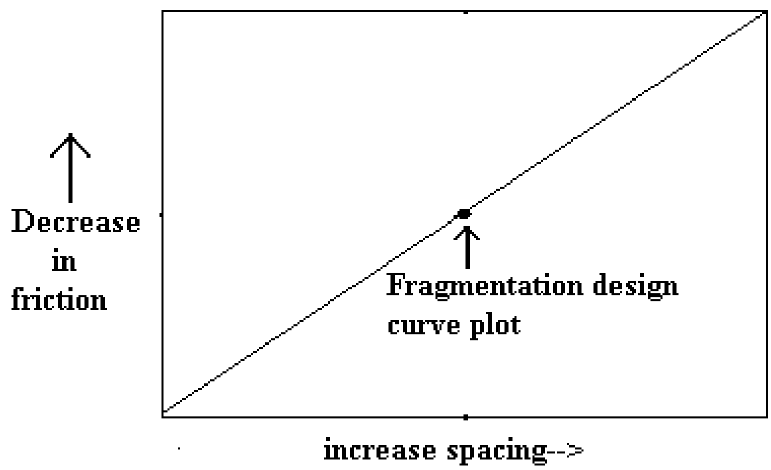
DESIGN PARAMETERS

Interactive draw

The concept of interactive draw was developed from the 3-D model studies and has been proved in practice. At Nchanga Mine in the original slusher layout, the dimensions were finger raise diameter of 1.8m and drawpoint spacing of 5m with slusher drift spacing of 12m. This resulted in isolated draw as could be seen when the pit exposed the previously mined areas. The dimensions in the current operation are a finger raise diameter of 2.2m, a spacing of 4.5m and drift spacing of 9.7m. These new dimensions have resulted in an interactive draw.

The frictional properties of caved material

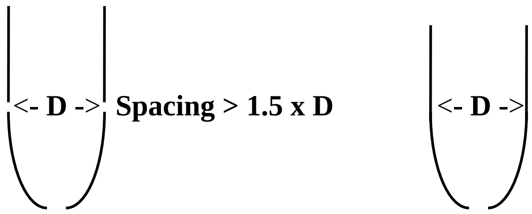
How do the frictional properties of the caved material affect the flow characteristics and therefore the drawpoint spacing. It is logical to assume that low friction material will flow at a lower angle than high friction material, but how do we define the friction of the material. The following diagram shows that if the fragmentation design curves are based on an average friction then a decrease in friction should mean an increase in spacing. The question is, by how much?



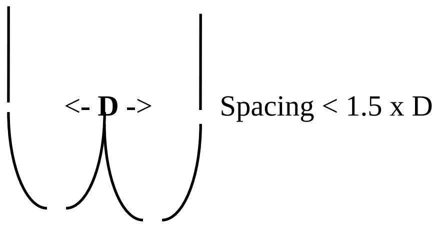
Isolated Drawzone diameter and interaction - role of PFC

The physical model tests and underground observations of finely fragmented ore, showed that drawpoints remote from the influence of other drawpoints had a drawzone with a finite diameter. This knowledge was used to create isolated drawzone diameters for different fragmentation ranges. However, the isolated drawzone diameter of the very coarse fragmentation range was not known, but based on underground evidence of arching and the area of influence of collapsed arches, it was assumed to be 12m to 13m depending on the width of the loading zone. The diameter for the fine material was known to be 6m, thus it was assumed that there was a linear relationship so that medium fragmentation would be 8m and the coarse 10m.

Three dimensional sand model tests have shown that there is a relationship between the spacing of drawpoints and the interaction between the drawzones that form above the drawpoint. Widely spaced drawpoints develop isolated drawzones or IDZ, the diameter of which is defined by the fragmentation sizes. The zone has a cylindrical shape tapering at the base to the drawpoint



Sand model tests and the interpretation of stresses around underground excavations show that when the IDZ is spaced at more than 1.5 times the diameter (**D**) then there is no interaction. If the distance is less than 1.5 x D then there is interaction, the degree of which increases with reduction in spacing



The sand model experiments have been confirmed by the drawing of fine material and the behaviour of material in bins. The question is whether the interactive theory can be totally applied to coarse material where arch spans of 20m have been measured in hangups. The collapse of these arches will affect a large area. As arches of different spans will be forming and breaking their influence must be recognised. If the span of the arch was assigned to the IDZ then drawzone spacings of 30m would be permissible and this is not the case because arching is an intermittent occurrence and has a local effect giving rise to a pulsating variation in drawzone diameter.

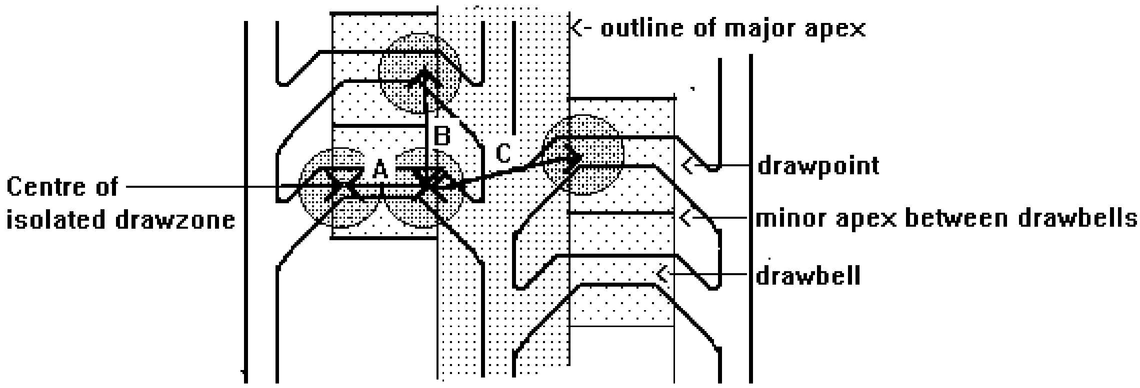
The centre of the drawzone is arbitrarily taken as 1m from the brow. It can be argued that this figure should vary according to the diameter of the isolated drawzone and that it should not be the same for a grizzly layout and a nominal 15m spacing. On reflection there is no real reason to change it. The following table was developed and has formed the basis for design. The principles remain and if more accurate data is available then the diameters could change, for example, the very coarse could be 14m. In the table isolated drawzone diameter - IDZ- is related to RMR and ff/m and in the lower section the permissible maximum and minimum spacing are also related to the RMR and ff/m. This table has proved to be a good guideline for design purposes:

Rockmass class	5	4	3	2/1
ff/m	50 - 7	20 - 1.5	5 - 0.4	1.5 - 0.2
Rock size m	0.01 - 0.3	0.1 - 2.0	0.4 - 5.0	1.5 - 9.0+
% +2m³	0	1 - 5	6 - 20	21 - 45
Loading width		Isolated drawzone diameter		
5m =			11.5m	13m
4m =		9m	11m	12.5m
3m =	6.5m	8.5m	10.5m	12m
2m =	6m	8m	10m	
Loading width		Maximum / minimum spacing of drawzones		
5m =			21/12m	24/14m
4m =		16/8m	20/11m	23/13m
3m =	10/5m	14/7m	19/10m	22/12m
2m =	9/4m	12/6m	17/9m	

Area of D/P influence m² ----> 95 -----> 180 ----> 290 -----> 380

Drawzone spacing - across minor/major apexes

Drawzone spacing - drawpoint spacings for grizzly and slushers generally reflect the correct spacing of drawzones, though there is a tendency to increase these spacings. However, in the case of LHD layouts the spacing of drawpoints is a nominal figure, for example, a drawpoint spacing of 15m could have drawzone spacings ranging from 6m in the drawbell to 25m across the major apex. It is often the case that drawpoints are made long to accommodate LHD's that are too large for the layout or under the misguided impression that a LHD has to be dead straight before the bucket / scoop enters a fine muckpile. Thus the major factor of optimum ore recovery is prejudiced by incorrect equipment selection. The following diagram shows how these distances vary in the drawbell, across the minor apex and across the major apex :-



A = distance between drawzones in drawbell e.g. = 8m
B = distance of drawzones across minor apex e.g. = 15m
C = distance of drawzones across major apex e. g. = 22m

As can be seen from the above diagram the spacing in the drawbell is out of phase with the other dimensions, therefore there is no problem in increasing the production drift spacing to 32m as this would only increase the drawbell spacing. The diagram also shows that the length of the drawpoint must be kept to a minimum so as to keep the spacing across the major apex down

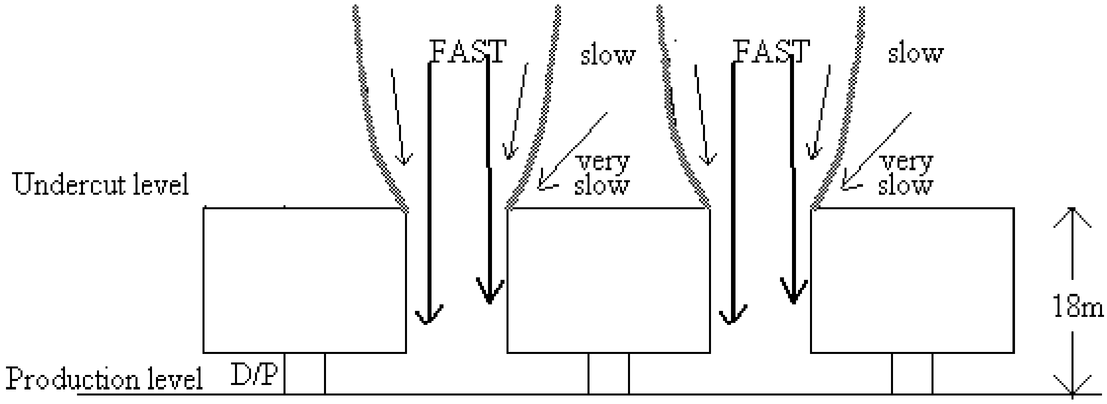
Influence of Draw Strategy

The draw strategy can have a significant influence on theoretical and practical drawzone spacing. The theoretical drawzone spacing is based on a sound draw strategy to ensure interaction between drawzones as described in section 25. If in practice, this is not done then ore recovery decreases and dilution increases.

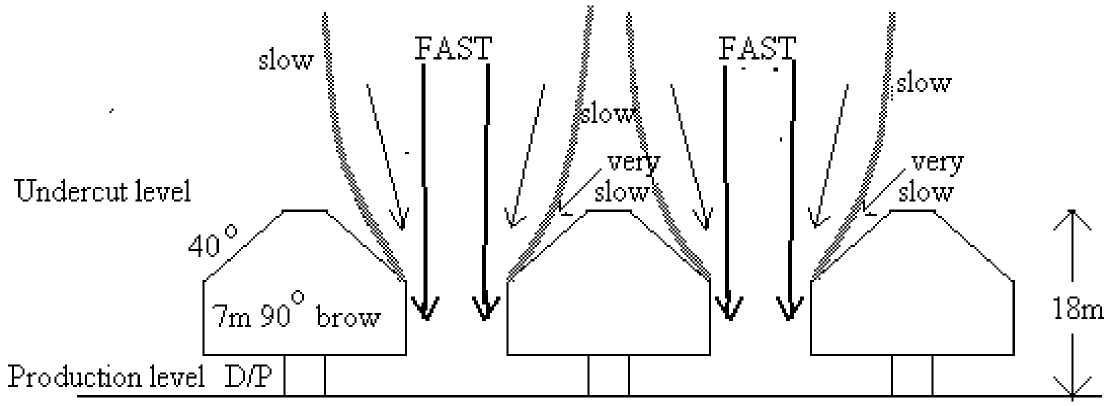
Preferred Drawbell Geometry

The term drawbell is descriptive in that in theory the idle shape of the drawbell is like a bell, so that ore can flow to the drawpoint. However, it is a compromise between strength and shape. The major and minor apexes must have sufficient strength to last out the draw life. It needs to be established how much influence the shape of the drawbell has on interaction. The following diagrams show that shaping the drawbell will improve interaction by increasing the size of the interaction zone. It has always been an empirical point that shaped drawpoints improve ore recovery. PFC modelling might confirm the practical belief.

The following sections through the drawbells across the major apex show that the shaped drawbell should have better flow characteristics than a drawbell with vertical faces and a large flat top major apex. The ideal draw situation should be if the drawbell boundaries are contiguous.

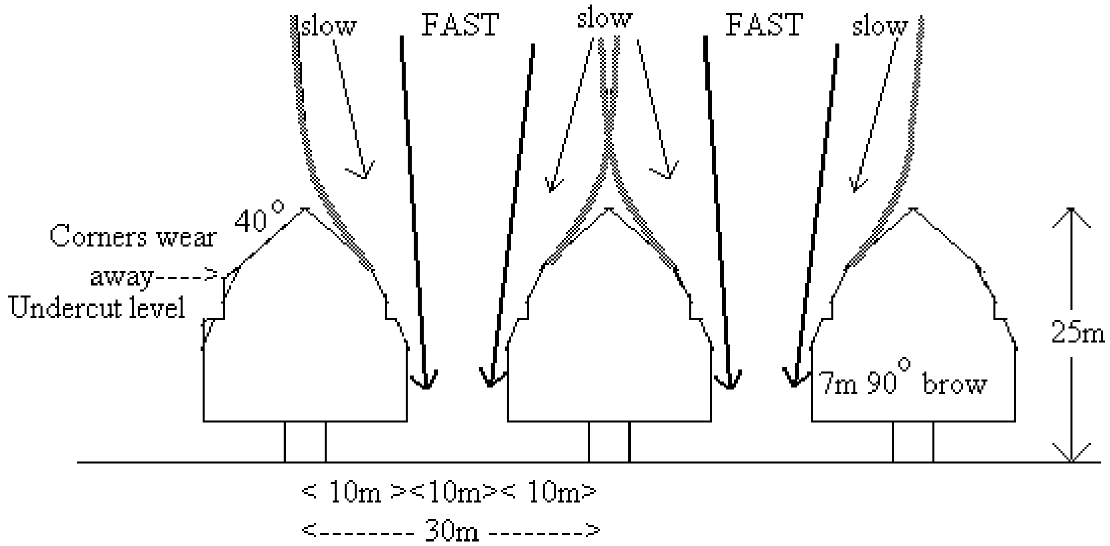


NARROW HORIZONTAL ADVANCE UNDERCUT - POOR INTERACTION



SHAPED DRAWBELLS - HENDERSON/CONVENTIONAL - GOOD INTERACTION

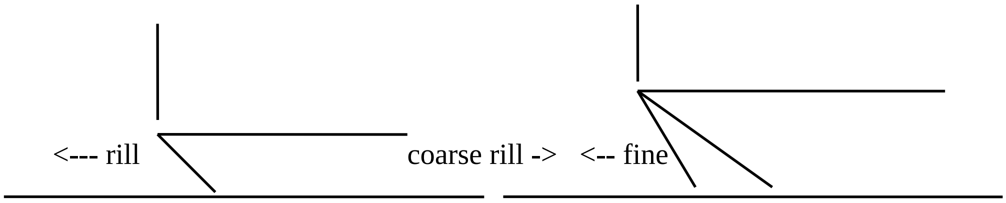
If the undercut level is raised to 20m the slope of the drawbell can be increased



ADVANCE INCLINE NARROW UNDERCUT - GOOD INTERACTION

Drawpoint size

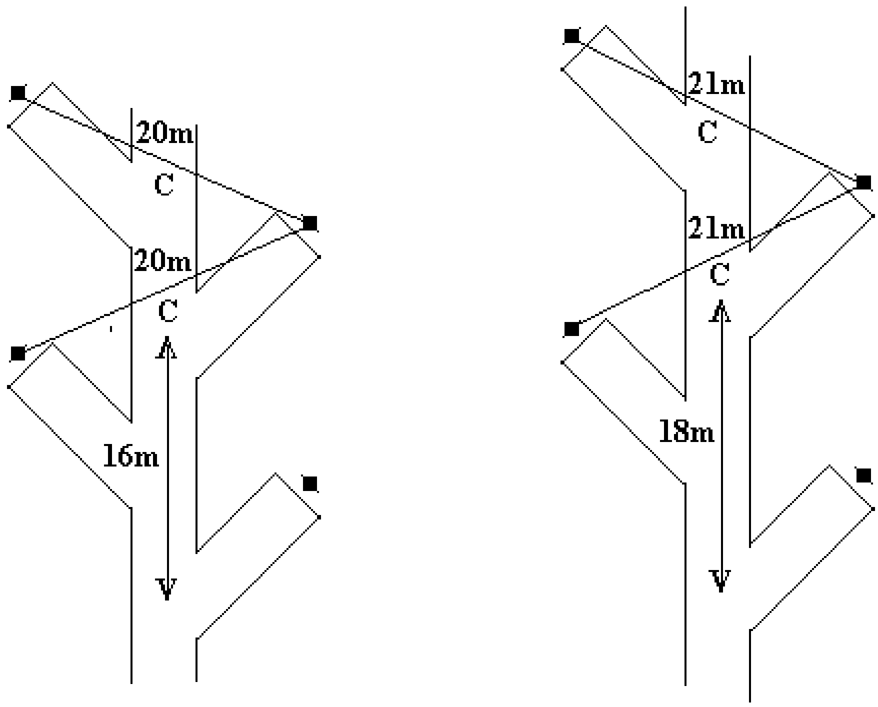
The size of the drawpoint is a function of the rock mass strength and the support requirements. The height of the drawpoint is a function of rock mass strength and will determine the rill distance from the brow. The rill angle is flatter for fine material than for coarse material.



As the LHD will load on the same line in a restricted horizontal layout there is no point in having a drawpoint much wider than the LHD as rocks can move behind the front tyres during loading and cause damage. This does not apply in an Incline layout or Front cave where the drawpoints are long and LHD's can be positioned to make full use of a wide drawpoint.

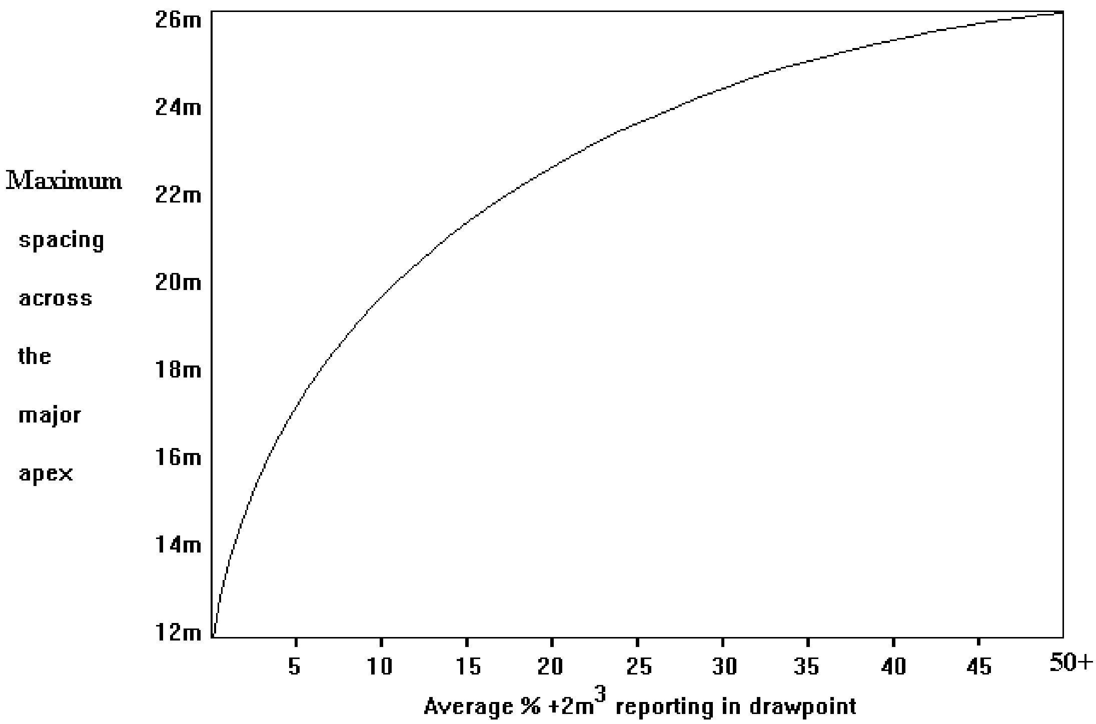
Drawpoint spacing - along drift - in Drawbell - drift spacing

The drawpoint spacing has to be related to an acceptable drawzone spacing and not according to the size of the LHD. A LHD must operate efficiently within the prescribed space. If it hits sidewalls regularly and damages support as well as the machine, it will be found that a smaller machine will out perform the larger machine, because of the better haul and load times. Unfortunately, layouts are often designed on the basis that a LHD must be straight to load, this has been disproved by Henderson Mine where LHD's start their loading at an angle and are straight when under full load. It is claimed that the extra wear on the articulation system is negligible when compared to the draw benefits.



DRAWPOINT SPACING BASED ON THE % + 2M³ OVERSIZE

The relationship between the minimum spacing across the major apex - C in the above diagram - and the average + 2m³ reporting in the drawpoint, is shown graphically overleaf. The curve flattens because there must be a limit to the movement of material. Another very important point is that as the rock mass competency increases so do the joint condition ratings which reflect the higher friction between the caved particles. Thus serpentine with talcose fines will move more readily across a major apex than primary diorite with high friction fines.

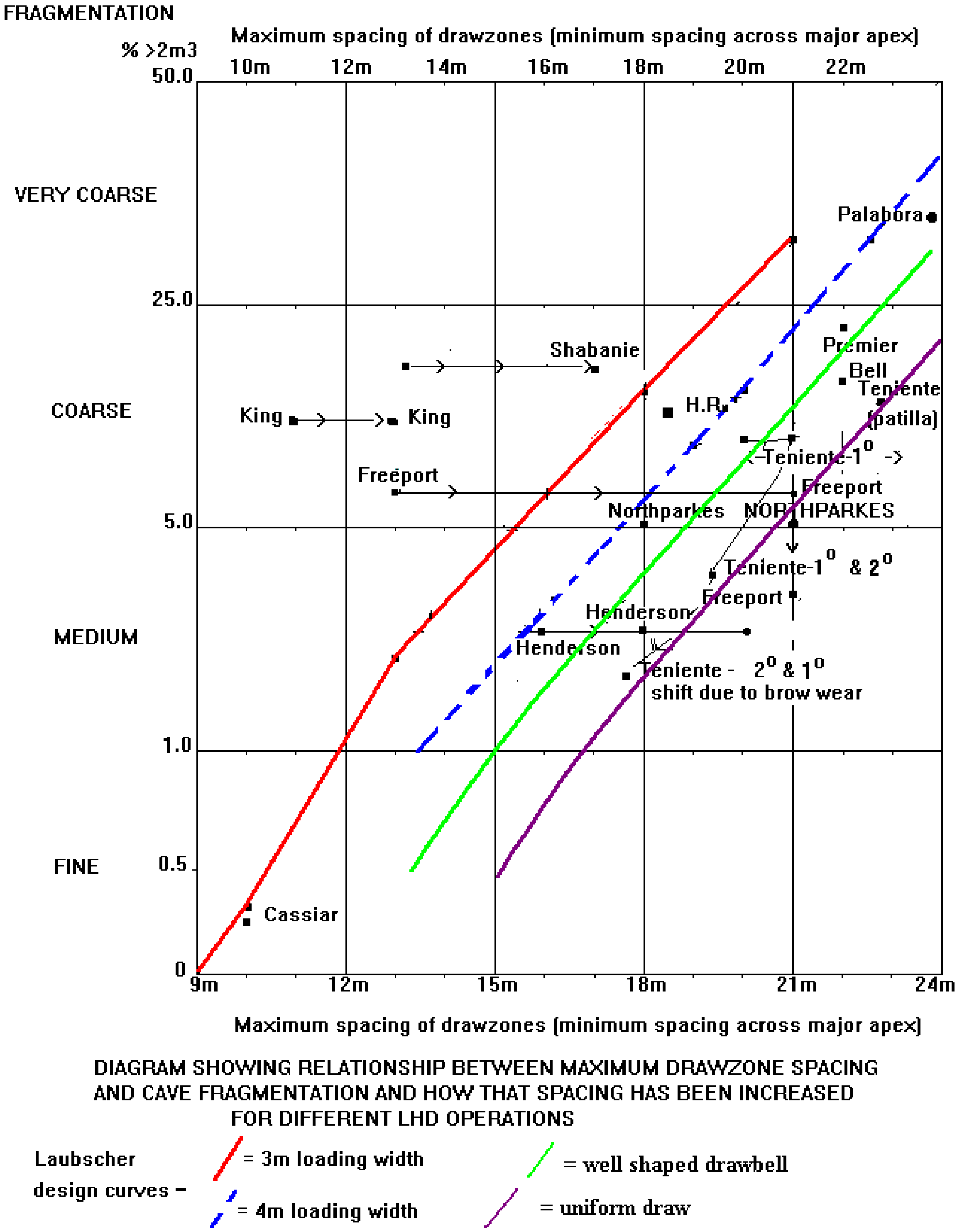


Examples of maximum spacing relative to the +2m³ content for different mines is shown graphically overleaf. Extra curves have been put onto this graph to indicate other factors which will influence the flow of caved material:

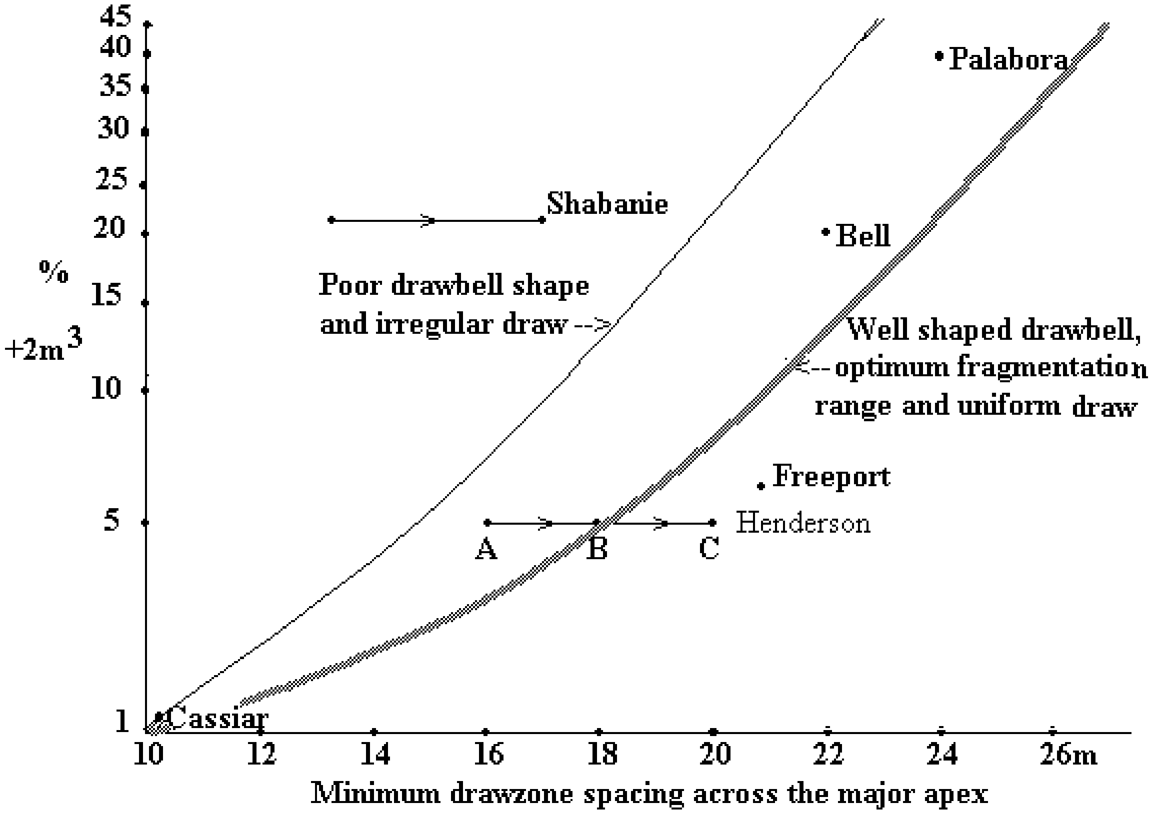
- Increasing the width and dig in the drawpoint will mean a larger draw area.
- A well shaped drawbell will allow unimpeded flow of material
- A managed draw control program where the object is to obtain interactive draw by short term high draw and long term uniform draw.

The spacing of the extra curves is purely arbitrary at this stage and are put in to illustrate a point.

It will be noted that there are several plots for Teniente draw zones. This is because there is a large range in fragmentation as finer extraneous material is drawn, also the drawzone spacing changes owing to the significant wear back of the brow.



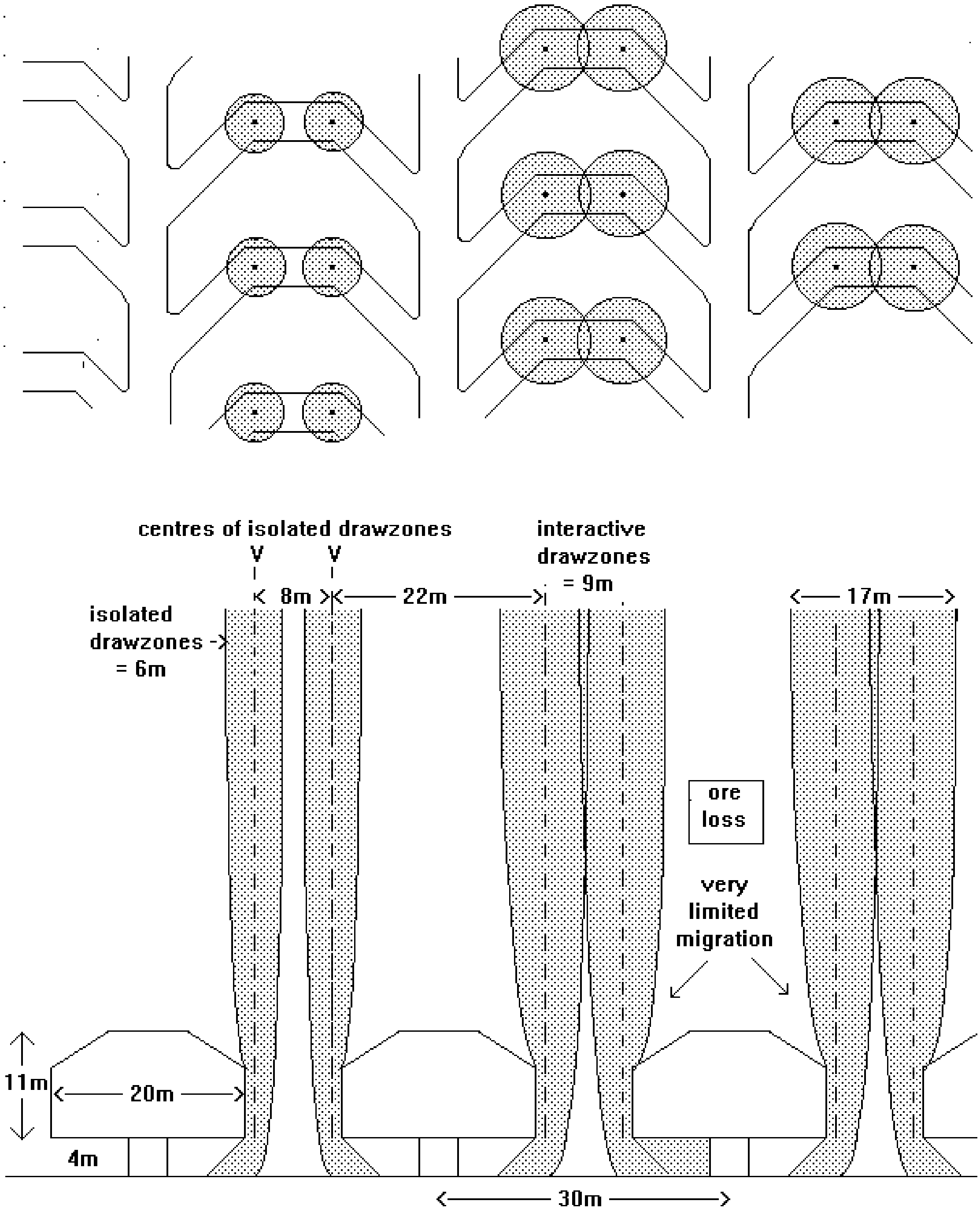
The following graph is an abridged version and can be used for design purposes.

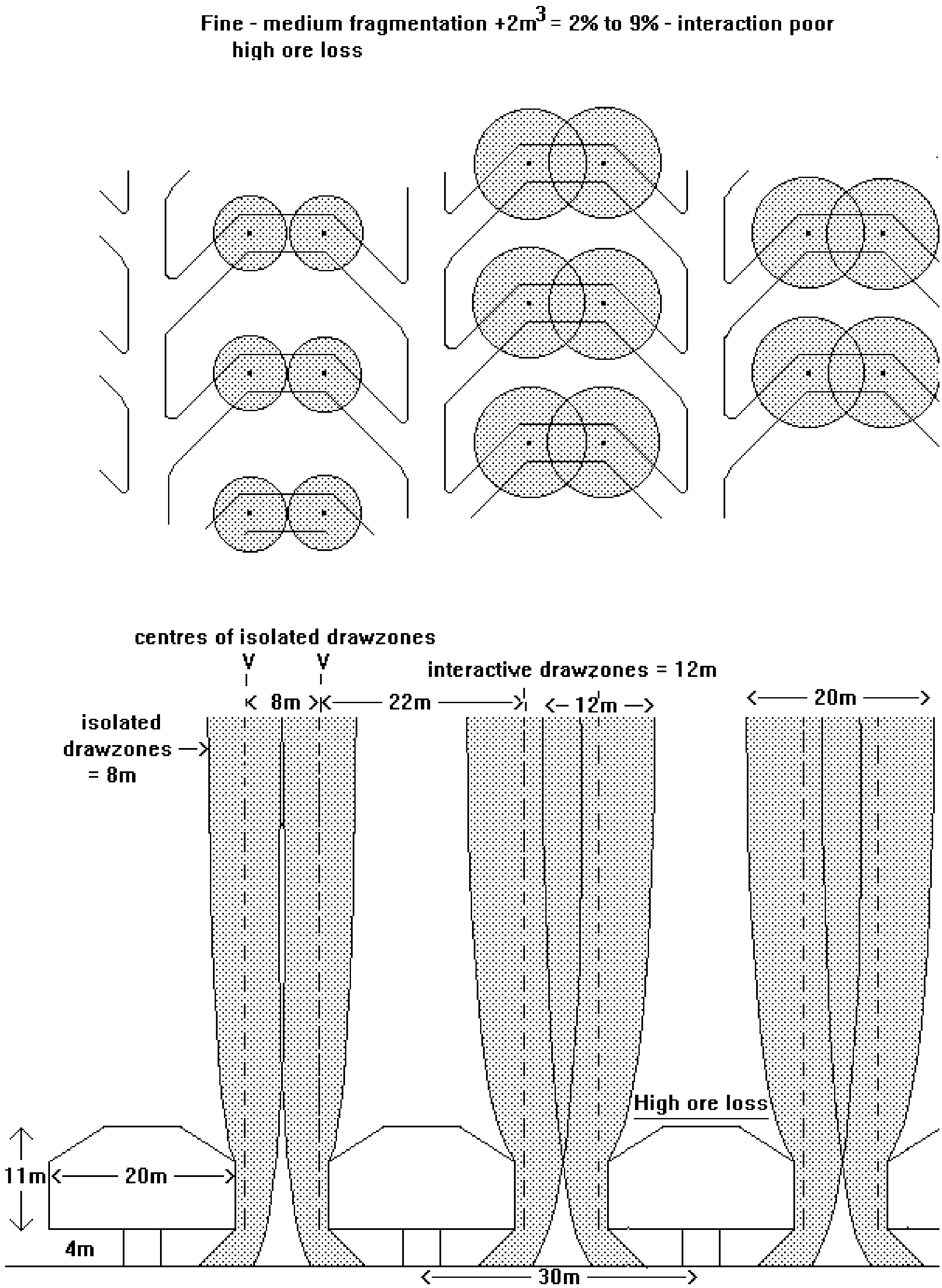


ISOLATED DRAWZONES

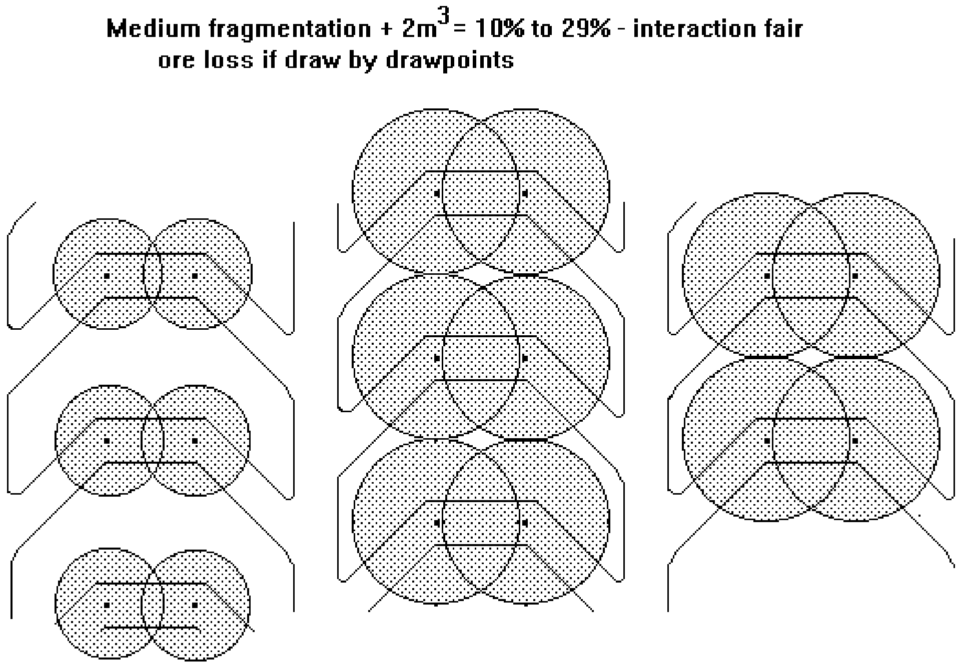
The following series of diagrams show plots of isolated drawzones for different fragmentation ranges from fine to very coarse in a standard nominal 15m drawpoint spacing layout. That is, drawpoints at 15m along the drift and the production drifts 30m apart.

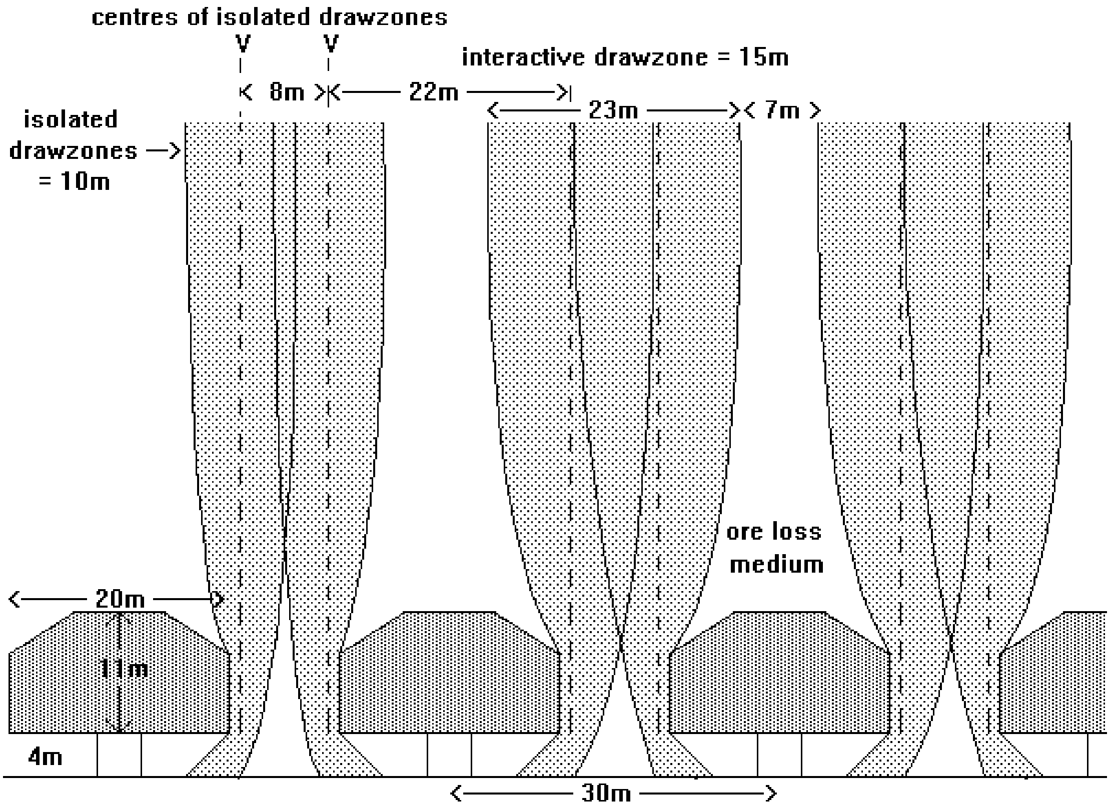
Fine fragmentation +2m^3= 0% - no interaction across major apex with very high ore losses



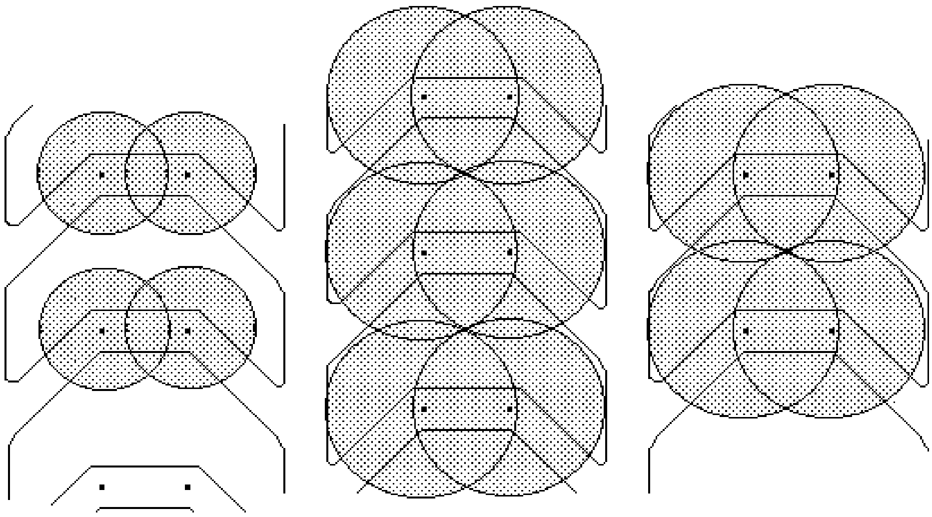


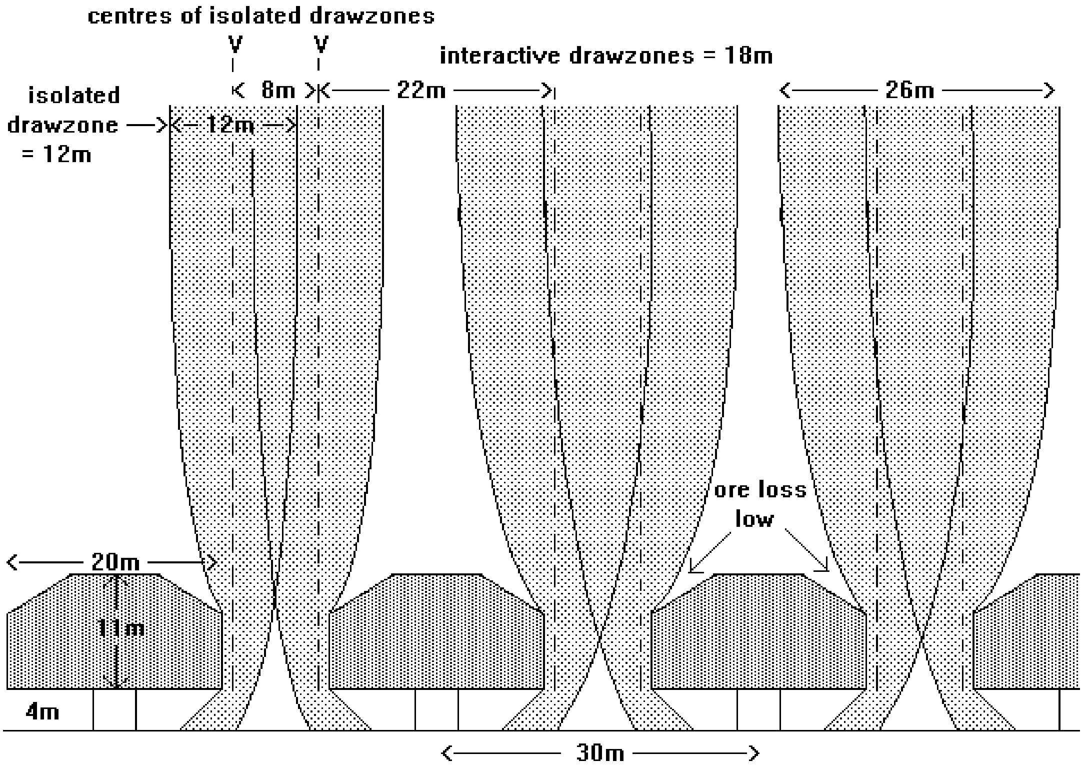
Fine - medium fragmentation +2m3 = 2% to 9% - interaction poor
high ore loss
centres of isolated drawzones
interactive drawzones = 12m
8m
22m
12m
20m
isolated
drawzones
= 8m
High ore loss
11m
20m
4m
30m

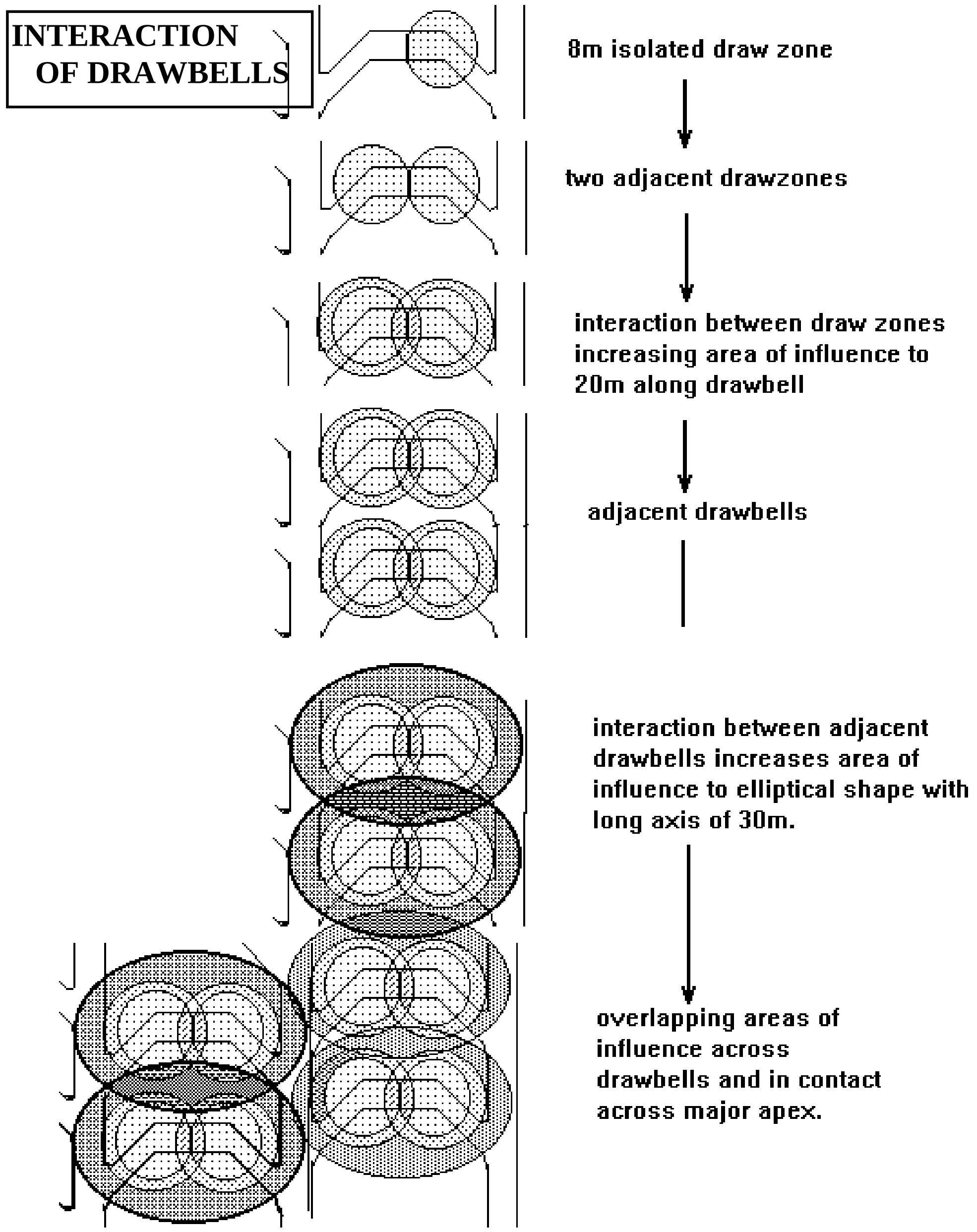

Medium fragmentation + 2m3 = 10% to 29% - interaction fair
ore loss if draw by drawpoints


centres of isolated drawzones
interactive drawzone = 15m
8m
22m
23m
7m
isolated
drawzones
= 10m
ore loss
medium
20m
11m
4m
30m

Coarse fragmentation at +2m^3 = 30% to 50% - interaction good







This drawing shows the development of interaction between drawpoints as the drawpoints are drawn:

- A single drawpoint with it’s isolated drawzone.
- Two isolated drawzones in the drawbell interact to form a large drawzone.
- The drawbell drawzones interact across the minor apex forming a larger interactive drawzone.
- The large drawbell interactive zones interact across the major apex.

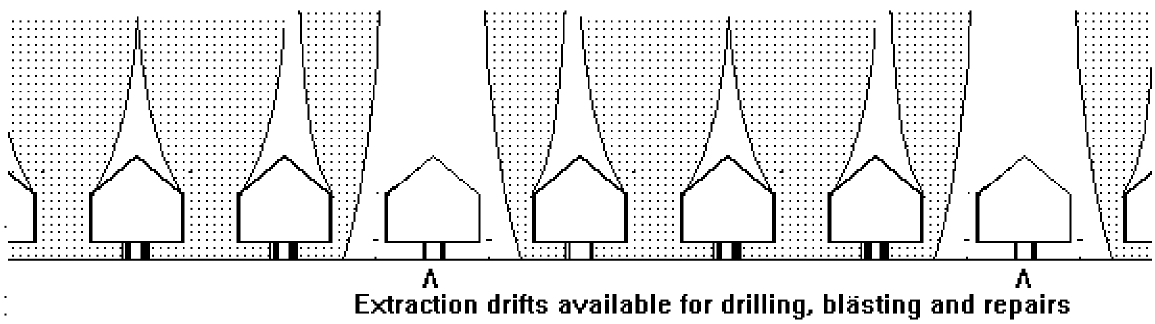
By drawing lines of drawpoints along adjacent drawbells good interaction across the major apex is achieved.

MASS FLOW

It is the contention of some operators that mass flow occurs from over the major apex towards areas of uniform draw of adjacent drawpoints, and that this flow occurs at low angles. There is no doubt that low angle flow occurs when there is interaction, but this low angle flow is related to a strict spacing law and the size of the material. Is the implication that finer material will flow further / at a lower angle than coarse material? There has to be a finite limit for the influence of a drawpoint and the interaction concept, recognising the role of friction and cave stresses is the best tool available at present.

DRAW STRATEGIES TO ENSURE OPTIMUM FLOW TO DRAWPOINTS

The active draw zone is a low density - low pressure area into which it is expected that material from above and the sides will flow. It is important to develop an understanding of the lateral flow characteristics to determine the drawpoint spacing. It is considered that with the correct draw the optimum spacing of drawpoints can be achieved. This means that there must be a defined strategy of drawing drawpoints / drawbells. It is known that there are often practical problems that can sabotage the best intentions, the object of this manual being to identify these problems and to recommend means of combating them. In the following figures, two scenarios are presented. Figures 1A and 1B show situations where alternating lines of drawbells are drawn i.e. all drawpoints of the specified drawbells are drawn. The following day the next line of drawbells is drawn. As it takes at least two days for a drawpoint to settle down, working adjacent lines of drawpoints will result in differential movement and lateral movement of material. This will also set up differential stresses which will encourage secondary fragmentation. However, there are practical problems with this method as all production drifts are in use all the time. The solution lies in not drawing a line in a planned sequence on a day or a shift so as to leave one production drift available for repairs or secondary breaking drilling.



In fig 2 alternate production drifts are drawn. If there is uniform draw, and the production drifts are drawn on alternate shifts as at Henderson Mine then in one day complete lines of drawbells will be drawn and the optimum interaction is obtained. At Henderson mine the closed production drift is used for repairs or if not, it is not ventilated, thereby saving in costs.

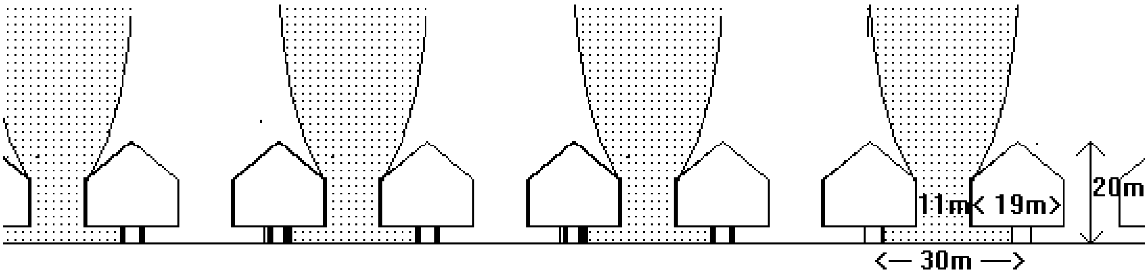


Figure 1 A - draw pattern on day1 with interactive drawbells

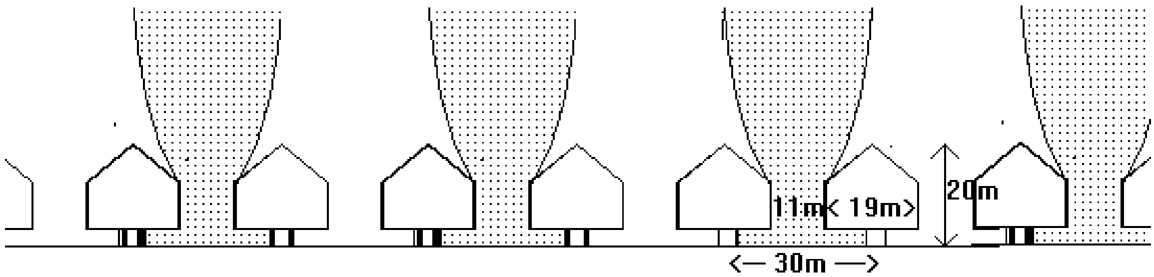


Figure 1B - draw pattern on day 2 with alternating interactive drawbells

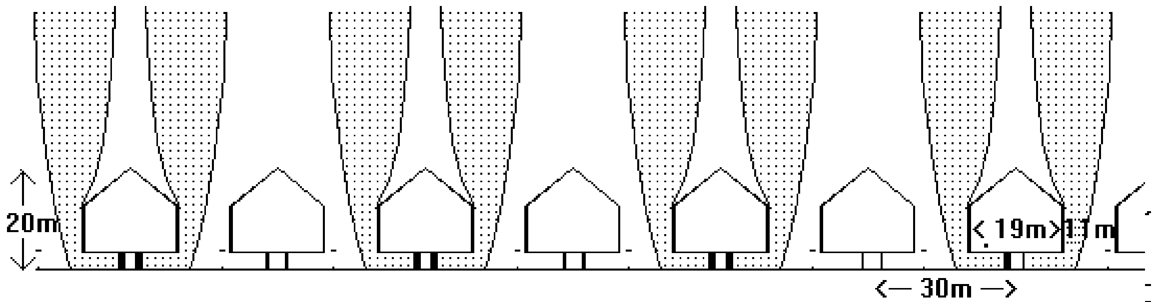


Figure 2A - draw pattern of drawpoints either side of major apex i.e. from extraction drift

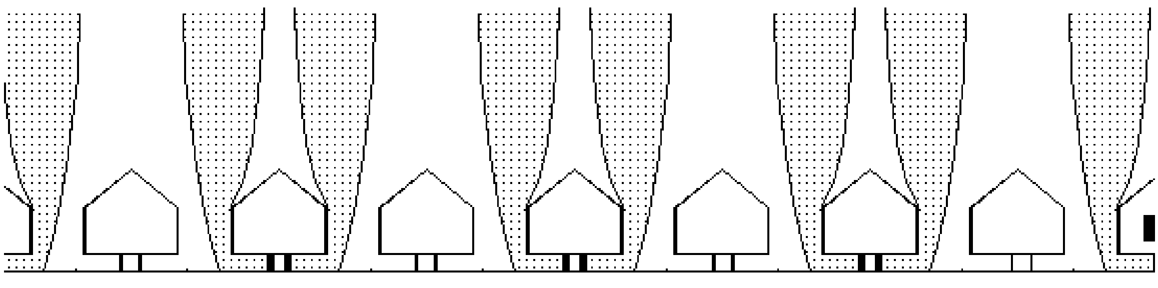
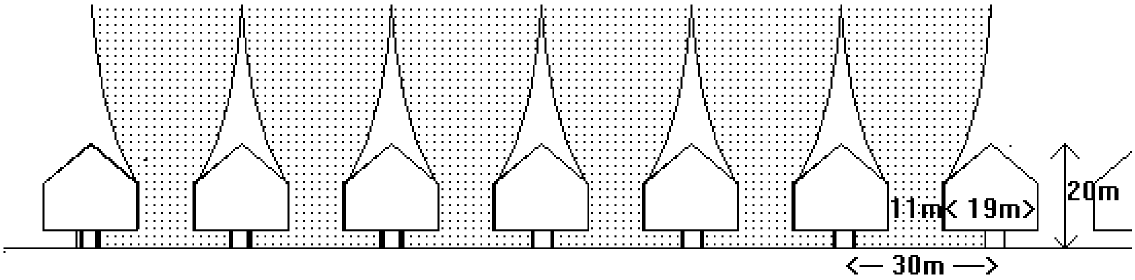
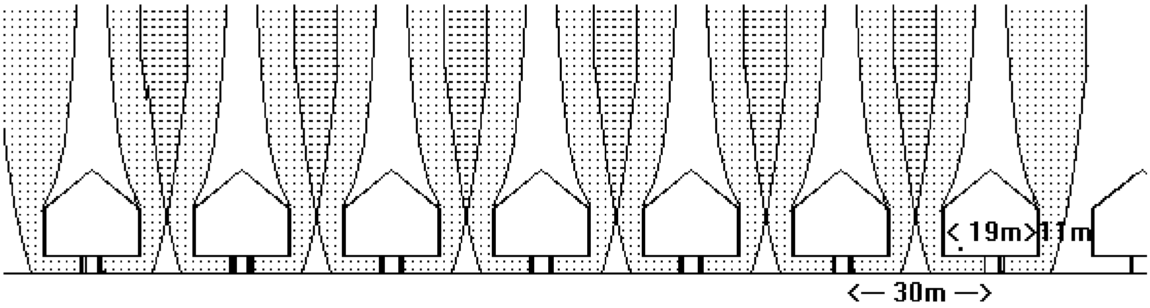


Figure 2B - alternating extraction drifts



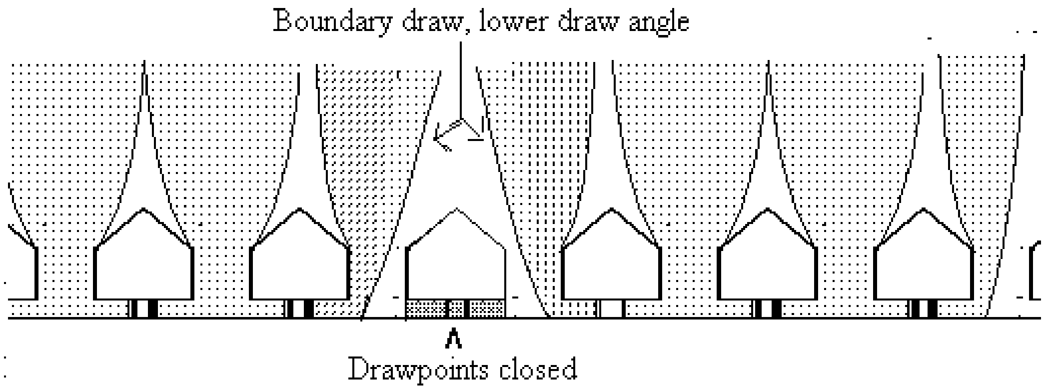
Figures 1A and 1B superimposed



Figures 2A and 2B superimposed

HENDERSON MINE INCREASES THE DRAWZONE SPACING

The drawzone area of influence at Henderson Mine was originally 148m² with a production drift spacing of 24.4m and drawpoint spacing of 12.2m. It was then increased to 190m² by increasing the drawpoint spacing along the drift to 16.2m. The next mining area will be laid out with further increases of 20.6m along the drift and 30.5m between production drifts, giving an area of influence of 300m². The logic behind these major increases is that they have mass flow as a result of the uniform draw, they also refer to the areas where some drawpoints have been closed off with concreted and they believe that they have recovered the ore overlying the drawpoints. This can be misleading, because if several drawpoints are closed, and there is uniform draw from surrounding drawpoints, a boundary situation occurs with reasonable ore recovery.



EFFECT OF INCREASING D/P DRIFT SPACING ON UNDERCUTTING

In the case of Conventional and Henderson undercutting systems, if the spacing along the drift is increased, it could mean that the spacing between the drawpoints becomes too wide to provide effective swell relief during undercutting, so that fully choke conditions prevail and pillars are left over the minor apex. In fact, this was the situation with the 20m spacing used at Teniente and later discarded.

EFFECT OF ABUTMENT STRESS ON UNDERCUT HOLES

Experience on several mines (Philex. San Manuel, Questa and King) has shown that horizontal and downholes are affected in the abutment by closing and cut-offs. This is more likely to occur if there is a wide spacing across the minor apex.

PRACTICAL ASPECTS OF INCREASING THE DRAWPOINT SPACING

Are there sufficient drawpoints to meet the tonnage requirements recognising that wide spacings are only permissible in areas of coarse fragmentation? The productivity in coarse fragmentation will be down with time spent on secondary breaking and bringing down hangups as well as allowing the hangup to stabilise.

COMMENTS FROM N.J.W.BELL - SHABANIE MINE

The size of LHDs and their digging depth in relationship to the length of the drawbell must be considered.

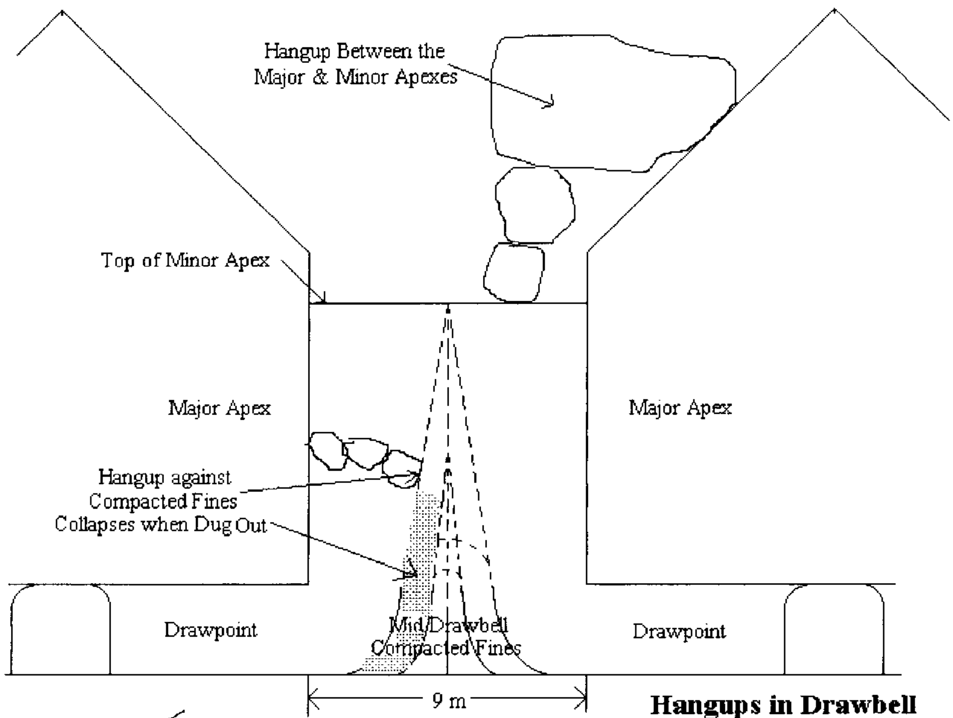
A study of the drawpoints at Shabanie and Gaths Mines (loading with 2 yd LHDs) indicates that during normal loading digging depth is only in the order of 1.0 metre. However, when hang-ups occur, the digging depth at floor level increases to 2.5 or 3.0 metres. Further, hang-ups are very rarely seen across the two halves of a draw pair (common drawbell). This is probably owing to the excavation of the compacted fines as the hang-up is exposed and hence the hang-up collapses. Hang-ups tend to occur against the minor and major apexes in the drawbell. These observations may just be the nature of Shabanie and Gaths rock, where the fines compact very solidly.

See Sketch over leaf – Hang ups.

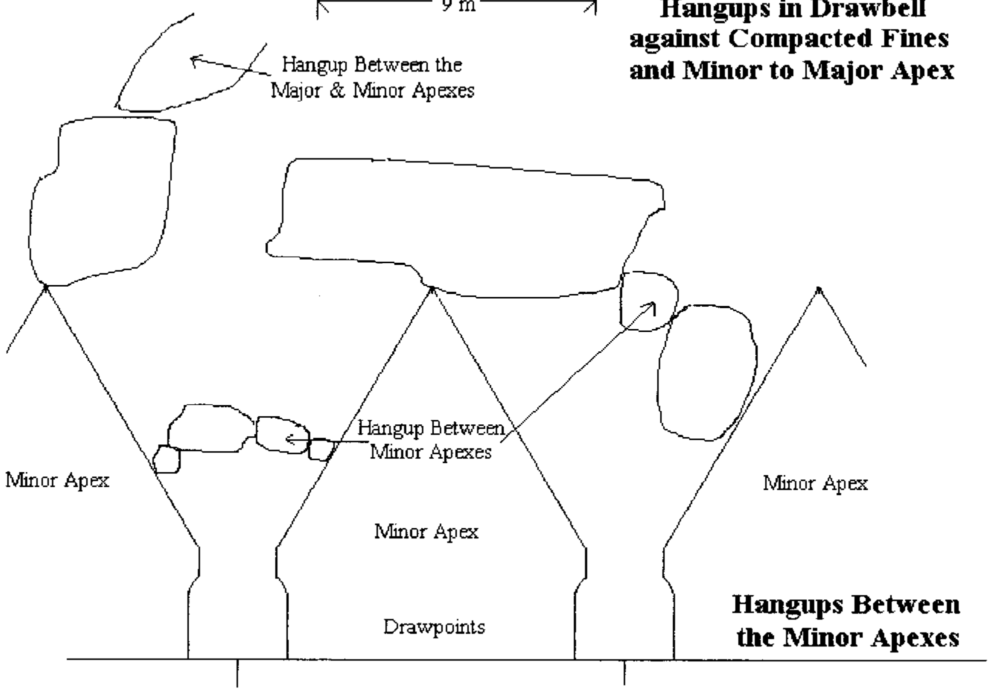
Owing to the nature of the digging action of an LHD, the draw zone (pattern) created in an individual drawpoint tends to be elliptical in the direction of digging. This more evident in the False Footwall layout at King where the hanging wall pressures accentuates it and the draw markers confirm this.

See Sketch over leaf – Draw Pattern.

In light of the above there could be an argument for increasing the drawbell length. However, this must be weighed against the interaction that occurs and the increase in effective draw area effected by that interaction. Longer drawbells will of course reduce the number of drawpoints available in a given area.

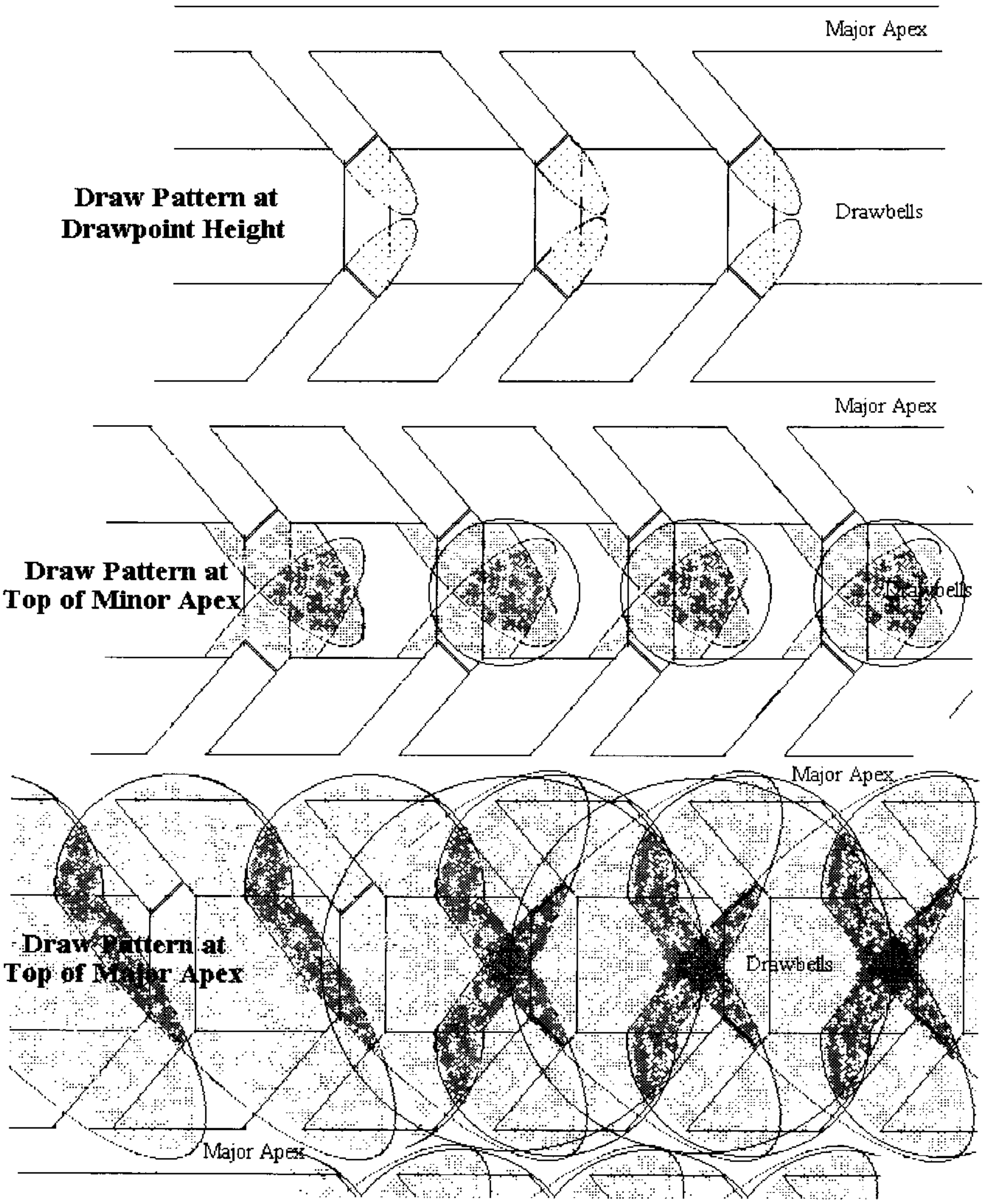


Hangups in Drawbell against Compacted Fines and Minor to Major Apex



Hangups Between the Minor Apexes

Hangups in Drawpoints



Over Lapping Draw Patterns as the Column Develops

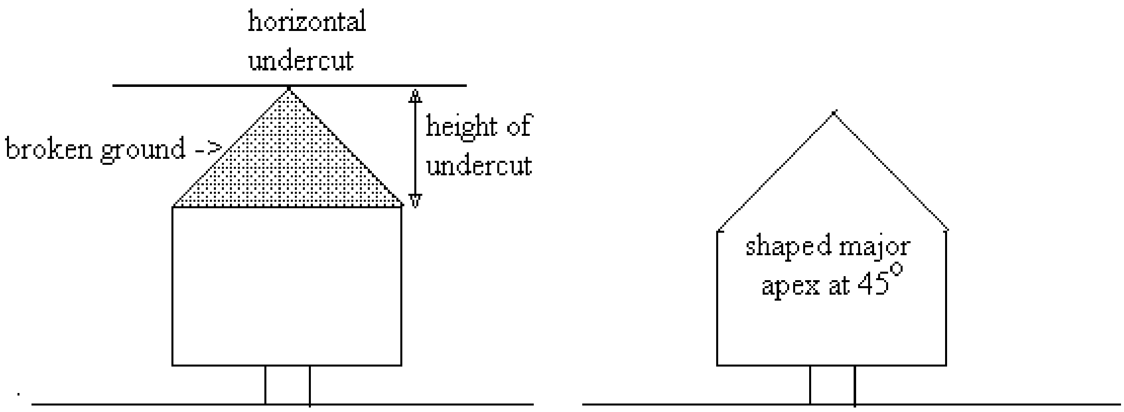
DESIGN TOPIC

Undercutting

GENERAL

Undercutting is one of the most important items in cave mining, as not only is a complete undercut necessary to induce a cave, but the design of the undercut and the sequencing of the undercutting also provides a facility to reduce the effects of the induced abutment stress. It is essential that the undercut is continuous and it should not be advanced if there is a possibility that pillars have been left. This rule, which is often ignored owing to the problems in re-drilling holes, results in pillars being left and the collapse of large areas and high ore losses. Operators have lost sight of assigning the right expenditure to undercutting - particularly advance undercutting - in relation to the tonnage made available and the low maintenance costs.. With a 200m draw height, every m² of undercut generates 560 tons, if the average operating cost were $3.00, then spending $56 per m² would only affects the cost by 10 cents. The obsession with pre-drilling needs to be re-thought in terms of assured holes and that the time required to undercut is not a critical item in bringing a block into production - the time consuming operation is creating the drawbell.

Abutment stress is a result of undercutting, but, because of the orientation of the drifts and the low percentage of development, abutment stress damage is seldom seen on the undercut level even though the production level is damaged. The magnitude of the abutment stress is a function of regional stress, direction of undercutting and undercutting technique. The undercut technique also determines the shape of the major apex and importantly the shape of the drawbell. Care must be taken that there is no stacking of large blocks on the major apex as this could prevent cave propagation. **A rule of thumb introduced years ago was that the top of the undercut should be at an angle of 45° from the edge of the major apex above the brow to the centre of the major apex:-**

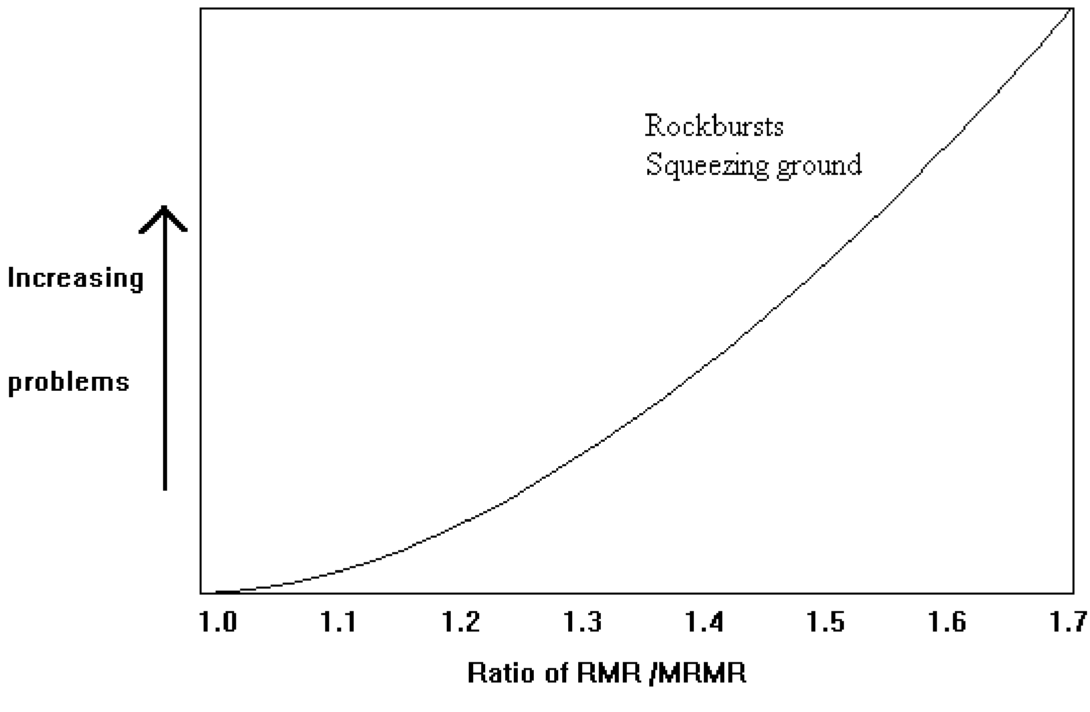


Height of undercut

PARAMETERS INFLUENCING THE UNDERCUTTING OPERATION

IRMR / MRMR

The ratio between IRMR and MRMR will indicate the changes that are anticipated for the rockmass and the need for caution in design.



Rockburst potential

The rockburst potential must be determined and will be the result of high regional stresses, abutment stresses, difference in moduli and the direction of advance. The difference in moduli or IRMR often leads to problems at contacts of the two rockmasses. The layout used and undercutting technique will be designed to reduce or eliminate the rockburst potential.

Geometry

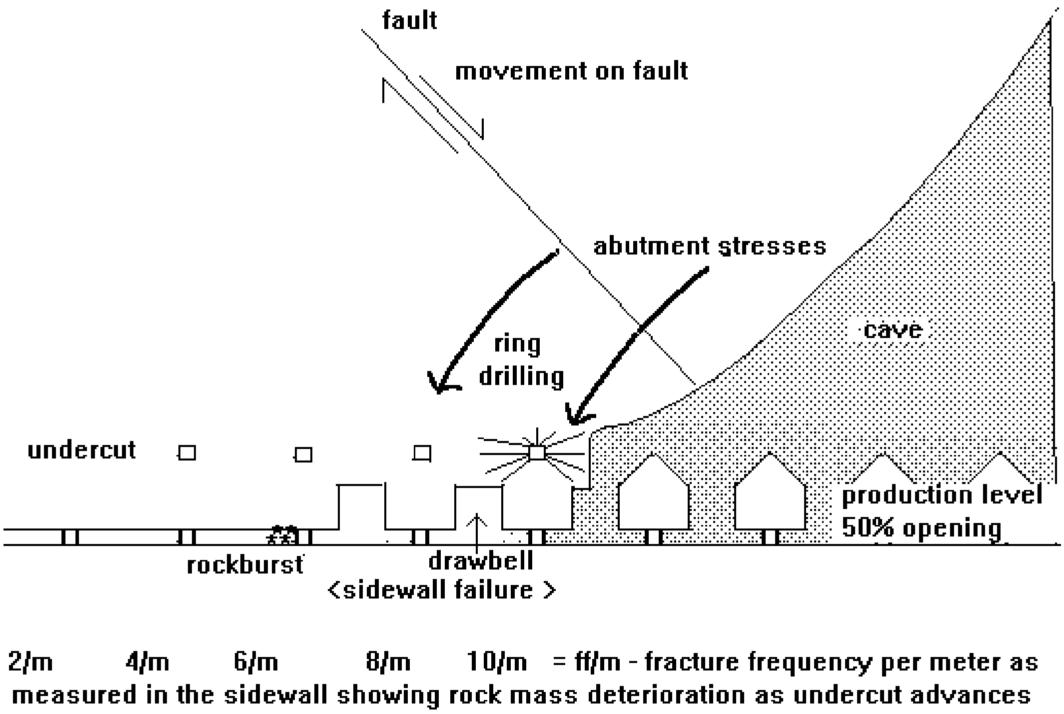
Simple layouts with the drifts at a large angle to the cave front are less likely to experience problems than those areas where there are irregular shapes and large leads between faces. Junctions with access drifts would have higher induced stresses and should be spaced as widely as possible. The accepted technique will be for the cave front to cross the access drift at an angle and to advance as rapidly as possible.

UNDERCUTTING TECHNIQUES

Conventional

The conventional undercutting sequence is to develop the drawbell and then to break the undercut into the drawbell. In theory, conventional undercutting should not present a problem as the undercut is broken into an excavated drawbell. However, there are numerous examples of undercuts freezing with pillars left followed by column loading and collapse of large areas. This is especially the case where ground conditions have deteriorated and /or drawpoint spacing has been increased and not enough attention has been paid to the blasting procedure. One of the major contributors to the problem is hole cut-off. As areas are pre-drilled there is a reluctance to bring back a drill so holes are charged as best possible and the blast is done in hope. There is no check until failure of the production drift occurs. On a major block caving mine a large area had collapsed owing to column loading caused by incomplete undercutting.

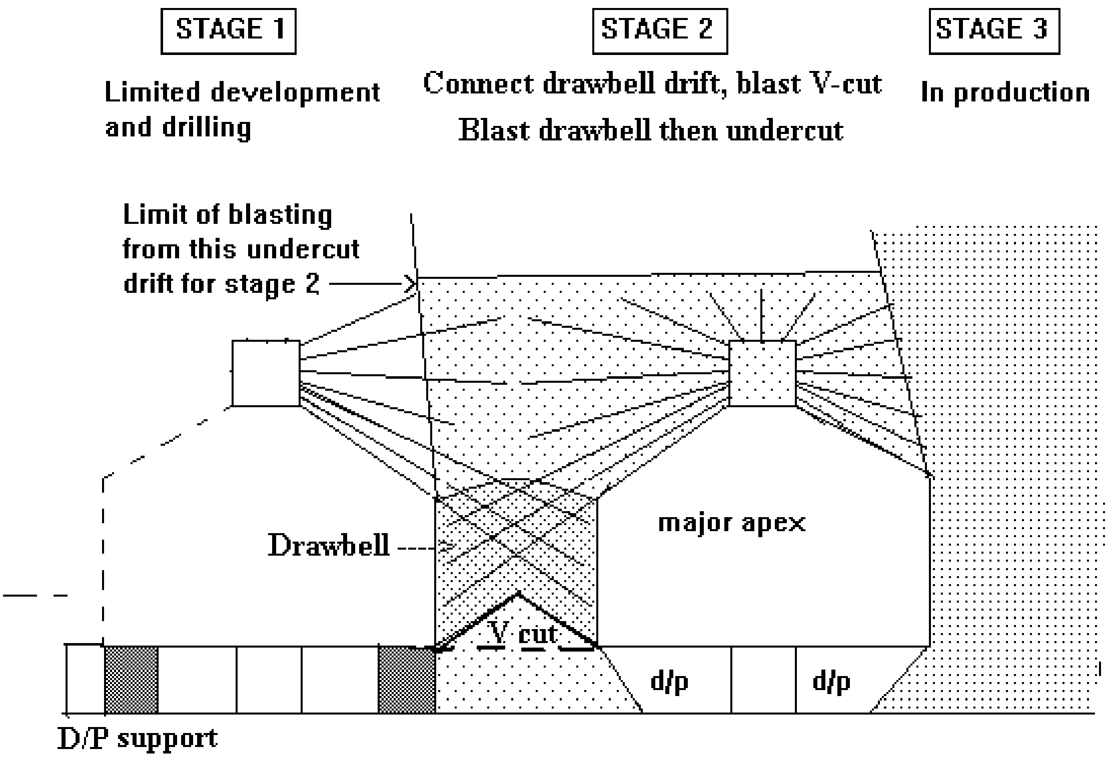
In high stress environments on the production level the pillars and brows are severely damaged by the abutment stress. In the following diagram of a typical situation on a major caving mine, the effect of the abutment stress on the rock mass is seen by the increase in the fractures per metre as the cave front advances towards a measuring site.



CONVENTIONAL UNDERCUTTING

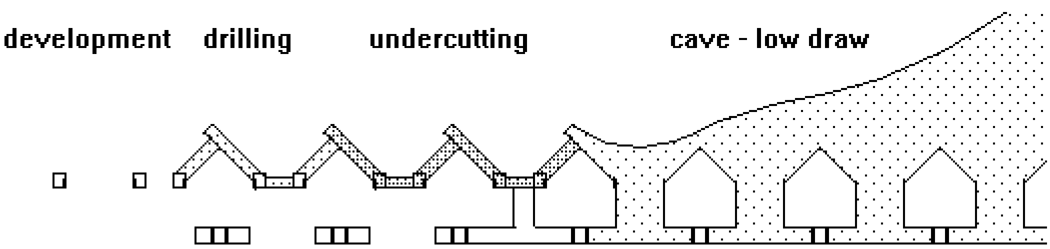
Henderson Technique

The Henderson Mine technique of blasting the drawbell with long holes from the undercut level just ahead of blasting the undercut reduces the time interval in which damage can occur. They have also found it necessary to delay the development of the drawbell drift until the drawbell has to be blasted so as to leave as much solid rock in place. This technique has proved successful at Henderson Mine where over two thousand drawpoints have been commissioned in this way. In fact it is surprising that no other mines have adopted this technique in preference to the conventional system. **The system does not work where there are squeezing ground conditions and hole closure occurs, particularly in the down holes.**



Advance Undercut

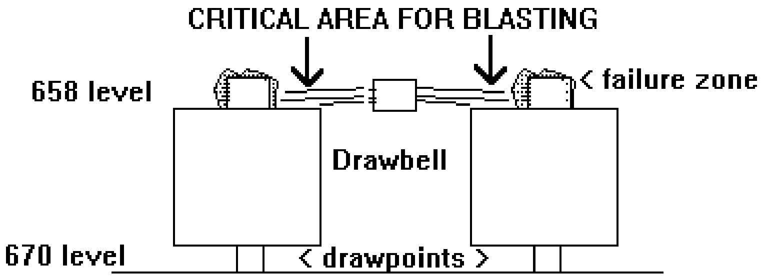
The advance undercut technique means that the drawpoints and drawbells are developed after the undercut has passed over, so that the abutment stresses are located in the massive rock mass with only the production drift or the production drift and drawpoint take-offs developed on the production level. Damage to the extraction level is avoided by developing the drawpoints and drawbells in destressed ground:-



Advance undercutting sequence

The advance undercutting techniques can vary from narrow `longwall' stopes to SLC operations. The narrow `longwall' stope with no or limited muck removal can be horizontal or inclined over the major apex , resulting in a ' saw tooth'. appearance.

In the advance undercut the principle is that the undercut development has sufficient space to absorb the volume increase from blasting the intervening ground. Operators tend to load a certain amount of muck out of the undercut because of concerns about choke blasting. It must be remembered that all SLC blasting is done under semi-choke conditions. The blasted muck acts as rock fill and therefore reduces the abutment stress.

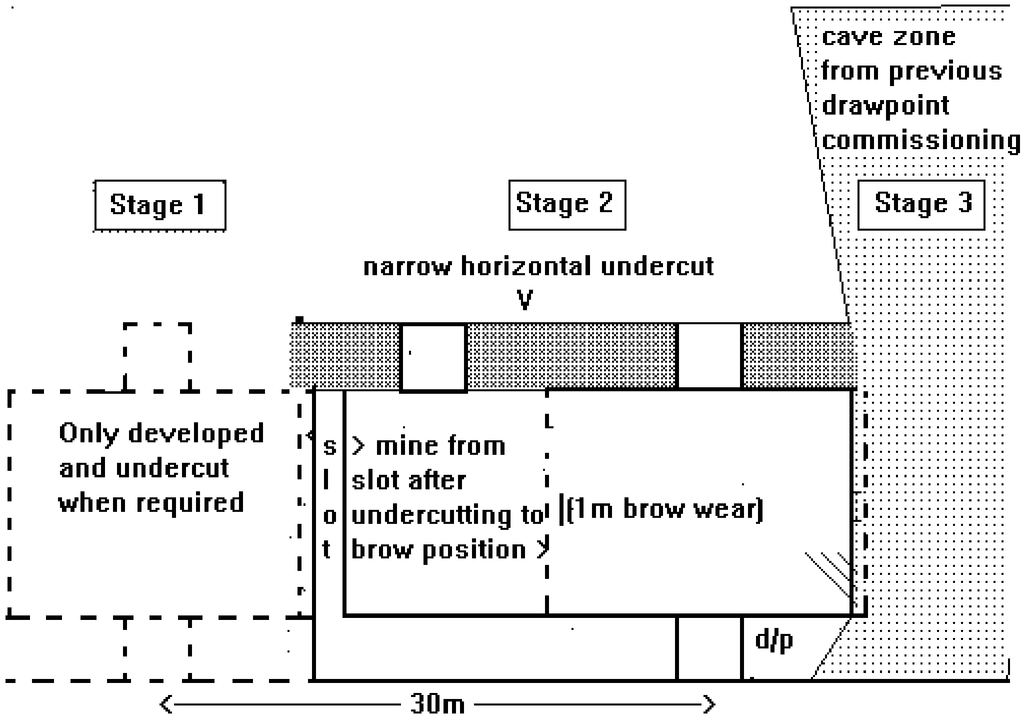


Because the advance of the undercut is done ring by ring it is possible to check that the undercut is through before taking the next blast. This is the procedure that was adopted in undercutting grizzly operations - the undercut would not be advanced until the operator was certain a connection had been made. In the large blasts of the conventional system it is assumed that the connection has been made - in many cases this is not the case.

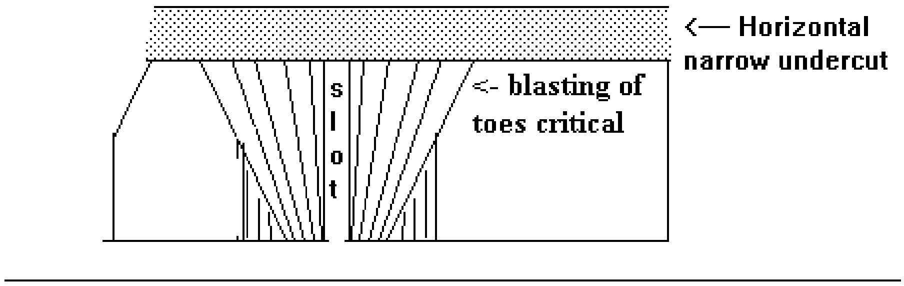
Horizontal Advance Undercut

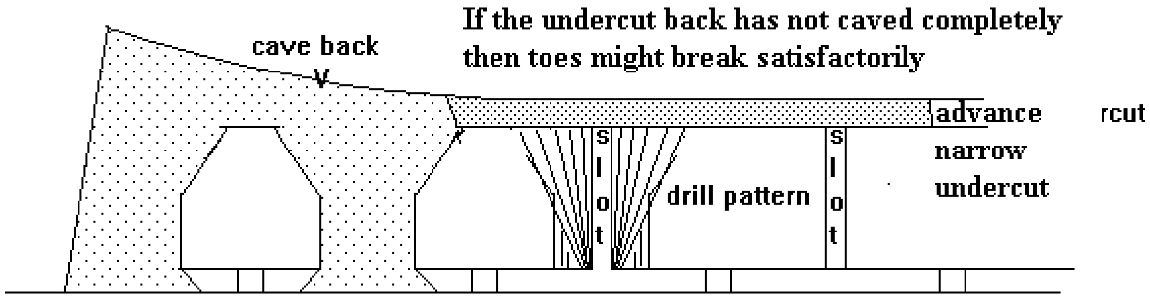
The following diagram shows a horizontal 3m high undercut face with the full ring blasted and the barrels evident in the wall rock. The person on the left is standing 2m from the start of the face in the undercut drift.

The following diagram shows the stages in the undercutting sequence for a narrow horizontal advance:-



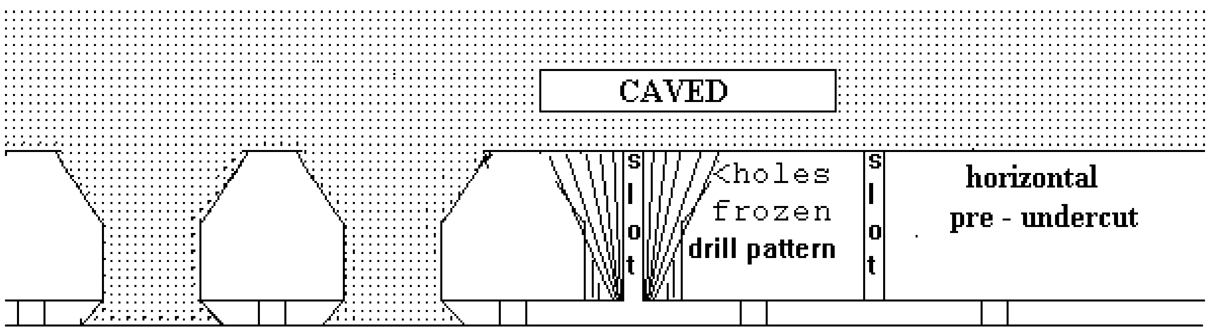
The result of this undercutting system is a flat top to the major apex and a high vertical brow. The flat top major apexes were successful at Shabanie Mine were the drawpoint spacing was 10m, angled structures meant rapid wear of the top of the major apex to an incline surface and the material though coarse had a low angle of friction. The flat top major apexes at Teniente are large and they have devised a technique to shape the major apex. The technique used at Teniente Mine to shape the major apex is to drill inclined rings in the drawbell back over the position of the brow, then to cut a vertical face at the brow. This might be successful if there is no load on the toes of the holes, which would be the case with an advance undercut with a limited lead or stable back. However, if this technique were used with a pre-undercut and caving had occurred, as at Esmeralda, then there would weight on the toes and these choke conditions could prevent effective breaking and an overhang could form





Shaping the major apex from the drawbell - advance undercut

In the case of the advance undercut the toes of the incline up holes might break satisfactorily as they would be blasted under semi-choke conditions as in above diagram. However, if this technique is used when the area has been pre-undercut and full caving has occurred then the toes would be under full choke conditions and that area might not break, as shown in following diagram.



Shaping major apex from drawbell - pre undercut

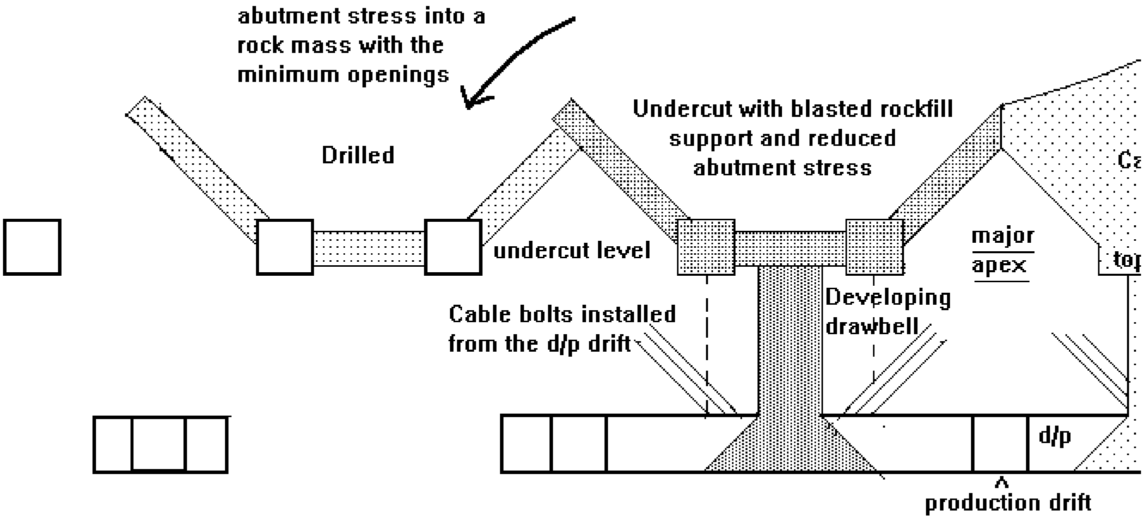
The shape of the major apex could be checked by drilling from the production drift through the major apex.

Disadvantages of the Horizontal Undercut.

- In the abutment horizontal holes are more likely to close than incline holes.
- If the drawpoint spacing is large 15m+ then the possibility of stacking and inhibiting caving is great.
- The poor shape to the drawbell will not encourage good ore flow.

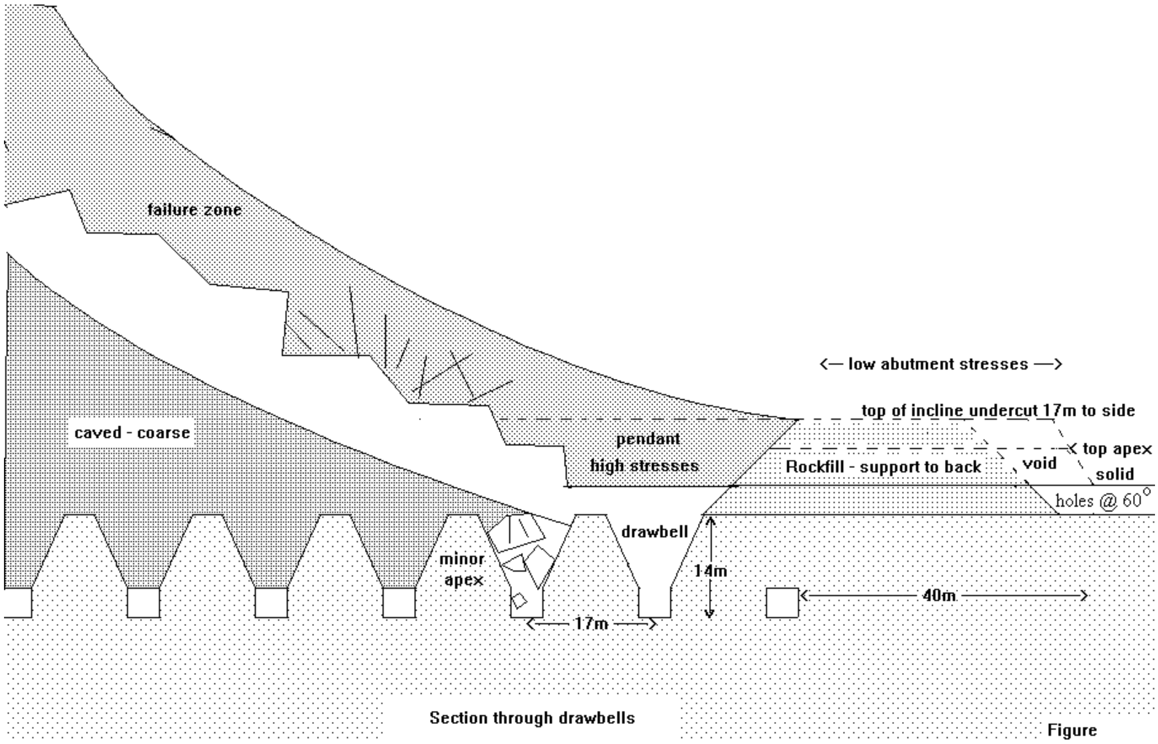
Incline Advance Undercut

The term incline undercut describes the attitude of the undercut surface after the undercut has been blasted. The objective is to shape the major apex by drilling incline up holes from the undercut drift so as to create the optimum drawbell. The sequence for an incline undercut is shown in the following diagram :-

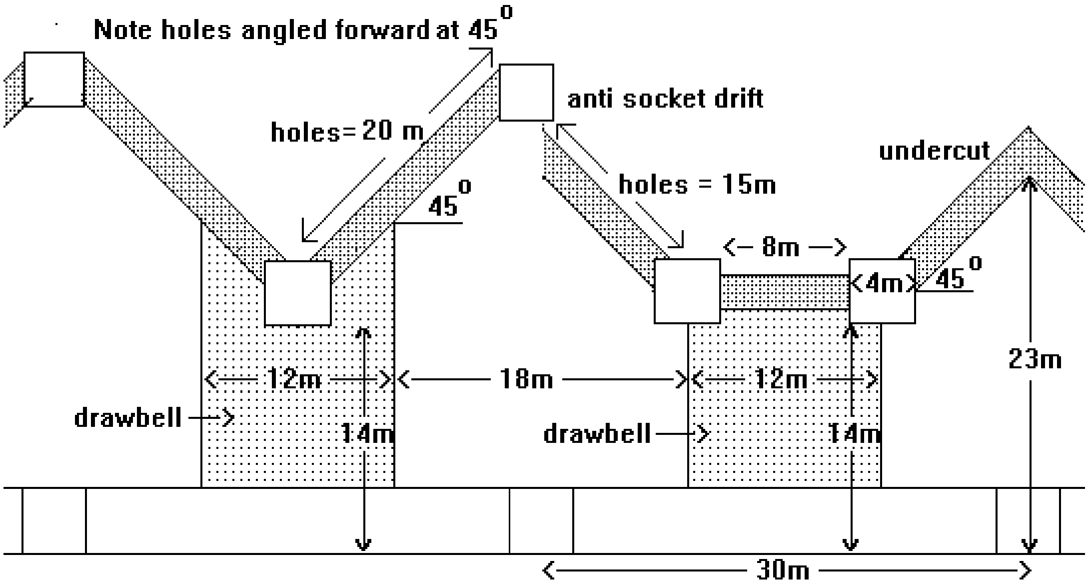


INCLINE NARROW ADVANCE UNDERCUT SEQUENCE WITH CABLES INSTALLED FROM DRAWBEL

The following diagram is a longitudinal section along the line of the production drift and shows how the back is in tension and caving cam start.

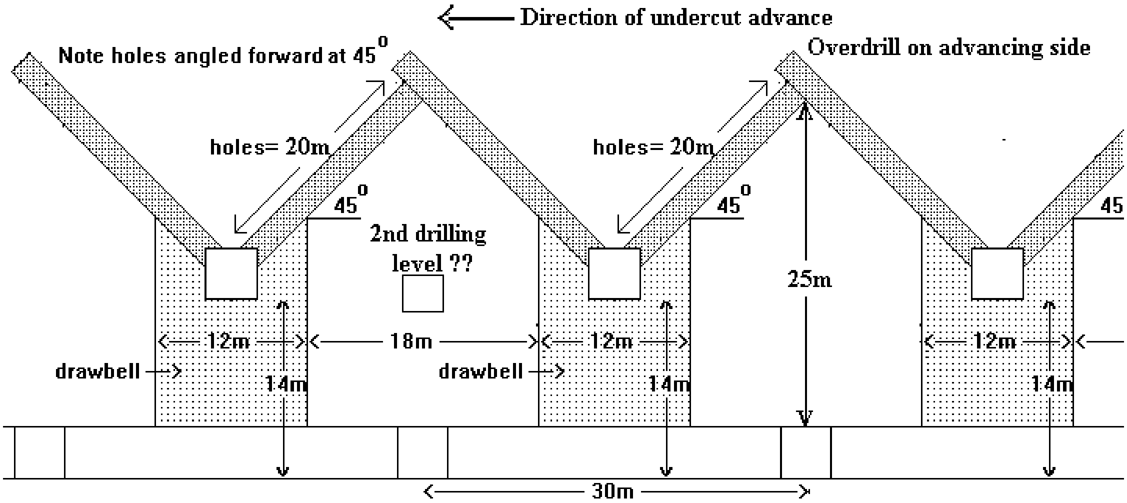


Undercut drifts -The following diagram shows two layouts one with the undercut drifts developed on two levels and the other on the same level. A drift is developed over the top of the major apex to provide a check on the accuracy of the drilling and also as an anti - socket drift. The long hole drilling is done from the drift over the drawbells and in the diagram 20m holes are envisaged, with modern drilling equipment this is no problem.



SINGLE UNDERCUT DRIFT VERSUS TWIN UNDERCUT DRIFT

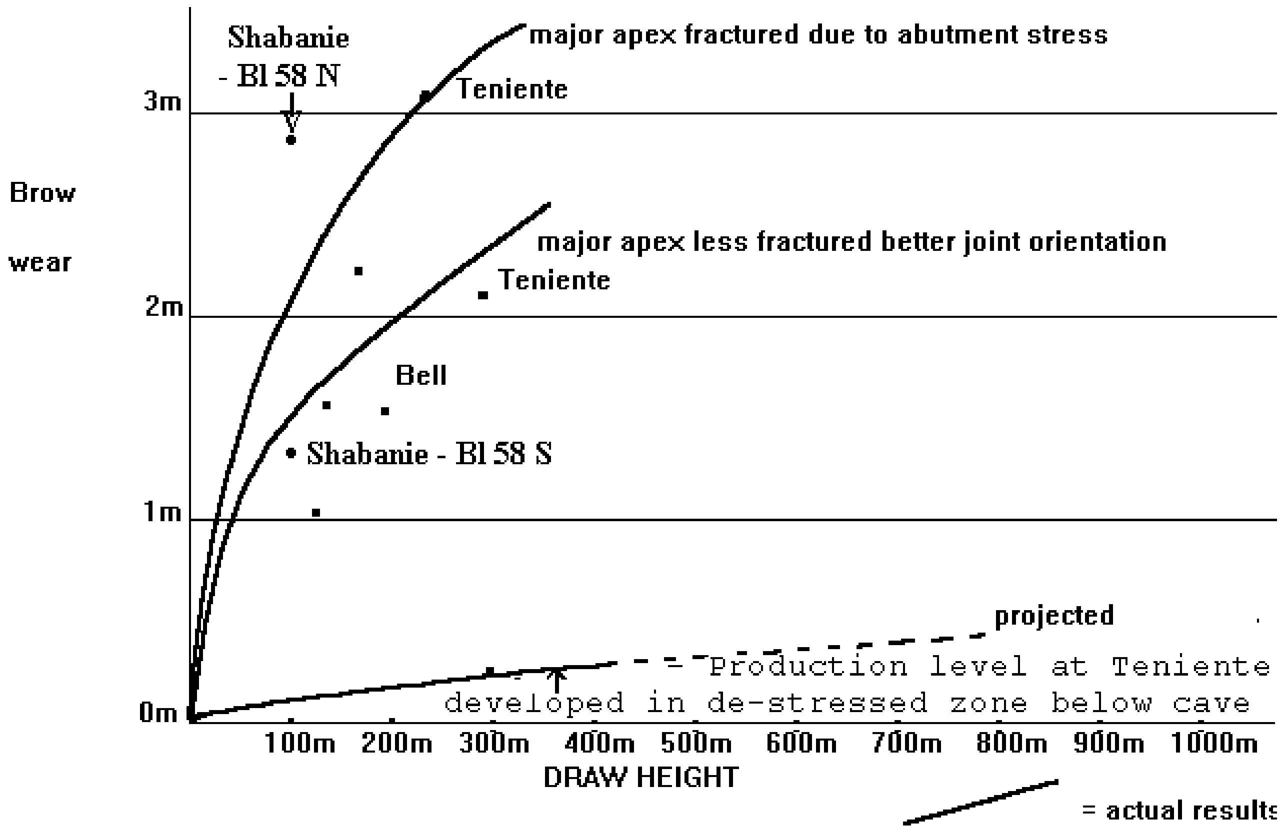
The advantage of the single drift layout is the assurance that the undercut has broken to the top of the major apex. Another advantage of the single undercut drilling drift is the reduction of development on a level, particularly at the access junction and this will minimise abutment stress damage. The major apex is higher, thus providing more space for a secondary drilling level. Certainly, this method does call for additional ramping, but is this a problem when draw heights of 400m are being considered? However, the holes are longer. In terms of cost per ton, it is negligible, however, if there is a capital cost problem, then the viability of the project is suspect. Finsch Mine are considering an incline undercut without the anti-socket drift. This means a reduction in undercut development, but does require accurate drilling and blasting or techniques to deal with the pillars that might form at the crest. The ‘King drilling rig could be used for this purpose.



SINGLE UNDERCUT DRIFT

Advantages of Advance Undercutting

One of the advantages of an advance undercut is the good condition of the rock in the brow and the significantly reduced brow wear as can be seen in the following diagram showing actual results of brow wear. The upper curves are from conventional undercutting operations. The lower curve is based on experience on a reclamation level located below a major collapse in Ten-4 with a resultant appreciable loss of ore. It was decided to develop the production level 15m below on the ventilation level. This meant that the development was done in a de-stressed environment. The result was that brow wear was negligible even though +300m of ore was drawn. It was this good practical example that finally convinced the Teniente management the merits of an advanced undercut. In this case it could be described as a pre-undercut.



RELATIONSHIP BETWEEN DRAW HEIGHT AND BROW WEAR

The narrow advance undercut technique is also favoured for the following reasons:-

- In a high stress environment the narrower the stope the lower the energy release as has been shown on high stress mines - examples available.

- It has been shown on South African gold mines that backfilling of stopes decreases the abutment stresses, thus by mining the narrow undercut under semi-choke conditions the undercut is effectively backfilled until the drawbells are commissioned.

- It has been shown at Teniente that the level of seismic activity is related to the extraction rate or the rate of propagation of the cave or the rate of increase in the size of the excavation. By blasting single rings, the undercut is advanced in a controlled fashion.

- The area under draw need not conform with the shape and orientation of the cave front. The cave front can cross a contact or be advanced against a contact at a certain angle, but the higher draw area can have a different shape to utilise stresses in the cave back.

- The connection between undercut drifts can be checked after each blast to ensure that the undercut is complete.

Undercut Planning

The timing of production level development and undercutting is dependant on the stresses and the rock mass response. The undercut is advanced from 30m to 40m ahead of the production level development, bearing in mind that there will always be a lead and lag between drifts. Planning must recognise the need for detailed scheduling and the acceptance that this process might take longer than with conventional undercutting. In order to speed up the development of the production level consideration could be given to increasing the level of development on the following basis:

- At up to 30% of the hydraulic radius the drawpoints and drawbell drifts could be developed and fully supported,
- Between 30% and 60% only the drawpoints are developed and fully supported,
- Beyond 60% only the production drift is developed.

This would only be done if the rockmass was competent enough, even in the initial 30% drawpoints in poor ground would not be developed.

Pre undercutting

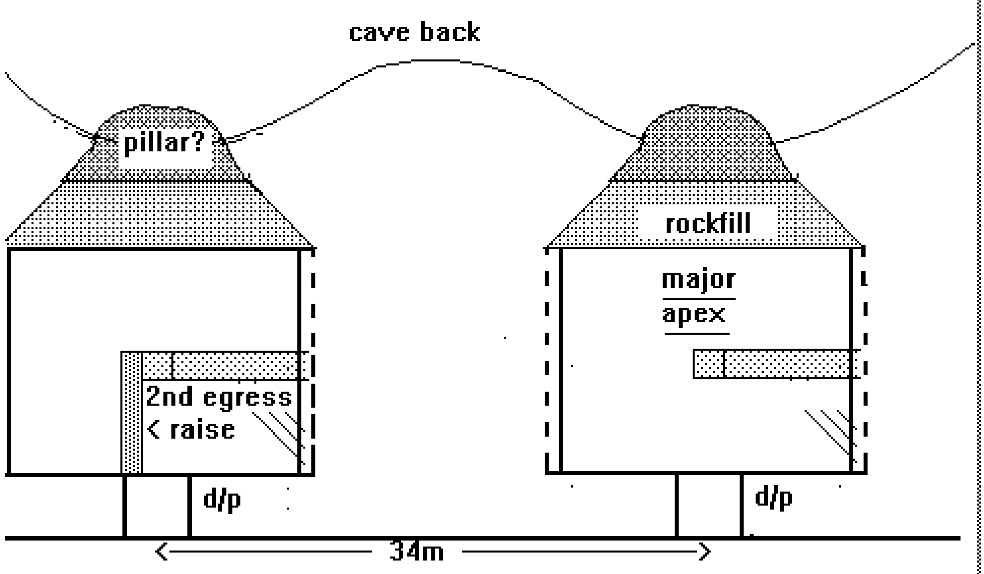
In theory a pre-undercut would have advantages. However, one of the prime controls available to a cave operator, if confronted with a weight problem, is to draw to relieve the weight. If the whole area is undercut and the drawpoints are developed thereafter, there is no opportunity to relieve weight. Also if a potential massive wedge is undercut it can `sit down' with minimum caving. If potential wedges are present, the undercut face should be angled to intersect wedge at such an angle as to allow the wedge to cave piecemeal and not en masse. With a pre-undercut this is not possible. Another problem with a pre-undercut is consolidation of the caved material before it can be drawn. There is also the large tie up of capital before the block comes into production.

HEIGHT OF UNDERCUT

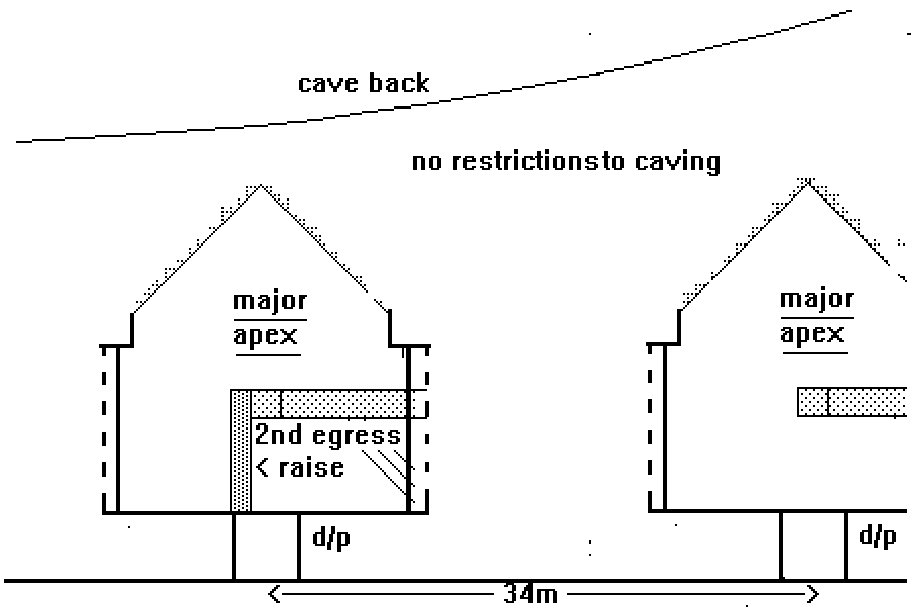
In the past it was considered that the height of the undercut had a significant influence on the caving and the flow of the ore. Asbestos mines in Zimbabwe had undercuts of 20m, the same caving or subsequent fragmentation results were achieved with 3m high undercuts. There is no reason why the undercut height should be more than one third the width of the major apex. Narrow undercuts of 4m can be used provided there is no impedance to the propagation of the cave and this will not occur if the major apex is shaped. At Henderson Mine the up-holes are seldom drilled. At El Teniente the upholes are not blasted after the undercut has advanced 20m from the slot. The undercut holes from the undercut drift shape the major apex so that the edge of the drawbell is at the drift.

PROFILE OF UNDERCUT / SHAPE OF MAJOR APEX

Where the fragmentation is coarse, and there is a large major apex with a horizontal top, caving could be impeded by formation of rock pillars on the major apex caused by the stacking of large rock blocks, this can be overcome if the undercut is inclined to form a shaped major apex. Factors which must be considered are the frictional properties and the fragmentation of the ore. Low friction material will move at low angles, whereas, high friction material and coarse material will stack.



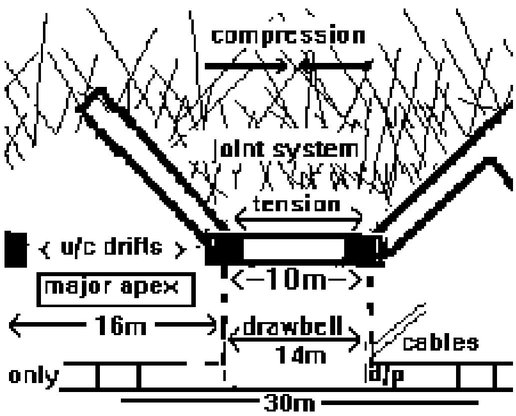
Stacking above a flat top major apex



Incline undercut with shaped major apex

The incline layout does mean a better shape to the major apex and would be a better operational method as it is self cleaning during mining. It is also easier for large rock blocks and high friction material to move on the incline thus avoiding any support to the cave back. In the long term, during the drawdown of the orebody, an inclined shape to the major apex leads to a better ore recovery.

The following diagram shows the stresses in the immediate cave back and the advantages of the incline shape in creating an immediate tension zone in the pendant.



CLOSURE OF HOLES / CUT-OFFS

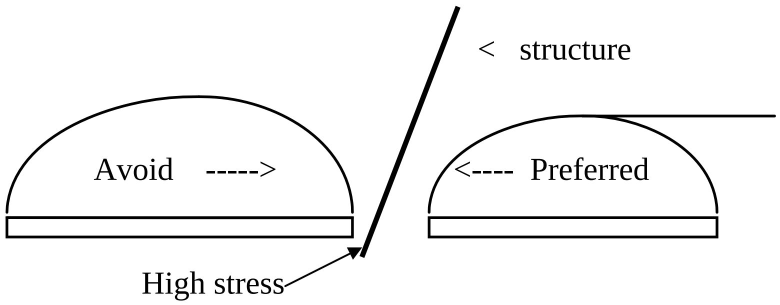
Hole closure has been a problem in some undercuts when the abutment stresses have caused the horizontal holes to close, particularly in squeezing ground. Hole closure is also experienced when there is a pre-break and drilling is done in a relaxing rock mass with holes being cut off by movement along joints. This situation happens when mining is done in the incorrect environment or the incorrect environment has been created by poor or optimistic mining. In squeezing ground, horizontal holes are more prone to closure than inclined holes. So, if, an advance undercut is used to create a more stable environment, then experience with a totally different undercutting procedure should not be used to condemn the use of this concept. However, if hole closure is going to occur, horizontal holes will be more affected.

There has been a concern expressed by some people that holes might be lost if a fracture zone were to develop in the abutment. There is no evidence that this would happen, in fact because the development makes a large angle to the cave front and the percentage of development is low, abutment stress damage is seldom seen on conventional undercut levels. However, in the remote possibility that this might occur, the problem can be overcome by drilling for each blast. A new system must be designed from the beginning and not stubbornly apply techniques that might have worked in benign environments. Drilling costs for the few holes required for a narrow undercut are low therefore drilling each ring on it’s own is not be unreasonable. Especially if that drilling is related to the tonnage drawn, but, more importantly it is the overall cost savings achieved as a result of less maintenance and ease of production owing to the stronger production level.

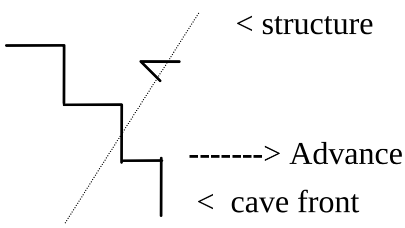
At King Mine the first 2m of the hole is blocked and removed when blasting is to be done A plastic liner in the collar of the hole would keep the hole open.

ORIENTATION AND DIRECTION OF THE FACE

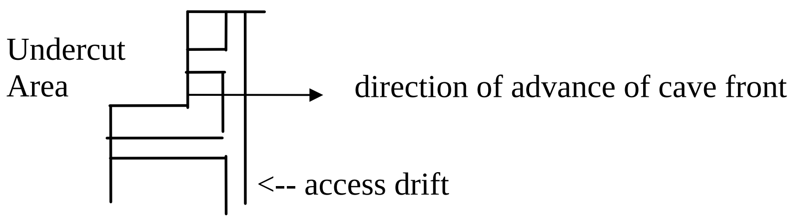
Massive wedge failures can result in major problems during the undercutting stage and can result in excessive weight causing collapse of major apexes and loss of large tonnages. The undercut must advance towards major structures from the correct direction:-



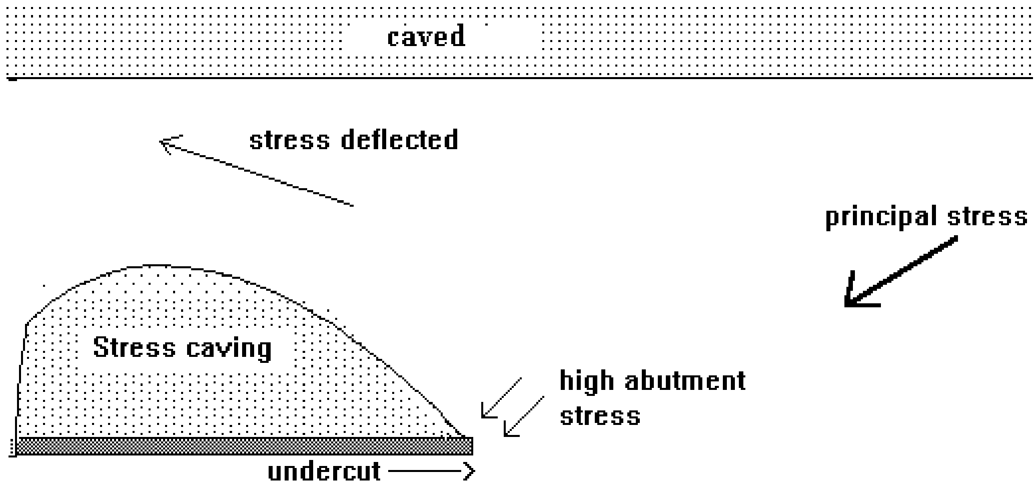
If possible, the cave front should not be advanced towards structures that could initiate massive wedge failures. However, if the cave front has to advance towards major structures these should be crossed at as large an angle as possible:



The cave front must be at a fairly large angle to the access drift as it moves across it to minimise abutment stress effects:

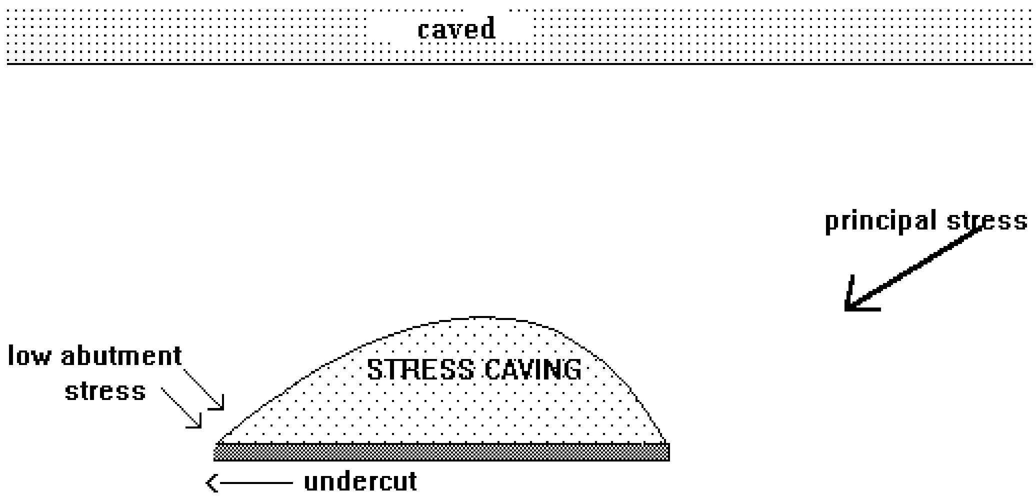


Advancing an undercut towards the principle stress should be avoided as this will result in high abutment stresses unless these stresses are necessary to induce caving and to improve the primary fragmentation:



Undercut towards principal stress = high abutment stress and good fragmentation

If the undercut is towards the principal stress then higher abutment stresses can be expected than if the undercut direction were away from the principal stress :

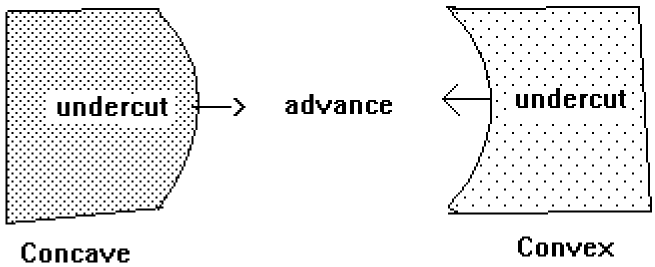


Undercut away from principal stress=low abutment stress and poorer fragmentation

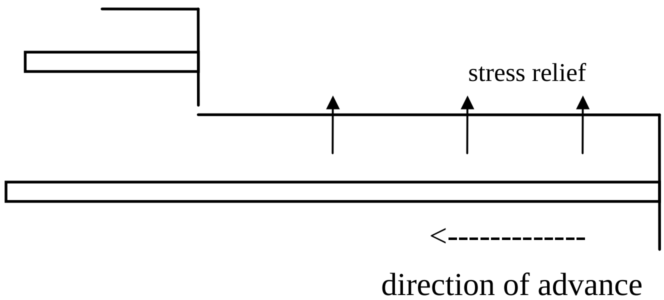
An undercut should not be advanced towards existing infrastructure such as haulage's or crusher sites.

SHAPE OF UNDERCUT FACE

A concave cave front will have the strongest shape owing to the confinement of structures and a convex face will be the weakest shape as it is in a relaxed state.



The lead between faces should be kept to a minimum as field evidence shows that a long lead results in relaxation and failure in the exposed drift, this also applies to SLC.



EXTENT OF UNDERCUT AROUND THE PERIMETER

It is generally accepted that junctions or large excavations close to the active cave must be overcut by extending the undercut. Concern has been expressed about the influence of toe stresses on peripheral development. Field evidence indicated that toe stresses are not a major factor.

RATE OF ADVANCE

The rate of advance of a conventional undercut is directly related to the availability of drawbells into which the undercut is broken. In the case of an advance undercut the distance ahead of the drawbell commissioning is set and therefore the rate of advance of the undercut is set by the drawbell commissioning rate. In high stress environments a rapid advance of an undercut can lead to seismic activity and rockbursts - the rock mass needs to adjust in it’s time not some scheduling figure. The undercut must advance at a uniform rate with only one ring blasted at a time. The same principles as apply for low draw for cave propagation apply for advance of the undercut. It is also essential that maximum quantity of rock fill is left in the undercut.

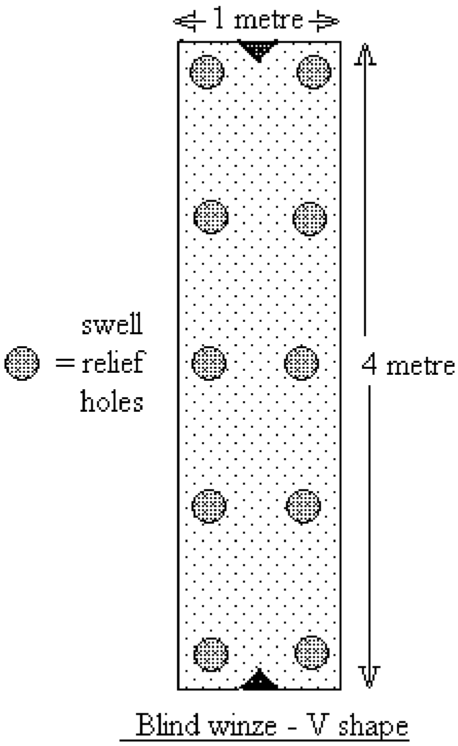
SUPPORT REQUIREMENTS

In the case of the conventional undercutting system the support on the undercut level is of a temporary nature and generally the support level is kept low, however access drifts and intersections need to be more heavily supported and the cave front must be at a fairly large angle to the access drift as it moves across it. Even in high stress areas there was little damage on the undercut level, the reasons for this were assigned to the orientation and limited development, this is partly true, but a major factor is that the abutment stresses were absorbed by the failure of the pillars on the production level.

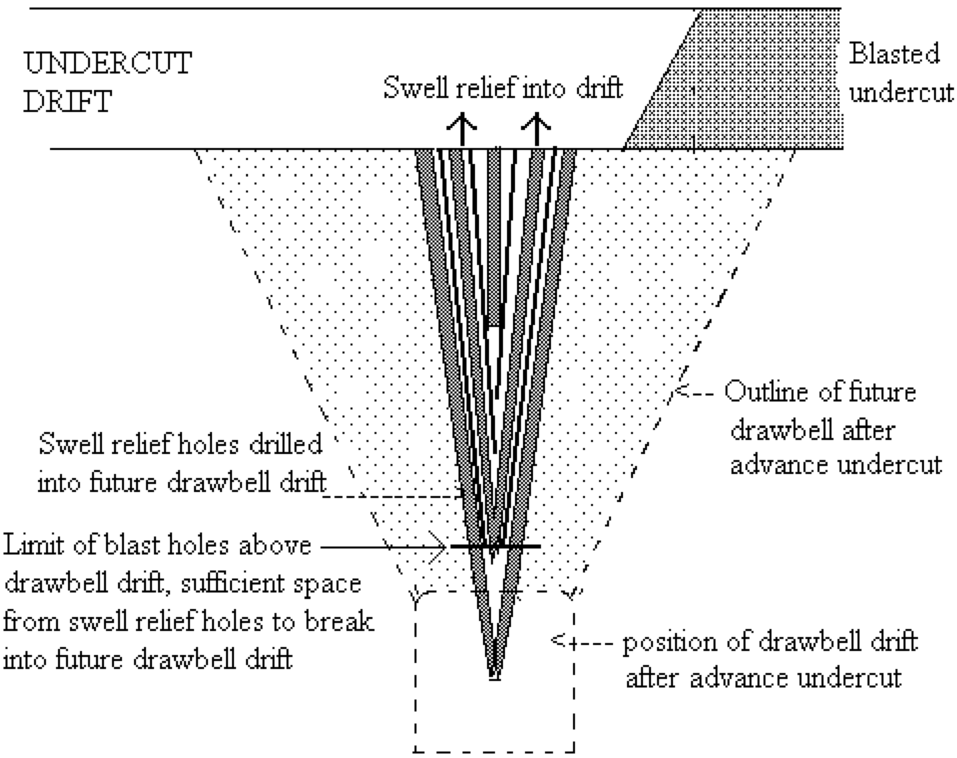
In the case of advanced undercutting there is an increase in the number of undercut drifts whereas on the production drifts the development is limited to the widely spaced production drifts with maybe drawpoints takeoffs. This means that the abutment stresses will not be dissipated by failure of pillars on the production level and therefore greater stress effects must be expected on the undercut level.. The support level must be increased in high stress areas with more extensive rock reinforcement consisting of yielding bolts (cone bolts) and the rock fill from the blasting must be left in place.

CREATING THE SLOT RAISE FOR THE DRAWBELL

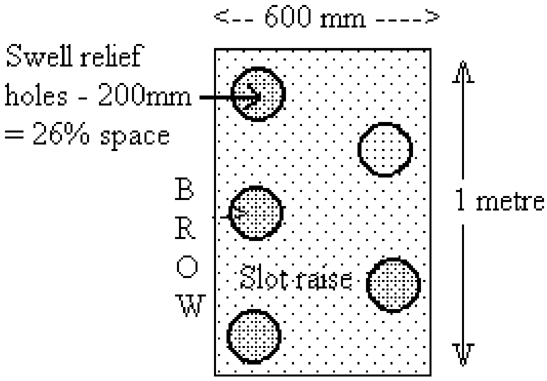
In an advance undercut situation the drawbell drift is developed after the undercut has passed and then a slot raise has to be put in. This often presents a problem and is time consuming. Blind hole raise-borers are used, but, are expensive and require the services of a contractor if the Mine does not have it's own machine. The following techniques appear to have possibilities and should be examined.



Plan above and section below of a V shaped blind raise drilled and blasted from the undercut drift in close proximity to the undercut blasting. The actual timing has to be sorted out, but it could be concurrent with the adjacent undercut blast, if some blasted rock has been loaded.



A possible problem is that owing to the V shape the broken rock will arch and not flow into the drawbell drift when it intersects the base of the winze. There could be sufficient relief for the adjacent ring holes to be blasted. However, if this is a permanent problem then a vertical winze might be required. Also the blasting could be done from the drawbell drift when it is developed. In weak rock or high stress areas it would be best to blast as a blind winze.



Blind winze - vertical

UNDERCUT BLASTING AT CASSIAR MINE - J Jakubec

Geometry

Undercut was 20m long at 10m spacing. Undercut excavation profile was 3.2 m high and 3m wide supported by 50mm of shotcrete with 1.5m long split-set in the back (shoulder to shoulder).

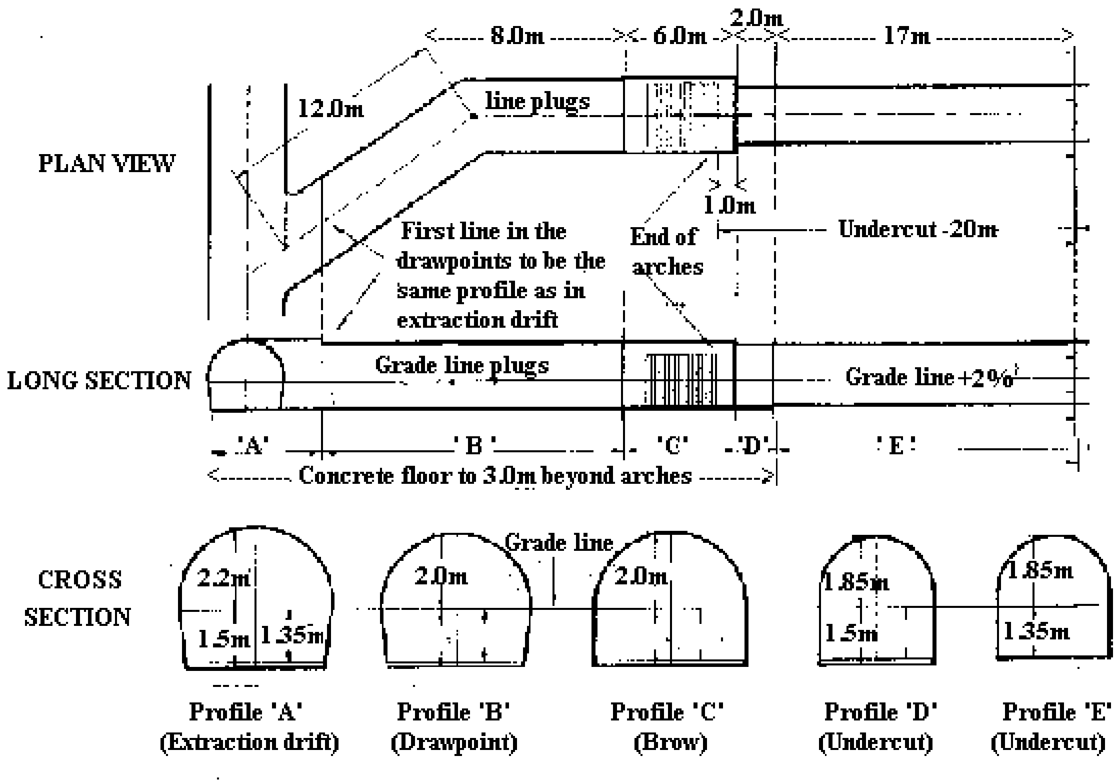


Figure 1 Geometry and excavation profiles of the drawpoints and undercut

Rock Mass

Mostly poor quality serpentinite (Class 4A-4B) is intersected by numerous shear zones of very poor quality – Class 4A to 5A (see Figure 2). Although the ground conditions were poor very little over-break was experienced during the development.

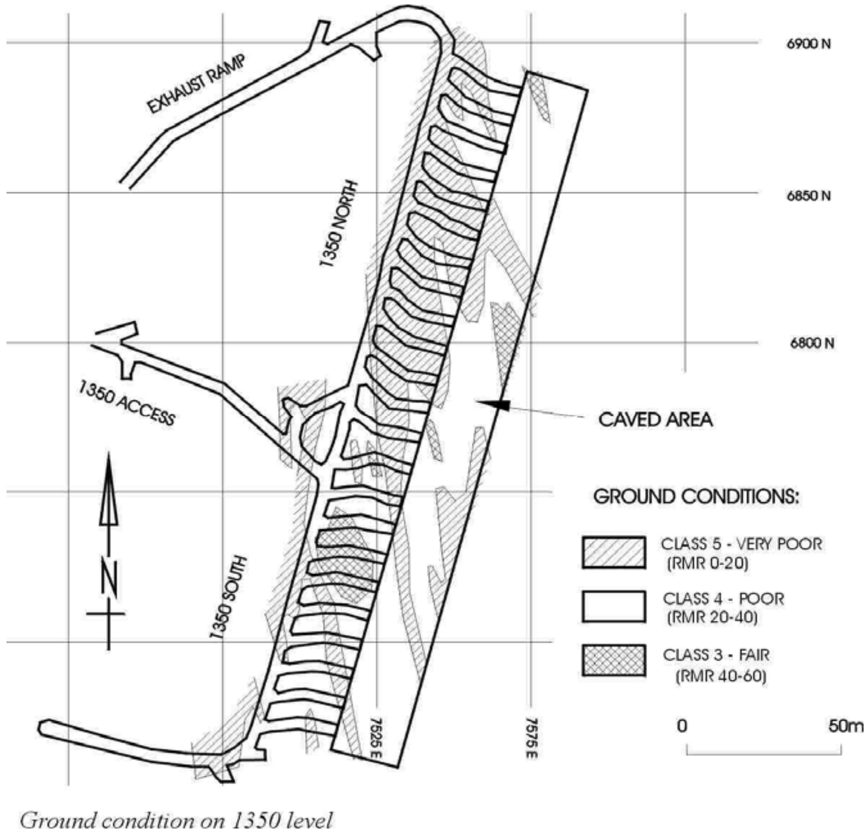


Figure 2 Ground conditions at 1350 level

DRILLING

Undercut blast holes were 50mm in diameter and the main undercut rings were drilled to a maximum height of 13m. In total 8 holes were drilled in the fan with 1m burden and 1.5m spacing. Although generally squeezing conditions were experienced there was no major problem in terms of keeping drillholes open. There were cases of drillholes being pre-loaded two weeks prior to successful blasting

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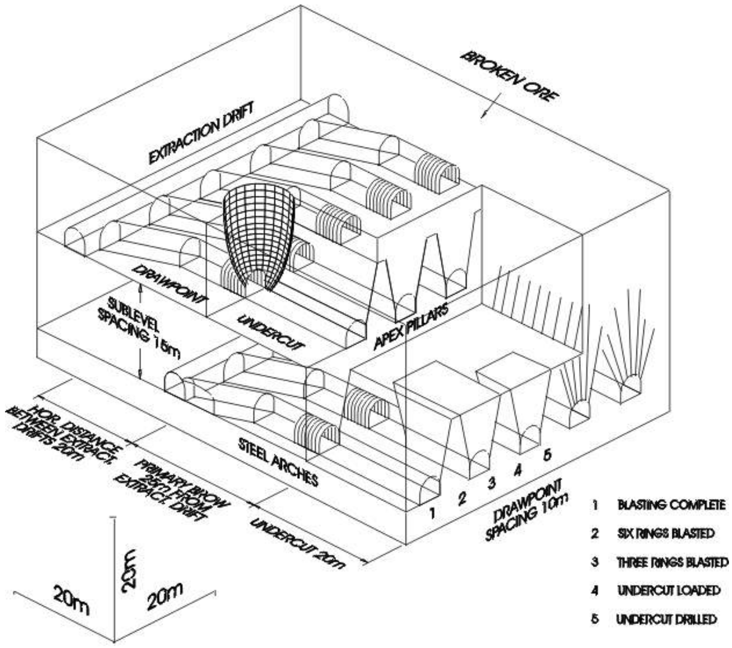


Figure 3 Front cave layout. With undercut drill rings geometry

Blasting

Holes were pneumatically loaded with ANFO explosives over the leg wires of electric caps (needed special permission from mining inspectorate). Pre-loading SG was 0.84 g/cm^3 and this increased to 0.95-1.05 loaded. Toe priming was initially exercised to ensure complete breaking and to minimise cut-offs. Because no major problem was experienced with blasting, collar priming was introduced later. Instantaneous blasting caps were used for every hole and the delays required between rings were programmed on CIL sequential blasting machine. Typically 2-3 rings were blasted at one time. Pre-loading and pre-priming was required because movement in the weak rock mass was anticipated. Because of the problem of contamination with plastic, NONEL blasting techniques could not be introduced.

Hand-drilled slash techniques were used to establish a breaking slot because of the difficulty of raising in the highly sheared serpentinite. 2.7 m^3 LHD units were used to pull out the swell from each blast until the drawpoint brow was reached. Lag of 3-4 rings were kept between the adjacent undercut tunnels to ensure smooth propagation of the caving and minimise induced stress damage (see Figure 5).

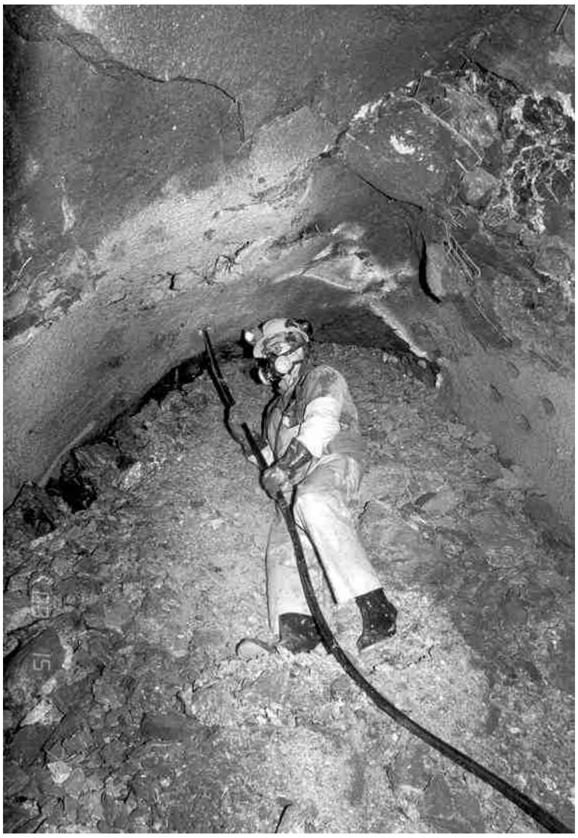


Figure 4 Pneumatic loading of ANFO explosives on the left and charged and primed holes ready for blasting on the right

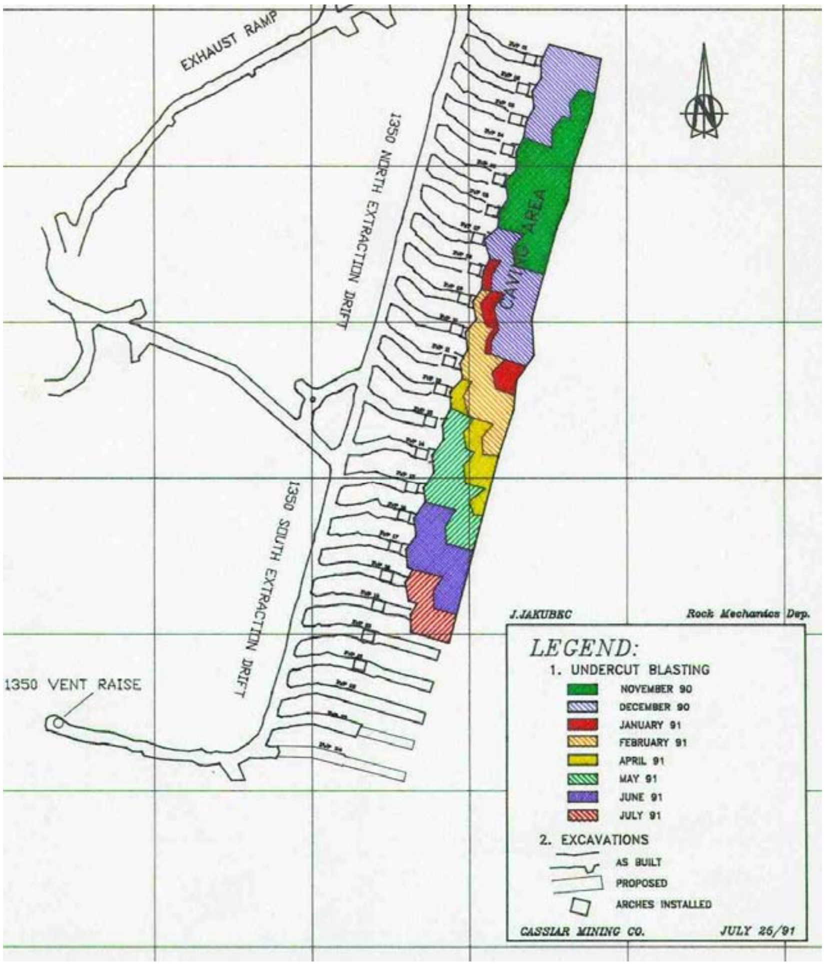


Figure 5 Undercutting sequence at 1350 level.

CONCLUSIONS

- No major problem was experienced during the undercutting of 1350 level
- Hand drill slash technique was not ideal but raising in highly sheared serpentinite was not feasible.
- Although undercutting was done in very poor ground conditions, some initial fragmentation was coarse as a result of sliding and gravity fall of competent serpentinite blocks from above the undercut.
- ANFO as “high gas” type explosive proved to be a good choice for such ground conditions.
- Good ventilation had to be in place for pneumatic loading – some initial problems with ammonia fumes were experienced.

UNDERCUTTING COMMENTS BY N.J.W.BELL

13 Undercutting

With out a doubt this is the most important activity with long term consequences and is an area that is often sadly neglected until too late.

General Description of Techniques and Potential Problems

The effective undercutting of a production block is critical to production, efficiency and ensuring that the planned production tonne are extracted as scheduled with minimum dilution. Failure to undercut a block properly results in the following:

1. Poor caving
2. Extreme mining stresses which can result in damage to or failure of the support systems on the production horizon.
3. Loss of production.
4. Early dilution entry and poor ore recovery.
5. Inability to meet production schedules.

Therefore, for all undercutting operations, no matter what cave mining method is employed, the laid out standards and procedures must be adhered to. The following are suggestions that need to be considered:

1. Undercuts shall be sufficiently supported to ensure that mining proceeds on schedule, smoothly and without interruption.

2. Suitable number and type of props shall be stored on the undercut level to deal with any such situation that demands temporary support urgently.

3. The area shall be fully serviced, the development services intact.

4. The area shall be cleared of all extraneous material and equipment.

5. To allow for the efficient lashing, if required (it might not be part of the planing but if required will have to be done urgently), of the daily blasts the tracks, roadways and tips shall be in good order, left fully operational from the development phase.

6. A standard should be set up as to the continuity of blasting of the under cut over public holidays/weekends etc. (Regular blasting reduces the stress build up and reduces the chance of 'seismic' events.)

7. The area that needs to be under cut to meet the ongoing production requirements should to be established together with the face shapes and monthly aim lines which should be marked on the master plans for guidance.

8. All areas being undercut shall have two sets of overlay plans properly superimposed and kept to within 24 hours up to date by the blasting supervisor of the area.

   One set underground the other on surface. They shall be available for inspection at all times. Particular attention being paid to sequencing to prevent any premature undercutting of the levels above when commissioning the lower level cones, overlapping undercuts or "A" undercutting of a False Footwall.

9. Areas being undercut shall have a suitable drilling rig/s (King rig, Jackhammers Etc.) with all equipment available on the undercut level to assist in any drilling required to ensure continuity of the undercut areas when required.

10. Original production drilling shall be to a high standard.

11. Charging of the undercut shall be under the control of the blasting supervisor who shall ensure that continuity is maintained and that the blasting advance is as planned. Each and every hole drilled for undercut breaking shall be checked for length and angle before blasting. No hole shall be charged and blasted that is not open to the correct depth unless the controller is satisfied that the blasting of the particular hole will not affect the break of the whole pattern.

12. All areas of doubtful breaks to be recorded and communicated with conformation when cleared.

13. The blasted rock shall normally only be lashed;
    To assertion continuity to adjacent crosscut/s or cones and that the last blast was to height and depth or
    To clear the way for additional development, drilling, blasting or recharging off partly broken rings to correct the situation if the above is not achieved.

14. Careful, full and continuous monitoring by all operations staff of the undercut operation is critical and any adverse or unusual conditions arising shall be reported immediately in order that corrective action can be taken immediately.

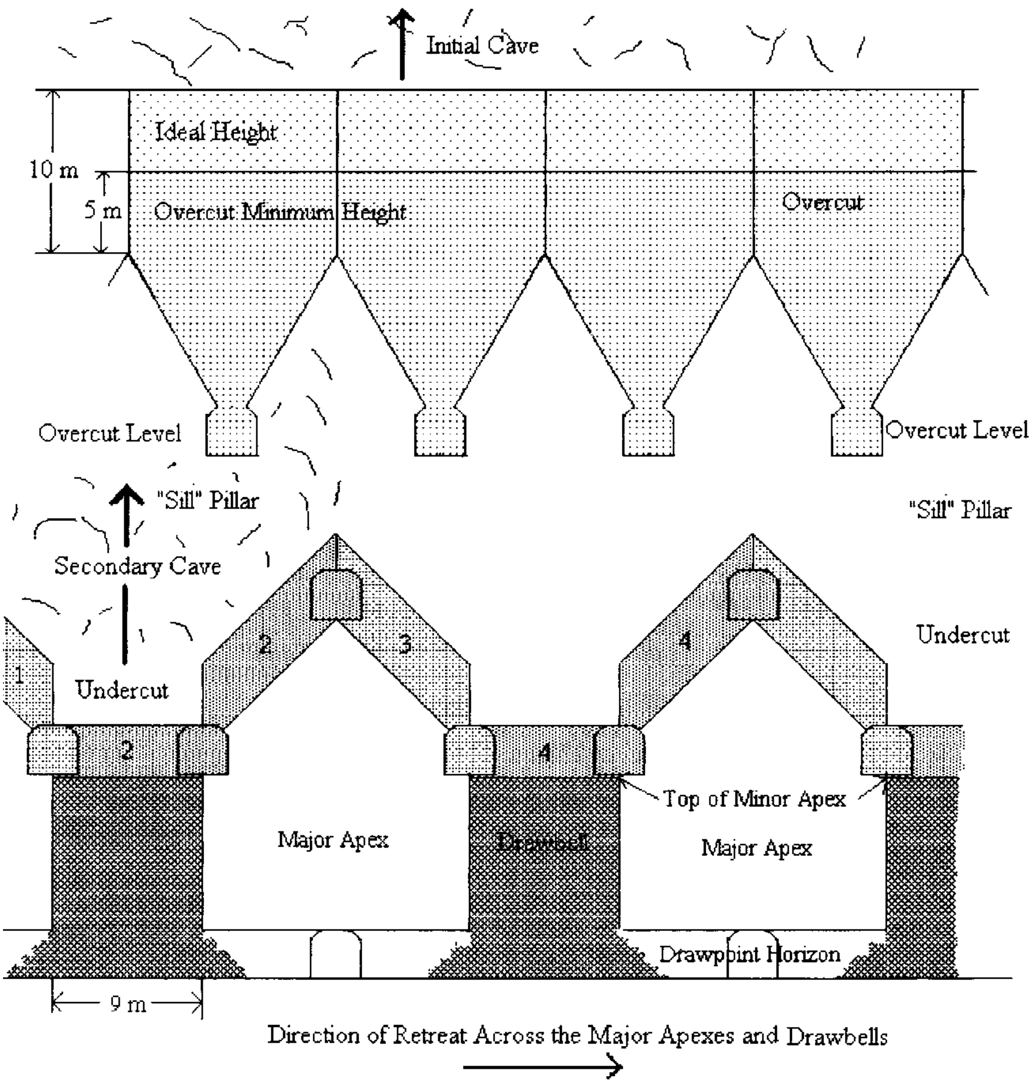
15. Should an extremely large span be opened up with no visible sign of caving, ring drilling drives or crosscuts shall be kept full of spoil that in the event of a sudden cave, personnel are not injured by an air blast. To do this a ring is blasted and not lashed at all if the end has reached stopping position no more rings left then an additional rings should be drilled and blasted, (NOT lashed) or the 'King Rig' used to blast additional spoil off the back of the undercut area. Alternatively suitable plugs, properly hitched, reinforced and robustly constructed, can be installed.

Pre-Undercutting

There are several instances where pre-undercutting has taken place and yes there can be a problem of consolidation and possible re-stressing. But if caving has taken place, it generally appears that the subsequent cave is easier. The consolidation problem might well be more applicable to fibrous ore, which can become particularly hard and resilient to subsequent breaking. At Shabanie however in our sub level cave operations where we have long time gaps between subsequent levels, no difficulties of consolidated ground on the levels above have been experienced.

Possible the answer lies in an undercut (overcut) at a higher elevation, taken either like a sub level cave or a narrow undercut across the ore body. Thereby creating a cave with sufficient room to swell relief when stress is building. Some early swell relief production is obtained. Then in the de-stressed/relaxed ground mine the drawpoints, support and construct them. In addition mine and drill the drawbells including the slot raises as required. Prepare the second undercut linking to under a small sill pillar, which will cave and collapse. In falling into the drawpoints will break up any consolidated material on its way to the drawpoints.

The drawing over illustrates this idea.



Pre-Overcut with "Sill" Pillar to Undercut and Drawbells

Preconditioning Ahead of the Undercut

What about the use of preconditioning blasting during undercutting?

It would be very appropriate in high stress areas, as has been found by the South Africans in their deep level gold, where they are after all undercutting continuously. The technique to blast a single line of holes, in conjunction with the advancing face so that a line of cracks is set up to disrupt the stresses that are going to get transmitted into the face area.

"The mechanism, optimization and effects of preconditioning" by A.Z. Toper, K.K. Kabango, R.D. Stewart, and A. Daehnke in the Journal of the South African Institute of mining and Metallurgy January/February 2000 describes this technique.

The technique could well be adapted to the cave situation where the ATC (advance track cutting) is mined completely in advance of the undercut and is the retreat out. Either blast what amounts to a pre split line of holes on the top of the under cut in panels or blast the top hole of the next ring as the face advances.

Advance Undercut

As a lot of people have found out advance undercutting does lead to sequence problems and possible production difficulties. This comes to light during the detailed planning stages:

Advance Undercutting
Development (including the slot raise) and Support
Drawpoint Construction - curing times
Drawbell Drill & Blast
Production

There are three basic stages of work that has to go on in the drawpoint leading to production and these will all be considered independently with possibly solutions to the difficulties that are encountered.

1. **Development and Support**

    The development lashing has to be linked to the production lashing and hence the direction of the drawpoints must be face the direction of the under cutting (see the sketches that follow.

    The tabulation over shows the cycles possible for a 1.8 metre blast and the various scenarios.

Development

1.8 m Advance Development and Support Cycle

2000/07/25

Activity	Amount	Unit	Time (Minutes) for Activity	Fatigue Factor	Progressive	**Remarks** Excludes Travelling Time
Spiling Bolts	11	Bolts	212	35	247	
Drill End (Jack Hammers)	40	Holes	200	33	481	
Charge	1	End	30	5	516	Pumped Emulsion
Blast	1	End	270	45	831	allow 4 hours Re-entry
Re-entry	1	End	30	5	866	Make Safe
Lash	1	End	85	14	965	100 m Tram to Tip
Install Services	1	End	35	6	1,006	Air, Water and Ventilation
Reinforced Shotcrete	1.8	metres	158	26	1,190	With Diamond Mesh
Straps	1.8	metres	126	21	1,337	with 3.0m Rock Bolts
Staples	5	Staples	525	88	1,950	
Install LHD Floor	1.8	metres	140	23	2,113	Not in Drawpoint Area

	Hours	**Days / Blast**
Drawpoint Development (pre Spiled - No Floor)	**28.4**	**2**
Drawbell Development (Full Requirement)	**35.2**	**2**
Slot Raise Development (No Support)	**12.6**	**1**

N.B. Holing Rounds and Spiling Bolts MUST be Remotely Drilled or have Been Pre- Examined

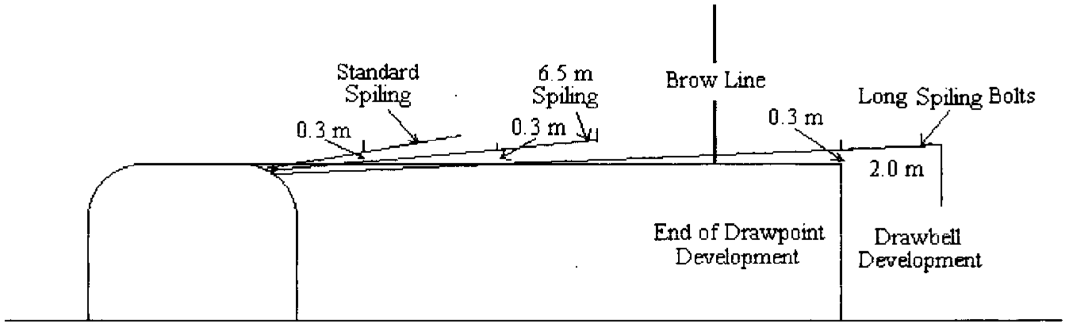
As can be seen the support is the most time consuming element of all. To try and reduce this it is suggested that the leg of the drawpoint to the drawbell could have spiling pre-installed and that this could even be extended to cover the drawbell.

This can be done in several ways. The actual situation on individual mines will depend on layout and the height of the development that is planned for the drawpoint to accommodate the drawpoint construction. Two suggestions are:

❖ **Pre-Spiling**

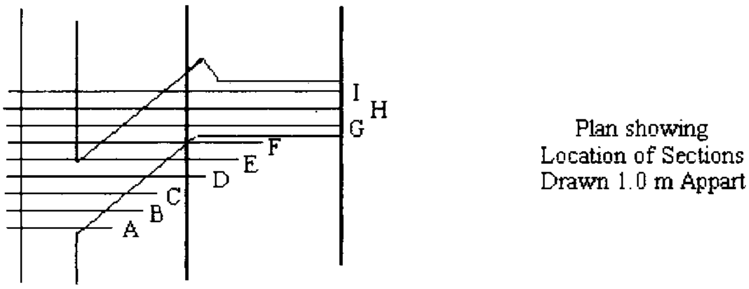
A series of holes are pre-drilled, along the line of the drawpoint, at flat angles from the extraction drift. The flattest holes drilled to the drawbell throat and further 2 metres beyond the anticipated limit of development. This is shown in the sketch over.

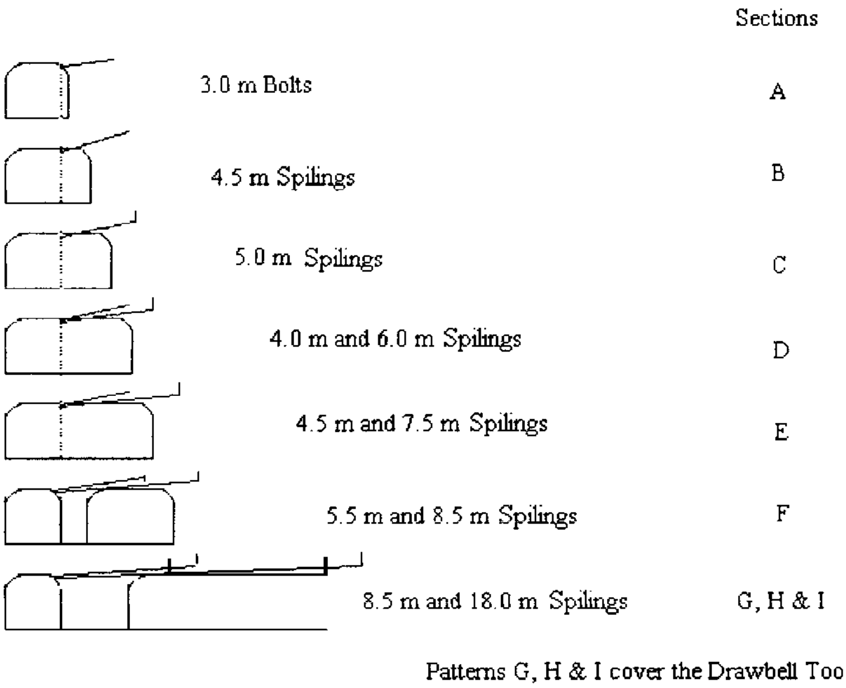
Into these holes are placed either heavy ropes or reinforcing bars that have threaded ends that can be coupled together. In either case the reinforcing is grouted. The ends of the reinforcing in the extraction drift are then secured using rock bolts and straps, which are ideally shotcreted.



Section along the Drawpoint
Showing Pre-Spiling for Advance Undercutting

The other suggestion is to drill a series of holes from the extraction drift at right angles. It must be remembered that the spiling bolts will need to be at least 2 metres on either side of the area to be supported and well grouted. This system can also cover the drawbell development if the slot has not been pre-blasted. This is shown in the sketch below.





Sections at 1.0 m Intervals Across the Collection Drift
Showing Pre-Spiling for Advance Undercutting

❖ Pre-Blasting Drawbell Slot

The other idea to reduce the time of development is to pre-blast the drawbell slot in its entirety, with sufficient relief holes. This being done from the undercut level, as the undercut is blasted back, provided the development is situated directly over the drawbell slot. There are however two disadvantages with this;

If it freezes, the blocks can be very difficult to dislodge and clear. However, there would be sufficient time to do this just before blasting the drawbell rings and

That any pre-spiling bolts and the last rounds holing into it would have to be, remotely drilled in case of miss-fires.

2 Drawpoint Construction

The remaining activities that have to be done for the construction of the drawpoint.

- Bullnose lacing
- Drawpoint floor
- Brow support
- Curing of the support concrete work

3 Drawbell Drilling and Blasting

Whilst the drawpoint construction is curing, the drawbell can be drilled. There after the drawbell can be blasted, the slot and rings.

4 Contingencies

A 15% contingency has been allowed. It must be remembered that all the development, support, construction, drilling and blasting is not happening in a single advancing end where everything is geared for it with whole crews in a high speed development situation. It is happening, possibly on a broad front, and certainly in at least four to eight drawpoints in an extraction drift that is also in production.

This gives a total of 56 days, which ties up with the 14 day advance of the undercut. The cycle is tabulated below.

Advanced Undercut Development and Preparation of the Drawpoints

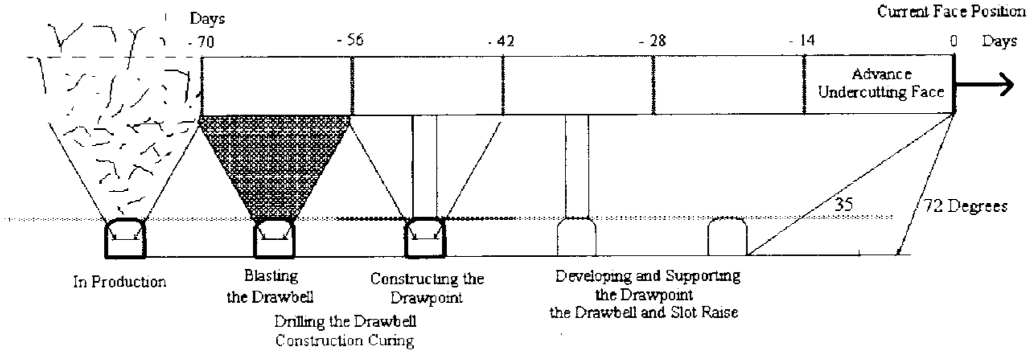
Activity	Amount	Units	Time (Days) for Activity	Time (Days) Progressive	**Remarks N.B. Days are 20 Hours to allow for Primary Blast Re-entry**
Develop and Support Drawpoint	10	Metres	11.1	11	Spiling Pre Done - See 'Development'
Develop and Support Drawbell	4.5	Metres	5.0	16	See 'Development'
Develop the Slot Raise	10	Metres	11.1	27	? Pre broken? from Undercut Level See 'Development'
Bulnose Lacing	2	Apexes	2.5	30	
Construct Drawpoint Floor	1	Rail Matt	1.6	31	with floor strapping
Brow Support	8	Metres	6.4	38	TH Arches or Other
Cure Concrete	7	Days	7.0	45	
Drill Draw Bell	600	Metres	**7.5**	45	**3 point drilling concurrent with the Construction**
Blast Slot	1	Slot	1.0	46	
Blast Drawbell	3	Rings	3.0	49	
Contingencies	15	Percent	7.3	**56**	Remembering this is NOT high speed development but one of a series of faces especially if on a broad front.
Total				**56**	Allowing for Full Days of Activities

N.B. Holing Rounds and Spiling Bolts MUST be Remotely Drilled or have Been Pre-Examined

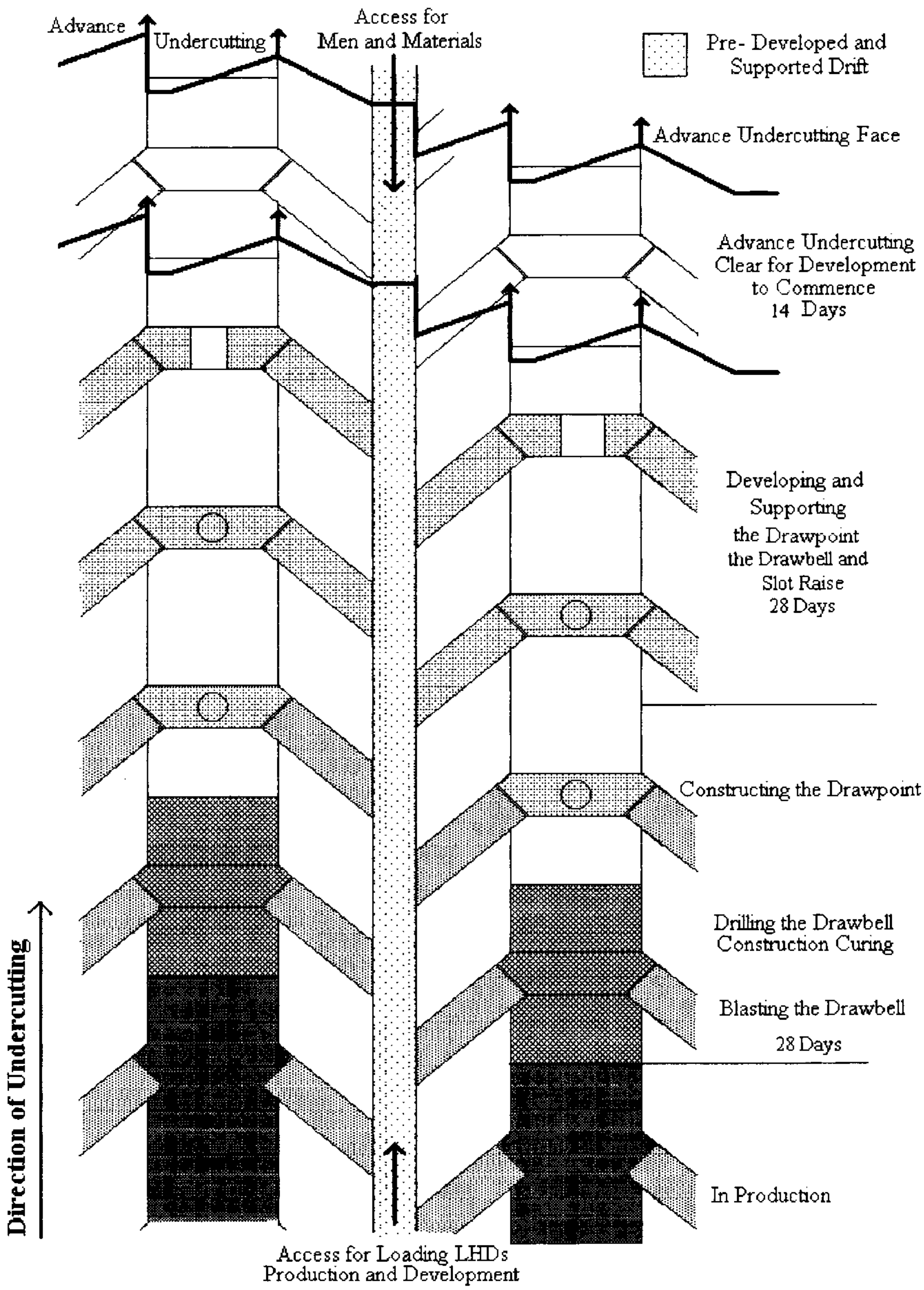
Using this scenario a new line of drawpoint will be into production every 56 days and the undercut will advance continuously. Depending on the drawpoint spacing and the complexity of the drawpoint construction this scenario could vary.

The cycle is shown in the three sketches that follow:

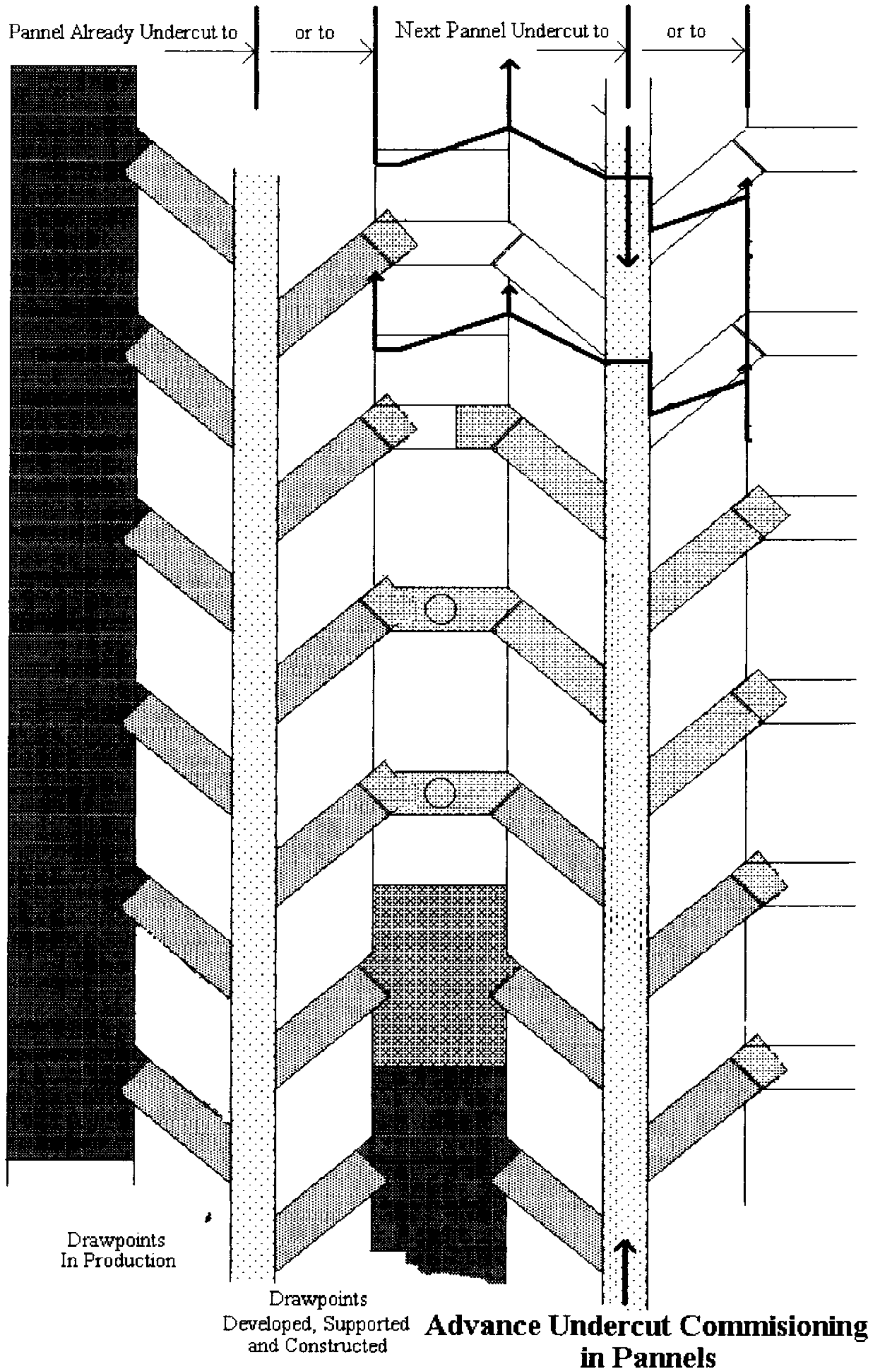
A section along the brows of the drawpoints,
A plan for a broad front approach,
A plan for a panel retreat and
A section across the major apex showing panel alternates.



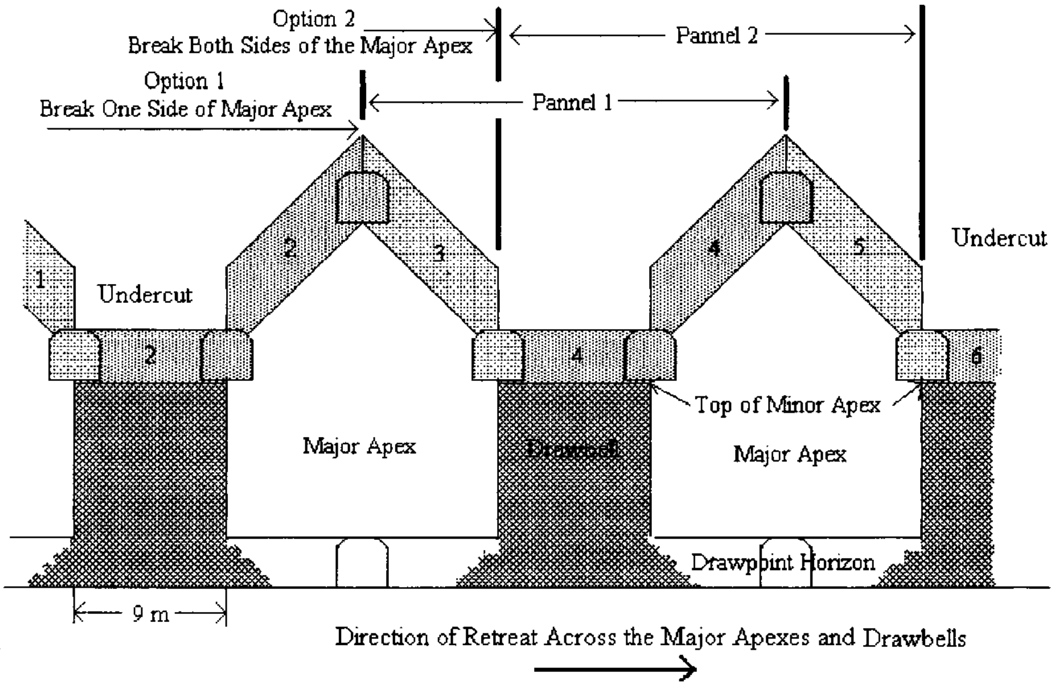
Section through Drawbells showing Advance Undercut



Advance Undercut Commisioning on a Broad Front



Pannel Already Undercut to
or to
Next Pannel Undercut to
or to
Drawpoints
In Production
Drawpoints
Developed, Supported
and Constructed
Advance Undercut Commisioning
in Pannels



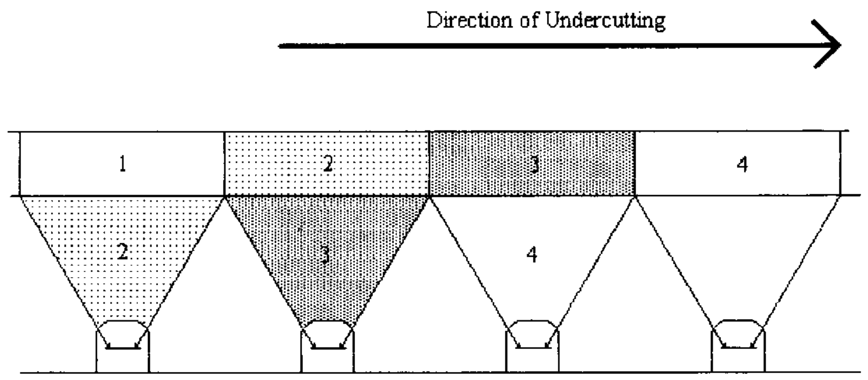
Breaks in Undercutting for Pannel Retreats and Shaped Major Apex

Post Undercutting

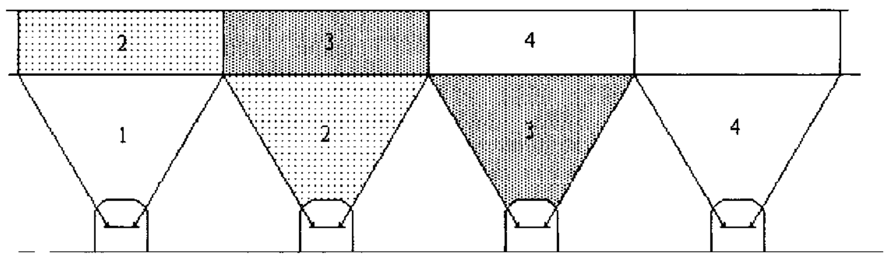
There is often a debate as to which the sequence should be – cutting the undercut and then the drawbells, or vice versa. Both sequences have been tried at Shabanie and Gaths and it was found that undercutting preceding the drawbells works best as the drawbell slot breaks more readily. The rings then follow very rapidly to bring the drawpoint into production.

When the cones are mined first there are less opportunities to deal with any problems on the undercut faces and the likely hood of pillar forming seems to be increased, possibly because loss of holes owing to the previous blasting of the drawbell.

The sketches over shows the sequences as discussed.



Undercutting Preceeds Drawbells



Undercut Follows the Drawbells

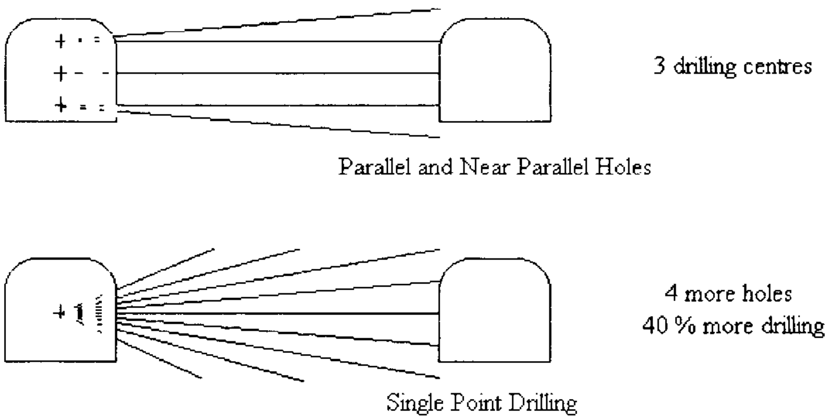
Design Shape and Height

When designing the undercut the amount of relief for swell that can take place into the development should be an ideal of 30 percent. If a design is less than this, one will undoubtedly have to do some lashing, even at 30 percent lashing will be required from time to time to clear build up in the development drift.

Several factors must be taken into consideration with the undercut design and the variations accepted will be what individual mines believe is the easiest for them to manage:

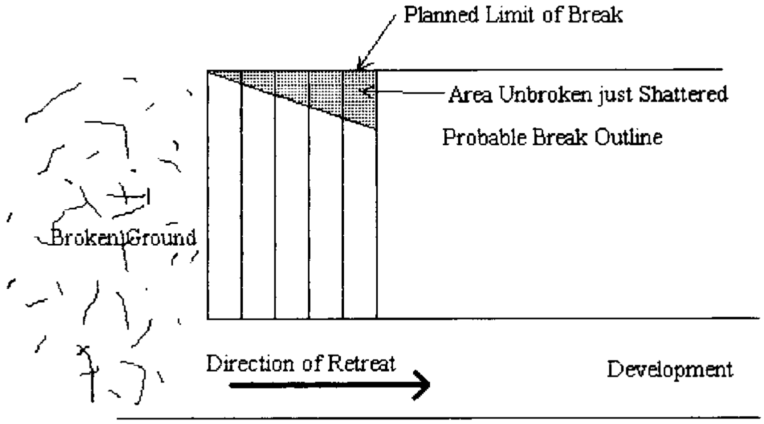
- The size and location of the undercut development - is there a case for wider drifts to increase the swell relief? Again, it is a question of economics as to what is the optimum width. It is a debate between width and additional support costs. The support being that which is required for the safety of personnel working both during the development, support and undercut blasting phases of the development's life cycle.

- Blasting Pattern design. With out a doubt for narrow undercuts, parallel holes yield the best tonne/metre drilled however unless the drill rig can do this simply, it can be laborious and time consuming, hence a regular single point pattern is preferred. See sketch below.

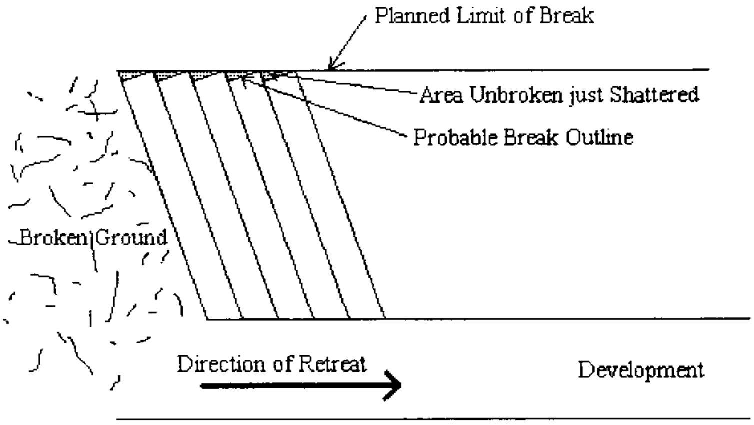


Undercut Drilling at 70 Degrees
Parallel Holes or Single Point

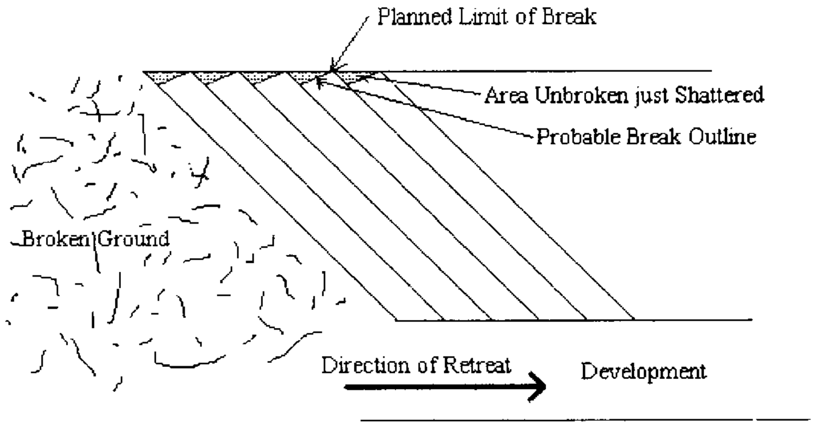
- Forward angled rings without a doubt give a better throw of material into the available swell relief space in the development. Angles somewhere between 45° and 70° should be considered. It has always been said that 70° was the natural rip angle for blasting and will probably be the point to start any discussions and debate regarding the angle for the rings. This is clearly shown in the sketch over, which for completion also shows 90° patterns.



Undercut with 90 Degree Rings



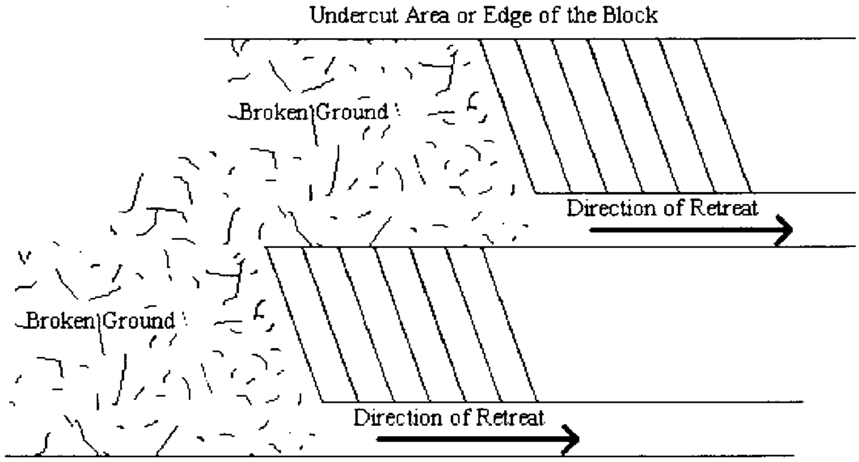
Undercut with 70 Degree Rings



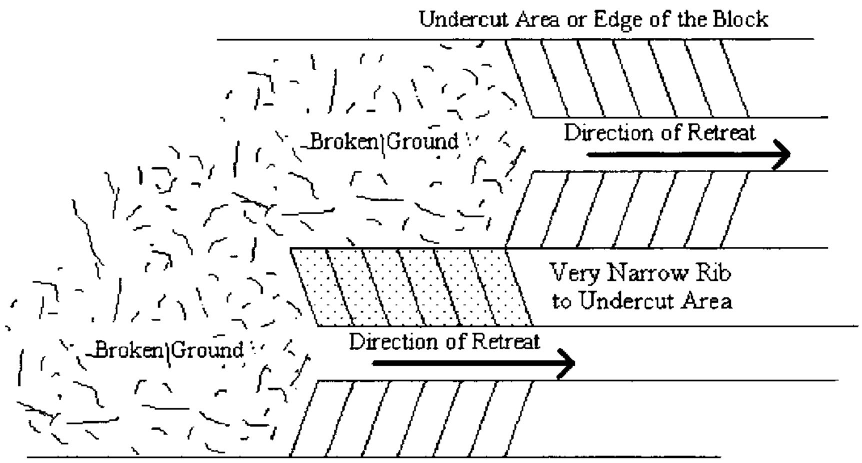
Undercut with 45 Degree Rings

Plans of Undercut Breaking Options

- Next consideration is; whether or not to blast from both sides into the central development drift. This is shown in the sketch and I believe this is dependent on the relative costs of development and the subsequent problems in dealing with any remnant pillars that might form. In addition there is always a concern about the small rib pillar that is formed to the next drift. Shown in the sketches below.



Single Side Undercutting

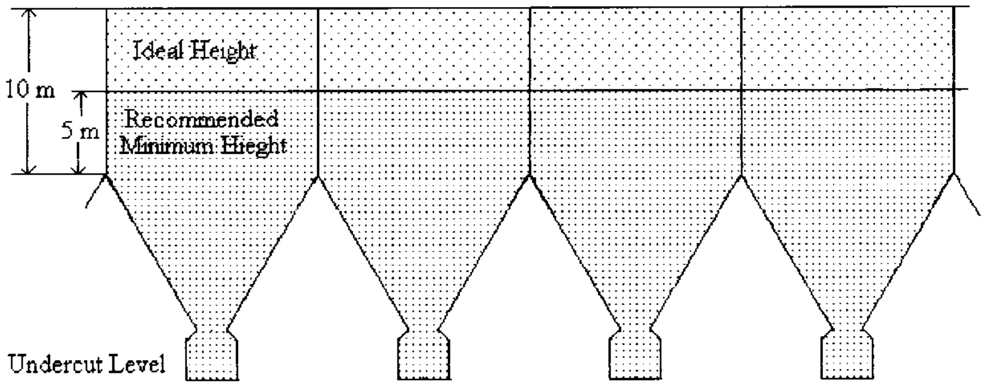


Both Sides Undercut

In Shabanie Mine's horizontal undercuts it has been found that single side blasting proved the easiest to manage as the previous development was used and available as an anti-socket drive and to reduce the chances of remnant pillars being formed.

When one is blasting from both sides into the centre, there is a tendency to be too optimistic about the space for swell relief. In this situation there is an argument for developing a wider undercut drift.

- Where the undercut is being mined like an initial cut in a sub level cave one must minimize the chances of a pillar developing on the top of the apex between the drifts. A minimum of 5 metres vertical should be built into the design – ideally this should be 10 metres. See the sketch below.

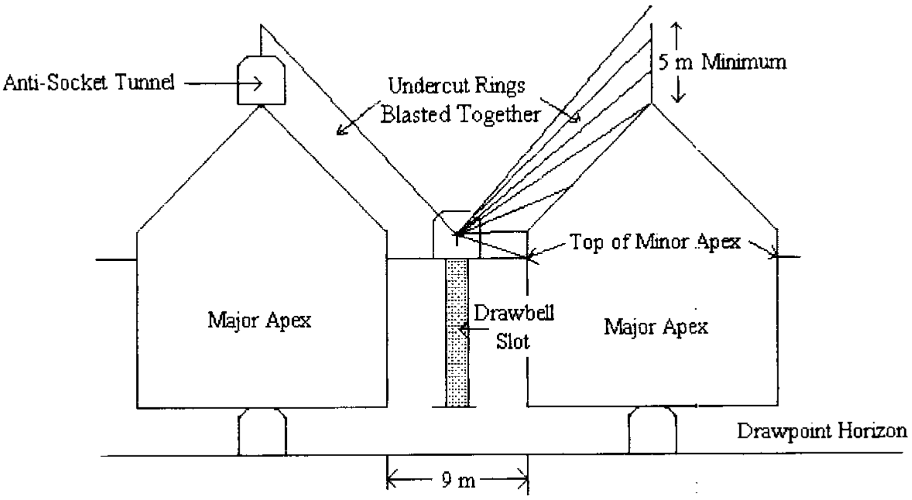


SLC Style Undercut

- If the drawbell slots are to break into broken ground they must hole a development drift on the undercut so that the holing point can be pre-examined for miss-fires and logged as such ready for holing.

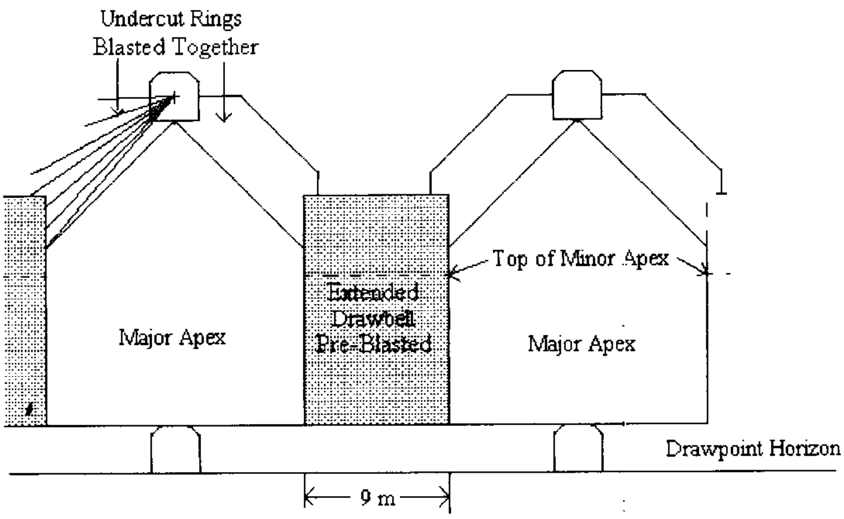
- The shaping of major apexes is possible either;

  From a single drift along the centre of the drawbells, coinciding with their slots. Swell relief is likely to be a problem.



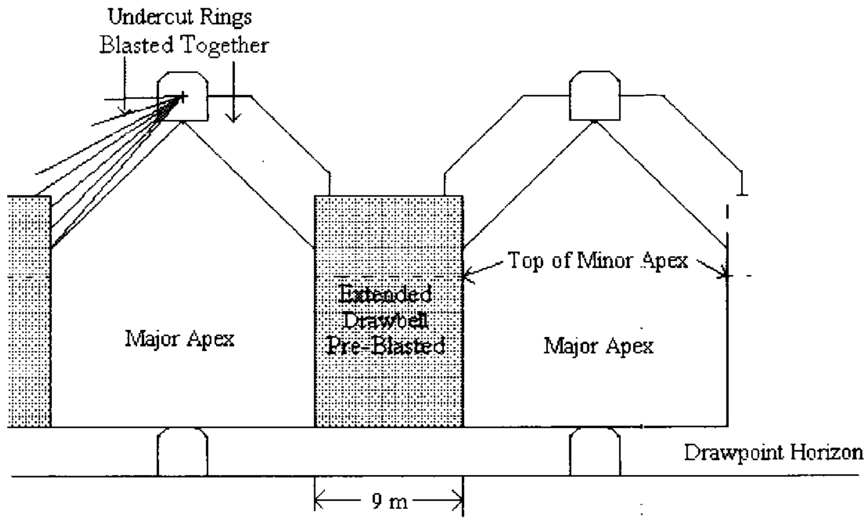
Single Breaking Drift Along the Centre of the Drawbells

  From a pair of drifts one along each brow line, in which case the drawbell slot should be located directly under the first one broken. Swell relief doubled.



Breaking from Top of Major Apex into Drawbell

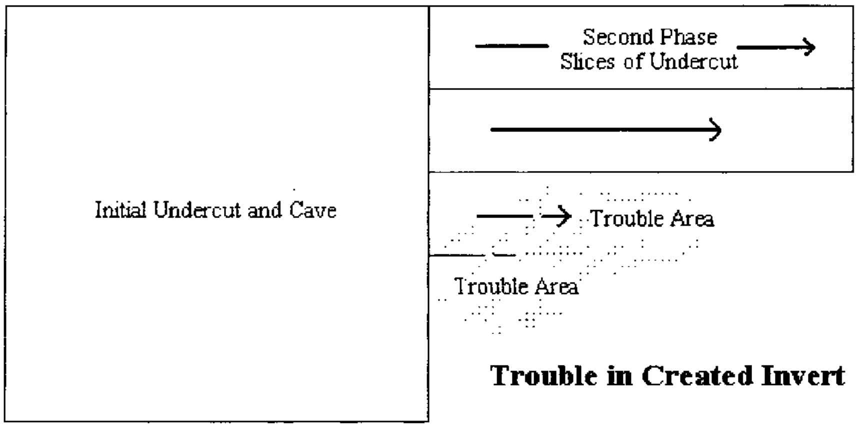
From a drift on the top of the major apex drilled down. Swell relief would be a major problem unless the drawbells are pre-mined and the swell drawn through them.

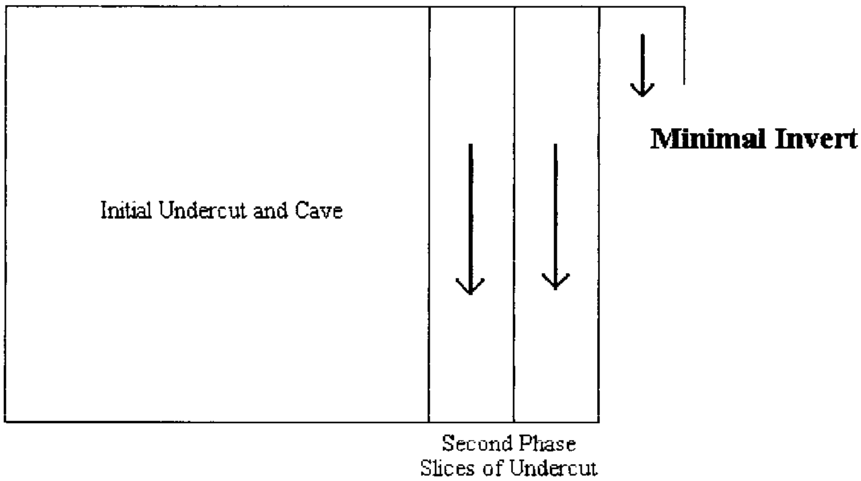


Breaking from Top of Major Apex into Drawbell

In many ways the two drift on either side of the drawbell are most advantageous. With the two drifts on the drawbells one two options where you could just undercut a set of drawpoints, you could undercut either post or advanced and mine a pre-undercut on the opposite face.

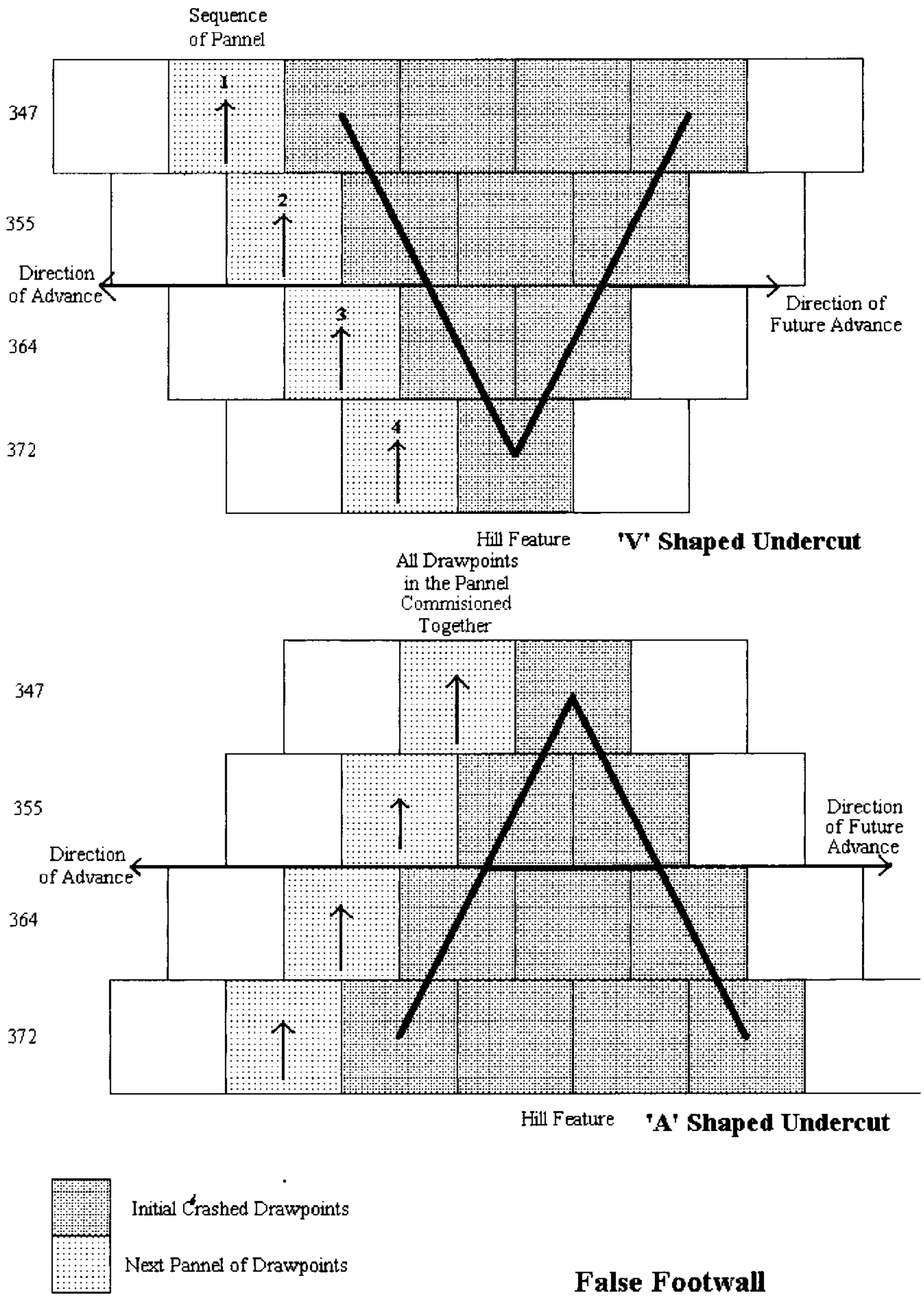
- In the design of the sequence of blasting it is important that the undercut area be expanded logically, uniformly and as "round" as possible. Ensuring that no large inverts are allowed to develop as shown in the sketch over.





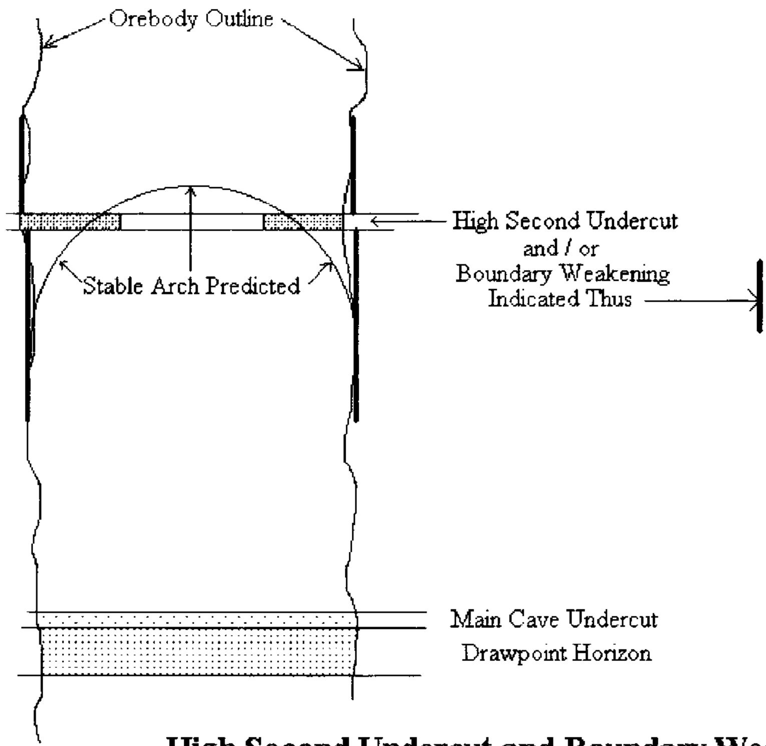
Problems if Undercut NOT Kept as "Round" as Possible

On incline drawpoint layouts, whether the face should be taken as a '**V**' or as an '**A**' was hotly debated. If incline draw as at Gaths Mine Section of Gaths Mine is anticipated, the ideal is definitely the '**A**' shape. However, this must be carefully managed and the sequencing of the blasting has to be absolutely controlled. Hence the production officials prefer the '**V**'. See the sketch over.

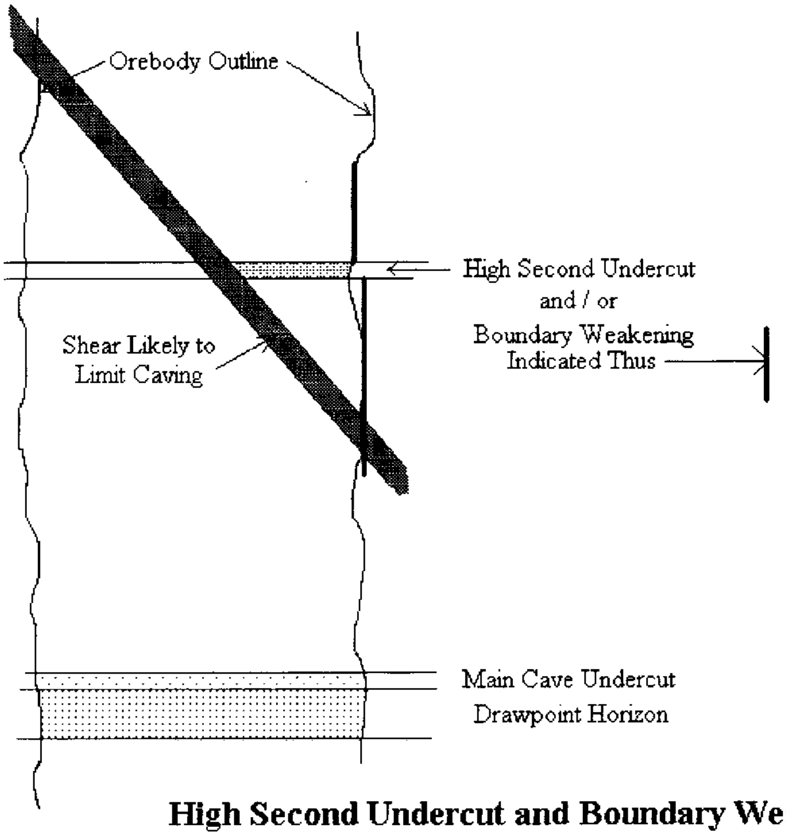


False Footwall Undercutting Sequence Options

- Consider if there is there a case for pre-undercutting and or boundary weakening at a higher level/s to prevent overhang? Predicting the stress effect, or possible effect of shears etc. which could lead to overhang situations. If one pre-undercut/boundary weakened at a higher elevation to again increase the hydraulic radius once the main cave reaches that elevation clearing the potential overhang. See the sketches below and over.



High Second Undercut and Boundary Weakening Where Stable Arch Formation is Predicted

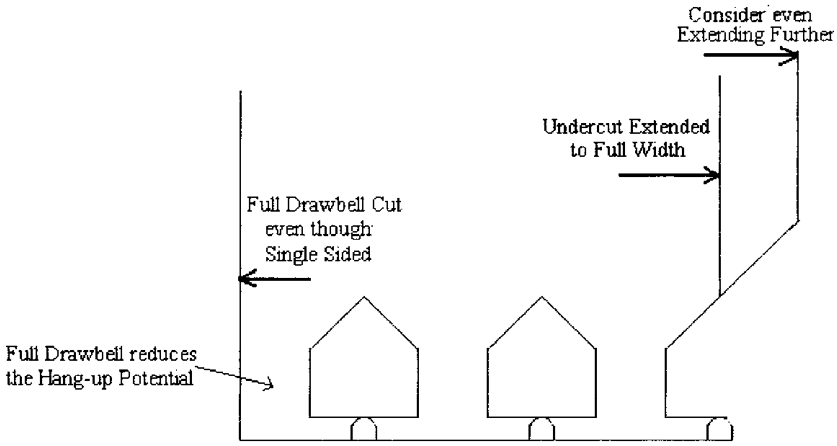


High Second Undercut and Boundary Weakening Where a Shear is Likely to Limit the Caving

❖ To avoid tight corners, inverts, and potential overhangs and to help the draw, undercuts should be extended e.g.

i. Gaths Mine upper level slot and bottom level narrow undercut.
* ii. Single side drawpoints a full nine metre long drawbell.
* iii. High slots on margin of cave area or final major apex.
iii. To break shears that might "punch" into access development when loaded in the abutment
v. To link to shears that might assist the caving process.

* See drawing over



Extention of Undercut Area to Reduce the Overhang Potential

Rate of Advance

The rate of undercutting is absolutely critical for stability in the undercut area and for ensuring continuity of the undercut.

The best form of undercut is an undercut that is expanded by regular small blasts. It is better to blast everyday a small portion, say a 1 metre x 10 metre area ($10m^2$).

Stress in the abutment (advancing face) do not build up because they are constantly moved further back before they can actually build up a large load. See keynote address by Diering. The blasting shock waves are also a key element in disrupting the stress build up and triggering seismic events during the four hour reentry period when no personnel are at the site.

By doing large areas irregularly damage is caused by the blasting shock waves and sudden release of the stresses in the abutment as they build up to a large load and can release abruptly. Large blasts of say $250m^2$ once a month cause more damage than a $10m^2$/day or $300m^2$/month; there is also a greater chance of "pillars" forming.

Once the undercutting has reached the size that is planned and no further undercutting is planned then one acknowledges there will be a steady build up of stress in the abutment that must be considered in the design phase particularly of the support.

Continuity of Undercutting

The undercut must be fully broken.

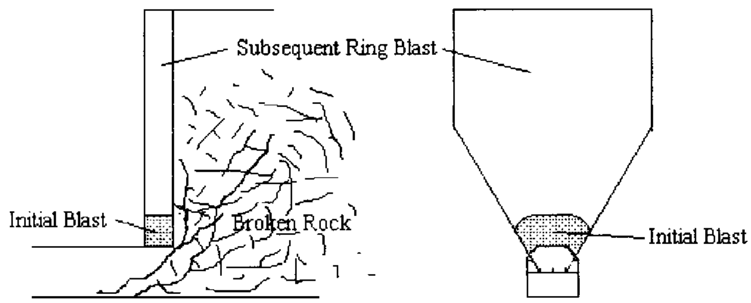
The vital importance of this needs to be stressed again and again.

The cost of mining and supporting an anti socket drifts on an undercut has been frequently debated, particularly with shaped crown pillars. If the anti-socket drift is not mined, management must decide how it is going to deal with any build-ups that occur at the top of the major apex. This could be with considerable difficulty and probably a great deal of danger. It will be up to individual mine managers to decide which is preferred.

Undercut Blasting

It has been found with ring blasting, particularly at Shabanie that where relaxation damage occurs around the development for a distance of some 1.5 to 3.0 metres owing to relaxation of the ground (no lining). Rings have to be blasted in two phases; first phase is to charge and blast off the relaxed (broken) portion of ground and then charge the remainder of the holes.

One ring at a time being blasted and the muck from the first blast is only lashed to clear the way in to charge the rest of the holes. The ground is left as a choke and a protection for the chargers. See sketch below.



Breaking of Rings in Two Sections when Relaxation Takes Place Near the Drift being Blasted

Mass Blasted Undercuts

These have worked, however, preparation time for these is large and it is uncertain that they actually achieve what is required. Sequencing the blast can reduce the blast damage; however, it has been found that a regular area increase is better.

8 cases of explosives / delay seems to be the maximum with out shock waves causing damage to support and development, particularly in a direction parallel to the structures.

Boundary Weakening

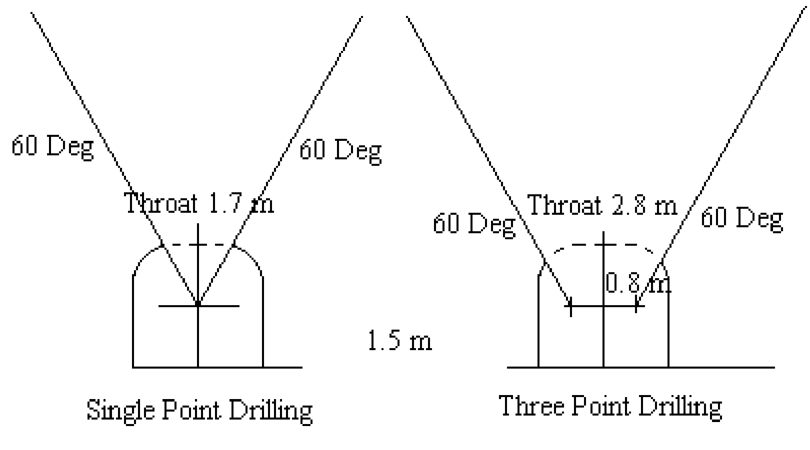
Boundary weakening is one of tools in the armory of the cave miner during the undercutting phase. Boundary weakening needs to be considered, particularly when high strength rocks exist in the perimeter area against weaker zones in its core leading to overhangs.

There are two ways to create the weakening;

> The most effective is to mine a slot in the required position; this can be done with conventional rings or large diameter long down holes.
>
> The alternate is to blast a pre-split, either conventional drilling or large diameter holes.

The boundary weakening is normally done from the undercut horizon drilling and blasting up holes. There is no reason why this could not be done effectively be for the undercut phase from a higher elevation as shown in the sketches earlier.

Drawing from N.J.W.Bell:-



The Advantage of using 3 Point Drilling to Create the Drawbell from the Drawpoint

DESIGN TOPIC

Ore Handling Systems

Previous standards for handling ore from block cave layouts were by slushers into orepasses or directly into cars, through drawpoint grizzleys down an orepass onto a conveyor or into cars / trucks and by LHD from the drawpoint into the nearest internal orepass. Panzer conveyors have been tried without much success. Gathering arm loaders have been used in drawpoints without success. Because the hauling distance was considered critical for LHD's orepasses were kept as close as possible. There is no doubt that productivity of LHD's is at its highest with short hauls and fine fragmentation.

As more block caves are being develop in competent orebodies with coarse fragmentation the ore handling process has been under review with some major changes. The LHD is being used to haul the ore long distances to crushers outside the orebody. In high stress environments long orepasses are often severely damaged and require costly linings, in these cases the focus has been on moving the ore horizontally to the crushers.

EFFECT OF ROCK BLOCK SIZE

There is little doubt that in a low stress environment and with fine to medium fragmentation - +2m³ less than 5% - that the internal orepass system down to a collecting level is the most efficient ore handling system. Once the fragmentation be comes coarse to very coarse - +2m³ greater than 20% then alternative methods need to be considered.

The one technique is to remove all the material that an LHD can handle to outside the production area and either put it straight into the crusher or onto a grizzly so that oversize can be handled without interfering with activity in the production drift. The system on Bell Mine, Quebec, was introduced many years ago. Here the 5yd LHD's haul the ore to a large in area grizzly where three pick hammers operate. The grizzly throughs go into a crusher then onto a conveyor to the shaft.

On Northparkes the 8yd LHD's haul directly to crushers on either side of the orebody in a mirror image horizontal herringbone layout. There can be delays with large strong rock blocks

It is not possible to drop the largest rock into the crusher and if the LHDs are large and capable of delivering a high percentage of strong large rock blocks then the pick hammers might not be able to cope, in which case the breaking in the drawpoint has to be more effective.

Effective secondary breaking system in the drawpoint are essential for high productivity - this will be discussed in a later section. The tip opening and orepass diameter or the size of the grizzly on the level will determine the maximum rock block that cam be handled by the LHD and the system. The efficiency of rockbreakers / pickhammers is a function of the RBS (rock block strength) and not the IRS.

OREPASS SYSTEMS

Orepass diameters / dimensions

The diameters of orepasses with respect to particle size are based on empirical rules established from extensive studies done on orepasses in pits. A rule of thumbs which states that if the diameter of the pass is five times the largest rock then 100% of the material will flow with no hangups, with four times the diameter then 90% will flow and with three times the diameter only 80% will flow with out hangups.

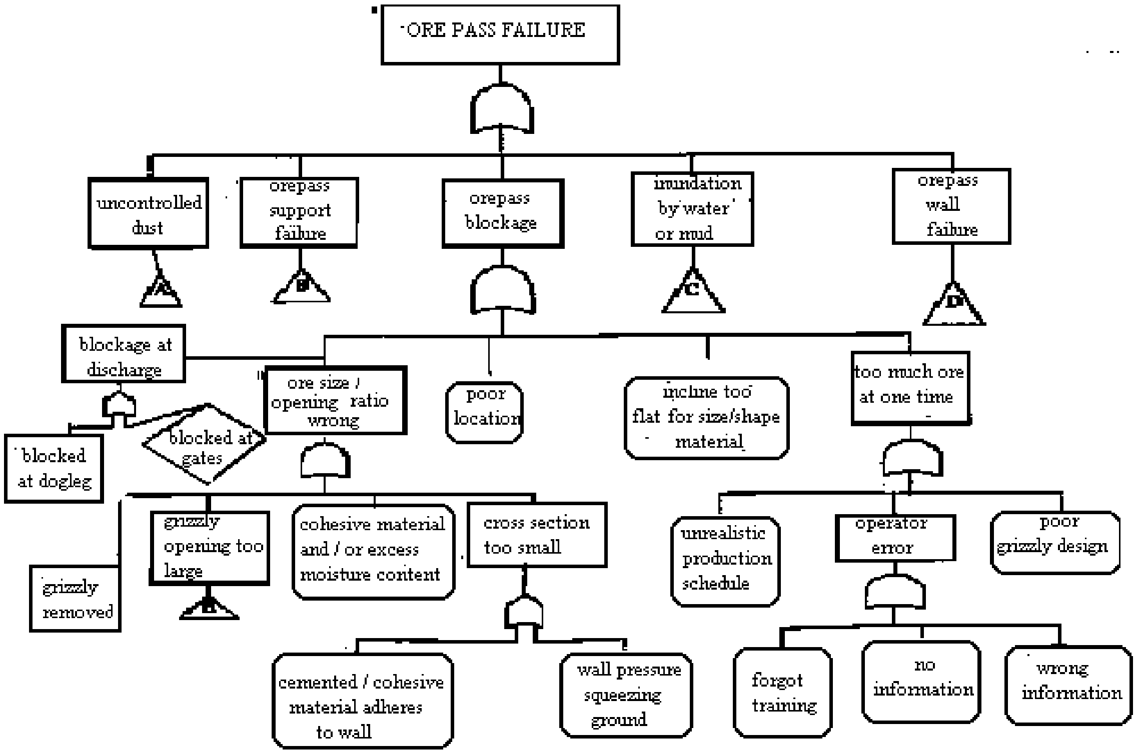
An interesting point is that raise bored orepasses are said to be prone to hangups. The ‘smooth’ rifled effect of the bored surface seems to have high friction characteristics. The same size pass developed by blasting does not hangup because of the very irregular surface and that there are faces inclined into the sidewall making it difficult for arches to form / anchor into the sidewall.

Orepass Length

Long orepasses have been effective in low stress environments, but are prone to stress spalling problems in high stress areas. Long (+40m) orepasses on high stress mines have spalled with the result that 2m passes become 12m In these situations the length should be limited to within in the stress relief shadow. On grizzly layouts draw control problems have been experienced with long orepasses when the control has been the filling of an orepass from a grizzly.

Orepass Wear Characteristics

The MRMR will indicate the wear characteristics of the orepasses and the need for support.. The following chart shows the various factors that affect the performance of orepasses and their inter-relationship :-



Orepass Linings

Orepasses need to be lined at the tip with concrete and rails. The extent of the lining will depend on the RBS and the tonnage that will flow from that drawpoint. Some mines have developed simple and effective systems such as lowering tub sections.

COMPARISON OF TRUCKS, CONVEYORS AND RAILROAD

Studies on the merits of trucks versus conveyors versus railroad seem to be done on a regular basis without conclusive results and decisions appear to be made on personal preferences. It would be useful to have an unbiased assessment or the conditions that pertain to the correct selection. It is not expected that there will be many improvements in railroad design or major changes to conveyor systems, however, there appears to be ongoing improvements to truck design for underground operations.

NORTHPARKES MINE COMMENTS ON DIRECT HAUL TO CRUSHER

The benefits of direct tramming to a crusher versus to an orepass with a haulage below is difficult to quantify. A comment on a few main areas;

- Capital costs - significantly reduced through less lateral development as on haulage level, no ore pass development (orepasses can affect the stability of the production level) and fixed pickhammer and equipment designed for large tonnages and out of production area.

- Operating costs - again significantly reduced due to less equipment, personnel, ventilation/power requirements and maintenance - however, the LHD’s are doing the hauling and this is not their specialised task .

- Other benefits - We see improved efficiency through direct tip - whatever fits into the LHD bucket goes into the crusher. Better availability of tip points As there no concern about grizzly / pickhammer / maintenance. General improvements that go with removing a line (stage) in the ore handling system.

- The only downside is perhaps the longer trams, however, with good roadways they are achieving very high speeds with the LHD’s which tend to negate this.

Statistics on the Northparkes crushers - Krupp double toggle jaw crushers, 88 inch by 54 inch, closed side setting of 180mm (closest we can get them), throughput rate of 600 t/hr and can handle rocks of 2m³ (can be smaller or slightly larger depending on shape.

Automation - Northparkes were operating one quadrant under teleremote, but, stopped due to poor productivity - 20% less than manual and costs - significant and costly damage was being done to the loaders due to collisions with the walls. They are sponsoring a CSIRO project into automation. Northparkes expect to have automation by August 2000.

INCLINE DRAWPOINT LAYOUTS

Incline drawpoint layouts lend themselves to the use of Large LHD’s and to the handling of large rocks. However, as it is a multi level layout the crusher will be on a lower level and ore handling will be down orepasses in the footwall. There will be space at the tips for pickhammers.

ASSESSMENT

DESIGN TOPIC

Horizontal LHD Layouts

GENERAL

Block caving is the lowest cost underground mining method, provided that drawpoint spacing, drawpoint size and ore handling facilities are designed to suit the properties of the caved material and that the extraction level can be maintained for the draw life. In the past caving has generally been considered for rock masses that cave readily and have fairly fine fragmentation with draw heights up to 150m. More recently draw heights of + 400m are being planned, where:

- orebodies where the extraction horizons are located in a competent rock mass,
- the caved material has coarse fragmentation
- the extraction horizon can be supported to handle the large drawpoint tonnage,
- the correct undercutting procedure is used to reduce abutment stress effects,
- consideration is given to a reclamation level (piggy back).

Cave mining is currently taking place in high stress environments with associated rockburst problems. These problems can be reduced or eliminated by using advance undercutting techniques. Once again the system has to be used correctly, one cannot apply conventional undercutting principles to advance undercutting.

Fragmentation of the caved material is a major factor in determining the drawpoint spacing and productivity. A block cave fragmentation program - BCF - has been developed to determine cave fragmentation and has been successfully used on many operations.. It is a tool and in DOS form is basically a simple program providing a lot of useful data for planning purposes.

When low cost underground mining methods are being examined for the extraction of large, competent orebodies then cave mining must be considered, particularly, in a high stress environment. The ability to define cavability and fragmentation, the availability of large LHD's, sound draw control knowledge, improved secondary drilling equipment and reliable cost data have shown that competent orebodies with coarse fragmentation can be exploited by cave mining at much lower costs than drill and blast methods. LHD layouts are the automatic choice for medium to coarse fragmented orebodies

The wear and condition of brows is critical with the greatest damage done during the conventional undercutting period. In the case of the drawpoint and the drawbell not being in line, the main area of wear in the brow is on the inside corner of the junction . If the vertical height of pillar above the brow is small, failure of the top section will reduce the strength of the lower section and result in aggravated brow wear.

More thought must be given to the design of LHD layouts in order to provide the maximum amount of manoeuvring space for the minimum size of drift opening so that larger machines can be used within the optimum drawzone spacings. Another aspect that needs attention is the design of LHD's to reduce the length and increase the width. The optimum width will be a function of extraction level stability. Whilst there might be an attraction in using large machines, it is recommended that caution be exercised and that a decision on machine size be based on the correct assessment of required drawzone spacing in terms of fragmentation. The loss of revenue that can result from high dilution and ore loss far exceeds the lower operating costs associated with larger machines. To the writers knowledge manufacturers have not been approached to design a machine for block caving operations.

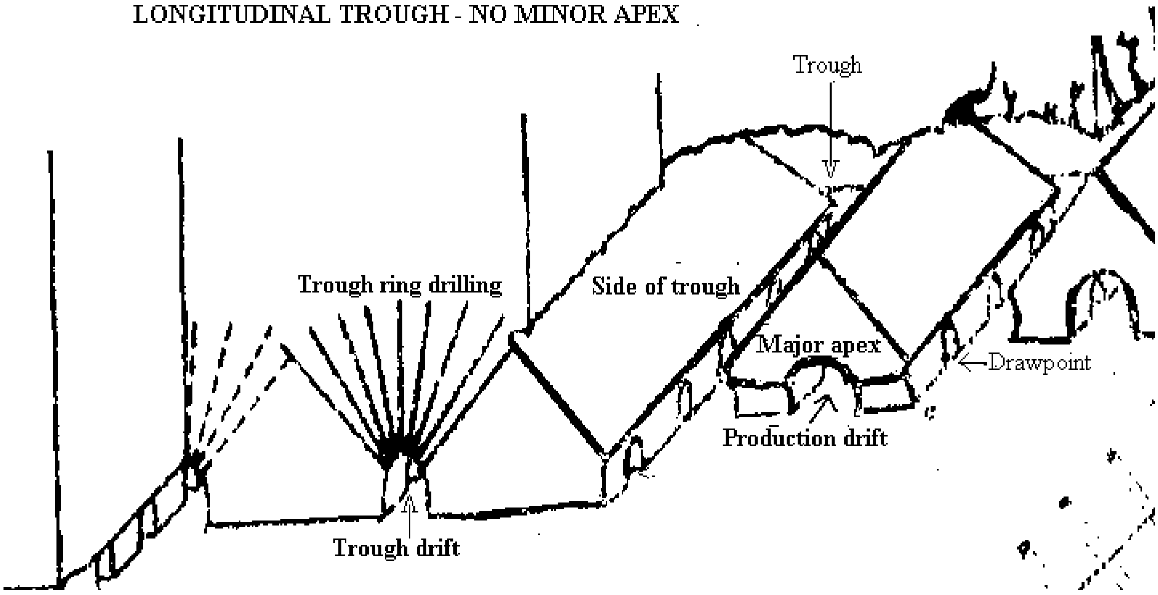
FRAGMENTATION

The fragmentation data determines the drawzone spacing, it also has a bearing on the height of the drawpoint so as to make large rocks available for secondary breaking, of course a high brow means a longer rill zone. The selection of the method of draw and its layout is dependant on the fragmentation. The shape of the major apex is also influenced by fragmentation.

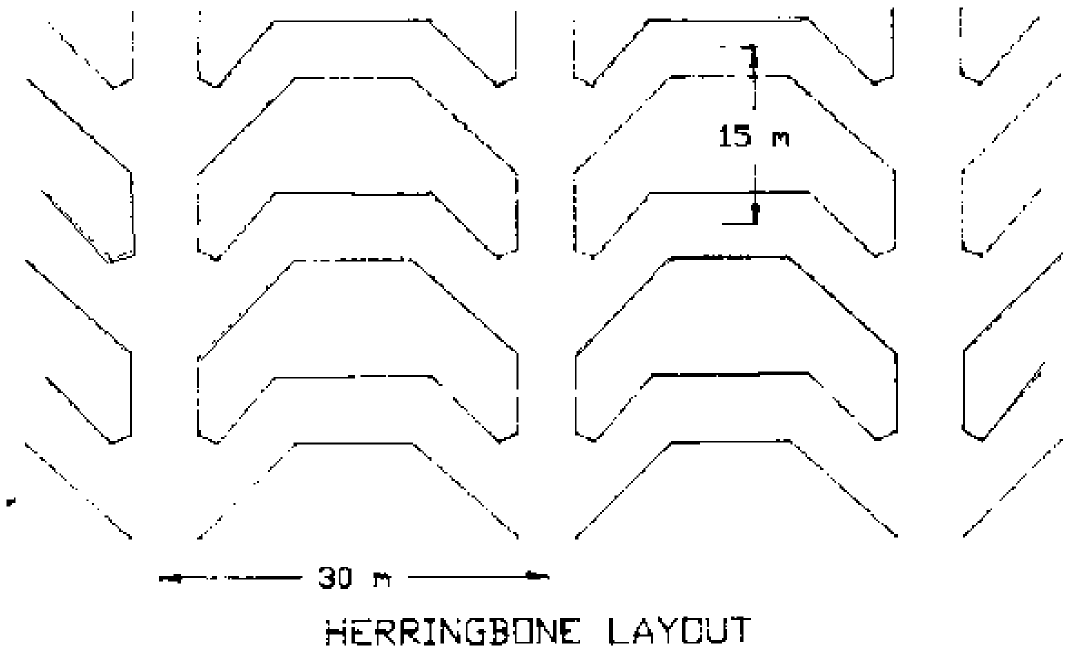
HORIZONTAL LHD LAYOUTS

Ten different horizontal LHD layouts are or have been in use at various block caving mines:

Continuous trough/trench at Austro - American Magnetite Co. Austria and also used at Shabanie Mine, Zimbabwe and San Manuel. .The layout used at American Magnesite is shown in the following diagram. The undercut trough was developed and blasted before the production drifts and drawpoints, that is, an advanced undercut. According to the description (Denver 1981) this helped the stability of the major apex. One of the big disadvantages of that layout is that drawpoints are using the same draw cone. In terms of stability there is no lateral restraint to the major apex as is normally provided by the minor apexes. If the trough / undercut drift is elevated then a small minor apex is created. If the drawpoints are retreated then the brows are more widely spaced resulting in better draw interaction.

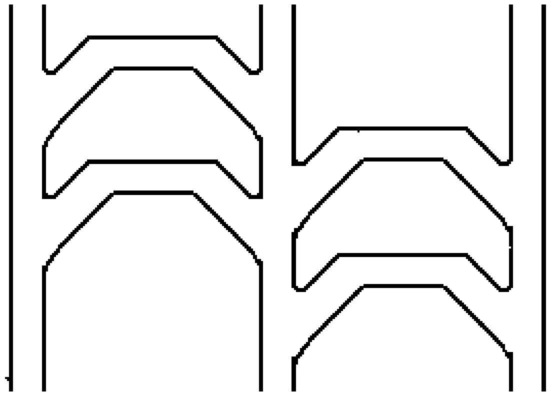


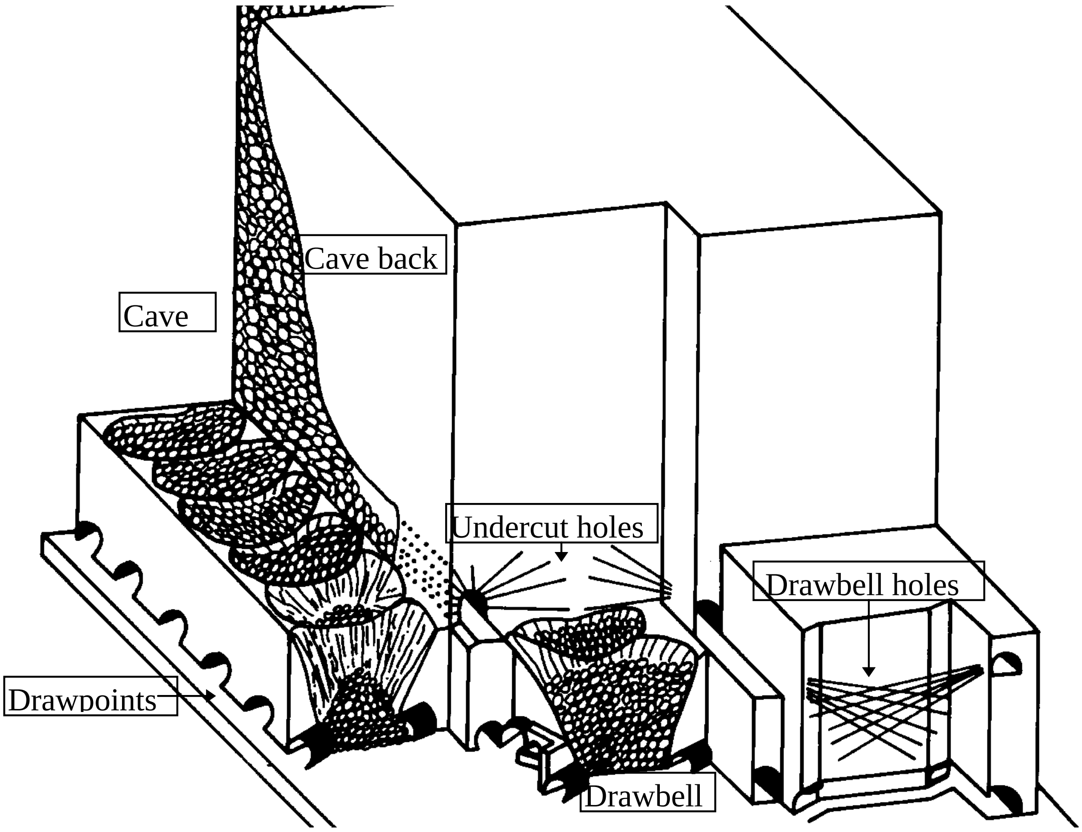
Herringbone with opposite drawpoints at King and Shabanie Mines, Zimbabwe



One of the major problems with this layout is the structural weakness owing to the large spans in the production drifts at the drawpoint takeoffs.

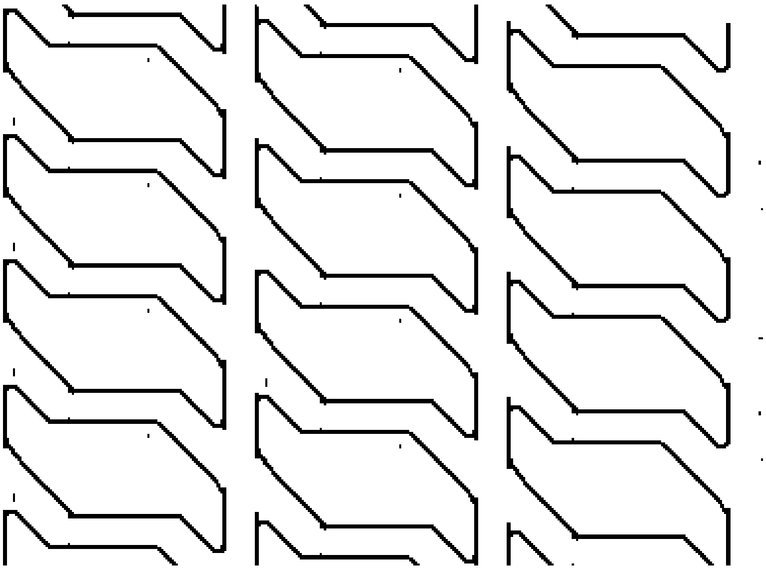
Herringbone with staggered drawpoints, Henderson Mine, USA , Bell Mine Canada and other more recent operations





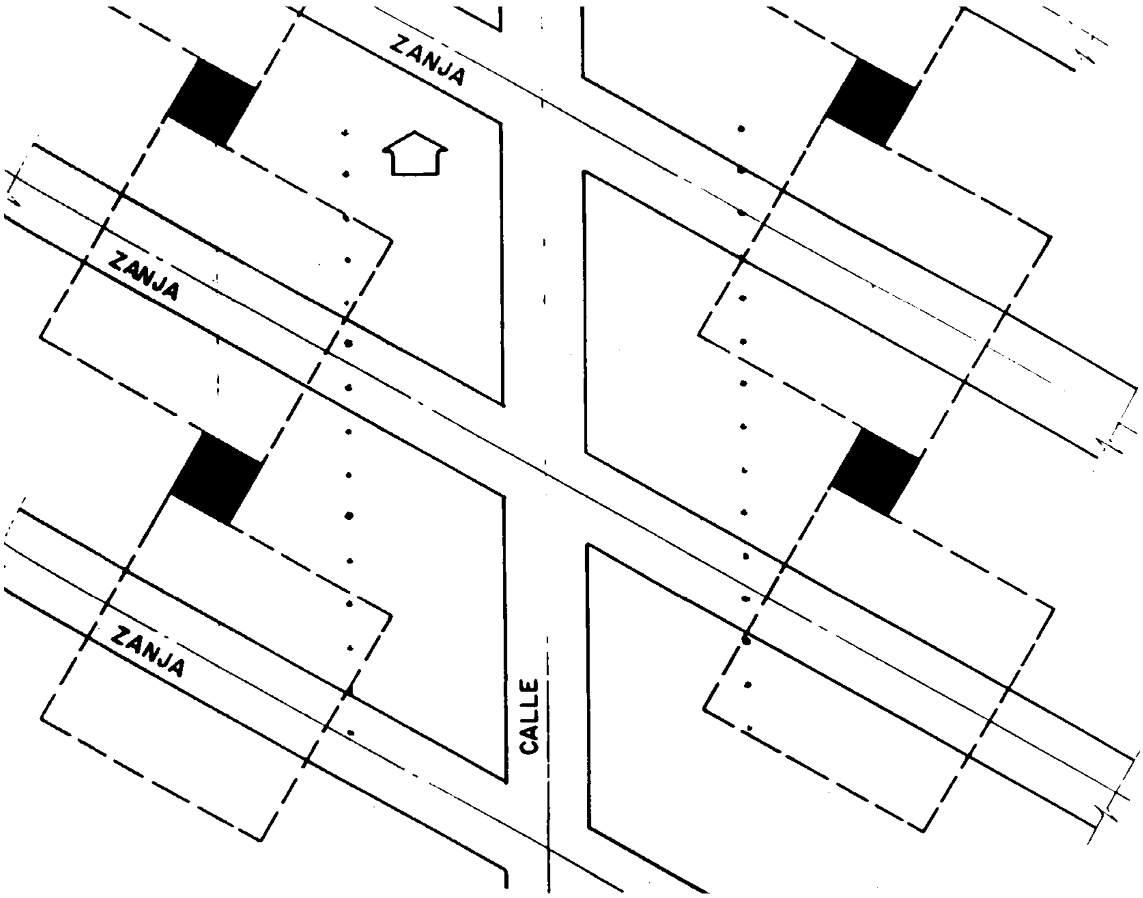
The above diagram is a ‘ cut out’ of the Henderson layout showing the configuration and sequence of operations. The drawbell drift is only developed immediately prior to blasting the drawbell so as maintain stability in the area. The drawbell is created by blasting holes drilled from the undercut level into a V cut in the back of hte drawbell drift. The undercut is then blasted into the drawbell. This technique was developed early in the life of Henderson Mine and has been used to develop thousands of drawpoints. The important point is the short period of time involved in creating the drawbell, thus limiting the effects of abutment stresses.   It is surprising that such a successful technique has not been used on other mines.

Z' design with drawpoints on line, but, drawbell drift and drawbells at right angles to production drift. Developed at Henderson Mine USA and no longer used.

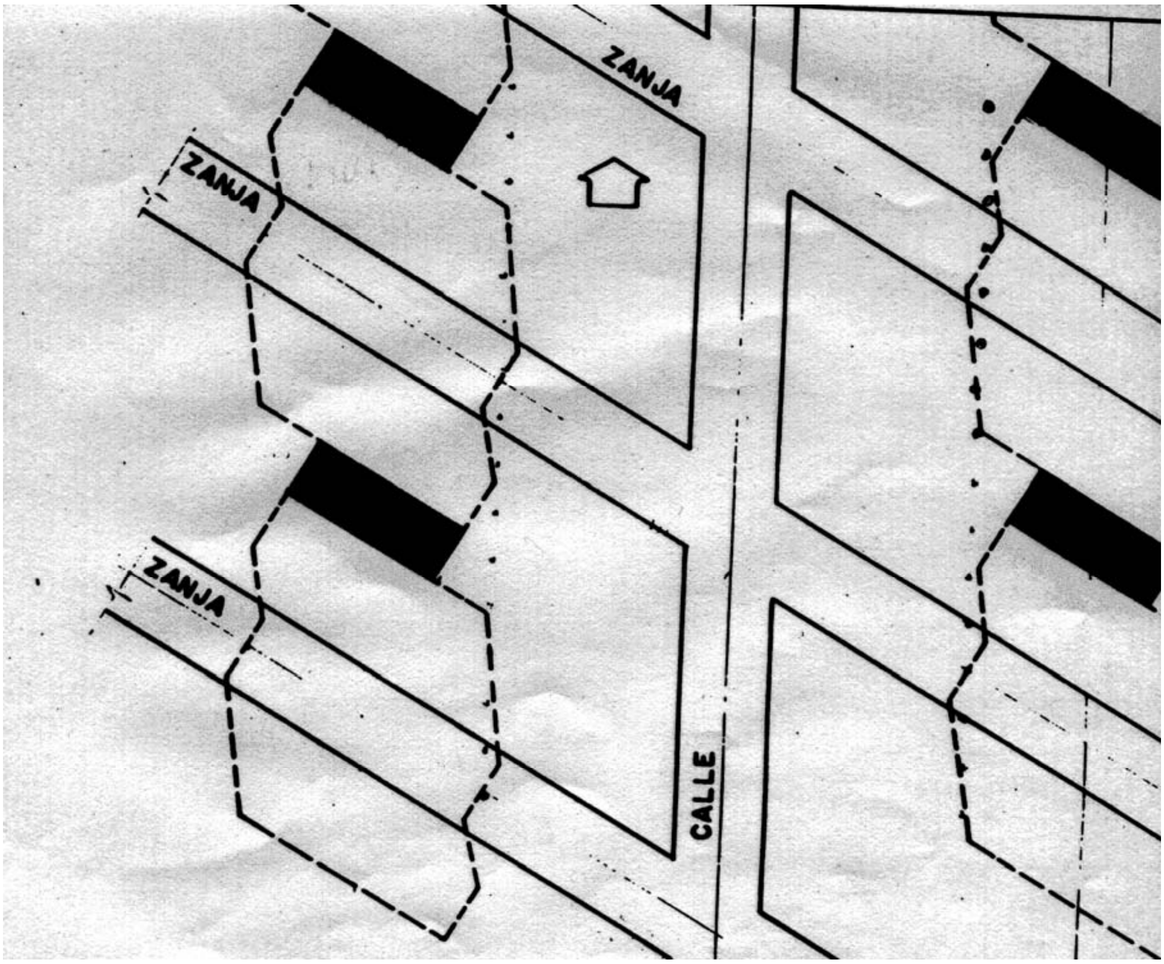


Parallelogram with drawpoints, drawbell drift and drawbells on same line at 60° to production drift. Developed at Teniente Mine, Chile - known as the **Teniente layout.**

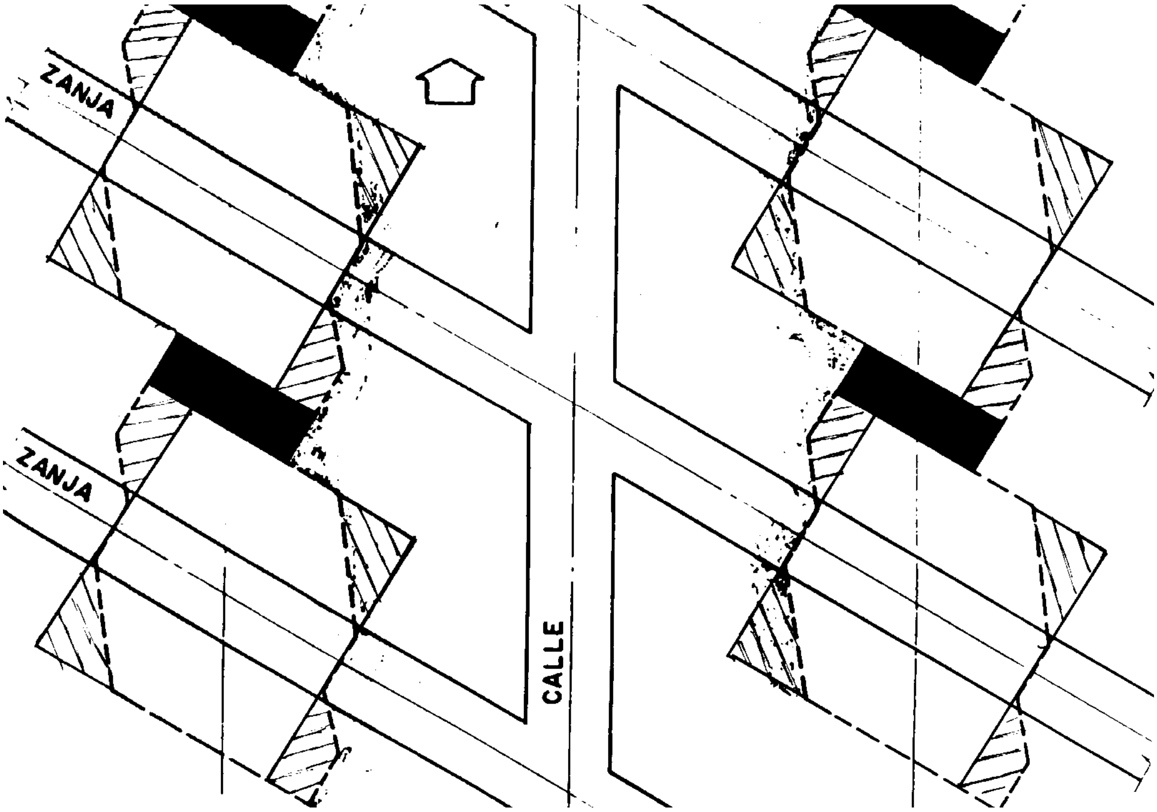
The following drawing shows the original LHD layout with square drawbells at right angles to the drawpoint drift. This layout had many advantages in that it is basically a strong structure, the brow is at right angles to the drawpoint drift, the drawzones are opposite each other, the square shape allows for the maximum recovery of material in the drawbell. When Teniente increased the drawpoint spacing they had some problems in breaking the undercut over the narrow minor apex.



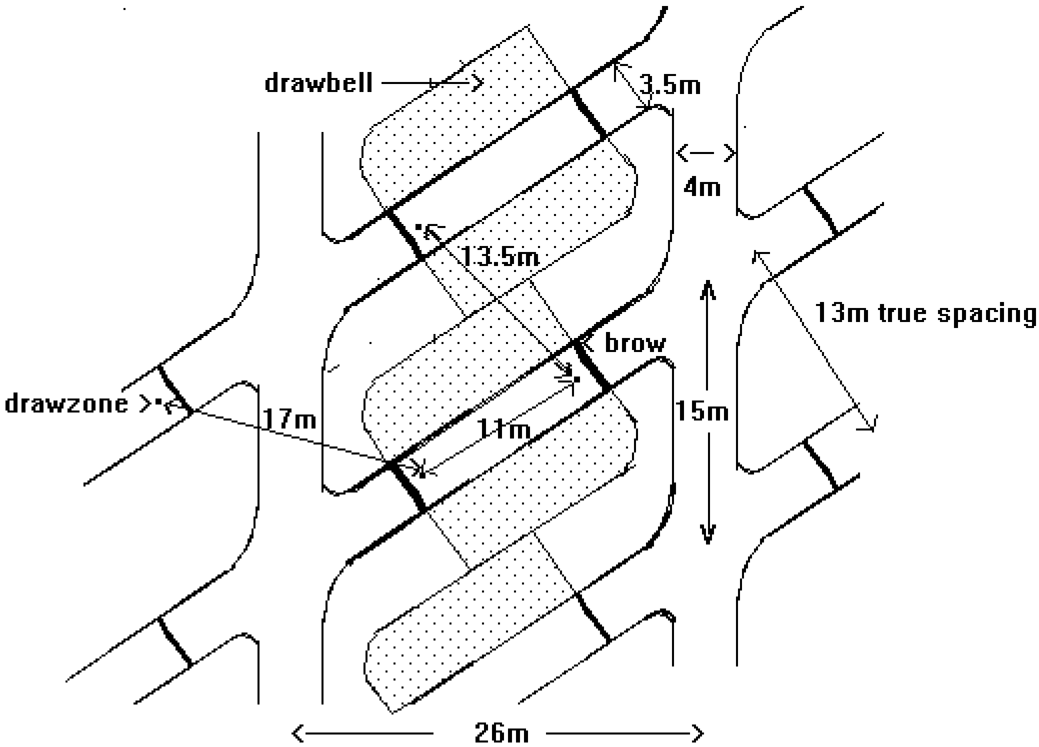
The following drawing shows a revised layout where the length of intersection of the minor apex has been increased to reduce undercut breaking problems, however this will result in a reduction in strength and there is a lot of wasted excavation which does not contribute to good ore flow. Numerical modelling of the herringbone and the Teniente layouts showed that the Teniente layout was 'stronger'. This layout also has the practical advantage that the LHD can back into the opposite drawpoint for straight on loading when there is brow wear. The only real disadvantage is that it is not suitable for electric LHDs.



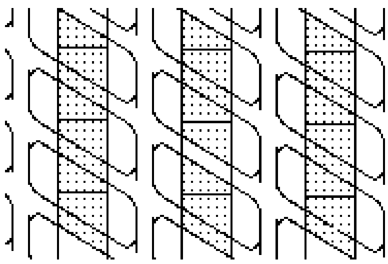
The following drawing shows the one drawbell shape superimposed on the other.



The following diagram shows the spacing of the drawzones in a ‘Teniente’ type layout



Drawpoints and drawbell drifts on same line at 60° to production drift. but, drawbells at right angles to production drift. Developed at Henderson Mine USA.



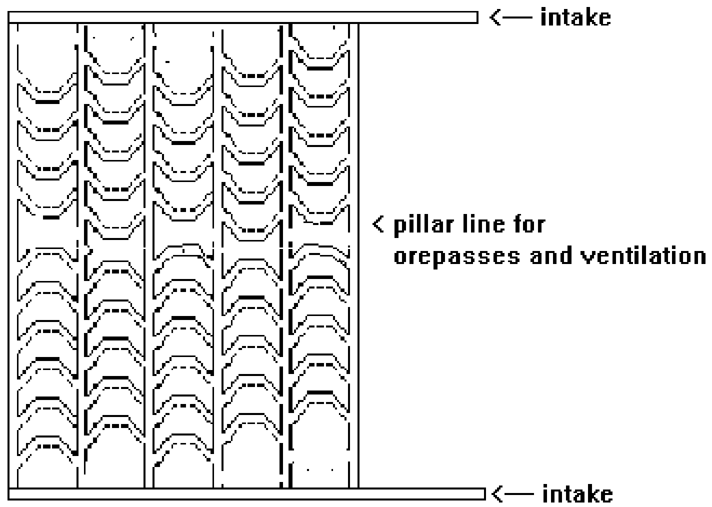
Common cone used in the past at Andina Mine, Chile.

Individual cones at Andina Mine, Chile

Individual Inclined Cones at El Salvador Mine, Chile

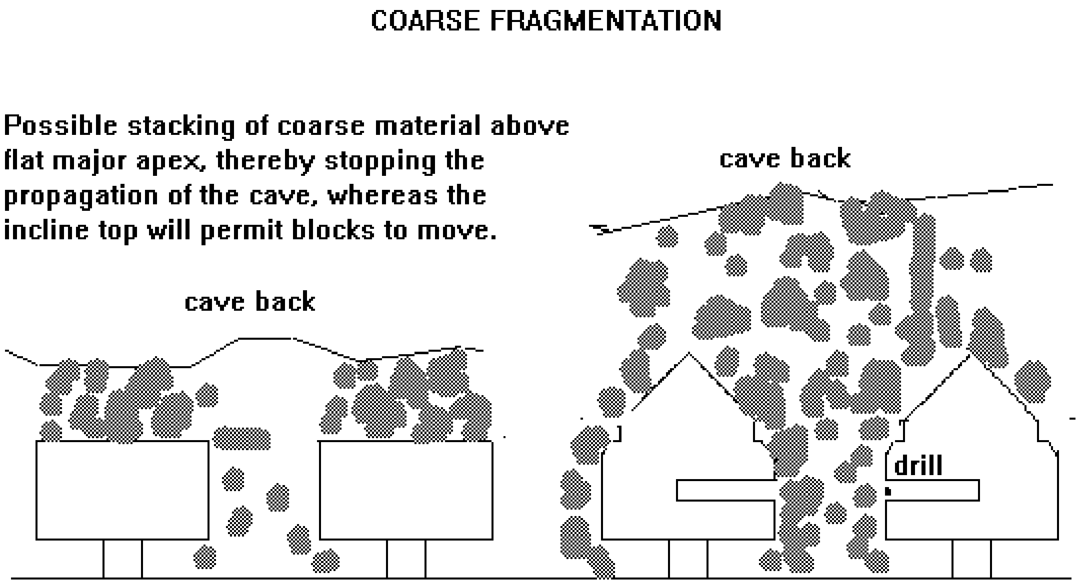
Mirror image of herringbone layouts 2.0 and 3.0. Northparkes Mine are using this layout but, with no internal orepasses and LHD haul to external crushers.   The mirror image layout is suitable for very steep dipping orebodies up to 200m wide.

The following diagram is a mirror image herringbone layout suitable for narrow orebodies and has a major advantage that the orepass and ventilation systems can be located in the centre of the orebody and out of the induced stress zone. Haul distances are reduced for high LHD efficiencies.

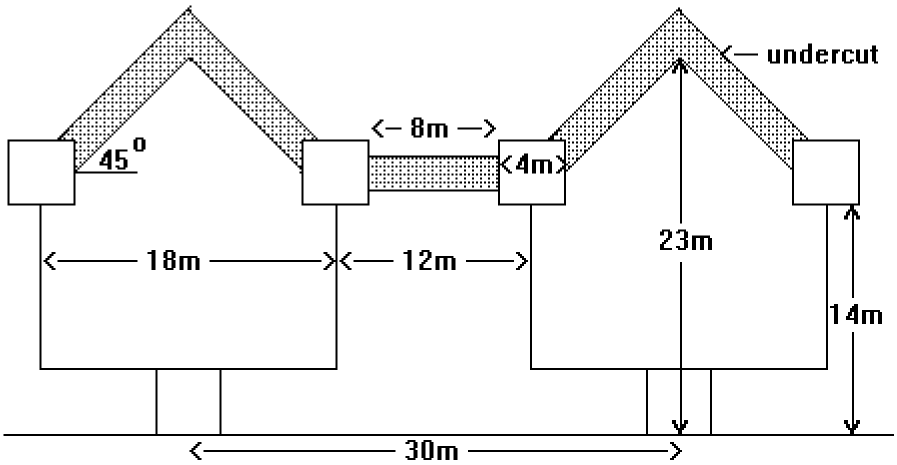


SHAPE OF THE MAJOR APEX

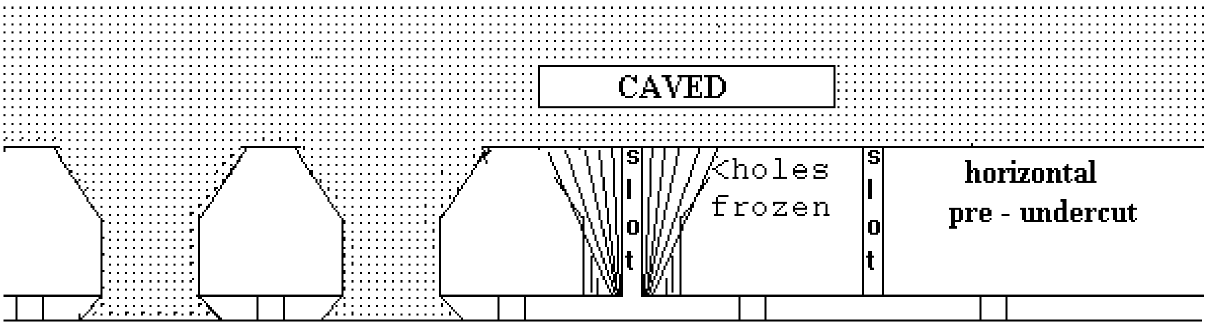
A factor that needs to be resolved is the shape of the major apex/pillar. A shaped pillar will improve the recovery of ore and will allow movement of undercut material. A flat top to the major apex is convenient in the undercutting process and serves a very useful purpose if excessive wear of the major apex is expected. However, if the fragmentation is coarse and has high friction then stacking could occur on the pillar and this would inhibit the propagation of the cave. Shaped major apex should allow for a better flow of material and this becomes critical as drawpoint spacings are increased and high friction material becomes coarser..



The incline advance undercut shapes the major apex from the undercut level.as shown in the following diagram



A technique is being used at Teniente Mine to shape the major apex by using inclined rings in the drawbell. This might be successful if there is no load on the toes of the holes, which would be the case with an advance undercut with a limited lead. However, if this technique were used with a pre-undercut and caving had occurred then there would weight on the toes and these choke conditions could prevent effective breaking and an overhang could form. There is a need to examine the blasting pattern.

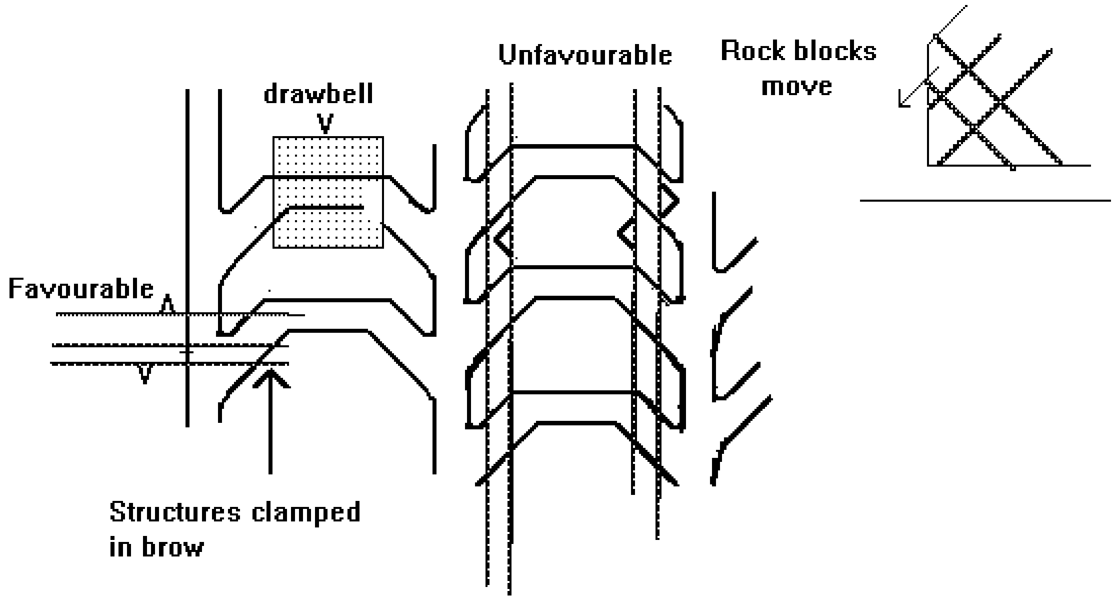


drill pattern

The shape of the major apex could be checked by drilling form the production drift through the major apex.

ORIENTATION OF DRAWPOINT AND PRODUCTION DRIFT

A decision has to be made whether the orientation with respect to structures of the production drift is more important that the orientation of the drawpoint.



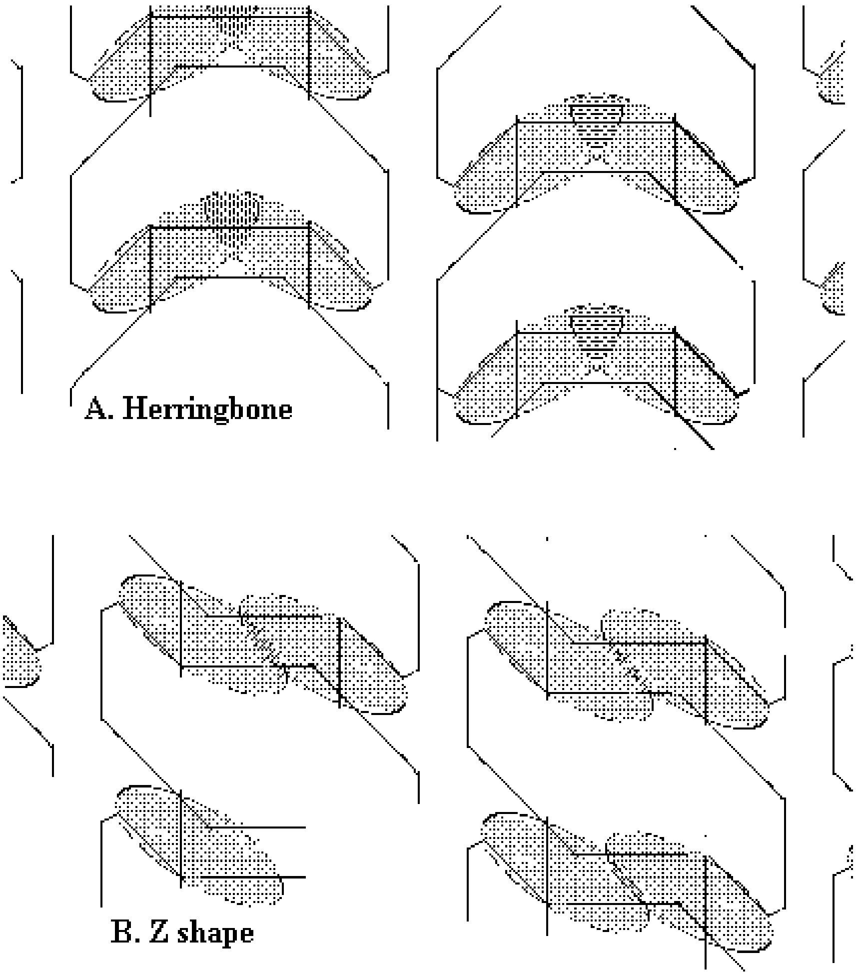
The tendency is to assess the structures and orientate the production drift in the strongest direction. **This approach is not correct.** The most important and the most vulnerable area is the drawpoint brow. The brow is a free face and therefore failure can occur at relative low stresses. It is a difficult face to support as it is in a relaxation mode and it is not possible to apply lateral restraint to the face in terms of plates or straps nor is it possible to use a lining. Abutment stresses, heavy drawbell blasting or undercut blasting result in a loosening of the rock mass so everything must be done to protect it by reducing abutment stress effects and to orientating it so structures are clamped in the brow and wedge failure cannot occur. The above figure shows the correct orientation with respect to structures, whereas the production drift is in solid rock and can be supported. A 3-D polystyrene model of the brow with the structures shown as planes conveys the right impression of the support requirements and potential hazards.

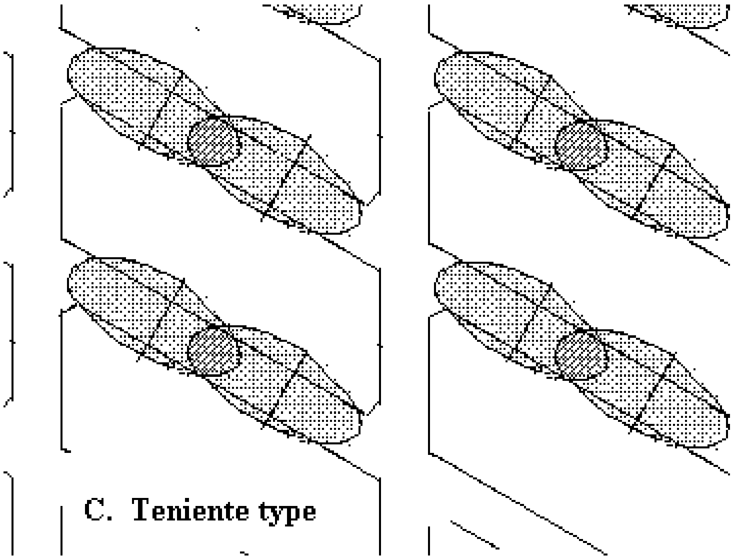
ORIENTATION OF DRAWBELL RELATIVE TO DRAWPOINT

There is a suggestion that the drawzone has an oval shape with it’s long axis orientated on the line of the drawpoint. The following observations should be noted:

- In a herringbone layout - figure A -the drawbell is at an angle to the drawpoint and the oval drawzones intersect over the minor apex.
- In the case of the Z shape - figure B - the drawzones do not intersect but are contiguous and form a wide zone in the centre of the drawbell. Certainly an interesting point and worthwhile following up.

- In the Teniente layout the drawzone intersect in the centre of the drawbell and that this will mean a better flow of ore and less hangups than when the drawpoint and drawbell are at an angle as in the herringbone layout. The concept that the drawzone has an oval shape on the line of the drawpoint could be tested in a 3D sand model and the merits of the layouts examined. **The potential benefits of the Z layout need to be examined.**





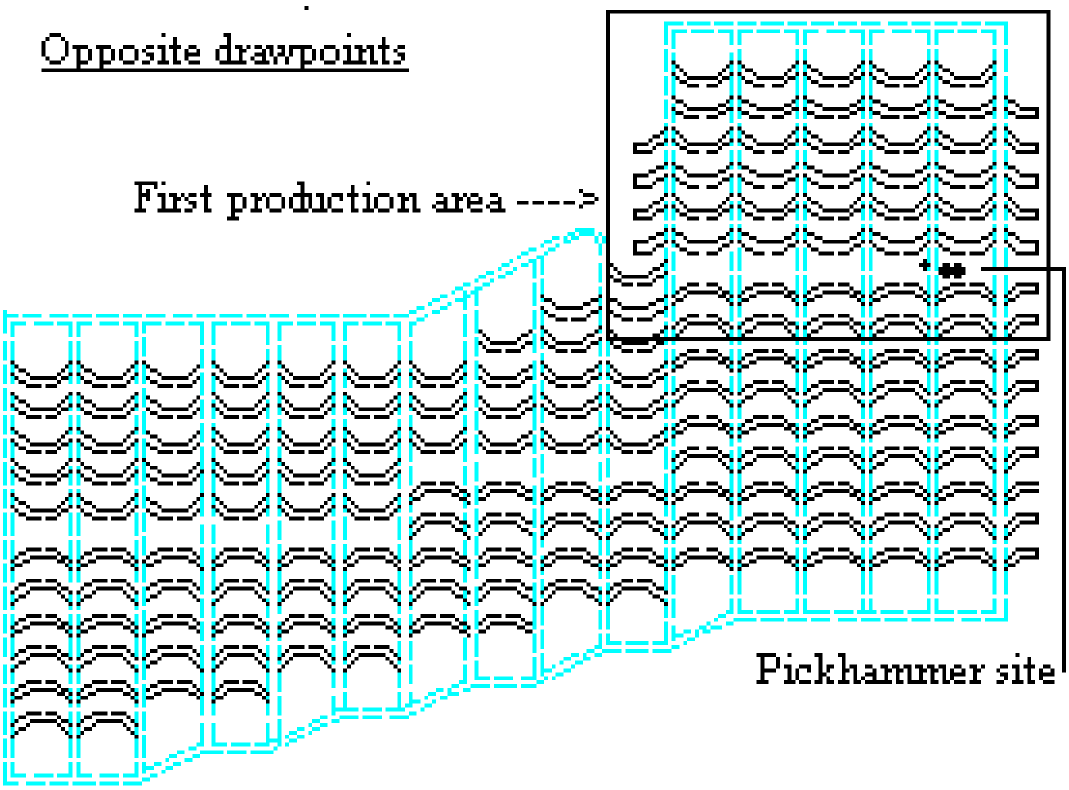
ORE HANDLING AND THE INFLUENCE ON THE LAYOUT

Ore handling plays a very important role in selecting a layout and with the caving of coarse fragmented orebodies and mining in high stress areas modifications to current layouts have been made. Various alternatives are discussed in the ore handling section.

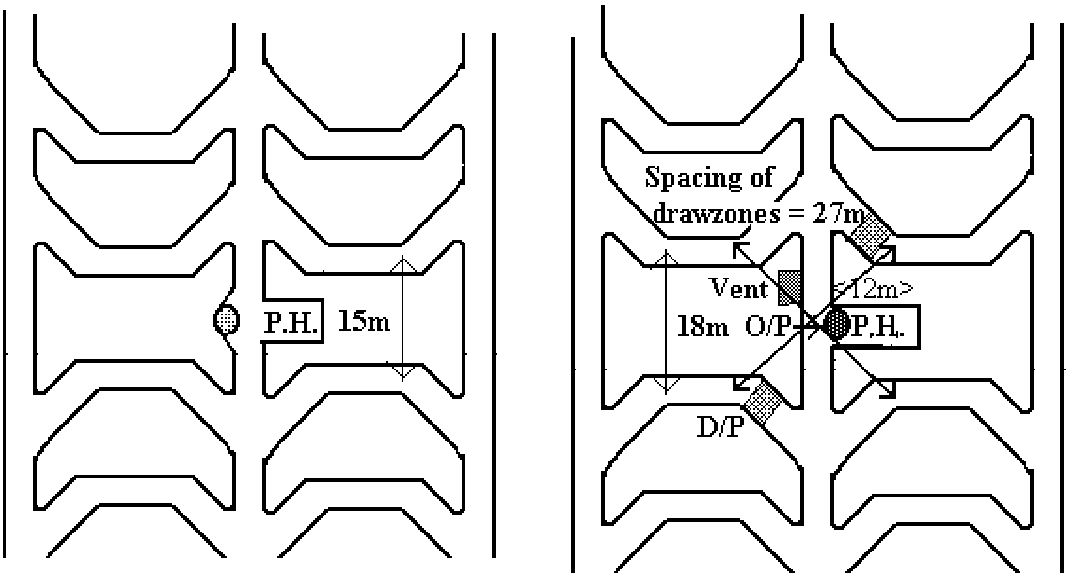
For example, there are various pickhammer / orepass combinations to consider if that is the route that is preferred. On some of the chrysotile asbestos mines large rocks in the 2m³ to 6m³ category are fairly common. These rocks contain weaknesses in the form of fibre seams or picrolite veins and are therefore amenable to secondary breaking by pickhammers were looked at.

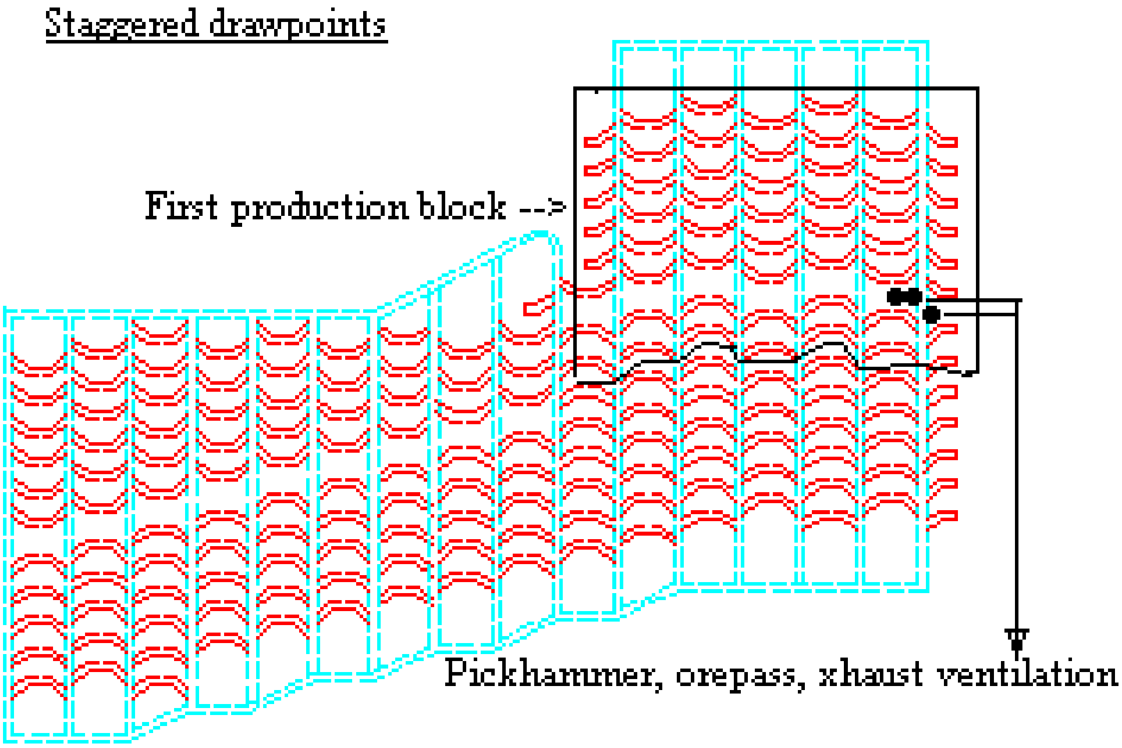
This section is discussion on how a layout is designed according to orebody geometry and ore characteristics. A pickhammer layout on the north and south side of the orebody had been considered. The problem here is that on the north side the installations could be affected by massive wedge failures and on the south side they would be in the abutment of the cave with the steep pit wall to the south. A mirror image layout was looked at with pickhammer, orepass and exhaust ventilation raise in the central pillar. To improve the stability the drawpoint spacing in the pillar would be increased to 20m. The mirror image has many advantages and actually results in a major saving in capital expenditure by reducing the footage of railroad haulage and the number of pickhammer sites. Extra exhaust ventilation development is required, but, the savings are far greater

The following two drawings show the opposite and staggered mirror image herringbone layouts. By locating the pickhammer in the pillar the orepass is to the side of the drift and does not interfere with access. The ventilation raise is located in the pillar in the opposite wall. These layouts do require a re-think on the current ventilation layout by having to put in a ventilation level some 15m below, but, this is more than compensated for by the saving in railroad development and equipping.

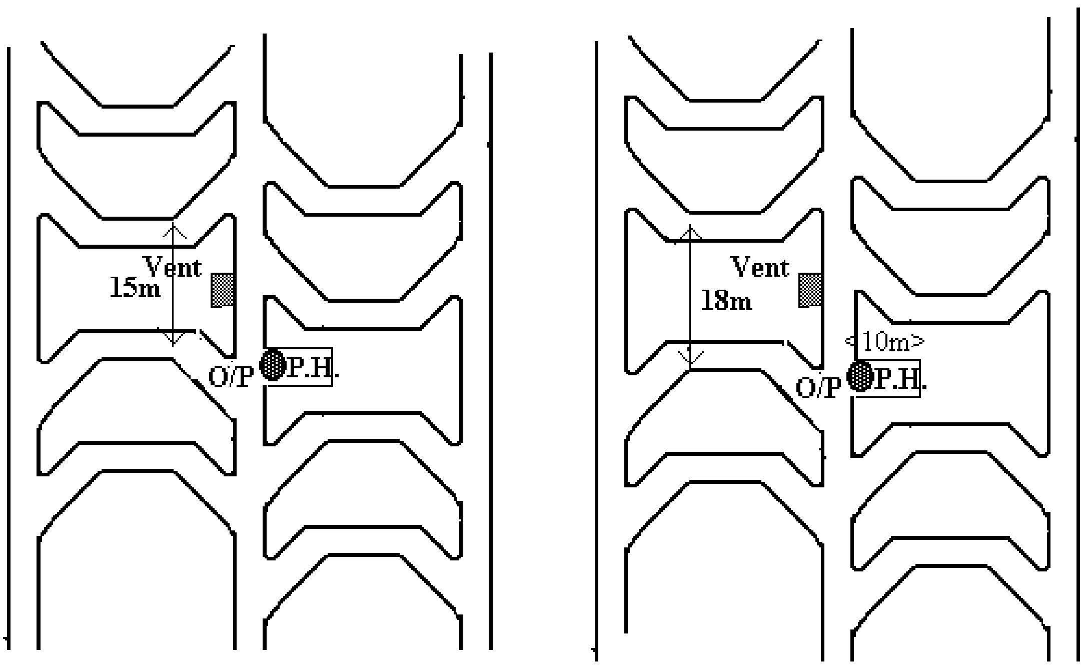


The details of the opposite mirror image herringbone layout is shown in the following drawing:

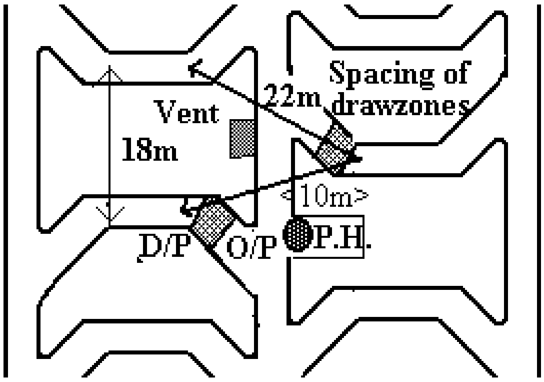




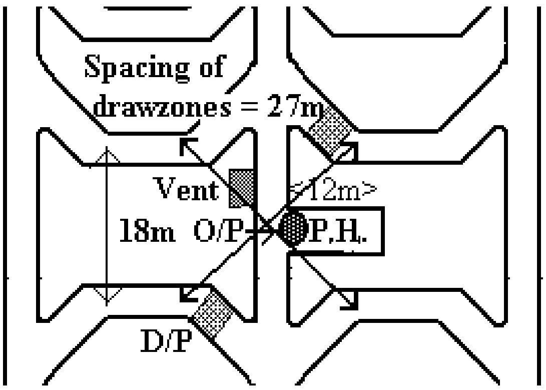
The details of the staggered mirror image herringbone layout are shown in the following drawing:



The staggered layout is better in terms of pillar location with respect to the shape of the orebody and the offset of the pickhammer and the ventilation raise results in a stronger layout.



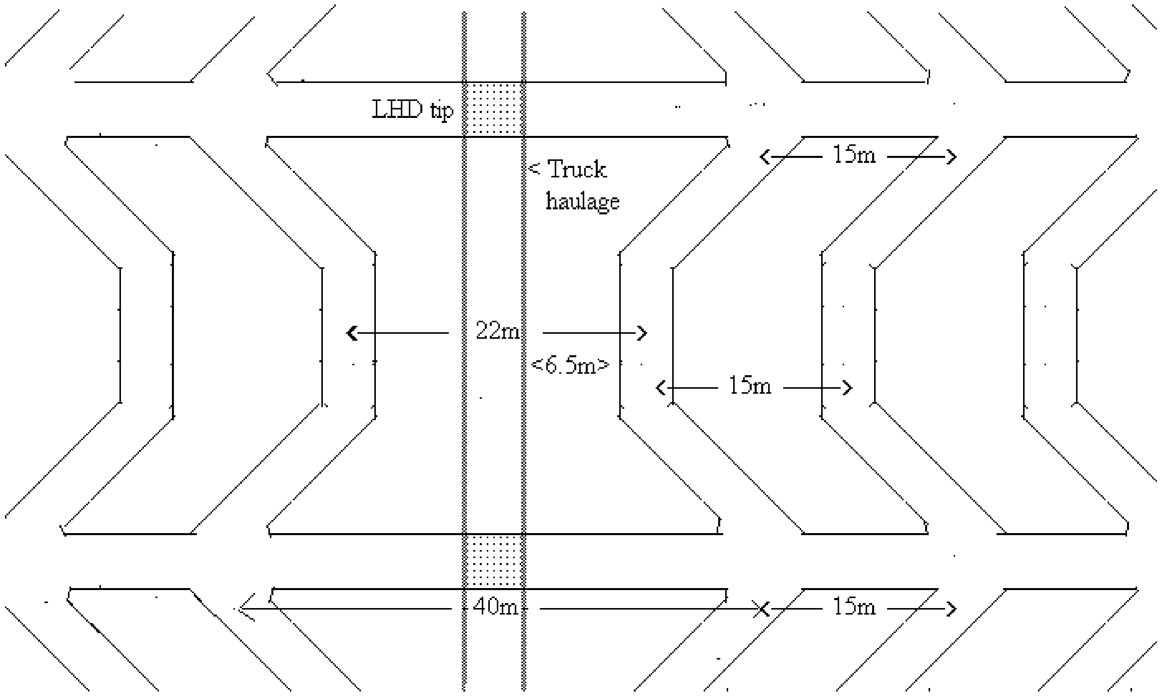
Another factor in favour of the staggered layout is that the drawzone spacing diagonally across the major apex is 22m compared to 27m with the opposite layout.

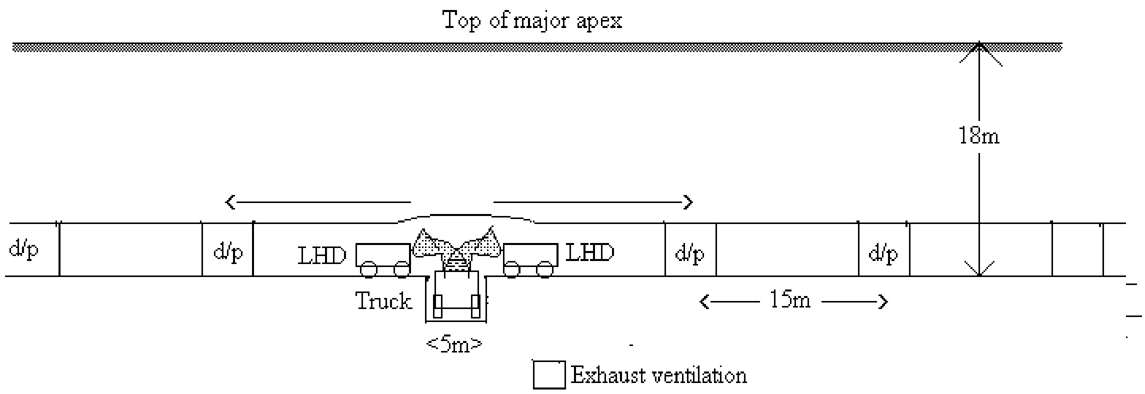


MIRROR IMAGE TRUCK HAULAGE

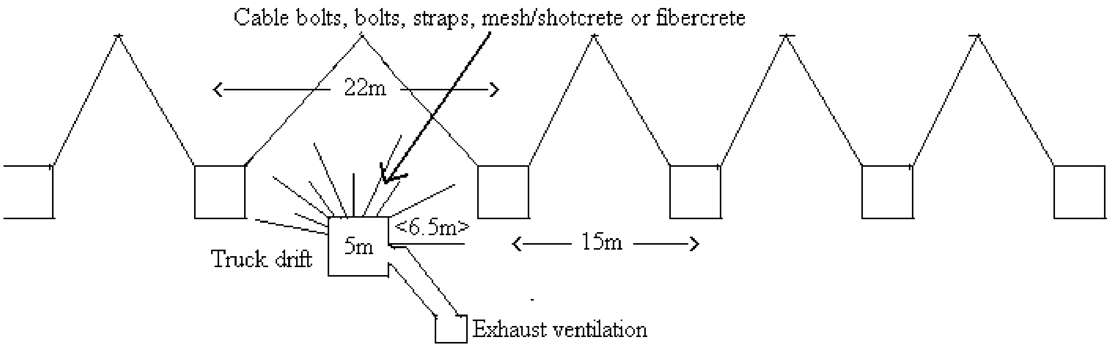
The mirror image layout can be used for various layouts if the object is to locate the infrastructure in the centre of the orebody. The concept here is that a central pillar does not present a problem, provided there is always continuous ‘uniform draw either side of the pillar thus avoiding any chance of column loading. The side of the pillar is a boundary zone and drawzone interaction will occur a short height above the pillar. The following drawings are examples of using trucks to remove the ore. This concept is based on the fact that the LHD is designed for loading and dumping and the truck for hauling. A large fleet of trucks is required, but the logistics can be solved with automation.

PLAN OF CENTRAL TRUCK HAULAGE AND LHD TIP





SECTION ALONG PRODUCTION DRIFT

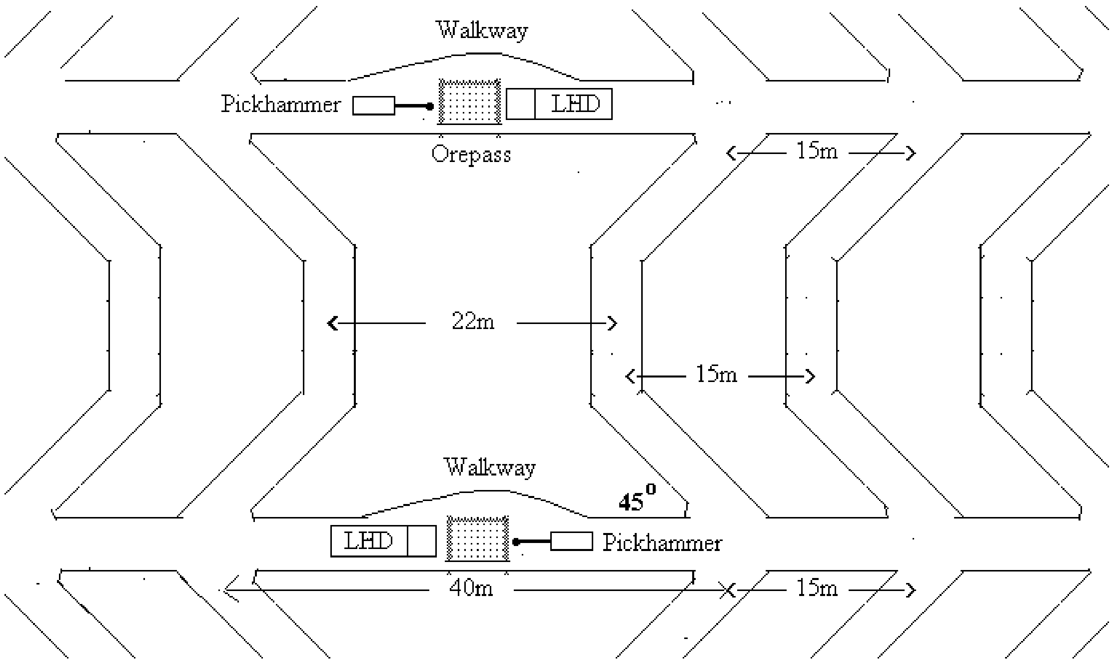


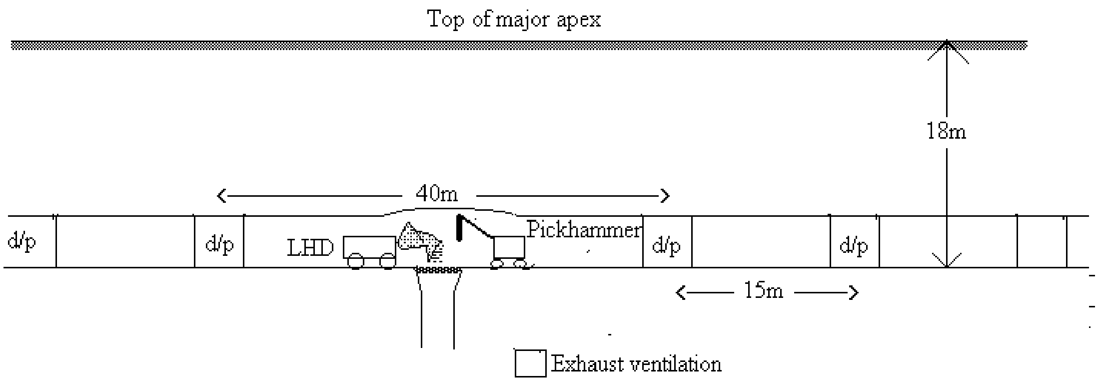
SECTION THROUGH CENTRE OF DRAWBELLS

Mobile Pickhammer

A mobile pickhammer operating on the central orepass would have several advantages in that use is made of the drift and it is not necessary to make a pickhammer excavation in the pillar. This means that the pillar can be reduced to have a 60' / 18m drawzone spacing. The following diagrams show the layout with a 22m spacing! The LHD would operate on the north side of the orepass for two shifts and then change over to load on the other side for one shift. If the orepass is recessed into the side of the pillar the drift is available for through access.

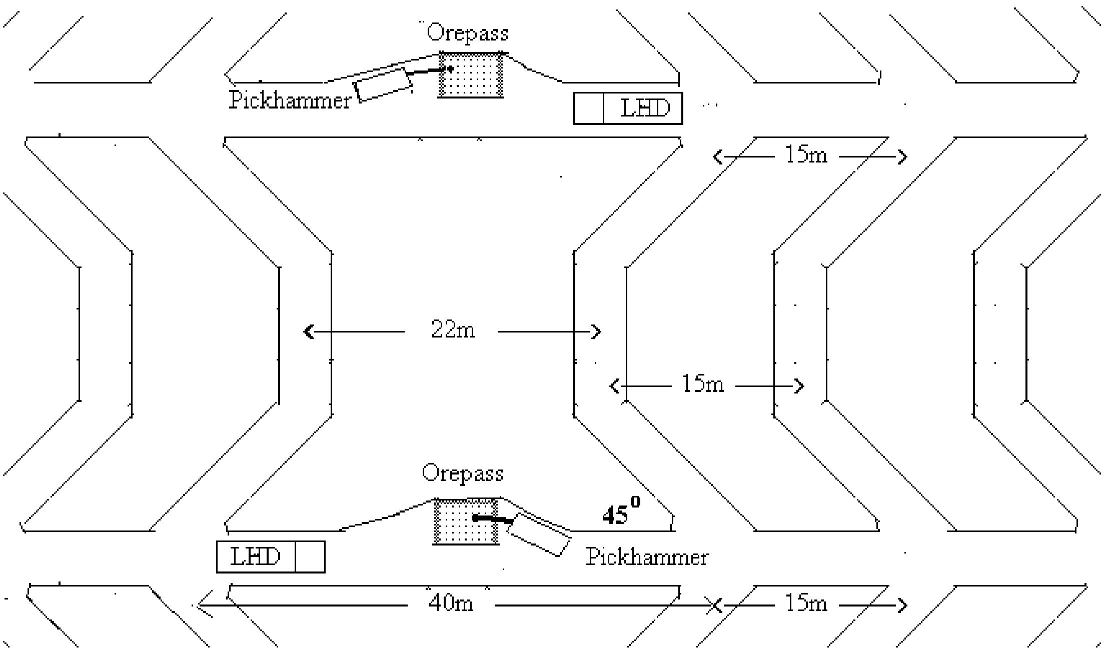
PLAN OF CENTRAL OREPASS WITH MOBILE PICKHAMMER AND LHD TIP





SECTION ALONG PRODUCTION DRIFT

PLAN OF OREPASS IN PILLAR WITH MOBILE PICKHAMMER AND LHD TIP



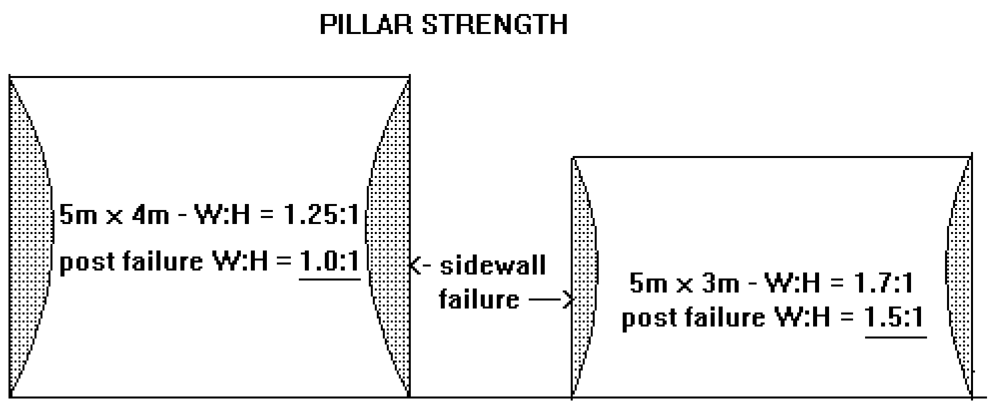
OREBODY GEOMETRY, STRESSES AND IRMR

The orebody geometry influences the selection of layouts. When the orebody dips at between 40°and 60° incline drawpoint layouts could be considered, particularly if there are high draw heights. Irregular shapes might call for closer drawpoint spacing so as to ensure greater draw control accuracy in limiting the amount of dilution.

The effect of regional stress on the selection of a layout is minimal except in the case of a narrow orebody where there could be advantages in utilising clamping stresses.

The effect of the structures on the drifts can be catered for with good support. Intersecting structures in the junction can create problems and need to be well supported prior to cutting the drawbell. The bullnose and camelback are also potentially weak areas which must be assessed in terms of structural failure.

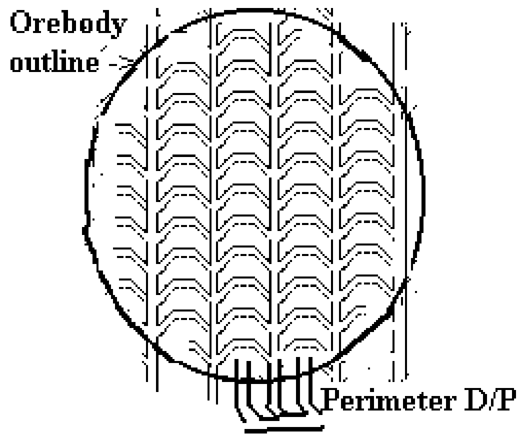
Sidewall failure as a result of high abutment stresses when conventional undercutting techniques are used is common on some mines. In case such as this if the drift height is reduced then the pillar strength is increased.



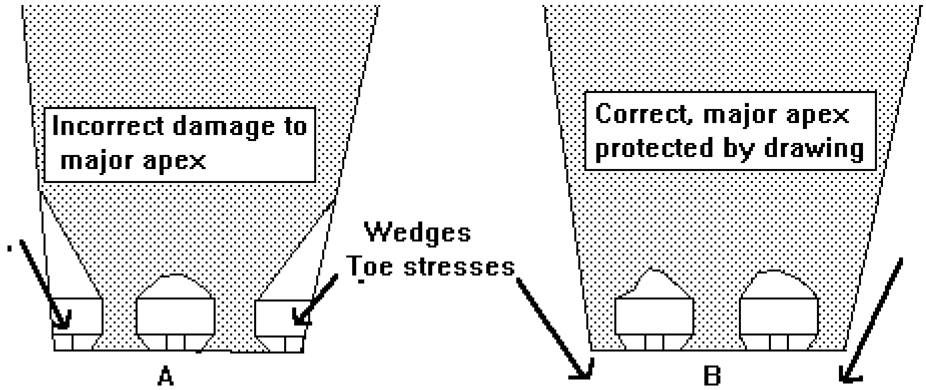
The pillar strength is increased by reducing the height of the drift/stope

PERIMETER DRAWPOINTS

In pipe like orebodies consideration can be given to using the undercut drifts as drawpoints, particularly if the country rock is stronger than the orebody. This means that the length of the drawpoint is not critical and therefore large LHD's can be used. The significant points are that the annular ring contains a high percentage of the ore. For example, if the orebody were 200m in diameter then a 15m annular ring would be 17% of the ore in a zone where ore recovery is difficult on the horizontal production level owing to the irregular spacing of the horizontal drawpoints with respect to the contact - see figure

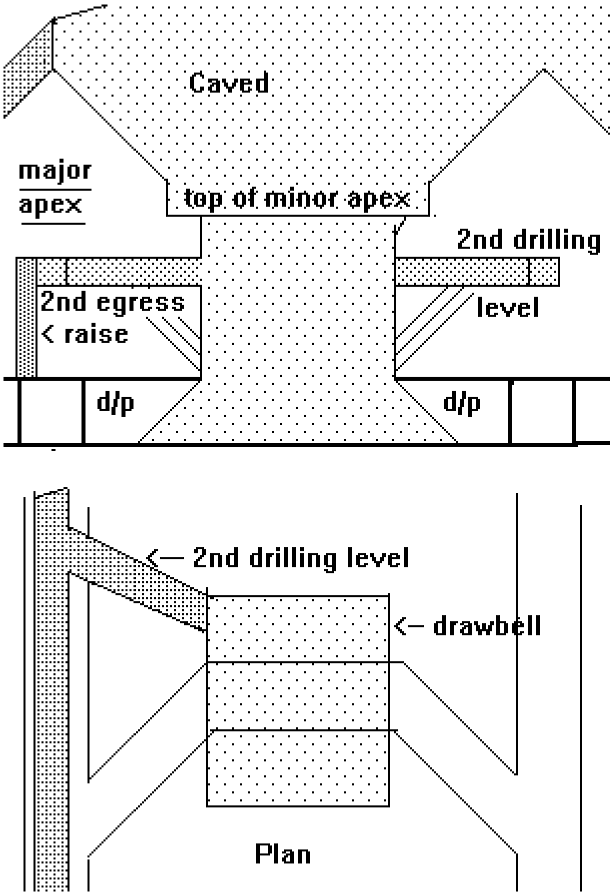


Another area that needs attention is perimeter of a horizontal layout where drawpoints should face outwards so that they draw into caved ground and not that the major apex can be loaded by toe stresses or wedges as shown in the following diagram.



MODIFICATIONS FOR SECONDARY BLASTING

A secondary drilling level in the major apex, located 4 - 5m above the brow and angled to intersect the corner of the drawbell will provide an ideal safe site for drilling of hangups



There is a reluctance among the mining community to accept this concept or even to try it out in suitable areas. .These negative attitudes have to change. The immediate response is that the major apex will be weakened, well , the weakest part of the major apex is at the base on the production level, where the pillars only represent 50% of hte area. The secondary blasting drift is not over the drawpoint but, in the corner of the drawbell. Another argument is that the drift will always be full of fines, well the reason for having this breaking level is because of the preponderance of large rocks and not fines, also, the fines move downwards not sideways.

DRAWPOINT PRODUCTION POENTIAL

The drawpoint production potential is the sustainable average tonnage per day that can be produced from a drawpoint for a defined period or draw height, this must be related to changes in fragmentation. The production potential is a function of :

- fragmentation,
- type of material is it free flowing or sticky,
- size of drawpoint which is also relates to the size of LHD,
- size of grizzly at the tip,
- distance to the tip,

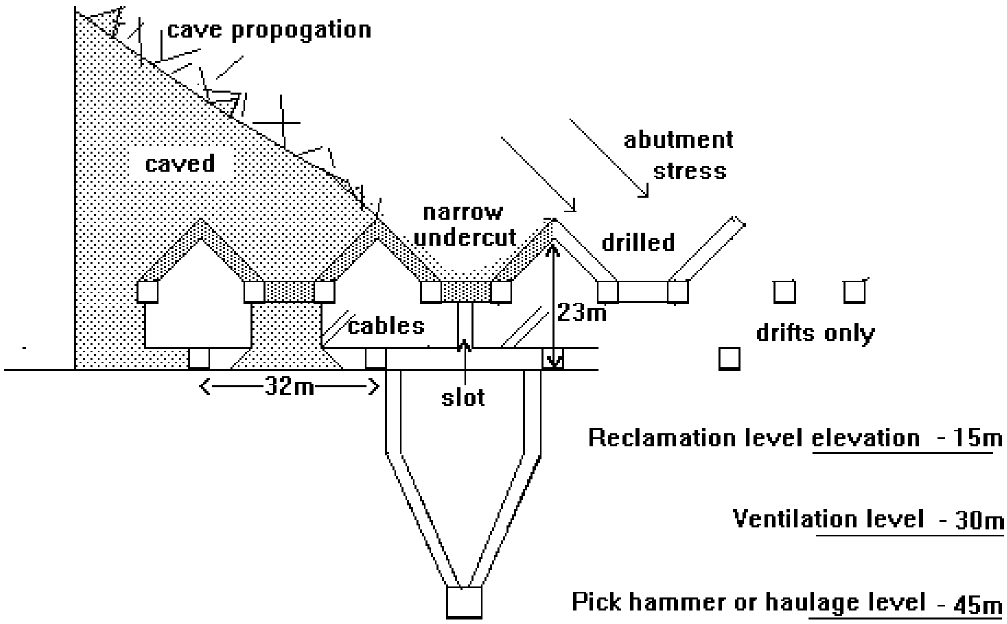
- secondary breaking procedure.

NUMBER OF DRAWPOINTS REQUIRED

The number of drawpoints required is based on production requirements, drawpoint production potential and drawpoint availability. Drawpoint availability is a function of fragmentation in terms of the time required to bring down hangups and maintenance of the drawpoint due to collapses or failure of support. The objective is to reduce drawpoint failure by better undercutting techniques and to reduce delays in bringing down hangups by having the right equipment on site.

RECLAMATION LEVELS

The infrastructure required for a production is a high capital cost item and therefore the tendency is to design for high draw columns, particularly where there is coarse fragmentation as high draws lead to better fragmentation. In the situation where the draw column is high and there is doubt whether the extraction level will survive the full draw height provision can be made in advance for a reclamation level by lowering the ventilation level etc. 15m so that the reclamation level would be 15m below the current level and developed in a destressed environment.



Advance undercut with narrow incline and horizontal undercut, drawpoints and drawbell only developed after undercut has passed over so as to eliminate abutment stress damage. Cables installed from drawbell drift. Undercut holes angled forward, only limited swell relief with muck handled through orepasses in good ground at a slot site.
Major apex = 23m, minor apex = 12m

LHD SPECIFICATIONS

Type of LHD	Length (L) mm	Width (W) Bucket mm	Turning Radius mm	Tramming Capacity (TC) kg	Ratio TC/W	TC/L
Elphinstone						
1500	9195	2482	6400	9000	3.6	1.0
1700	10640	2720	6680	12000	4.7	1.1
2800	10697	3048	7390	16200	5.7	1.5
Wagner						
HST-1A	5283	1219	3505	1361	1.1	0.3
ST-2D	6593	1651	4700	3629	2.2	0.6
ST - 3.5	8223	1956	5465	6000	3.1	0.7
ST-1000	8530	2040	5800	10000	4.9	1.
ST-6C	9490	2610	6320	9525	3.6	1.0
ST-7.5Z	9800	2590		12272	4.7	1.2
ST-8B	10287	2769	7010	13608	4.9	1.3
ST-15Z	12396	3404	8443	20412	6.0	1.6
Eimco (Obsolete see Tamrock Below)						
925	8800	2300	6200	8160	3.8	0.9
Toro (Obsolete see Tamrock below)						
300D	8520	1900	5715	6200	3.3	
501DL	10530	3040	7230	14000	5.2	1.3
Tamrock						
Micro-100	4597	1050	3191	1000	1.0	0.2
EJC 61	5486	1448	3734	2727	1.9	0.5
TORO 151	6970	1480	4730	3500	2.4	0.5
EJC 100 D	7341	1702	5004	4540	2.7	0.6
EJC 130 D	8407	1930	5511	5897	3.1	0.7
TORO 301	8620	2100	5780	6200	3.0	0.7
EJC 210 D	9957	2718	6553	9545	3.5	1.0
TORO 400	9252	2440	6590	9600	3.9	1.0
TORO 450	10003	2700	6537	12000	4.4	1.2
TORO 1250	10508	2700	6672	12500	4.6	1.2
TORO 1400	10508	2700	6887	14000	5.2	1.3
TORO 650	11410	3000	7180	15000	5.0	1.3
TORO 2500E	14011	3900	9440	25000	6.4	1.8
E.M. France Loader						
CTX 6	8375	2550	6080	8500	3.3	1.0
CTX 6S	8900	2550	6080	9500	3.7	1.1

SELECTION OF LHD, SIZE, POWER

The size of the LHD is a function of the permissible layout. It is not based on the independent wishes of management. Where the geotechnical and draw control issues dictate small openings and close drawpoint spacing, it is often found that people persevere with large machines. If a smaller machine were used with close orepass spacing then a better production would have been achieved as there would not have been the long tram and in particular the support problems and high dilution and ore losses. A case in point is a mine in USA where 3.5yd machines average 2200 tpd.- excellent, this is owing to return trips of 45m.

The power is either diesel or electric. It is interesting to note that there are still problems with trailing cables. Decisions will need to be based on these experiences rather than on current successful operations. There are also conflicting assessments in the selection of power. A large mine did a comprehensive study and decided that electric power was best particularly for improved ventilation in the hot environment, however with a change in staff, the decision was reversed.

DIESEL VERSUS ELECTRIC LHD'S

Comments from Northparkes mine are attached:

Diesel or Electric?

A comparison of diesel and electric loader types is given below:

Diesel Loaders	Electric Loaders
Operational Flexibility	
The great advantage of diesel units is their increased flexibility of movement. This not only results in quicker relocation between the production drives, but also enables the unit to be used for other miscellaneous jobs such as spillage clean-up or sump mucking.	Electric loaders are largely tied to the production area. This limits their use and causes delays when access is required to the workplace. An advantage of this is that the tendency to divert production units away from production duties can be easily overcome.
Diesel units have the full flexibility to tram to the current 9 800 level workshop for major services and repairs, negating the need for additional maintenance facilities at Lift 2.	Though more limited, electric units can be trammed to the existing workshop using generator sets mounted on a vehicle travelling in tandem with the loader. Basic servicing facilities would be required at Lift 2.
Secondary Breaking	
Diesel units allow secondary breaking activity to take place behind the operating machine (on the non-crusher side) without unduly impeding either production or breaking activity.	Operation of secondary breaking activity in the same drive as a production loader would require a mid drive anchor point and connection cable slung along the back or shoulder of the drive. Care would have to be taken with secondary breaking adjacent to the electric cable.
Ventilation	
Though requiring greater airflow volumes, sufficient ventilation exists through the drives as the volume must support diesel secondary breaking activity.	Electric loaders can operate under minimal ventilation volumes
Automation	
Automation of diesel units is viable. No direct connection can be made between electronic safety barriers and the unit's power source raising some safety concerns.	Electric loaders can be automated and a direct connection made between electronic safety barriers and the unit's power source enabling shutdown in emergency situations.

around 84% of current running costs are attributable to cabling ,cable reeling

COMMENTS BY N.J.W.BELL ON LHD PERFORMANCE

Brow stability

In block 58 at Shabanie it was found with production that the north facing drawpoints eroded back tremendously whereas the southern drawpoints were stable and did not wear back. Investigations indicate that this was predictable and owing to the structures. It might well be that if we had considered the data differently that we would have used a different configuration for block 58 collection X/Cuts as opposed to the drives selected. We went for stability the development of draw points, across the major features and structures.

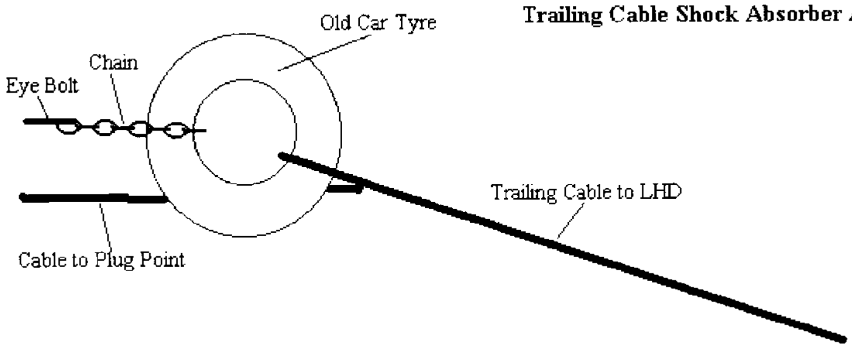
Electric LHDs - Trailing Cables

General

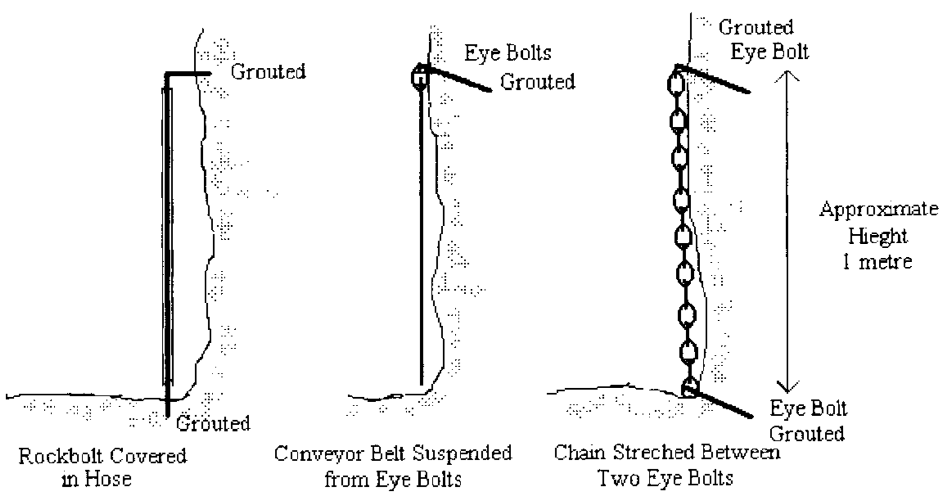
Trailing cables are anchored near the plug in point using old car tyres as a shock absorber to reduce the damage on the cable owing to shock.

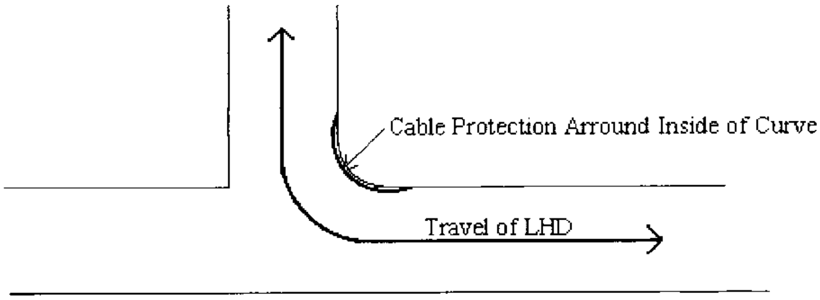
The electric trailing cable is protected from wear on the corners around which it has to run when loading. This is done using either conveyor belt hung from chains on the sidewalls or with bent rockbolts covered with 1" hose or lengths of chain suspended against the sidewall.

Trailing Cable Shock Absorber Anchor



LHD Trailing Cable Protection on Corners





Electric LHD Trailing Cables

Cable Life

Electric trailing cables from Gaths Mine, King Section have an average operating life (between repairs) of 375 hours (range from 148 hours to 738 hours). If seven joints total life +\- 2,600 hours or 1 year on a 2 x 8 hour shift/day operation.

Life is dependent on the condition of the:

i. Area – corner protection, water on the floor and spillage.

ii. LHD cable reel mechanism and cable guide rollers.

Cables are rejected when they have approximately seven joints in them (not a hard and fast rule, but an indicator).

Cable Repair Equipment Required

1. Surge generator
2. Insulation resistance tester
3. Vulcanising machine
4. Trimming knife
5. Linen tape
6. Vulcanising rubber
7. Emery paper
8. Binding copper wire
9. Combination pliers
10. Side cutters
11. Hacksaw

Procedure

1. The cable is first checked on site using an insulation resistance tester. If it is established that one or more phases are down to earth, the cable is removed for repair in the cable bay.

2. In the cable bay the cable is spread out and cleaned.

3. The phases are checked for leaks again using the insulation resistance tester. If the reading is below 0.5 megaohms, then the phase is leaking.

4. After establishing the leaking phases, the leaking phase is connected to the surge generator. The surge generator is turned up gradually until a cracking sound is heard from the position where the cable is faulty.

5. Cut and open the faulty section and prepare the two ends for joining.

6. Do a married joint on the conductor. The joints should be staggered, insulated with linen tape and vulcanising tape. Ensure that the joint is flexible and of the same length as the vulcanising mould.

7. Vulcanise the joint for about 4 hours.

8. Test joint using the insulation resistance tester.

9. Store the cable in a dry place in neat loops until it is required for use.

LHD Selection

Electric or Diesel

The selection of whether the LHDs should be electric or diesel is primarily a consideration of cost (capital and working) and effectiveness for the duty. In general diesels are best suited to development where tramming distances tend to be greater and the machine often has to operate over a large area and also carry out transport duties for support and construction materials.

For production electric LHDs have the edge as they load better and have higher availability than their diesel equivalents and are cheaper, quieter and cooler to operate. Electric LHDs do have the major disadvantage relative to diesels, namely flexibility; they are anchored and not readily moved unless linked to a portable generator set, which the LHD tows.

LHD Size

The selection of the LHD size is an iterative process, starting with an ideal for the production rate required and then adapting to meet the considerations below.

From duties required, the specifications of LHDs and the ground conditions RMR and MRMR one can select the size of LHD that is practical. This is determined primarily by the size of unsupported area that can be safely exposed during the development phase and the potential for **F**alls **o**f **G**round. The legal requirements regarding tunnel dimensions and clearances must be met. Further the route, by which the equipment will be introduced into the mine, ramp or shaft has to be taken into account.

Once the LHD size has been determined the minimum design parameters become fixed. The following have been found to be reasonable :

- Tunnel Width = Machine Width + 1.5 m
- Tunnel Height = Machine Height + 1.3 m
- Drawpoint Length (brow to centre line of access) = Tunnel Height + Machine Length
- Radius of Curves (for reasonable tramming speeds) = 2.5 X (IR + OR)/2

From the data above once the angle of the drawpoint off the collection drift is determined so is the minimum width of the major apex.

The data discussed above are in the tabulation below for the different basic size LHDs and shown diagrammatically in the sketches for ‘small’ LHDs. Similar sketches need to be drawn for the LHD size selected.

2000/06/13

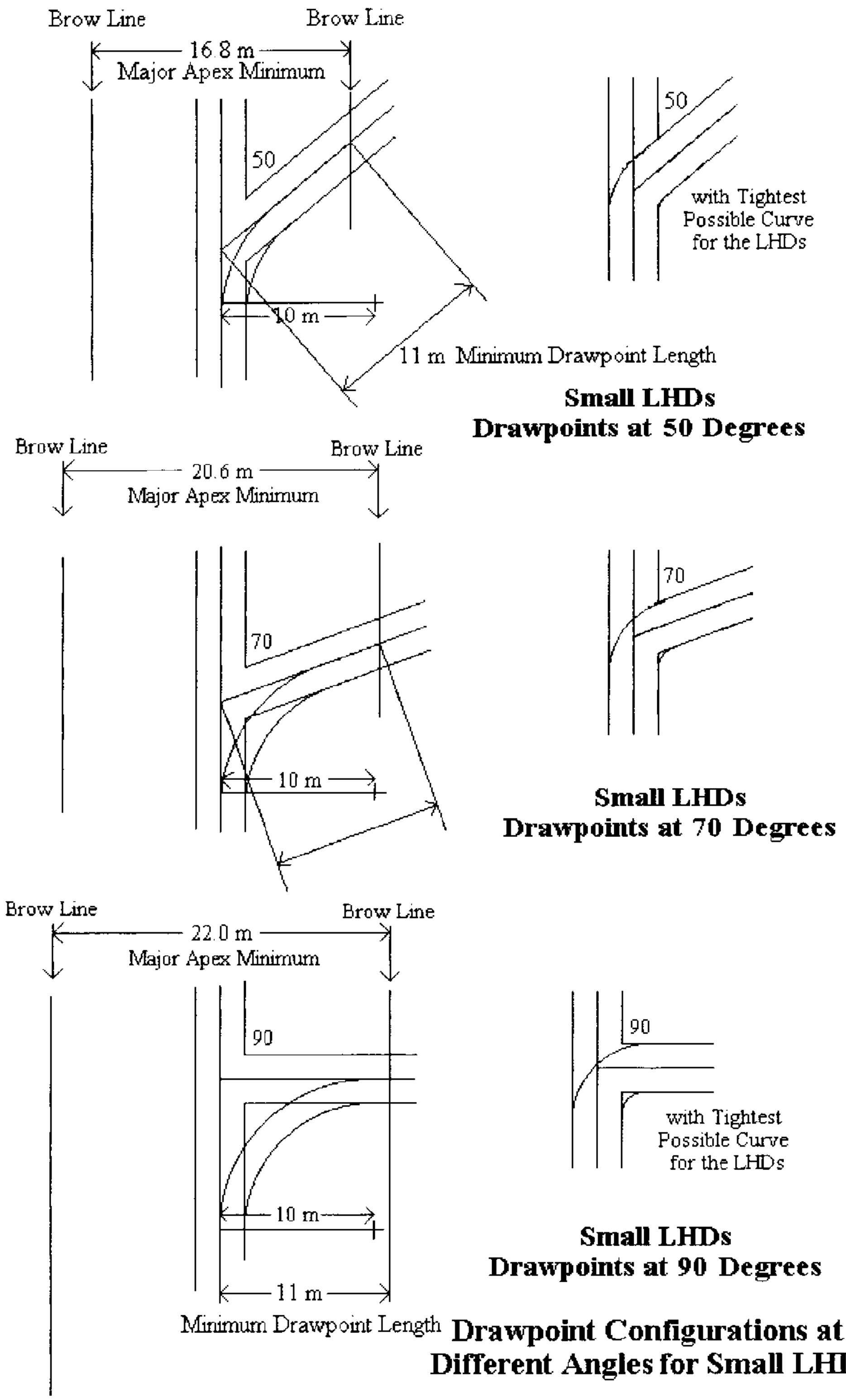
LHD Size and Planning Dimensions

Tunnel Dimensions	Units	LHD Description V Small	Small	Medium	10 tonne	Large	15 tonne	V Large
LHD Size	tonne	> 2	3 to 5	6 to 8	10	12 to 14	15	> 20
Width (+ 0.5 m on Curves)	m	2.8	3.2	3.4	4.0	4.4	5.0	5.4
Height	m	3.1	3.2	3.4	3.6	3.8	3.9	4.5
Minimum Drawpoint Length	m	9.0	11.0	12.0	13.0	15.0	17.0	19.0
Radius for Curves (Reasonable Tramming Speeds)	m	7.0	10.0	12.5	12.5	15.0	17.0	18.0
Major Apex Minimum Width								
Drawpoints at 50 Degrees	m	13.8	16.9	18.4	19.9	23.0	26.0	29.1
Drawpoints at 70 Degrees	m	16.9	20.7	22.6	24.4	28.2	31.9	35.7
Drawpoints at 90 Degrees	m	18.0	22.0	24.0	26.0	30.0	34.0	38.0

LHD SIZE

9/10/00 **LHD Size and Planning Dimensions**

Tunnel Dimensions	Units	LHD Description						
		V Small	Small	Medium	10 tonne	Large	15 tonne	V Large
LHD Size	tonne	> 2	3 to 5	6 to 8	10	12 to 14	15	> 20
Width (+ 0.5 m on Curves)	m	2.8	3.2	3.4	4.0	4.4	5.0	5.4
Height	m	3.1	3.2	3.4	3.6	3.8	3.9	4.5
Minimum Drawpoint Length	m	9.0	11.0	12.0	13.0	15.0	17.0	19.0
Radius for Curves (Reasonable Tramming Speeds)	m	7.0	10.0	12.5	12.5	15.0	17.0	18.0
Major Apex Minimum Width								
Drawpoints at 50 Degrees	m	13.8	16.9	18.4	19.9	23.0	26.0	29.1
Drawpoints at 70 Degrees	m	16.9	20.7	22.6	24.4	28.2	31.9	35.7
Drawpoints at 90 Degrees	m	18.0	22.0	24.0	26.0	30.0	34.0	38.0



Drawpoint Configurations at Different Angles for Small LHDs

DATA ON LHDS OF INCREASING SIZES

LHD Comparisons

Comparison of Diesel LHDs

Very Small Less than 2 tonne

9/10/00

Factor	Units			**Reasonable Data**
Make		**Tamrock**	**Wagner**	
Model		**Microscoop**	**HST1A**	
Bucket Size	**m^3**	**0.54**	**0.76**	
Tramming Capacity	**tonne**	**1.0**	**1.4**	
Ratios				
Tramming Capacity for bucket size	**tonne/m^3**	**1.9**	**1.8**	
Ratio Tramming Capacity to Breakout Force	**1 :**	**1.1**	**2.3**	
Ratio Tramming Capacity to Vehicle Mass	**1 :**	**3.2**	**3.7**	
Tramming Capacity for machine width	**kg/mm**	**1.0**	**1.1**	
Tramming Capacity for machine length	**kg/mm**	**0.2**	**0.3**	
Breakout Force	tonne	1.1	3.2	
Mass Empty	tonne	3.2	5.1	
Length	mm	4597	5283	
Width	mm	1050	1219	
Height of Machine	mm	1169	1219	
Height of Drivers Head - **Canopy Where Standard**	mm	1779	1828	
Tipping Height - top of Bucket	mm	2321	2972	
Tipping Height - Bucket Lip	mm	600	1181	
Turning Circle Inside Radius	mm	1838	1753	
Turning Circle Outside Radius	mm	3191	3505	
Speed Full on Flat	kph		12.1	
Speed Empty on Flat	kph		12.1	
Speed Full on + 20 % Gradient	kph		4.4	
Speed Empty on + 20 % Gradient	kph		6.0	
Tunnel Dimensions				

Width (+ 0.5 m on Curves)	**m**	**2.6**	**2.8**	**2.8**
Height	**m**	**3.1**	**3.1**	**3.1**
Minimum Drawpoint Length	**m**	**8.0**	**9.0**	**9.0**
Radius for Curves (Reasonable Tramming Speeds)	**m**	**7.0**	**7.0**	**7.0**
Width	m	2.55	2.72	
Height	m	3.08	3.13	
Drawpoint Length	m	7.68	8.41	
Turning Circle Ideal Radius **2.5 X (IR + OR) / 2**	m	6.29	6.57	

Comparison of Diesel LHDs - Small 3 to 5 tonne

Units	Wagner ST 2D	GHH LF 4.1	GHH LF 5.1	Tamrock EJC 61 D	Tamrock TORO 151	Tamrock EJC 100 D	Reasonable Data
m^3	**1.9**	**2.0**	**2.5**	**1.2**	**1.5**	**2.3**	
tonne	**3.6**	**3.8**	**4.2**	**2.7**	**3.5**	**4.5**	
tonne/m^3	**1.9**	**1.9**	**1.7**	**2.3**	**2.3**	**2.0**	
1 :	**1.6**	**2.4**	**2.5**	**1.4**	**1.8**	**1.7**	
1 :	**3.2**	**3.2**	**2.4**	**2.5**	**2.5**	**2.8**	
kg/mm	**2.2**	**2.2**	**2.5**	**1.9**	**2.4**	**2.6**	
kg/mm	**0.6**	**0.5**	**0.6**	**0.5**	**0.5**	**0.6**	
tonne	5.8	9.0	10.5	3.8	6.2	7.7	
tonne	11.5	12.1	10.2	6.8	8.7	12.7	
mm	6593	6920	7206	5486	6970	7341	
mm	1651	1690	1700	1448	1480	1702	

mm	1448	1660	1659	1500	1450	1676	
mm	1981	1900	1900	**2134**	**1740**	**2235**	
mm	3733	4100	4060	3048	3500	3962	
mm	1463	1760	1800	991	1235	1473	
mm	2700	2840	2725	1905	2735	2870	
mm	4700	5130	5005	3734	4730	5004	
kph	18.2	19.1	19.0				
kph	18.7	19.7	19.0				
kph	3.4	3.4	3.6				
kph	4.6	4.6	3.6				
m	**3.2**	**3.2**	**3.2**	**3.0**	**3.0**	**3.2**	**3.2**
m	**3.3**	**3.2**	**3.2**	**3.4**	**3.0**	**3.5**	**3.2**
m	**10.0**	**11.0**	**11.0**	**9.0**	**11.0**	**11.0**	**11.0**
m	**10.0**	**10.0**	**10.0**	**8.0**	**10.0**	**10.0**	**10.0**
m	3.15	3.19	3.20	2.95	2.98	3.20	
m	3.28	3.20	3.20	3.43	3.04	3.54	
m	9.87	10.12	10.41	8.92	10.01	10.88	
m	9.25	9.96	9.66	7.05	9.33	9.84	

Comparison of Diesel LHDs - Medium 6 to 8 tonne

Units	GHH LF 6.1	GHH LF 7.3	Tamrock EJC 130 D	Tamrock TORO 301	Wagner ST 3.5	Reasonable Data
m^3	**3.0**		**2.7**	**3.0**	**3.1**	
tonne	**6.0**	**7.3**	**5.9**	**6.2**	**6.0**	
tonne/m^3	**2.0**		**2.2**	**2.1**	**1.9**	
1 :	**1.9**		**1.3**	**2.0**	**1.3**	
1 :	**2.8**	**3.0**	**3.7**	**2.3**	**2.9**	
kg/mm	**3.24**	**3.32**	**2.77**	**2.95**	**3.07**	
kg/mm	**0.73**	**0.82**	**0.70**	**0.72**	**0.73**	
tonne	11.5		7.7	12.3	8.0	
tonne	16.9	21.7	22.0	14.5	17.5	
mm	8230	8910	8407	8620	8233	
mm	1850	2200	2133	2100	1956	
mm	1760	2000	1752	1735	1572	
mm	2000	2400	**2057**	**2200**	1956	
mm	4080		4445	4597	3962	
mm	1520		1397	1430	1295	
mm	3222	3500	2997	3030	2710	
mm	5902	6400	5511	5780	5465	
kph	26.0				18.3	
kph	27.0		19.0		18.9	
kph	4.3		3.6		4.3	
kph	6.1		3.6		6.3	
m	**3.4**	**3.8**	**3.2**	**3.0**	**3.6**	**3.4**
m	**3.3**	**3.7**	**3.2**	**3.4**	**3.3**	**3.4**

m	**12.0**	**13.0**	**11.0**	**9.0**	**12.0**	**12.0**
m	**12.0**	**13.0**	**10.0**	**8.0**	**11.0**	**12.5**
m	3.35	3.70	3.63	3.60	3.46	
m	3.30	3.70	3.36	3.50	3.26	
m	11.53	12.61	11.76	12.12	11.49	
m	11.41	12.38	10.64	11.01	10.22	

Comparison of Diesel LHDs - 10 tonne

Units					**Reasonable Data**
	Tamrock EJC 210 D	**Tamrock TORO 400**	**Wagner ST 6 C**	**Wagner ST 1000**	
m^3	**4.6**	**3.8**	**4.6**	**5.0**	
tonne	**9.5**	**9.6**	**9.5**	**10.0**	
tonne/m^3	**2.1**	**2.5**	**2.1**	**2.0**	
1 :	**1.0**	**2.1**	**1.5**	**3.2**	
1 :	**2.6**	**2.4**	**2.5**	**2.5**	
kg/mm	**3.5**	**3.9**	**4.2**	**4.9**	
kg/mm	**1.0**	**1.0**	**1.0**	**1.2**	
tonne	9.9	20.4	14.5	31.8	
tonne	25.1	22.8	24.2	25.0	
mm	9957	9252	9490	8530	
mm	2718	2480	2261	2040	
mm	1778	1900	1684	1480	
mm	**2311**	**2370**	2134	**2108**	
mm		5002	4547	4967	

mm		1810	1448	1801	
mm	3302	3533	3312	3275	
mm	6553	6590	6320	5800	
kph			25.3	22.6	
kph			26.9	24.3	
kph			3.2	4.2	
kph			5.7	5.6	
m	**4.2**	**4.0**	**3.8**	**3.6**	**4.0**
m	**3.6**	**3.7**	**3.4**	**3.4**	**3.6**
m	**14.0**	**13.0**	**13.0**	**12.0**	**13.0**
m	**13.0**	**13.0**	**12.0**	**12.0**	**12.5**
m	4.22	3.98	3.76	3.54	
m	3.61	3.67	3.43	3.41	
m	13.57	12.92	12.92	11.94	
m	12.32	12.65	12.04	11.34	

Comparison of Diesel LHDs - Large 12 to 14 tonne

Units						Reasonable Data
	Wagner ST 8 B	**GHH LF 12**	**Tamrock TORO 450**	**Tamrock TORO 1250**	**Tamrock TORO 1400**	
m^3	**6.5**		**4.6**	**5.0**	**5.5**	
tonne	**13.6**	**12.0**	**12.0**	**12.5**	**14.0**	
tonne/m	**2.1**		**2.6**	**2.5**	**2.5**	

^3						
1 :	**1.6**		**2.1**	**2.2**	**2.0**	
1 :	**2.7**	**2.6**	**2.6**	**2.6**	**2.4**	
kg/mm	**4.9**	**4.6**	**4.4**	**4.4**	**5.0**	
kg/mm	**1.3**	**1.1**	**1.2**	**1.2**	**1.3**	
tonne	22.4		25.2	28.0	28.0	
tonne	36.8	31.5	31.2	33.0	33.5	
mm	10287	10770	10003	10508	10508	
mm	2769	2600	2745	2825	2825	
mm	1930	1890	2205	2335	2335	
mm	2591	2400	**2540**	**2540**	**2540**	
mm	5105		5350	5937	5937	
mm	1676		1880	2473	2473	
mm	3531	4050	3274	3246	3307	
mm	7010	7540	6537	6672	6887	
kph	13.5					
kph	14.0					
kph	2.2					
kph	3.2					
m	**4.4**	**4.2**	**4.2**	**4.4**	**4.4**	**4.4**
m	**3.9**	**3.7**	**3.8**	**3.8**	**3.8**	**3.8**
m	**15.0**	**15.0**	**14.0**	**15.0**	**15.0**	**15.0**
m	**14.0**	**15.0**	**13.0**	**13.0**	**13.0**	**15.0**
m	4.27	4.10	4.25	4.33	4.33	
m	3.89	3.70	3.84	3.84	3.84	
m	14.18	14.47	13.84	14.35	14.35	
m	13.18	14.49	12.26	12.40	12.74	

LHD Comparisons

Comparison of Diesel LHDs - 15 tonne

9/10/00

Factor	Units			Reasonable Data
Make		**GHH**	**Tamrock**	
Model		**LF 15.1**	**TORO 650**	
Bucket Size	**m^3**		**6.5**	
Tramming Capacity	**tonne**	**15.0**	**15.0**	
Ratios				
Tramming Capacity for bucket size	**tonne/m^3**		**2.3**	
Ratio Tramming Capacity to Breakout Force	**1 :**		**2.3**	
Ratio Tramming Capacity to Vehicle Mass	**1 :**	**3.1**	**2.4**	
Tramming Capacity for machine width	**kg/mm**	**4.3**	**5.0**	
Tramming Capacity for machine length	**kg/mm**	**1.2**	**1.3**	
Breakout Force	tonne		33.8	
Mass Empty	tonne	46.5	36.4	
Length	mm	12370	11410	
Width	mm	3500	3000	
Height of Machine	mm	2110	2120	
Height of Drivers Head - **Canopy Where Standard**	mm	2500	**2640**	
Tipping Height - top of Bucket	mm		6164	
Tipping Height - Bucket Lip	mm		4210	
Turning Circle Inside Radius	mm	4750	3575	
Turning Circle Outside Radius	mm	8850	7180	
Speed Full on Flat	kph			
Speed Empty on Flat	kph			
Speed Full on + 20 % Gradient	kph			
Speed Empty on + 20 % Gradient	kph			
Tunnel Dimensions				
Width (+ 0.5 m on Curves)	**m**	**5.0**	**4.6**	**5.0**
Height	**m**	**3.8**	**3.9**	**3.9**

Minimum Drawpoint Length	**m**	**17.0**	**16.0**	**17.0**
Radius for Curves (Reasonable Tramming Speeds)	**m**	**17.0**	**14.0**	**17.0**
Width	m	5.00	4.50	
Height	m	3.80	3.94	
Drawpoint Length	m	16.17	15.35	
Turning Circle Ideal Radius **2.5 X (IR + OR) / 2**	m	17.00	13.44	

LHD Comparisons

Comparison of Diesel LHDs

Very Large Greater than 20 tonne

9/10/00

Factor	Units			**Reasonable Data**
Make		**Tamrock**	**Wagner**	
Model		**TORO 2500**	**ST 15 Z**	
Bucket Size	**m^3**	**12.0**	**11.5**	
Tramming Capacity	**tonne**	**25.0**	**20.4**	
Ratios				
Tramming Capacity for bucket size	**tonne/m^3**	**2.1**	**1.8**	
Ratio Tramming Capacity to Breakout Force	**1 :**	**2.2**	**3.7**	
Ratio Tramming Capacity to Vehicle Mass	**1 :**	**3.0**	**3.3**	
Tramming Capacity for machine width	**kg/mm**	**6.4**	**6.0**	
Tramming Capacity for machine length	**kg/mm**	**1.8**	**1.6**	
Breakout Force	tonne	54.0	74.5	
Mass Empty	tonne	76.0	67.1	
Length	mm	14000	12396	
Width	mm	3900	3404	
Height of Machine	mm	2896	2413	
Height of Drivers Head - **Canopy Where Standard**	mm	**3161**	**3098**	
Tipping Height - top of Bucket	mm	7370	7046	
Tipping Height - Bucket Lip	mm	3156	2821	

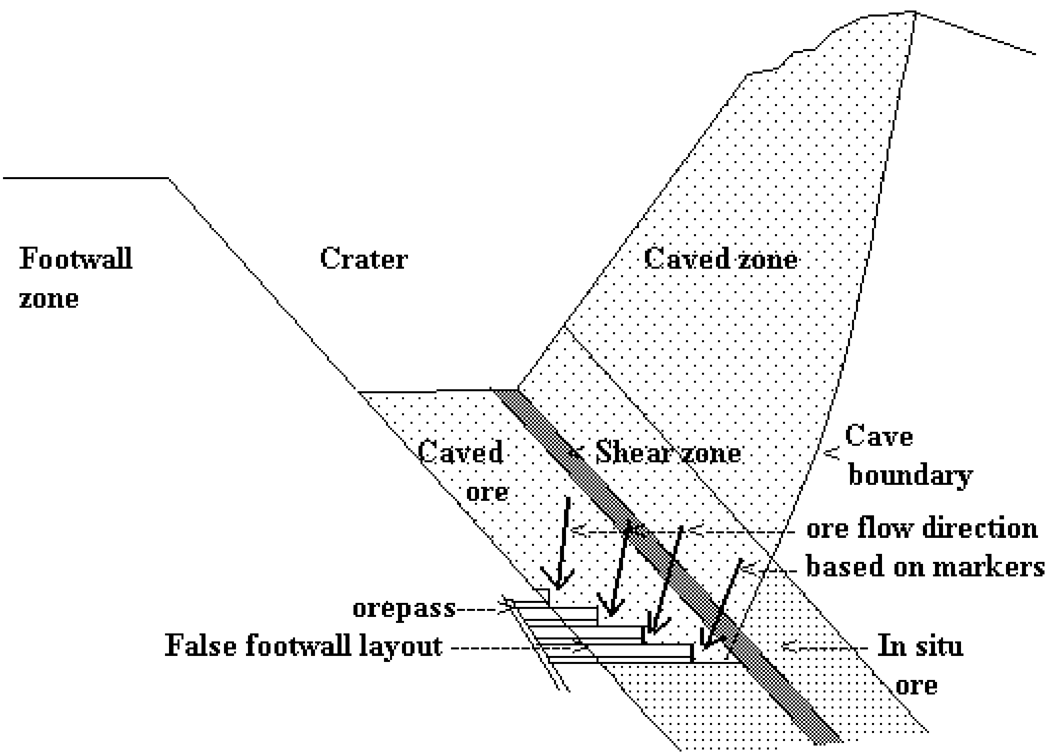
Turning Circle Inside Radius	mm	4774	4148	
Turning Circle Outside Radius	mm	9436	8443	
Speed Full on Flat	kph		14.7	
Speed Empty on Flat	kph		15.8	
Speed Full on + 20 % Gradient	kph		1.6	
Speed Empty on + 20 % Gradient	kph		2.3	
Tunnel Dimensions				
Width (+ 0.5 m on Curves)	**m**	**5.4**	**5.0**	**5.4**
Height	**m**	**4.5**	**4.4**	**4.5**
Minimum Drawpoint Length	**m**	**19.0**	**17.0**	**19.0**
Radius for Curves (Reasonable Tramming Speeds)	**m**	**18.0**	**16.0**	**18.0**
Width	m	5.40	4.90	
Height	m	4.46	4.40	
Drawpoint Length	m	18.46	16.79	
Turning Circle Ideal Radius **2.5 X (IR + OR) / 2**	m	17.76	15.74	

DESIGN TOPIC

LHD Incline Drawpoint and Front Cave Layouts

GENERAL

An incline drawpoint layout was first introduced at King Mine, Zimbabwe as it was not possible to maintain a horizontal layout through the major internal shear zone. The layout was based on a successful inclined grizzly layout used at Nil Section, Shabanie Mine, Zimbabwe. The King Mine layout was termed the 'False Footwall layout' because the inclination of the plane of the drawpoints was flatter than the footwall dip.



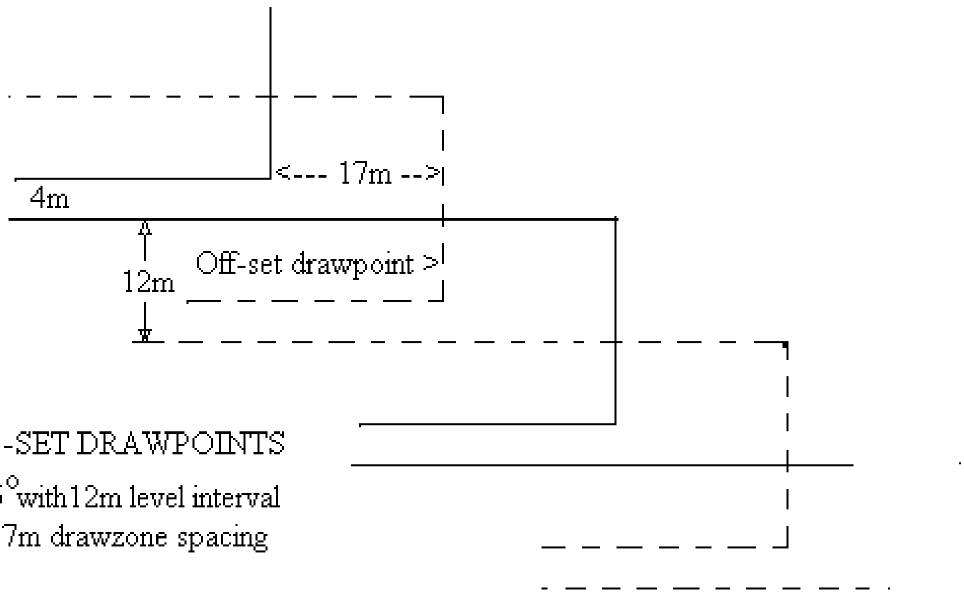
Diagrammatic section through King Mine showing Incline layout, major shear zone, caved zone and the direction of ore flow as determined by markers.

There are two incline drawpoint layouts, namely, the truncated ‘SLC’ and the inclined ‘egg-box’ and these will be described in separate sections.

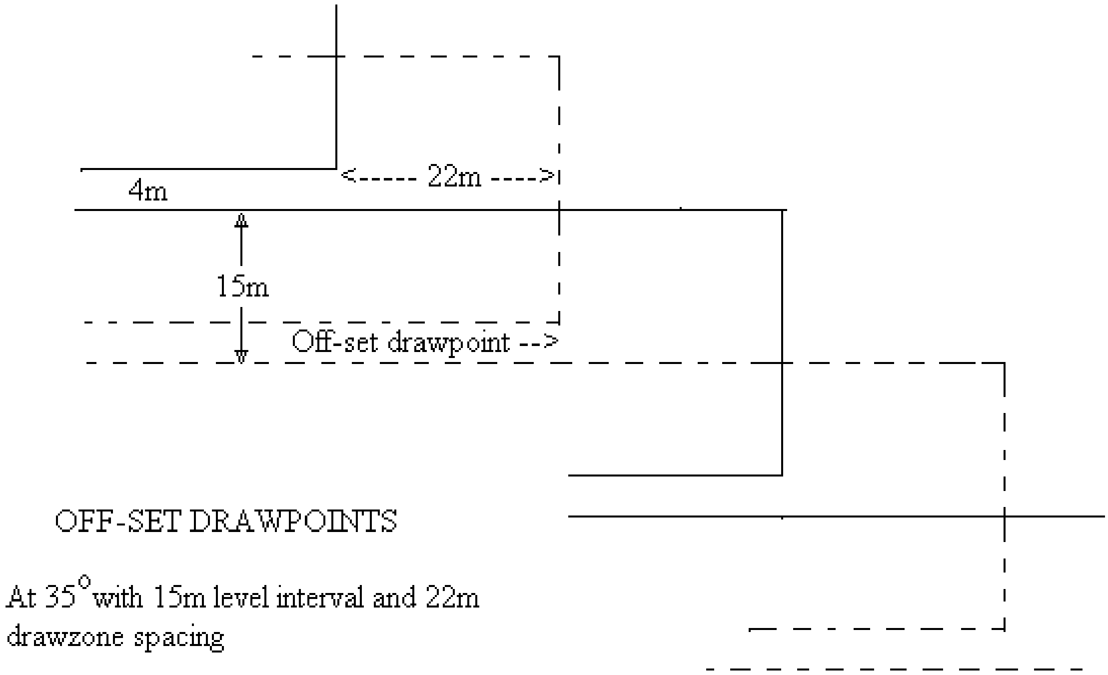
SPACING OF DRAWZONES

One of the major issues that has to be decided is the spacing of the drawzones. For the want of better information at present , the spacings must relate to empirical data from horizontal layouts, which means that the spacings could be:-

Dip of incline plane	Fragmentation	Vertical	Along strike	Down dip
35°	Medium	12m	12m	17m
35°	Coarse	15m	15m	22m
40°	Fine	10m	10m	12m
40°	Medium	12m	12 m	14m
40°	Coarse	15m	15m	18m
40°	Coarse	18m	15m	22m



OFF-SET DRAWPOINTS
At 35° with 12m level interval and 17m drawzone spacing



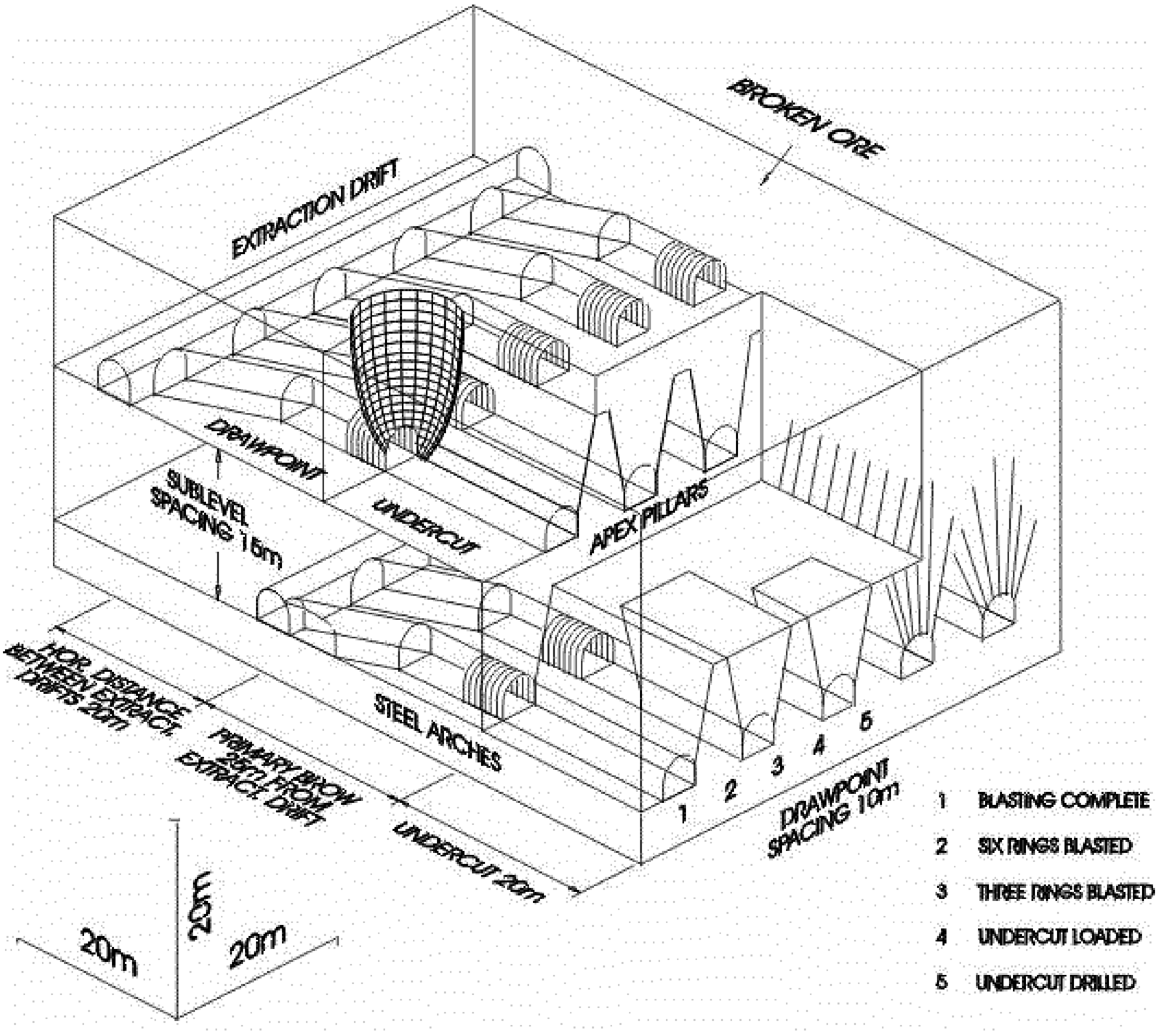
Thus the spacing from level to level can be 15m, from drawpoint to drawpoint on the level from 12m to 15m depending on pillar strength and from level to level it would be 22m which would relate the 22m across the major apex in some horizontal LHD layouts.

There is no reason why the ‘false footwall ‘ technique should not be applied to other mining operations.

TYPES OF INCLINE DRAWPOINT LAYOUTS

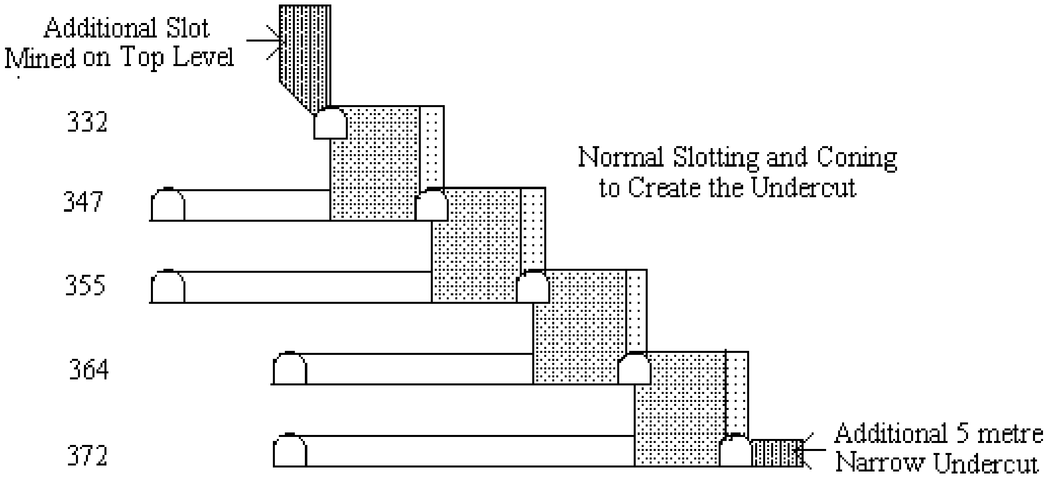
Truncated SLC layout

The truncated SLC layout is simply the SLC layout with the correct spacings of production drifts and extraction drifts to ensure stability and ore extraction. The details of the Cassiar Mine layout are shown in the following diagram.



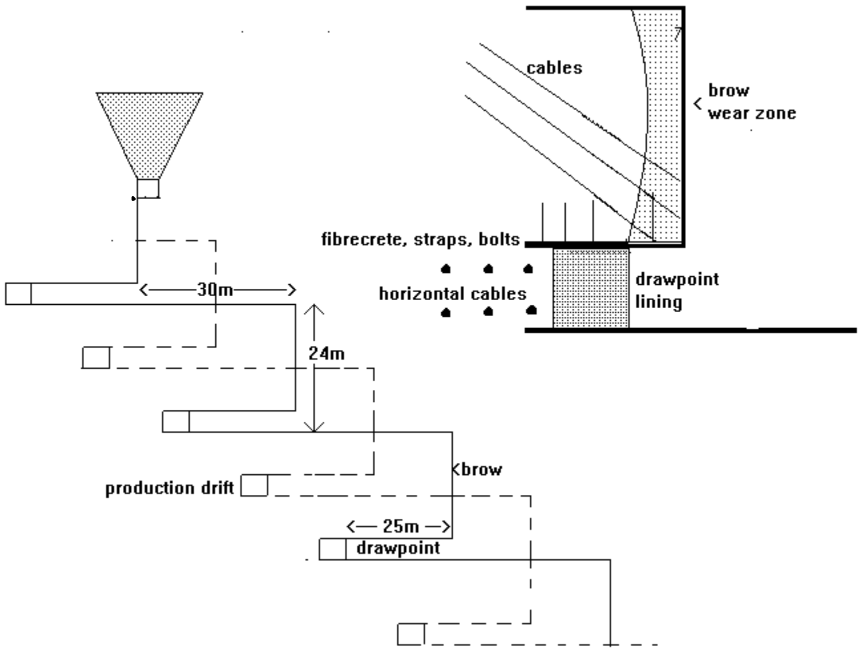
Cassiar Mine Incline Drawpoint Layout

The details of the King Mine layout are shown in the following diagram. Contrary to usual practice the undercutting was done as an inverted ‘V’ from the lower level upwards. This proved to be successful despite misgivings at the start. The reason for this approach was to create a cave of the hangingwall zone, namely the higher ground so as to have topographical caving pressure.



Gaths - King Section Main over 4
Showing the Additional
Top Slot and Bottom Narrow Undercut

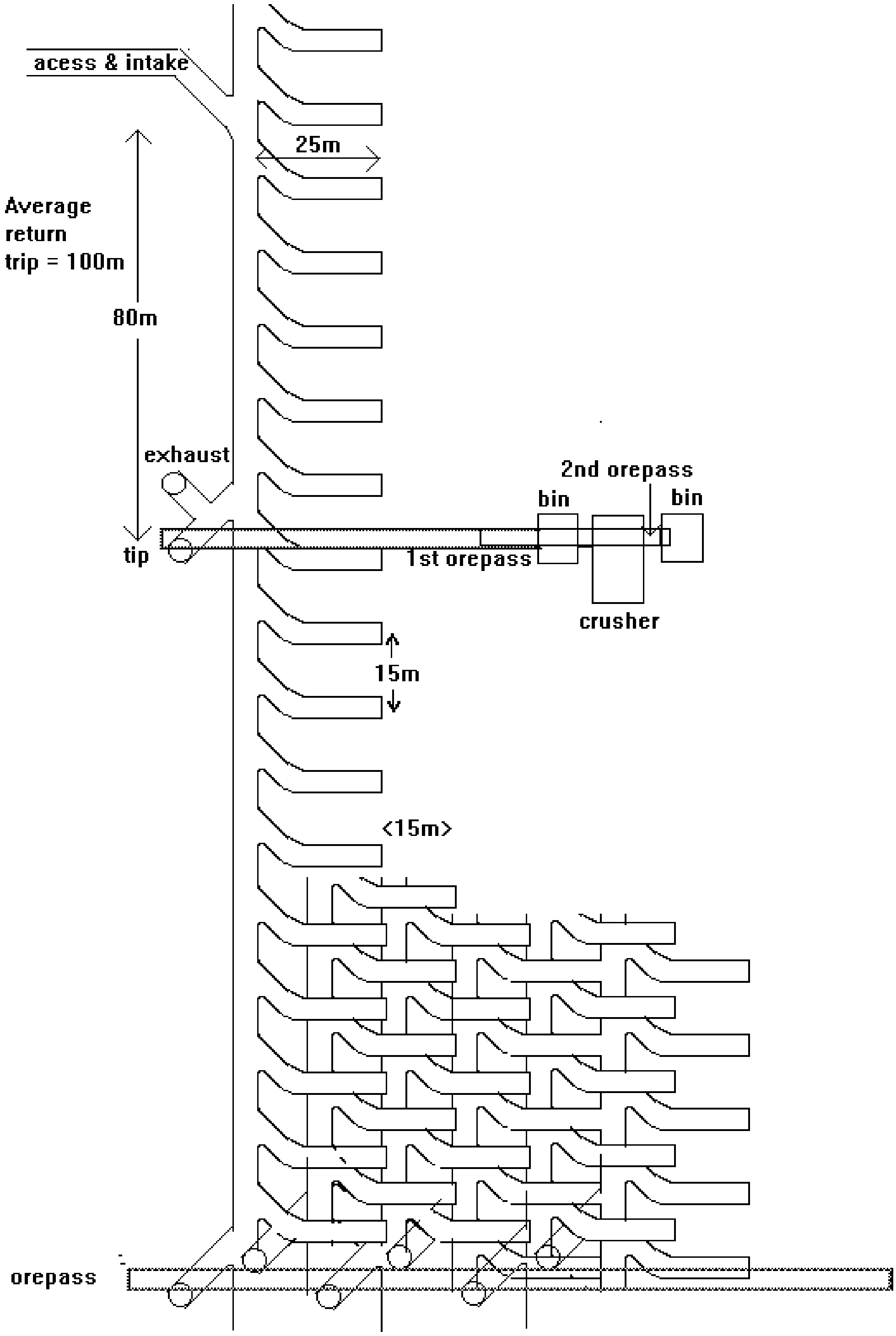
The following series of diagrams show various proposed layouts based on different mining environments.



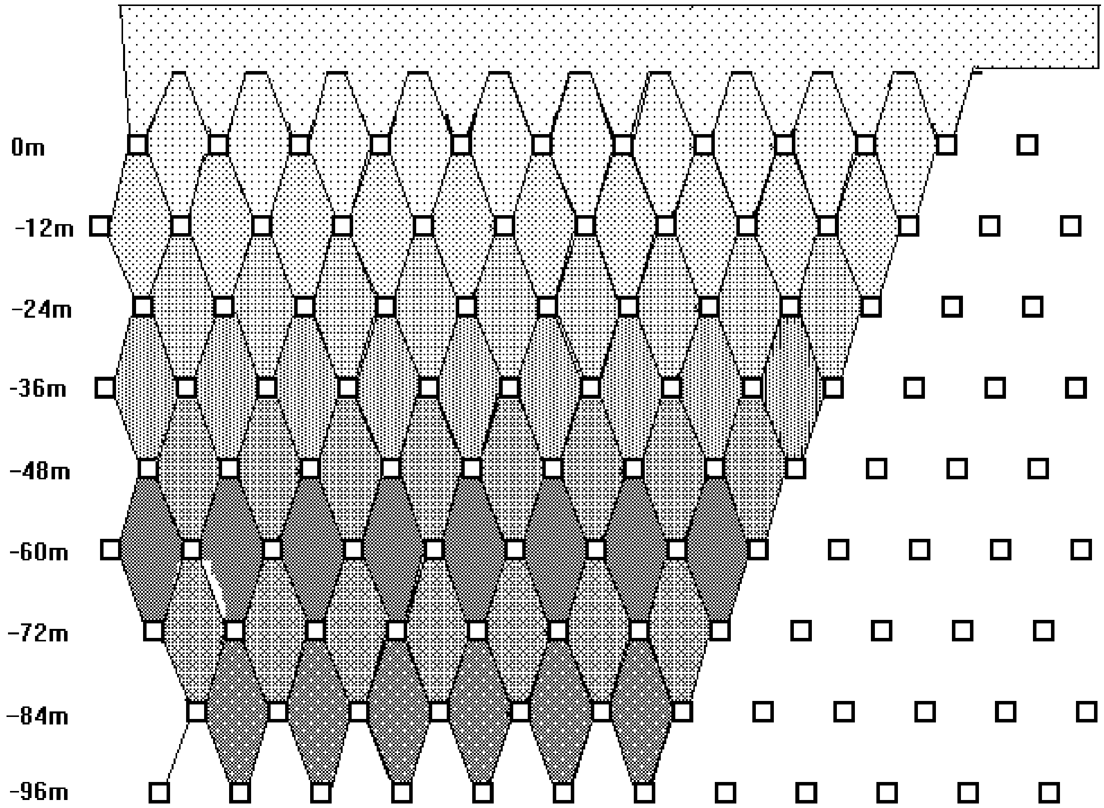
INCLINE DRAWPOINT LAYOUT - OFF-SET DRAWPOINTS AND SUPPORT

Drawpoint spacing at 15m on horizontal on strike and dip, vertical at 12m
Brow height 20m, brow wear back 15m

Fig.22



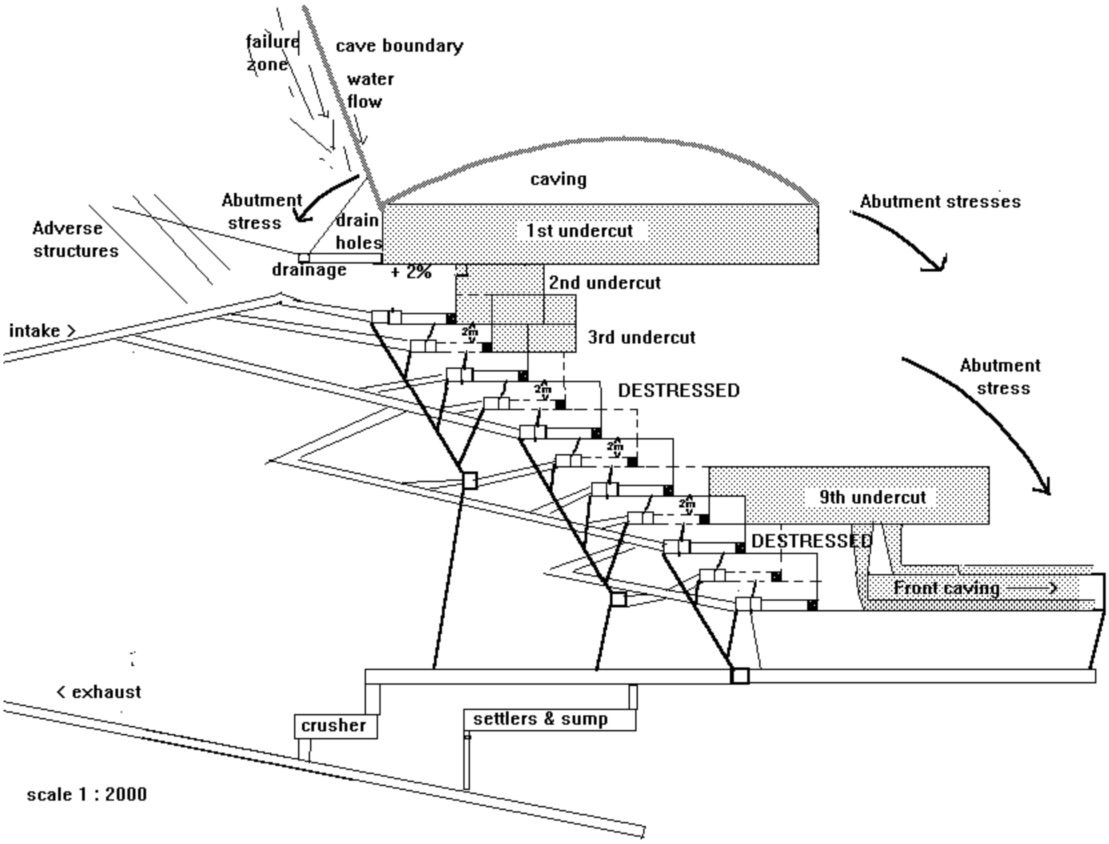
PLAN OF INCLINE DRAWPOINT LAYOUT - LEVEL INTERVAL 12M - DRAWPOINTS AT 15M



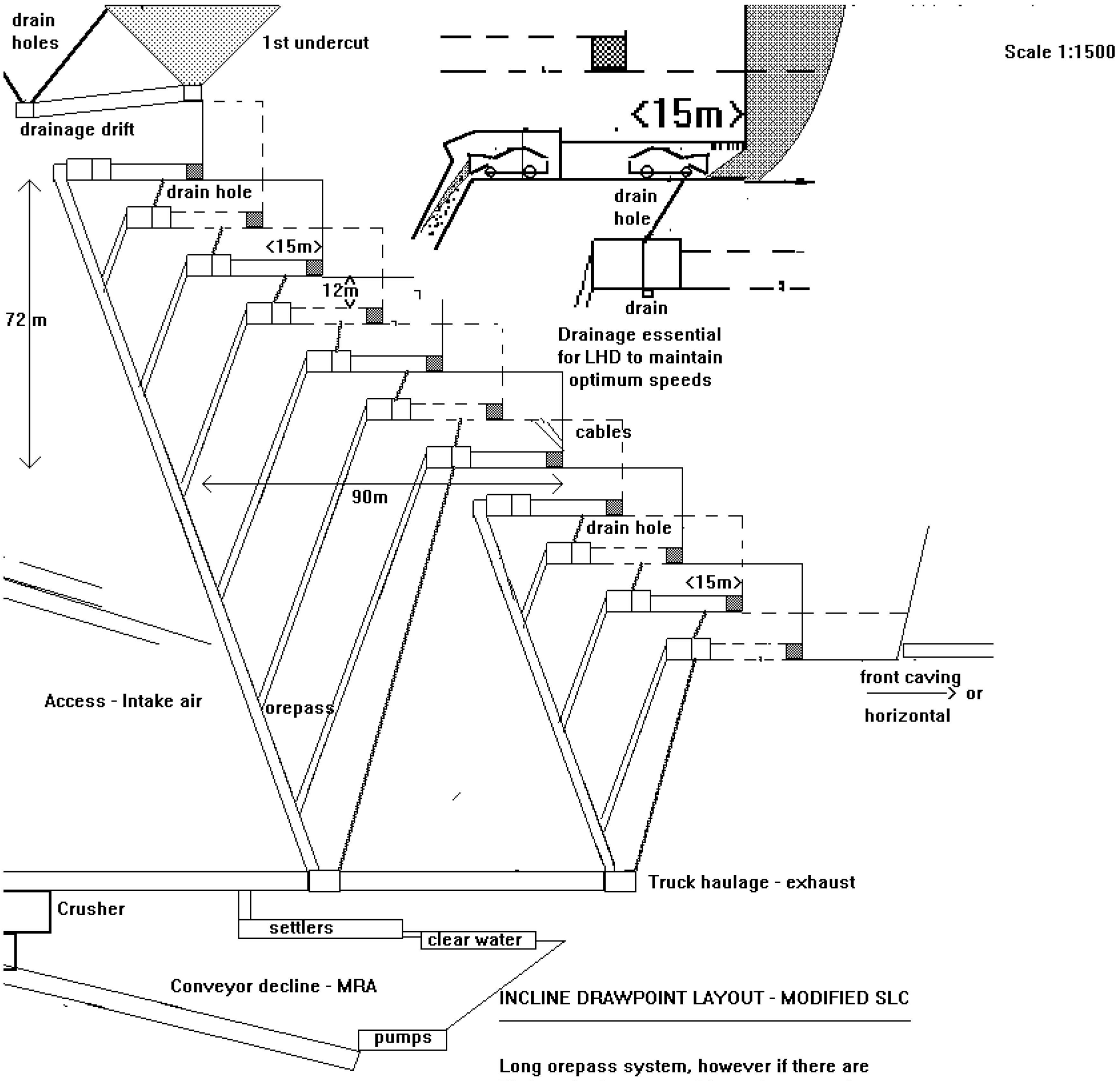
End on view of Incline drawpoint layout showing brows of the drawpoints on levels 12m vertical interval and set back 15m from level to level.

Undercutting sequence is from top to bottom

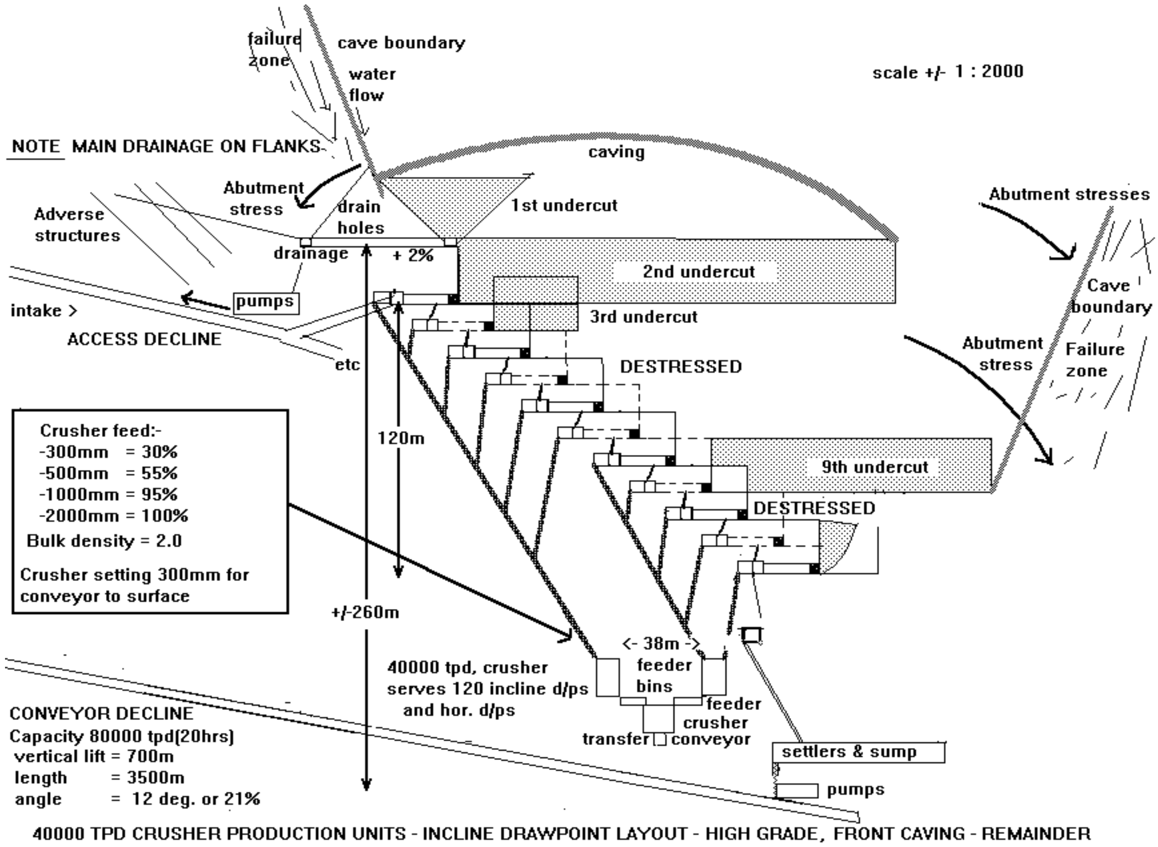
In the following diagram an upper undercut level has been developed to ensure the correct hydraulic radius for propagation of the cave, allowing for stress relief cover over the upper footwall drifts.



INCLINE DRAWPOINT LAYOUT - HIGH GRADE
FRONT CAVING - REMAINDER

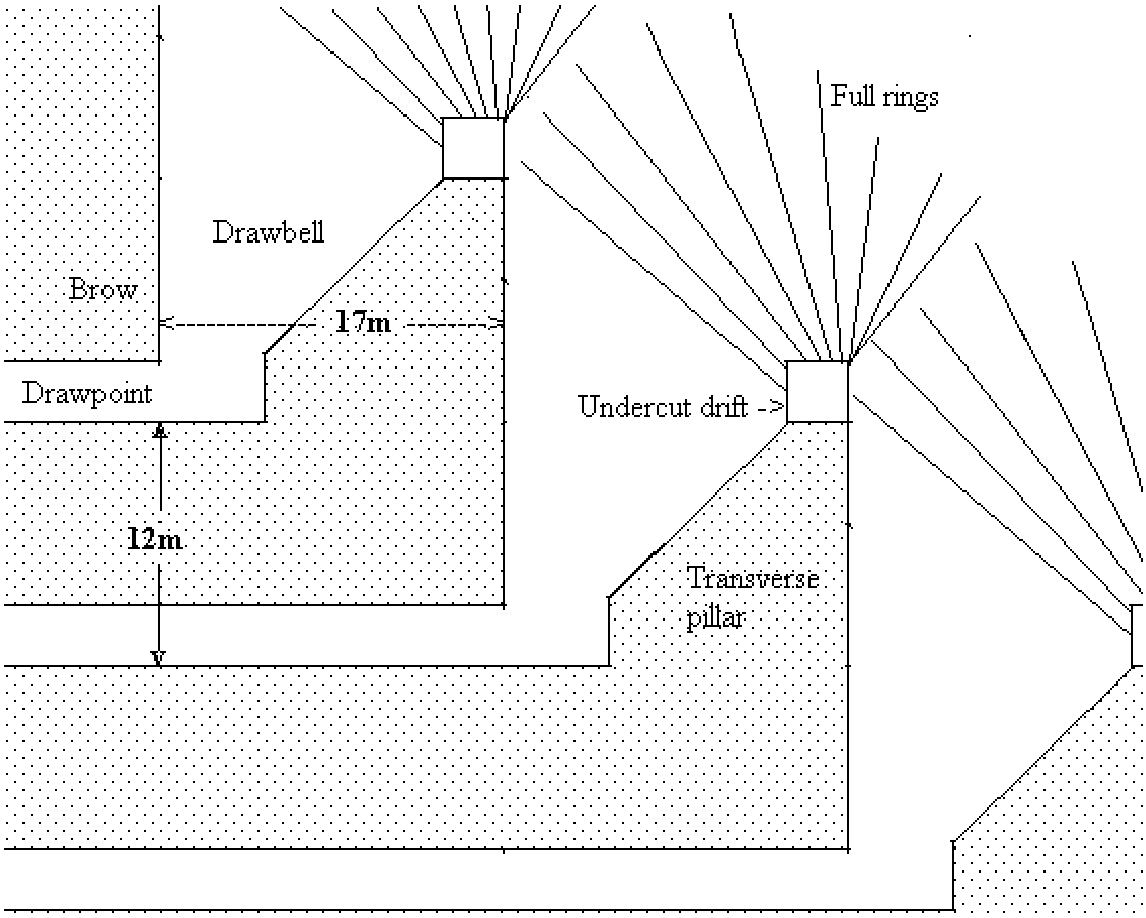


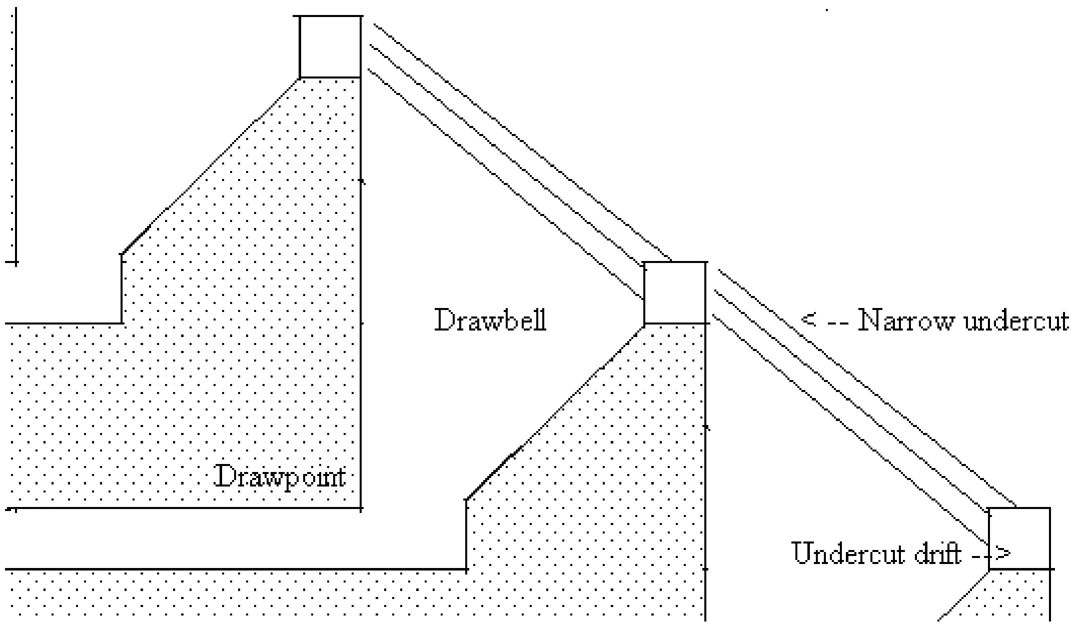
Long orepass system, however if there are likely to be hangup problems due to mud or packing then orepasses will be shorter and collecting levels are required - see Batu 1.

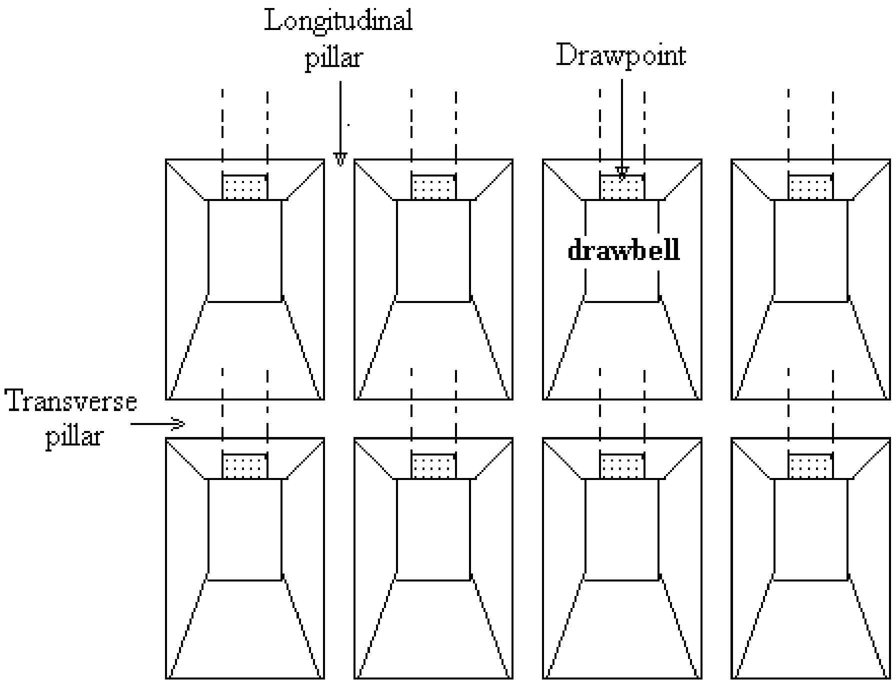


Incline Egg-box

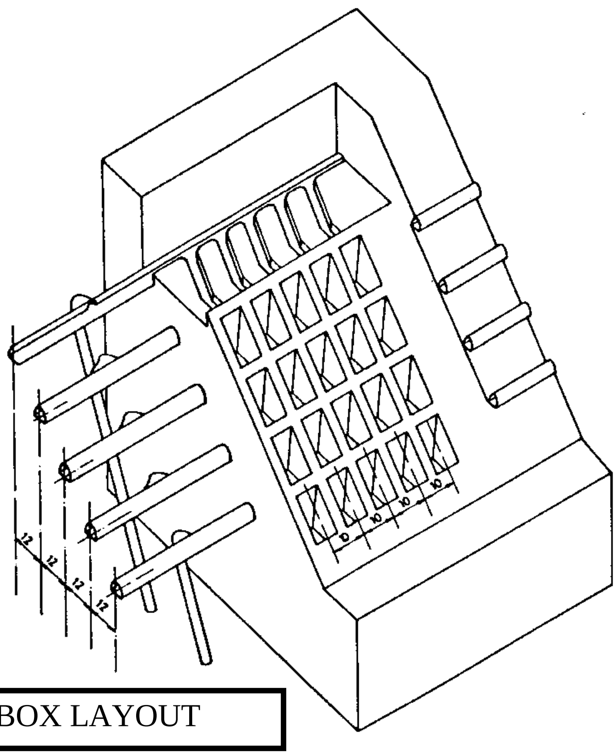
The incline egg box was designed to provide a stronger structure, a smooth incline plane and to provide the facility to do advance undercutting. This system was used at Bell Mine with a certain measure of success, some drawpoints producing over 100 000 tons whilst others had problems owing to wedge failures along major shear zones. Footwall stability is of paramount importance in designing any footwall layout. This is not only during the undercutting period ,but also the production period. Suspect drawpoints must be prevented from collapse as a result of weight and poor draw control or draw management owing to lack of understanding or suitable equipment.


Full rings
Drawbell
Brow
17m
Drawpoint
Undercut drift ->
12m
Transverse
pillar

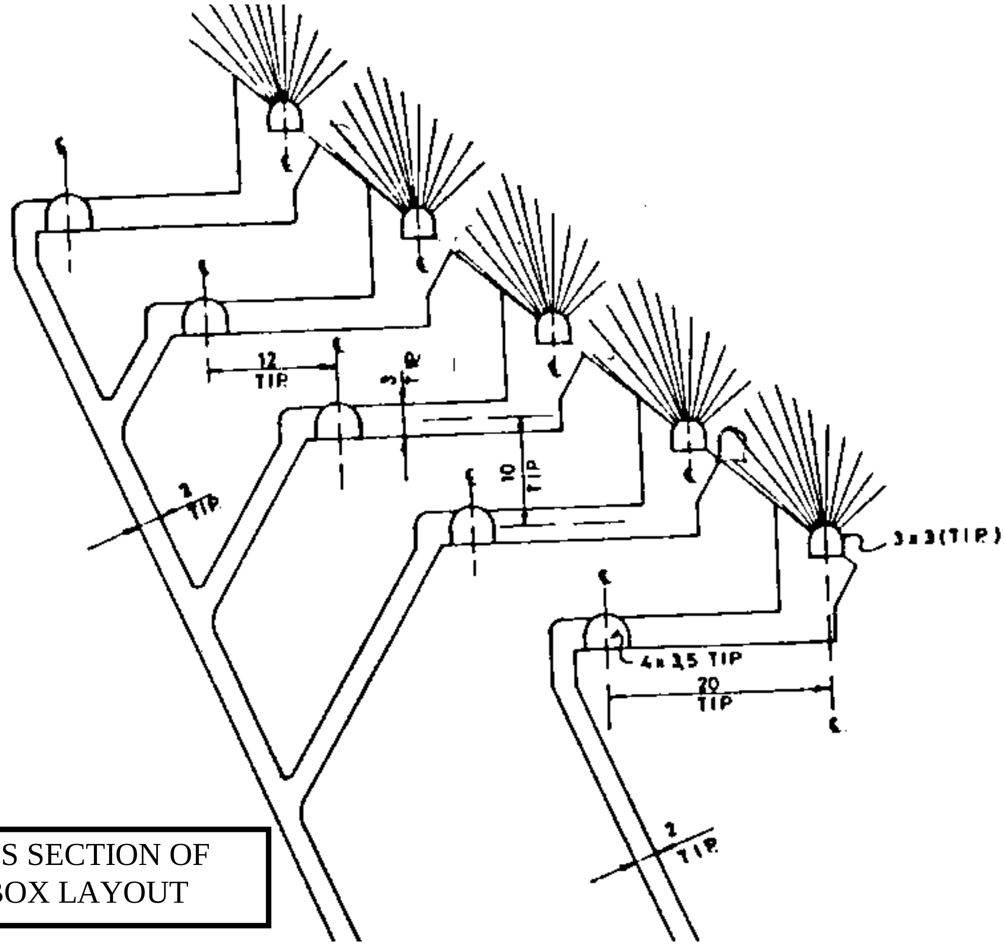

Drawbell
< -- Narrow undercut
Drawpoint
Undercut drift -->



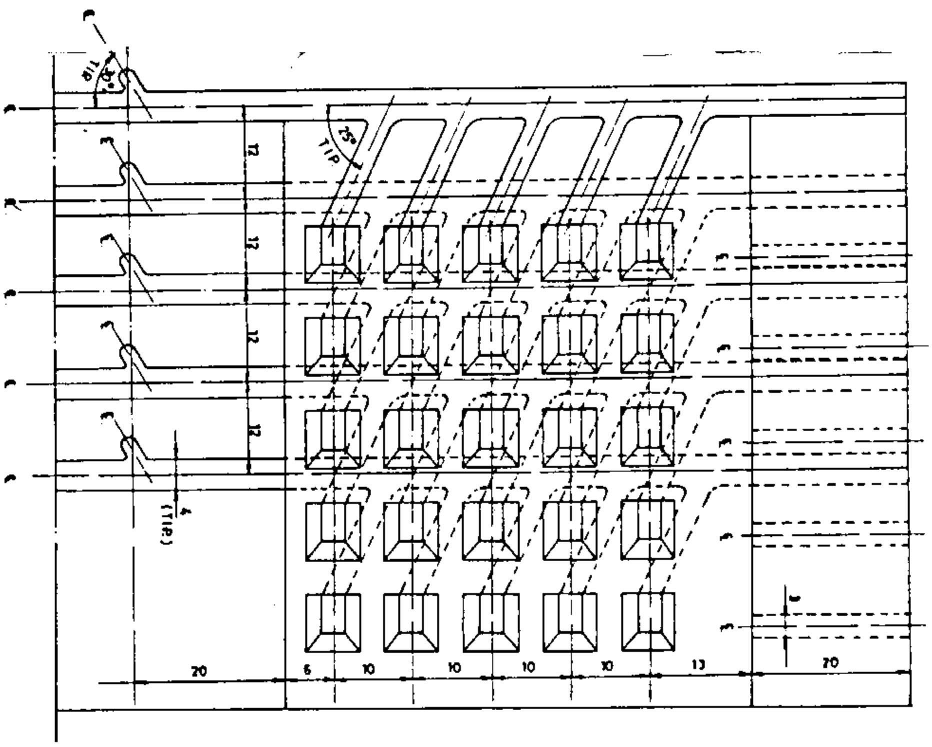
PLAN VIEW OF INCLINED EGGBOX LAYOUT



3-D VIEW OF EGGBOX LAYOUT



CROSS SECTION OF
EGGBOX LAYOUT



PLAN OF SUPERIMOSED LEVELS AND DRAWBELLS IN AN EGGBOX LAYOUT

PRACTICAL EXPERIENCE OF INCLINE DRAWPOINT LAYOUTS

The success story of the incline drawpoint layout was King Mine where the method saved the mine excessively high production costs and possible closure. The method was introduced at Bell Mine with identifiable problems in the form of massive wedge failures because the upper level was not overcut and the mining was done from the lower level upwards. At a later stage the truncated SLC was changed to the inclined eggbox with good results in the more stable areas where 240000 tons were drawn from some drawpoints. However, in highly sheared areas only 60% of the tonnage was drawn. Larger pillars, advanced undercutting and a better sequence would have resulted in a more stable mining environment.

Mine	Layout	D/P spacing	tons per day - tpd
King mine	False footwall	10.0m / 12.5m	2000
Bell Mine	Truncated SLC	15m	5000

ADVANTAGES AND DISADVANTAGES OF THE INCLINE DRAWPOINT LAYOUTS

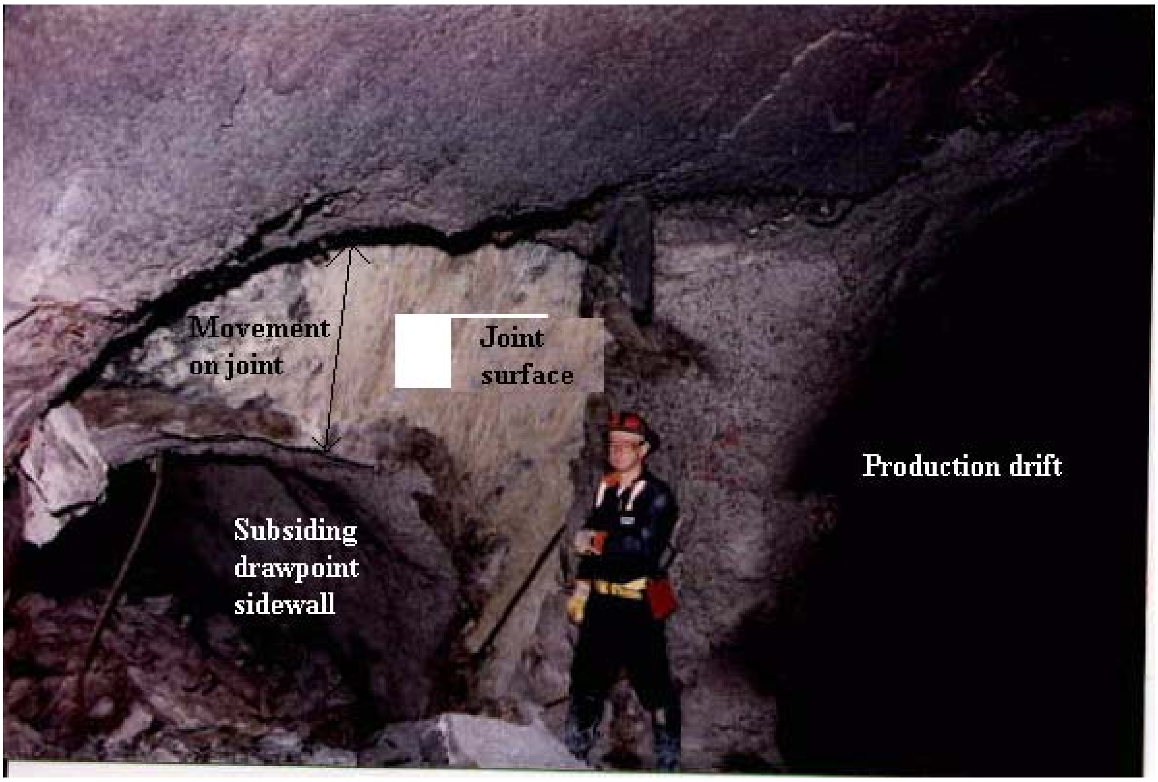
The advantages of incline layouts are:

- High productivity with close drawpoint spacing.
- The length of the LHD is not a restriction on spacing as the drawpoint length is not critical.
- Brow wear appears to be less than with a horizontal layout.
- Brow wear is not a problem as brow retreat can be catered for by the length of the drawpoint drift and a difference of 2m between adjacent drawpoints is not important.
- With the truncated SLC layout the undercut can be broken to any distance from the drawpoint.
- Complete undercutting is easy to achieve
- The incline eggbox layout permits advance undercutting.
- The extraction level is backed by solid ground. Better ore recovery in dipping orebodies and where there is a potential for <u>inclined draw</u>
- Drawpoints handle big rocks owing to the better flow characteristics and presentation.
- Better secondary drilling facilities
- Can handle wet muck more efficiently and will permit a more uniform draw
- Drawpoint spacing can be reduced to cater for weaker ground and to improve recovery.
- Drawpoint repair easier as there is no interference with production
- The stability of an incline drawpoint layout can be likened to the stability of a pit slope.

Disadvantages compared to a horizontal layout are:

- The high capital costs
- Ventilation is more costly.
- Supervision not as good.

- Electric cable lengths can be excessive
- Liable to severe damage from wedge failure unless area properly overcut and effect of structures recognised as shown in the following photograph at Bell Mine..



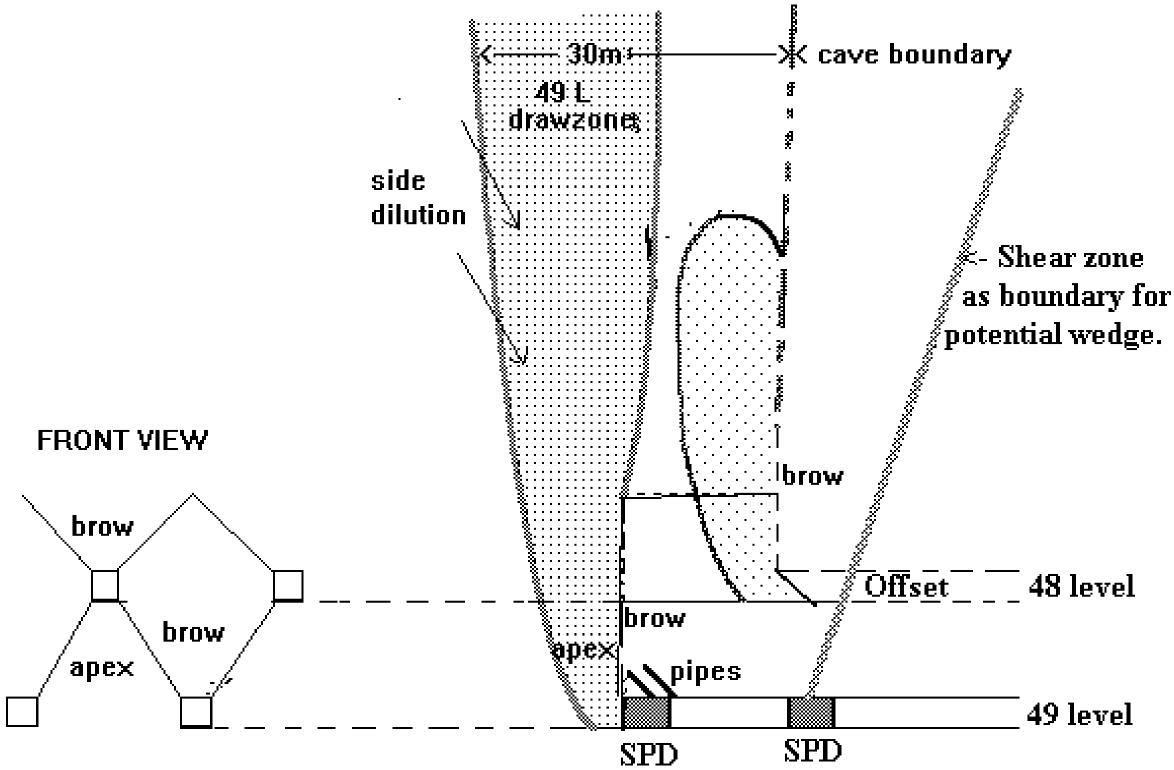
FRONT CAVE LAYOUTS

The front cave technique was developed from the overdraw system used on the two lower levels of SLC operations on Shabanie Mine, Zimbabwe. The technique involves a retreat on one or more levels from an initiating slot. The slot can be in the centre of the orebody or against the boundary. A central slot reduces the haul distances and provides double the number of semi permanent drawpoints - SPD. The lower level is the production level with the upper level being the undercut level. If three levels are used then the two lower levels are production levels. The number of levels used is a function of the cavability of the orebody so that when the next SPD is reached the overlying column has caved and can be drawn at a high rate. It is essential to maintain a fairly straight line between the SPDs to ensure interaction between the drifts.

The drilling, blasting and loading sequence is quite complicated, particularly at the start. Broad ground rules can be laid down, but each ring must be assessed on its own. Sections are required along each front cave drift and these sections should show lithology, geological structures and cave face. Draw tonnages must be related to the rings and analysed. Ventilation requirements are as for SLC operations

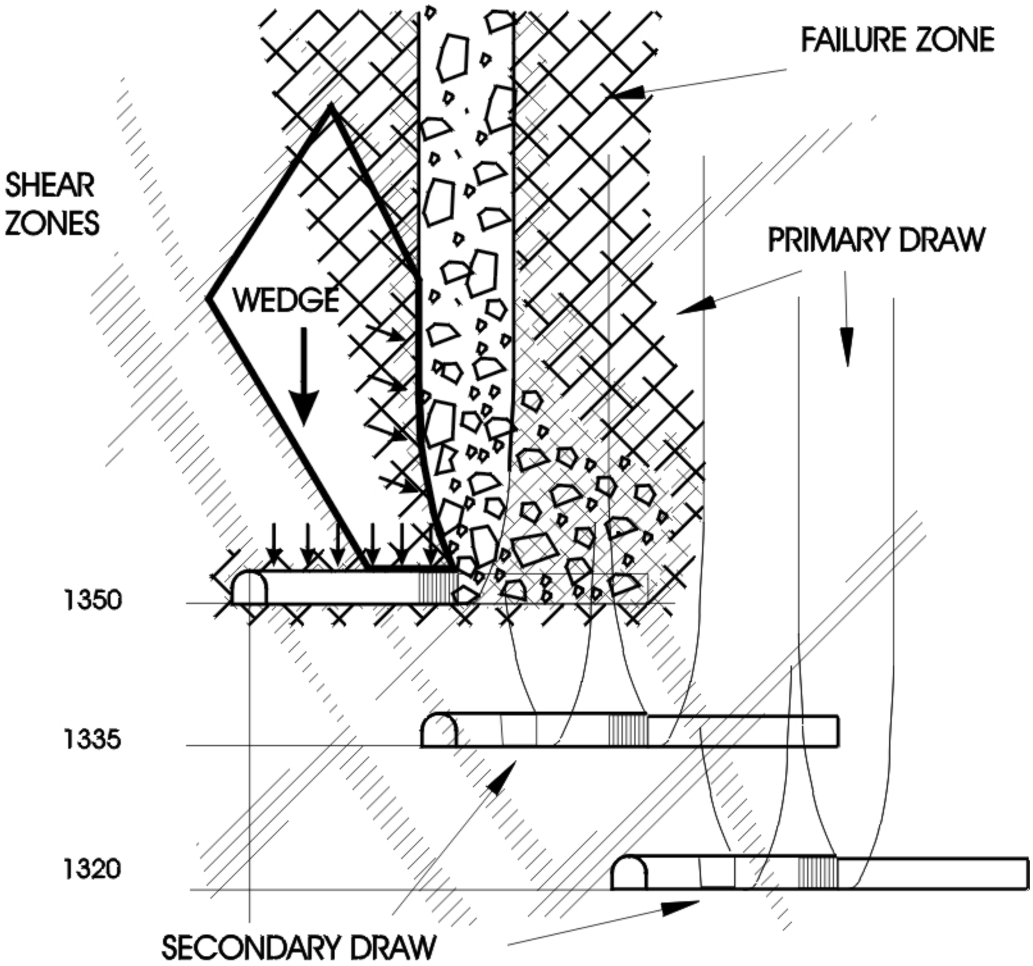
The big danger with Front Caving is the formation of an overhang or large wedges which will severely damage the production level. Front caving can only be used where the cave follows the undercut. If one level is used the caving must be virtually instantaneous

Front caving is a method that requires strict discipline in terms of the loading and blasting sequences.



A diagram showing a two level layout with SPDs on 49 level at 15m spacing. 48 level is the undercut level with caving occurring with a 15m face advance. The diagram shows a major shear zone which will form the boundary of a large wedge which would cause extensive damage on 49 level. In this case 48 level would be advanced through the shear zone and the wedge destroyed by undercutting and drawing.

The following diagram shows the type of wedge failure that occurred at Cassiar mine.



Wedge failure and drawpoint loading

DESIGN TOPIC

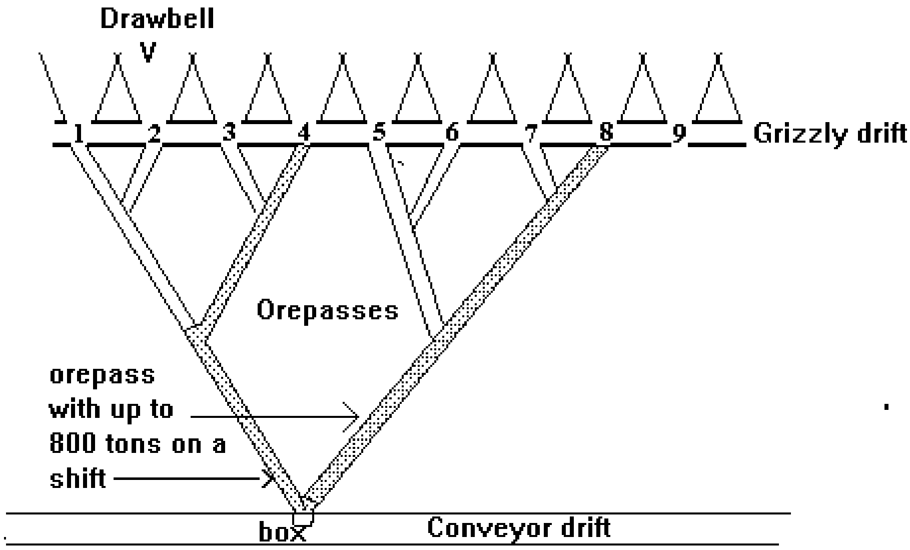
Grizzly and Slusher Layouts

GRIZZLY LAYOUTS

Grizzly layouts were to be found on most of the old mines that were block caving finely fragmented ore. The concept is the gravity flow of material through the drawpoint, over a grizzly and down an orepass into a box which fed trains, conveyors or trucks. The size of the grizzly was a function of the ore handling facilities and ranged from 0.3m to 0.5m (12” to 20”). Large rocks on the grizzly were broken by a manually wielded hammer or secondary blasting. Productivity was high in the grizzly layouts with up to 800 tons per man shift at very low costs. Capital expenditure is fairly high owing to the amount of development required.

The grizzly method of cave mining is a highly efficient and cost effective method with the right fragmentation.. Grizzly layouts were used at King and Shabanie mines, but owing to the coarse fragmentation, productivity was low and secondary blasting costs high, with 700 gm/t of explosive used. As the secondary blasting was usually done with lay-on charges, the damage to drawpoints was extensive. The Shabanie and King layouts had individual orepasses for each drawpoint which meant that it was possible to set up good draw control.

On Andina mine, the approach was to reduce the number of orepasses to the transfer level by having connecting legs from each drawpoint. The result was that the legs could take as much as 800 to 1000 tons per shift. The drawpoints were worked in rotation, that is, drawpoints 4 and 8 might be worked on the same shift, but drawpoint 3 might only be worked 2 days later. Because of the high draw rate and the small diameter isolated drawzones, there was a tendency to isolated draw and early dilution.



The solution to the problem is to have short orepasses, a flow control at the drawpoint and a fairly uniform draw. Where this is the case, because of the close spacing of the drawpoints, the ore recovery is good and dilution is low.

The following diagram shows the grizzly layout that was used at Bell Mine, Quebec for many years. The undercutting is done from a raised undercut level and by retreating fans of holes which intersect over the major apex. The troughing along the length of the major apex removes lateral restraint to the top of the apex. It was abandoned in favour of a LHD layout when massive wedges flattened the grizzly drifts. A similar layout with the raised undercut drift, but with single sided drawpoints was used at King Mine in Zimbabwe. In retrospect, there would have been many advantages if an advanced undercut concept had been used by undercutting ahead of the finger raise development or if a stronger design were used.

DESIGN AND OPERATION OF CAVING AND SUBLEVEL STOPING MINES

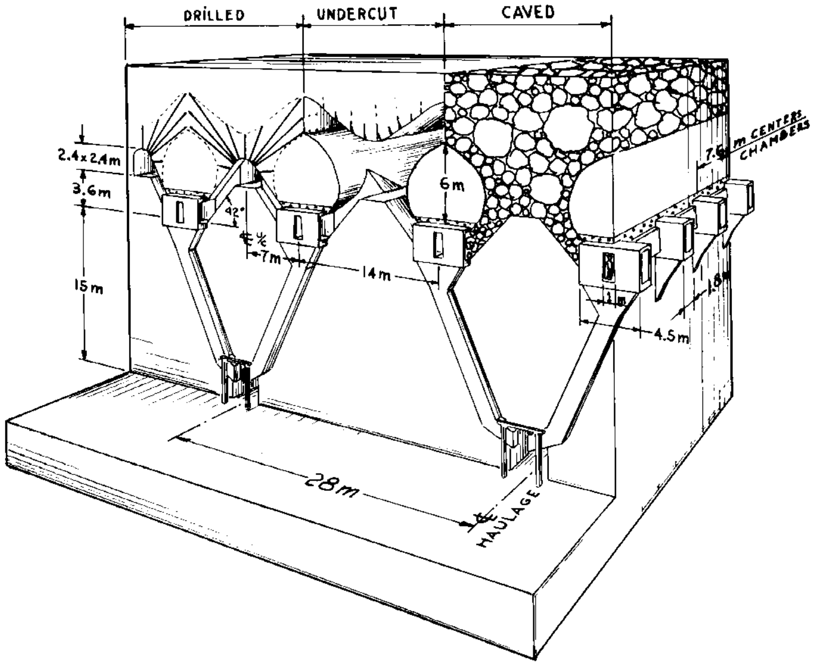
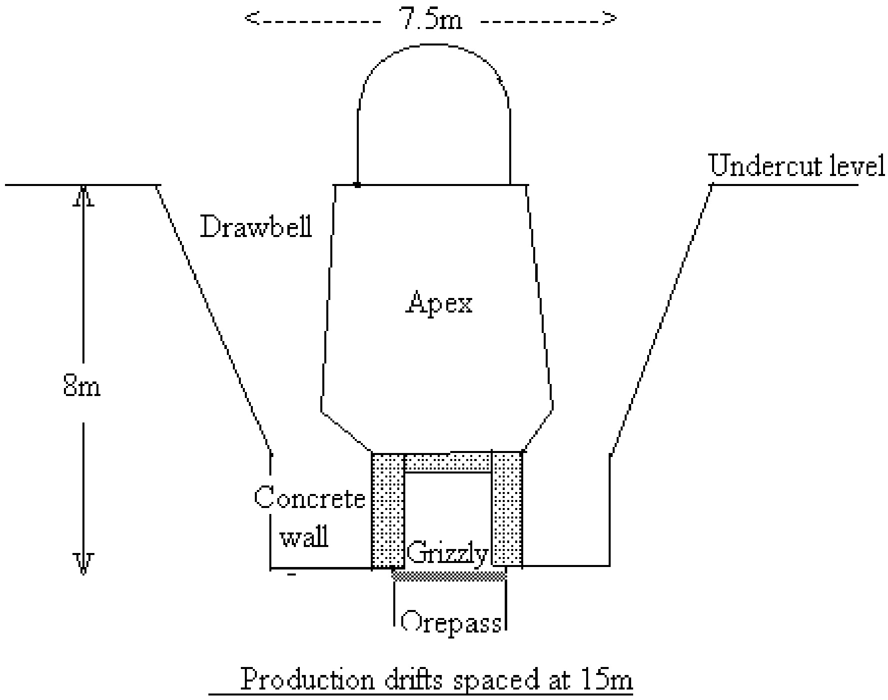


Fig. 1 CAVING STEPS AND DESIGN DIMENSIONS OF THE GRIZZLY METHOD

The conventional grizzly method used on the Chilean mines, consists of an undercut level at the top of the major apex and finger raises / cones from the grizzly drift that holed the side of the undercut drift and became the drawpoints. This is a strong structure as the major apex has good lateral support.



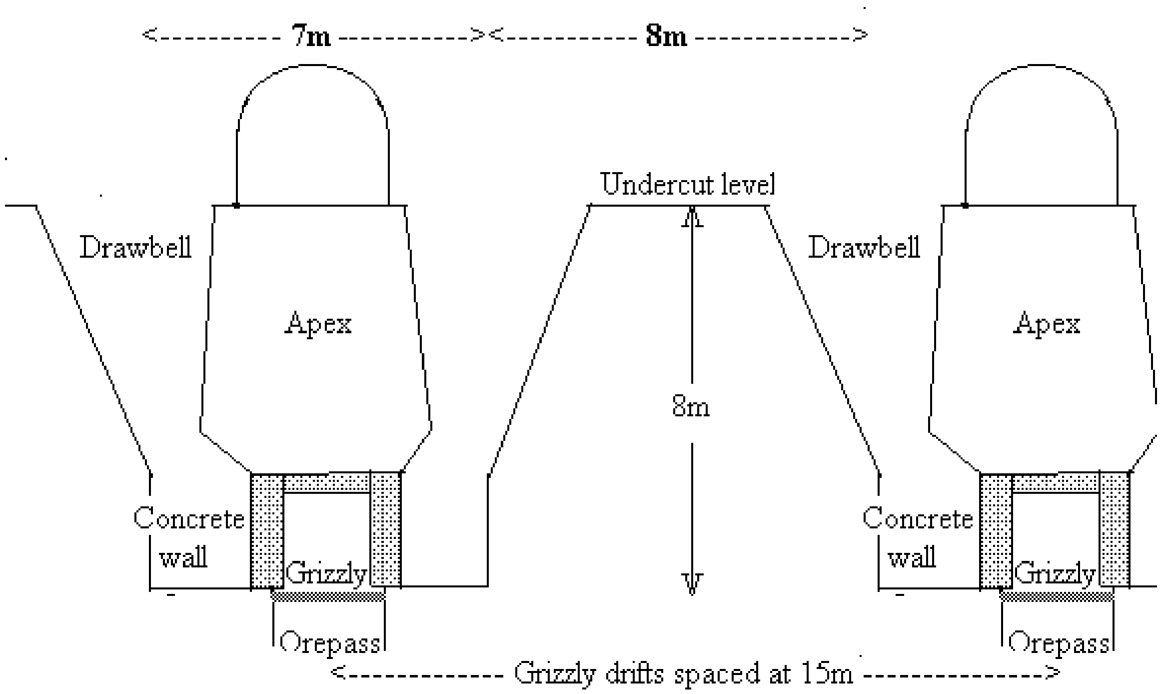
A grizzly layout was developed for Andina mine 3rd panel, to combine some of the design principles of a LHD layout with the low operating costs of a grizzly layout mining finely fragmented ore. In the 3rd panel at Andina it was possible to locate the grizzly layout in strong primary rock and to draw the 250m of overlying secondary ore. It is not often that that situation occurs. The big danger is to try to reproduce a layout like this in weaker ground - which would be the case in Andina when the next block is developed in secondary ore.

PRODUCTION STATISTICS

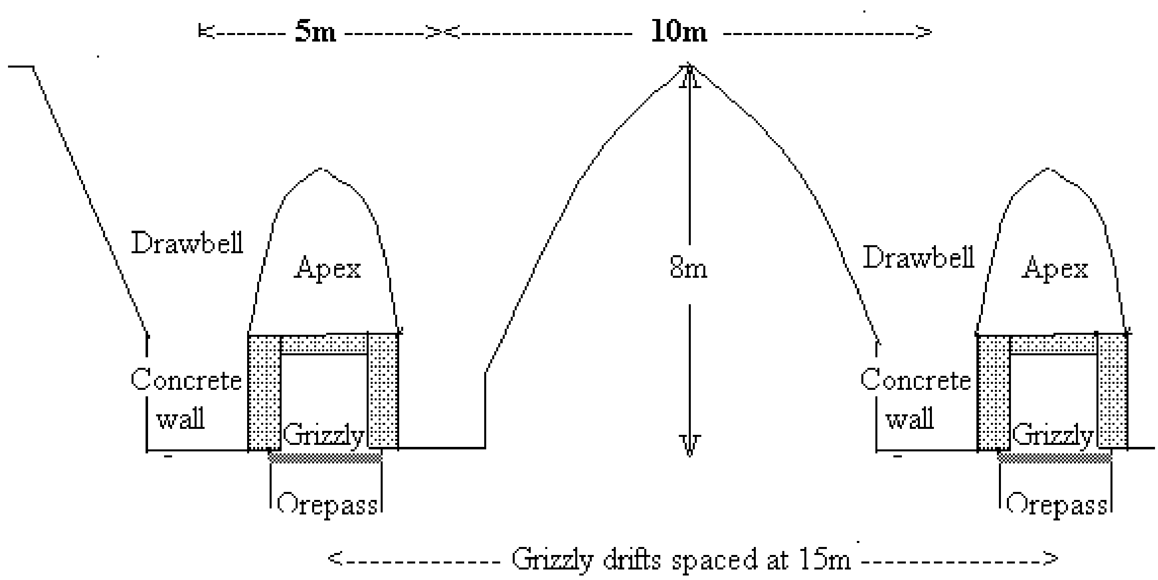
Mine	**Layout**	**D/P spacing**	**tons per day - tpd**
Andina	Diagonal	9.0m	20000
Teniente	Opposite	7.5m	15000
San Manuel	Opposite	5.0m	45000

DRAWPOINT SPACINGS

It is common practice to increase the grizzly drift spacing and the drawpoint spacing along the drift without looking at the drawzone spacing. The following two diagrams show the spacing at the start of mining and the changes after the brow has worn. This results in a change in drawzone spacing.



DRAWZONE SPACINGS AS DEVELOPED



DRAWZONE SPACINGS AFTER BROW AND APEX HAS WORN

Over the years there has been a tendency to increase the drawpoint spacings. Original layouts were 15' x 15' or 4.57m x 4.57m. These have been increased to 6m x 6m and then up to 9m x 9m with 7m x 7m being fairly common. By increasing the spacing in finely fragmented ground, draw control becomes very important and every effort must be made to ensure interactive draw. It is important that the drawpoint design is also changed when the spacing is increased.

PHOTOGRAPHS OF A GRIZZLY LAYOUT

These photographs were taken on a large tonnage grizzly operation which has experienced ‘weight’ problems owing to column loading.

Concrete brow in drawpoint

View down a grizzly drift in early stages of draw. The opposite drawpoints with boards in place to control the gravity flow into the central orepass

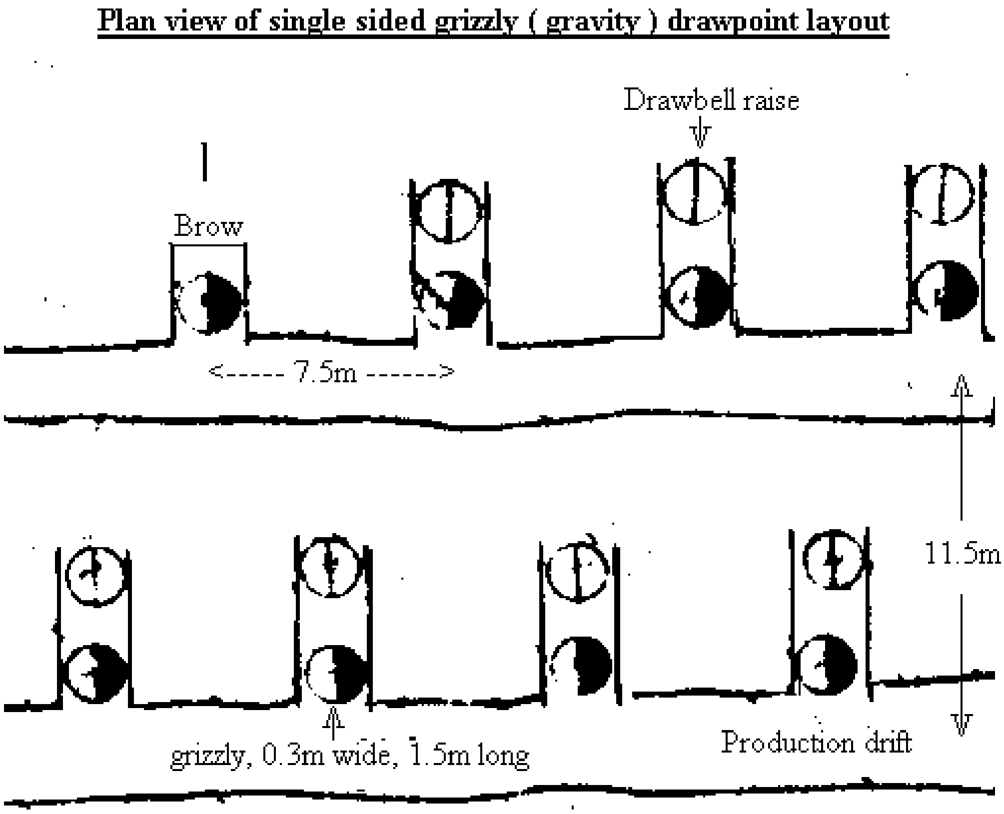
View down an orepass with grizzly bars on the top of the orepass

Well fragmented ore flow controlled by board in bottom left of photograph

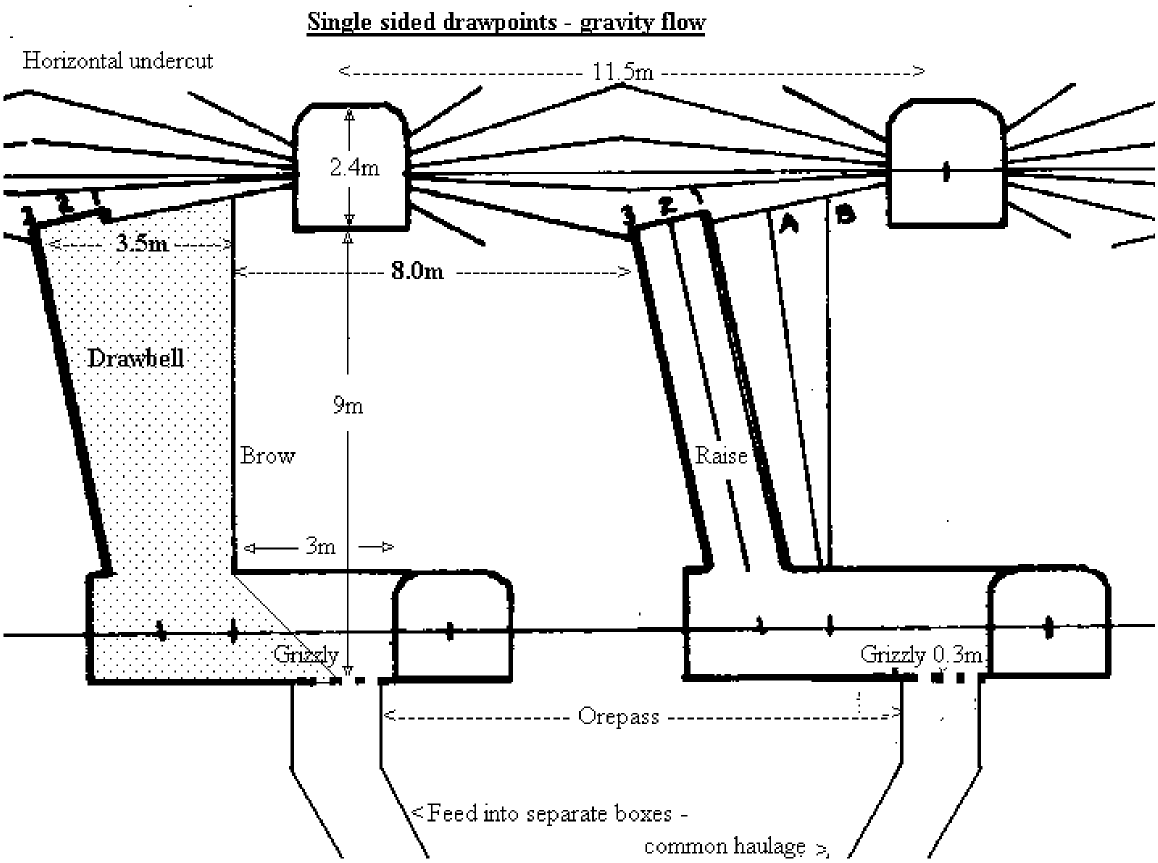
Concrete brow has failed and the steel repair beam is taking 'weight'

SINGLE SIDED DRAWPOINTS

Single sided grizzly drawpoints were used in the Western Section, King Mine so as to increase the size of the major apex with the intention of increasing the stability of the extraction level. The layout is shown in the following plan and section . Whilst the major apex increased in size the drawbell opening was small compared to the coarseness of the fragmentation. In retrospect the spacing of the drawpoints along the drift should have been increased and the drawbell opening increased. After all, the only control that the operator has is to be able to draw the cave material and these changes would have made the large rock more accessible.



Plan of Single Sided Drawpoints

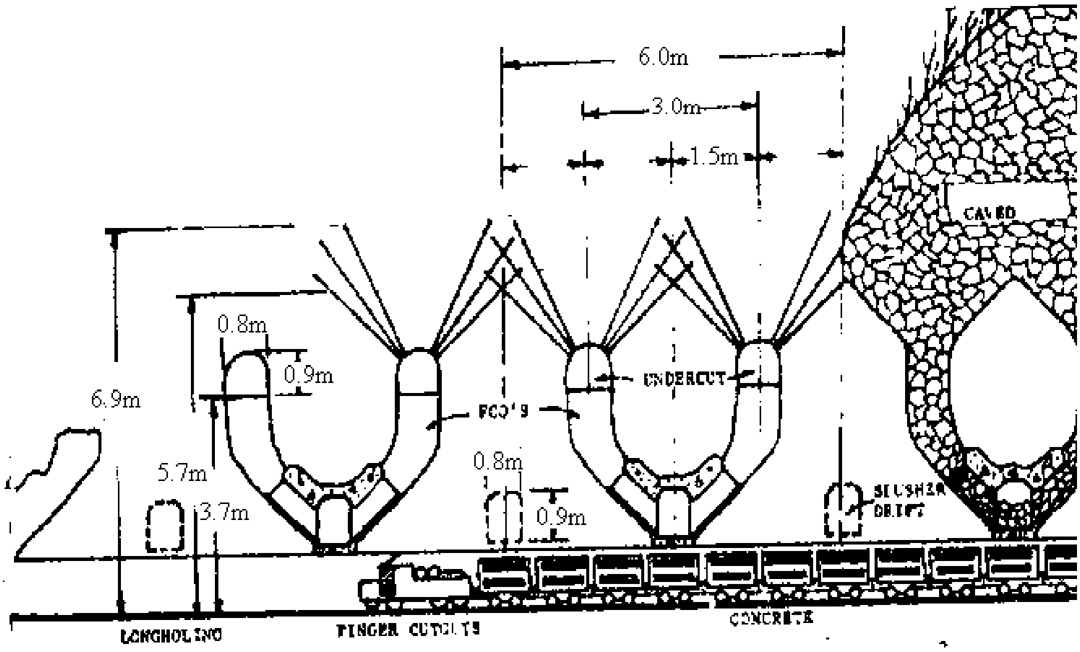


Section Through Single Sided drawpoints

SLUSHER LAYOUTS

Slusher layouts were and are still in use on many mines. The principle of moving the caved ore horizontally from a drawpoint into either cars or an orepass had limitations and the only reason why it was used, instead of the grizzly method, could have been because of the reduced development required. The production potential was a function of the slusher capability. Slusher layouts are not recommended as they are not highly productive or cost effective One of the big failings of the slusher method is the poor draw control when some drawpoints run freely and others are hung-up, which results in high dilution. In dusty operations it is difficult to see into the drift. High production is obtained from the nearest drawpoints. Controls are used to record the various distances that the slusher has been moved.

The following diagram is a diagrammatic section through a slusher layout showing the various stages in commissioning the block and the direct scraping into cars and hauling by train.



DIAGRAMMATIC SECTION THROUGH SLUSHER LAYOUT

DESIGN TOPIC

Induced Stresses

DESCRIPTION

The object is to identify all the mining induced stresses. This means mining the deposit on paper in order to have a 'feel' for all facets of the operation. Thus, this section will record the induced stress values for different areas based on the selected or examined layouts. The stress values will vary according to changes in size of excavations and the orientation of openings or cave front, with respect to the regional stress. There should not be any surprises at the end of the day.

REGIONAL STRESSES

The starting point is a good assessment of the regional stresses. The importance of ensuring that the regional stresses are realistic becomes apparent when the induced stresses are calculated and these values approach or exceed the rock mass strength.

NUMERICAL MODELLING

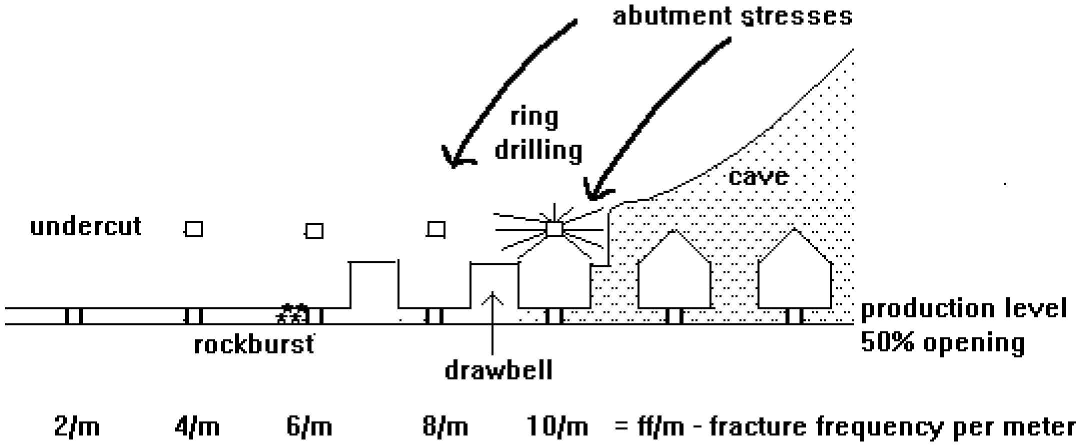
Numerical modelling plays an important role in calculating induced stresses for the different areas and situations that are likely to develop in the course of mining. If numerical modelling techniques are not available, then theoretical stress distribution diagrams provide useful information.

ABUTMENT STRESS

In order to induce a cave, it is necessary to undercut a large enough area of the orebody so that the rock mass fails and that the cave propagates. The stresses redistributed by this large opening are concentrated in the advancing abutments where the induced stresses are much higher than the regional stresses. The direction of advance with respect to the principal stress influences the magnitude of the abutment stress.

The effect of the abutment stress on the rock mass in the production level, reaches high levels when the abutment consists of pillars between drawbells and drawpoints as is the case with conventional layouts.

There is a multiple increase to the regional stress firstly, from the cave opening and secondly from the openings on the production level. The result of this is a failure of the pillars and sometimes rockbursts occur ahead of the cave front. Repeated ff/m measurements done on El Teniente mine in Chile showed that the ff/m as measured on the production level at regular intervals ,increased as the cave front advanced. The effect of the abutment stress and the pillar induced stress is to cause the sidewalls and brow to fracture, so that by the time the drawpoints are in production the condition of the rock mass has deteriorated significantly.



INDUCED STRESS ON THE UNDERCUT LEVEL

Because of the limited development on the conventional undercut level where the drifts can be 30m apart the abutment stress effects are not so noticeable, except at access drift intersections. As mentioned, the high induced stresses are on the production level where the stress relief occurs with rock mass failure. However, in the case of advance undercuts the development on the undercut level could be more closely spaced and the development on the production level limited, as a result the induced stresses on the undercut level would be higher as there would be no stress relief on the production level.

STRESSES ON THE PRODUCTION LEVEL

Abutment induced stresses on the production level are the main problem in a conventional layout. However there are also the uplift / relaxation stresses once the undercut has passed over with subsequent loading on the pillars once the cave has propagated. Uniform pillar loading is not a serious problem. It is only when columns loading occurs due to poor draw that the stress level exceeds the strength of the major apex..

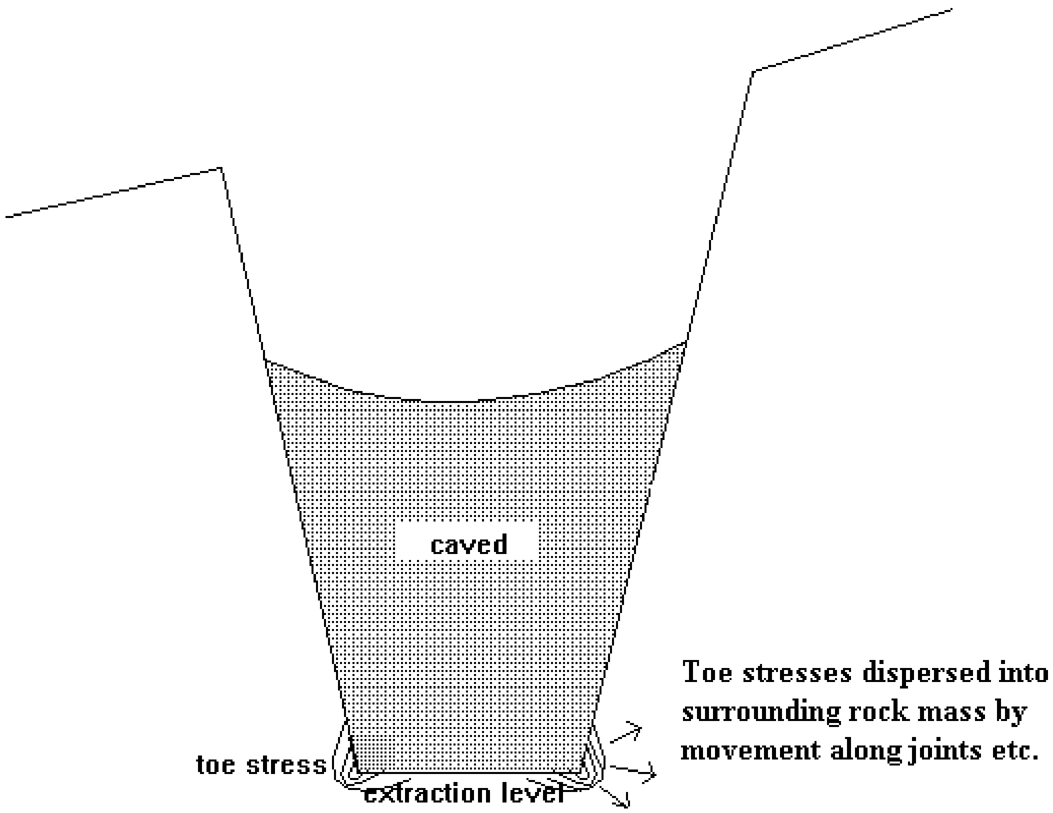
In high rockburst areas no rockbursts have been recorded on production levels overlain by a cave.

STRESSES BELOW THE PRODUCTION LEVEL

A low stress envelope forms below the production level, the shape of which will depend on the orientation of the principal stress. In this envelope damage is minimal. Below the envelope there can be extensive orepass failure and damage to ventilation drifts, pickhammer levels and transfer / haulage levels.

STRESSES IN THE PERIMETER - TOE STRESSES

It is important to distinguish between potential wedge failures and stress concentrations. Toe stresses are there for the life of the operation and it is necessary to distinguish between abutment and toe stresses. There is much field evidence on this subject and the field evidence suggests that toe stresses are not a major problem. The tendency is to overcut important installations and junctions as a safety precaution.



The following photograph shows a drawpoint in the toe of a 1400m high face and there is no real damage. The wear to the brow is normal for a conventional undercut and the abutment stress effects.

At Shabanie Mine modelling showed that there should be high stress effects in the perimeter of a 110m by 120m opening, but nothing unusual was noted, as stress relief occurred with very small movements on joints in the surrounding rock mass.

CAVE BACK STRESSES

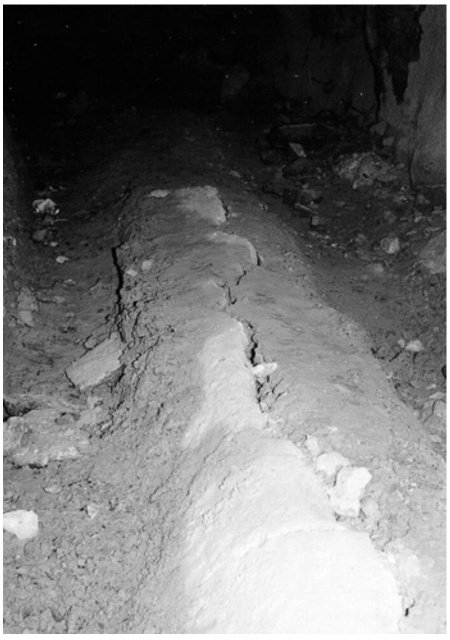
The orientation and magnitude of the induced stresses in the cave back have a major influence on fragmentation and cavability. Numerical modelling should be used to determine the stress fields with different orientations. The BCF program shows how the primary fragmentation varies with the orientation of the cave back stresses with respect to the structures.

DESIGN TOPIC

Roadways

GENERAL

The design of roadways is still a little controversial in terms of whether rails should be placed in the concrete floor, the quality of concrete, whether there should be two layers of concrete of different qualities or whether gravel will suffice. It is important to remember that the higher the quality the more difficult it is to achieve a consistent product and the end result is a variation which could result in failure of the weaker sections. Floor conditions and water are major factors in the design of roadways. Heaving floors can develop after the roadway has been installed.



Heaving floor - Cassiar mine

Cross section through a floor that has heaved

Extensive floor support as done at Cassiar mine did help prolong the roadway, but was an expensive operation.

Rebars in a grid and concreted in with 3m ropes into the floor

NORTHPARKES COMMENTS ON ROADWAYS

Currently 80 MPa steel fiber reinforced concrete with a 15 MPa binding and 25 MPa infill. The roadways have been wearing and breaking up due to wear in the drawpoints, former practice of dropping oversize from LHD bucket on to the roadway contributes to wear as does infiltration and ponding of water on the top of the binding and slabs. The 'slabs' are independent and hence rocking with the passage of LHD's over them (large slabs were used at Cassiar some 15 years ago and they did not work).

Northparkes use a 28 day cure time for the roadways. They have just gone to a lower strength 50 MPa concrete because of cost of rebuilds. They expect similar wear as with 80 MPa concrete. A couple of modifications which they believe will improve the life of the roadways are a tongue / grove link between the slabs (to stop rocking effect), a peak shaped binding to improve drainage i.e. high ridge in centre of drift so that water drained to sides and to use wax on top of this binding instead of polyurethane sheets they had used before - also to improve drainage. They seem to be having lots of roadway problems in the short life of the mine. They were not interested in rail reinforced concrete floors.

PREMIER MINE COMMENTS

In the original tunnels in BA 5 (about 1.5 km) they went to a lot of trouble and spent a lot of time in roadway support. They constructed a steel rail framework and bolted this into the footwall (see following photograph) and then filled in the framework with concrete of not less than 35 MPa . Overall thickness was seldom less than 500 mm. Once construction was complete the rails were flush with the concrete surface. LHD's put their buckets down onto the rails to clean the floor. Some of the roadways that we still use are over 9 years old and still in very good condition. They have also helped us control water effectively.

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

These types of roadways have been phased out, however, for the following debatable reasons:-

- Cost; these roadways cost around US$125 (1990) per square metre for labour. steelwork and concrete.
- The construction was time consuming and it was not possible to carry out other activities in the tunnel whilst the roadways were being constructed.

- Where footwall heave occurred the tunnel became impassable and the concrete had to be drilled out.

They subsequently approached the roads section of the CSIR and asked them to look into alternatives to steel reinforced concrete roadways (why not look at better design). Their conclusions were that we could make adequate roads by compacting aggregate. but. these would require ongoing maintenance, probably every weekend. Final costs would be the same. Areas where LHD's turned would require additional measures!!!

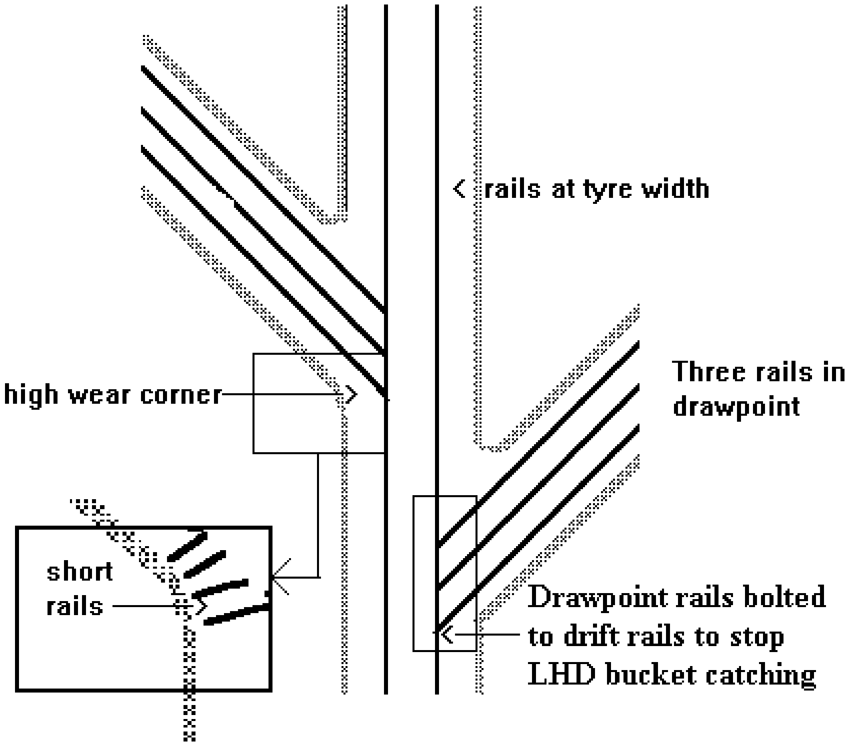
Their current strategy is to place aggregate on the roadways and compact this. (Why not use roller compacted concrete??) The roadway is then treated with a product called 'dust-aside', a hydro-carbon product from Sasol which binds the aggregate. This is done on a contract basis and before the undercut passes over. Once the undercut has passed the roadway is concreted with a minimum of 300 mm of 35 MPa concrete. There is steel rail reinforcement only in the drawpoints. These roadways need constant maintenance and lead to production losses. The initial cost of these roadways is about US$80 per square metre. There has been no calculation on the cost of maintenance, lost production, tyre wear and LHD costs

PLANNING ROADWAYS

A mine's approach to roadways can be a little confusing as one often gets the impression that they are a necessary evil so put down something as cheap and as quickly as possible. There is little thought given to repair costs, tyre wear and low productivity. Often because the people involved with construction have nothing to do with production. Good planning will ensure sound roadways and low operating costs. It is difficult to understand why some mines consider it necessary to depart from proven practice of massive concrete with embedded rails for roadways. Objections to rails can only be based on experience of poor installation.

At Henderson mine the roadways are basically very simple. The floor is not cleaned to bed rock and a 30 MPa concrete is used in both the production drift and drawpoints. Generally these roadways, with minor repairs, last for the life of the area under draw when approximately one million tons would have been drawn. Water is not a significant factor at Henderson. Henderson do not consider it necessary to install rails as they have had problems with the bucket hooking onto the rail - note this does not occur with proper installation. *A noticeable feature at Henderson is that the roadways are not cleaned to the concrete and that a layer of fine material is left on the roadway as it is felt that this protects the roadway*.

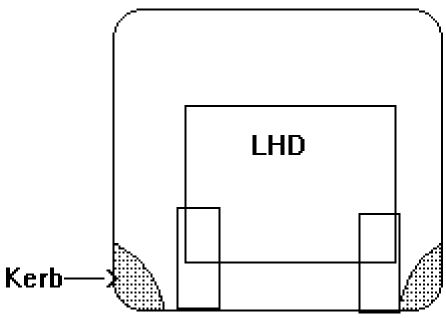
The measure of success of roadways are the floor conditions and the tonnage hauled before repairs and the ease of repairs as well as time in terms of loss of production.



Rails embedded in concrete to stop wear and to assist the cleaning of the drift.

Bell Mine spend large sums of money on their roadways with high quality concrete and a layer of reinforcing. The tonnage hauled between repairs was quoted as only 500 000 tons and this is a low tonnage for the amount of work put into the roadway. However, this figure must be related to the poor floor conditions on chrysotile asbestos mines, the problems being exacerbated if water is present.

A technique employed at Bell Mine in Canada to protect the sidewalls from LHD damage is to shotcrete a kerb during the final shotcrete stage:-



ROLLER COMPACTED CONCRETE

Roller compacted concrete has a role in the construction of roadways and is currently being used in the main ramp on the JM - Asbestos block cave. The concrete itself was in good shape, but the undulating profile of the completed floor was not that good - possibly a problem in putting the mix down on an incline surface with a low friction base. Roller compacted concrete has been proposed for many years as the answer to LHD roadways but, there has been a reluctance for mining companies to follow this up with large scale experiments. Apart from the simplicity in laying the dry mix the repair techniques are simple and the roadway is available for use in a few days.

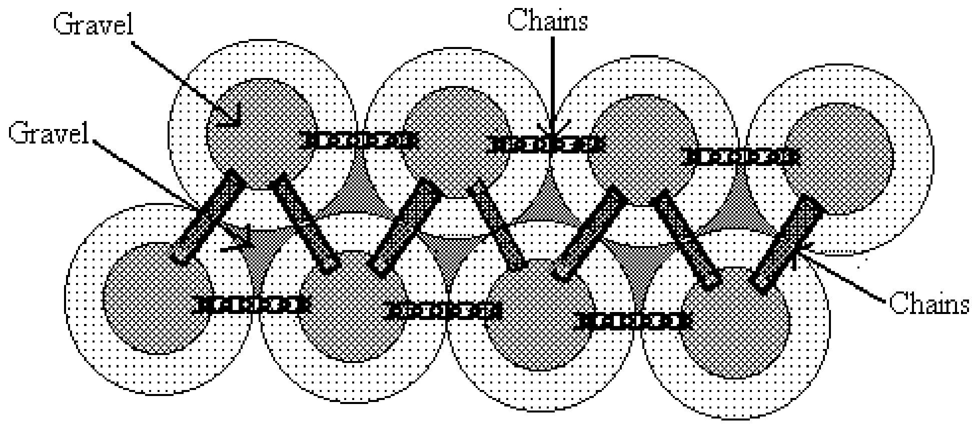
Roller compacted concrete is used extensively in timber yards in Canada where big vechiles with high axle loads are maneuvering all the time.

REPAIR TECHNIQUES

Once a normal concrete roadway fails, the repair technique is to use a quick set concrete. These areas always have a problem as production starts as soon as the concrete has hardened, but not at it 's full strength.

GRAVEL IN LINKED TYRES

A roadway surface consisting of discarded tyres linked together and filled with gravel has been developed on Australian coal mines to provide a stable surface where the floor is very soft. This technique could have application on other mines where soft floors occur. The system could be used during the development stage and then the permanent roadway laid. If the mine is subjected to continuous floor lift then this system could be used on a well reinforced floor. Removal of excess material, resulting from excessive heave, is simple as the tyres can be easily moved.



Tyres joined with chains and filled with gravel on weak floors

INTERLOCKING BRICKS

These have been used by African Associated Mines for many years. They were first developed at Havelock and then introduced to Gaths, King and Shabanie Mines. The bricks were originally 150mm thick and have now been reduced to 100mm and found to be effective. However, the LHDs are usually small in the 2yd to 3yd category.

Interlocking bricks on King mine, Zimbabwe

The advantages of interlocking bricks are:

- The brick concrete fully cured and good quality being made in controlled conditions on surface.
- Simple to lay and immediately available for use.
- Do not require curing time before being driven over.
- The floors are readily repaired in the event of floor heave or potholes.
- Good traction surface for LHDs.
- Bricks are reclaimable particularly in SLC type layouts for reuse.

The disadvantages are:

- The bricks are time consuming to demould.
- Transport and Laying are labour intensive.

Method of Construction

1. Suitable drain holes established according to plan.

2. The floors are canted so that water will drain off on one side of the drive. A difference of 0.1 m across the drift.

3. Lash the floor to 150mm (for 100mm bricks) below the required final floor level creating an even uniform base.

4. The floor is then compacted using a plate compactor. Should the area be wet the spoil is first stabilised with cement roughly mixed in at one sack per 6 metres square of drive.

5. Place approximately 50mm of sand on top of the compacted rock left in situ, again using cement to stabilise if the area is wet. The sand is then compacted and levelled to form the final base to the elevation required.

6. Lay the first row of bricks with the 'noses' towards the low side of the drift to form the drain or to the edge of the drain line, if one is to be constructed separately.

7. Lay the second row and successive rows in turn interlocking the bricks correctly.

   a. **N.B.** Odd numbered rows have 'noses' towards the drains and even numbered rows have 'noses' away from the drains. The bricks are laid so that each interlocks with those in the adjoining row.

8. Once the area has been laid completely, sand is packed into the joints between bricks. Between the bricks and the sidewalls or drain, concrete is poured. Across the end of the section of flooring, laid sand and rock are compacted into place to form a ramp for the LHDs to traverse minimising the damage to the bricks.

9. At loading points rail, reinforced concrete is used, the concrete locking into the final line of bricks.

10. Brick floors in ramps are constructed in the same manner but at every 10m a concrete lintel is cast 0.3m wide and pinned to the floor locking the bricks against creep.

GOOD HOUSEKEEPING

The Benefits can be listed as :-

- Great improvement in safety
- LHD costs are reduced by as much as 50% and production efficiencies are increased owing to:-
  - a reduction in tyre wear,
  - a reduction in wear on the center articulation,
  - increased tramming speeds,
  - a reduction of sidewall impact (affects machine and support).
- Improved working environment, resulting in improved efficiencies.

Comments From N.J.W.Bell

The importance of good roadways

Without a doubt money spent on good roadways is returned multiplied owing to the improved productivity obtained from the LHD’s. This is owing to higher speed and the reduction in maintenance, both on the tyres and such things as the centre articulation, which are costly items and could lead to considerable down time.

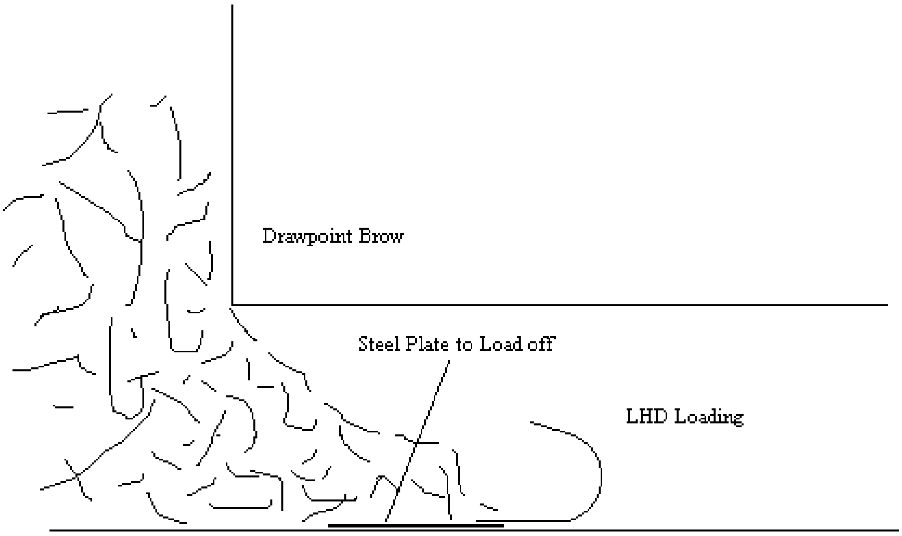
On good roads there is less chance of sidewall strikes which again do tremendous damage to machines. There is also the matter of driver comfort, which again leads to greater productivity.

Good initial roadways are safer, easier to housekeep and maintain - overall lower cost/tonne.

Several roadway variances have been tried over the years and each has its difficulties. The selection of good roadway material will undoubtedly depend on the actual mine in question.

Some of the options are:

ROADWAY TYPE	ADVANTAGES	DISADVANTAGES
Conventional concrete	A good finished surface that can be well controlled to give a strong floor in intermit contact with the hard rock beneath. Can be reinforced with weld mesh panels and protected from LHD digging by inclusion of rails.	Requires a minimum of seven days curing time with no traffic on it, so the end is not available for other work during this time and the floor should perforce be dug out to solid or the material left on the floor properly compacted before the final concrete is poured. Repairs are very difficult as the concrete has to be dug out and a further seven days waited for the replacement concrete to cure.
LHD Interlocking Roadway Bricks	Readily laid on a compacted floor, which does not have to be dug to solid. No delays to travel. Readily repairable in case of damage e.g. floor lift. Concrete is guaranteed to be of good strength to stand the wear and tear of the LHD travel	Manpower to install the bricks. The costs of the bricks and their moulds etc, which are very high. Can't be used in drawpoint loading areas without concrete or steel plate.
Compressed Bricks – e.g. G pattern, cubic or other varieties.	Same as the interlocking brick, except easier and cheaper to make.	Same as the interlocking brick, but as laborious if not more so to place and tend to tear out on corners.
Reinforced Concrete with rails/RSJ's	As for concrete floors, however the rails have to be put in place before hand and this increases the costs. However, it does improve the wear characteristics and maintenance. These are particularly recommended for loading areas where the wear and tear of the bucket into the floor, if not done, can be horrendous leading to large holes being dug in the floor	If footwall heave occurs the whole floor lifts and access into the area can be prevented or huge damage to tyres results.
Roller compacted concrete	If the logistics can be overcome and controls put in place this could well be a very useful material for initial floors and for repairs of existing concrete floors or other floors.	Underground this has proved difficult to install as the controls have to be exceptional to make the concrete to the required strength.
Steel Plate in the loading area	These have been successfully used in development and trials are to be conducted at King in the next set of draw points to see if this idea has merit.	See sketch below.
Run of Mine Gravel	Easy to lay	High rolling resistance to vehicles. Maintenance costs ongoing and higher. Material must be suitable.



<u>Roadway maintenance</u>

It is imperative that roadways are maintained and kept clear of spillage.

DESIGN TOPIC

Pre-break / Forced / Induced Caving

GENERAL

Pre-break or forced caving is often attempted in a more competent zone which is required to be caved. It is mistakenly thought that if the lower section is pre-broken, a high productivity will be achieved at the commencement of the operation and continuous caving occurs thereafter. Unfortunately, in more cases than not the forced cave is not successful and ends up by being far more costly than if a straight cave had been employed.

The problem is, that if costs are going to be kept down then the forced cave calls for a long hole mining method suited to a stable mining environment. This is a misnomer as the purpose of a caving operation is to create an unstable environment above the extraction horizon. The only safe method is sub level caving -SLC -, however this is an expensive method compared with the cost of caving coarsely fragmented material. The reason given for a prebreak is that a large tonnage becomes available at an early stage. This is not entirely correct. Compare the cost and time taken to develop the pre- break levels, complete the blasting and develop the production level, with a straight cave. Although the fragmentation is coarse it will be noted that the straight cave produces tonnage at an earlier date and at a lower cost. After all, the pre-break has to undercut sufficient area to ensure a propagating cave and in a high stress area it has to act as an advance undercut The only time it is rally applicable is if the production level is in very good rock overlain by a high column of good caving ground as at the contact between primary and secondary rock in Chile.

Also the mine personnel are deprived of the learning curve in how best to deal with the coarse fragmentation during the build up in production. With a pre-break easy tonnage is available, followed by a rapid change to when it is more difficult to maintain the tonnage call.

The Northparkes example clearly shows that the method was costly and time consuming and achieved very little in the long run.

LAYOUT, LEVEL INTERVAL, DRIFT SPACING

Various methods have been used and suggested to prebreak the competent zone. Sub level caving has been used on several operations and is possibly the best method. Level spacing and drifts would be at the maximum possible spacing to ensure a full break, e.g. 20m level interval and 15m drift spacing with drift dimensions of 4.5m x 4.5m. The connection from drawbell to the pre-break has to be carefully designed. All the correct procedures must be maintained to ensure good brows, no verandas form and that there is continuity of mining. When pre-break drifts are developed far apart to keep development costs down, then large volumes of rock are broken per blast. If the same approach is used on the undercut level then those large blasts into drawbells have resulted in blast damage to the major apex and drawpoint pillars, this rather defeats the object of maintaining the integrity of the rock mass.

The lead - lag between drifts must be kept to a minimum particularly as the hydraulic radius is approached.

LOCATION OF SLOT AND DIRECTION OF BREAKING

The location of the slot is very important. A central slot might appear to be the best arrangement in terms of having more ends, but it might not be the best location in terms of cavability of the overlying rock mass. After all, the object is to produce the bulk of the tonnage from a block cave. The layout must be designed and mined in a direction that is going to give the optimum caving and fragmentation. If there are no other governing factors, logical place to start is in the high grade zone.

BLASTING METHOD, DRILLING PATTERNS AND AIR BLASTS

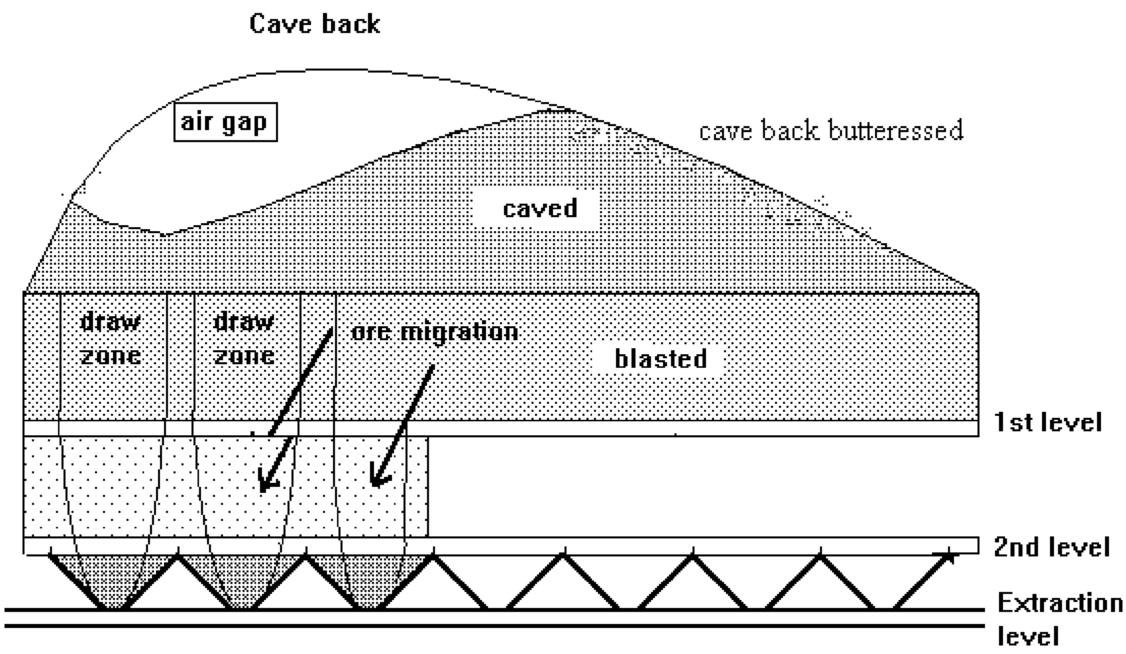
The blasting method must be well designed, as heavy blasting of the ground between the pre-break and the drawbell, could cause damage to the rather frail apexes of the production level. As one of the objectives is to mine a large enough area to ensure that the overlying orebody has caved, precautions have to be taken against air blast damage.

Whilst the blasting can be done to void conditions at the start, once the undercut area approaches the hydraulic radius and back failure has not occurred then the ends have to be left full of broken rock and any other openings sealed off. Thus if the drift have been widely spaced, problems might occur when blasting is done under choke to semi-choke conditions.

A mass blast was used at Shabanie mine to pre-break a large tonnage. Sufficient space was mined and the blast was well timed so that the breaking was effective and there was little damage to the underlying development. However, it was a far more costly exercise than the normal straight cave. Long hole crater retreat blasting was also used to create a well fragmented rock mass. The blasting was expensive, but the main problem was that relaxation occurred behind the high faces so that there was opening up along joints with the result that large blocks were not broken, but moved into the blasted rock mass. The result was high secondary blasting costs in the drawpoints and extensive damage to the drawpoints.

DRAW CONTROL

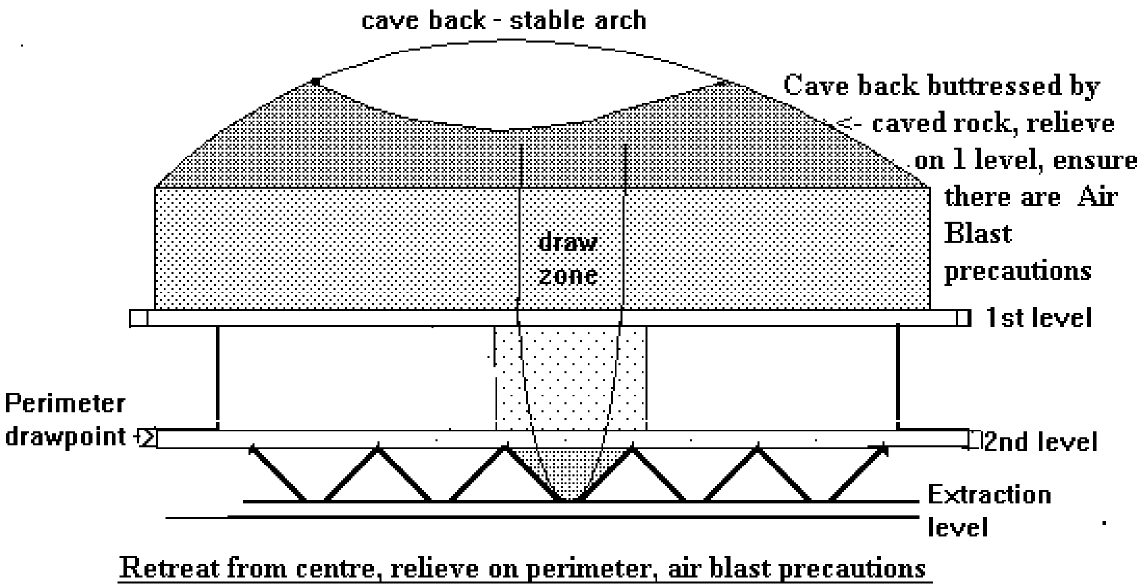
The tonnage blasted must be noted as well as the tonnage drawn to relieve swell, so that draw control records can be kept. A pre-break can have an adverse effect on draw control because of the high column of broken rock and the need to keep this moving so as to propagate the cave.



Migration of material remote drawpoints

In the early stages, until all drawpoints are operational and the cave is propagating the drawzones will tend to be isolated with movement of material from high pressure to low pressure areas. This means that material will report in drawpoints from areas out of the normal zone of influence of those drawpoints and the reallocation of tonnage becomes impossible. This situation will worsen if the cave back arches and stabilises because of buttressing by the broken / caved ground, as there is no means to pull tonnage until the drawpoints are developed. A better result will be obtained if the mining were started in the centre and worked out wards to the flanks. This has the advantage that the block is commissioned more quickly and drifts on the 2nd level can become perimeter drawpoints to assist with the cave propagation.

In the following case the cave back is supported by the broken rock and a stable arch has formed.



WEDGE FAILURES

Because of the scale of the operation there is limited flexibility in coping with large potential wedge failures. The presence of potential wedges must be noted and the technique adapted to cater for this situation.

BOUNDARY WEAKENING

Boundary weakening by blasting high slots along the side of a cave to cut off clamping stresses is a proven and successful technique. The location of the slot with respect to the high horizontal stresses must be planned from the geotechnical data. The height of the slot must be such that:

- Sufficient tonnage is made available so that subsequent intermittent caving above will meet production requirements.
- There is a change in the rock mass with height which will result in complete cave propagation.

Boundary weakening by blasting a pre-split is not a successful technique as vertical pre-splits are no different from vertical major joints and are therefore clamped in the same manner.

HYDRAULIC FRACTURING

Hydraulic fracturing was attempted in 1968 on Shabanie Mine to induce the cave over block 16. Boreholes were drilled into the back and water pumped into the holes under pressure using a cementation pump. The system was rather crude and was not successful. This technique was suggested for Northparkes, where, with more advanced technology it has been possible to generate high pressures and the failure of zones of the jointed rock mass was achieved.. The impression is that if the back is fractured and close to caving then hydraulic fracturing is a method that can induce some or all caving. However, the south-west corner overhang, where the structures are clamped and which was inhibiting the overall caving, hydraulic fracturing has not been successful.

There is hope that where the rock mass is competent or that there is clamping of structures and the 'footprint' of the orebody is similar to the calculated hydraulic radius, hydraulic fracturing will induce and propagate the cave. Until the method is proved, this approach cannot be contemplated as failure of such a costly exercise means the added expense of trying to recover the situation with drill and blast methods. It has also been suggested that where caving is intermittent, that hydraulic fracturing could be used as a production tool to generate caved material at the planned production rate. If high draw columns are taken into consideration then the amount of drilling required to give the right coverage for a doubtful technique is enormous and cannot be justified. It is far better if the money were spent on a mass drill and blast situation.

ASSESSMENT

It is suggested that pre-breaking ore to initiate a cave or to provide easy tonnage at the start of operations is a fallacy as the cost and time involved can be compensated for by the early availability of drawpoints and the provision of good secondary blasting equipment.

DESIGN TOPIC

Excavation Stability

GENERAL

This section summarises the factors that can affect the stability of the excavations and discusses the failures that various caving operations have experienced   The rockmass strength is known or should be known, not as an average classification rating, but for distinct rock mass rating zones. Whilst it might appear to be more economical to design support for different rockmass zones, practical problems could be encountered in maintaining the required standard when changing from one support design to another over short distances (*it is difficult enough in getting the right standards with one support method).* The best procedure is to accept that the strength of the chain is based on the weakest link.

INDUCED STRESSES

The induced stress values are required for the specific areas if there are likely to be significant variations. The emphasis is on the correct design and undercutting sequence to reduce the induced stress effects for the bulk of the development. However, the effect of induced stresses, particularly the abutment stresses, will depend on how much development is done ahead of the advancing undercut.

STRUCTURAL CONTROL

Major structures in the drifts will require a high level of support once the potential failure pattern has been established. A major problem can occur if or when major structures form massive wedges. Experience has shown that these wedges can lead to uncontrollable collapses. At Havelock Mine, mining continued in a grizzly drift at the base of the footwall wedge employing a continuous repair cycle using yielding steel arches. This effect was localised to the base of the wedge because the adjacent grizzly drift on the hangingwall side suffered little damage.

ROCKBURSTS

The rockburst potential is a function of the abutment stress, the number of openings and variations in rock type. With advanced undercutting techniques the rockburst problem on the extraction level can be reduced, but, precautions must be taken on the undercut level, particularly at junctions with access drifts. In high stress areas the rapid advance of the undercut or cave propagation may give rise to seismic events which can result in rockbursts hundreds of metres away in highly stressed pillars or at contacts of rocks with different moduli.

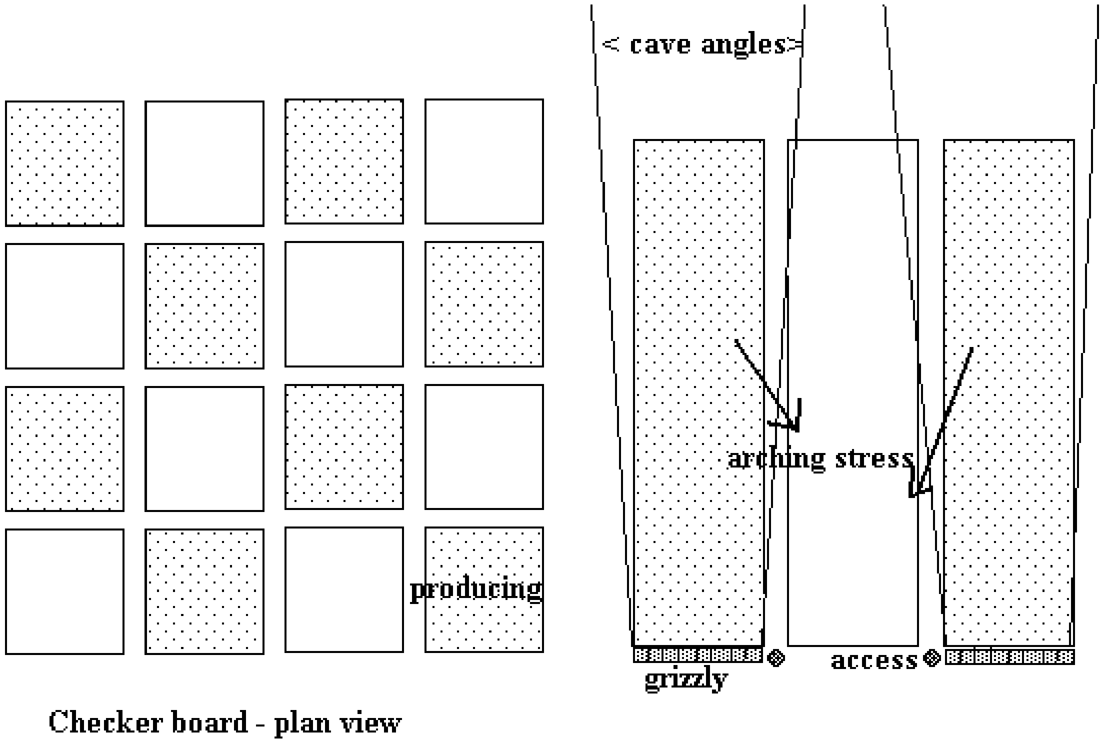
MINING FACTORS

Excavation size - It is hoped that the excavation size has been based on good geotechnical investigation and not a design based on the largest LHD available.

Drilling and blasting - Sound drilling and blasting techniques are required to ensure minimum damage to the rockmass.

Support -The support has to be designed for the life of the operation which is a function of draw height and rate of draw. The critical and most difficult area to support in a caving operation is the drawpoint brow. Every effort must be made to preserve the rock mass and to support it in the best possible manner, whether it be rock reinforcement or drawpoint linings. It is pointless installing widely spaced cable bolts in the brow in a highly jointed rock mass, drift rock reinforcement and a strong lining are required.

Mining sequence -The mining sequence must recognise the above factors so as to minimise damage to the extraction level The checker board system was used on mines using the grizzly method e.g. San Manuel. The first (primary) few blocks did not give trouble, but, once it was necessary to mine the remaining (secondary) blocks, major collapses and squeezing were experienced owing to the settling of the solid column and increased arching stresses on the unmined ground

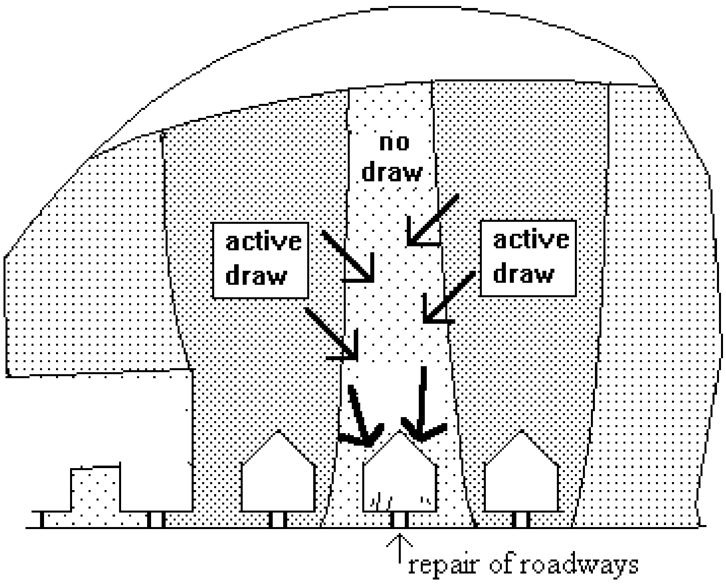


This method was eventually discarded for a panel retreat system. The consequences of any sequence must be carefully analysed, because what might appear to be most satisfactory for an early high grade recovery could lead to very high repair costs.

At Shabanie mine in block 4/5 the lower down dip section was mined after the upper section and then the middle section was mined. This meant that the middle section had no down dip support causing it to move down on the footwall talc which resulted in extensive squeezing of the grizzly drifts. This meant continual repair cycles and the introduction of yielding steel arches which proved to be extremely useful in controlling the deformation and made it possible to mine all the ore under adverse conditions.

COLUMN LOADING

Column loading is a major problem that can occur during the life of the operation, and can flatten drifts if not corrected. Sound draw control is the answer, however, if there are indications that the apex is being loaded the solution is to increase the draw in the affected area to relieve the 'weight'. Column loading at one of the Kimberley mines led to the failure of the slusher drift with it's one metre thick , 60MPa concrete lining and an underlying drift 15m below.



Weight of broken ground plus the arching stresses of the adjacent drawzones imposes a significant load on the major apex and thus on the pillars of the extraction horizon.

MAJOR COLLAPSES

Large areas at Teniente Mine have been affected by collapses which cover up to twenty drawpoints, representing a significant tonnage loss. The following photograph taken in the production level of a collapsed area shows the back to be in a good condition, but the sidewall / pillars have failed.

In the conventional layout it is common for the pillars to show a high degree of fracturing after the undercut has passed, these ’pillars’ only represent 50% of the solid rock on a production level, the fracturing most likely means that the solid core of the pillar is about 20% of the area and hardly sufficient to withstand any traumatic event A combination of poor draw or a seismic event could set up a domino effect.

ASSESSMENT

This section summarises the factors that can affect the stability of the excavations and discusses the failures that various caving operations have experienced The rockmass strength is known or should be known, not as an average classification rating, but for distinct rock mass rating zones. Whilst it might appear to be more economical to design support for different rockmass zones, practical problems could be encountered in maintaining the required standard when changing from one support design to another over short distances (*it is difficult enough in getting the right standards with one support method).* The best procedure is to accept that the strength of the chain is based on the weakest link.

DESIGN TOPIC

Extraction Level Support

GENERAL

In areas of high stress, the rock mass will respond by plastic deformation in the weaker ground and by brittle - often violent - failure in the more competent ground. If there is a big difference between the RMR and the MRMR ratings then yielding support systems are required. Pre-stressed cables have little application in underground situations unless it is to stabilise fractured rock in a low stress environment. If cables are installed ahead of the major excavation they become stressed, as the induced stresses increase. The need for lateral restraint on the rock mass cannot be too highly emphasised.

A factor that is often ignored is the long term economics of installing the correct support at an early stage, as opposed to the attitude of expediency or convenience of contractors to have the ideal development cycle time. The cost of the support must be related to the tonnage mined and to the ability to mine safely in that environment. Only on the undercut level can the term 'temporary support' be used to describe the support, recognising stand-up time and time dependant failure. However, with advance undercutting the number of drifts can be increased with resultant higher stress levels.

An extraction level with a DRMS of 15 to 30 MPa that will draw 15000 tons per drawpoint, can be used as an example. The support that would have been installed in this environment would have been rockbolts, shotcrete and TH (yielding) arches or rockbolts and concrete at a cost of $ 0.70/t. Failure of these systems would lead to repairs that would usually involve sliping and installation of replacement arches at reduced spacing, as well as loss of production. At King mine, Zimbabwe, the direct cost of rehabilitating extraction areas has been assessed at $10000 per drawpoint or $0.67/t, excluding the cost of the loss of production. If the same area had been comprehensively supported in the first instance the support cost would have been an overall of $0.83 instead of the actual $1.37.

The repair costs are lost in working costs and are only highlighted when requested, the indirect costs of loss of production are never available, but are appreciable. The chain effects of poor initial support is to affect other areas which should not have failed. In fact, areas are sometimes abandoned with high ore losses. Feasibility studies must reflect the correct support costs. This is not an area to cut costs.

For support to be effective, it must be economically and safely designed for the life of the excavation taking into consideration the stability or instability of the surrounding rock mass. The stability or instability of the rock mass is defined as the quantitative unconfined rock mass strength in relation to the mining environment stresses. The rock mass strength is derived from the geomechanics ratings after the necessary adjustments have been made. The mining environment stresses are derived from the field stresses with allowance for the mining induced stress. The object of support is to increase the unconfined rock mass strength so that it will match the mining environment stress. Where the difference is too great, the deformation can be controlled within the period of mining - in which case the rate of extraction could be increased. The timing of the support installation must be such that the rock mass is not allowed to fail and therefore, the installation should be early rather than late. The importance of this last sentence is still not appreciated by many operators.

A support system should be designed and agreed to before the development stage so that there is interaction between components of the initial and final stages. The initial support will be installed concurrently with face advance to control deformation and will preserve the integrity of the rock mass. The final support is to cater for induced stress and stress changes that result from the mining operation. An integrated support system consists of components that are interactive and the success of the system depends on correct installation and the use of material of the right quality.

EXCAVATION STABILITY ASSESSMENT

An assessment of the long term stability of the excavation is required at an early stage, preferably before the excavation is developed so that the support can be designed and installed at the earliest opportunity:

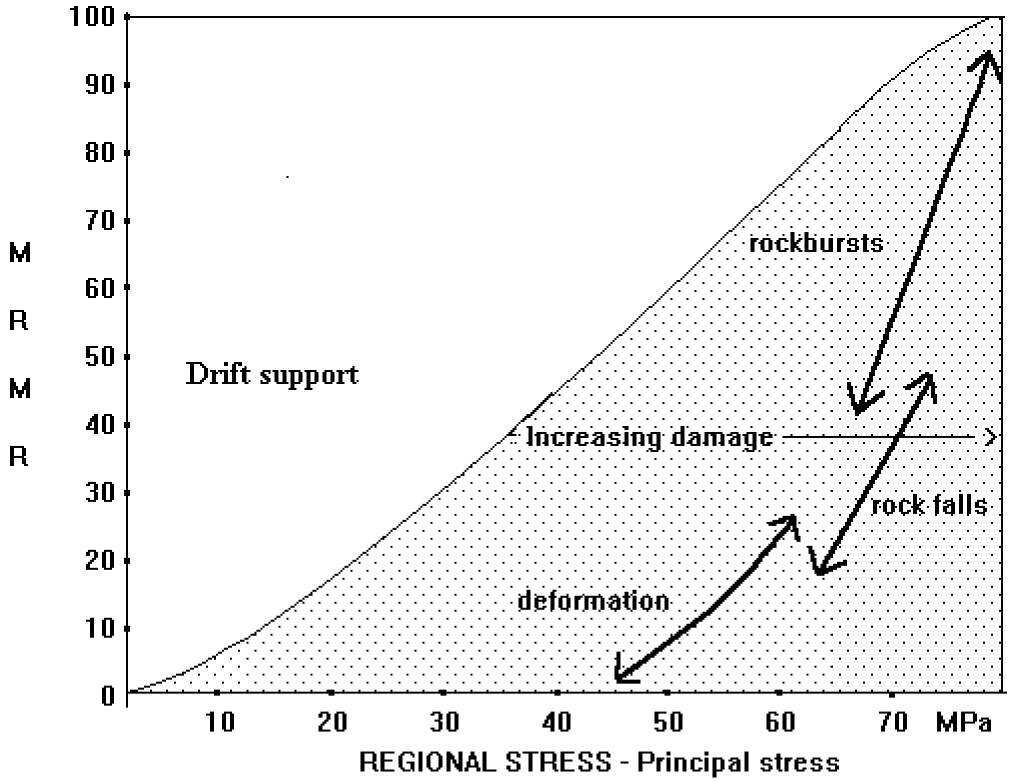


Diagram stability relationship between MRMR and the regional stress

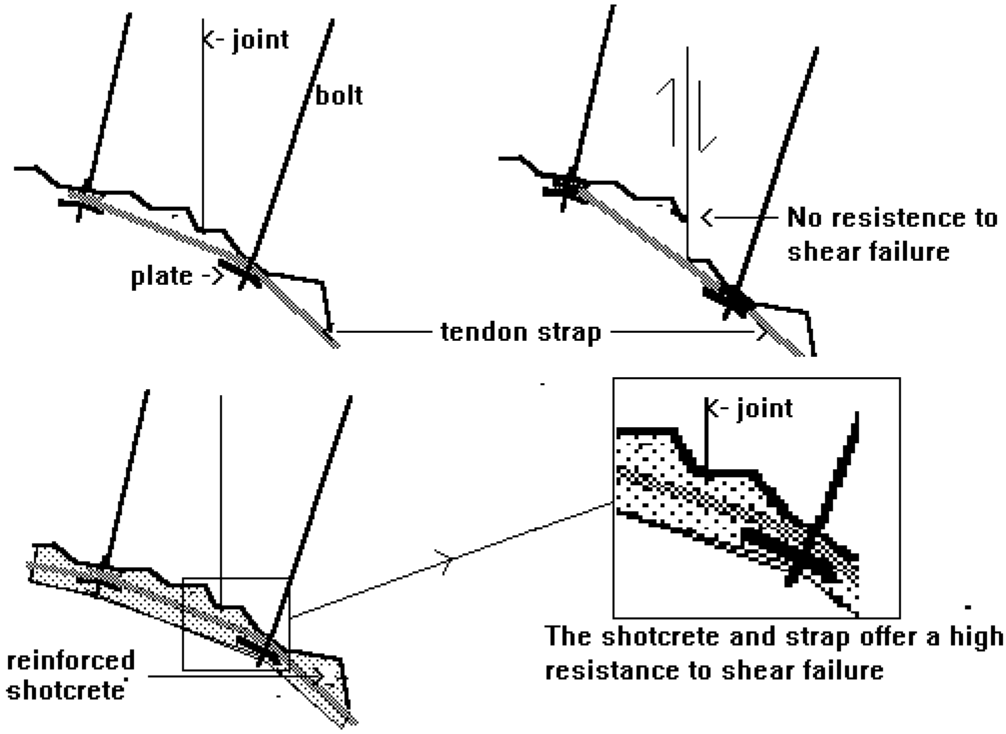
ROCKBURST POTENTIAL

Cave mining is taking place at greater depths and often below large open pits. The induced mining stresses can be high and therefore, these mines can expect to experience rockbursts unless precautions are taken. Where the rockburst potential has been identified**,** yielding support systems are required. Rockburst support proposals are in the Rockburst section in a paper by Stacey and Ortlepp.

ROCK REINFORCEMENT CONCEPT

Rock reinforcement systems are sophisticated and only achieve 100% support pressure when all components are correctly installed and are interactive. If the support system consisted of bolts, plates, mesh reinforced shotcrete/fibercrete and straps, then the percentage increase in support pressure would be:-

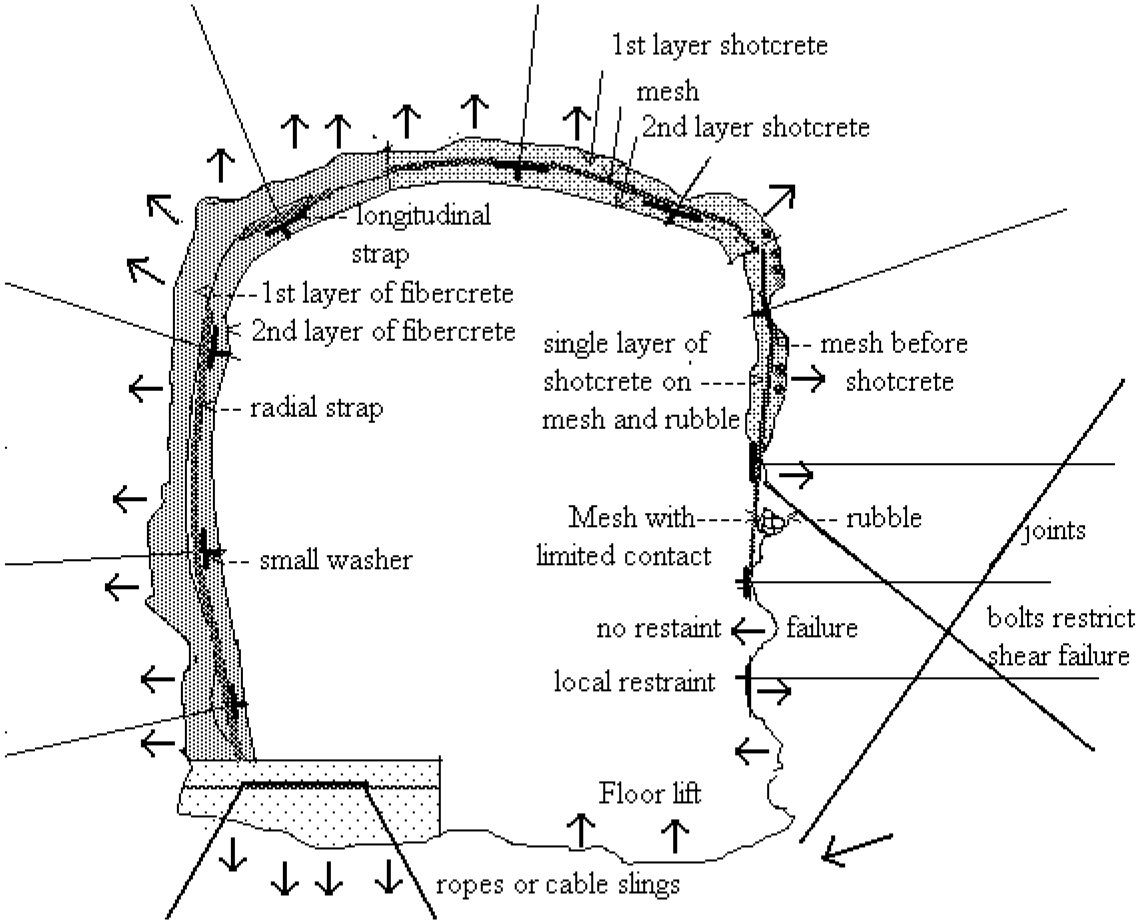
Item	Accumulative % Support pressure
Bolts	25%
+ Plates	30%
+ Mesh	35%
+ Straps	40%
+ Shotcrete	100%



Role of shotcrete in completing the rock reinforcement system

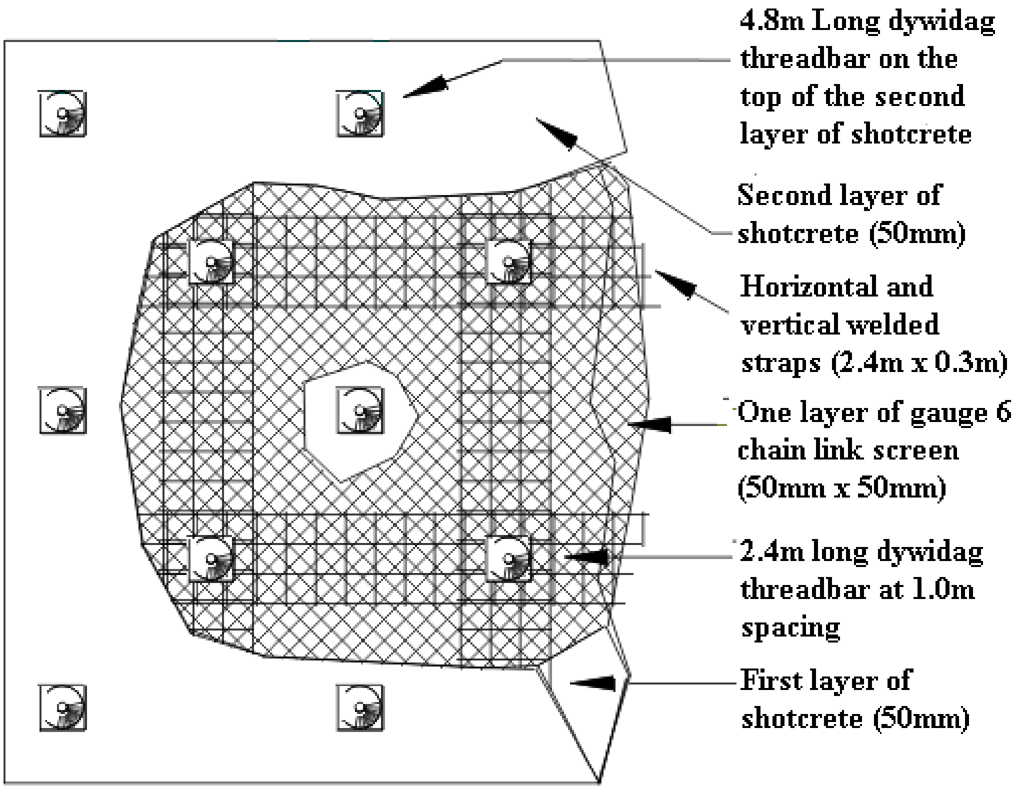
Reinforcement systems without shotcrete allow the rock mass to deform/spall between bolts until the mesh/straps are extended. Shotcrete against the rock and encasing the reinforcement does not allow this failure zone to form under low stress.

The interaction of components in a rock reinforcement system is shown in the following two diagrams:



Early installation of full rock reinforcement will result in a sound support system

The earlier the full system is installed the better. If the initial support allows the rock mass to deform then failure can occur even if a full rock reinforcement system is installed. In rock masses that are prone to significant deformation with limited development, the procedure of installing the minimum support to achieve the maximum development rate is not acceptable



Installation sequence to obtain a sound rock reinforcement system

Shotcrete without reinforcement offers very little deformation resistance and fails rapidly into large slabs..

Shotcrete without reinforcement fails at low deformation

In fact, plain shotcete cannot be considered if there is not good bonding to the wall rock and the object is keep rock blocks in place with no major deformations . The following photo shows failure of plain shotcrete into large slabs at low deformations. In fact shotcrete on its own can be a hazard where deformation is expected

DRIFT SUPPORT STANDARDS

The MRMR support table is a useful tool for this purpose, however it must be included in the mine standards, otherwise there will always be the excuse that it is not convenient to install the support at the right time. It is obvious that a simple system is required to ensure that support standards are maintained, the problem with underground support is usually not the support recommendations, but the installation procedure. With due respect to contractors, they do not understand or are not concerned with the long term requirements of a stable rock mass to effect the draw of thousands of tons through a drawpoint. In fact this point is often not appreciated by mining personnel. The only way that support installation can be effective is to have a support system and code of practice included in the mine standards. If these standards are not adhered to then disciplinary action is called for. The following is an example of rock reinforcement systems

MRMR	DRIFT SUPPORT SYSTEMS	UNSUPPORTED AREA TO FACE	LENGTH OF ROUND
0 - 10	Spiling at 2m(4m bolts at 300mm spacing), bolts (5m cables/bolts @ 2m), plates, cables, straps, chain link mesh. shotcrete- minimum 100mm thick	0 m^2	1.0m to 2.0m
11 - 20	Spiling??. Bolts + (5m), plates, cables, straps, mesh / fibre. shotcrete- 100mm thick	0 m^2	2.0m to 2.4m
21 - 30	Bolts, plates, straps, mesh / fibre. shotcrete- 100mm thick	12m^2	2.4m to 2.8m
31 - 40	Bolts, plates, mesh / fibre. shotcrete- 100mm thick	14m^2	2.8m to 3.2m
41 - 50	Bolts, plates, mesh and straps	28m^2	3.2m to 4.0m
51 - 60	Bolts, plates	80m^2	4.0m
61 +	Bolt localised well-jointed areas	100m^2	4.0m

BOLTS

Bolt length - The bolt length is based on the following formula which the writer devised after considering all the factors that affect rock mass reinforcement:-

* <u>Low stress</u> :- $L = 1m + (0.33W \times F)$

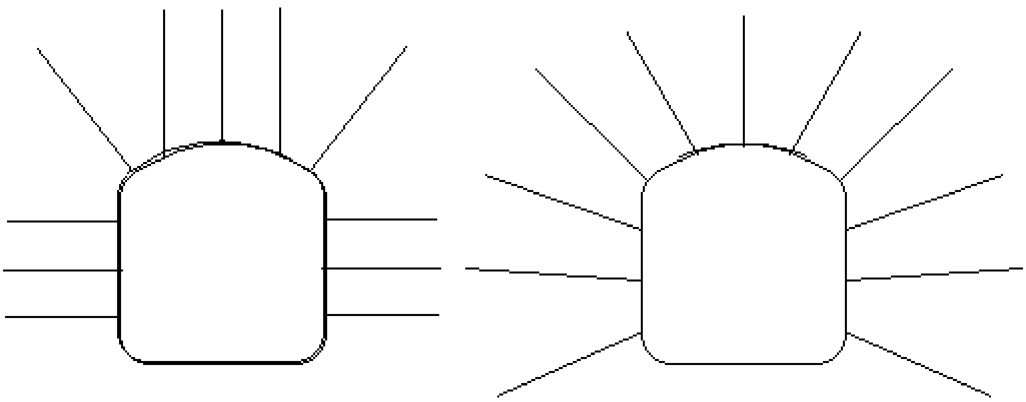
* <u>High stress or squeezing ground</u>:- $L = 1m + (0.5W \times F)$

F is a factor based on the MRMR values:- 0 - 20 = 1.4 ; 21 - 30 = 1.30 ; 31 - 40 = 1.20 ; 41 - 50 = 1.10 ; + 50 = 1.00.

W = Width of drift

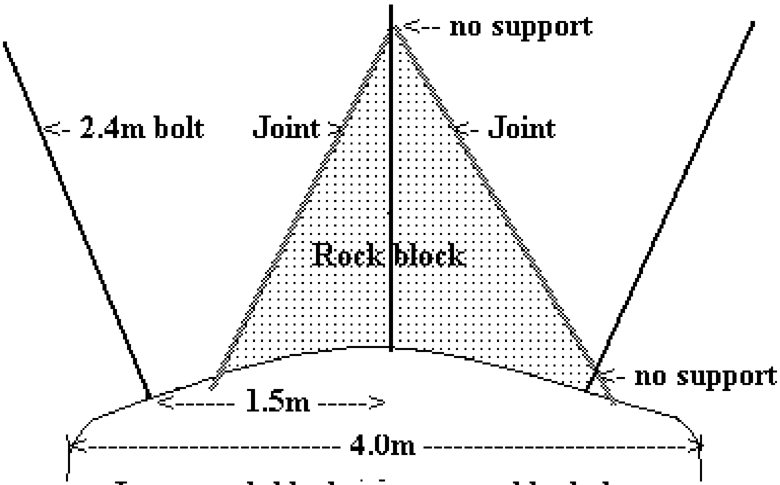
For the bolt to be effective in retaining the surface reinforcement - shotcrete / straps - the bolt must be securely anchored beyond the fracture zone that has developed around the drift. This length must relate to the rock mass strength and the stress environment.

Bolt pattern - There is tendency to avoid installing bolts in the lower corner, this can only be done when the MRMR exceeds 50.



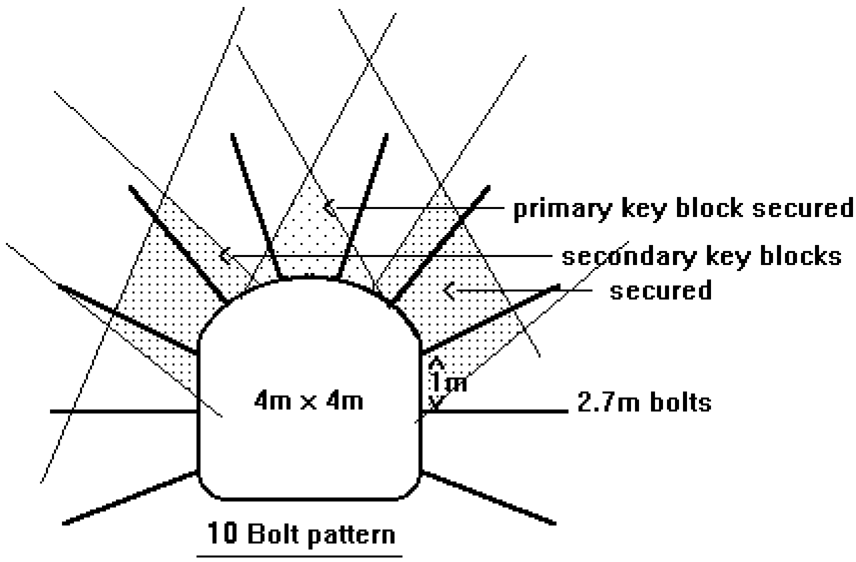
Far better coverage for rock reinforcement

The practice of increasing the bolt spacing in more competent the ground is common. As the object of the exercise is to ensure that blocks do not fall from the back then this approach is not correct as widely spaced bolts will not cross widely spaced joints and will be contained within the large rock block. Also bolts at 1m cover $1m^2$ whereas a bolt at 1.5m spacing covers $2.25m^2$.



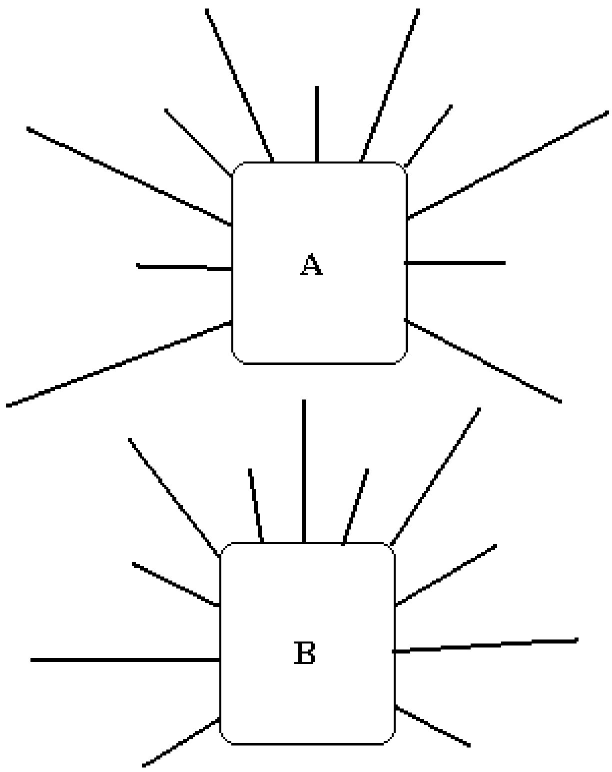
Large rock block unsupported by bolts

The following bolt pattern is suitable for most situations, the V in the back is preferred to the vertical bolt on the centre line:-



10 Bolt pattern

Where deep anchorage is required in weak rock, cables or long coupled bolts must be installed. Bolt length should be at least 2.4m long whilst the cables and long bolts should be 5m. It is recommended that the bolt and cable pattern be simplified by having bolts and ‘cables’ at 1m spacing in the ring and between rings. The cables or long bolts must be incorporated into the standard pattern. It is important that the sidewalls are also supported by deep anchored support. The following two diagrams show the pattern for alternating rings with alternating long and short bolts. If only one pattern is going to be used then pattern B would be recommended.

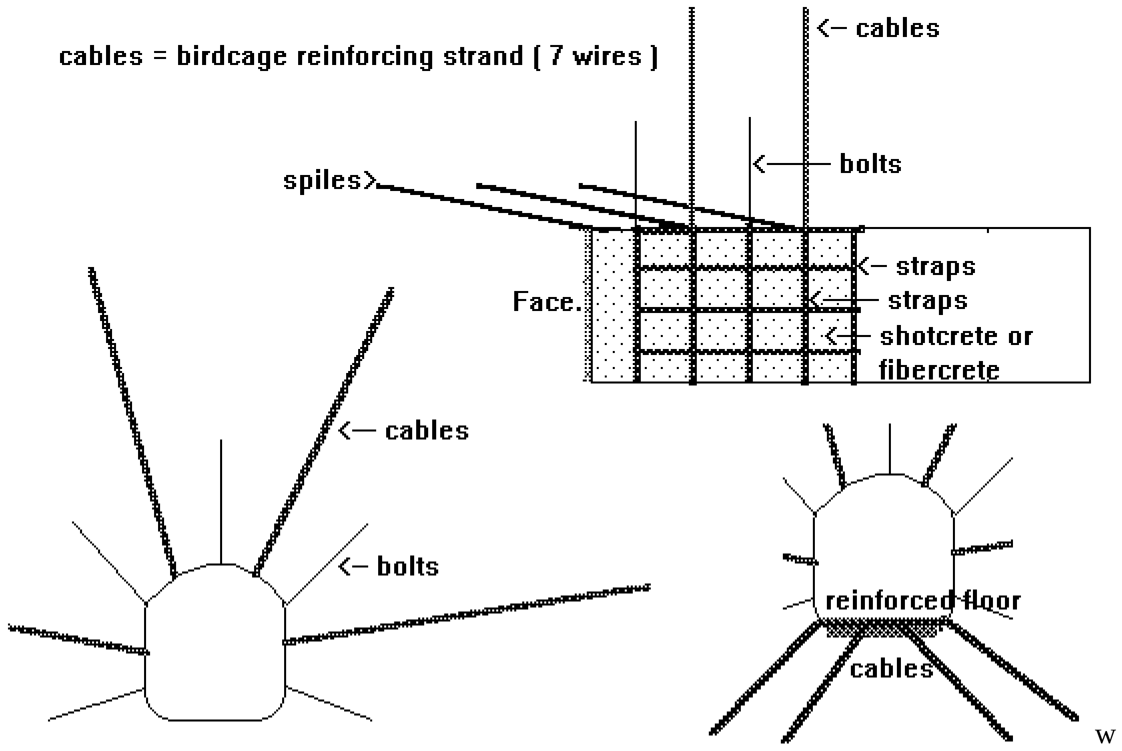


Alternating bolt and cable patterns

INITIAL DEVELOPMENT SUPPORT

It is preferable that the initial support of bolts or shotcrete lining form part of the designed support system. There is a tendency to install split-sets, for example, as an initial support and then to come back later and install the final support. This is only acceptable if the split-sets prevent any failure/ deformation / loosening of the rock mass. The split set is a friction bolt and therefore the ability of split-sets in upholes, to support large blocks is limited owing to the short length of bolt beyond the block boundary. It is far better to install resin grouted bolts as part of the final system. Split sets can provide effective support if the tube is clamped, as seen in the following photograph showing the ground deforming around the split-set, but held in place by the mesh.

In weak ground or when traversing major shear zones the support must be up to the face and sufficiently comprehensive to prevent any deformation or caving. The spiling technique has proved to be effective if done correctly. A procedure is to use 4m x 25mm rebar installed at 300mm spacing at the face for every 2m advance. The ends of the spiles strapped into place and the straps shotcreted. As soon as possible and as part of the system cable bolts or long coupled bolts are installed. If the floor is deforming then the floor support must be done at the same time. The object is to contain the rock mass before loses its integrity

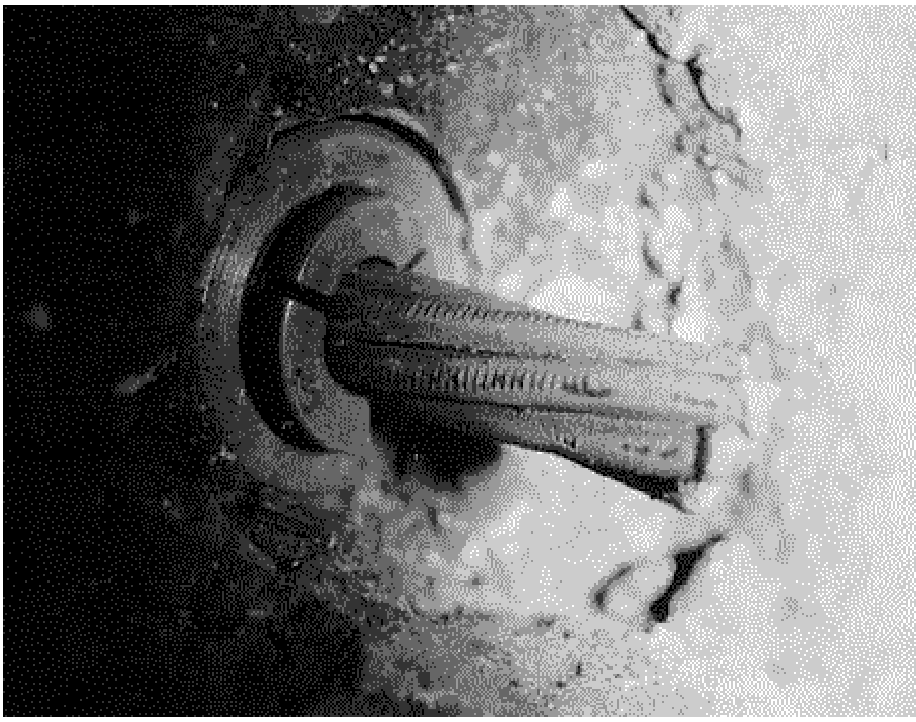


Polystyrene / Styrofoam models should be used to display the bolt patterns to determine if the coverage is sufficient. The drifts are modelled as the solid and the bolts cut to the right length from coloured blasting wire and inserted into the model. The end result appears as loaves of bread resembling a porcupine.

CABLE BOLTS - INSTALLATION PROBLEMS

Cable bolts are used and misused extensively in underground situations. The installation of the cable has to be done under good conditions. At Cassiar Mine it was found that cables did not perform as well as expected, due to installation difficulties. The cable clamps were not always clean owing to the rough conditions underground with the result that dirt filled the gap between the clamps. Another aspect often ignored is shotcreting over the clamps. This means that the gap becomes filled with shotcrete preventing the wedges from moving and clamping the cable. Failure to grease the wedges and clamp can result in rusting and once again the wedges rust in place and are prevented from moving.

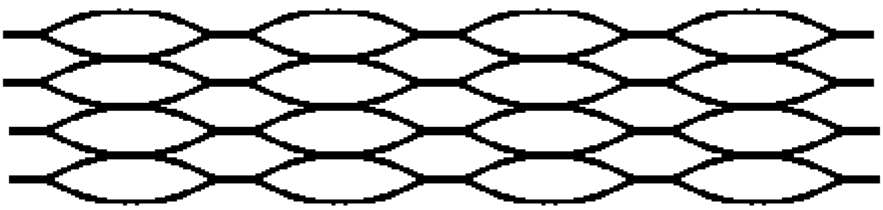
The following photograph shows an interesting situation where the wedge grooves have been imprinted on the cable which was inside the clamp as the sidewall deformed outwards. Afterwards the sidewall must have moved inwards with the result that the cable pushed through the clamp.



Cable bolt pushing through the clamp

STRAPS

Over the years various forms of straps have been developed from the W strap to the open tendon straps to the expanded metal straps. The open straps were developed so as to form part of the rock reinforcement support systems. The W strap lay on top of the rock and it was difficult to have it provide initial restraint.

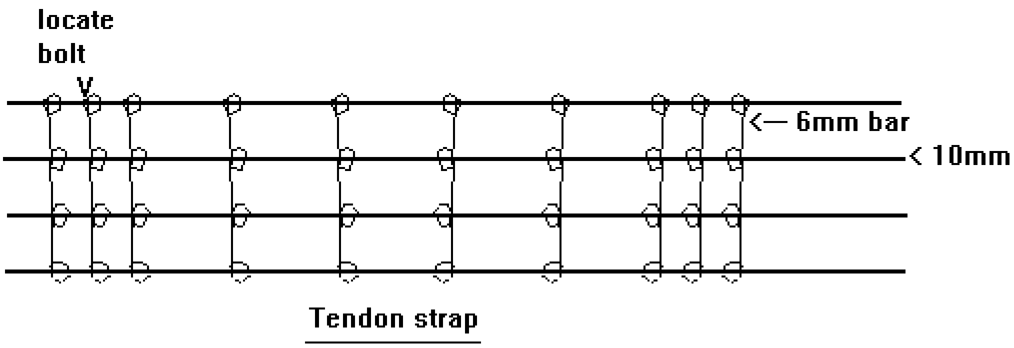


Expanded metal straps

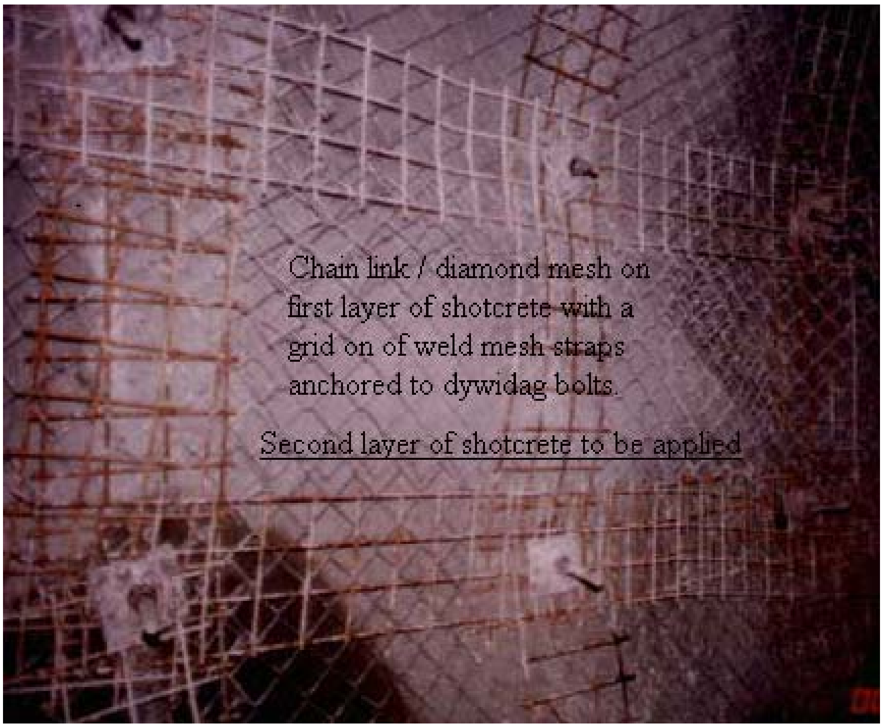
Expanded metal straps elongate at low sidewall deformation so they must be encased in shotcrete to provide sufficient support pressure. When encased in shotcrete the yield is at a high level.

The tendon strap provides good longitudinal restraint. The following diagram depicts a tendon strap made from 2.4m lengths of 10mm diameter rebar as the longitudinal component. To provide rigidity and maintain the correct spacing, 6mm diameter rebar is wrapped around the longitudinal lengths.

Where the bolts and plate are located, say at 1.0m intervals, the number of transverse rebars are increased :-

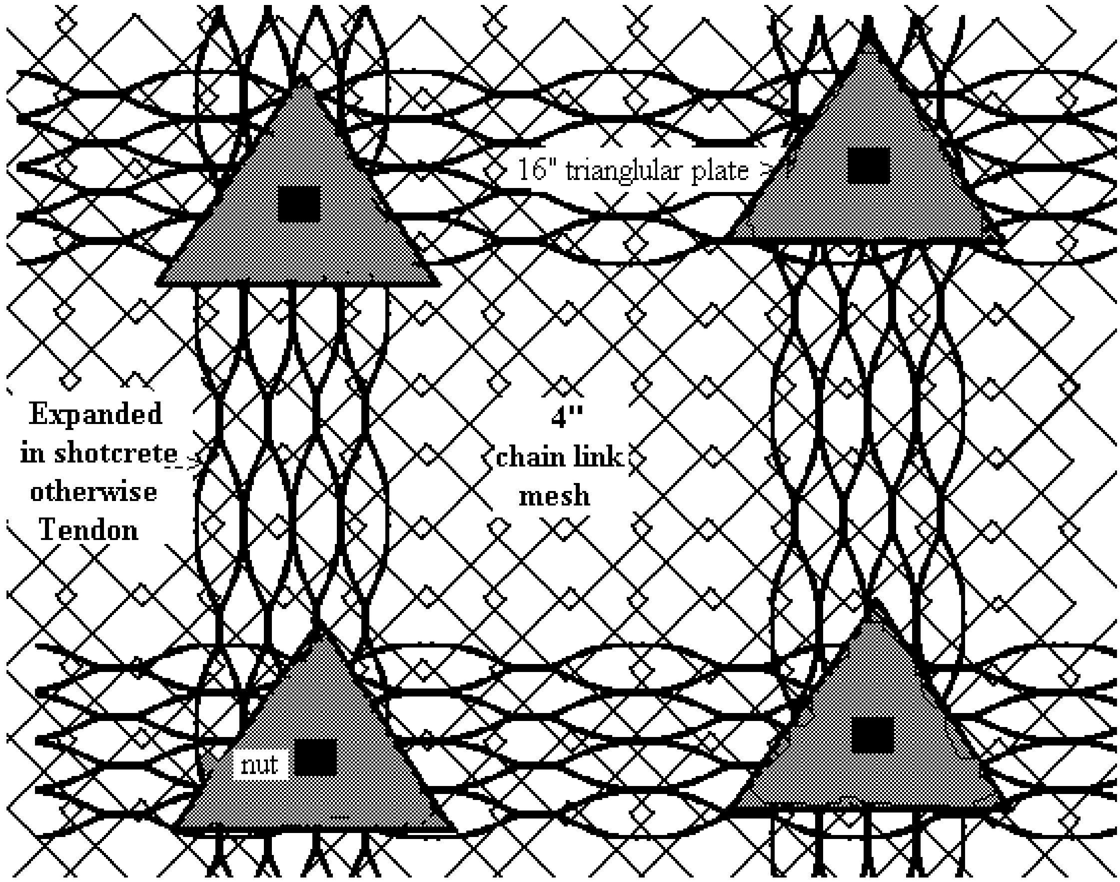


The manufacture of these tendon straps can be labour intensive and in some situations heavy duty weld mesh was used as shown in the photograph below. The weld mesh straps performed a useful function, but did fail at the welds and could not compare with tendon straps or expanded metal straps in terms of rock mass restraint

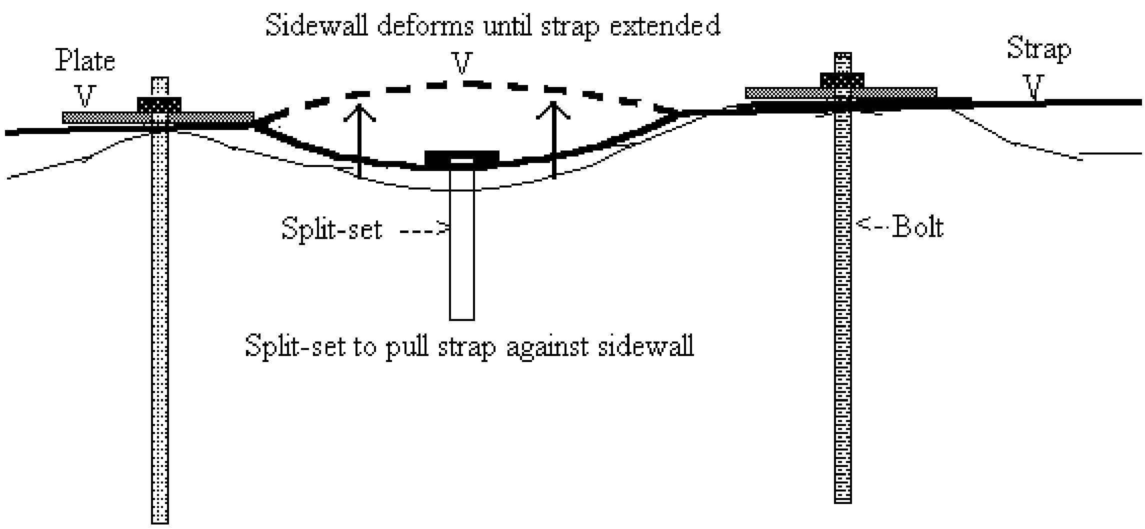


Heavy duty weld mesh straps on chain link mesh, anchored by dwyidag bolts

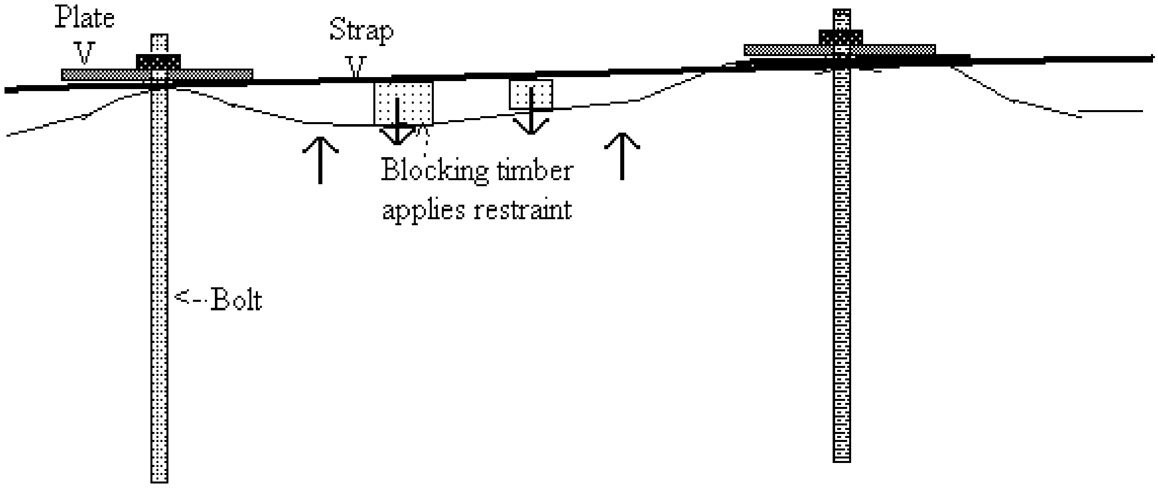
Orientation of straps - Longitudinal (horizontal) and radial (vertical) straps are only used on the sidewall. In the back longitudinal straps are preferred, however, if these straps interfere with the ring drilling then radial straps can be used. If the area is to be shotcreted, expanded metal straps or tendon straps must be used.



Where there are gaps between the strap and the sidewall, the strap only becomes effective once the rock mass has failed. If the area is going to be shotcreted then the strap is in intimate contact with the sidewall. If the area is not shotcreted then short split-sets have been used to pull the strap or mesh flush up against the wall rock. If the area is not to be shotcreted, they will only apply restraint when extended in the opposite direction: It is far better in these situations to block the strap against the sidewall. The following diagrams illustrate these points:-



Straps anchored against the wall rock with short split-sets



Straps blocked against wall rock - provide immediate support

MESH REINFORCED SHOTCRETE

The first question is what mesh to use. Weld mesh is the popular choice by contractors as it is easy to use, even though it's reinforcing and deformation capabilities are low compared with diamond / chain link mesh. This photograph shows typical failure of weld mesh at the weld.

When weld mesh is subjected to deformation, the mesh fails at the welds with sidewall deformations of 50mm - this was readily observed at Cassiar mine under squeezing ground conditions. Chain link, however with 4.5mm diameter wire and 75mm openings was still providing support with deformations of 100mm. Chain link is available with wires of 5mm and openings of 50mm, 75mm and 100mm. It was surprising to read in a recent publication on support, that a 'leading authority' had condemned chain link because it was difficult to shotcrete. The article was backed up by a photo graph clearly showing poor workmanship, this is also evident in the following photograph of shotcreted weld mesh:

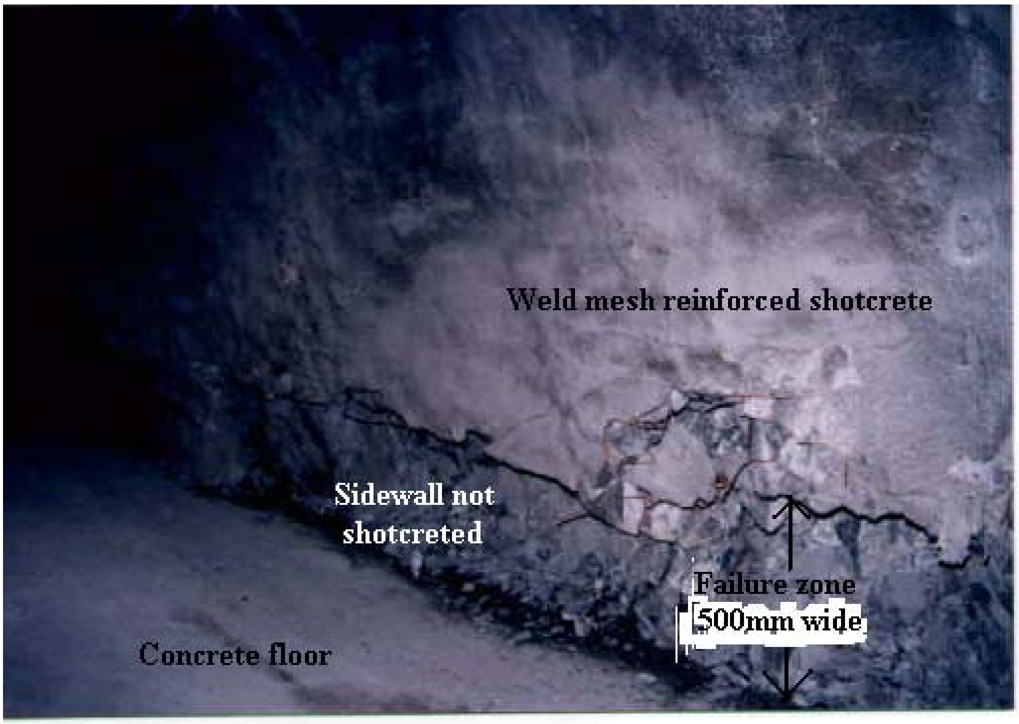
Poor shotcreting of weld mesh

Weld mesh is primarily used not because it has any strength properties but because contractors find it easier to install. The following photograph shows correctly installed chain link mesh.

On those mines using diamond / chain link mesh there is no difficulty in shotcreting the mesh when done by competent personnel. This photograph shows that shotcrete can be placed in tight areas if the support installation procedure demands that this should be done. There are no excuses for poor workmanship.

Several layers of weld mesh reinforced shotcrete and a dwyidag bolt

The following photograph shows how by not shotcreting down to the concrete floor that the support system can be compromised by failure of the lower sidewall. This particularly the case if rock bolts have not been placed in the lower corner. If muck is left in this area the failure does not become noticeable until the damage progresses up the sidewall.

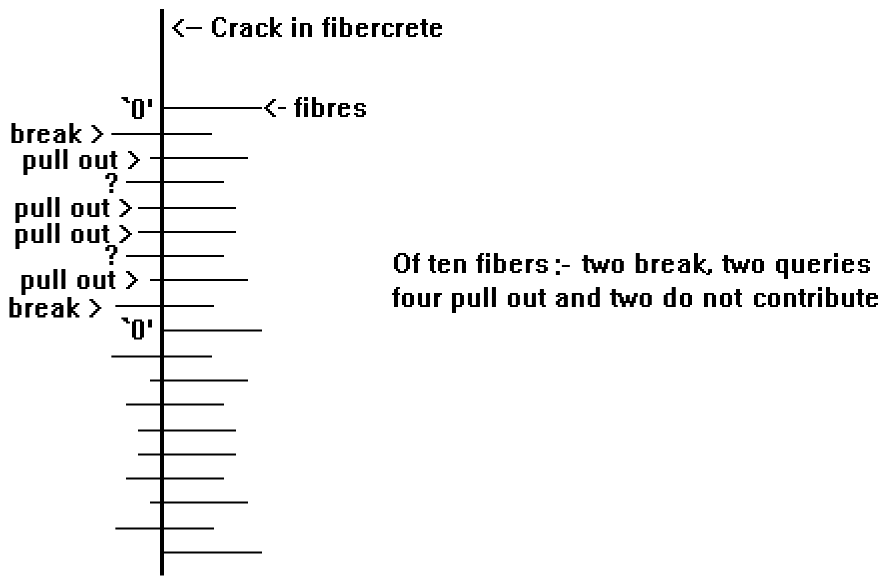


Incomplete shotcreting results in failure 500mm into sidewall

FIBRE REINFORCED SHOTCRETE - FIBERCRETE

Fibercrete is an easy to apply reinforced shotcrete and has proved to be successful where the deformations are limited. The support capabilities are very much a function of the length of wire used. When fibercrete is used in conjunction with tendon or expanded metal straps the lateral restraint of the wall rock is good, particularly when the bolt spacing is 1.0m. The close spacing of the bolts means that the area between the straps is limited and the straps prevent extensive crack development in the fibercrete. Underground exposures show that once a crack has formed the fibercrete does not appear to cope with deformation of fractured ground as compared with chain link mesh reinforced shotcrete where the wires only fail after extensive deformation. The important feature is the effect of a crack on the fibercrete as opposed to uniform loading of the whole surface. In the case of the loading of the whole surface all the fibres contribute, whereas if a crack forms - movement along a joint in the rock mass - only a few fibres contribute and the degree of contribution is very much dependant on the length of the fibres. This is shown in the following photograph.

Typical crack failure of fibercrete.

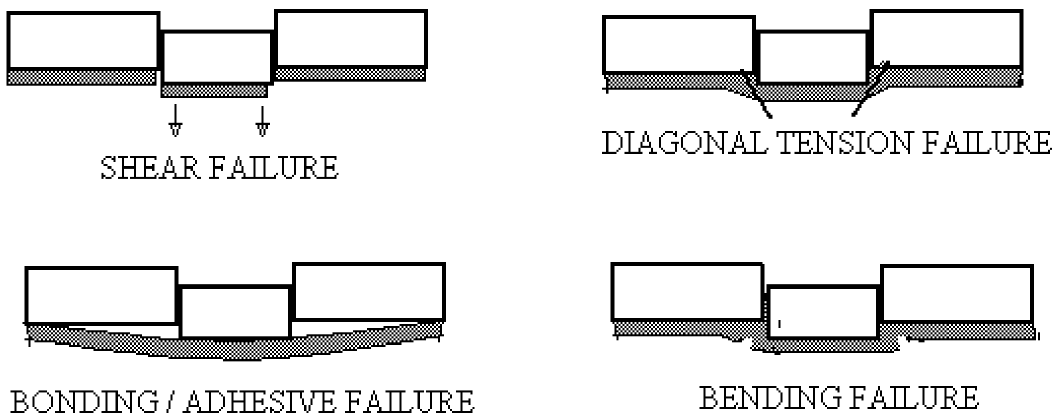


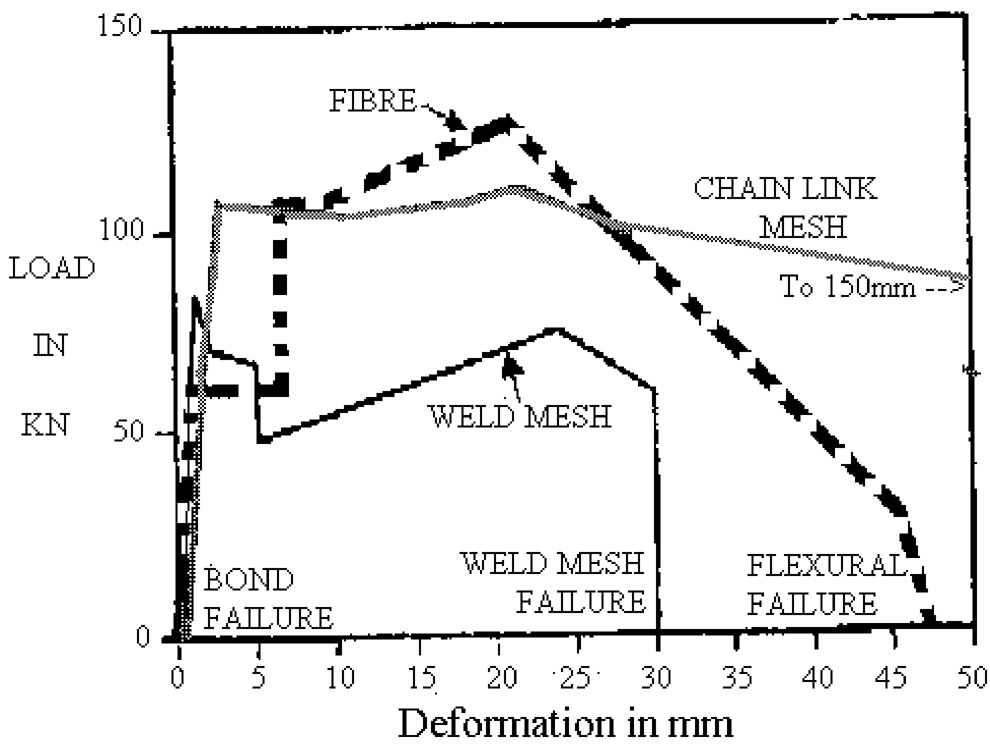
Situation repeatedly observed underground – conclusion that fibercrete does not perform as well as mesh when subjected to normal crack failure.

Another side issue with fibercrete is that the loose steel fibres lying on the roadway penetrate the electric LHD cables because of the magnetic field generated by the cable. This problem has been solved by sweeping the roadway at regular intervals. That of course presupposes good roadways and good housekeeping.

LABORATORY TESTING OF SHOTCRETE

Tests done on fibercrete panels has shown that the fibercrete has good deformation capabilities up to 50mm. However, with continued deformation it does not compare with chain link / diamond mesh. The fibercrete is superior to weld mesh. Tests carried out on fibercrete and mesh reinforced shotcrete panels have shown good results for the fibercrete as the load was applied over a large area and all the fibres come into play. ‘Falling block’ tests as depicted in the following diagram showed that the fibrecrete out-performed the weld mesh.



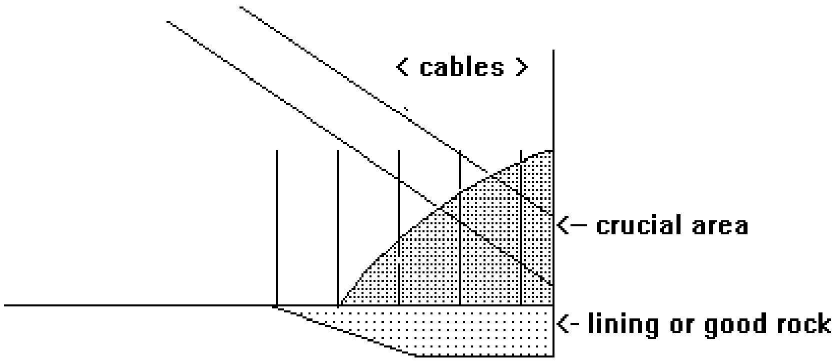


A factor which is ignored in these tests or possibly not emphasised is the proportion of steel in the reinforcing. There are published results comparing a 4% fibre content with shotcrete with 1.4% mesh.

In Chile, chain link of 5.5mm diameter with 50mm, 75mm and 100mm openings is available. In South Africa tests were conducted using mesh wire of only 3.5 mm diameter - a difference of 240%. *This is going to have a significant effect on the performance of the reinforced shotcrete.*

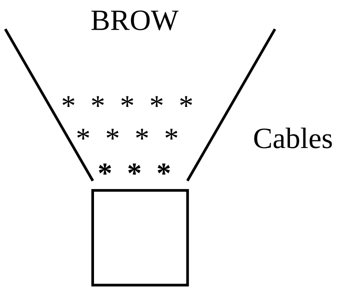
BROW SUPPORT

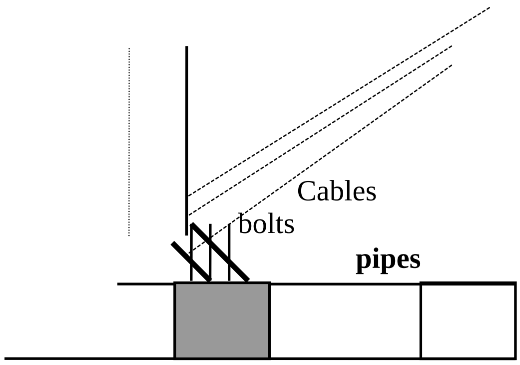
The brow is a difficult area to support as no restraint can be applied to the face. If the abutment stresses are high there is extensive damage to the brow and it fails rapidly when production starts. In these cases the drawpoint lining must be designed to prevent further failure. The first step is to preserve the brow as much as possible by using the correct undercutting system. Brows can be supported by cable bolting from the drawbell drift or from the undercut level. Because no restraint can be put on the brow itself the object of the cables is to assist the drawpoint rock reinforcement in holding the rock blocks in place. Thus widely spaced cables are of little help in highly jointed ground.



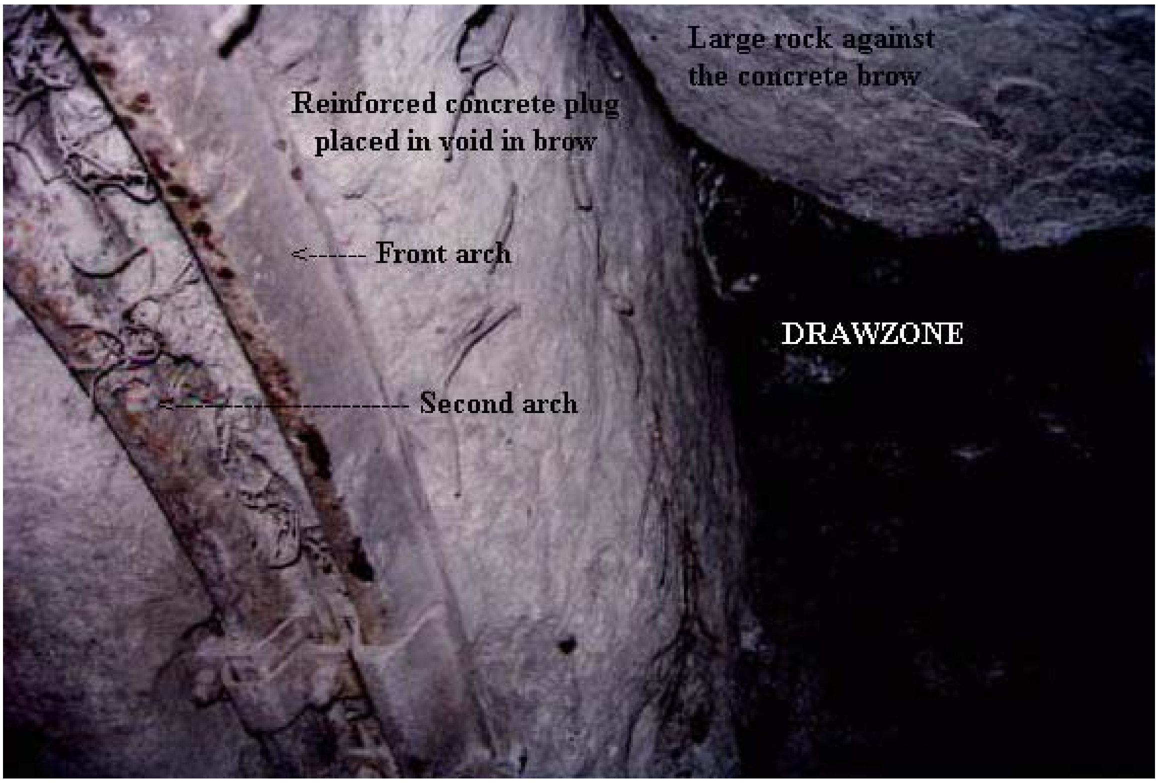
Drawpoint

There is little point in placing one line of cables in a brow immediately above the arches. If the object is to effectively support the critical area of the brow, and cables are considered to be the right tool, then they must be installed in a pattern that will stabilise the area. The brow is not restrained and is subjected to attrition, point loading from arches and column loading of the major apex. In some situations the cables could be sub-parallel to the structures and thus would be less effective. In these cases 50mm pipes are installed at a large angle to the structures so as to supplement the cables. If the ground is sheared or highly jointed, the cables serve little purpose and the effort should be spent on the lining. The following two diagrams are views are looking at the brow from the drawbell and along the drift:-





Reinforced concrete brow at Bell Mine - The following photograph shows a brow of reinforced concrete that was placed after the collapse of the proposed brow area. The reinforcing was a simple rectangular grid of reinforcing bars. The photograph was taken after 20000 tons had been drawn. The drawpoint produced its quota of 150000 tons.



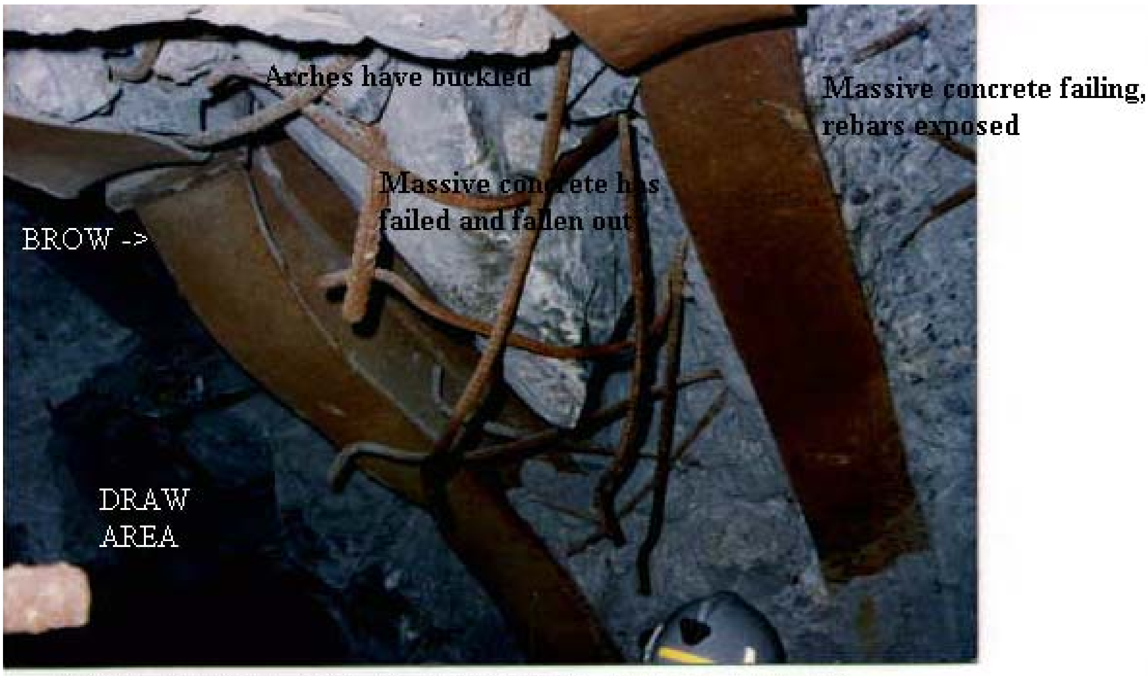
Reinforced concrete brow at Bell mine

Orientation - The drawpoint brow is the most important single unit of the extraction/production level. It is often very difficult to support the brow and therefore it's orientation with respect to structures takes precedent. The object is to have the structures confined in the brow by the lateral stresses along the major apex. A classic example of this occurred at Bell Mine where in the initial LHD block the structures were east-west and the brows were north- south and no brow problems were experienced. However, in the next block the layout was turned through 90 degrees with the structures no parallel to the brow resulting in wedge failures

DRAWPOINT LININGS

The purpose of a drawpoint lining is threefold. To provide support to the bottom of the brow, to back up the rock reinforcement system by providing support to the base of rock blocks and to stop wear of the brow. The benefits will be greatest if the lining makes good contact with the rock, failing which, it will only assist after deformation has occurred. If the rock mass is weak then the lining is, in effect the brow and must be strong enough to withstand the attrition and impact from the ore tonnage which is to be loaded. from that drawpoint.

In low stress environments massive good quality - 60 MPa - concrete linings up to 1 metre in thickness proved to be successful and handled ‘weight’. Massive concrete lost favour when it could not stand up to the high abutment stresses and was replaced by steel arches set in concrete. However, if one examines the steel arches after being subjected to abutment stresses, they are often twisted with the intercalated concrete in fragments :-

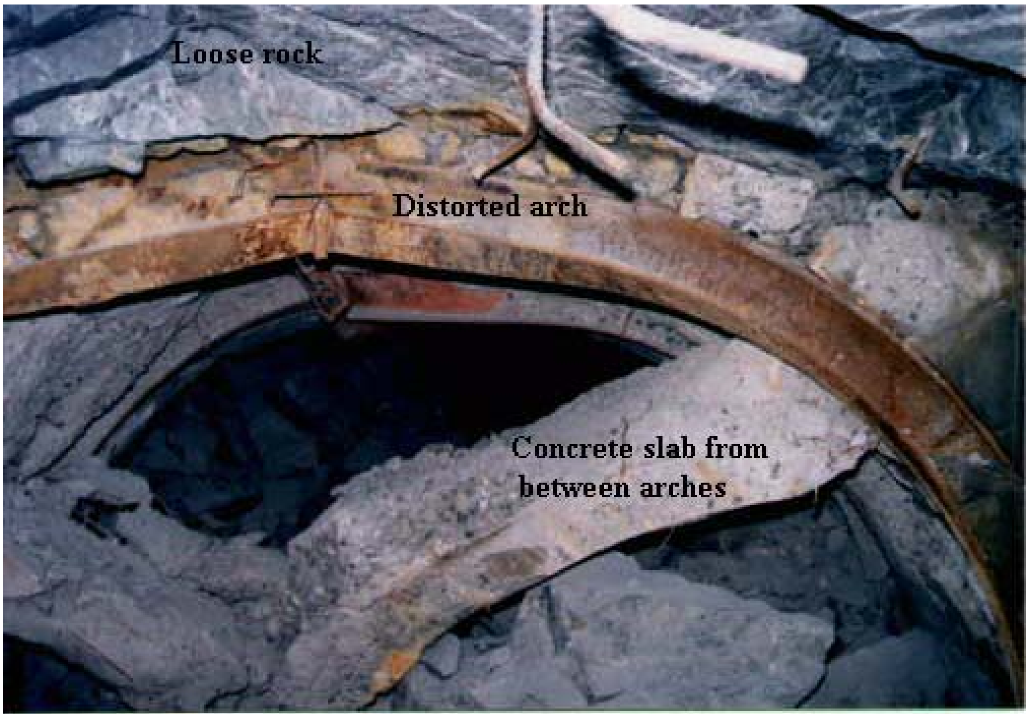


EFFECT OF ABUTMENT STRESS ON DRAWPOINT LINING

If the advance undercut system is used and the drawpoints are no longer subjected to major stress changes then high quality massive concrete could be used as drawpoint linings. This would mean a major cost saving.

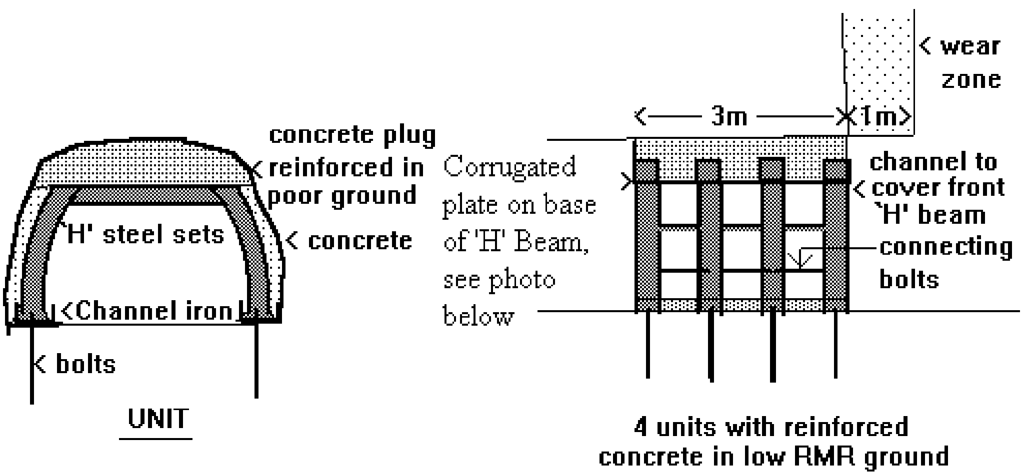
There are cases where steel arches have been installed as linings with a spacing of 0.5m between the first three arches and 1.0m between the last two. If the arches are to perform a useful function in supporting a retreating brow then the same spacing should be used over the whole lining distance. The arches need to be securely tied into the sidewall with bolts and clamps. The arch legs to be secured in channels and concrete.

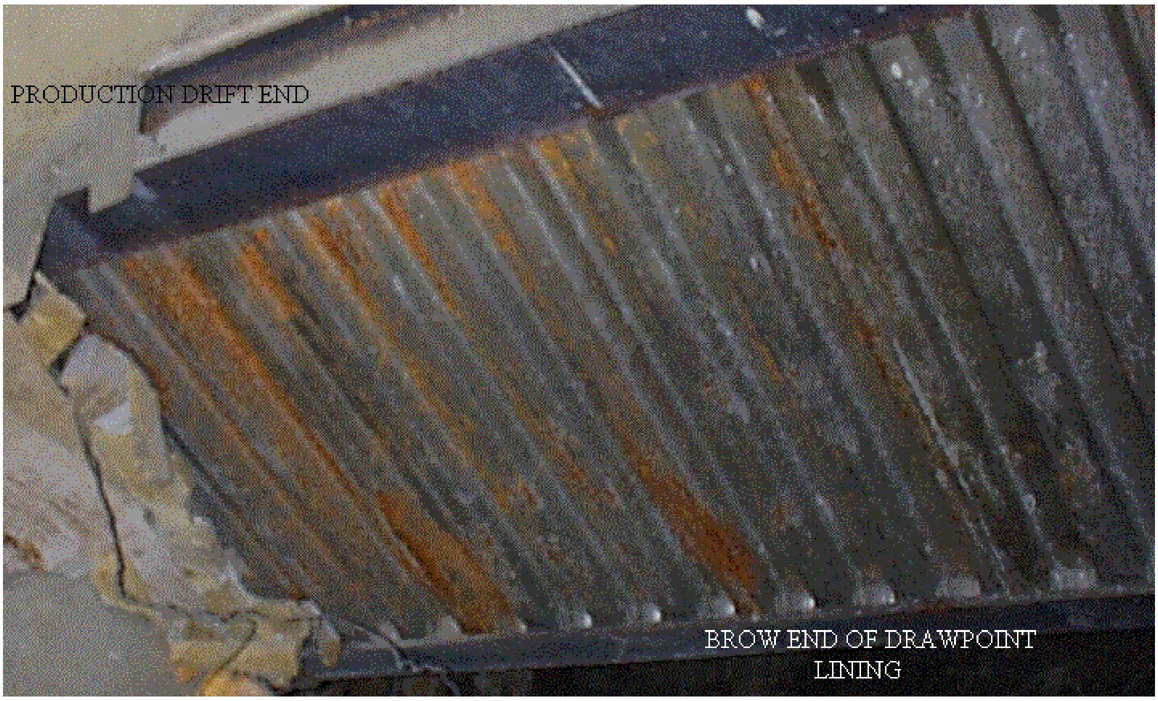
When arches are concreted in place, the quality of the concrete is often low. If the concrete is to be part of the rigid support system it must be in contact with the arches and the wall rock so as to ensure that the concrete takes load at an early stage. The significant difference in modulus between the steel arch and the concrete leads to failure at the contact between the arch and concrete and rapid failure of the concrete. The drawpoint shown in the following photograph had only drawn 5000 tons out of a possible 100 000 tons and was due for repair

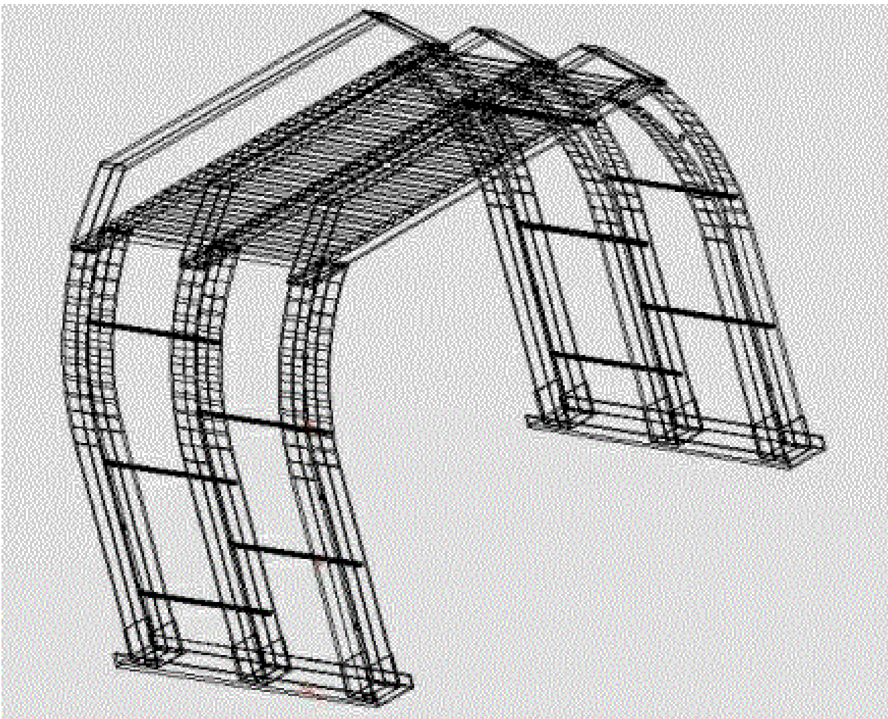


Concrete separating from arches in drawpoint subjected to high stress

Henderson lining - The drawpoint lining used at Henderson Mine referred to as the Henderson lining has proved to be successful because the steel is not incorporated in the massive concrete **:-**

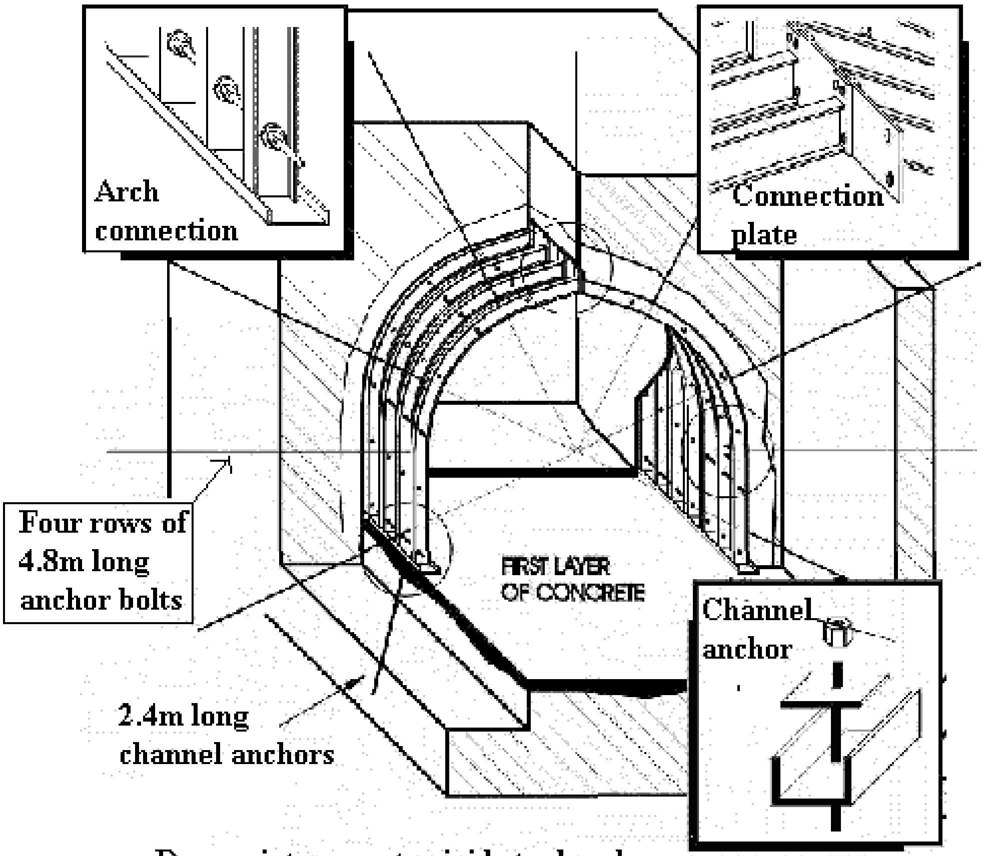






Reinforced concrete lining - A reinforced concrete lining was successfully developed at Bell Mine. Yielding steel arches were installed and backed with chicken mesh to act as a former. A reinforcing bar cage was installed between arch and sidewall and the opening filled with concrete. The inside of the arches are shotcreted for LHD protection.

Rigid steel arch - A rigid steel arch and shotcrete lining was developed at Cassiar mine. The logic was to ensure that the arches were properly connected so that they operated as one unit. The following photograph and drawing show how the arches were connected along the crest, to a longitudinal beam and then longitudinally down the sides with 25mm dywidag bolts. The whole system was anchored to the sidewall with 4.8m long dwyidag connected bolts.



Drawpoint support - rigid steel arches

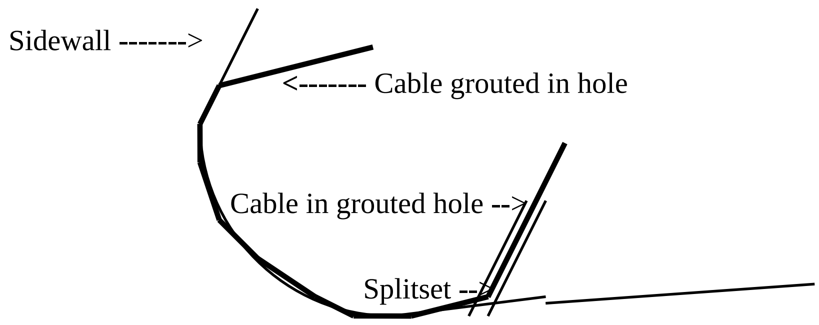
Yielding steel arches - This system is discussed in detail in the section dealing with drawpoints in squeezing ground at Cassiar Mine.

Unlined drawpoints - On some mines unlined drawpoints in the higher MRMR ground have performed well with over 100 000 tons having been drawn through them.

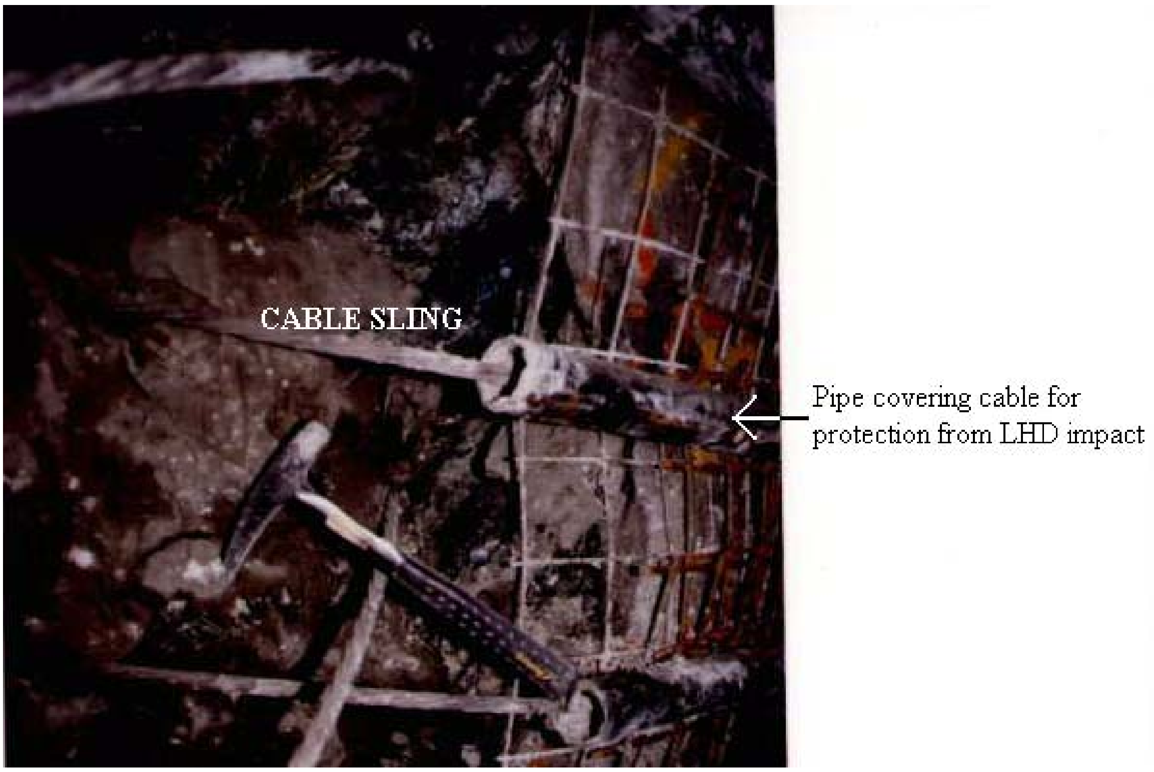
BULL NOSES, CAMEL BACKS AND JUNCTIONS

The legs of the large spans at the junctions (drawpoint takeoffs) are the sidewalls of the production drift and the corners into the drawpoints. The acute angle corner is known as the bull nose and the obtuse angle corner as the camel back. Stresses are concentrated mainly in the bull nose and to a lesser extent in the camel back, therefore show more damage than the sidewalls. Apart from the normal sidewall support a wrap around system of using ropes or cables has been developed and has proved to be an excellent backup to the standard methods. When introduced in Zimbabwe, 70 ton Koepe ropes were used and these stabilised areas that were approaching a state of collapse. A system of using cable slings was developed on Cassiar mine and also proved to be effective and easier to install than the ropes.

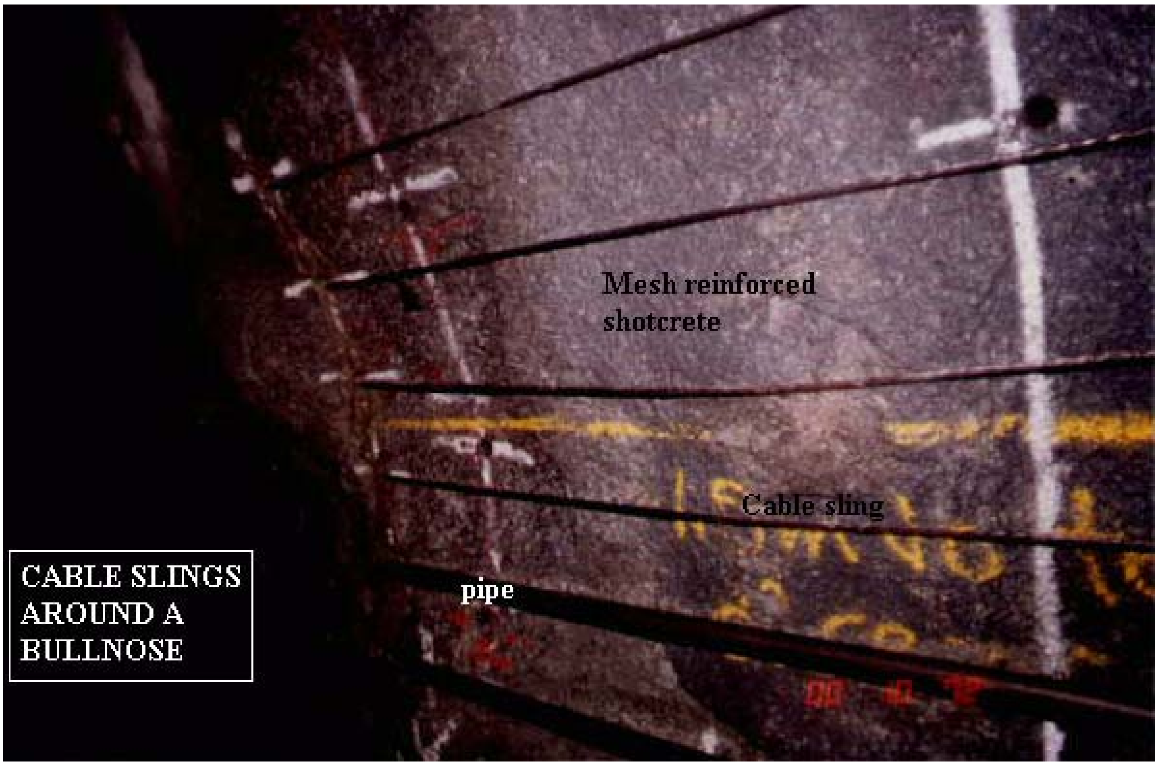
The technique is to grout the one end of the cable in a hole and when the grout has set then to drive in the other end into a grouted hole with a split set.



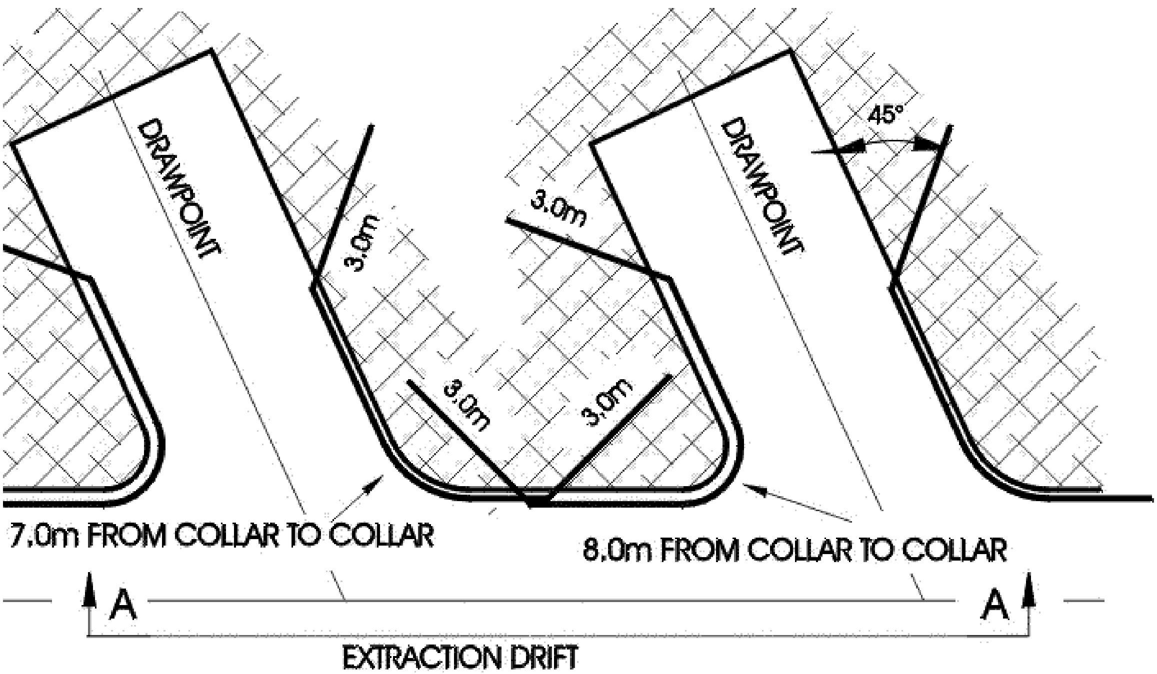
This photograph shows the actual installation.

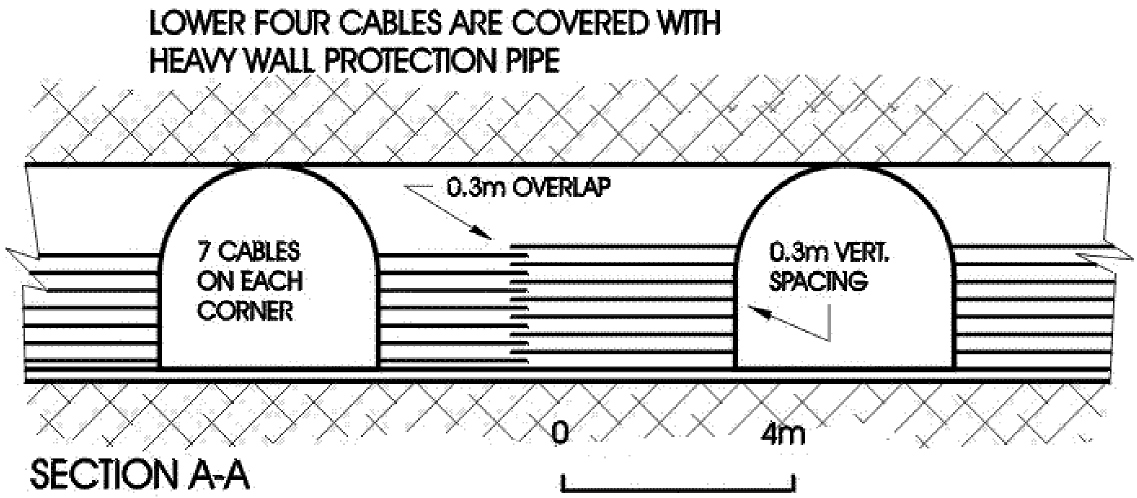


Cable sling with pipe protection from LHD damage



The length of the cable is kept as short as possible to limit elongation and thus provide more effective support. The cables are grouted into the rock mass for 3m. Not only does the cable provide excellent restraint to the rock surface, but it is also contributes to the rock reinforcement.





On some mines the cables have only been anchored in the drawpoint and then joined on the sidewall between the drawpoints. This is not advised as a long cable and does not provide good restraint owing to the greater elongation. The following photograph shows cables connected midway between the bullnose and camel back with the anchorage in the drawpoint.

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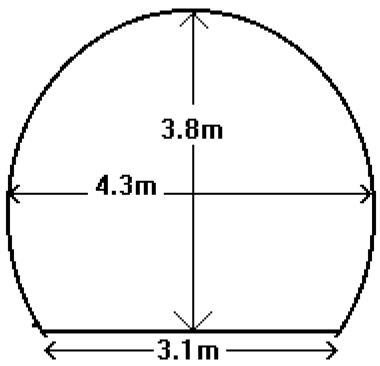
JUNCTION STABILITY

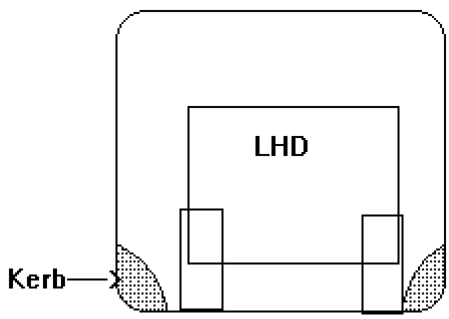
The largest spans in the extraction drifts occur at the junction with the drawpoints. These spans are of concern, but seldom do they fail as they are the base of the major apex. It is the pillars that support the spans that fail. Thus we find emphasis on supporting the spans and not enough attention paid to the pillars. Apart from the standard rock reinforcement, cable bolts are usually installed in the back. However, it is common practice to install the cables without plates or straps connecting the cables. This is not making use of the support potential of the cables

DRIFT SHAPE

The following shape would be ideal for a LHD drift for strength and sidewall protection:-



A technique employed at Bell Mine, Canada, to protect the sidewalls from LHD damage is to shotcrete a kerb during the final shotcrete stage:-



SQUEEZING GROUND - STRESS RELIEF

The concept of a stress relief system is on the basis that in time it is impossible to stop squeezing ground with standard rock reinforcement in production drifts. It can be done in isolated drifts by extensive cable bolting with deep seated anchors and mesh reinforced shotcrete often reapplied -

Austrian tunnelling method. As the production drifts have a limited life a simpler method is required. The following system is easy to install and allows the squeezing ground to be removed through the plain mesh by drilling large diameter holes into the wall rock. These holes provide space for swell relief so that the drift dimensions are maintained. Deep seated rock reinforcement is required and relief is only practised when convergence occurs. This system was tried on King mine and was successful, but it was not pursued because they did not have the right drilling equipment or no doubt the full commitment. The cost of the stress relief operation was not compared to the cost of rehabilitating the drift and the production losses. This technique requires further testing with the right equipment.

If the stress relief technique is not used then the support design for squeezing ground requires thought, for example, does one do a lot of rock reinforcement when it is expected that it is going to squeeze and rehabilitation is necessary. The rehabilitation requires removing the previous heavy support, slashing and then installing yielding steel arches. It might be better to limit the rock reinforcement and install closely spaced yielding arches which are removed, straightened and reinstalled. Details of support methods used in squeezing are contained in a separate section.

A collapsed drift (squeezed) being rehabilitated with timber slats and arches

LATTICE GIRDERS

Lattice girders were used at King mine some 20 years ago and proved to be as effective as T H yielding arches provided they were incorporated into the rock reinforcement system. In practical terms the girders could protrude from the sidewall and therefore be damaged by LHD’s unless the sidewall is smoothed with shotcrete. Spacing the lattice girders at 2m is too much if they are to be effective in the shears, maybe 1m is a better interval.

COSMETIC SUPPORT

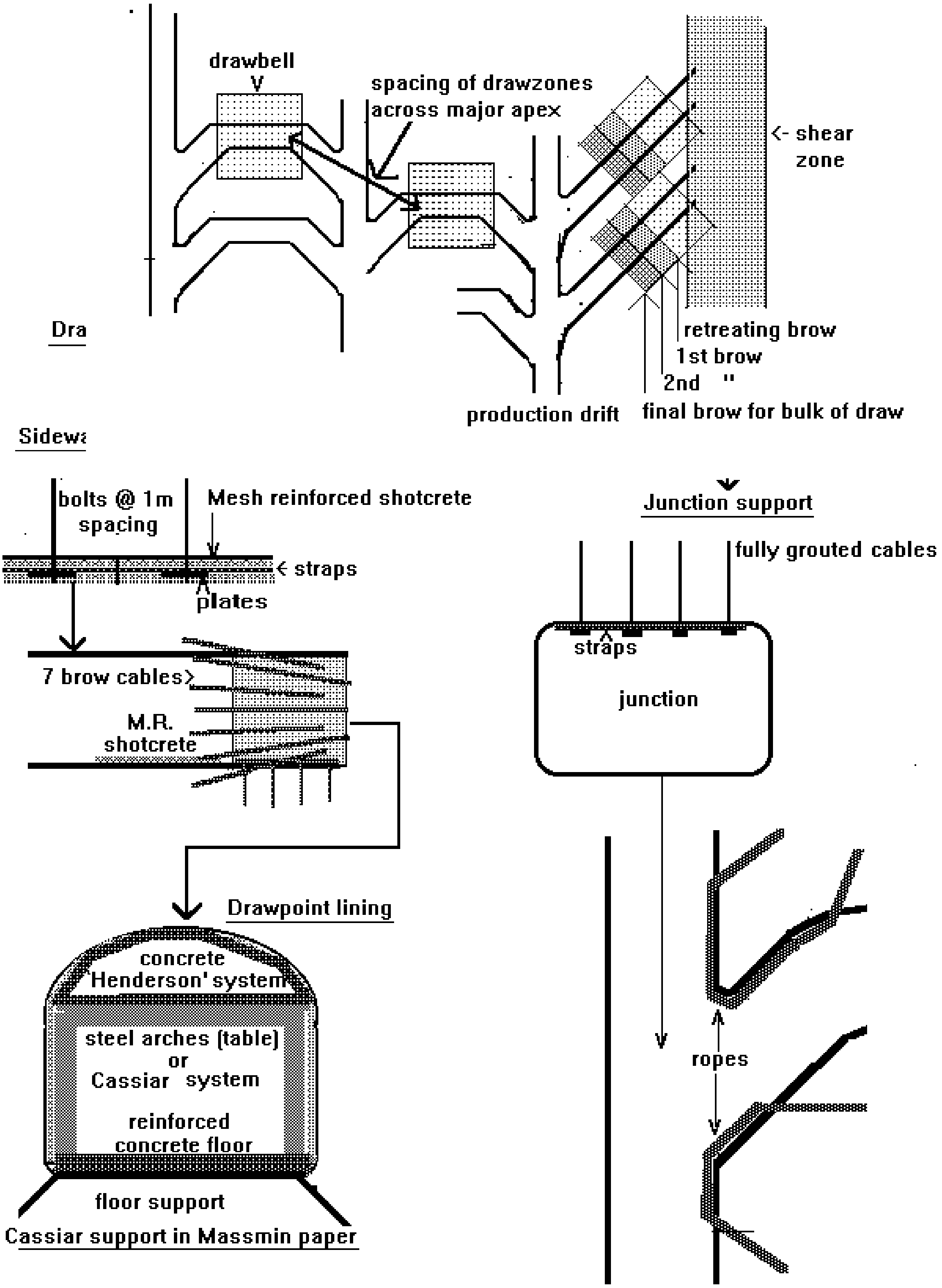
It has been observed that support has been installed in stable areas without any sound logic. It can only be assumed that this has been done for cosmetic purposes. The danger with cosmetic support is that when it is installed in a stable environment, the technique becomes the standard, only to be inadequate and dangerous when the need for good support occurs. A sense of false security is developed.

SUPPORT TECHNIQUES

	Low stress	High stress
Bolts length	= 1m + (0.33 x W x F)	= 1m + (0.5W x F)
Spacing	= 1m	= 1m
Type	= Rigid	= Yielding cone
W = width of drift	F = factor based on RMR :- 0-10 = 1.5, 11 -20 = 1.4, 21 - 30 = 1.3, 31 - 40 = 1.2, 41 - 50 = 1.1, + 51 = 1.0	
Deep seated	Cables = 1m + 1.5W	Ropes = 1m + 1.5W
Mesh	0.5mm x 100mm	0.5mm x 75mm
Linings	Mesh reinforced shotcrete Rigid steel arches Massive concrete	Mesh reinforced shotcrete Yielding steel arches Reinforced concrete
Surface restraint	Large plates - triangular Open straps	Large plates - triangular Open straps
Corners	25mm rope / cable slings over shotcrete at 1m vert.	25mm rope / cable slings over shotcrete at 0.7m vert.
Brows	Birdcage cables into brow <--- 75mm pipes inclined towards brow from d/p----->	Birdcage cables into brow
Repairs	<-------------- Grouting , mesh reinf. shotcrete, extra bolts, plates, straps and cables-- -> Arches	Yielding arches

LHD HERRINGBONE STAGGERED DRAWPOINT LAYOUT

FOOTWALL DRAWPOINTS



SUPPORT IN SQUEEZING GROUND AT CASSIAR MINE - J. JAKUBEC

1. DRAWPOINT LINING - ARCHES

If installed properly, arches tolerated severe deformation without jeopardising the production (Figure 1) Three types of steel arches were used as the drawpoint lining at Cassiar Mine:

I. Rigid “H” profile 2 section steel arch (Figure 2)
II. Yielding “TH”? profile 3 section steel arch (Figure 3)
III. Flat back “Henderson” type 3 section steel arch (Figure 4)

The following could be summarised:

I. Arches should be installed immediately after the excavation and support of the brow area is completed – not after the undercut was developed. This would minimise the ground deformation.

II. Arches should be initially as stiff as possible – rigid ”H” profile or Henderson “flat type” type offer possibly the best solution.

III. Once the deformation exceeds the strength of the rigid arch, the yielding arch should be installed to control the deformation.

2. ARCH PERFORMANCE

The most common failures experienced include:

Toppling of front arches - due to the excessive wear of the brow and inadequate concrete lining (plug) on top of the arch. Once the brow worn off and front arches were exposed, large blocks entering the draw bell point loaded the front arch causing toppling of the one or more steel sets into the caved area. Such deformed arches prohibited proper draw and were difficult to extract. Loss of brow had negative impact on adjacent drawpoint. The top connecting plate improved the arch stability but did not help (see Photo Figure 4) without solid concrete plug being placed on top of the arch in very weak ground.

Buckling of arch legs – due to the missing or poorly installed wall anchors. Initially the cable bolts and wedge clamps were used as side anchors. Due to the poor performance of the clamps cables slipped and arch legs buckled under the pressure prohibiting the LHD to operate (see Figure 6).

Sinking of arches and floor heave – due to the absence of floor support or missing base channels.

Toppling of yielding arch – due to the failure of connection channels. The original design of the yielding arch connection proved to be inadequate. “J” type bolts failed very easily under the minimal deformation and severely impacted on the integrity of the structure. Complicated system of connecting channels and “J” bolts (see Figure 7) was replaced with brackets and dywidag bolts. This system not only improved stability but also enable change arch spacing as required. Detail of the new system is illustrated on the Photo Figure 8).

Toppling of rigid arch – due to the failure of connection bolts. The original design using connecting bolts and nuts prove to be inadequate. The nuts were pulled through the middle flange of the arch destroying the integrity of the lining (see Photo Figure 9). Thick washers were later successfully used in combination with dywidag thread bars.

Collapse of the middle section of the yielding arch - due to failure of connecting “U” clamps. This failure was experienced where the yields exceeded 75 – 100 mm (see Photo Figure 10). Since the whole arch assembly was shotcreted over there was no access to the ”U” clamps to re-adjust.

Excess arch deformation – due to the inadequate concrete lining above the arches. This failure was experienced if inadequate profile was excavated during the tunnel development. In order to “fit” arches after the excavation and support, the reinforced shotcrete has to be cut and the integrity of the support was destroyed (see Photo Figure 11).

In all cases the negative impact of above-mentioned poor practices was observed only after the production commenced making the repairs very costly and disruptive.

THE FIGURES THAT ARE REFERRED TO IN THE TEXT, FOLLOW:

Figure 1 : If installed properly arches tolerated severe deformation without jeopardising the production.

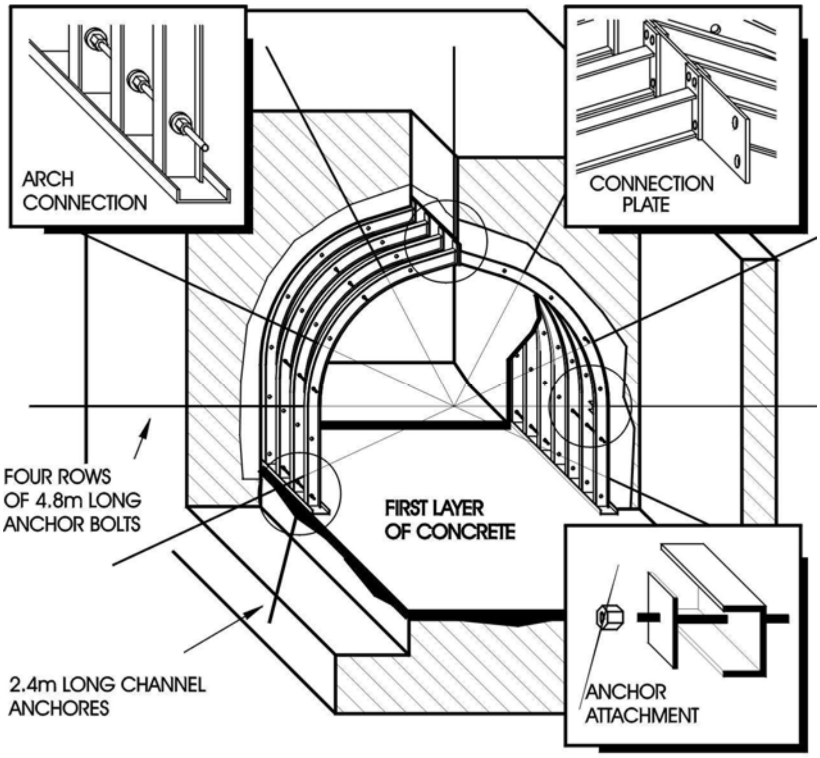


Figure 2a : Rigid Steel Arches

Figure 2b : Rigid arch

Figure 3 : Yielding arch

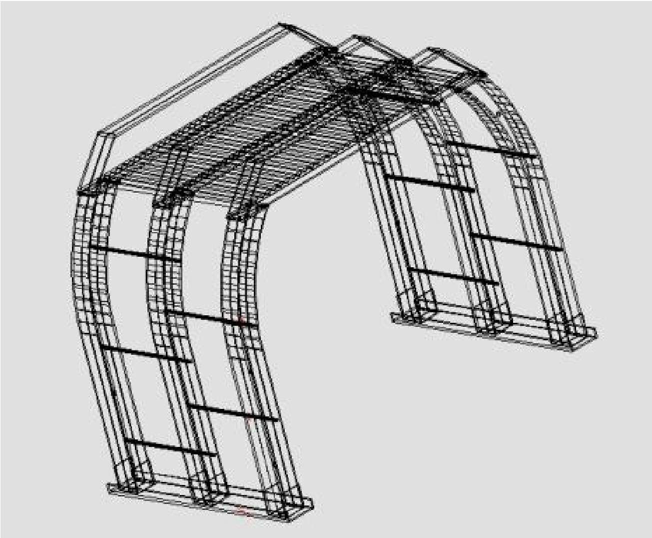


Figure 4a : Flat back "Henderson" type arch

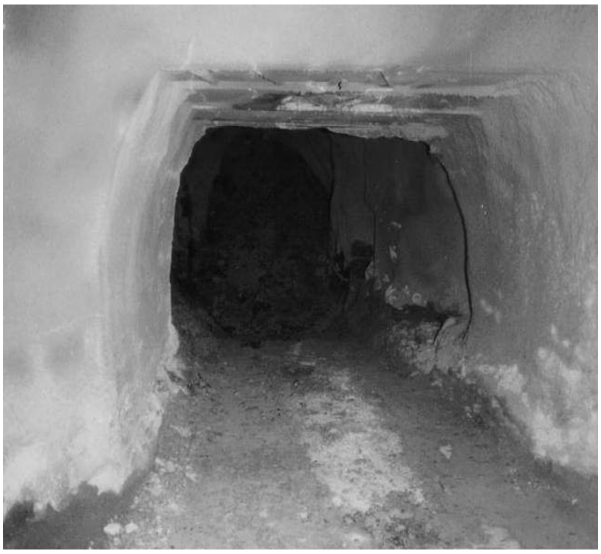


Figure 4 b : Henderson type arch

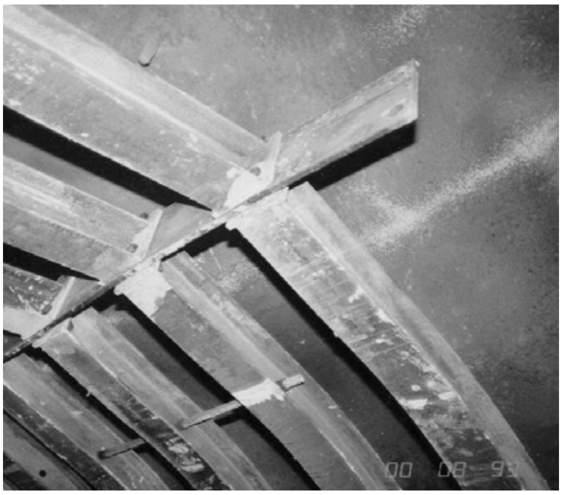


Figure 5a : Connecting plate

Figure 5b

Figure 5a, 5b ***Connecting plate - 5a, greatly improved rigid arch stability but it did not work without the reinforced concrete plug - 5b.***

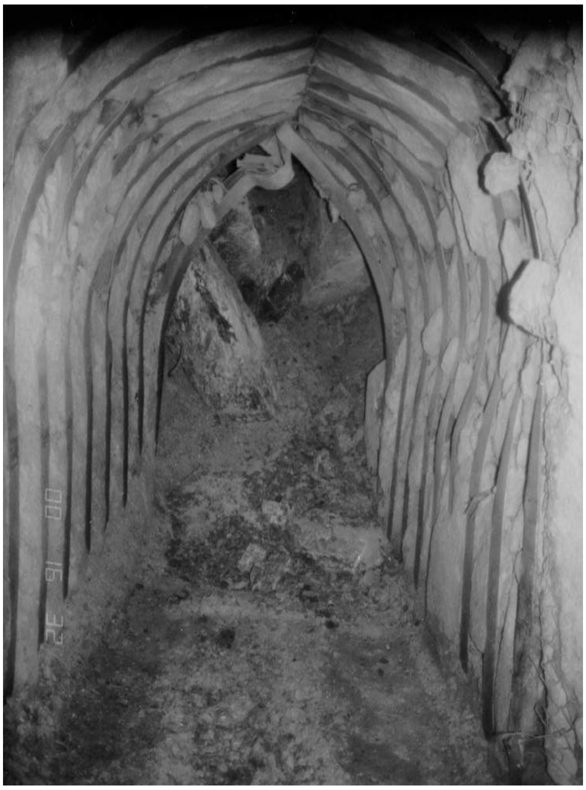


Figure 6a : Buckling of arch legs.

Figure 6b : Buckling of arch legs.

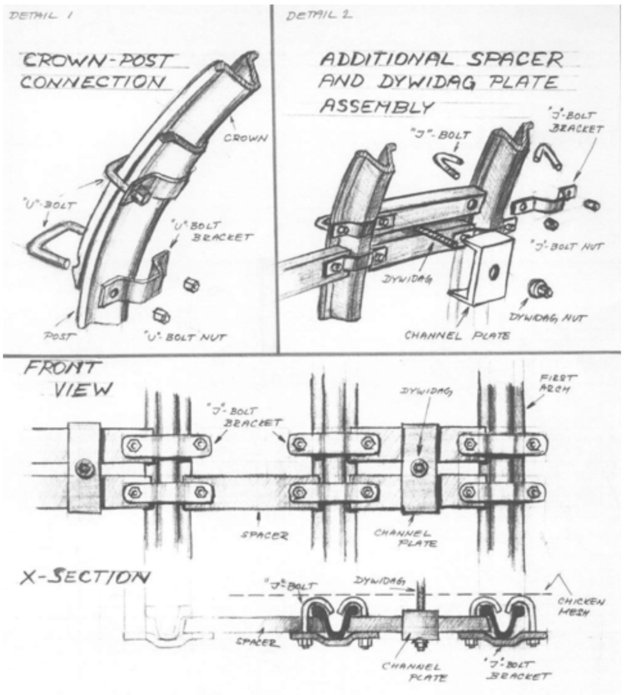


Figure 7a : Connecting channels and 'J' bolts of yielding arch

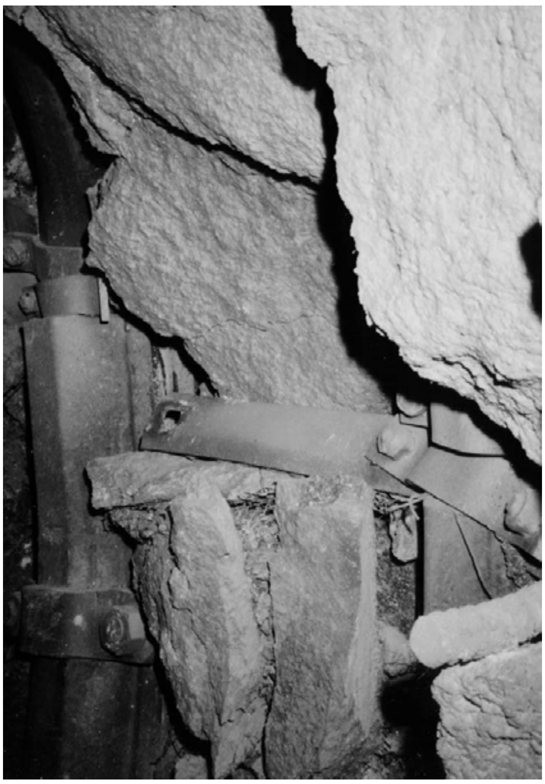


Figure 7b : Failed spacer between two arch legs

Figure 8 : New system offered flexibility and strength to the yielding arches

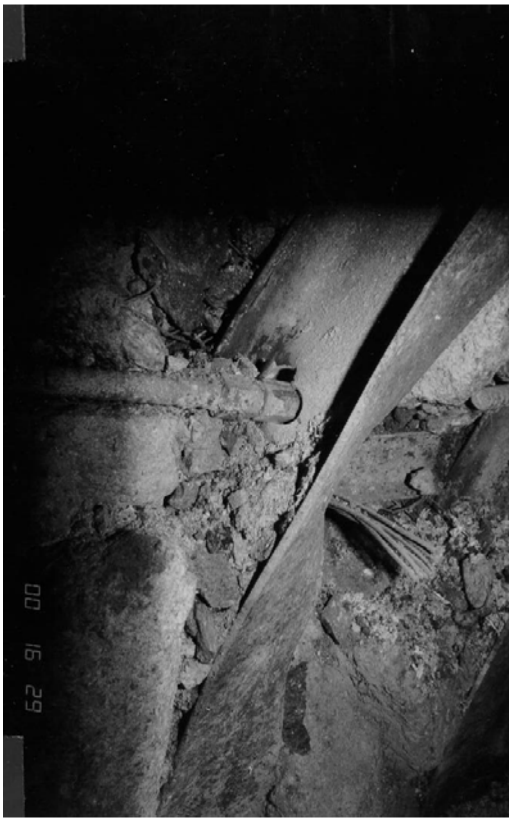


Figure 9 : Failed connecting rod between the rigid 'H' profile arches. The connecting rod pulled through the central flange of the arch.

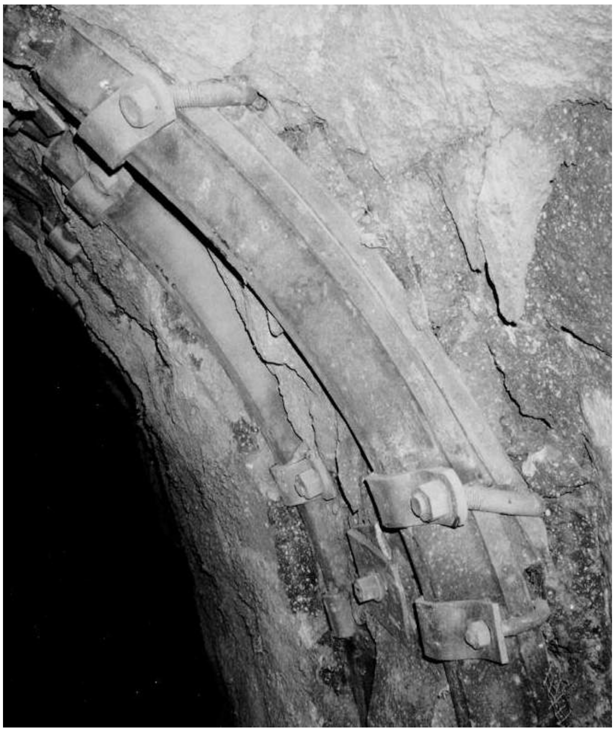


Figure 10a : Failed 'U' clamp -yielding arch

Figure 10b : Failed ' U' clamp - yielding arch

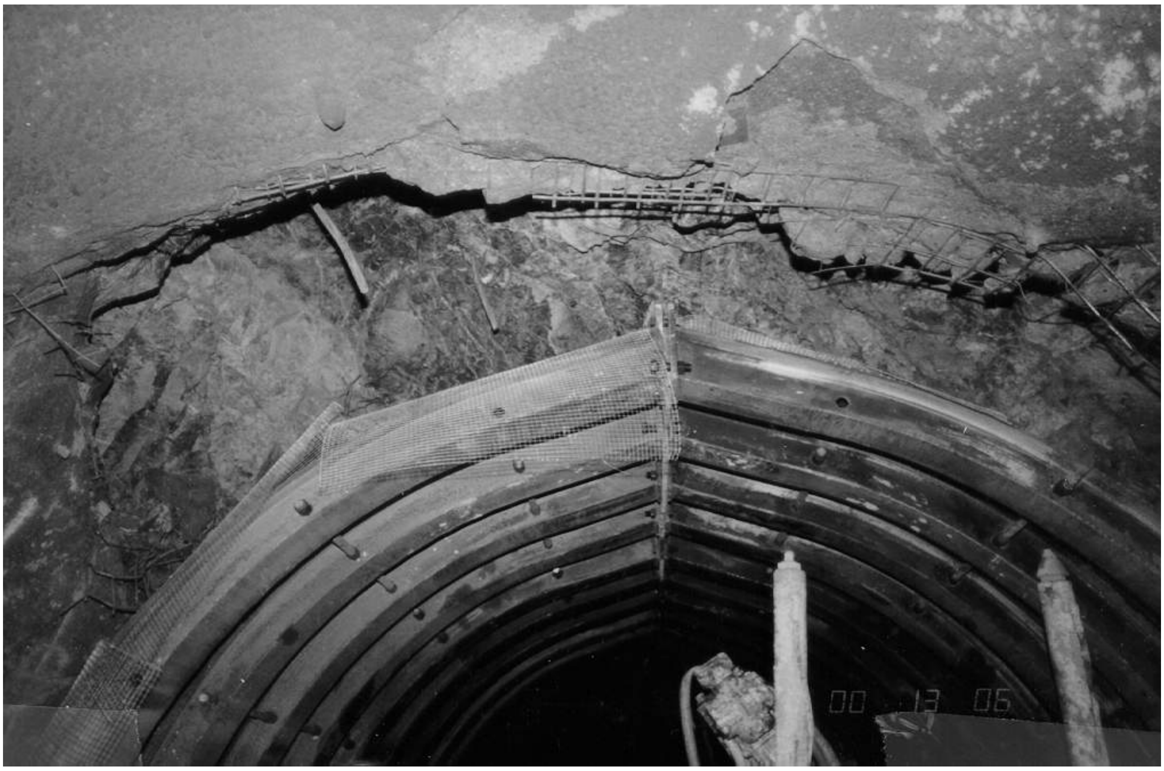
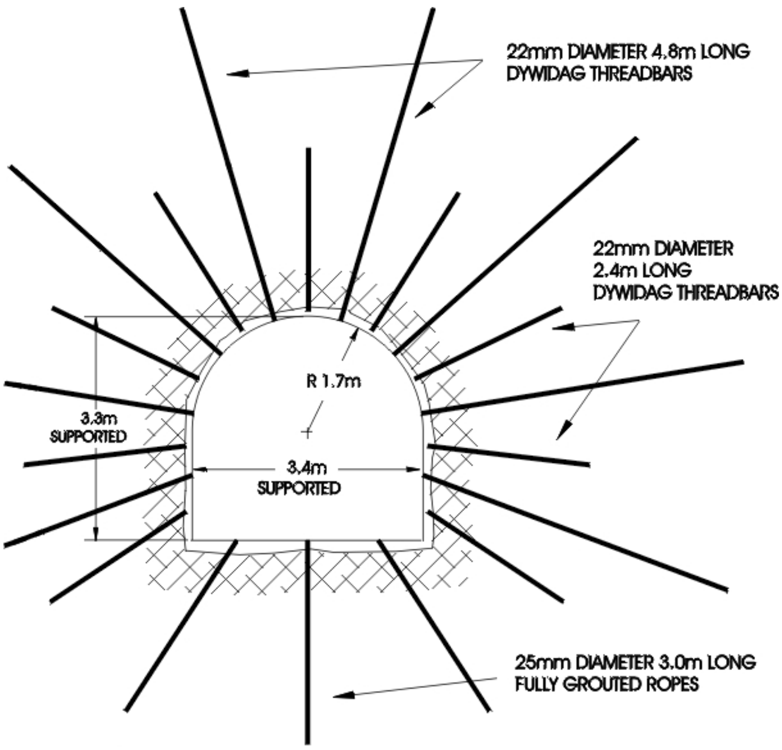
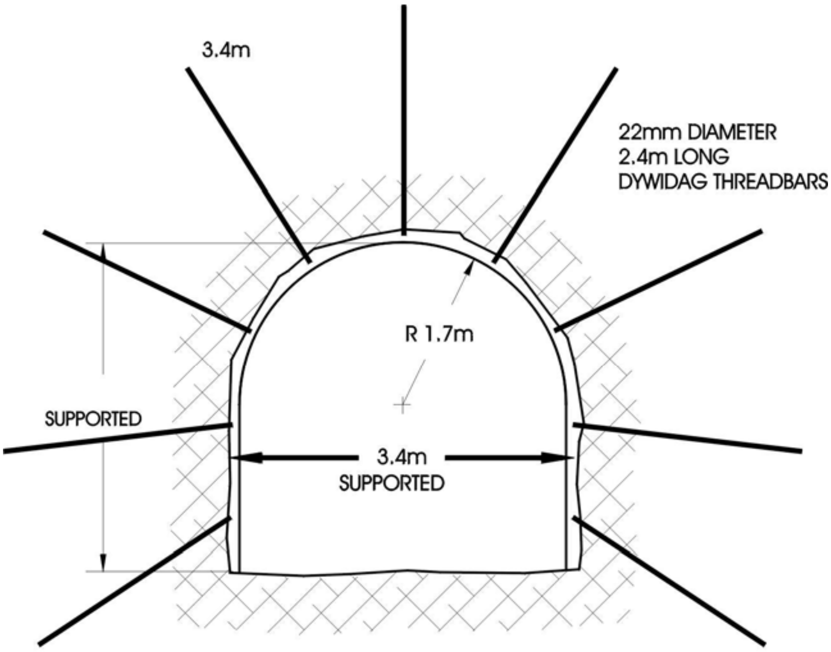


Figure 10 : Initial reinforced shotcrete had to be slashed to increase the size of the tunnel in order to fit arches.



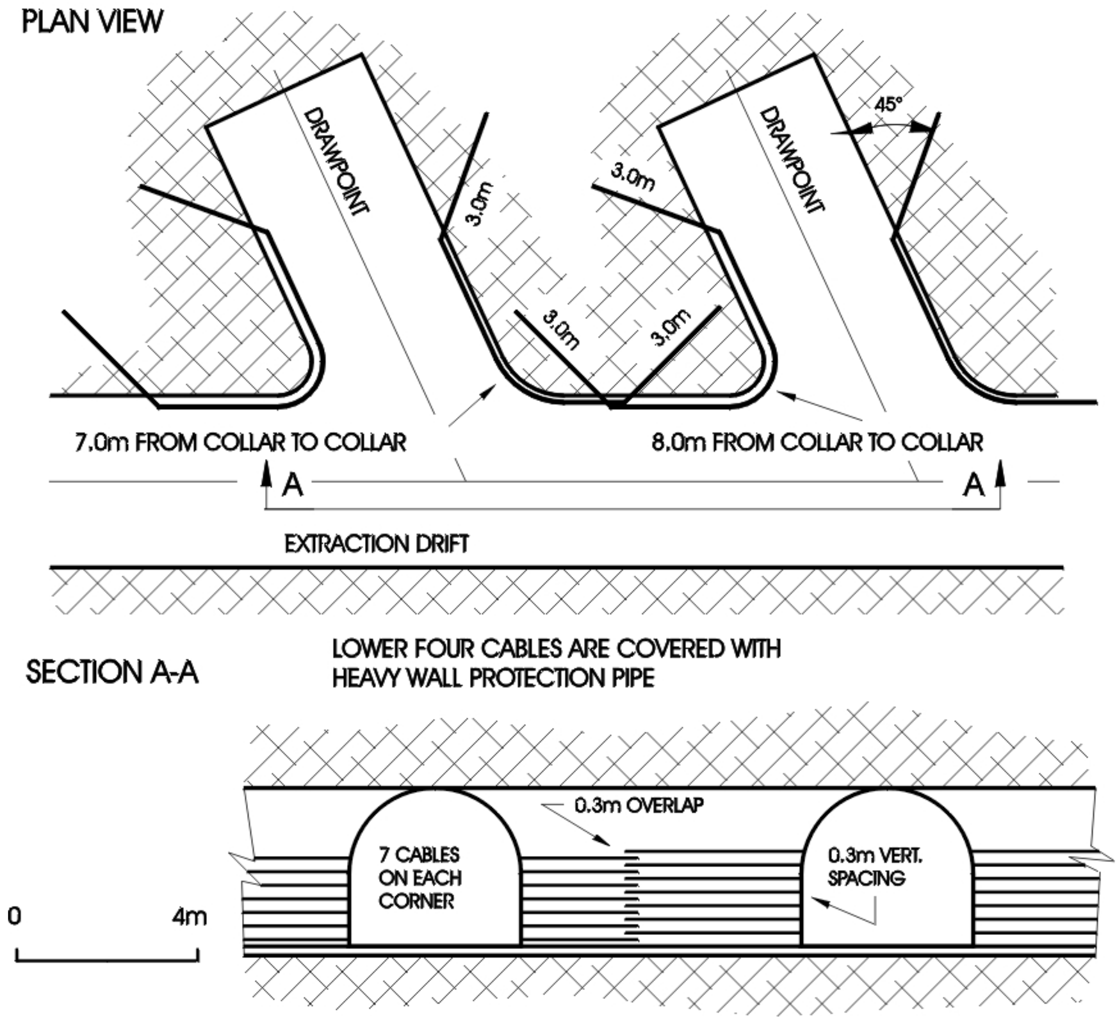
Layout of rock support on 1350 South

11- 25mm Ropes and 9 - 2.4m dwyidag bolts, fully grouted to provide rock reinforcement support in squeezing ground.

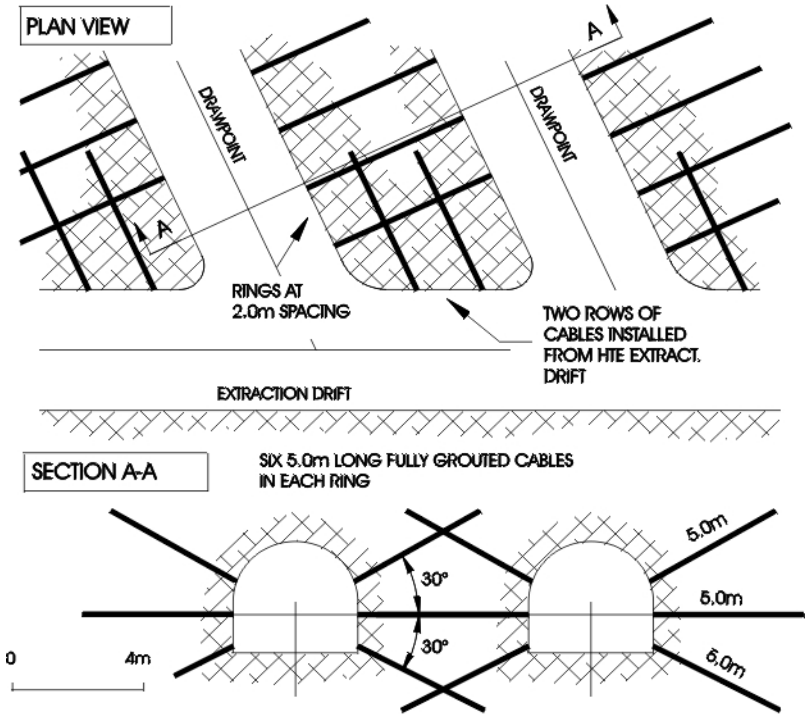


Layout of rock support on 1350 North

9 rock bolt pattern



Bullnose and camelback support in potential squeezing ground



Pillar cable reinforcement

Cable bolt support in extraction drift and drawpoint

Comment From N.J.W.Bell

TH Arches for Support

It has been found that by increasing the arch height by some 0.2 metre using a 1 metre length of arch and another pair of clamps on each of the legs a longer life is obtained as the floor can heave and the vertical joint takes up the movement. The overlap also reinforces the bottom of the leg, which is held ‘rigid’ in footing channel irons, cross bracing etc. Previously the lower half of the leg had proved to be a weak point and where they buckled. These are now reinforced and this strengthens the whole structure.

DESIGN TOPIC

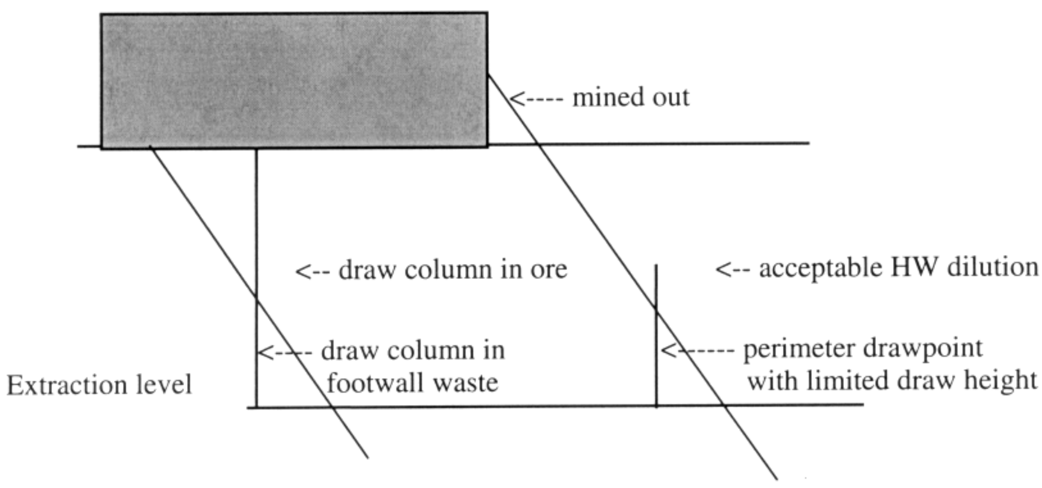
Draw Column Heights

GENERAL

The draw column height is the economic height of diluted ore that can be drawn from that extraction level under the defined capital cost, operating cost and profit conditions. The average draw height determines the viability of the operation. There is the possibility that there are lesser ore heights in the perimeter. Here the mineral tonnage / value in the column need only to pay for the capital cost of the extension to the production drift, drawpoints, ventilation drift and haulage and the value can meet working costs plus an acceptable profit.

EFFECT OF OREBODY GEOMETRY

The effect of orebody geometry is most noticeable in dipping orebodies.



POTENTIAL DRAW HEIGHTS AND DILUTION

If the interactive theory is accepted, then the higher the draw column the lower the dilution. This is on the basis that the ore / dilution interface will be maintained as a distinct zone and that dilution will only enter the ore column when the ore/ waste contact reaches the height of the interaction zone:-

- Ore column height = 300m, height of interaction zone = 80m, dilution into draw point after 220m drawn, dilution entry percentage = 73%
- Ore column height = 200m, height of interaction = 80m, dilution into drawpoint after 120m drawn, dilution entry percentage = 60%

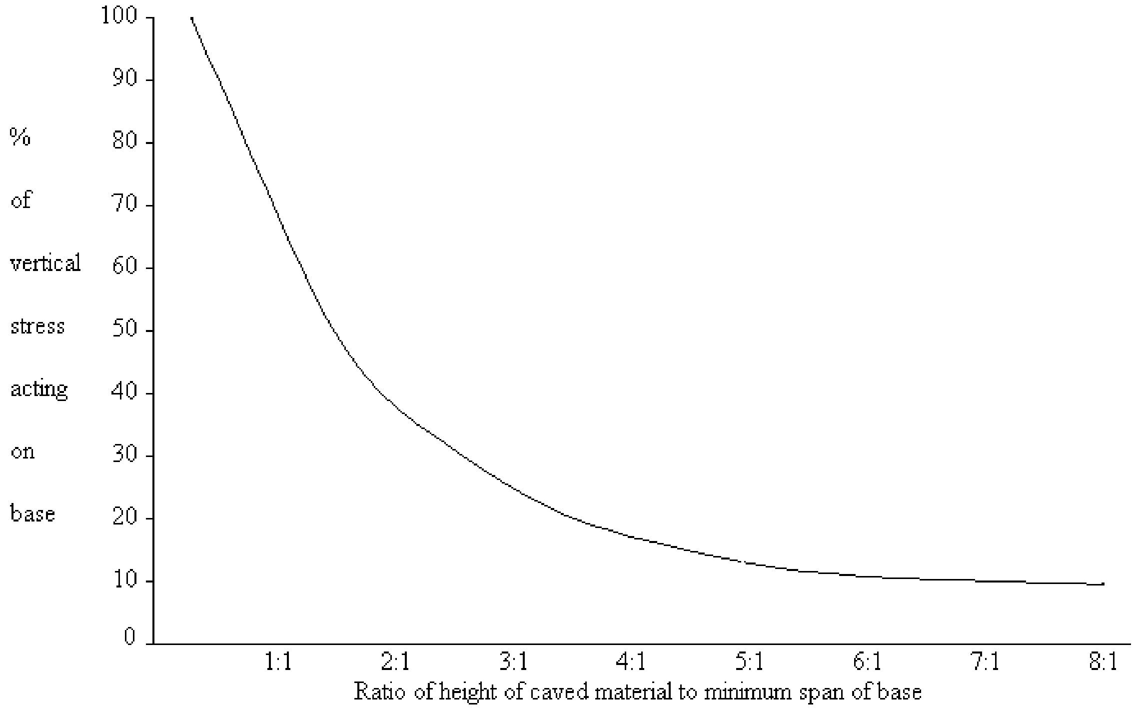
In the case of coarse fragmentation, the height of the interaction zone is very much a contact zone rather than a distinct line, in fact the height of interaction represents the degree of mixing - discussed elsewhere. Where the basis for calculating dilution entry is based on the percentage drawn, then the quantity of dilution in the draw column increases as the column height increases. If the entry percentage is 60%, that would apply to the ore column regardless of height. Compare this with the height of interaction zone calculation where the dilution entry is 73%.

EXCAVATION STABILITY

High draw heights result in a larger volume of ore through the drawpoint which means the attrition effects must be identified. It is a fact that 300000 tons have been drawn through drawpoints in good rock. Excavation stability means that the draw program can be adhered to, while repeated repairs result in irregular draw and an increase in the draw control factor and earlier dilution entry.

5.0 GEOMETRY OF DRAW ZONE

The vertical load on the base of the caved material is a function of the minimum width to height ratio of the draw column. In the case of narrow columns, the bulk of the load is carried by the sides. If the column height is high and the mining area has an appreciable width, then the load on the extraction horizon can be significant, particularly if the pillars are not strong. The following graph was developed from experiments conducted in a 3-D model with load cells on the base and sides. The base 750mm x 750mm with a height of 2400mm. The model was loaded to different heights with sand in the lower 500mm for draw purposes and then with aggregate, rock particles, core randomly placed and core vertically placed. There was very little difference in the results for the different material s- the vertically placed core having a 15% higher base load than the aggregate:-


100
90
80
70
60
50
40
30
20
10
0
%
of
vertical
stress
acting
on
base
1:1
2:1
3:1
4:1
5:1
6:1
7:1
8:1
Ratio of height of caved material to minimum span of base

DESIGN TOPIC

Mining Sequence

GENERAL

This section will list all the items that will influence the sequence and assess their influence. These items are described in detail in the appropriate sections. In some cases there might be conflict, in which case there has to be a rating system and a decision taken.

CAVABILITY

A potentially difficult caving orebody might require a certain sequence and this would override other sequence considerations. For example, in order to ensure caving, the best sequence might be towards the high stress, but this would result in high abutment stresses in a conventional layout. Advance undercutting and the optimum level of support would cater for those problems. It is essential that overhangs are not allowed to develop. The possibility of this happening must be examined and remedial action taken such as undercutting from the strong to weak ground.

OREBODY GEOMETRY

The plan shape could offer limited options, the best and worst directions must be recorded. Should the mining start at a narrow end with problematical caving or the wide end with assured caving? If the production demands are not severe then starting at the narrow end would be better so as to give the rock mass time to fail. This might require a larger undercut area, but in the long term the block can settle into a steady production rate.

INDUCED STRESSES

Proper identification of the induced stresses with different sequences is required. The influence of induced stresses has been discussed and needs to be incorporated into sequence decisions.

PRIMARY FRAGMENTATION

Primary fragmentation is influenced by orientation and direction of the cave front. This information would be available from the fragmentation section and fed back here. If there is a large variation in primary fragmentation then this must be recognised in deciding on the mining sequence. In the long term, the best results could be achieved by starting in the more competent areas and ensuring the cave was complete, even though the build-up might be slow. In the mining is started in the less competent then early production is achieved, but this could lead to a chimney cave or the formation of overhangs with the subsequent loss of ore.

PRODUCTION REQUIREMENTS

Obviously production requirements are very important, but can only be achieved if the caving has occurred at the right rate and ore is available through the drawpoints. tonnage. Block caving is a long term mining operation and short term expediency must not influence sequence decisions.

INFLUENCE OF GEOLOGICAL ENVIRONMENT

All factors that could affect a sequence decision must be identified and there is a need to list them otherwise something might be ignored.

GRADE DISTRIBUTION

There is always the temptation to start the mining in the high grade and work outwards. This can only be done if there are no long term ill effects.

ROCKBURST POTENTIAL

A very important factor but effects should be reduced with advance undercutting and correct procedure. Advance undercutting does not eliminate the possibility of seismic events occurring with the rate of cave propagation and the direction of undercutting with respect to structures and major infrastructure.

MASSIVE WEDGE FAILURES

Massive wedge failures have occurred on Bell, Teniente, King and Shabanie mines and there are sure to be others. These failures have been the result of mining towards major structures that dip towards the advancing face.

ZONING OF MRMR

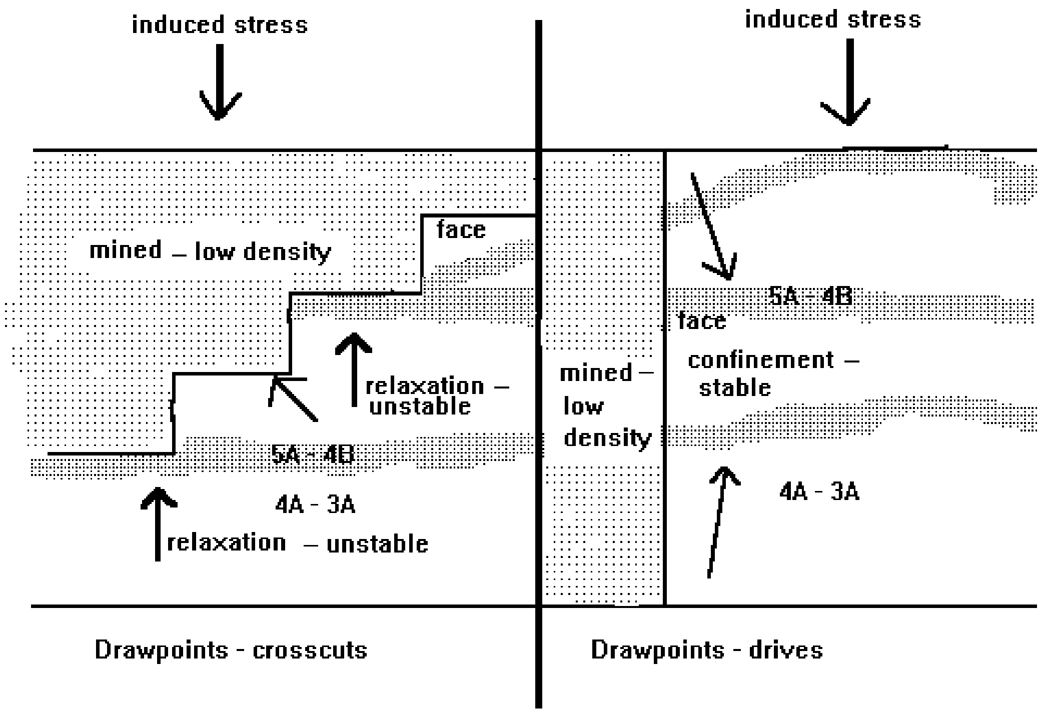
It is sound mining policy to stope from poor to good ground, however, in block caving, particularly with advance undercutting, it might be better to mine from competent to less competent, this will permit the induced stresses in the cave back to work on the competent zone and not have the undercut reach the competent ground when the cave has progressed upwards and the competent zone no longer in a stress caving mode. This has been the experience at San Manuel and King Mine. This might also have been the situation at Northparkes, where the south-west area was rapidly drawn allowing the cave to move up rapidly and have the caving stresses located well above the critical zone. It is significant that where the draw was slow (north-east) and stresses had time to work on the rock the cave had a vertical attitude.

INFLUENCE OF AND ON ADJACENT OPERATIONS

The influence of adjacent operations on the sequence must be recognised - do not advance towards old workings and create remnant pillars. Also do not advance towards a current operation and create problems with high abutment stresses.

KING MINE SEQUENCE

The following diagram shows how the incorrect sequence with leads and lags can result in major instability problems compared with a straight face, where the rock mass is confined:.



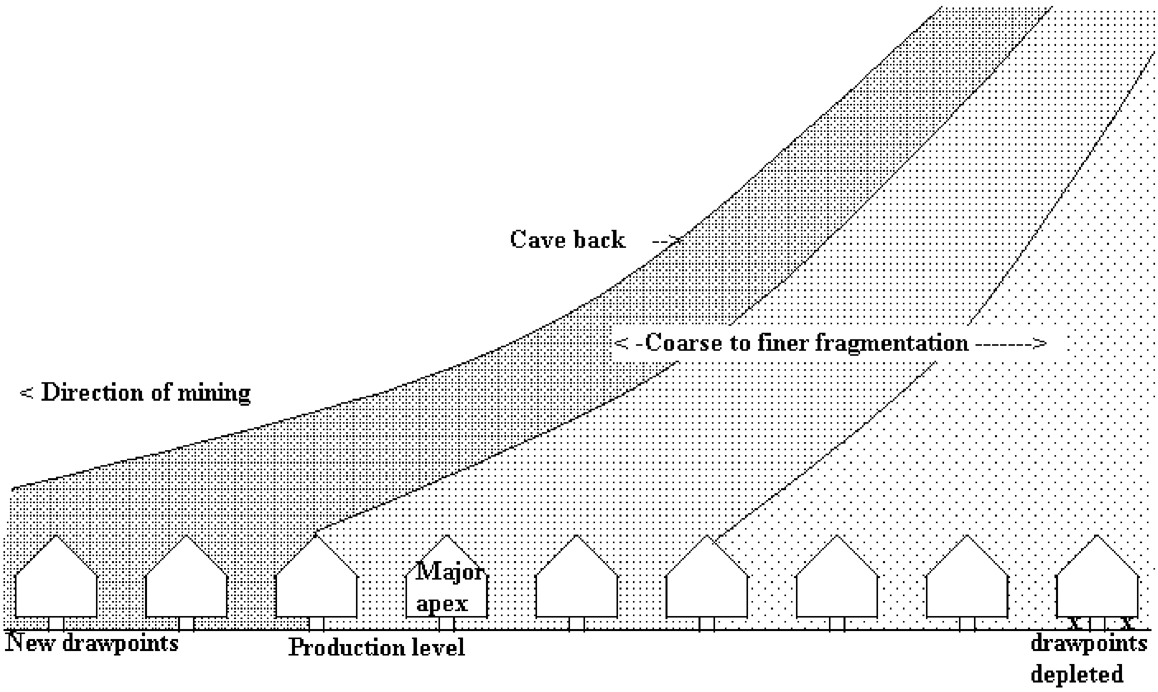
KING MINE - WEST/CENTRAL fig.

MUD / WATER INFLOWS

If there is a potential for mud / water inflows the sequence must recognise this and if it is not possible to adjust, then precautions must be taken

PANEL RETREAT

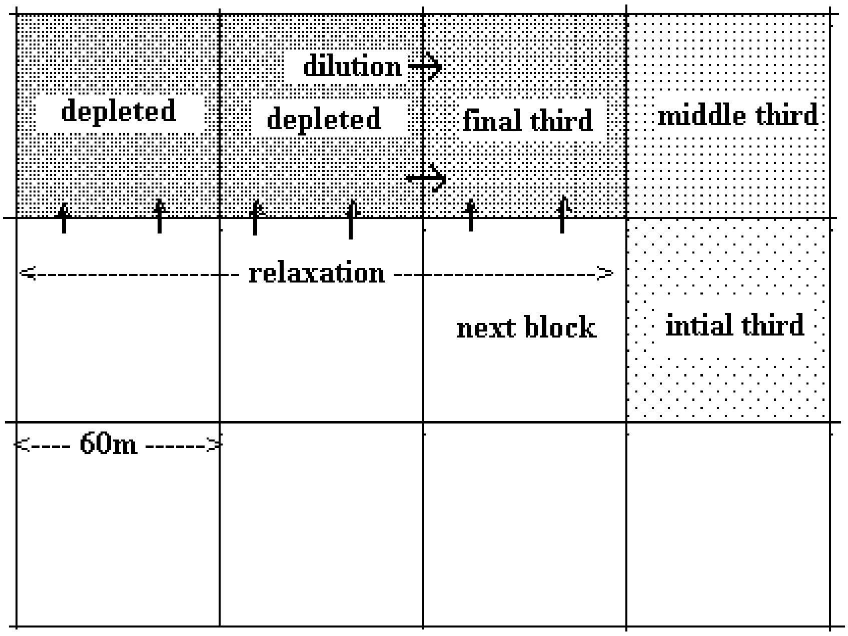
Panel retreat sequence consists of starting the undercutting / drawpoint commissioning from a start line, and then advancing the face in a certain direction and with a specified face shape and length. The start line could be the orebody boundary or a line anywhere in the orebody. The principle is that once the production area is established,then new drawpoints are commissioned as drawpoints are depleted. A straight or concave face shape ensures maximum stability to the development by confining the rock mass. The sequence results in an average grade as the drawpoints are drawing material from different heights in the column and an average fragmentation. This means that equipment selection can be optimised and mineral production can be accurately forecast. The ideal angle to the cave back is 45 °, but this is not always possible, particularly if the draw columns are high.



This is a continuous operation which makes production scheduling simpler. It is essential that an air gap is not allowed to develop otherwise there can be migration of material down the cave back, as this could lead to the early introduction of dilution.

MINING BY BLOCKS

In this sequence the footprint is divided into blocks of specified dimensions e.g. 60m x 60m as at Andina Mine. A block is fully undercut and mined out. If the object is to have an average grade and an average fragmentation then at least three blocks must be in production at any one time and in different stages of draw.



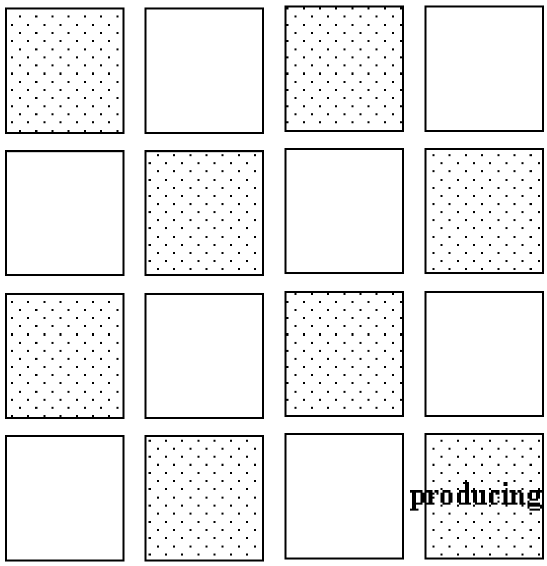
Mining by blocks in sequence

The above pattern follows a sequence across and down the footprint. Another sequence that has been used is the 'flying wedge' which is basically a 'V' with apex in the centre of the footprint. In all cases the new block will have sides that have had a long period of relaxation. With dilution entry from the overlying ore / waste contact and side infiltration from the adjacent depleted block. The side dilution is limited by leaving an ore pillar in the depleted block by restricting the draw from the boundary drawpoints. This tonnage must be reclaimed at a later date and hopefully the drawpoints remain in a good condition.

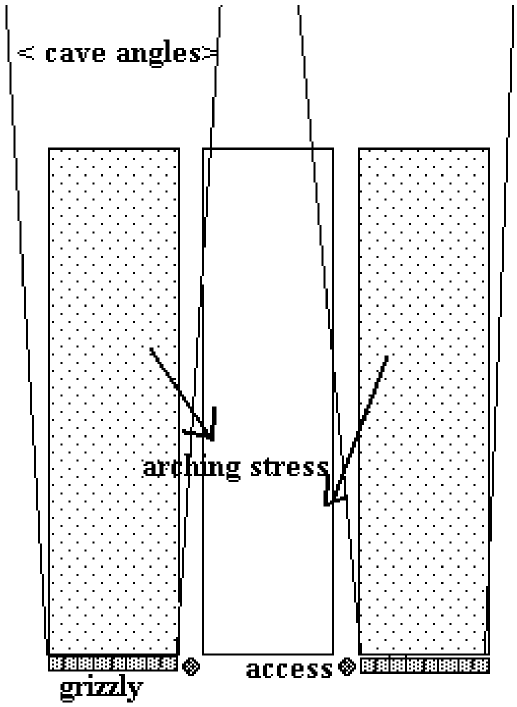
There is also the situation where mining by blocks is planned over a long period of time say fifteen years and not much thought is given to any possible complications ten years down the line. For example, it is convenient to draw in a narrow block, but when it comes to mining the block is too narrow to cave and an overhang develops.

CHECKERBOARD

The checkerboard system was used on some grizzly operations, but discarded because of failure of the production development in the secondary blocks owing to 'pressure' from the laterally free column of rock over the production horizon. The mass of this column was increased by the transference of arching stress from the drawdown of the surrounding blocks.



Checker board - plan view



DESIGN TOPIC

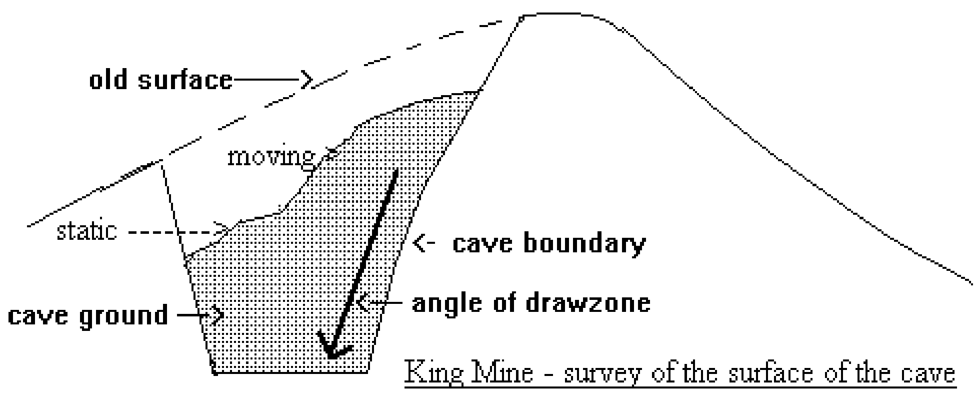
Angle of Draw

GENERAL

This section shows that under different topographical and geological conditions the angle of draw or movement of caved material into the drawpoint can vary. On some operations these characteristics have been recognised in the design.

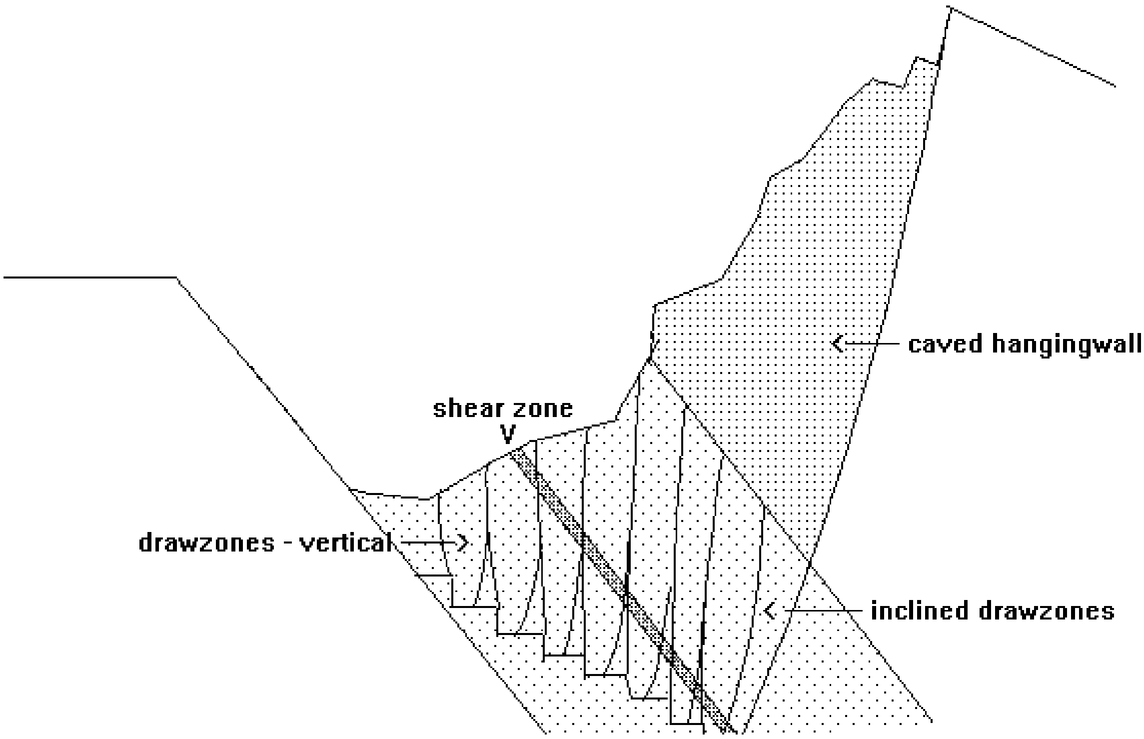
INFLUENCE OF TOPOGRAPHY

At King mine it was established that the drawzones angled towards the caved high ground:



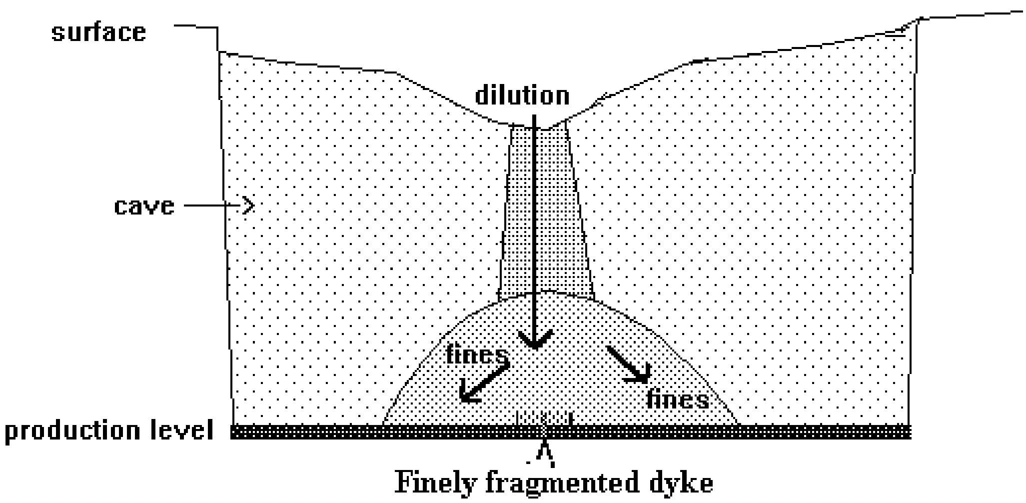
It was noted during the course of surveying the cave surface that the area vertically above the bulk of the drawpoints was static, whilst the slope against the hill was moving down. This implied that drawpoints need only cover half the orebody to recover the ore on the hangingwall side of the orebody.

As a major internal shear zone prevented the development and mining from a horizontal drawpoint layout in it, the angled draw phenomena meant that an incline drawpoint layout could be designed for the next lift so as to recover the ore on the hangingwall side of a large shear.



King mine - Incline drawpoint layout with angled draw towards high ground.

INFLUENCE OF INTERNAL STRUCTURES

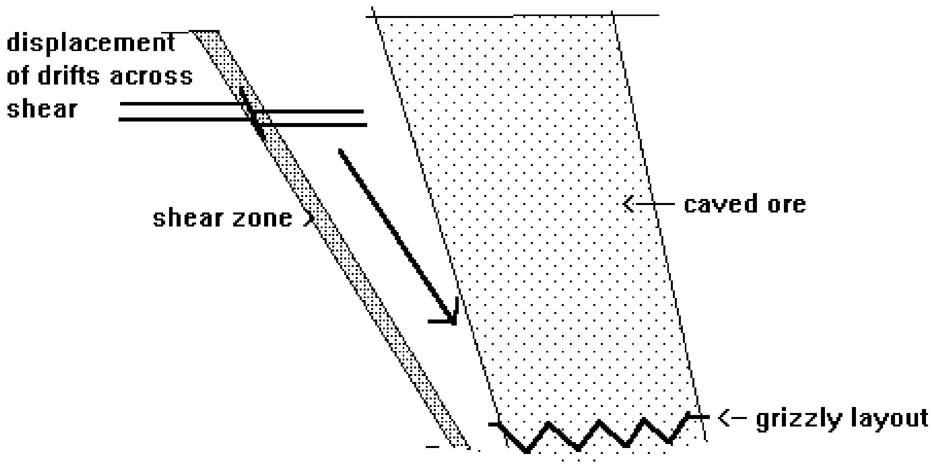


Preferential draw of fines into central drawpoints with more rapid lowering of the surface in the centre and rapid inflitration of the dilution

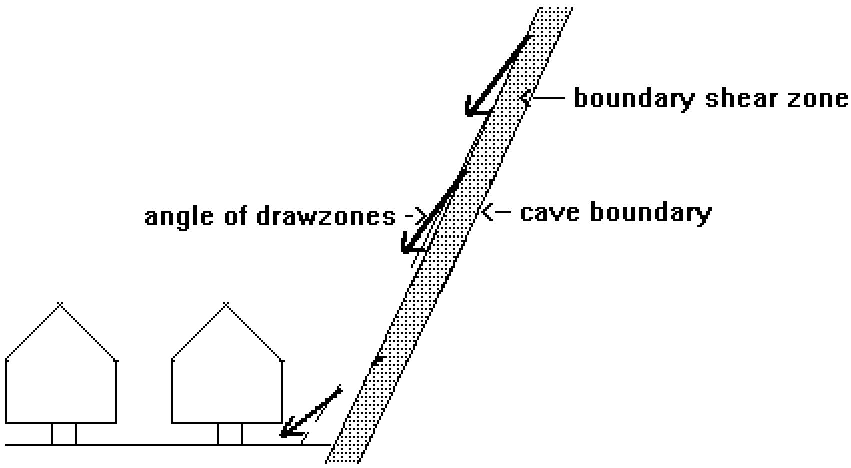
Major shear zones within the orebody will localise draw and lead to angled migration of material along the line of the shear zone.

INFLUENCE OF EXTERNAL STRUCTURES

At Havelock Mine in Swaziland, there were massive movements on the hangingwall of a footwall shear as the ore was drawn as shown by the downwards displacement of a drift on the hangingwall side of the shear -



This behaviour can be used to advantage in order to draw ore sitting on a footwall shear:



Major structures often define cave boundaries and material will be drawn down the structure resulting in an angle to the drawzones.

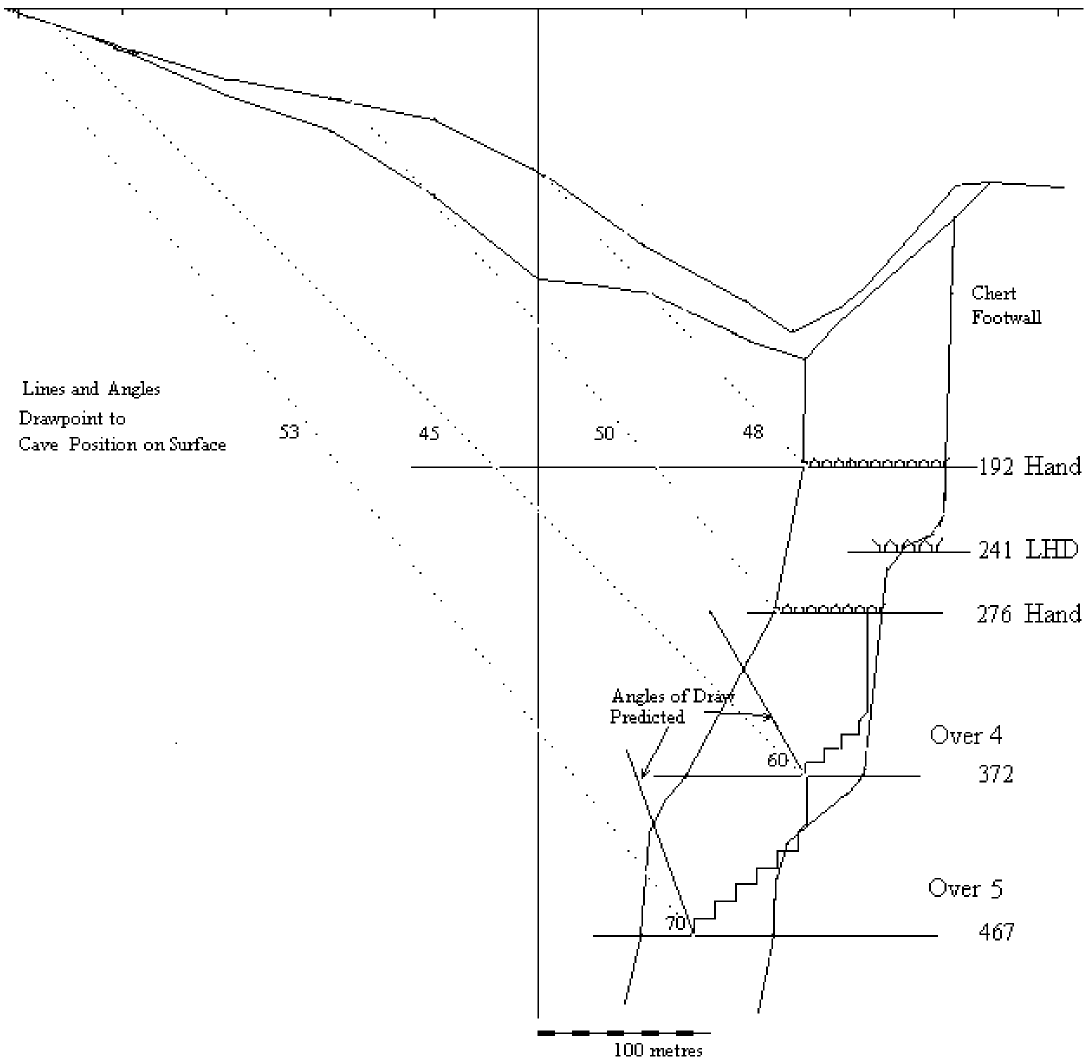
INFLUENCE OF RANGE IN FRAGMENTATION

Fine fragmented material will move through coarse material and if there are distinct fragmentation zones then drawzones will be angled. The only solution is to break coarse material as soon as it reports in the drawpoint

COMMENTS FROM N.J.W.BELL

Gaths Mine, King Section

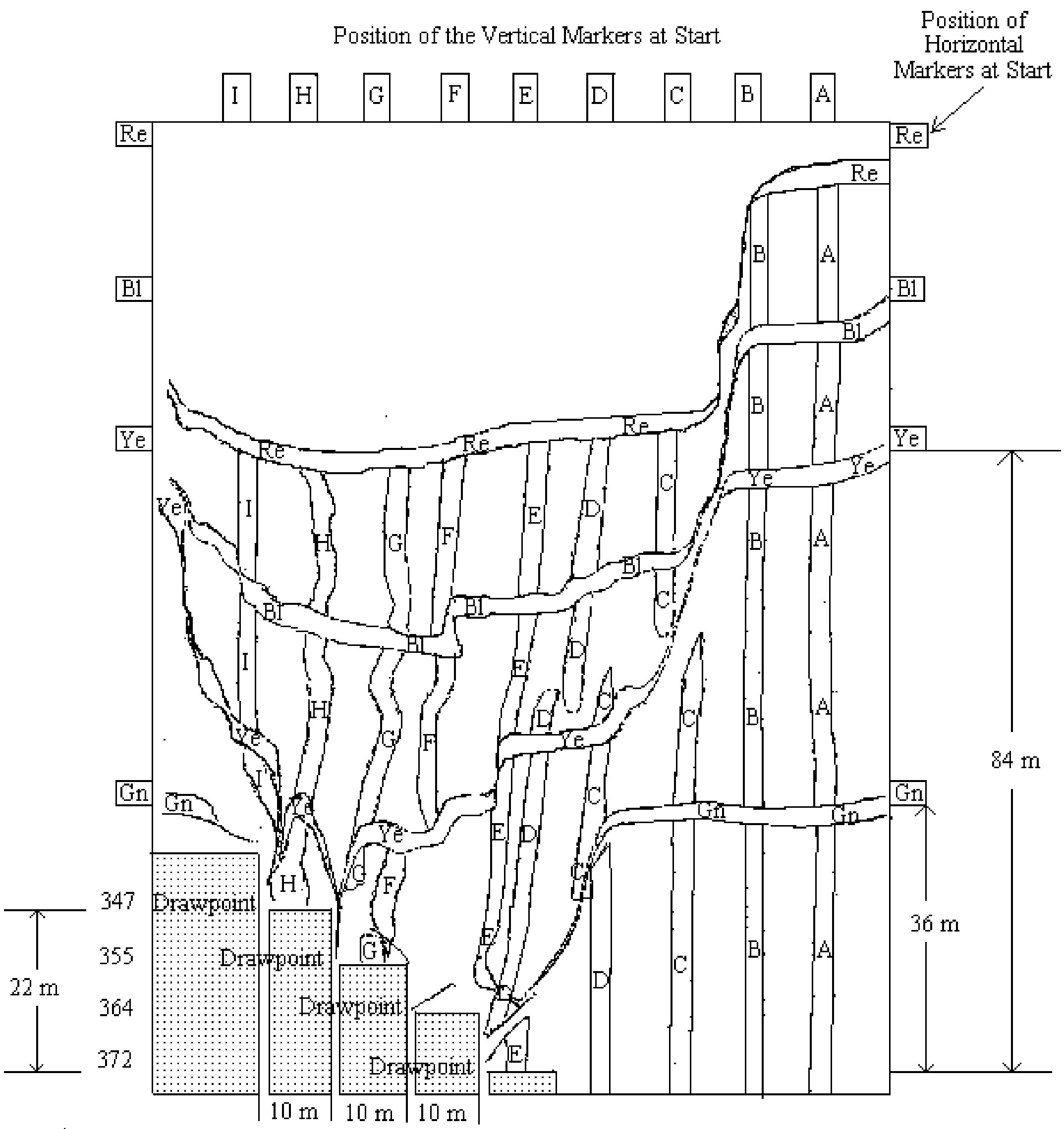
It was noticed during the course of surveying cave surface that the area vertically above the draw points was static while the slope against the hill was moving down.



Gaths Mine - King Section

Also noted that the TH arches in the extraction X-Cuts on 276 level were leaning back away from the hill as the load through the crown pillar pushed them. The displacement of drifts seen along structures at King mine where the main draw affected development in the Western and West Central ore bodies. Model tests were carried out in the draw model at Shabanie Mine to simulate the hill feature and showed conclusively that the incline draw takes place in a very significant manner. Dividing the model and loading the side opposite to the false footwall created the hill feature.

A Section Through One Of The Experiments



Draw Control Model

Experiment 4/82 - Diagonal Section Through False Footwall with Hill Pressure and Differential Draw for Uniform Lowering of Ore - Waste Interface

Viewing all these factors and the various angles involved, the false footwall mining method was devolved using a 60º bottom angled draw into the hillside. This was the layout parameter for Main Over 4. After the completion of draw and collection and analysis of the draw markers the angle was modified to 70º.

PHOTO 1 & 2 – show two different configurations for the false footwall layout. Photo 1 shows where the ‘minor apexes’ are all in line and the drawpoints all in line and hence have very little in the way of ‘major apex’. Photo 2 shows the staggered arrangement that was modeled and is currently in use at Gaths, King Section.

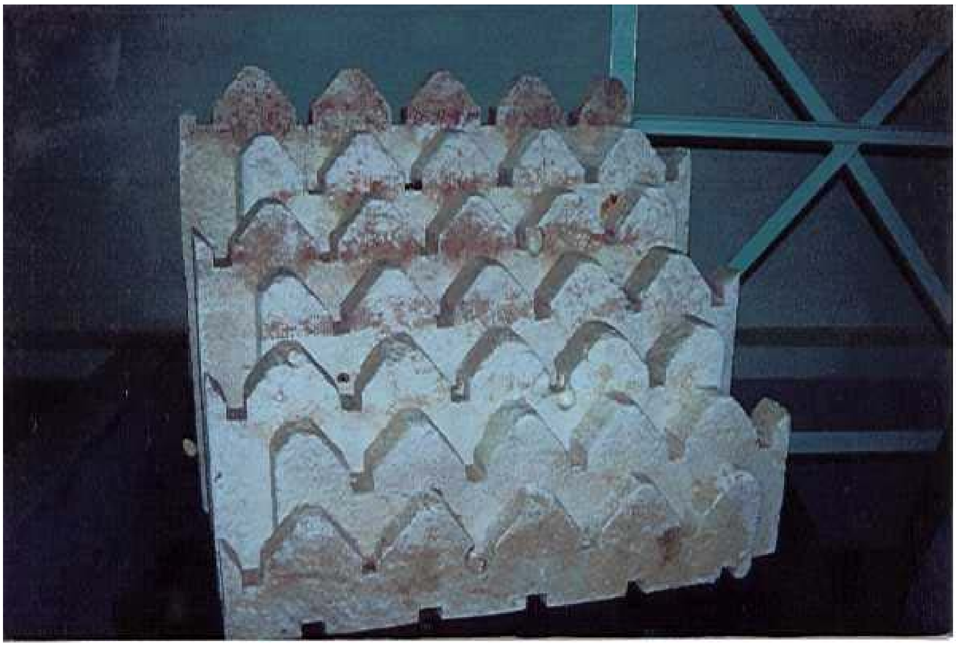


Photo 1

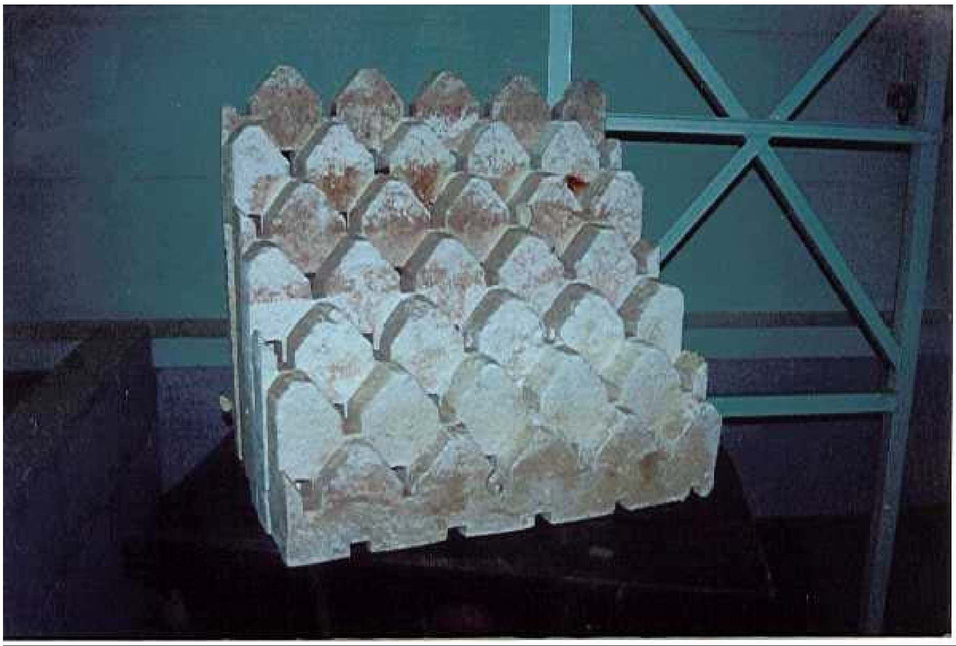


Photo 2

DESIGN TOPIC

Potential Draw Rate

GENERAL

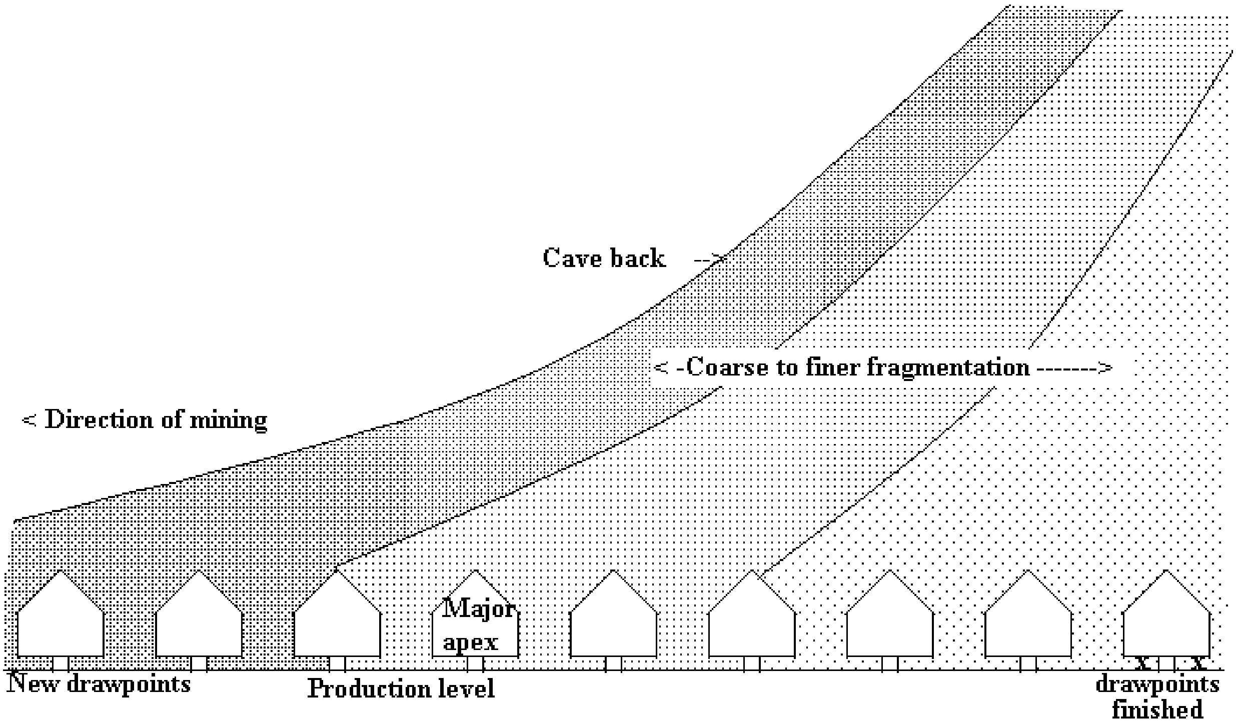
Draw rates must be based on :-

- the requirements for the safe propagation of the cave,
- avoidance of seismic events,
- adhering to the draw control program.
- variations in fragmentation
- And not on some idealised production rate.

This section identifies the relevant items which have to be taken into consideration in deciding on the optimum draw rate. These items are described in detail in their relevant sections

FRAGMENTATION

Fragmentation plays a major role in deciding on draw rates, particularly in the early stages of commissioning drawpoints. The advantages of a panel retreat system are that once the first panel is mining, commissioning of new drawpoints is balanced by the drawing of finer material in the latter stages of the back drawpoints.



To achieve the same result with the caving of individual blocks, requires at least three blocks in operation simultaneously, but at different stages of draw. This requires excellent scheduling and can result in draw rates that are too low.

CAVE PROPAGATION

The draw rate has to match the rate of cave propagation otherwise large air gaps form which could create an unfavourable air blast situation. It has been shown at Teniente Mine that when the rate of draw is in the order of 300mm per day or more, seismic events occur in the cave back. It has been found that draw rates of 150mm per day are acceptable until the full cave column has developed.

OVERSIZE AND HANGUP INCIDENCE.

The draw rate is influenced by the ability to clear drawpoints of oversize and prior knowledge of the fragmentation will ensure that the right equipment and techniques are available. The incidence of hangups and the time required to bring these down, affects the rate of draw and the number of drawpoints required to maintain production. Once again a panel retreat sequence balances out these factors resulting in far better production planning.

ORE HANDLING - SIZE OF LHD, SIZE OF OPENINGS

There is little point in having very large LHDs which can handle 4m³ rock blocks when the orepasses can only accept 1m³ material. The large LHD will have to wait for suitable sized material to be made available by secondary breaking. Obviously the larger the material that can be handled by the ore handling system, the higher the productivity.

REPAIR CYCLE

Repair cycles form part of many old cave layouts which were designed without recognising the effect of abutment stress on a pre-developed rock mass. This meant that production areas would be described as having 70% availability, resulting from repairs or hangups. Recognition of all the factors that contribute to ground failure and production problems means better design and an increase in available drawpoints.

PRODUCTION POTENTIAL

Is a combination of all the above factors and is a number that can be assigned to a drawpoint of a certain size. For example a grizzly drawpoint with a 1.5m x 1.5m opening and discharging into a long orepass can fill that orepass with 1000 tons in a shift, that is 125 tons per hour. However, a LHD drawpoint of 4m x 3m might only produce 90 tons per hour owing to the coarse fragmentation.

The planned draw rate cannot exceed the production potential of the drawpoint for different stages of draw.

Comments From N.J.W.Bell

General Description - Draw rates must be based on facts and not hopeful production calls which are always over optimistic and unobtainable and causes friction between management and production personnel. Draw rates can increase as caving progresses and fragmentation improves. Being constantly aware of the light of the cave back relative to the broken ground (air blasts and seismic risks.)

Production Potential - The planned draw rate cannot exceed the production potential.

Production Status - The rate at which the undercut is to be blasted and lashed, is to be planned and recorded. This will give the initial call that is possible from block dependent on the size (height) of the undercut face that is to be mined.

It has often been stated that the early draw from drawpoints should be at a reduced rate building up to the final rate planned. The reduced rate should be calculated as mm per day after the coning is complete, as per the tabulation below.

Draw Rates at the Start of a Cave Block Single Drawpoint with Time					
		Drawpoint Spacing			**Remarks**
Drawpoint Spacing		15 X 15	12.5 X 12.5	10 X 10	
Drawpoint Area		225	156	100	
Density used this example					2.7
Days after Coning	**Mm Per day**	**tonne**	**Drawn / day /**	**Drawpoint**	**Remarks**
1 to 14	50	30	21	14	
15 to 28	75	46	32	20	
29 to 42	100	61	42	27	
43 to 56	125	76	53	34	
> 56	150	91	63	41	Recommended Maximum Rate
	200	122	84	54	AA Mines Normal Rate
	300	182	127	81	} Watch for Seismic Events
	400	243	169	108	} at these higher rates
High Draw Rates are only possible in free flowing easily loaded small uniform material					
Pre broken ground Production					
Source	Days	tonne	Drawn / day /	Drawpoint	Remarks
5 m high Undercut	10	152	105	68	50 % of broken ground lashed
Coning	5	152	105	68	50 % of broken ground lashed

What the maximum production rate planned should be must be stated and recorded. 150mm to 200mm a day to be seems to be a reasonable maximum. Gaths and Shabanie Mines fall into this rate of draw.

During the coning and undercutting, lashing from the draw point should be limited to a maximum of 50% of the tonne broken. There is a tendency to want to pull at higher production rates because the tonnage is pre-broken, but this does not allow the cave the time to develop and leads to subsequent difficulties and potential danger.

Control must be exerted if the planned production rates are to be achieved satisfactorily. This is clearly indicated in the spreadsheet below.

Draw Rates at the Start of a Cave Block					
Days	**Mm/ Day**	**Tonne Drawn / day / Drawpoint**			**Remarks**
		15 X 15	12.5 X 12.5	10 X 10	Drawpoint Spacing
		225	156	100	Drawpoint Area
		Undercut and Cone tonne Lashed			
Initial		2280	1575	1020	50 % During the Undercut/Coning Phase
Remainder		2293	1593	1019	Remainder of the Pre-Broken Ground
1	250	152	105	68	}
2	250	152	105	68	}
3	250	152	105	68	}
4	250	152	105	68	}
5	250	152	105	68	} Undercut Swell 50% Relief
6	250	152	105	68	}
7	250	152	105	68	}
8	250	152	105	68	}
9	250	152	105	68	}
10	250	152	105	68	}
11	250	152	105	68	]
12	250	152	105	68	]
13	250	152	105	68	] Coning Swell 50% Relief
14	250	152	105	68	]
15	250	152	105	68	]Swell Relief complete
16	50	30	21	14	)
17	50	30	21	14	)
18	50	30	21	14	)
19	50	30	21	14	)
20	50	30	21	14	)
21	50	30	21	14	)
22	50	30	21	14	) First Phase Main Draw
23	50	30	21	14	)
24	50	30	21	14	)
25	50	30	21	14	)
26	50	30	21	14	)
27	50	30	21	14	)
28	50	30	21	14	)
29	50	30	21	14	)
30	75	46	32	20	)
31	75	46	32	20	)
32	75	46	32	20	)
33	75	46	32	20	)
34	75	46	32	20	)

35	75	46	32	20	)
36	75	46	32	20	) Second Phase Main Draw
37	75	46	32	20	)
38	75	46	32	20	)
39	75	46	32	20	)
40	75	46	32	20	)
41	75	46	32	20	)
42	75	46	32	20	)
43	75	46	32	20	)
44	100	61	42	27	)
45	100	61	42	27	)
46	100	61	42	27	)
47	100	61	42	27	)
48	100	61	42	27	)
49	100	61	42	27	)
50	100	61	42	27	) Third Phase Main Draw
51	100	61	42	27	)
52	100	61	42	27	)
53	100	61	42	27	)
54	100	61	42	27	)
55	100	61	42	27	)
56	100	61	42	27	)
57		61	42	27	)
58	125	76	53	34	) Fourth Phase Main Draw
59	125	76	53	34	)
60	125	76	53	34	)
61	125	76	53	34	)
62	125	76	53	34	Last of the Undercut/Cone Material lashed
63	125	76	53	34	Start of Free Caving Ground
64	125	76	53	34	) Fourth Phase Main Draw Continues
65	125	76	53	34	)
66	125	76	53	34	)
67	125	76	53	34	)
68	125	76	53	34	)
69	125	76	53	34	)
70	125	76	53	34	)
71	125	76	53	34	)
72	150	91	63	41	} Main Production Rates Begin
					At the pre-determined rates of draw

DESIGN TOPIC

Draw Strategy

GENERAL

The draw strategy will determine the draw control program. As the object is to remove caved rock, the production call to achieve the right economic goals is of paramount importance. However the studies leading to the decision to go ahead should have identified the optimum production targets and would have identified problem areas.

The capital investment in bringing a caving mine into operation is large and can only be recouped if the mine operates for the planned period. Production, and the planned draw strategy must be implemented and only changed once enough experience has been gained on how the rock mass responds to the mining operation.

The planned strategy must be implemented on the production level. This can only be achieved if all operating personnel are fully aware of the logic behind the strategy.

DRAWPOINT SPACING / DRAWZONE SPACING

The drawpoint and the subsequent drawzone spacing would have been decided after considering all the factors described in that section. The draw strategy must be to obtain the optimum interaction with that spacing. Widely spaced drawzones will require a strict discipline, particularly in the time required to cater for oversize and hangups.

FRAGMENTATION

Accurate fragmentation data for different areas and draw heights is required. This might show that fragmentation predictions are too conservative and that production calls can be increased or that modifications are required to the draw control program to ensure better interaction.

DRAW RATE

The optimum draw rate would have been determined to allow for cave propagation.

Also to avoid seismic events, to stop dilution entry or the possibility of an air blast resulting from the development of a large air gap.

DRAW CONTROL - UNIFORMITY OF DRAW

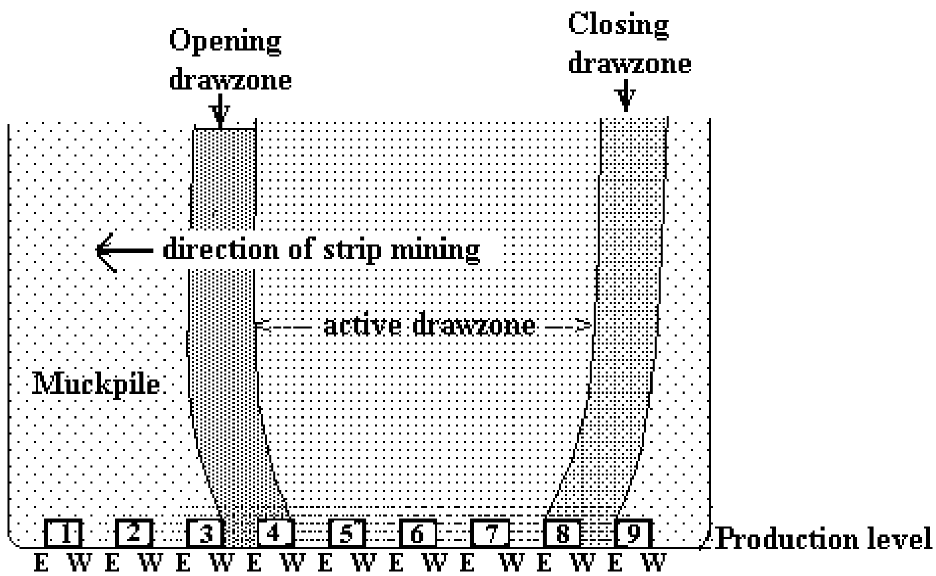
What techniques are going to be used to control the draw from a drawpoint, tons / shift, tons / day or for a longer period? How is the grade of the material in the drawpoint determined, by assaying samples taken in the drawpoint or by a visual assessment of the rock types in the drawpoint? How is the dilution percentage determined?

DRAW PATTERNS - STRIP / DRAWBELL / DRAWPOINT

Draw patterns to obtain optimum interaction have been described in the appropriate sections.

STRIP DRAW

Where a large area has been undercut, but only portion is required for production, the strip method was developed and successfully used on the diamond mines in Kimberley, RSA. The object was to concentrate draw in a strip for a specified time and move across the caved area from one side to the other. This would ensure that there were interactive draw zones rather than draw from scattered drawpoints which would create an isolated draw situation. The strip draw method consists of producing from say four lines of drawpoints for a week then closing a line on the one side of the strip and opening a line on the other side.



Strip draw with drifts numbered and east(E) and west(W) drawpoints

INTERACTIVE DRAW

In the section on drawpoint spacing various draw scenarios were described to obtain the optimum interaction. Drawing lines of drawbells is the best, but as pointed out this does commit all the production drifts to use. Drawing lines of drawpoints - that is bounding the major apex and changing from shift to shift as practised at Henderson is the most practical and offers the best compromise.

RANDOM DRAW

Random draw, as practised on many mines, is to be avoided.

PERIPHERAL DRAWPOINTS

It is not just footwall drawpoints that can be a problem but all peripheral drawpoints. Peripheral drawpoints tend to be dominated by the side of the cave, especially if geological conditions are unfavorable and tend to an overhang, which is always a danger. It is all single side draw points that can have an isolationist effect and draw narrow.

COMMENTS FROM N.J.W.BELL

Uniformity of draw – this is vital to control dilution entry and the following must be considered:

The number of draw points per LHD

These should be sufficient to allow for:

- The fragmentation expected,
- The size of LHD,
- The grizzly apertures and rock breaking facilities that are to be used there.
- Together with the tramming distance from mean loading point to tip.
- The ore characteristics in respect to the ease of loading.

The number of draw points that are allocated are dependent on productivity required from that area or LHD and might well have to be a compromise (inefficient LHD operation but higher tonnages from a smaller area).

Remembering with electric LHD's that these require their own unique area and cannot cross. If LHDs have to share a tip, or a tramming route from the draw points to the tip, productivity diminishes considerably as there is always a wait time. This must be calculated into the productivity calculation.

Where draw control/uniformity of draw techniques are going to be used to control the draw from a draw point – is it tonne per shift or per working cycle, or per day, or per week, or per month? The basis must be set out and adhered to by all.

It has been found that if the draw is controlled on a shift basis and reviewed weekly it gives a practical workable situation. In that the correct emphasis is put on draw points that need to be pushed and holding back on those draw points that are tending to over draw. The actual redistribution and calculation of actual tonnages from draw columns being done on a monthly basis. In a month it is estimated that the rock will have moved some 5 metres in the draw column. (See Section 27 Potential Draw Rates).

For strip draw patterns it is pointed out that hang-ups will take place and are semi predictable. It has been found in practice, that you don’t actually have to close the extraction drifts for drilling and charging. This can take place in the draw point without affecting the production going on behind it, in the collection drift, particularly if a compact rig such as the King rig is used. Once it is in place and operating, even if the driller is operating it remotely from the opposite drawpoint the production LHD can get past. It might be different if a large jumbo type rig is being used to do the drilling. Blasting can be sequenced into the production cycle/change overs.

Drawpoint repairs are however a different matter. These should be on going and continuous as a form of planned maintenance. Drift that is to be closed on the following day should be inspected and the work required for maintenance agreed and planned for the next day. Early maintenance tends to be more rapid and cost effective than delayed major repairs.

DESIGN TOPIC

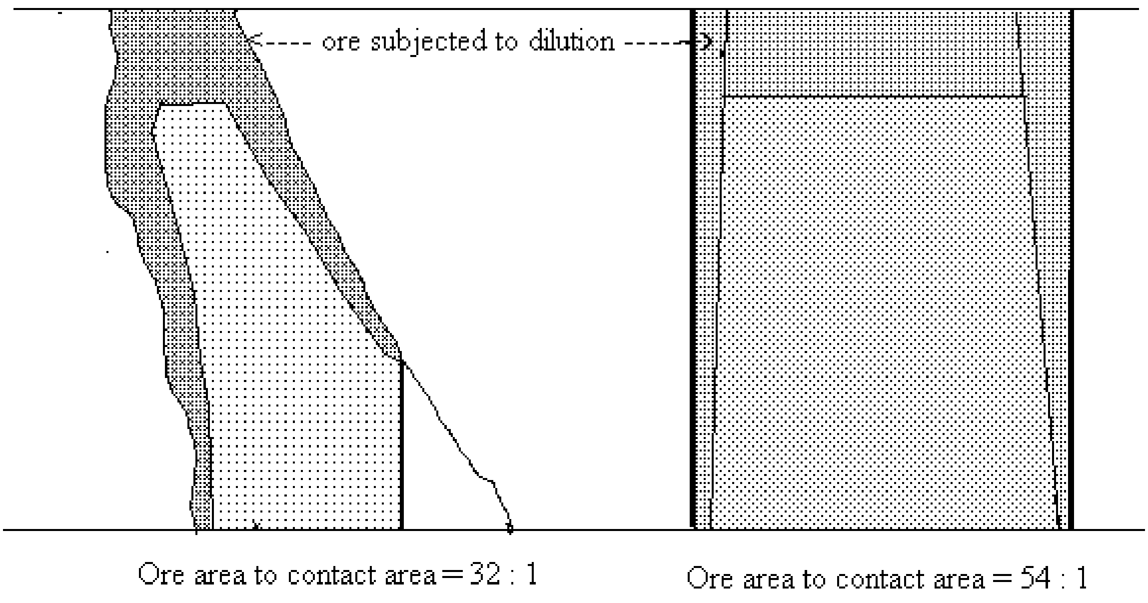
Dilution

GENERAL

Dilution is an integral part of cave mining operations and the object is to keep the dilution down, however, there are situations where the unpay zone is extensive and of slightly lower value than the ore cut-off grade. In these cases it might pay to draw a high dilution - increased mixing - so as to draw a large tonnage and to recover a larger mineral tonnage than is available from the ore reserve.

ORE VOLUME TO ORE/WASTE INTERFACE AREA

The higher the ratio of ore volume to the surface area of the ore/waste interface the lower the overall percentage dilution.



In the above example, the diagram on the left shows a large percentage of the ore is liable to dilution while there is a significant difference between the ratios of the two diagrams. As the orebody become wider the side dilution becomes insignificant.

ATTITUDE OF ORE/WASTE INTERFACE

As can be seen from the previous diagram, dilution will be severe with the dipping irregular outline compared to the simple geometry of the vertical orebody on the right.

FRAGMENTATION RANGE OF ORE AND UNPAY

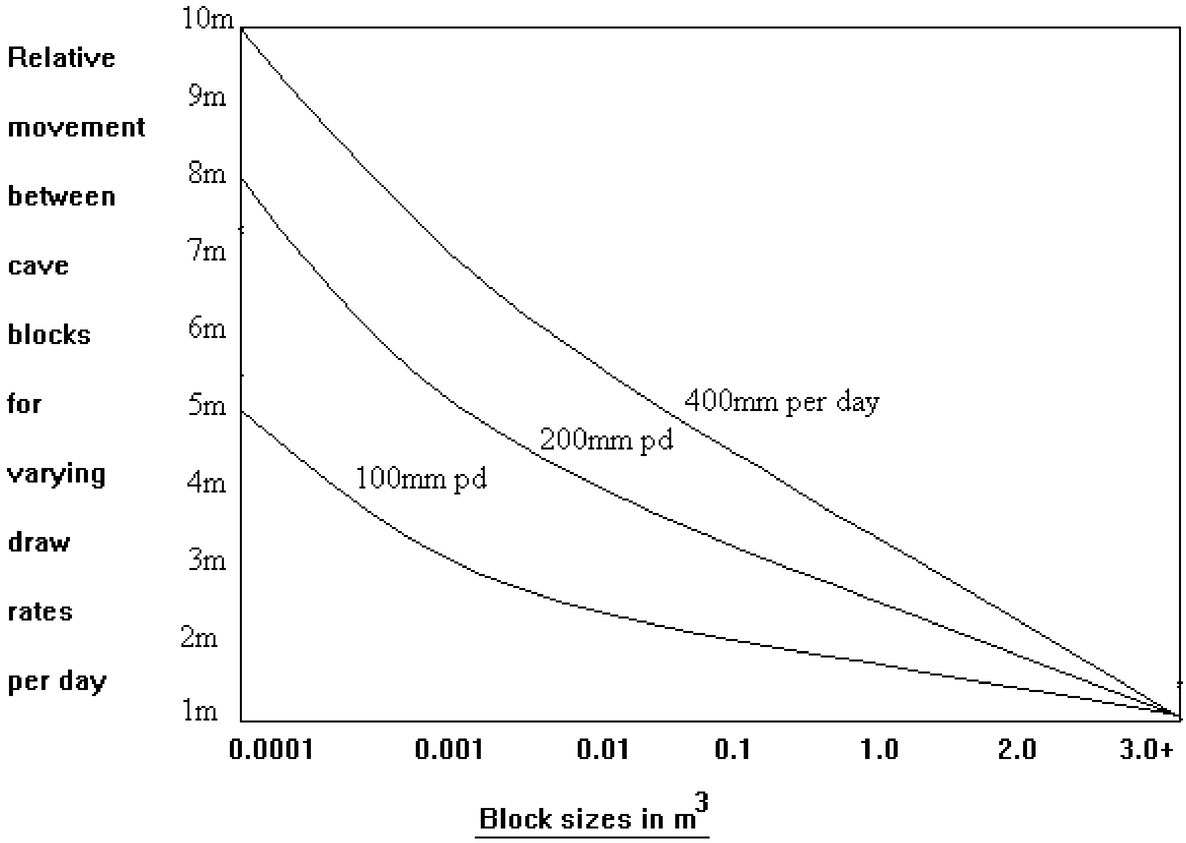
Finely fragmented waste / unpay means early and extensive dilution whilst coarse unpay and fine ore means low dilution.

GRADE DISTRIBUTION IN UNPAY

High grade patches in the unpay zone can lead to appreciable overdraw as this leads to erratic sample values and visual assessment. In some asbestos deposits blobs of enrichment in the hangingwall led to excessively high overdraws of up to 300% as a result of the visual impact of the fibre in the drawpoint.

MINERAL DISTRIBUTION - DISSEMINATED / FINES IN UNPAY ZONE

If the mineral in the dilution zones occurs as fines, there could be an enrichment to the ore as the fines move more rapidly than the coarse.



DRAWZONE INTERACTION AND DIRECTION OF FLOW

A good drawpoint interaction and parallel flow will represent the optimum conditions. Poor drawpoint interaction and drawzones angled according to local variations will lead to high dilution. The flow of material is discussed in the section on drawpoint spacing. Wide drawpoint spacing requires that the ore does not remain behind as columns between drawpoints.

DIFFERENCES IN DENSITY

High density ore and low density waste leads to low dilution and vice versa.

BLOCK / PANEL / STRIP MINING

These draw strategies have to be reviewed in terms of dilution percentage

Block Mining - The orebody is divided into a series of blocks and the mining follows a sequence of extraction of the blocks. If the blocks were 100m x 100m and the production requirements meant that two blocks had to be in full production at any one time, then for the bulk of the time three blocks would be operational. In this way one block would be going out, one in full production and a new block coming into production. The problem with mining by blocks is the large area of boundary with a previously mined block requiring boundary broken ore pillars to reduce the waste inflow.

SOURCE OF DILUTION

The source of the dilution has to be defined in terms of fragmentation, mineral distribution and grade. If the mineral in the dilution is readily released then the value assigned to the 'dilution' in the drawpoint can be higher than the in situ grade. The converse applies if the mineral is disseminated in the competent blocks and fine unpay dilution forms.

HEIGHT OF INTERACTION ZONE - DEGREE OF MIXING

The height of the interaction zone was established from 3-D sand model tests where it was shown that for close spacing of drawpoints, there was a uniform drawdown above a certain height. As the drawpoint spacing was increased, the boundary between active movement and the upper zone became less distinct, with irregularities decreasing upwards. At a wide spacing there was no interaction. In practice, with a large range in fragmentation and wide drawpoint spacing ,the top of the interaction zone is not a narrow zone, but, a broad zone with troughs and peaks. The number assigned to the height of interaction zone is not a precise figure, but, refers more to the relative degree of mixing.

DILUTION ENTRY

The dilution entry percentage is the percentage of the ore column that has been drawn when the first dilution appears in the drawpoint and is a function of the amount of mixing that occurs in the draw column. The mixing is a function of:-

- Draw column height
- Range in fragmentation
- Drawzone spacing
- Range in tonnage's drawn from working drawpoints
- The range in tonnages and the maximum drawzone spacing will give the height of the interaction zone

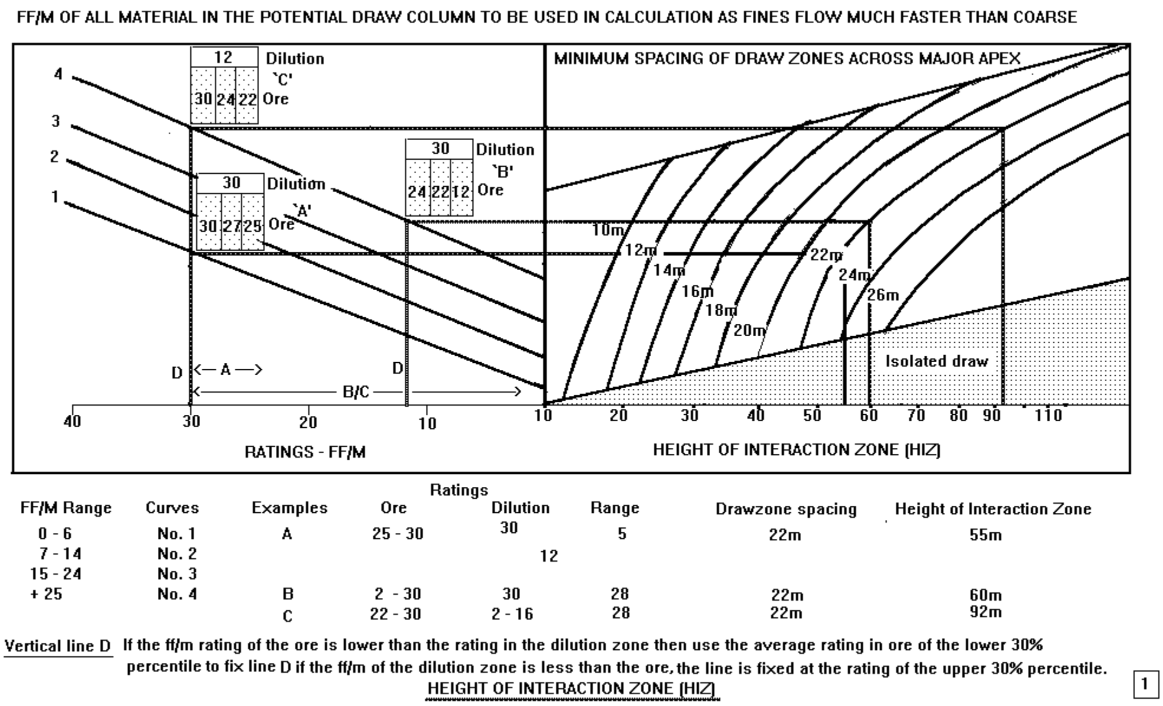


Figure 1

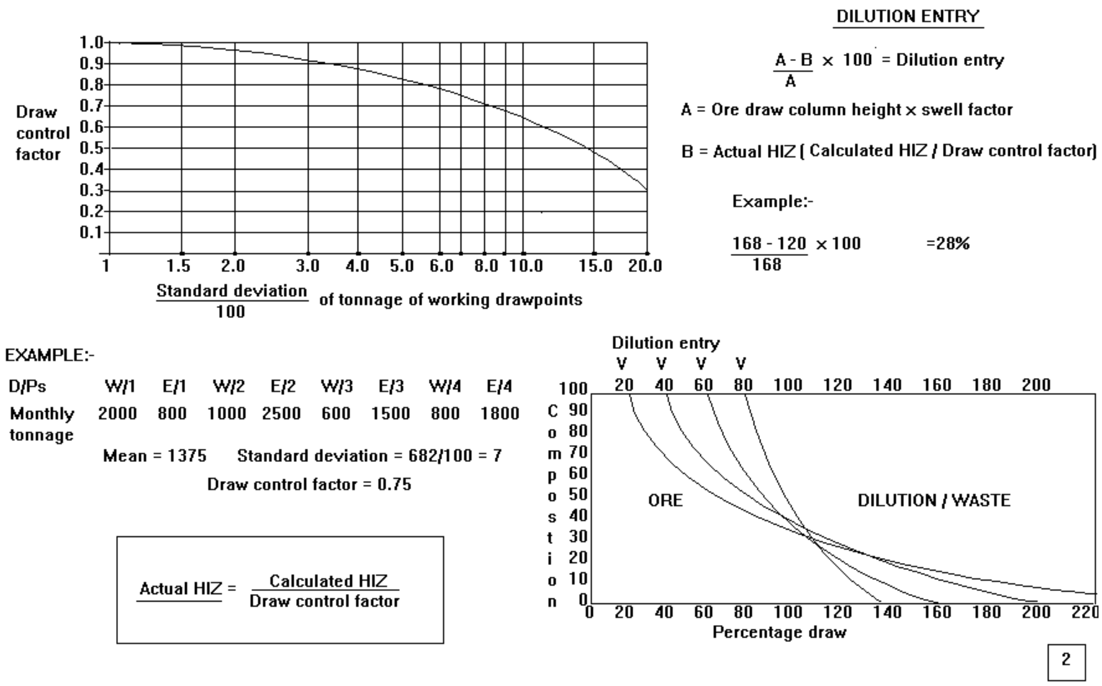


Figure 2

The range in fragmentation size distribution and the minimum drawzone spacing across the major apex will give the height of the interaction zone (HIZ). This is illustrated in Figure 1. There is a volume increase as the cave propagates, so that a certain amount of material is drawn before the cave reaches the dilution zone. The volume increase or swell factors are based on the fragmentation and applied to column height. Typical swell factors are: fine fragmentation 1,16, medium 1,12, coarse 1,08. A draw control factor is based on the variation in tonnages from working drawpoints. This is illustrated in Figure 2. If production data is not available the draw control engineer must predict a likely draw pattern. A formula based on the above factors has been developed to determine the dilution entry percentage:-

(A - B)/A x C x 100 = Dilution entry

A = Draw column height
B = Height of interaction
C = Draw control factor

The dilution entry curve was originally drawn as a straight line, but underground observations show that where there is early dilution, the rate of influx follows a curved line with a long ore 'tail' as shown in Figure 2. Dilution entry will also be affected by the attitude of the drawzone, which can angle towards higher overburden loads. Dilution entry is a function of the height of the interaction zone (based on the range in fragmentation and drawzone spacing), the volume increase during the propagation of the cave and the draw control practice.

Dilution will not report in a drawpoint until the ore tonnage for the swell relief or volume increase of the propagating cave, and the tonnage to create channelways has been drawn. These tonnages are a function of the range in fragmentation during the propagation of the cave and the initial draw rates - whether uniform or not. The primary fragmentation, expressed as a percentage oversize (+ $2m^3$) is used to determine the swell relief percentage:-

Primary fragmentation; % + $2m^3$ oversize	**Percentage volume increase**
0	18%
10	15%
20	12%
30	10%
40	8%
50	6%

It is assumed that the draw will be uniform and that large rocks will be blasted as soon as they report in the drawpoints. Failure to do this will lead to the rapid formation of channelways, particularly in caved zones with a large range in fragmentation. The range in fracture frequency (ff/m) **ratings** will indicate the uniformity of the material and whether channelways will form. The following table is used to derive the percentage to be added to the volume increase percentage to obtain the minimum percentage draw before dilution enters the drawpoint:-

ff/m rating range	**% drawn to create migration paths**
6	12%
14	10%

24	8%
+ 25	6%

The minimum dilution entry figure is based on the volume increase and the amount of material required to create channelways. Thus, in uniformly fine material with a uniform draw down, the earliest dilution can report in the drawpoint, regardless of the height of the draw column, is after 30% of the column has been drawn.

The formula to calculate dilution entry is:-

Column height = A
((Volume increase % + ff/m %) x A) = Draw height of undiluted ore = B
Draw control factor = C
Height of interaction zone = Calculated HIZ ÷ Draw control factor (C) = D

B ÷ A x 100 + [(A - B - D) ÷ A x 100] = Dilution entry %

NOTE; if C > (A-B) then B ÷ A = Dilution entry

Recommended dilution entries for different draw heights are:-

Ore column height	Draw rate	HIZ	Dilution entry %
40m - 60m	100 mm	50m	20%
61m - 80m	100 mm	50m	29%
81m - 100m	100 mm	50m	45%
101m - 120m	200 mm	55m	50%
121m - 150m	200 mm	60m	56%
151m - 200m	200 mm	65m	63%
201m - 250m	300 mm	70m	69%
251m - 300m	300 mm	70m	74%

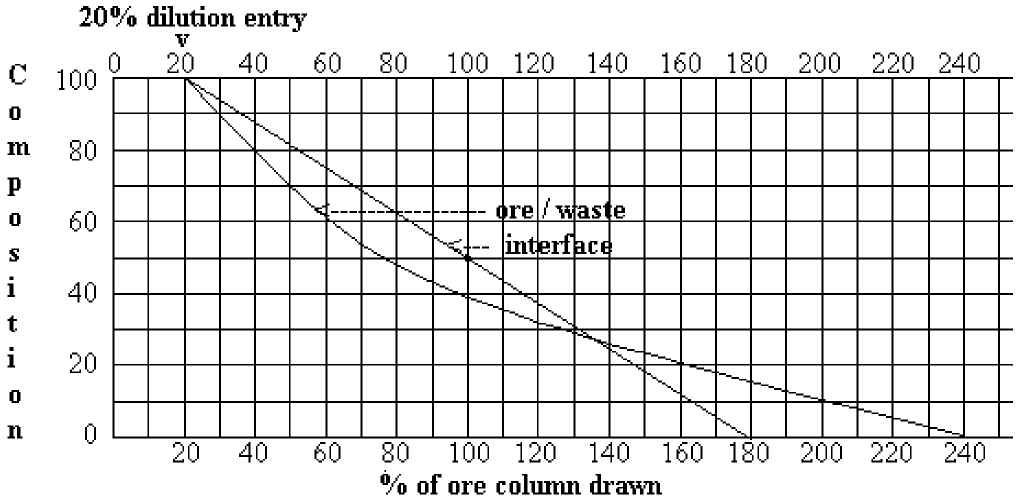
A = Solid ore height
B = Swelled ore height
C = HIZ from curves
D = Draw control factor
S = Swell factor

Dilution entry = 100 x ((S -1) + (A -C) ÷ B) x D or with swell factor as a percent

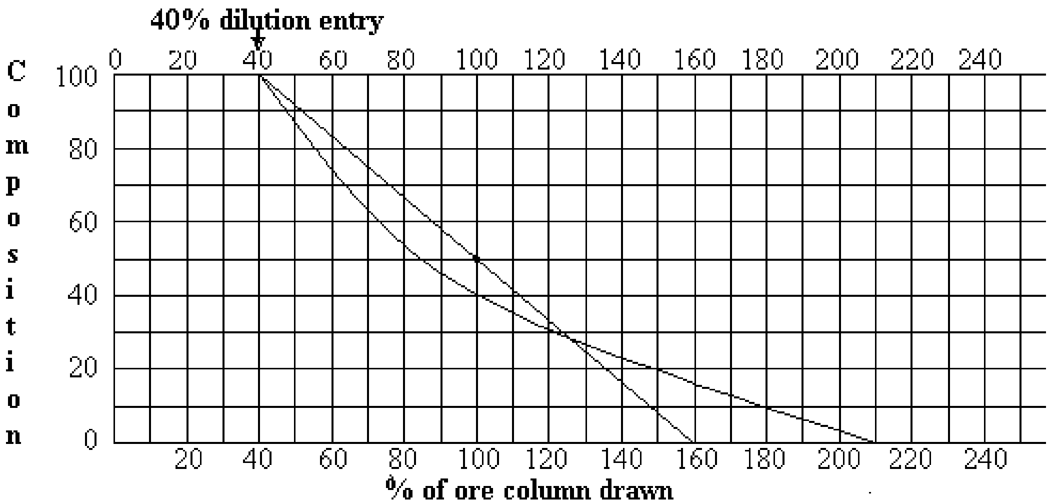
$$(S + (\frac{A - C}{B} \times 100)) \times D = \text{Dilution entry}$$

PERCENTAGE DILUTION

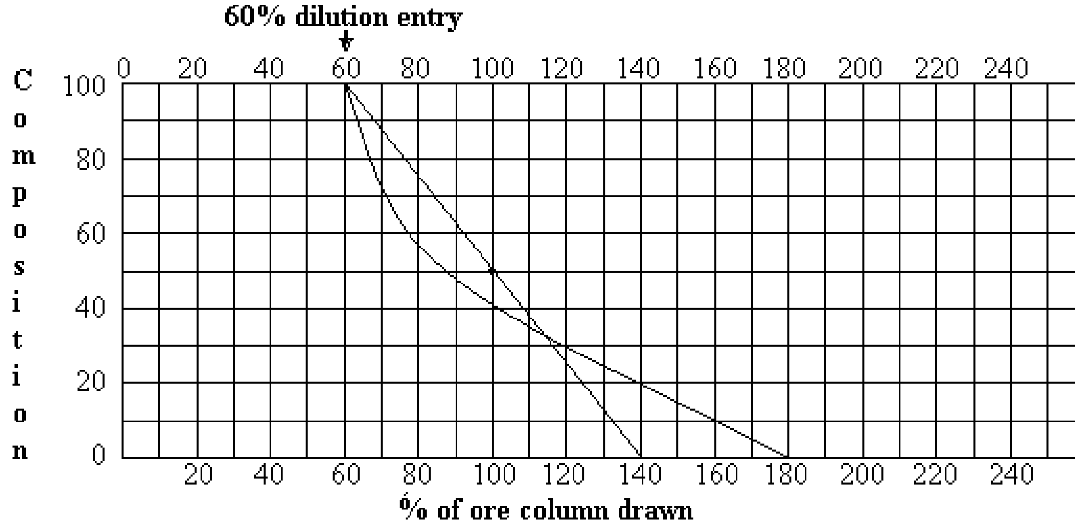
Various dilution entries are shown in the following diagrams which have two ore/waste interface curves, for simplicity the straight line is used. The percentage dilution for each situation is shown. It is assumed that the ore has a value of 2.0% and the dilution zone a value of 0.5%. The shut -off value is 1.0%



At 125% draw the shut-off value = 1.0%, percent dilution = 30%, ore recovery = 90%, the average grade = 1.6%



At 120% draw the shut-off value = 1.0%, percent dilution = 23%, ore recovery = 93%, the average grade = 1.7%



At 115% draw the shut-off value = 1.0%, percent dilution = 17%, ore recovery = 94%, the average grade = 1.8%

It can be seen that if the curve is used the influence on grade is greater at low dilution entries. Also, if the grade of the dilution is fairly high, then large tonnages can be drawn to the shut-off grade.

DESIGN TOPIC

Draw Control

(With Major Contribution from T.G.Heslop)

GENERAL DESCRIPTION

Draw control is the practice of controlling the tonnages drawn from individual drawpoints with the object of:

- Minimising overall dilution and maintaining the planned ore grade sent to the plant.
- Ensuring maximum ore recovery.
- Avoiding damaging load concentrations on the extraction horizon.
- Avoiding the creation of conditions that could lead to air blasts or mud-rushes etc.

The draw strategy will change during the life of the block; the largest change will come when caving is complete and the block can go into production. In the undercutting and caving phase, we need to control the draw for several reasons such as influencing the shape of the cave back, keeping the gap between cave back and caved material small enough to prevent unintended rilling and air blasts should the back collapses. This is discussed in Section 3 below

By the time it has fully caved we need to have investigated:

- the local factors that will influence the caving and drawdown processes,
- calculated the potential tonnages and grades that will be available from each drawpoint;
- implemented a draw control system - a set of methods and procedures to:-
- devise a draw strategy,
- record and analyse the tonnages drawn,
- manage the draw following the adopted draw strategy,
- estimate of the remaining tonnages and grade for future production scheduling and planning.

This is discussed in Section 4 below

A prerequisite for designing and operating a draw control program is an understanding of the mechanisms of draw down that occur in the cave block. These are outlined in the next section.

DRAW MECHANISMS

Whilst no one can directly observe what goes on in a block cave under draw, the principles of draw have been established though years practical experience. This has been reinforced by marker experiments in block cave and sublevel caving operations, sand and gravel scale model experiments and more recently, in numerical modelling.

From all this work, we know that there are three basic mechanisms of draw in the cave material.

- In the upper portion of the cave, the subsidence is orderly and uniform, generally not influenced by the rate of draw from individual drawpoints below, but may reflect steps defined by zones of drawpoints worked at higher rates. The mass flow is underlain by a zone of interaction and intermixing in which two modes of draw may occur.

- The first of these is the classical granular flow (Kvapil's "gravity flow") in granular materials in which particles flow under the pressure from overlying and lateral materials and, to a much lesser extent, their own weight towards lower pressure zones.

- The second mode of flow occurs in coarser grained materials where transient voids or hangups form. This allows finer material to rill in from above or the sides before the arch forming the void collapses. It is replaced higher up by one or more smaller voids and the process repeats itself.

MASS FLOW

In the mass flow in the upper portion of the cave, subsidence is orderly and uniform. Particle trajectories are nearly vertical, converging or diverging according to the shape of the cave, responding to side pressures and deflecting towards lower density areas above zones of drawpoints being drawn at higher rates. The particles are not influenced by the rate of draw from individual drawpoints below, but may reflect steps defined by zones of drawpoints worked at higher rates. However, the uniform draw-down may be punctured by relatively small "ratholes" or vortex-like subsidence zones where drawpoints have been worked in isolation. Other than these small zones, there is no horizontal or vertical no mixing in the mass flow zone. The rates of flow of fine and coarse materials are the same as the larger voids into which the fines can flow have dissipated into numerous smaller cavities, generally short lived as they consolidate, collapse and allow the whole zone to subside as in a mass flow zone.

GRANULAR FLOW MECHANISMS

The first of these flow mechanisms was extensively investigated in the early sand model experiments, usually small glass-fronted sandboxes. Janelid and Kvapil(1965) interpreted the principles of "gravity flow" through a single drawpoint from two dimensional model experiments. They have identified the characteristic ellipsoid of draw, limit ellipsoid and particle flow trajectories.

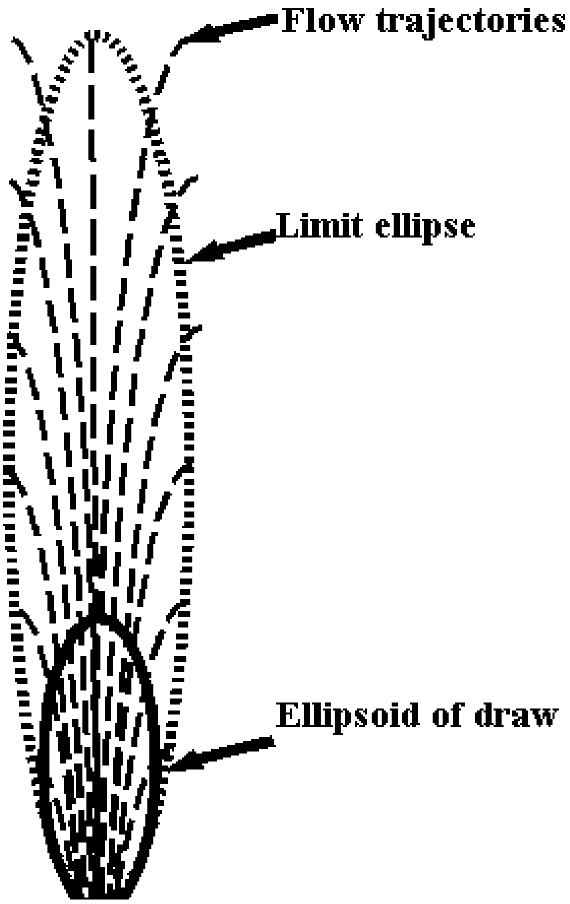


Figure 1 **Kvapil's "Gravity Flow" ellipsoid of draw (the amount of material drawn out), the limit ellipsoid, i.e. the limit of disturbance when the ellipsoid of draw is withdrawn, and the particle flow trajectories.**

This is the model on which many sublevel caving mines were designed. In sublevel caving mines, the ring spacings, sublevel and drift spacings were chosen to match the ellipsoid of draw dimensions. In many early caving mines, drawpoints were laid out so that the limit ellipsoids overlapped and no ore was lost between the ellipsoids. The experiments in a large three-dimensional sand model (discussed in Chapters 12 and 40) have confirmed their conclusions for a single drawpoint worked in isolation. However, when many drawpoints are worked at the same time, the characteristics of the draw change dramatically. There is a zone of mass flow and uniform subsidence in the upper draw column. In the lower draw column, there is a zone of stress interaction that induces dynamic lateral migration of material from slow drawpoints and inter-draw column areas into the live draw columns. Material also moves from the slower working draw columns to the faster and this evens out the rate of subsidence of the mass-flow zone.

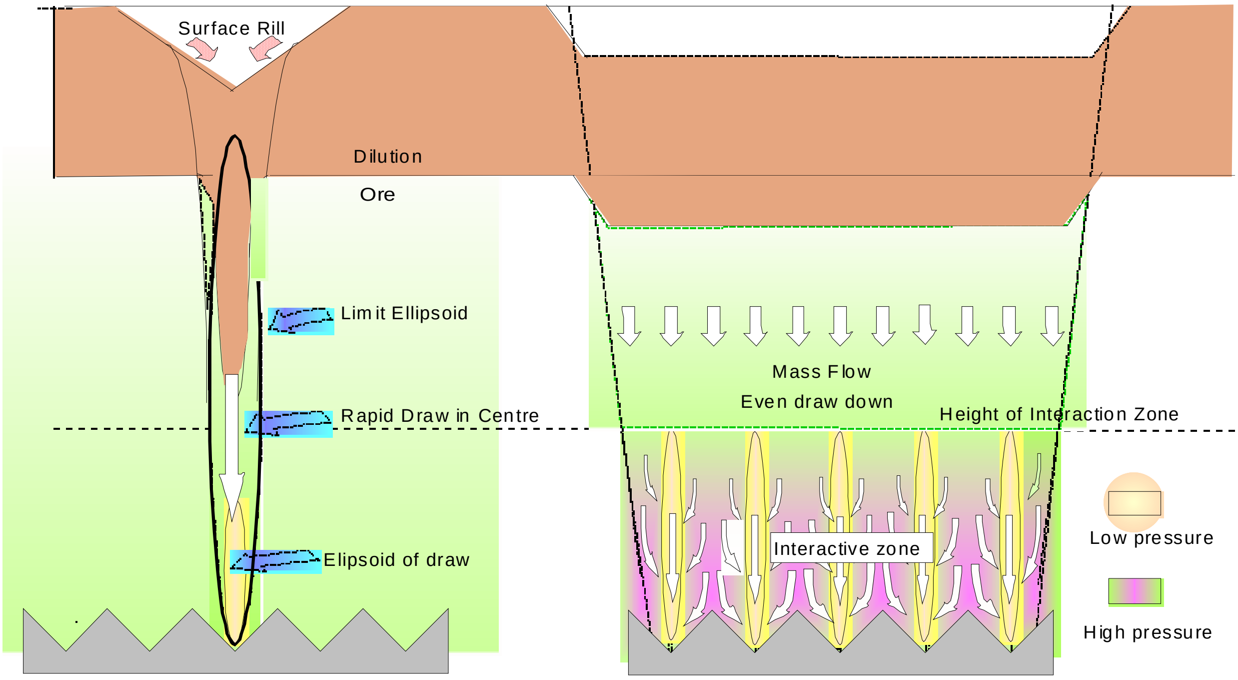


Figure 2 **Granular flow mechanisms – Material flow paths for drawpoints worked in isolation (left) and worked concurrently (right) where higher pressures in inter-drawpoint areas forces material into the low- pressure zones above working drawpoints.**

VOID DIFFUSION

The second of these flow mechanisms is known as stochastic flow, probabilistic flow and void diffusion. The term "void diffusion" (VD) is preferred as it conveys an easily imagined dynamic picture of voids diffusing through the cave material by filling, collapsing and reforming at successively higher elevations. The mechanism has been investigated and extensively computer modeled as a stochastic process by a number of researchers. Attempts have been made in glass-fronted boxes to investigate the effects of larger angular particles on the draw behaviour, but the characteristics of the draw have been more difficult to observe and interpret, and certainly more difficult to illustrate in diagrams. Confirmation of the mechanism has come from two extensive marker experiments in sublevel caving operations in Sweden.

In the experiments, Gustafsson(1998) found that the recovery of markers did not conform to the accepted ellipsoid of draw. They indicated an irregular "drawbody" shaped like a hand with fingers extending upwards, some into the waste left by the previous ring. When the ring was fired and drawn, alternating runs of ore and waste indicated a changing pattern of draw, with some markers from higher up in the ring being recovered before lower markers. Gustafsson believes that these indicate transient voids that were filled by material from above or one or both sides, and reform a little higher up above, to one side or another of the filled void. He modeled the draw using a probabilistic function that determined whether the void was filled from above or to one side or another. By applying a modifying flow factor to the finer and more mobile waste, he reproduced the runs of waste and ore observed in the experiments.

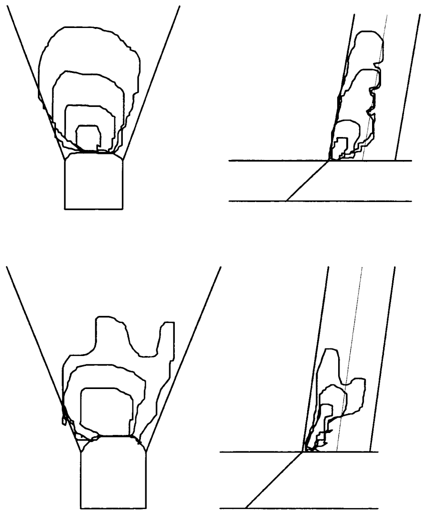


Figure 3 **Gustafsson's "palm and finger" drawbody shapes from full scale marker experiments at Kiruna Mine**

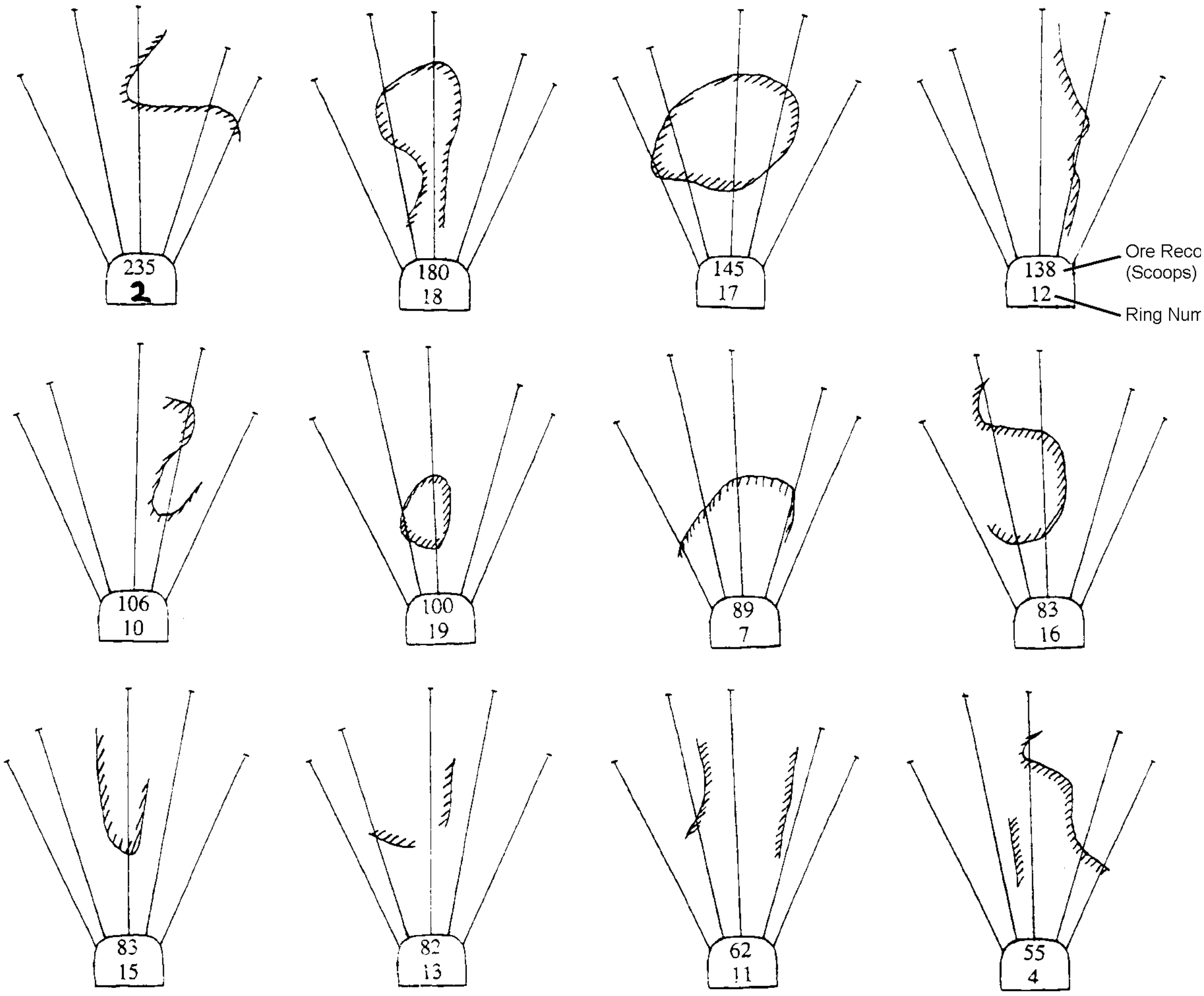


Figure 4 **Some drawbodies from Gustafsson's marker experiments at Kiruna.**

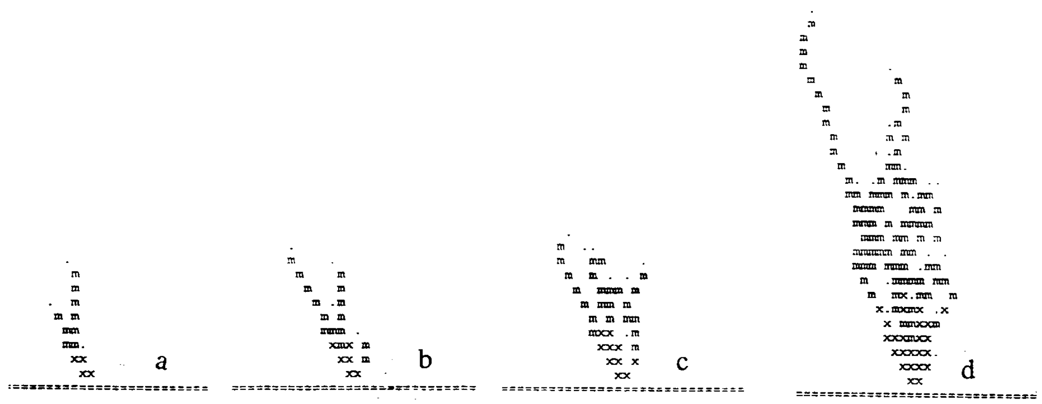


Figure 5 **Drawbody shapes from computer modelling. a: 6 blocks removed, b: 10 blocks removed, c: 16 blocks removed, d: 33 blocks removed.**

In block caving, the hypothesis is that transient voids form above the drawpoint, collapse, and reform as in the sublevel caving experiments. The voids are partially filled with material from the sides or they collapse and are filled from above, reforming a little higher up, and the process repeats itself. Fine non-cohesive or loose material may fill a cavity even as it develops. While some voids may coalescence, it is expected that generally the voids would split and become smaller and probably more stable. At some elevation in the draw column, the effect of numerous voids from many drawpoints is to de-stabilise adjacent voids thus favouring vertical flow and limiting the faster flow of fine material. This effectively transforms the draw mechanism into mass flow.

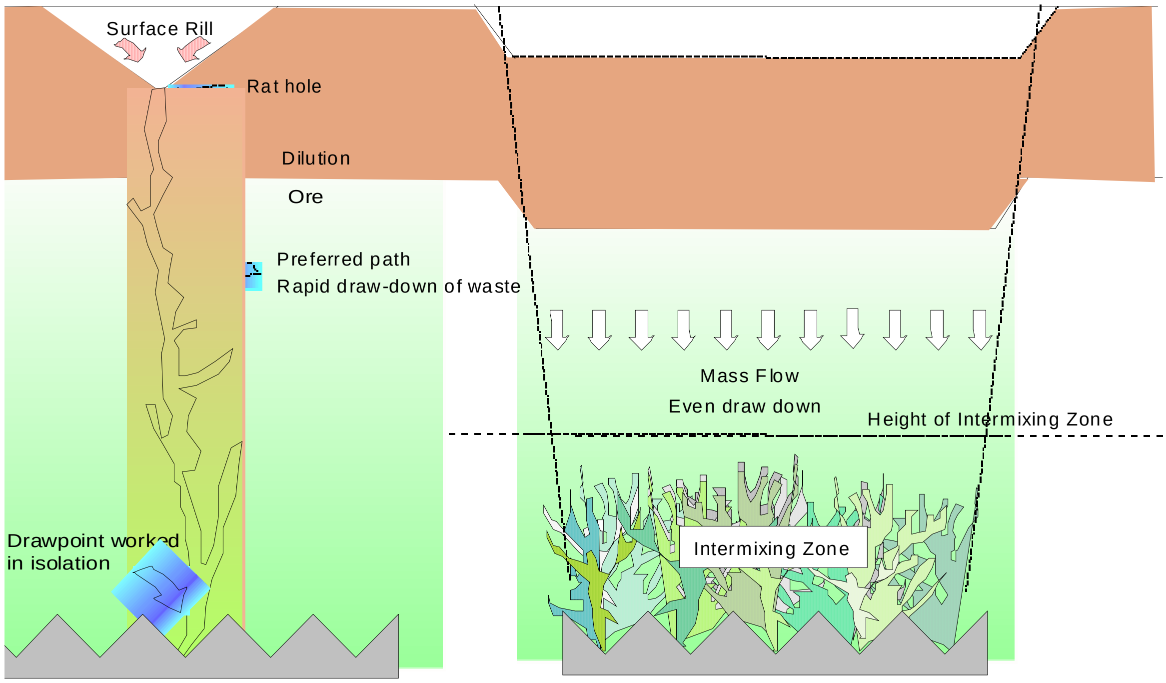


Figure 6 **Void Diffusion (VD) mechanisms – The formation of preferred channel leading to rat-holing for a drawpoint worked in isolation (left) and overlapping of multiple "palm and finger" drawbodies when drawpoints worked concurrently (right).**

However, if a drawpoint is worked in isolation, the passage of successive voids from the same drawpoint will form a zone of less dense, more mobile material above the drawpoint while compacting the surrounding material. "Piping" occurs when a succession of cavities works its way through the caved material to surface. Once the path is established and filled with mobile fine material, it becomes a preferred draw-down channel that can pull dilution deep into the ore.

The following photograph of a hungup drawpoint shows primary rock which came from 0 -100m, secondary rock from 100m - 300m and talus from +300m. The percentage drawn at that stage was 280%.

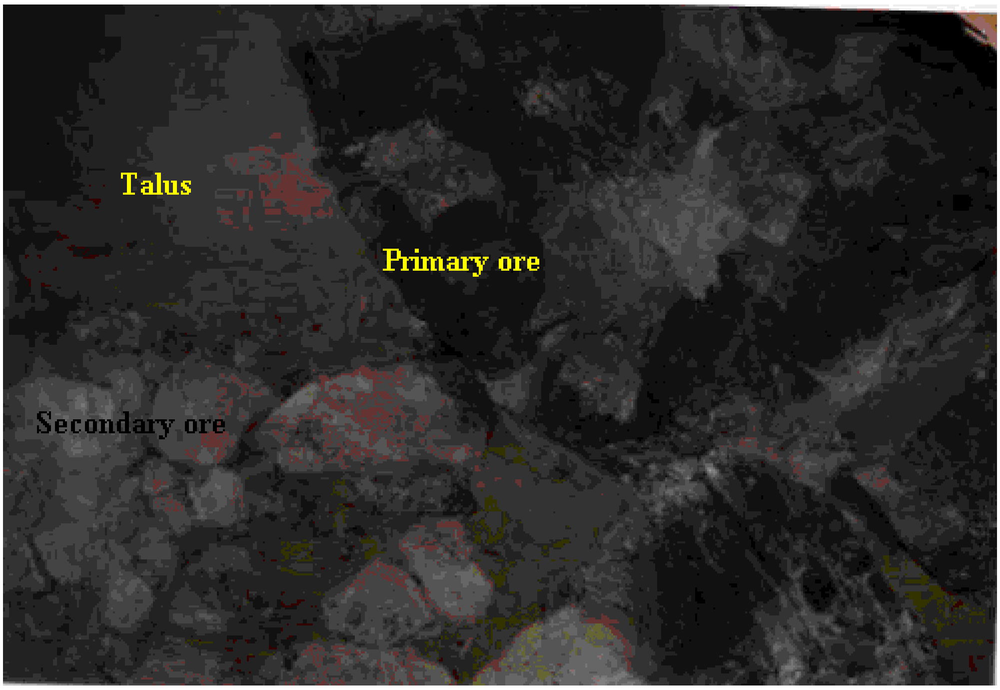
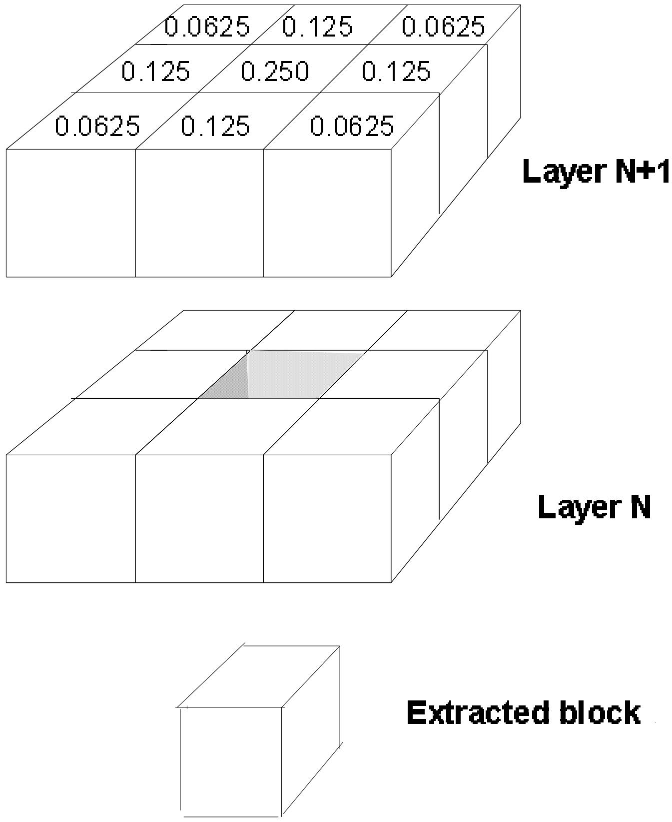


Figure 7 **Photograph illustrating the ingress of fine waste (talus) into coarse ore. The primary ore came from 0 -100m, the secondary ore from 100m - 300m and the talus from +300m.**

In the numerical modelling of this process, the void is modeled as a vacant cell that is filled with material from one of nine cells in the next layer of cells above, chosen on a probabilistic basis. In Jolley's model (Figure 8) the probability of a block in layer N+1 filling a void in layer N is based on its position relative to the empty block below. In the Swedish sublevel caving modelling, the probability factor is modified to suit the type of material occupying the cell. Thus, the more mobile finely fragmented material would have a higher probability of moving into a vacant cell than coarse, angular, freshly blasted ore. The factors can be further modified to allow density differences, as the more loose material towards the centre of a draw column can be expected to be more mobile than compacted denser material, in the peripheries of the draw column.



Jolleyb.emf

Figure 8 Block flow probability after Jolley (1968)

THE INFLUENCE OF MATERIAL PROPERTIES ON DRAW MECHANISM.

The interactive and VD flow models represent two ends of a spectrum of block caving material behaviour. Clearly, if the caved material is made up of blocks that are dominantly equidimensional and rounded with a well graded range of material sizes, the behaviour in a block cave will be similar to that observed in sand models. On the other hand, if the material is gap-graded or has a wide range of material sizes and high proportion of larger angular or slabby blocks, the cave draw action will follow the VD model. Intermediate materials will behave somewhere between these two extremes. The proportions of interactive and VD can be assessed by a simple scoring system as this:

Score 20 % for each affirmative answer

	Interactive draw	Score	VD draw	Score
Particle size distribution	Well graded		Gap or poorly graded	
Intact block strength	Weak, veined or foliated		High strength, defect free	
Rock Block Shapes	Rounded or equidimensional		Slabby, orthogonal angular	
Joint condition	Soft or low-friction infills		Clean or hard infill	
Fines fraction	Sandy or platy fines		Sticky Fines	
	Total Scores			

The following material properties could have a bearing on draw mechanisms through the propensity to from arches, the stability of the arches or flow characteristics:

Primary and secondary fragmentation

The rock mass classification can be used to outline domains or zones with different primary fragmentation in the cave. The primary and secondary fragmentation analyses (discussed in Chapters 10 and 11) can provide estimates of the particle size distributions of freshly cave material, and the material size distribution after it has been drawn down to the drawpoint.

A narrow range of smaller rock sizes or a well-graded mix of small, intermediate and large sizes favours interactive draw. A “boney” or gap graded mix favours arching and accelerated fines migration. Figure 9 shows typical primary or secondary fragmentation particle size distribution (grading) curves, and the probable proportions of VD draw that will occur. Thus, it is possible that a higher proportion of VD draw could occur early in the life of a block and decrease as draw progresses due to secondary fragmentation and rounding of blocks. The proportion of VD draw should be assessed for all stages of draw.

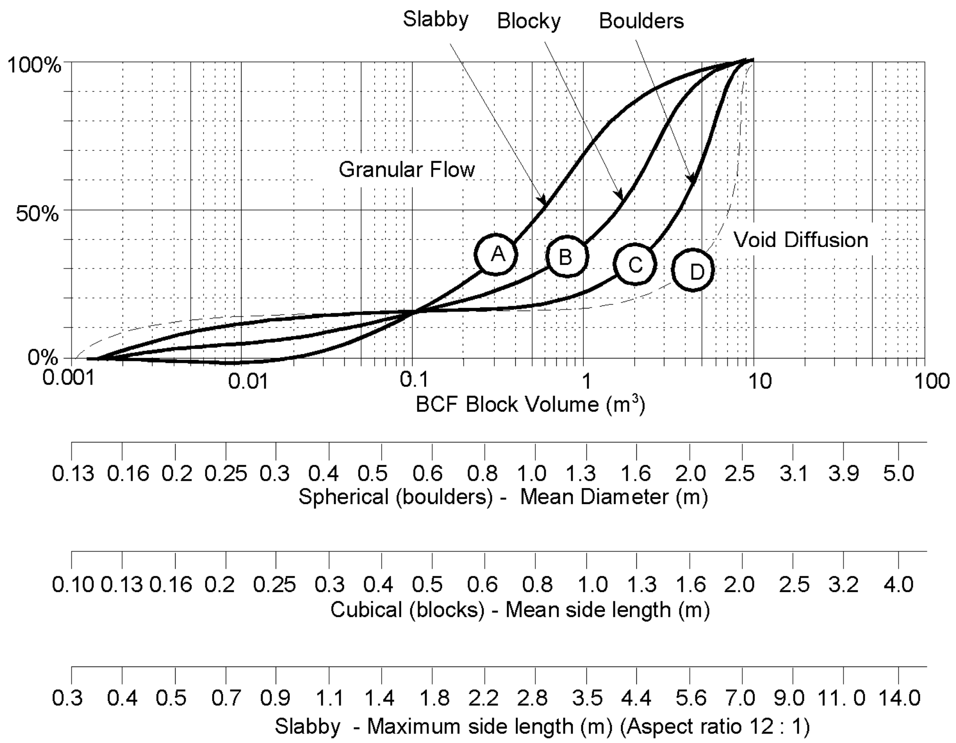


Figure 9 **Conceptual particle size distribution - draw mechanism relationship.**
Curves A to D are increasingly gap-graded, favouring VD Draw. The thresholds for VD draw decrease with angularity and aspect ratio. A blocky material with a particle size distribution curve shaped between B and D , will have an increasing proportion of VD draw, and with an A-shaped curve will behave as a granular material in interactive draw.
Curves require experimental confirmation.

Distribution Curve Shape	Slabby	Blocky	Rounded-Spherical
A	Granular	Granular	Granular
B	Some VD draw	Granular	Granular
C	Half VD Draw	Some VD draw	Granular
D	All VD Draw	Half VD Draw	Some VD draw

Intact block strength

Weaker materials, or materials with a high proportion of weak veinlets, will tend to reduce the stability of arches and hang-ups and thus reduce the proportion of VD draw. The difference between the primary and secondary grading curves is an indication of the probability of blocks splitting or breaking up in arches and so destroying the arch.

Rock Block Shapes

Slabby materials, or materials with a high proportion of rectangular (brick) shaped blocks would form more durable arches and hang-ups and thus favour VD draw. These would normally be formed by three orthogonal joint sets, or, two joint sets plus bedding or foliation. Additional joint sets that will produce pyramid, tetrahedral or more rounded blocks will tend to reduce the tendency to hang up.

Joint condition

Rock surfaces coated with lower friction materials or soft vein filling materials will tend to reduce arch stability. It is suggested that the percentage of VD draw from the primary fragmentation be reduced by 5% for soft vein filling materials (gypsum, propylite) and by 10% for sericite and or chlorite, and 15% for graphite, talc, clay or serpentine minerals. As the joint condition rating in the rock mass classification system is a combination of joint planarity and roughness, joint filling materials and water, it can not be used for this purpose. In caved ore, the planarity is meaningless given the scale of the rock blocks; the interlocking of rough joints is greatly reduced as the surfaces no longer match, asperities are abraded and pits filled with dust. The water conditions will have changed.

Fines fraction

The presence and nature of fines can also influence the stability of arches. Fines may either bind blocks or reduce inter-block friction. Wet clays may lubricate inter-block contacts, while stiff moist clays, silty or clayey materials may act as a binder (as in "sticky" ore) and stabilize arches. Sandy materials on contacts will tend to reduce friction by a rolling action. Platy minerals such as micas, antigorite, and clays also have a friction-reducing effect.

COMPARISON OF GRANULAR FLOW AND VOID DIFFUSION DRAW CHARACTERISTICS

The following comparison of the draw characteristics of the interactive and VD mechanisms may help in appreciating draw characteristics

Table 1 - Comparison of Draw Characteristics.

Condition	Narrow range of material sizes, multifaceted blocks. Granular and Interactive Flow	Wide range of material sizes, large interlocking blocks Void Diffusion
Isolated drawpoint	Regular shaped ellipsoid of draw; all material is drawn sequentially, with the highest rates of subsidence in the core of the draw column decreasing outwards to the limit ellipsoid.	Voids diffuse upwards and outwards according to a probabilistic function that favours the less dense areas and fine materials. Eventually "ratholes" form as preferred channels for drawing in fine mobile waste, into the ore
Simultaneous draw from adjacent drawpoints	Draw produces low-pressure zones above a drawpoint and high pressures in the surrounding ground. When many drawpoints are worked at the same time, the stresses above each working drawpoint interact; increasing the pressures in inter-drawpoint material are greatly increased. The pressure differences produce lateral migration in the form of plastic deformation from high-pressure zones to the low-pressure live draw columns, effectively widening the draw from each drawpoint.	Voids formed by working of drawpoints or blasting of hang-ups diffuse upwards and outwards, into the inter-drawpoint material. Working adjacent drawpoints at the same time will increase the number of voids in the inter-drawpoint ground. These de-stabilise each other and this effectively widens the draw from each drawpoint. As the voids diffuse upwards they tend to split, become smaller and more numerous and are more likely to disturb the stability of other voids. This favours vertical flow and limits opportunities for fine material to enter cavities. At a particular elevation in the draw column the material above subsides as in mass flow.
Uneven draw and slow or drawpoints stopped for repair	Interaction between stress fields produces higher pressures in the inter- drawpoint material and in the adjacent slower or temporarily stopped drawpoints; this promotes lateral migration into live draw columns, evening out the draw. This action occurs below the HIZ. No "catch up" tonnage should be drawn from the slow drawpoint.	Faster drawn drawpoints produce more voids that diffuse upwards and outwards into adjacent drawpoints. This outward diffusion is not compensated for by a similar number of voids diffusing from slower neighbours. This tends to even out the draw rate as stress interaction does in interactive draw., but it favours lateral migration of the fine fraction. "Catch up" tonnage should only be drawn if the ore has a high proportion of large material sizes.
External influences (peripheral structures and lateral pressures) – inclined draw	Inclined draw due to sliding on peripheral structures or lateral pressures that can widen the draw column.	VD is probabilistically controlled and is not influenced by lateral pressures, but may be constrained by peripheral structures. In the mass flow it would be influenced by lateral pressures as in interactive draw

IMPLICATIONS OF DRAW MECHANISMS

The two mechanisms, granular flow and VD draw have the following factors in common:

- Both have a zone at the base of the cave in which material moves laterally from slowly drawn areas towards the faster-drawn drawpoints.
- In both mechanisms, there is an upper zone in which mass flow is the main mechanism. Material trajectories are influenced by lateral confining structures, pressure differentials due to regional differences in draw rates, height of caved material, etc.
- And in both mechanisms, a drawpoint worked in isolation will produce a narrow zone of rapid material draw-down above the drawpoint. In interactive draw this is vertical and in VD draw it

could be inclined and it could tap into a pre-existing preferred channel of draw formed by an adjacent drawpoint.

The fundamental differences between the two are:

- In interactive draw the caved mass deforms under pressure differences, coarse and fine materials move together towards low pressure zones i.e. active drawpoints..
- In VD draw, the fines are more mobile than coarse material and the probability of them rilling into a void, especially from the side, is greater. Preferred channels filled with materials that are more loose and thus more mobile, could form, especially when a drawpoint is worked in isolation. If one of the neighbouring drawpoints is then worked in isolation, it may tap into this channel. If the dilution is more mobile than the ore, dilution will be drawn well into the ore.

On a practical level, hungup drawpoints are more probable with VD draw and greater secondary blasting will be required. This means that with VD draw, a greater number of drawpoints will be required to produce the same tonnage than with interactive draw. In VD draw situations, there will also be the temptation by LHD operators to work those drawpoints running fine materials when others are rough or hung-up.

The mechanisms have implications for how we manage the draw and how we calculate the tonnages available for draw, record the tonnages and analyse them. We have a number of established terms for interactive draw that have equivalents in VD draw. These terms could be used for both mechanisms such as:

Isolated drawpoint diameter

In granular flow, this is the diameter of the ellipsoid of draw above a drawpoint. It is related to the size of the large rocks in the draw column and the propensity to form hang-ups. It is recognised in VD draw that there is no ellipsoid boundary, that “fingers” of material drawn will extend into the material beyond the ellipsoid, but the ellipsoid is an area of more frequent voids, lower densities and therefore more mobile material.

In traditional block cave design, the maximum spacing of the drawpoints is set at the diameter of the ellipsoid of draw. In designing block caves for interactive draw it is accepted that the drawpoints could be spaced at 1.5 times the isolated drawpoint diameter. Interaction of the stress fields would effectively widen the draw from each drawpoint when they are worked at the same time. In VD draw, the penetration of the inter-drawpoint areas by voids produced by simultaneous working of adjacent drawpoints, produces the same effect.

Height of the Interaction Zone

This is the vertical dimension of the zone at the base of a block cave where the stress above active drawpoints interact and produce lateral migration of material. In VD draw, it is postulated that in diffusing upwards, the voids become smaller and more frequent and transform into mass flow at some elevation above the drawpoints. In both mechanisms it is recognised that the height is dependent on the range of material sizes, and the maximum drawpoint spacings (i.e. across the major apex). If the block is worked such that the active drawpoints are more widely spaced than the layout spacing, the height of the interaction zone increases. In interactive draw and VD draw, both vertical mixing and lateral migration or horizontal mixing occurs within this zone. In both mechanisms, the higher the interaction zone, the earlier dilution will appear and the more dilution will eventually be drawn.

Dilution entry point

In both interactive draw and VD draw, the dilution entry point is dependent on the height of the interaction zone, and not the height of the ore. It defines the slope of the slice lines on the grade analysis graphs. Where drawpoints are worked in isolation, it is dependent on the total height of the ore.

Isolated drawpoints

For both types of draw it is necessary to have a criterion for distinguishing between those drawpoints that are being worked in isolation and those that are worked simultaneously.

In interactive draw, drawpoints can be considered to be worked in isolation if they are worked after the majority of the adjacent drawpoints have not been worked and the low stress zones above them have dissipated. In VD draw , drawpoints can be considered to be worked in isolation if they are worked after the halo of voids around the majority of the adjacent drawpoints have collapsed. When draw ceases in a block cave it is known that settling noises can still be heard for several hours. It is commonly assumed that the stresses or voids will only last for about 24 h.

Tonnage redistribution

Due to the cross-drawpoint migration of material, when drawpoints are worked simultaneously, some of the tonnage recovered from those drawpoints that are worked faster than their neighbours will have come from the slower neighbours. In the tonnage accounting system, the tonnage drawn from each *drawpoint* is recorded and redistributed to *draw columns* from where the ore came. This can be done by a simple moving average of the eight neighbouring drawpoints and the centre drawpoint being considered. This should be done on daily or shift tonnages and any isolated drawpoint credited with the full tonnage drawn. In interactive draw, no “catch up” tonnage should be drawn. from slower drawpoints, although in VD draw, only the finer fraction may flow into adjacent active draw columns. Redistributed tonnage should be limited to this, and the lag in draw down should be made up by faster draw

Redistribution of excess tonnage drawn should be confined to the interactive zone i.e. below the HIZ.

DRAW CONTROL IN THE UNDERCUTTING AND CAVING PHASE

In small to medium sized orebodies, it may be possible to undercut the whole orebody, allow it to cave and then draw it in a controlled manner. However, in larger ore bodies, this is normally not practical, and the ore has to be developed and undercut in stages. Planning the mining of larger orebodies requires careful consideration of:

- the area to be undercut to initiate caving and how this can be extended if required,
- how many drawpoints are needed to meet the production requirements,
- where undercutting should start and the initial undercutting direction. How it should be extended into the remainder of the orebody, the options being panel or block caving,
- the undercutting direction(s), face shapes and potential abutment pressures and damage, and
- the initial and subsequent caving mechanisms as they can affect fragmentation.

These are discussed in greater detail below.

INITIAL UNDERCUT SIZE AND DIRECTION OF UNDERCUTTING

Careful consideration should be given to the direction of undercutting with respect to ground competency, the presence and orientation of major structures and their effect on abutment loads. Preference should then be given to the undercutting direction that will produce better fragmentation. Better fragmentation is more conducive to interactive draw and this gives better ore recoveries and lower dilution.

The minimum undercut size required for caving can be estimated from the MRMR with the aid of Laubscher's Stability graph. (i.e. the in-situ RMR after appropriate adjustments for the shape of the undercut, the possible clamping effect of stresses, orientation of joints etc.). Where it is possible, provision should be made to increase the minimum span dimension of the undercut should the caving not start when fully undercut, or, stalls after a small amount of caving.

It should be remembered that as the cave back progresses upward, the stresses in the back change. In a high lateral stress environment, the stresses in the cave back change from tensile in the initial flat undercut back, to compressive, particularly if the back is allowed to form an arch. If a flat back can be maintained the change to compressive stresses is delayed. Further, the cave back may pass into more competent ground conditions. With either of these, the caving action may slow or stop. If it passes into poorer ground conditions the caving could accelerate. See Chapter 6 for more details.

CONTROLLING THE RATES OF DRAW DURING CAVING

It is sometimes possible to influence the rate of caving and thus the shape of the initial cave back, by controlling the amount drawn in different areas of the undercut. As discussed above, the shape of the back influences the stresses in the back and that affects the primary fragmentation. Thus it is important to manage the size and location of the gap(s) between the back and the caved material by controlling the quantities drawn through the drawpoints below the gap. Small to moderate gaps are admissible where caving is to be encouraged. Large gaps / voids are an air blast hazard and should be avoided. Inclined gaps may allow waste to rill in to the ore. Where there is a weak zone that may "chimney cave", great care is required to keep the gap as small as possible in the vicinity of the weak zone. If it is allowed to run, it may bring dilution into the heart of the orebody; and, it may leave overhangs. See Chapter 6 for more details.

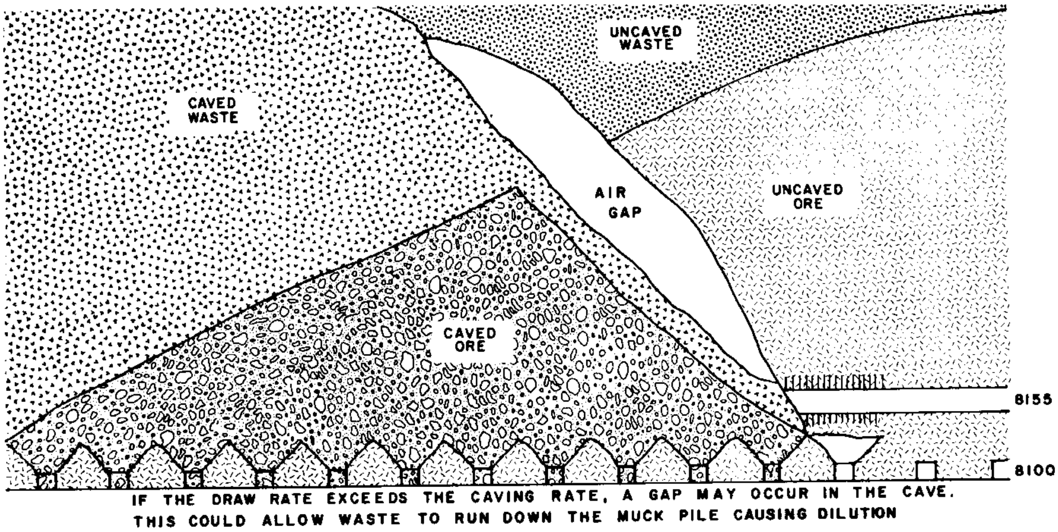


Figure 10 **Unintended rilling of waste into the ore can occur when the rates of draw in new drawpoints exceed the rate of caving and an air gap forms.**

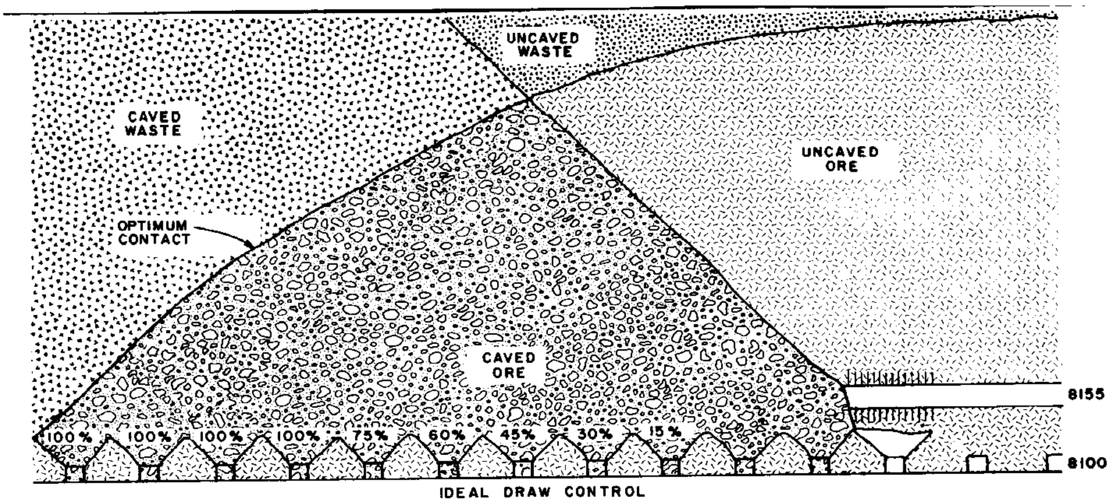
SECOND AND SUBSEQUENT BLOCKS - UNDERCUTTING DIRECTION AND CAVING MECHANISM

Once the initial block has caved, the stresses in the backs of adjacent blocks will alter, increasing in one direction and reducing in the other, and the caving mechanism may change. This reduction in lateral stress could allow simple differential subsidence to occur on any sub-vertical joint or fault structures. This has been called "subsidence caving" and it is characterised by the orderly subsidence of the large sub-vertical columnar blocks with very poor fragmentation. Where a choice of undercutting direction exists, this situation should be avoided, as it can cause point loading on major apexes, detrimentally affecting production efficiency and allow ingearly dilution entry.

PANEL OR BLOCK CAVING.

In many of the early caving mines in very large orebodies, the ore body was divided into a series of blocks (commonly 50 m x 50 to 80 m). These were often defined by boundary weakening drifts and / or slots. For continuity, several blocks could be in production at the same time. Commonly alternate blocks were mined as primary stopes followed by the pillar blocks later when the waste in the worked-out primary stopes had had time to consolidate. On some mines the blocks were worked in a chequer-board pattern. The claimed advantage for block caving (as opposed to mass or panel caving) was reduced pressure on the extraction development and therefore reduced maintenance costs, minimum dilution and a number of operating advantages. It is clear that the low dilution could only be achieved where the dilution in depleted blocks would consolidate and remain on place when the adjacent block is mined. Some mines left ore pillars between cave blocks to avoid side dilution.

In the 1960s Climax Mine (Colorado) introduced the panel caving system. Once the initial caving block had been established, extensions to the undercut and cave could continue in one, two or three directions simultaneously. Draw was controlled so that there was a 10% differential in the quantity of ore drawn between each line of drawpoints. This was to allow for the initial volume increase when the ore caved and then to lower the ore/waste continuously from the newest to the oldest drawpoints. This system has also been adopted at Henderson (with a 15% differential- see Figure 11 below) and several other mines such as El Teniente.



100, 100, 100, 100, 75, 60, 45, 30, 15, 5, 2 = % drawn of tonnage available to the draw column on a daily basis.

Figure 11 Draw control with panel caving at Henderson Mine

DRAW CONTROL IN THE PRODUCTION PHASE

In the production stage, the prime objectives of controlling the draw are normally to maximise the discounted cash flow (or ore recovery), while minimising dilution and avoiding stress damage to the drawpoint level.

The starting point is the calculation or estimation of the initial drawpoint tonnages and grades. This is normally based on the geological block model, which has to be transformed to match the drawpoint layout and the potential directions of draw, which may not be vertical. This requires careful consideration of several factors that could influence the direction of draw, and the potential for intermixing of ore and waste. These are not necessarily independent.

Then we need to devise a suitable strategy for drawing the ore. There may be several options that will need to be evaluated to determine the best strategy. The common strategies include working all drawpoints at the same rate, working the drawpoints at rates proportional to the height of draw for each drawpoint, and working the highest grade drawpoints faster.

Finally we need systems for controlling the draw, recording and redistributing the tonnages drawn, and analysing the state of draw for the whole block.

CALCULATING THE INITIAL DRAWPOINT TONNAGES AND GRADES

It is assumed that a block model with the mineral grades of the deposit is available. This block model should extend well beyond the limits of the orebody, to surface or at least 50% higher than the ore, and laterally to the limit of the mineralised halo. This block model will need to be transformed to a draw control model that takes into consideration the drawpoint layout and angles of draw.

The steps in calculating the tonnages available are:

1. Assess the angles of draw.
2. Assess the influence of ore and dilution characteristics on draw.
3. Assess the practical constraints – layout features.
4. Assess the draw mechanisms, height of the interactive zone.
5. Transform the geological block model to match the drawpoint layout and expected directions of draw.
6. Calculate the tonnages and grades available for draw from each drawpoint.

Step 6 can be done manually on groups of drawpoints or by computer using commercially available software. There are limitations in the latter approach (See Section 7 below)

Factors influencing inclination of draw

Draw is not necessarily vertical, it can be influenced by external structures that constrain the subsidence zone, such as; external pressures, material properties, caving mechanism, methods and rates of draw. These are discussed below:

Limits of subsidence
Natural features that can define the limits of the cave include faults, shear zones, and dominant joint sets, or combinations of joint sets that form a step path failure surface. Examples include the 60° dipping footwall shear zone at Havelock Mine (see Chapter 26), and the Pinto Fault at Miami Mine (Fletcher 1959). The Pinto fault was a major gouge-filled fault that dipped at 45 to 50° towards the ore. Markers placed in development above the fault were recovered in the next stope down-dip and had moved down at angles of 50° and 54°. The fault in this area dipped at 50°

It is has also been noted that the plan dimensions of the cave zone affect the subsidence angles, as there is a lateral buttressing or horizontal arching effect that resists slippage of the walls. This arching is reduced when the size of the caved area is increased, and is accompanied by a flattening of the draw angles. There are numerous examples of draw down occurring on steeper dipping structures dipping towards the ore. Overhanging sections, where structures dip steeply away from the ore are also common.

Pressure
The relative heights of materials within the subsidence zone have a strong influence on the angles of draw. The most dramatic example of this is the case of King Mine in Zimbabwe, where the hill slope to

the north of the orebody has caused a 69° inclination of the draw towards the hill. As mining has progressed, the surface elevation of the caved ground immediately above the drawpoints remained almost unchanged while the higher hill slope subsided. The inclined draw was confirmed by the recovery of markers (See Figure 12).

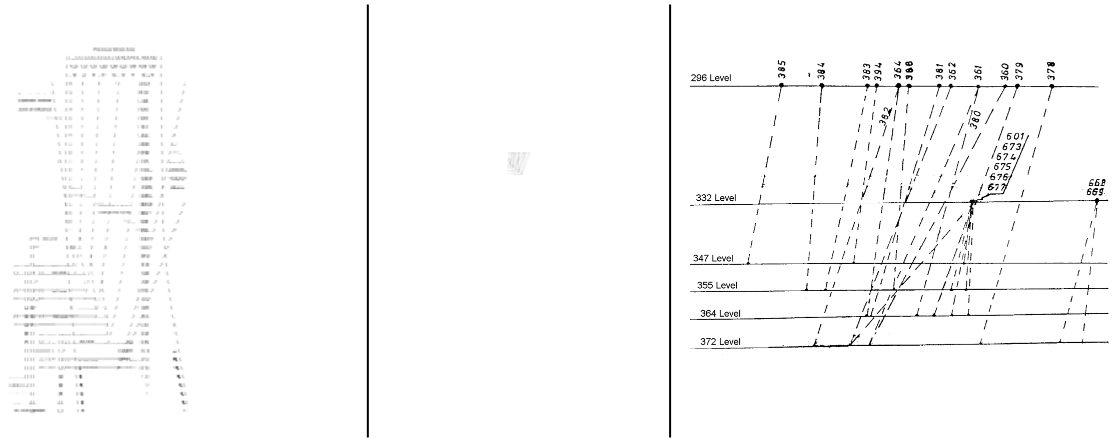


Figure 12 **Marker draw trajectories at King Mine, Zimbabwe, Main/4 Block (left) and NNE/4 Block (right) Refs Bell (1992) and Mujuru (1995)**

Draw Rates

The King Mine experience can be extrapolated to the higher areas in the cave profile over slower working drawpoints surrounding zones of drawpoints with higher draw rates. This interpretation is borne out in the sand modelling tests and in some of the full scale marker experiments at El Teniente (Alvial 1992). When the elevation difference reaches a critical point, the higher pressures in the peripheries will incline the draw towards the higher peripheries. However, this is only part of the story, the central faster drawn zone would also have a lower density and lower pressures, and it would exert a lower restraining lateral stress on the peripheries. The effect increases with the size of the high draw rate zone as the influence of horizontal arching is reduced.

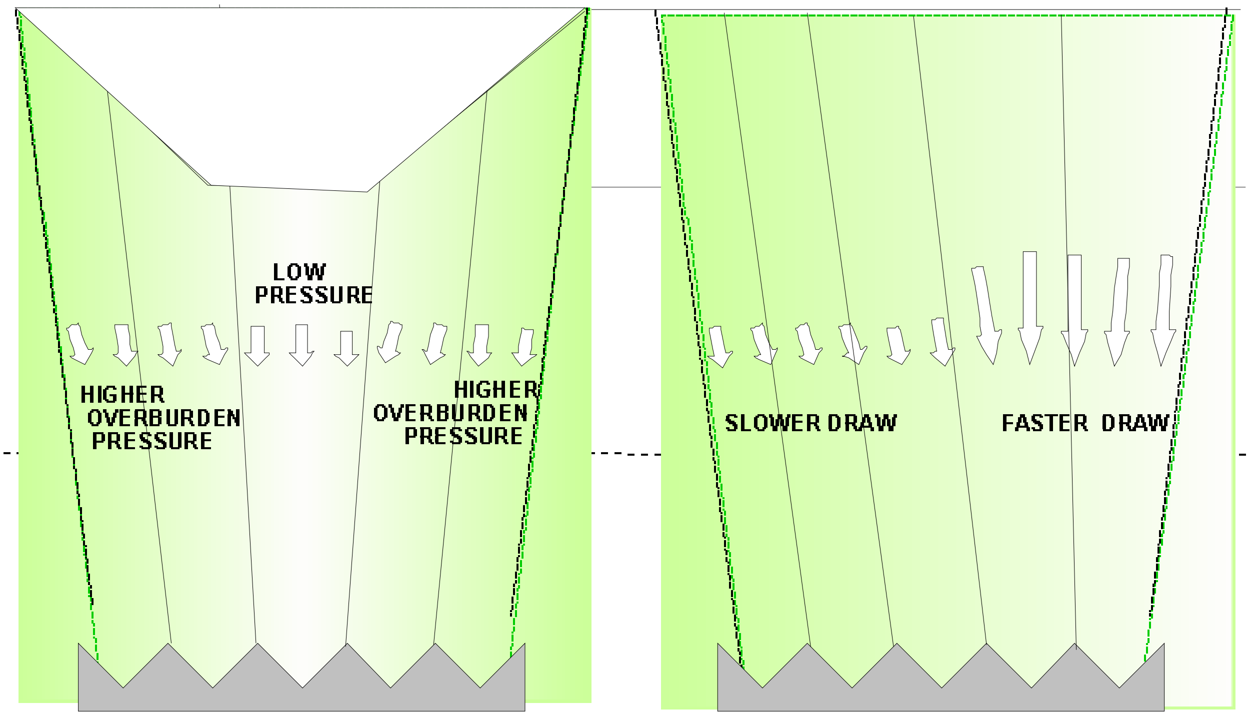


Figure 13 **Diagrams illustrating the inclination of draw columns towards higher pressure zones and areas of slower draw.**

The influences of drawpoint layout, drawbell design and methods of working.

Drawpoint layout

In most block caving operations the drawpoint layout is a compromise between a uniform spacing of drawpoints, for good draw and greater spacings across the major apex than between them. Increasing the spacings across the major apex provides more space for LHDs and improves the strength and stability of the major apex and the extraction development in it. The effect of this is to create bands of relatively closely spaced drawpoints that could have a strong interaction if worked together. These bands have been called “drawzones”. if the drawpoints are worked together, they effectively form a continuous trough a few metres above the drawpoints elevation.

The height of the interaction zone, and therefore the dilution entry point, is dependent on the distances separating the draw zones.

Methods of working and Draw Mechanism.

The interactive and the VD flow models represent two ends of a spectrum of block caving material behaviour. Section 2.4 outlines the effect of primary and secondary fragmentation, intact block strength, rock block shapes, joint fill materials and the nature of the fines fraction on the draw mechanism. Where the dominant draw mechanism is VD, and the dilution is finer and more mobile than the ore, the preference should be for a draw system for uniform simultaneous draw. Where this cannot be achieved, the estimation of tonnage and grade becomes very difficult and unreliable.

Calculating the Tonnage for draw and estimation of dilution

The calculation of the tonnages available for draw can be calculated manually aided by a spreadsheet, or by using the commercially available software discussed in the Section 7 below.

Manual Methods

The steps involved are:

1. Decide on the potential direction of draw and define the draw columns
2. Transform the block model to suit the layout and draw inclinations.
3. Divide the block up into like areas of inclination of draw, and grade distributions
4. Select an appropriate grade graph analysis graph type from figures 13 to 18 and calculate the average grade for increasing stages of draw until the grade drops below the grade cut-off value. The remaining draw column above this point should be tested to determine whether this is a local drop in grade and /or whether there is more economic ore above this point.

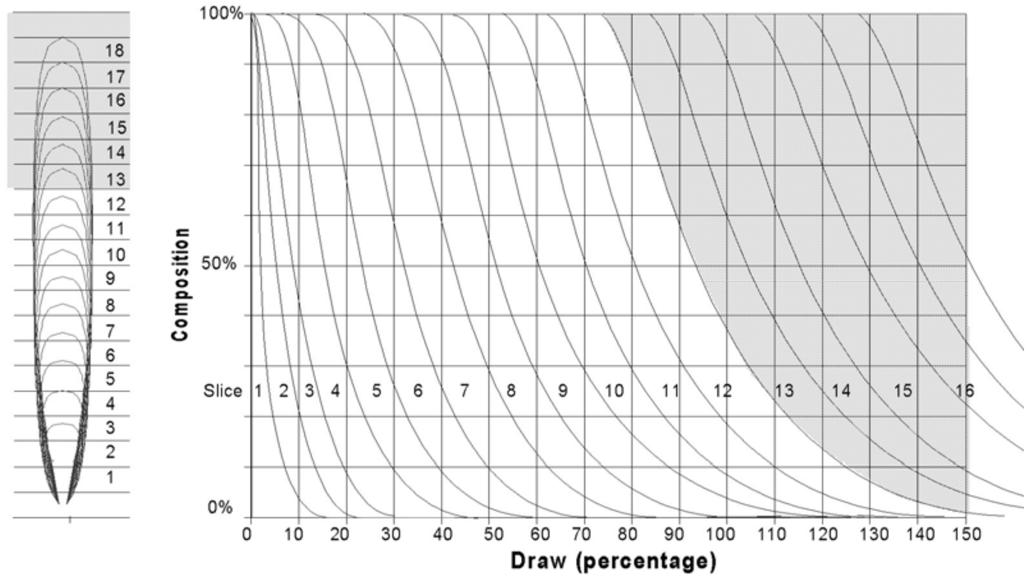


Figure 14 Grade analysis graph for an isolated drawpoint in a granular material.

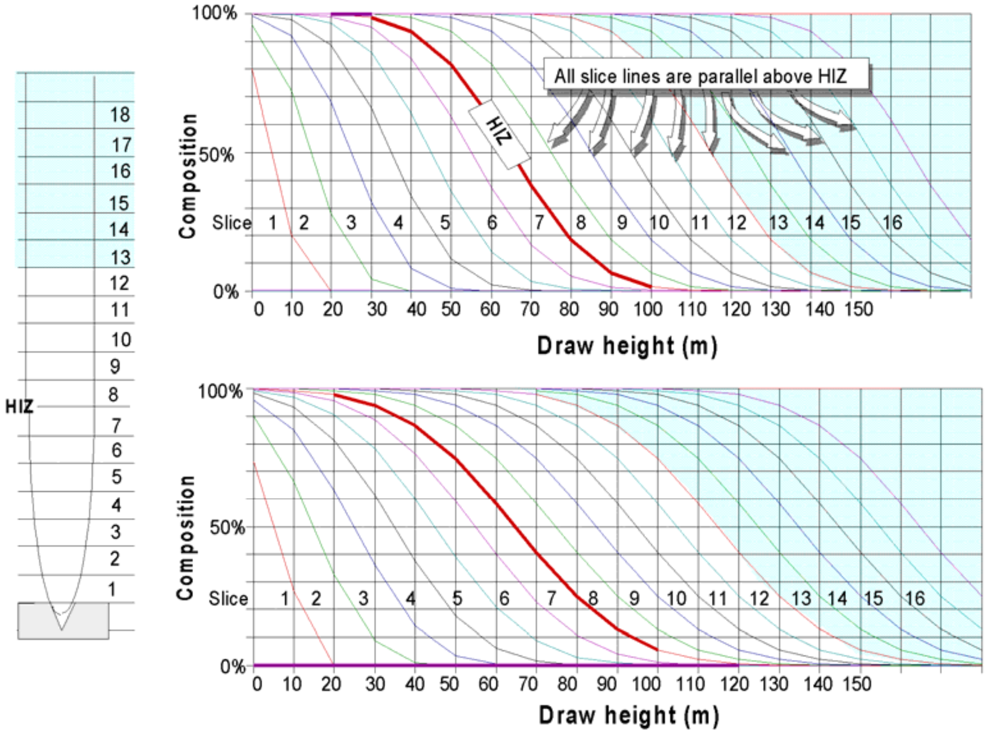


Figure 15 **Grade analysis graph for an isolated drawpoint in a VD draw. Top graph: Theoretical proportions Bottom graph: Simplified graph for manual grade calculations.**

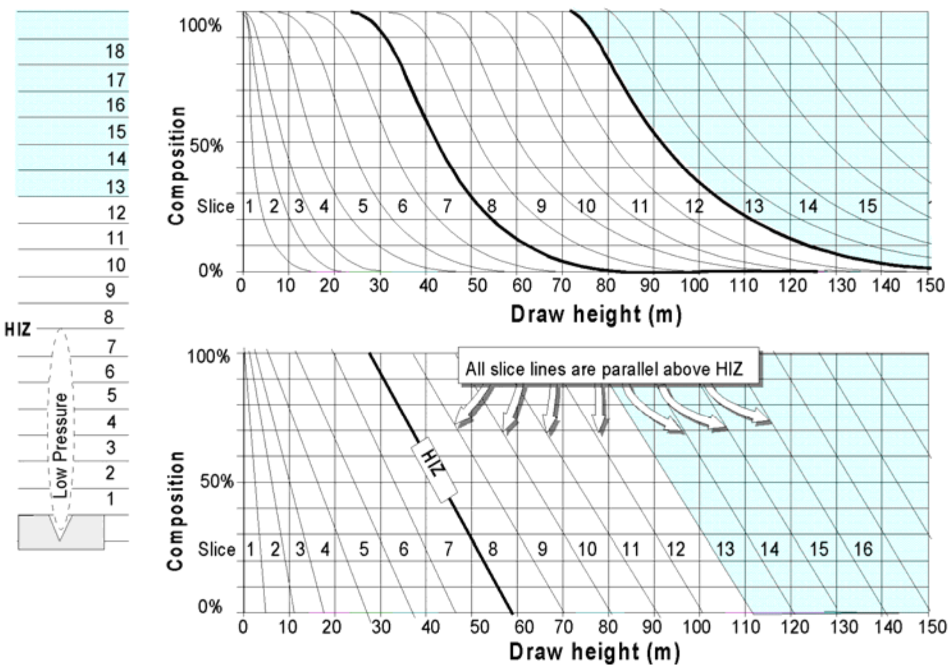


Figure 16 **Grade analysis graph for interactive draw**
Top graph: Theoretical proportions Bottom graph: Simplified graph for manual grade calculations.

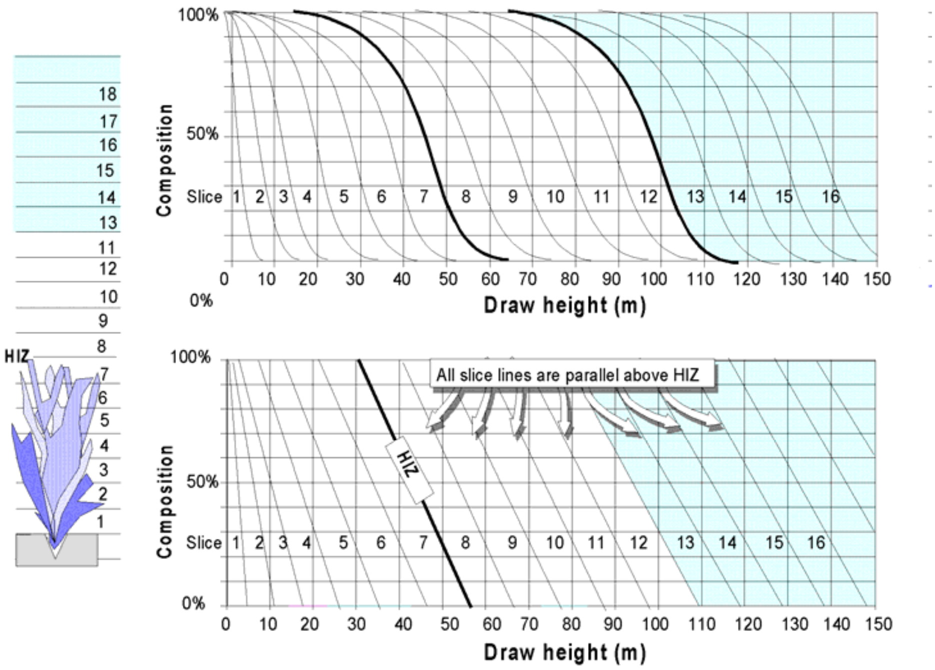


Figure 17 | **Grade analysis graph VD draw**
Top graph: Theoretical proportions Bottom graph: Simplified graph for manual grade calculations.

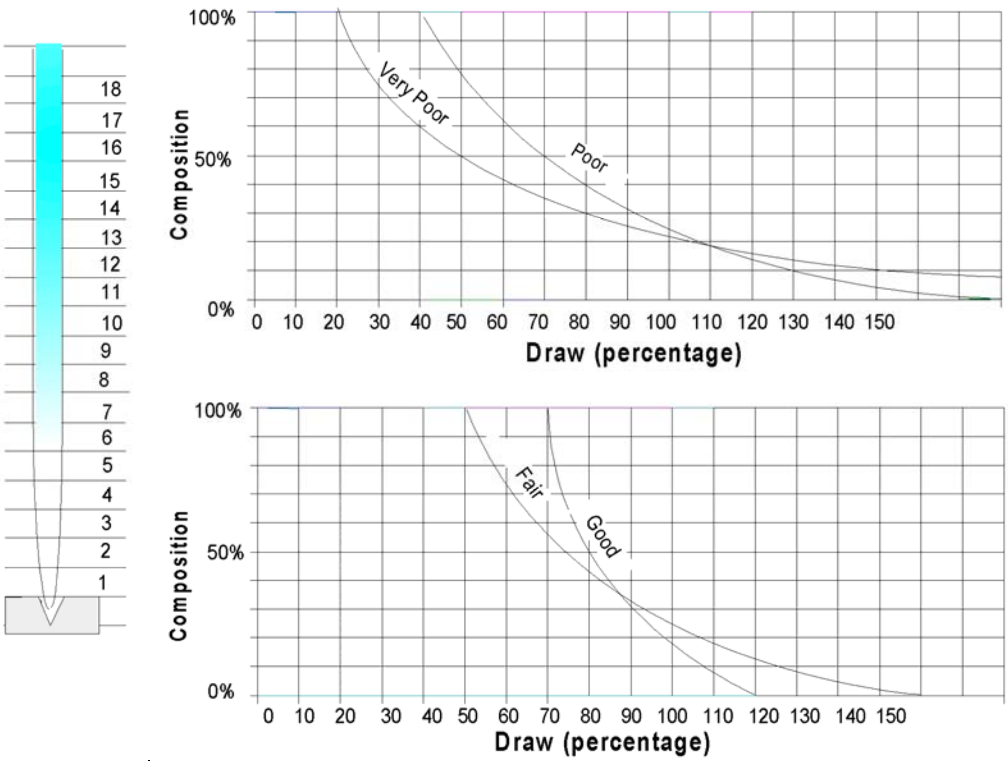


Figure 18 | **Laubscher's (1994) dilution curves with very poor to good draw control deduced from observations in producing block caves.-**

Computer Methods

The tonnage and grade calculations can also be computerised. The commercially available PC-BC code implements the grade graph shown in figure 18 only. This is known as “vertical mixing” in this package.

The steps involved are:

1. Decide on the potential direction of draw and define the draw columns profiles
   Draw these out by hand as a wide draw column profile will give unrealistically large tonnages in peripheral drawpoints, while too narrow a profile will produce gaps between the draw columns for high drawpoints.
2. Transform the block model to suit the layout and draw inclinations.
3. Select an appropriate mixing factor for vertical mixing. Note the difference between the programme implementation of vertical mixing and the interactive and VD mechanisms. (figures 15 and 16)
4. Calculate the heights of draw.

PRODUCTION PHASE – DRAW CONTROL STRATEGIES AND SYSTEMS

Strategies

The usual strategies that can be employed in drawing a block or panel cave are:

1 ***Uniform draw down***. This is the best option for recoveries and dilution, but towards the end of the life of the block may leave drawpoints with initially high draws to be worked in increasing isolation. If the ore at that stage of draw is coarse and the dilution is fine, then the remaining tonnages may be lost due to lateral migration of fines dilution.

2 ***Height of draw*** In this, the rates of draw from individual drawpoints are set proportional to the height of the draw so that theoretically at least, all drawpoints in a drawpoint drift or area are depleted at about the same time. If there are large differences in draw heights, it should be remembered that the drawpoints that have to be worked faster, will draw more from their neighbours. It may mean redefining the draw columns and recalculating the tonnages available for draw.

3 ***High Grading*** Earlier or faster draw of the high grade drawpoints may be done to improve the cash flow early in the project, and again, faster draw means more drawn from neighbours, redefining the draw columns and recalculating the tonnages may be required. The effect on the dilution will need to be considered carefully.

Other strategies may be adopted such as those aimed at avoiding steep ore - waste interfaces with fine mobile dilution.

In planning the strategy and scheduling the production, care must be taken to ensure that the production schedules and draw strategy can be achieved with the number of drawpoints available with an adequate allowance for drawpoints under repair, hung up etc. Next, the operators and supervisors need to understand what the strategy is and what the potential consequence could be if they deviate from the strategy. This particularly applies to the temptation to work drawpoints running fines in a VD draw in preference to neighbouring drawpoints with coarse ore or even within a single drawpoint (figure 19)

If the number of drawpoints is greater than can be accommodated in orebody footprint, then either the production schedule calls need to be reduced or other reasonable means of increasing the production should be considered.

Consideration of the theoretical aspects of draw control must be matched with consideration of the practical problems facing production personnel in meeting production schedules. When production personnel are faced with poor drawpoint layouts, lack of equipment, or poor supervision, they may have real difficulty in meeting tonnage calls from some drawpoints. The reason for this is a result of hangups, coarse fragmentation, drift or drawpoint damage. To meet overall calls they may preferentially work drawpoints that are flowing freely with fines. This is could have a very detrimental effect on the ore/waste interface and introduce early dilution onto adjacent drawpoints. Everything must be done to ensure that production calls can be practically met and that production personnel have the equipment and sufficient drawpoints to produce the planned tonnage without prejudicing the whole block.

In LHD layouts, a major factor in poor draw control is the drawing of fine material at the expense of coarse material. That is, the operator continues to load the drawpoint as long as fines are available. As the fines can run for a long time and through small openings, means that fines come from remote areas. A strict discipline is required if a uniform drawdown is wanted. Coarse rocks must be blasted in the drawpoint on the day that they report there, i.e. if management have provided the means to do this.

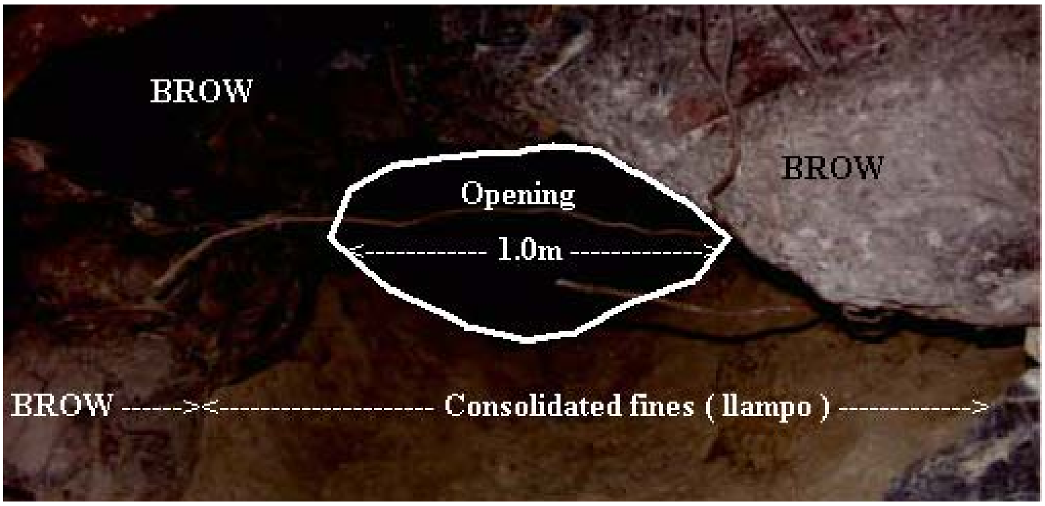


Figure 19 **The above photograph shows a LHD production drawpoint with an opening of 1.0m by 0.4m through which fines were drawn. No attempt was made to remove the consolidated fines to establish proper draw.**

DRAW CONTROL PROCEDURES

Having set a draw strategy, procedures are needed to control the draw. These can be entirely manual or manually assisted by computer or other devices. The procedures needed include:

- Methods of informing operating staff of the drawpoint production calls or priorities. These are normally set by the draw control engineer or officer and transmitted to operational staff by paper or electronic call sheets.

- Methods of capturing and recording the tonnages produced. These are commonly manual procedures such as simple manual counters or tally boards to record the buckets pulled from each drawpoint. These counts are transferred to paper report sheets. Recent developments include radio frequency identification systems(RFID) for automatic draw control data collection and transmission to a base computer (Knights et al, 1996). Research work on the automation of mining machinery is developing rapidly. Recently introduced are: on-board micro computers that assist in completely filling LHD buckets, computer and remotely controlled operation of LHDs.

- Methods of reporting the production figures and analysing them. These are usually site specific computer systems or spreadsheets. The PC-BC program could be used for this purpose, but it is understood that few if any mines are using this program for this purpose.

The draw control engineer has the choice of:

- Working all drawpoints on every shift - this provide the best conditions for both interactive and VD draw, but may not be practical where the calls per drawpoint are small, or where there is a need for frequent secondary blasting or repairs.

- Working half the drawpoints on one shift (or day) and half on the next shift (or day), which may be achieved by working alternate drawbells (figure below) or alternate extraction drifts crosscuts (figure below). The sequence given in figures 1A and 1B is superior to that shown in figures 2A and 2B for both interactive draw and VD draw.

When figures 1A and !b are superimposed there is a uniform appearance to the drawdown. When figures 2A and 2B are superimposed, the darker overlap zones could become preferred channels in VD draw, and pull fine dilution into the both drawpoints.

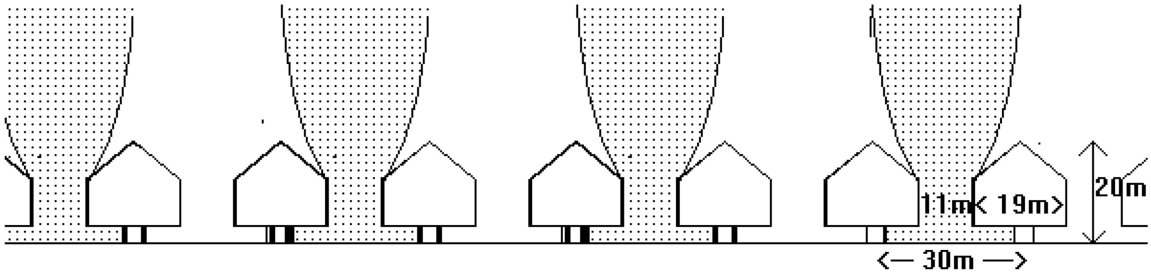


Figure 1 A - draw pattern on day1 with interactive drawbells

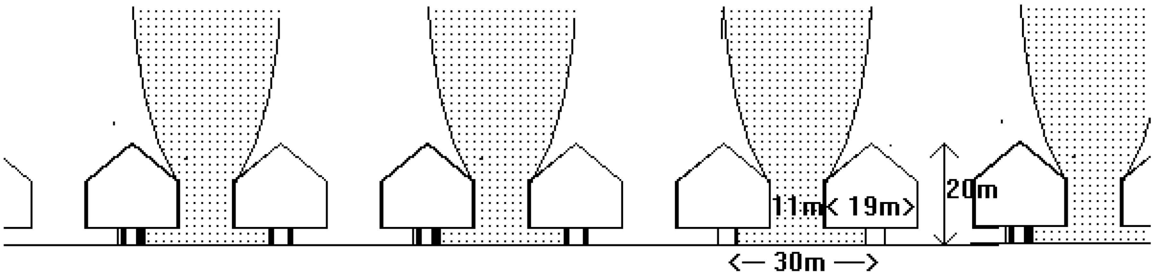


Figure 1B - draw pattern on day 2 with alternating interactive drawbells

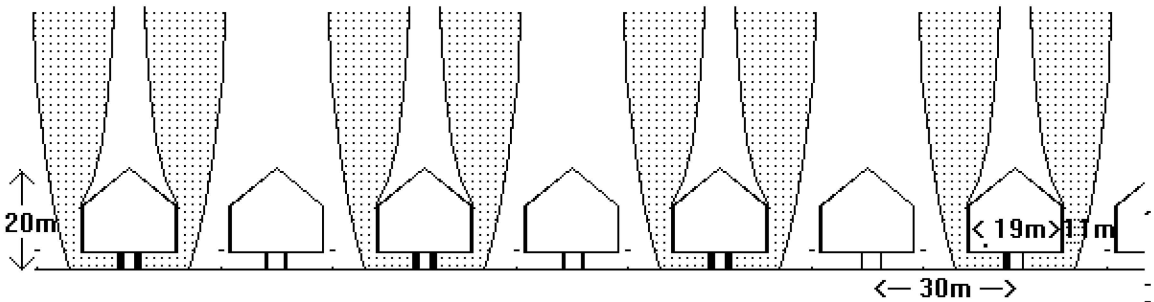


Figure 2A - draw pattern of drawpoints either side of major apex i.e. from extraction drift

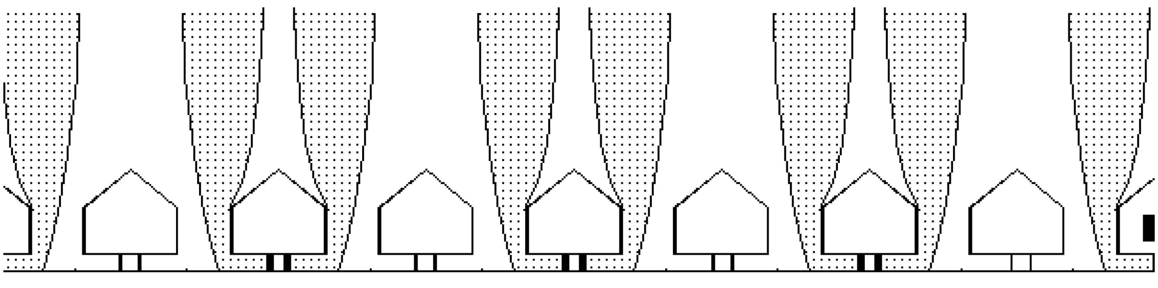
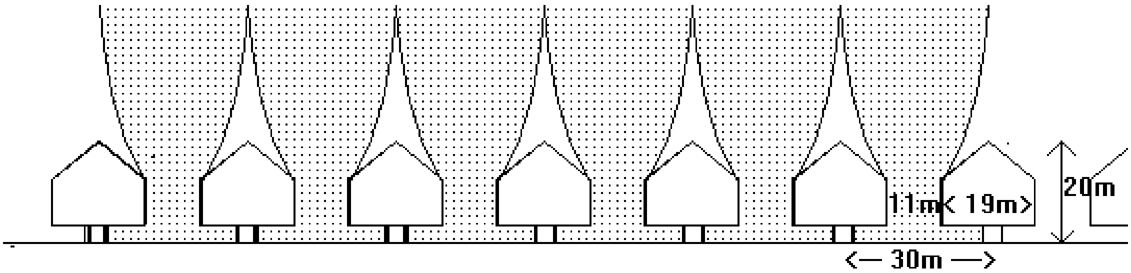
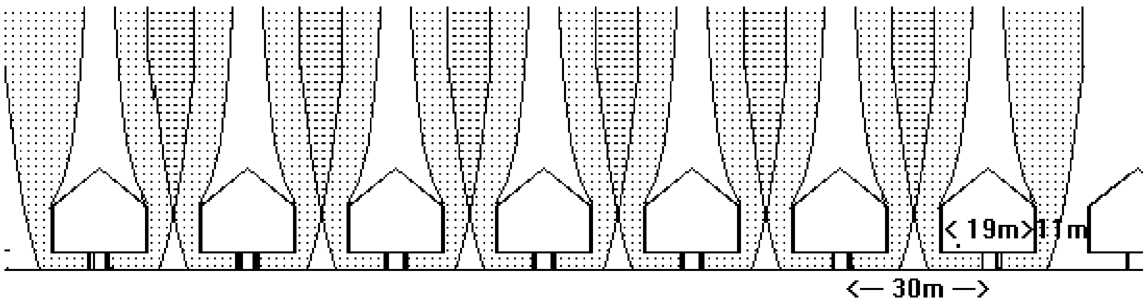


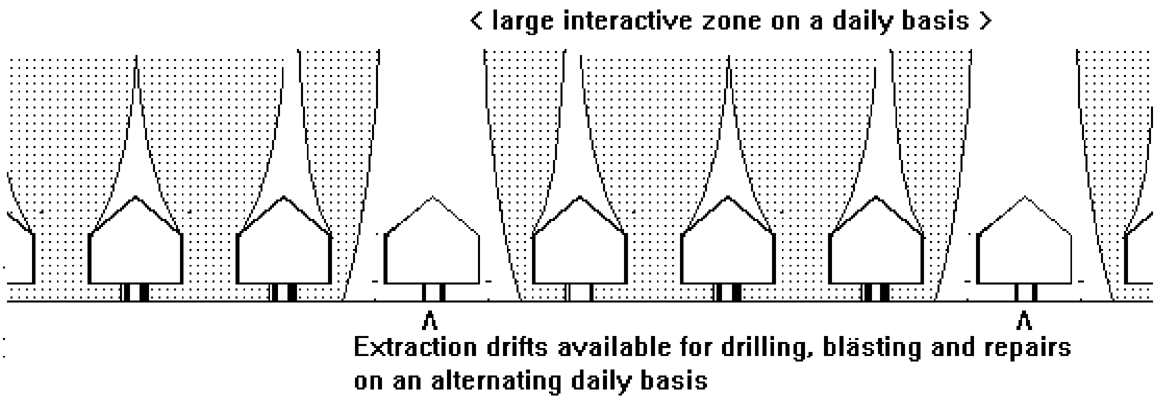
Figure 2B - alternating extraction drifts



Figures 1A and 1B superimposed



Figures 2A and 2B superimposed



DRAW MODELLING AND DATA PROCESSING - THE PC-BC PROGRAM.

The PC-BC program is a major suite of computer routines designed to assist an experienced user to model the draw in a block cave using a number of user-defined control parameters. Potential users require one-on-one training and on-site assistance from Gemcom in setting up the initial models. The documentation that accompanies the program is insufficient for the potential user to understand how the program manipulates the tonnage and grade data except in an elementary and conceptual way.

The writer cannot unreservedly endorse this program. In our use of the program we have identified several problem areas and these have not been resolved. The requested evaluation copy of the software to test the significance of these has not been made available because of the need for Gemcom to provide on-site assistance in setting up the test models.

The following description of the operation of the program has been based on information kindly provided by Dr Tony Diering, Chief Technology Officer, Gemcom Software International, Vancouver, Canada.

This program has been used mainly for modelling the tonnages and grades that could be produced from a block caving operation with various draw scenarios, and to evaluate draw strategies. It maybe used to do the tonnage accounting during production and to control the draw.

The program relies in a number of user-defined draw parameters that are used in ways defined by the authors of the programme. There is some latitude for fine tuning these, based on actual results. While it purports to be based on the theory of interactive draw, in its implementation it is distinctly different. In the program the draw is modelled as a series of overlapping draw cones (equivalent to Kvapil’s ellipsoids of draw). Each drawpoint has a draw cone above it, shaped like a tapering cylinder with a semi-ellipsoid shape at the base. The shape of these draw cones is defined by the user who specifies its diameters at specified elevations. It is always circular, but the user can define its inclination. Where the draw cones overlap, the overlap area is called "shared" and the remaining area is called "unique".
When the draw cones are defined and drawpoint locations entered, the block model can be transformed to PC-BC slice file in which each drawpoint has a series of records for set increments in elevation.

The slice file record contains:

- the volume of ore and volume of waste,
- the tonnage of ore and tonnage of waste,
- the dollar value of the ore waste mixture,
- the proportion of fine material (%),
- the proportion of "unique" material (%) and
- the initial elevation.

In the program, the criterion for "isolated" drawpoints is user-defined in terms of the ratio of the tonnages drawn from a drawpoint and its neighbours. A ratio of three is suggested, i.e., a drawpoint is deemed to be working in isolation when it has drawn three times as much as its neighbours, irrespective of how long it has taken to draw the tonnage. When a drawpoint is worked in isolation, cross draw point mixing between that drawpoint and its neighbours is de-activated. The tonnage drawn while the drawpoint is being worked in isolation is taken from the full user-defined draw cone i.e. both the unique and shared portions. This is then equivalent to the maximum possible horizontal mixing.. "Vertical" mixing parameters are not changed for isolated drawpoints and vertical mixing stops at the HIZ. There is no horizontal mixing for isolated drawpoints.

When adjacent drawpoints are worked simultaneously, the "shared" portion is split between the adjacent working drawpoints. If the "horizontal mixing" feature is enabled, the division of the "shared" material between the drawpoints follows user-defined parameters to allocate more of the shared material to the drawpoint with the higher draw rate. Tonnage in the "unique" portion of the draw cone is not available for redistribution to adjacent faster drawpoints. In both VD and interactive draw, this tonnage would be available for redistribution.

Two forms of mixing are recognised "vertical mixing" and "horizontal mixing" (or redistribution). The program recognises that when a drawpoint is worked faster than its neighbours, it will draw some tonnage from its slower neighbours. "Horizontal mixing" is the tonnage accounting process that debits the reserves of slow drawpoints with a user-defined proportion of the tonnage differential between the slower and faster drawpoints. The user has some control over the proportion of the tonnage differential that is transferred and he can control how the transfer is implemented. This process smoothes any large differences in tonnage drawn between adjacent draw points by moving some of the shared tonnages drawn from the slower draw column to an adjacent column that is being drawn faster. Only the shared tonnage is moved, unique tonnage can't be transferred. It is not applied to isolated drawpoints.

"Vertical mixing" is the mixing that occurs between slices as they are drawn down and approach the drawpoint. The user may define the proportion that one slice mixes with the slice(s) above it when the lower slice is removed. Different mixing factors can be specified for the fine and coarse fractions Separate grades for fines and coarse components can be tracked. However, during input, each block in the block model is assumed to have a single grade. Thus, for example, if a block with high amount of fines and low grade mixes with a block of low fines and high grade, the separate grade components are tracked..."Vertical mixing" may be implemented as a "pre-mix" to enable the program to calculate the tonnages available for draw, the recoverable grade and dilution percentage. Alternatively it may be run iteratively as for modelling a draw scenario. In this application the routine is run whenever the tonnage in the lowest slice is drawn. As each slice is moved down it is mixed with the slice above. The mixing process works from the first slice up to the "height of mixing" (equivalent to the HIZ). However, in our modelling runs, changing the HIZ has no significant effect in dilution and grade estimates. The HIZ can be modified "on the fly" by Laubscher's Draw Control Factor (i.e. the standard deviation of tonnages drawn). PC-BC has an algorithm for estimating HIZ, but this is different from the one recommended in this manual.

CAVE AND DRAW MODELLING - THE PARTICLE FLOW CODE (PFC).

This is a research tool for studying the failure of material in the cave back and then its behaviour while it is being drawn down. It is not suited to calculating recoverable grades or tonnages, for financial modelling or for production scheduling, but can be used to track particle flow trajectories that can be used in other modelling and analysis processes.

PFC models the movement of spherical particles by a distinct element method, which uses a time-stepping explicit method of calculation which allows systems that are unstable to be modelled without numerical difficulty (Lorig et al 1995). The program can operate in either two or three dimensions, with large numbers of particles. But the computational speeds can be long where it is necessary to model large numbers of particles, or the particles have to move large distances or move slowly. PFC has the ability to accurately represent the mechanics of gravity flow, but as in other modelling processes, it suffers from our poor knowledge of the strength of caved rock.

PFC has been used to investigate caving sequence(s) and the subsequent trajectories of the caved material at Northparkes where a large void collapsed suddenly. The modelling suggested that the ore in the back caved in a particular sequence and that some of the initial material that fell from the back had a sideways displacement as well as a vertical displacement.

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Bell N J W (1992) “Technical Note on M/4E draw Markers” (AA Mines Group reports)

Mujuru FC (1995) “Geotechnical cave modelling at King Section, Gath’s Mine, Zimbabwe” *African mining 1995* Institution of mining and Metallurgy London p 401-410

Alvial J (1992) “Analysis of the extraction at El Teniente 4Sur LHD “ *Massmin 92, SA Inst. Mining* Metall, Johannesburg p 233-246

Gustafsson P (1998) “Waste Rock Content Variations During Gravity Flow in Caving” *Doctoral Thesis No 1998:10*, Lulea University of Technology, Lulea, Sweden.1998

Lorig L., Gibson W., Avial J. and Cuevas J. (1995) “Gravity Flow Simulations with Particle Flow Code (PFC)”, Intern*at. Soc. for Rock Mech. News Journal* V3 N1 Summer 1995, p18-27.

Kinghts P.F., Henderson E. and Daneshmend L. K. “Drawcontrol Using Radio Frequency Identification Systems”, *CIM Bulletin* September V98 N1003, 1996,

Technical Note - Draw control Workshop - **MASMIN**

Draw Control at King Mine, Mashaba, Zimbabwe

Summary

The draw control system on the initial section of the false footwall block caving mining method which has recently been depleted at King Mine was a success. The draw was monitored using markers to determine the flow of rock to the drawpoints.

Introduction

From the observations both on surface and underground during the mining of the Over 3 blocks, horizontal hand worked drawpoints on 276 level, it was predicted that inclined draw was probable. This was described in the "false footwall" block caving method for the Main/4 block [3], a hangingwall draw angle of 60° was predicted [1 & 2]. Figures 1 and 2.

Further theoretical experimental work was carried out using the draw model at Shabanie which showed that flow was away from the high load side of the model to simulate the hill feature.

In order to monitor this prediction, a series of markers was placed on 296 level (in existing development).

The markers were recovered from the drawpoint as they were drawn and the results are now analyzed.

The Markers

The markers were installed on 296 level in the haulage development for the previous mining operation. The markers were thus at a position where, in theory, they would report at between 50 and 75 percent of the ore tonne for draw.

These markers were of two types: the first was reinforced 1,0 m. cubes of concrete poured in situ on the level; the reinforcing having numbers welded to it for identification, the other type was old 2 cu. yd. LHD tyres which had been cleaned of all loose rubber to reduce the contamination hazard and with appropriate identification tags bolted inside the tyres. The tyre markers are the easiest to install.

Recovery of the Markers

To June 1990, 15 markers were recovered out of 39 placed in the area concerned. Their recovery was facilitated by awarding a bonus to the finder.

We are currently collecting markers and data for the North-East Extension draw and will continue to place markers at suitable locations for further monitoring.

Results

See Table 1.

The results have been sorted as follows :-

Those with a horizontal movement of less than 6,0 m (the radius for full overlap of circles drawn from the brow positions) regarded as being normal gravity draw (five markers).

Those with a horizontal movement of between 6,0 m and 12,0 m (a radius to cover the draw centres of the immediate neighbours) regarded as being an extension of normal gravity draw with interaction (five markers).

This left 5 markers which were plotted on a plan as if they all reported to the same drawpoint (Figure 3) which shows a scatter round the predicted bearing of 310°, actual mean 305°. It was therefore regarded as being a good approximation. Using predicted bearing of 310°, the true draw angles were calculated giving a mean of 69°, indicating that we had anticipated too flat an angle at 60°. A plot of the true draw angles on a composite section (Figure 4) shows that there was a fair degree of interaction between drawpoints as is predicted by the theory.

The draw percentages at which the markers reported has not been studied fully but it is interesting to note that they were generally higher than anticipated indicating that dilution enters earlier than normally anticipated probably around the 50 % mark. There is also a strong possibility of the draw pattern changing during the life of the block - starting as normal gravity flow until a link to the previous cave and then inclining owing to the increasing influence of the hill. Despite the obvious early entry of dilution the grade did not fall off as it did in the over 3 probably owing to the better draw control - no loss of drawpoints.

Conclusions

It was worthwhile installing the markers and the practice will continue as the mine moves down. The data for the remaining areas of the Over 4 will be analyzed as the blocks are completed, this should give us invaluable data on the changes of direction and draw angle relative to the hill feature for the various mining areas.

The data from the Main/4 has been used for the planning of the Main/5 and a steeper draw angle of 70° will be used and the bearing of 310°.

References:

1. **D.H. Laubscher** and **T.G. Heslop**
1981 "Draw Control in Caving Operations on Southern African Chrysotile Asbestos Mines"
Society of Mining Engineers of the American Institute of Mining and Metallurgy. pp 755 - 774.

2. **Heslop, T.G.**
"The Application of Interaction Draw Theory to Draw Control Practice in Large Chrysotile Asbestos Mines"
Mining and Metallurgy in Zimbabwe 1983 Vol. III pp 290 - 313.

3. **Buchanan, G.F. et al.**
"The 'false footwall' mechanised caving method, King section, Gath's mine, Zimbabwe"
IMM African Mining '87 pp 53 - 62

TABLE 1

Shabanie and Mashaba Mines Pvt Ltd.

King Mine

Draw Markers Recovered in the Main / 4

Drawpoint Level	Drawpoint Number	Marker Number	State of Draw tonne	State of Draw %	Angle	Bearing Marker to Drawpoint	H.D. m	Corrected for a Bearing of 310: Draw Angle	Bearing diff. deg	HD m	Vertical Distance
339	8E	423	18362	165	86	90	2.6	87	40.0	2.0	41.2
347	11E	346	24403	144	85	128	4.3	85	2.0	4.3	49.9
355	4E	325	25581	108	86	59	3.5	89	71.0	1.1	55.9
355	11E	344	18928	71	85	260	4.4	87	50.0	2.8	56.7
364	4E	324	38651	82	88	138	2.1	88	-8.0	2.1	66.2
				114	86			87			
339	5E	326	4015	104	79	306	7.6	80	4.0	7.6	41.3
347	14E	369	20810	95	80	313	8.7	80	-3.0	8.7	50.3
347	13E	370	39167	158	78	25	10.5	87	285.0	2.7	50.3
347	12E	345	13692	54	78	231	10.2	88	79.0	1.9	50.2
364	11E	343	19693	50	83	236	7.6	88	74.0	2.1	66.7
				92	80			85			
347	13E	366	39070	179	51	324	39.9	52	-14.0	38.7	50.3
364	4E	322	36486	77	75	310	17.7	75	0.0	17.7	66.2
364	5E	321	41163	101	68	288	26.1	70	22.0	24.2	66.2
372	11E	339	67103	186	68	290	29.3	70	20.0	27.5	75
372	12E	365	35774	99	79	314	14.4	79	-4.0	14.4	75.1
				128	68	305		69			
		Grand Total		112		310 Predicted bearing					

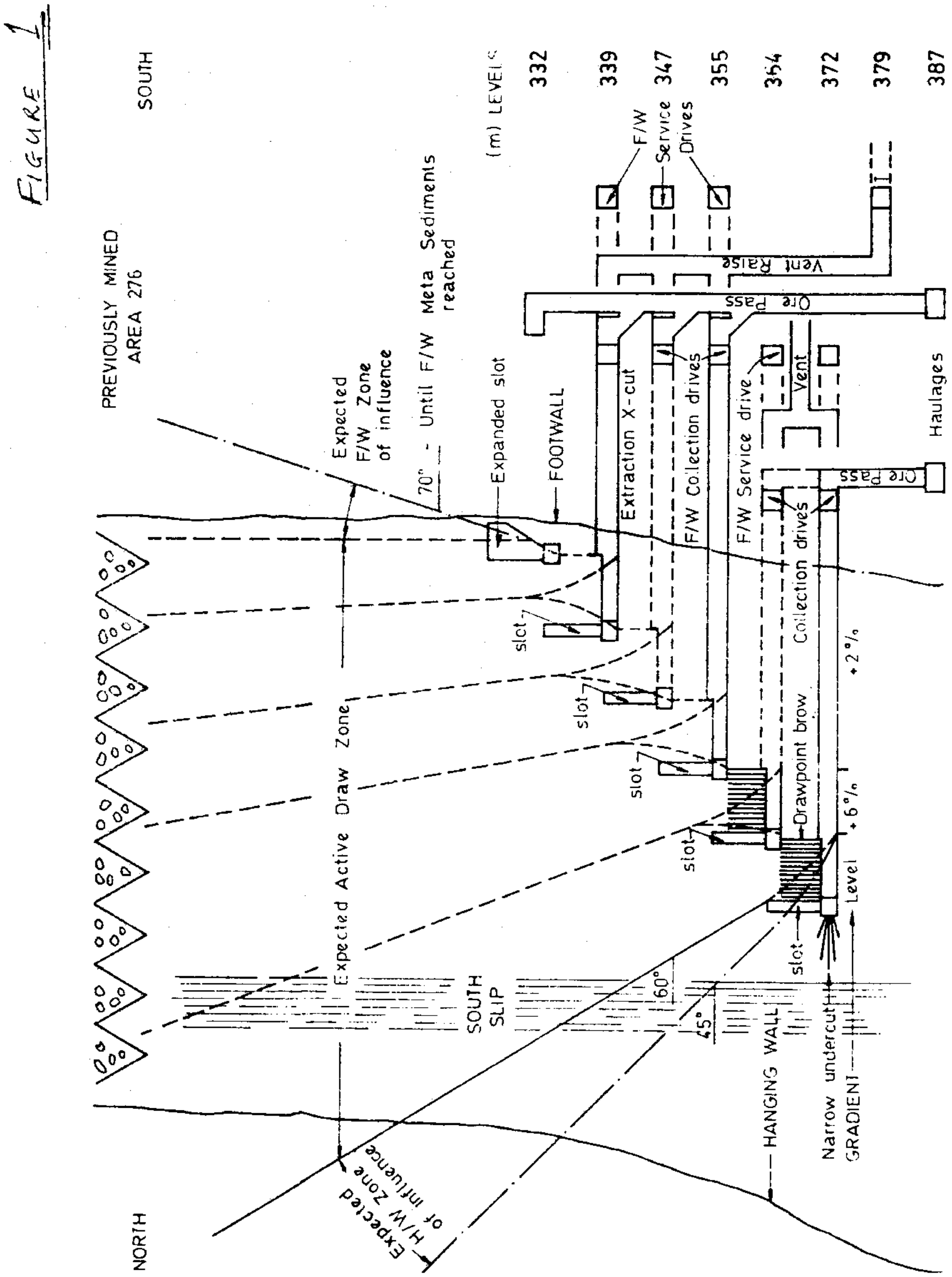
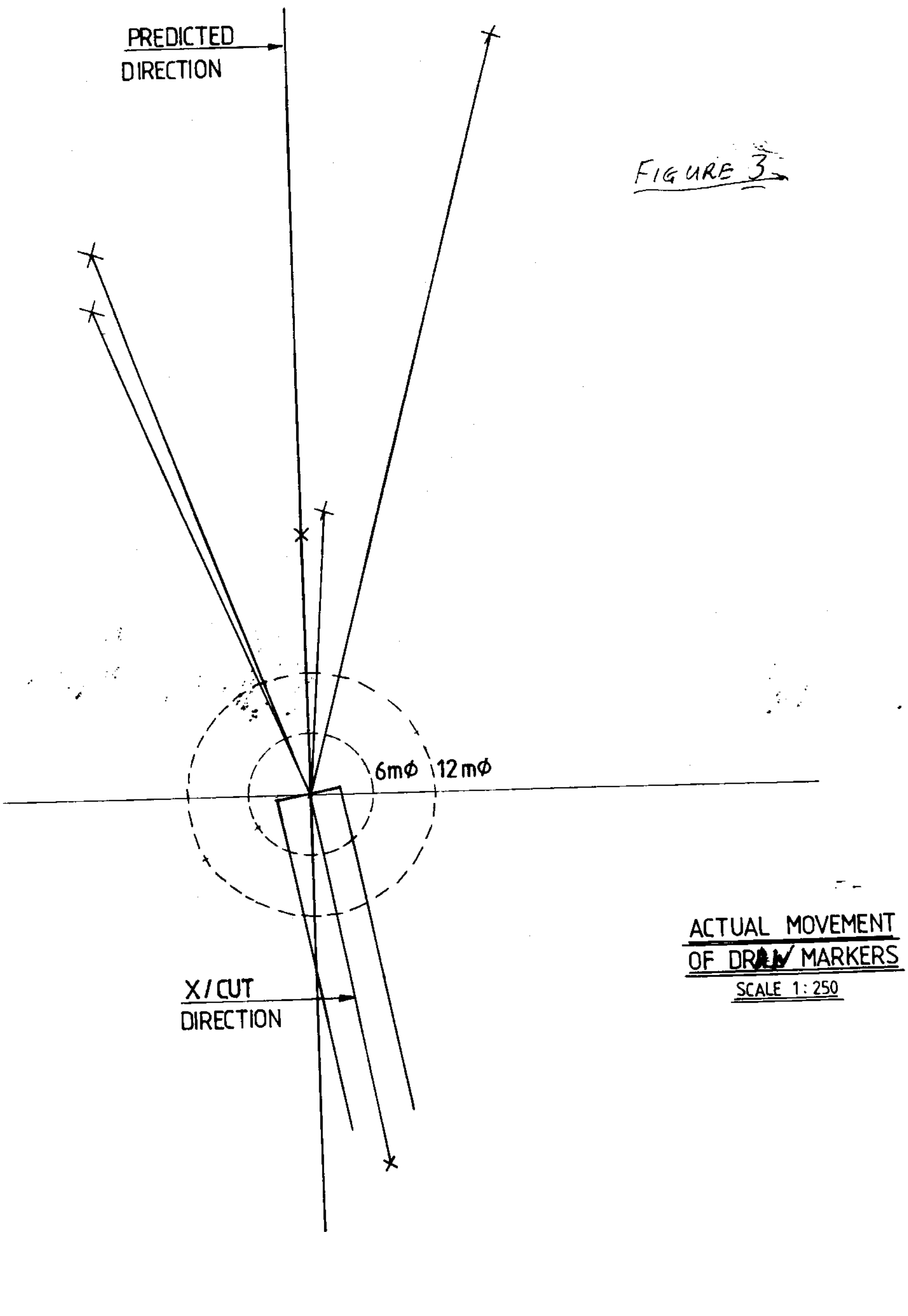
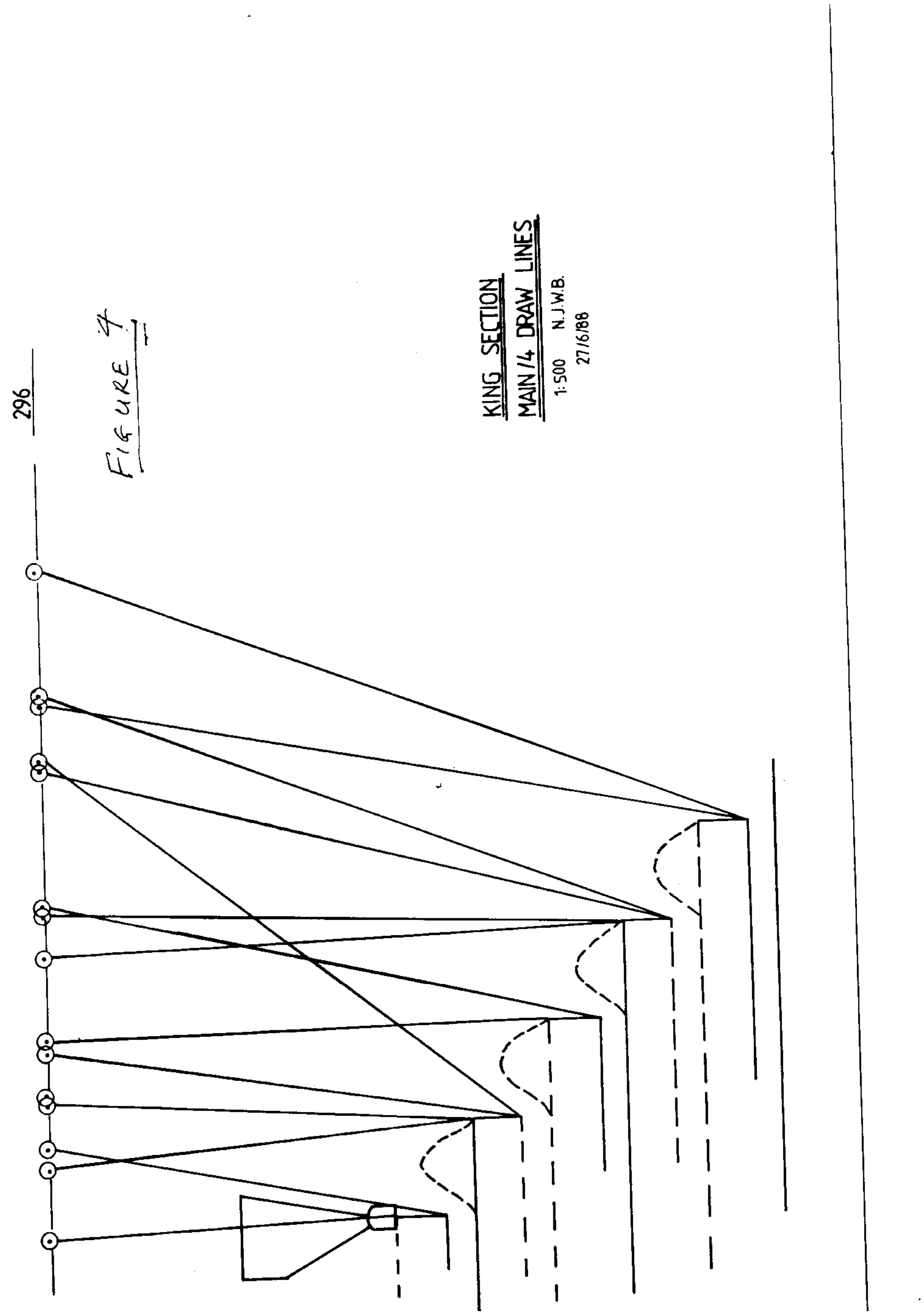


FIGURE 1
SOUTH
NORTH
PREVIOUSLY MINED AREA 276
Expected F/W Zone of influence
70° - Until F/W Meta Sediments reached
Expanded slot
FOOTWALL
Expected Active Draw Zone
Expected H/W Zone of influence
SOUTH SLIP
60°
45°
HANGING WALL
Narrow undercut GRADIENT
Level
+6%
+2%
slot
Drawpoint brow
Collection drives
F/W Service drive
F/W Collection drives
Extraction X-cut
Vent
Haulages
Ore Pass
Vent Raise
F/W Service Drives
(m) LEVELS
332
339
347
355
364
372
379
387


PREDICTED
DIRECTION
FIGURE 3
6mØ
12mØ
X/CUT
DIRECTION
ACTUAL MOVEMENT
OF DRAW MARKERS
SCALE 1:250


296
FIGURE 4
KING SECTION
MAIN/4 DRAW LINES
1:500 N.J.W.B.
27/6/88

COMMENTS FROM N.J.W.BELL ON PHYSICAL MODEL TO DETERMINE RELATIVE PARTICLE SPEEDS IN DRAWDOWN

SPEED DRAW MODEL

A. Aim

1. Draw model to get relative rates of movement of particles within a draw column.

B. The Model

1. A 3m length 50cm diameter pipe from Shabanie to be used. Mounted on suitable solid tripod, with a flange at the bottom to carry the drawpoints and the draw control mechanism. Pipe to be cut open and hinged in sections at 200mm each to allow for ease of loading and subsequent unloading after draw tests are complete in each case.

2. System will be to construct the model, complete with major and minor apexes.

3. Scale of the model will be 1:100, ½m diameter pipe representing 50m on the ground. 1m of material in the pipe vertically representing 100m of ore vertically. Total model 300m.

4. Will need a scaffold to get to the top for loading.

5. At the bottom of the pipe where the draw control mechanisms will be, there needs to be a approximately 1½m height above the ground. The whole will need a firm and level base so that the pipe column is vertical.

6. The model drawpoint centers will be at 12½m scale with major apex 15m wide and the cone between the major apexes at 10m. Major apexes to be shaped and minor apexes to be installed.

C. The Tests

1. 5 tests are planned at this time.

2. The bottom 1.0m in each case will all be regarded as ore. The model will be packed in 100mm layers representing 10m of ore in scale.

3. The first test will be assuming the whole model was in 3B ground situ type fragmentation. Start from the bottom 15m representing the cones and undercut, which will be 3B blasted fragmentation.

4. The second test will be with class 3A with the same "blasted" for the cones and undercuts.

5. The third test Class 2 with the same "blasted" for cones and undercuts.

6. The fourth test will be Class 2 blasted for cones and undercut, followed by 40m of class 2 in situ followed by 20m of Class 5. (We will need to look at using the same mechanism that we used for the sands to give the class 5 a colour, so it is identifiable as being fine fragmentation from that level.)

   The rest of that model will then be class 2 to the top.

7. The fifth model will be as for test 4. Instead of the 20m of class 5 a 5m layer of different SG materials (namely ball bearings and kaylite pellets) to be installed. The particles to be of the same size and matched with granite. Installed at 1/3 of each of the three products to form that layer and see how quickly these migrate relative to each other.

8. After the first test, if deemed practical and necessary, we might have to coat the different sized particles with different colours to aid the sorting of material if screenings is proving inefficient.

D. __Test Procedures__

1. With all the draw plates in place and with the 10mm holes in the bottom plate plugged.

2. Tamping pad to be a round plate 100mm diameter with a suitable handle.

3. Packing the model. The cones will be packed first with "blasted" material and then packed in with a tamping pad.

4. The material is spread then lightly compacted into place. The undercut material is then put into place and again compacted, followed by the rest of the bulk of the ore in 100mm (10m) scale layers with correct mix for that each time compacted in and the next layer added in until the model is full to the top.

   Remembering that in each case the packing of the model will be done by weighed fractions in the correct ratios.

5. The material drawn from each of the twelve drawpoints will be collected separately.

6. The drawing procedure will be: open each draw point at a time, firstly the 10mm holes and collect the material that comes through those holes. Once each of the drawpoints is worked to that limit the 25mm holes are then opened. The draw from each one by free flow and with suitable working using a piece of wire on a handle so that the wire cannot protrude into the scale model further than scale of 6m.

   Once all the drawpoints have been drawn to this level, open the 35mm holes and repeat the draw pattern again. Finally move to the 50mm holes.

   If the material starts to flow through the hole the next smallest aperture shall be put in place to restrict the flow till back to the 10mm hole and let it run until again it hangs up.

   Need to calculate what we need for 10m of draw, or the equivalent of 10m of draw from each drawpoint in terms of volume and this should be marked carefully in the receptacles, to collect the material drawpoint by drawpoint. When this volume is reached the drawpoint is stopped and shut off (if necessary plugging the 10mm hole again). Move on to the next drawpoint.

7. At the end of each cycle: i.e. after the 10mm or 25mm or 35mm or the fully open hole at 50mm in the base plate is complete. If the equivalent of 10m of draw is reached then stop. The collected material from each draw point is then to be screened back to the constituent sizes and the weights taken compared with the weights that were put in in the first place.

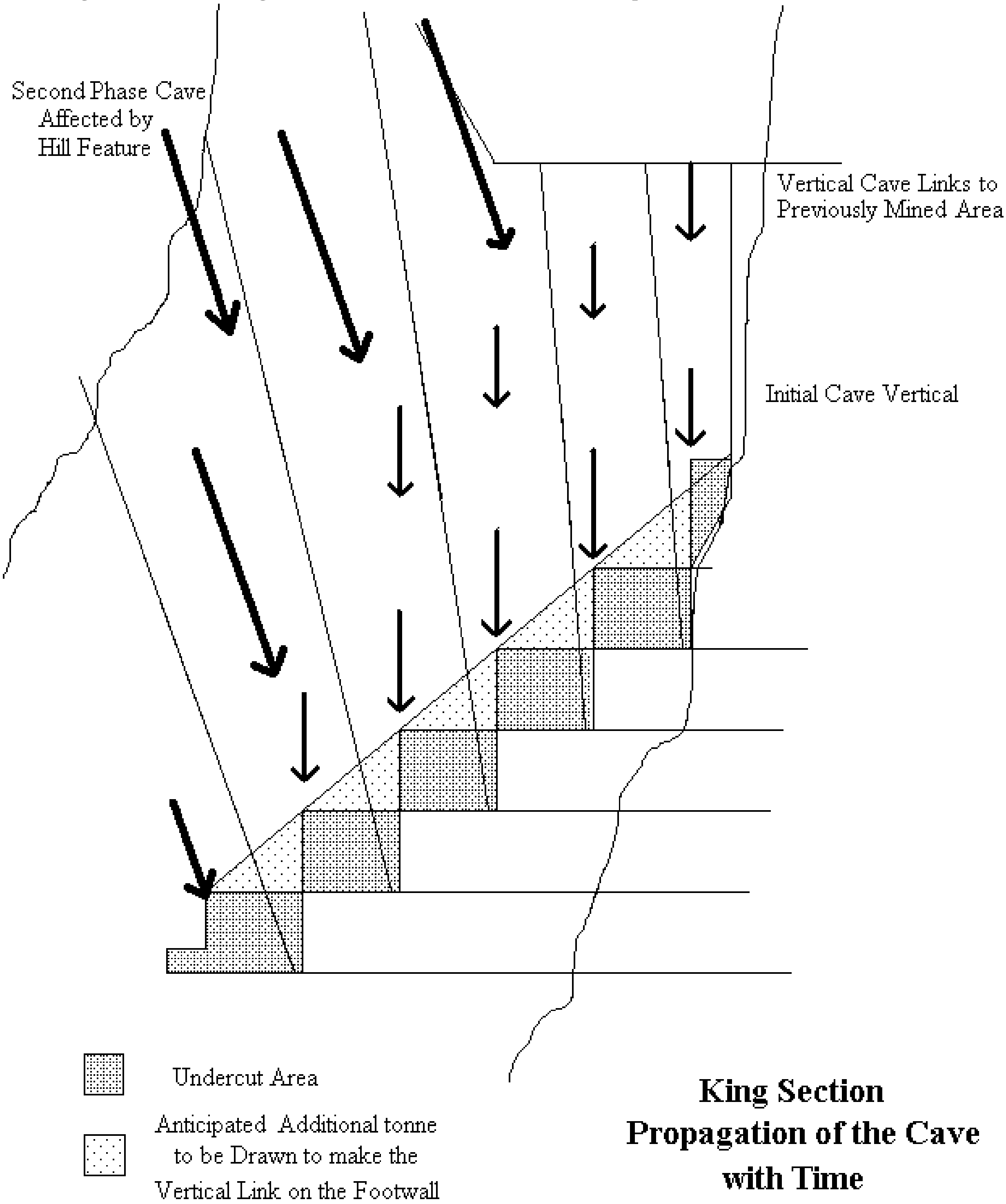
8. Drawing will continue until four drawpoints are hung up with rocks that cannot be maneuvered through the 50mm holes. The cycle being completed when this fourth drawpoint is found with a rock that won't come through.

9. The model is then to be cleared from the top down, taking off 100mm layers and each of these layers will then be screened in the same fashion as the production from the bottom to get a distribution of the particle sizes left in the column above the draw. This will give a very good matching off how the different size material moves through the column and at the different elevations in the column.

NJWB/jal
2000/06/14

DRAW CONTROL - THINKING IT THROUGH

There is a need to visualize how a cave will progress from undercutting to the anticipated final draw columns. This might well influence the way the draw control is managed through the life of the cave block e.g. Gaths Mine King Section Main. The sketch below helps illustrate this.



The caving of King Section Main Ore Body is believed to take place in phases. The initial cave being under gravity and the horizontal stresses, this cave progressing up the footwall side until it links to the previously caved block above. At this point once the strike has been opened far enough, a cave propagates through to surface on an incline and creates a loading of broken rock that 'pushes' the ore into an incline draw.

GENERAL

The Draw shall be controlled rigorously.

1. Only the drawpoints issued for Draw shall be worked, but if this is not possible, any additional drawpoint that is worked shall be clearly shown with the reason stated in the remarks column of the daily return. Draw Control sheets shall be filled in neatly, completely and accurately.

2. **All Drawpoints** shall be numbered clearly underground and the numbering maintained in a legible state to ensure easy and correct identification. Any Drawpoint not being worked shall be chained off. All Drawpoint chains shall be maintained to standard.

3. It is imperative that tonnes drawn from a drawpoint do not exceed that as laid down by the Draw Control Officer as this will introduce waste dilution and reduce the value of the ore.

4. Drawpoint closures: these must be carefully controlled and arrangements made to cater for both Temporary and Permanent Closures. The walling off of drawpoints only needs to be done if the ventilation dictates warrant it or if there is a need for a solid concrete wall to reduce spans and to strengthen an area.

REDISTRIBUTION SYSTEM & DRAW CONTROLS AS DEVELOPED AND USED BY AFRICAN ASSOCIATED MINES

The data from the draw model experiments was analysed and the following system for redistribution devised which was then tested against the data from several of the experiments and found to give a reasonable correlation. A simple data base programme in HP BASIC was devised and has stood the test of time as a useful tool for draw control guidance.

DEFINITIONS

A. 'Effective Area' of a drawpoint is the area obtained from:

$$\text{Effective Area} = \frac{\text{Recoverable Ore Tonne}}{\text{Density x Height of Ore}} \ m^2 \ \text{density set at 2.7 (for AAM Ore)}$$

B. ‘Not Crashed’ Drawpoints

Any drawpoint not covered by the undercut is regarded as not crashed. Where sub level shrink is operative, the Draw Controller will decide where the change will take place, ‘not crashed’ to ‘crashed’, normally after the second level.

C. ‘Working’ Drawpoints

Are those which are crashed and have a tonne / m^2 factor drawn from them equal to or greater than $^1/_{10}{}^{th}$ of the average tonne / m^2 drawn / drawpoint actually producing.

D: ‘Minimum - Interactive Draw Area'

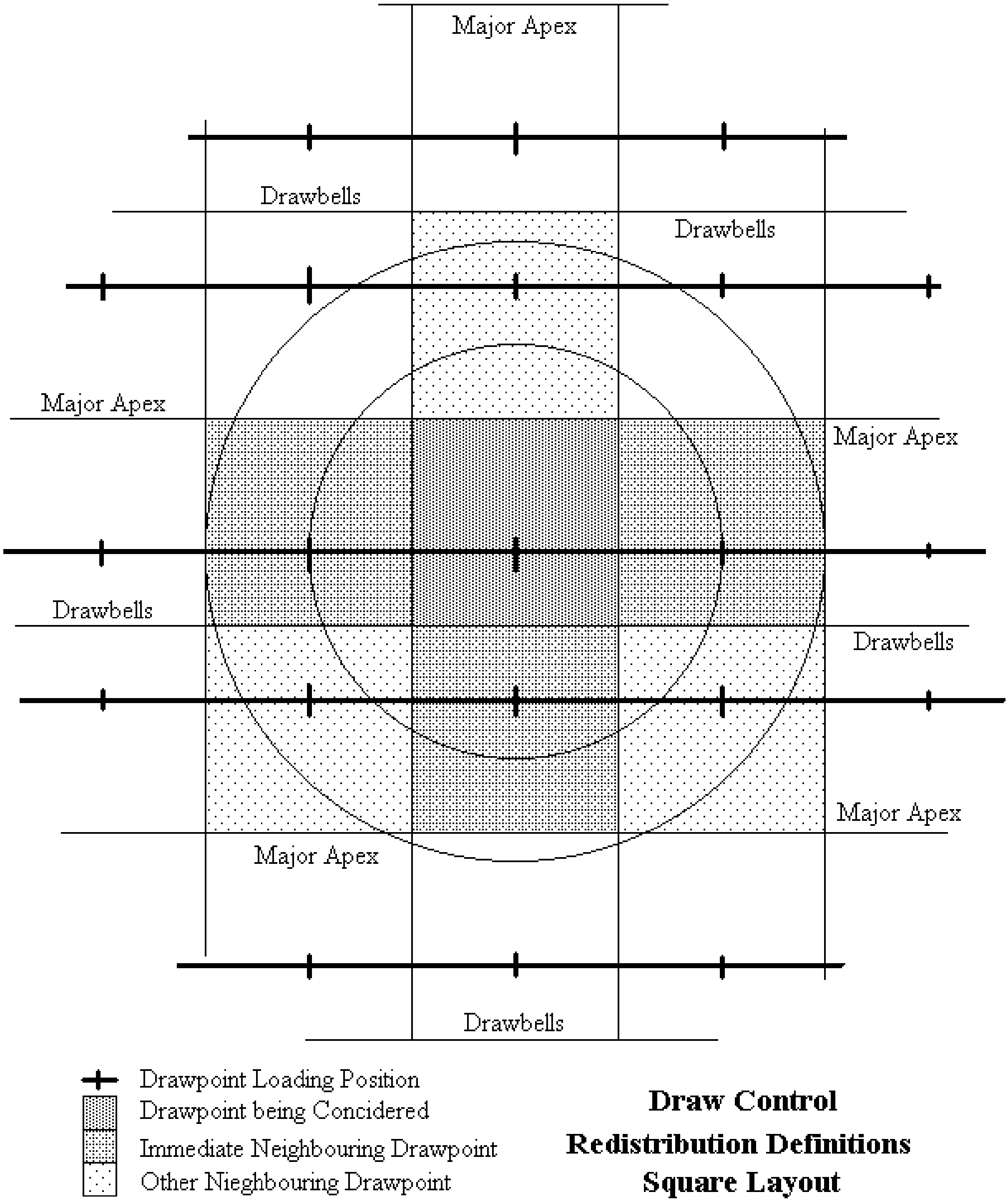
The minimum area, defined by the hydraulic radius, that is required for interaction is dependent on the drawpoint spacing, as follows:

Spacing	7.5 m X 7.5 m	Hydraulic Radius	11 m
	10 m X 10 m		15 m
	12.5 m X 12.5 m		19 m
	15 m X 15 m		22 m

In this minimum area all drawpoints must be ‘working’ and overall have a tonne / m^2 factor greater than the mean of the ‘Working’ Drawpoints less their standard deviation.

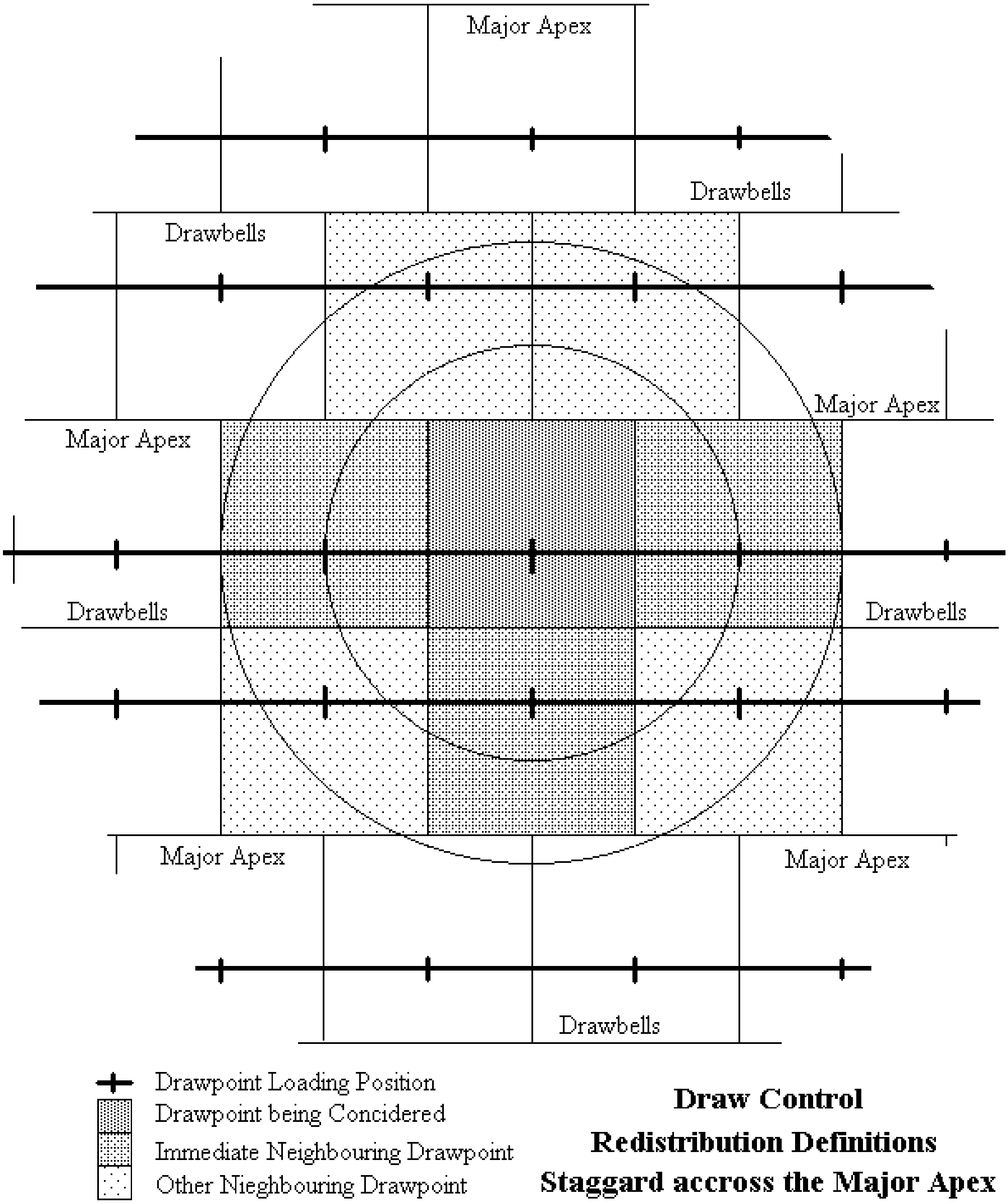
E. ‘Immediate Neighbouring’ Drawpoints

Are drawpoints where their centre of draw lies at or within the standard drawpoint spacing for the block, from the centre of draw of the drawpoints being considered. See sketches below and over.



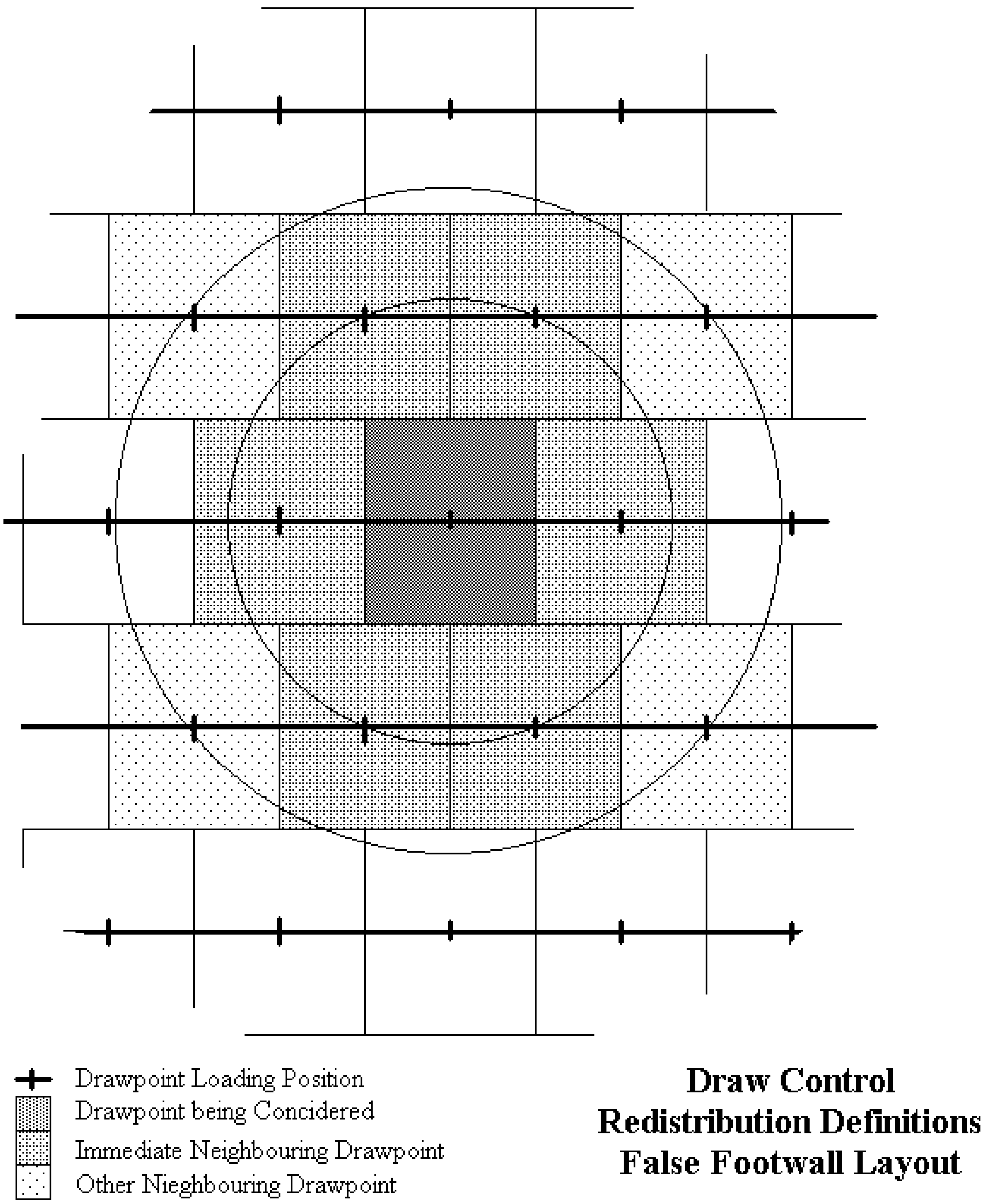
F. **'Neighbouring' Drawpoints**

Are drawpoints where their centre of draw lies within 1.5 X the standard drawpoint spacing for the block, from the centre of the drawpoint being considered. See sketches over.



For E and F

It should be noted that regular layouts are straight forward calculations, however irregular areas or special cases must be dealt with on an individual basis by inspection for a logical approach.



G. Interactive Draw Area

Building on the 'minimum Interactive Draw Area', if a drawpoint is 'working' and has two 'neighbours' which lie within the interactive area it, too, is 'interactive'. In addition, any drawpoint, which has all its neighbours in an interactive draw zone, is regarded as forming part of that zone.

H. Boundary Drawpoints

Are those ‘immediate’ neighbours forming the boundary of the area of interaction and where partial redistribution of tonnage takes place via drag zones, from:

i.	Not ‘crashed’ drawpoints	5% of difference.
ii.	Permanently Closed drawpoints & ‘Major Repairs’ to drawpoints	10%
iii.	Other drawpoints	15%

I. Isolated Drawpoints

Are ones which :

- lie outside the ‘interactive’ or ‘boundary’ areas;
- lie within the ‘interactive’ area but have drawn three or more times the average tonne / m^2 factor for the period for that interactive area.

APPLICATION

A. For each drawpoint from the monthly tonnage drawn and the effective area of the drawpoint calculate a factor ‘monthly tonne drawn per metre square area’

B. From the mean of ‘A’ determine the ‘working’ drawpoints.

C. Determine the mean and standard deviation of the ‘working’ drawpoints.

D. Find a ‘minimum interactive draw area’ (up to 6 independent interactive draw areas catered for in the AA Mines programme).

E. Extend the ‘minimum interactive draw area’ to the full ‘interactive draw area’.

NOTE : For ‘F’ & ‘J’ the ‘interactive draw area’ only is considered.

F. Determine the mean for ‘Interactive Draw Area’ (IDAM) and determine the Moving Average tonne / m^2 for each drawpoint in the interactive draw area with its neighbours in each independent interactive draw area (Moving Average - MA) using the following weighting :

Self	X 3
‘immediate neighbours’	X 2
‘other neighbours’	X 1

G. Establish the ‘boundary’ drawpoints.

H. The ‘boundary’ drawpoints redistribution is now calculated for :

‘Not crashed’ drawpoints - 5 %

‘Permanently Closed’ drawpoints	- 10 %
Drawpoint undergoing ‘Major Repairs’	- 10 %
All other drawpoints	- 15 %

Of the difference between the ‘immediate neighbouring’ drawpoint in the ‘interactive draw area’ and the ‘boundary’ drawpoint being considered. This figure is added to the ‘boundary’ drawpoint and subtracted from the drawpoint in the ‘interactive draw area’.

I. Determine the ‘isolated’ drawpoints and allocate to them the tonne / m^2 factor which they actually drew.

J. Calculate the Redistributed tonnage for each drawpoint in each independent area in the interactive draw area using the following :

$$\text{Redistribution Tonne / m}^2\text{ factor} = (MA) + \frac{(IDAM - MA)}{(3)}$$

K. The Initial Redistribution tonnage is now calculated for each drawpoint :

Initial redistribution tonne = Redistribution Tonne / m^2 factor X Effective areas of draw.

All the Initial Redistribution tonnages are now added together and compared with the total actual tonnage drawn. The Initial Redistribution is then proportioned so that the total equals the actual tonne drawn.

DRAW AIM CALCULATION

The monthly increase in Aim is calculated for each type of Drawpoint: Grizzley, LM Loader, and LHD – by independent interactive area and the whole Block. Using the mean monthly Redistribution Draw data for each type for all the Drawpoints that are Crashed and NOT Permanently Closed except when these are either Boundary or Intermediate, this is then added to the progressive Aim for each Drawpoint by Type by interactive area. If a drawpoint is not in an interactive area the average block figure is used.

DRAW CONTROL

Draw Control is set so that if Redistributed % draw is :

> Aim + 10 % then ** Temporary Closed **
< Aim – 10 % then ## PRIORITY ##

Drawpoint is Crashed and Produces too early or out of sequence and it is desired to hold it back this can be done by altering the Draw Aim for that Drawpoint using the correction routines. The same can be done for Lines / Panels where fixed controls are required.

PERCENTAGE DILUTION AND DRAW

The weighted percentage dilution for the month is now calculated from the input entry point of dilution. The weighted mean dilution can then be calculated for the year and used for the Ore Reserves.

The weighted average percentage drawn from drawpoints worked that month is also calculated.

HIGH DRAW TONNE

Mean tonne drawn from drawpoints Producing more than the mean +1 standard deviation. This figure to be used as a GUIDE for CALL Calculations & Forecasting.

ISOLATION FACTOR

This factor is obtained from the tonne Drawn in Isolation relative to the total tonne drawn.

REDISTRIBUTION FACTOR

This factor is obtained from the amount of redistribution that is calculated, obviously the lower the better.

Tonne REDISTRIBUTION to ‘Permanently Closed’ DRAWPOINTS

This factor is obtained from the tonne Redistributed after permanent closure.

CALL TONNE

This is based on the Call tonne for the Month and the Maximum Production capability of individual drawpoints – Working for 26 days.

a) ## PRIORITY ## Drawpoints are called at Maximum.

b) The rest are relative actual % to aim % drawn using :

Call 1 = Call / No of Drawpoints Open for Draw

If under Aim :

(Call 1 + (Max – Call 1)X(Aim – Actual)/10)X Correction factor for effective Draw Area to Maximum

If over Aim :

(Call – Call 1X(Actual-Aim)/10)X Correction Factor for Effective Draw Area with a Minimum of 10% of mean tonne / drawpoint or 50 tonne, whichever is greater.

c) Drawpoint in Over-Draw allocated Calculated Call X 0.75 ONLY

Add all the call tonne and then correct to exceed required call for the Month up to a maximum of Call X 1.1 by adjusting the Call 1 and rerunning the Data.

Checks made:

i) that NOT more than 25 % of the Drawpoints are on maximum call.

ii) that the Call is reasonable for the number of drawpoints available when compared with the average possible NOT, too small or greater

NJWB/jal
2009/04/29

DESIGN TOPIC

Ore/Grade Extraction

GENERAL

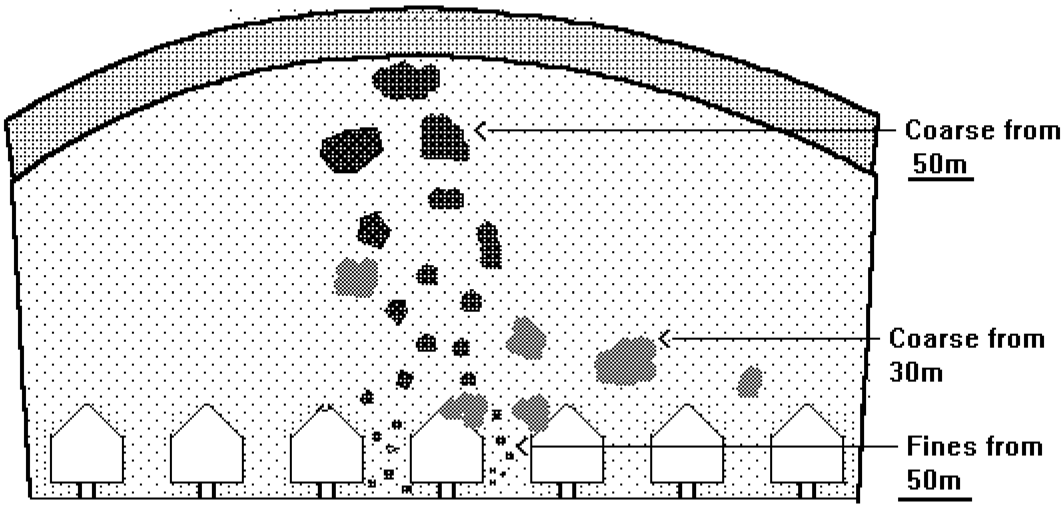
This section is a prediction of what the recovery will be from the caving operation. It is essential that the value and characteristics of all mineralised zones are known, as dilution enrichment can lead to mining considerably larger tonnages than are shown in the ore reserve.

SHUT-OFF GRADE(S)

This is the grade at which a drawpoint is closed - recognising economic aspects, life of mine and grade requirements. The shut-off grade could vary over the life of the mine. Drawpoints could be shut down and then reopened if mineral prices increase or as a reclamation exercise at the end of the life of the deposit. With no further major capital expenditure, shut down drawpoints can now be worked to a lower grade at a profit.

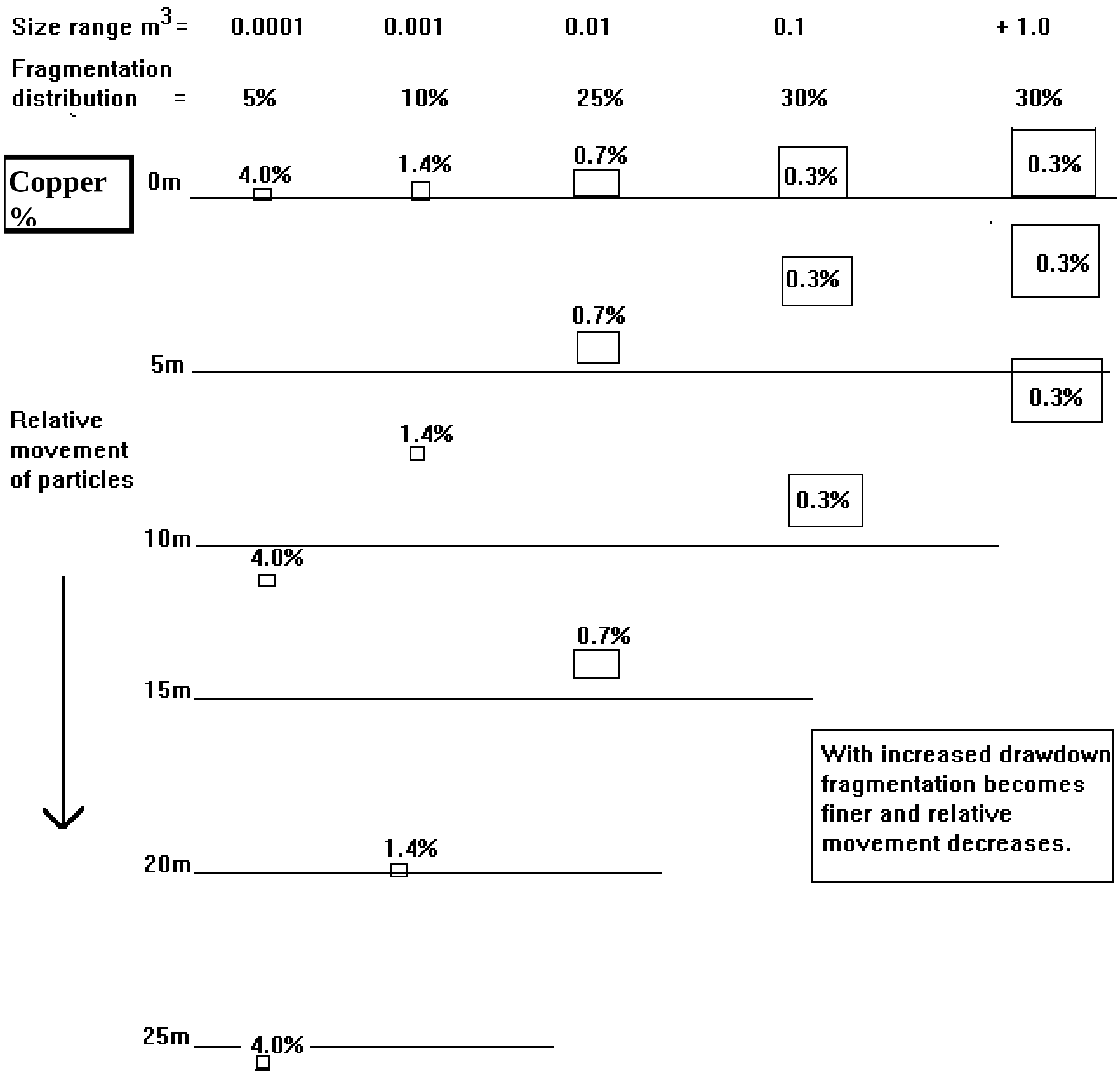
MINERAL DISTRIBUTION - FINES ENRICHMENT

This is a very important aspect which surprisingly enough is often ignored, but could make a big difference to the viability of a deposit. The draw column acts as a large 'jig' resulting in fine material moving more rapidly through the draw column.



FINES MIGRATION RELATIVE TO COARSE PARTICLES

The following diagram is an interpretation of the relative movement of material with a uniform distribution in an active draw column where all particles are moving. The copper grade increases in the lower level owing to the rapid flow of the fine copper minerals:

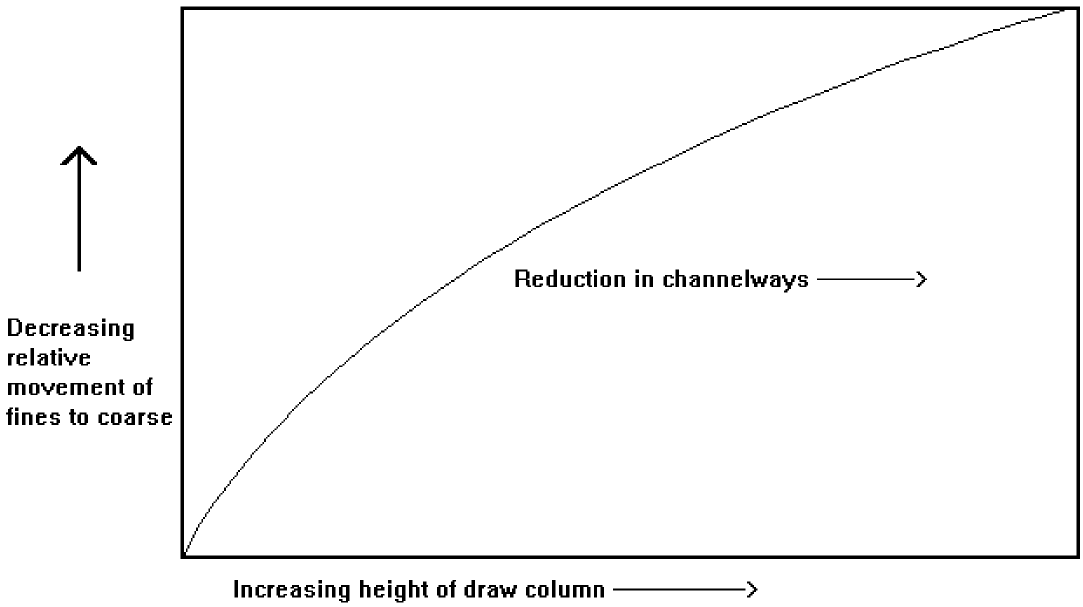


Average grade = 0.7% composed of :- 5% x 4.0% + 10% x 1.4% + 25% x 0.7% + 60% x 0.3% = 0.7%

Migration of fines with drawdown showing relative movement of particles and enrichment of lower portion of draw column, based on geological assessment of the following mineral distribution:-

Location	% of Cu
Open breaks	50
Veins	30
Host rock	20

In the above case, the draw strategy would be to recover as much fine material at the expense of the coarse. However, if the mineral is uniformly distributed through the rock mass and there is a decrease in the grade upward in the column, then the object must be to recover coarse and fine at the same time. In practice this is often not the case, as fines will be drawn at the expense of coarse material because it is easier for the operator to load this tonnage than to break the large rocks. Fines report in a drawpoint in sufficient quantity for loading to continue even though the drawpoint is hungup. We cannot prevent the fines from moving faster than the coarse, but we can prevent this getting out of hand by breaking large rocks as soon as they report in the drawpoint. The flow of the fines could also be interrupted by them settling on large rocks or on the top of an arch, but the relative movement will always be faster for finer material. It will be impossible to accurately predict the relative movements so an engineering judgement has to be made, based on underground observations, as per the following diagram.



PRODUCTION TONS AND GRADE

It is important that the ‘ore’ from a drawpoint be defined as production ‘tons’, which is ore plus dilution. The calculation of that tonnage shows ore losses due to layout, ore losses due to dilution and the quantity and grade of dilution.

At the start of planning there is an ore reserve tonnage and outline. The mining layout will be located within this reserve and this will define the mining tonnage available to the layout. This tonnage will be mined to a shut-off grade which will include a certain percentage of dilution at a value and will represent the production tons. At the end of the draw there will be a loss of ore as the dilution becomes too high. This loss is recorded as a draw loss.

The format to be adopted is :-

1) Ore Reserve Tons, Grade 2) Available Reserve Tons. Grade

3) Ore Loss, % Available Ore Tons 4) % Dilution , Dilution Tons, Grade

5) Production Tons, Grade 6) Recovered Mineral / Ore Reserve Mineral x 100

A spread sheet with figures relevant to the above headings will ensure that much thought has gone into arriving at the correct production 'tons' and that the planner will have a better understanding of the potential problems.

PERCENTAGE ORE RECOVERY

The percentage ore recovered will depend on achieving a good overall performance in all the items that contribute to a block caving operation. This table shows the relative importance of the different items and how they could affect recovery.

ITEMS		**RATINGS**	
	GOOD	***FAIR***	***POOR***
Draw column height dilution / fragmentation	+ 200m =10%	80 - 200m = 7%	-80m = 5%
Mining area - m² large = less side dil.	+90 000 = 7%	± 40 000 = 5%	± 15000 =3%
D/P spacing optimum - good interaction	= 10%	= 7%	= 5%
Drawbell shape, well shaped - good flow	= 10%	= 8%	= 6%
D/P availability, high > good interaction	= 10%	= 7%	= 5%
Interactive draw area	> 100m = 8%	50m - 100m = 6%	< 50m = 4%
Ore fragmentation	uniform, fine to medium = 10%	uniform, medium to coarse = 7%	large range = 5%
Dil. fragmentation	Coarse frag. = 10%	medium frag. = 7%	fine frag. = 5%
Mineral distribution	In fines = 10%	disseminated = 6%	in coarse = 5%
Geometry - drawzones	Contiguous = 10%	Stepped = 7%	Scattered = 2%
Draw strategy	Lines of drawbells per shift = 10%	Lines of drawpoints = 7%	Random. irregular = 5%
Total	**95%**	**75%**	47%

For example 95% ore recovery can be expected from a layout where the production level is in a rock mass with a MRMR of 70, the ore column is in a rock mass with a MRMR of 45, the orebody dimensions on the level, are 400m x 600m, the ore column height is 300m, the orepasses are at 70m spacing and 6 yd LHDs are used with drawpoint spacings of 14m. If the height were reduced to 100m, the MRMR on the production level to 45 and the ore column now has a MRMR of 30 to 70 and the drawpoint spacing increased to 17m then the ore recovery could drop down to 70%. The percentages in the table are a first pass and will be reviewed periodically based on field data.

DESIGN TOPIC

Drawpoint Support Repair

GENERAL

Drawpoints might need repairs in very poor ground conditions such as squeezing ground, when abutment stresses have damaged the rock mass prior to production, or when the original work in developing the drawpoint was not good. It is obvious that initial effort in installing good support more than pays for itself. Poor support could eventually incur repair costs, a loss in production and there is always a possibility that adjacent drawpoints may also be affected. Because repairs are done in a failed rock mass they can never achieve what the original support could have achieved, with a significant increase in the cost per ton.

POOR UNDERCUTTING

Incomplete undercutting leaves pillars (stubs) which load the major apex leading to failure of drawpoints at an early stage of draw. The rock mass begins to fail and usually nothing is done in the hope that the pillar will crush and all will be well. There are no known examples where this wishful thinking has succeeded.

TONNAGE DRAWN - OVERALL / DRAWPOINT - REASONS FOR FAILURE

The tonnages drawn per drawpoint, with reference to the original support, rock mass quality and draw control, needs to be assessed to relate to the reasons for the failure. The reason might be a very high tonnage resulting in wear back of the brow and that the original drawpoint support did not address this problem.

TONNAGE REMAINING PER DRAWPOINT

The decision whether to repair a drawpoint will depend on the remaining tonnage and grade. In some cases repairs are not justified or production can continue by installing arches inside the drawpoint and using a smaller LHD, say a 3yd instead of a 6yd LHD. If only one drawpoint is affected, it is often best to seal off the drawpoint and draw the surrounding drawpoints in a uniform manner.

COLUMN LOADING

Column loading is a major factor in the failure of production drifts and drawpoints. This situation occurs when :

- Difficult to draw drawpoints are left, and easily flowing drawpoints are pulled at a high rate. The arching stresses supplement the column load resulting in increased' weight'.
- A production drift is closed for a period and the drawpoints on both sides are drawn such that the arching stresses are thrown on to the narrow column.

In these cases the cause must be removed before repairs are attempted

WEDGE FAILURES

Massive wedge failures usually result in major collapses or squeezing conditions such as existed at Cassiar Mine, Havelock Mine and Shabanie Mine. In these cases, where there was a steady downward movement of the wedge, production was maintained by the cyclic replacement of yielding steel arches. Some arches were removed and straightened as many as nine times. Thus in the case of squeezing conditions and if the ore grade warrants it then the support techniques described in the Support section can be used.

However, where there has been a major collapse, with little chance of regards repairs, it would be better to do a reclamation exercise by recovering the ore from a lower level.

SECONDARY BLASTING

Lay on charges used indiscriminately in placement and magnitude can cause unnecessary damage to support necessitating repair of the original support. As it is usual to affect repairs only when the damage is considered significant, at that stage the rock mass would also have been damaged by the blasting.

BROW WEAR / DRAWPOINT LINING

The brow can be damaged during the undercutting period, particularly with conventional undercutting techniques, so that when production starts the wear is rapid and the leading arch support fails. In other cases, the wear back is gradual owing to the attrition from the drawn rock. Drawpoints can tolerate some wear before it is necessary to go in and rebuild the drawpoint. The repair techniques consist of reinforcing the remainder of the major apex and in stabilising the muckpile with grout and shotcrete. Arches are placed in the drawpoint under spiles. If there is sufficient space below the spile then a reinforced concrete layer can be placed between the arch and the spiles. Totally collapsed drawpoints have been rebuilt in this manner and gone on to produce the tonnage call.

BULL NOSE / CAMEL BACK /JUNCTION

The start of failure in the bull nose, camel back and junction can be observed by cracking of the shotcrete. Simple visual monitoring of the crack growth can be observed by putting a filler in the crack. The pillars and the junction back can be strengthened by installing more ropes and cables. Badly damaged corners have been rebuilt with reinforced concrete. There is no excuse, however to let them deteriorate to that extent.

DESIGN TOPIC

Rockmass Response to Block Caving

INTRODUCTION

The response of the immediate surrounding rock mass to block caving is referred to in several of the previous sections as:

- Cavability
- Air Blasts
- Rockbursts
- Massive wedge failures
- Induced stresses
- Subsidence and cave angles

However, the influence of changes in size of excavation, the influence of several distinct caving operations and the geometry of the excavations on the remote surrounding rock mass have not been considered. It is also worthwhile in this section to summarise the response in the immediate surrounds and to predict response in subsequent operations. There are cases when the effect of greater depths have not been recognised in designing the next lift and serious problems have occurred.

SIZE OF MINING AREA

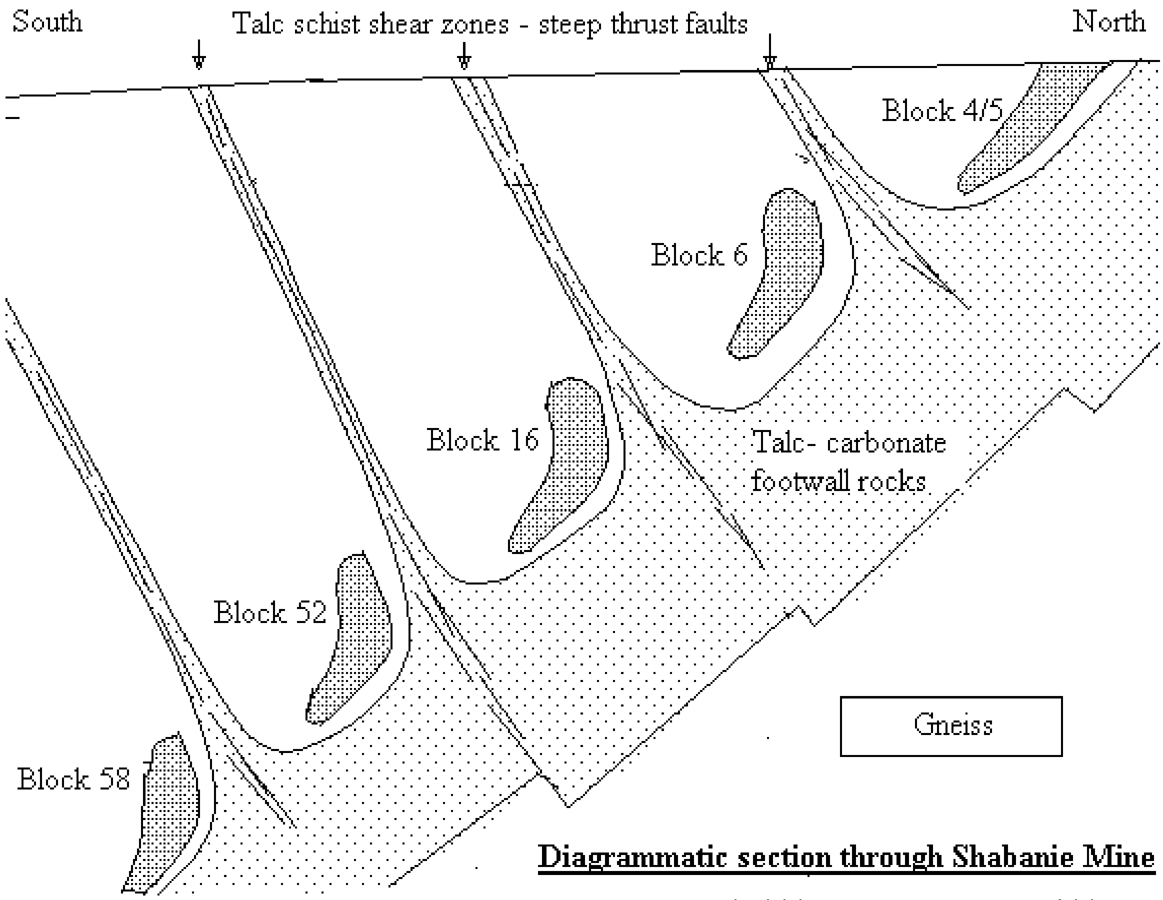
It is often the case that problems only occur once the cave area exceeds a certain size. At Havelock Mine, Swaziland, rock mass response became significant once the strike length exceeded 120m. Rock mass response was noted by movement in shear zones some 200m from the operation.

EFFECT OF DEPTH/ STRESS

The effect of depth was noticeable at Shabanie Mine where a whole series of orebodies ranging in size from 1 to 10 million tons are separated, down dip, by major talc shear zones and along strike by east dipping dykes. In this complex mining environment the MRMR ranged from 10 to 70. The effect of depth on caving was to go from gravity failure to clamping to shear failure. As the mining depth increased, so the induced stresses increased and the aura of failure and movement along major structures increased.

Block 6 was mined in 1964 and caved with a hydraulic radius of 25. This was the start of the rock mechanics program and the progress of the cave was monitored with boreholes and lead to the stress caving concept. Block 6 was near surface and the up-dip blocks 4/5 had been mined out, so there was no dip confinement, but there was strike confinement. Thus there was a stress difference in the back. When it came to mining block 16 the same caving parameters were used as the geology was identical. The importance of increased horizontal stresses and clamping was not appreciated. When block 16 had a hydraulic radius of 30, caving had not occurred. Mining was stopped and caving only occurred in block 16 when the adjacent block 7 was mined and the horizontal clamping stresses were removed.

In view of the block 16 experience, Block 52 was planned as a series of large open stopes with pillar recovery. The geology was still the same, however, the stresses had increased significantly and caving occurred owing to shear failure along major weak structures.



Diagrammatic section through Shabanie Mine

This was an excellent learning curve and taught one that projecting upper level experience can be very dangerous

In other operations the effect of depth has not been that significant - as the case with Henderson mine where the first production level was at 8100` and the next on 7700` level an elevation difference of only 400 feet or 123m in mountainous terrain with an elevation range of 1285m to 584m.   The bulk of the 7700 level is overcut by the 8100 layout.

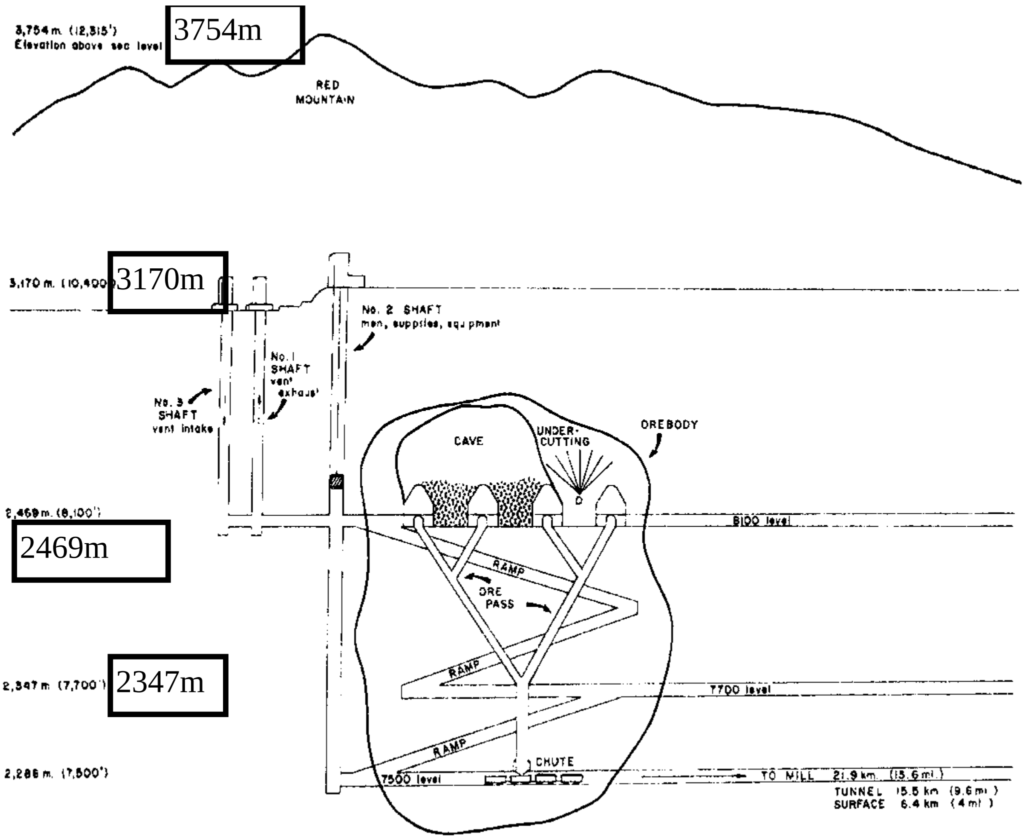


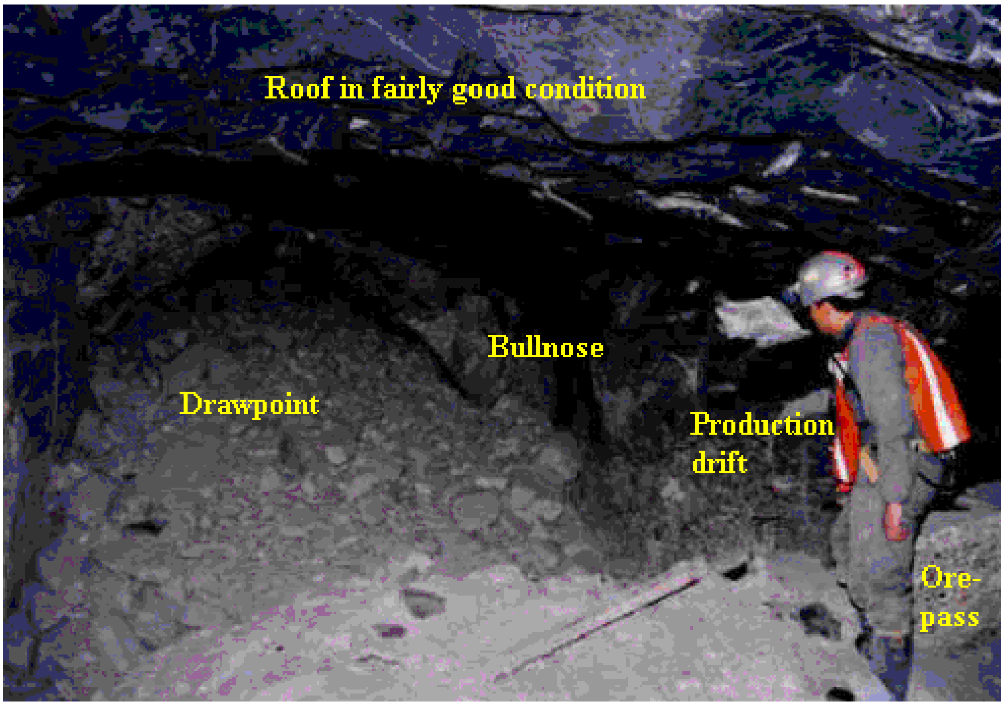
Figure 2. Generalized section, Henderson Mine.

USE OF MAJOR STRUCTURES TO MONITOR RESPONSE

At Shabanie Mine the minor movements on a fault in the footwall of Block 7 were monitored using a dial gauge. A significant factor was that it was possible to record movement in response to mining activity up to 300m away.   It is always advisable to set up these simple monitoring devices to obtain a better understanding of rock mass behaviour.

COLLAPSE AREAS

The following photograph is of a production level with a drawpoint where the sidewall has failed, but, the back is still in good shape. The opening is 1.8m high compared with the original 4m.   An area of some 40 drawpoints was affected after only 15% of the tonnage had been drawn.   This tonnage was later recovered by reclamation from a drawpoint layout developed on the original ventilation level.



Collapsed area with sidewall failure

DESIGN TOPIC

Subsidence

GENERAL

A sound assessment of the cave angles / subsidence in the development of the cave zone is very important in determining the final size of the crater, the extent of the failure zone in terms of location of shafts, declines, orepasses etc. and the environmental aspects. A chart has been developed to give cave/subsidence angles for different MRMR's. The cave angles must be calculated for different depths. Chimney caves do not conform to this pattern. This section summarises many factors that have been discussed in detail in previous sections.

REGIONAL / INDUCED STRESSES

The orientation and magnitude of the regional stresses will determine whether joints are clamped, in tension or shear. The regional stresses also have a major influence on behaviour of the crater wall.

ROCKMASS STRENGTH - IRMR / MRMR

The MRMR is derived from the RMR and can vary depending on whether it is on the minimum or maximum span. If the crater has a circular shape then 'hoop' stresses will play a part. The MRMR must be calculated at vertical intervals to show changes in MRMR.

GEOMETRY

The geometry of the mining area will determine the shape of the crater and the response of the rockmass on the different sides. A narrow zone will have high arching stresses acting on the sides and a steeper cave angle.

DEPTH OF MINING

The depth of mining will have a significant bearing on the cave angle. It will also affect the induced stresses.

HEIGHT OF ORE COLUMN AND CAVE COLUMN

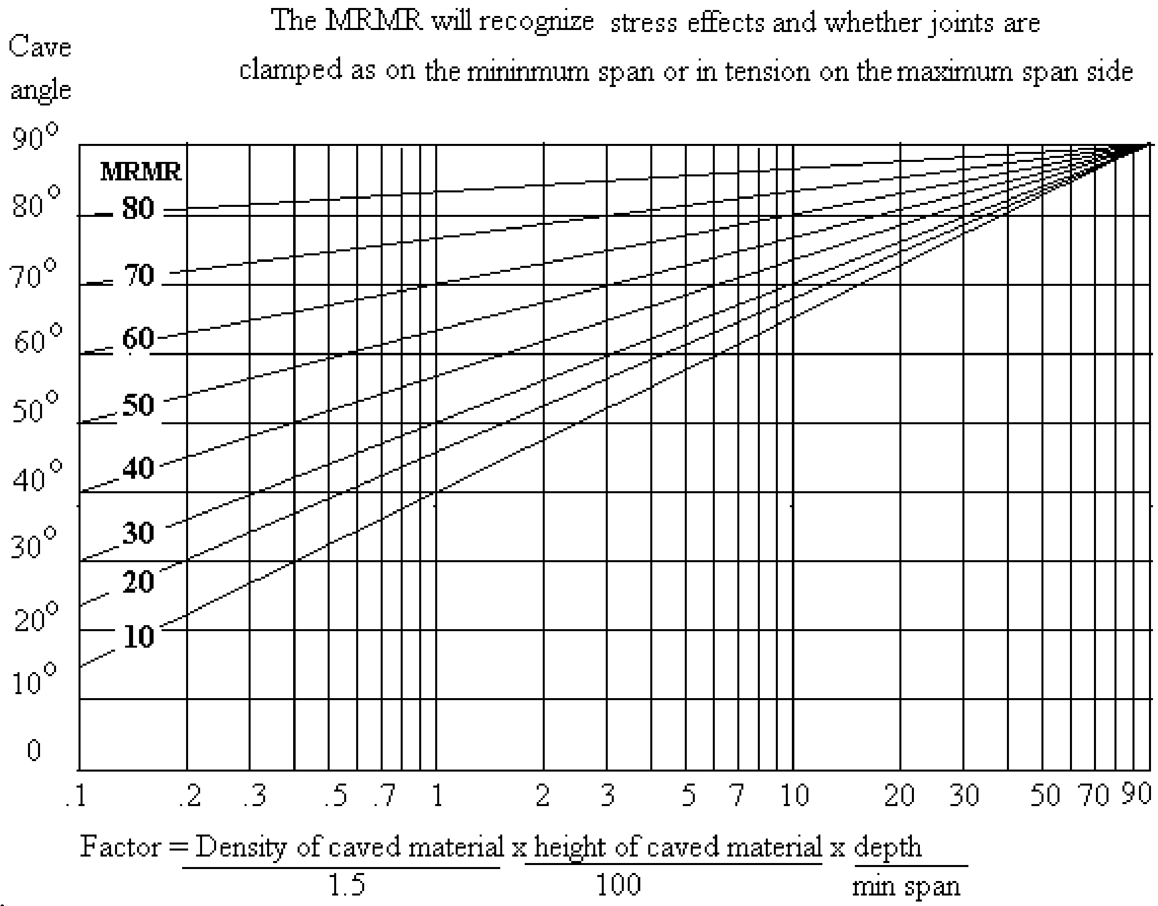
Where the ore column is high, and the overlying waste low, the cave angle will decrease with drawdown as the sides of the crater are exposed.

TOPOGRAPHY

Where the cave comes to surface on the side of a hill / mountain, toppling of the upper slopes often occurs. The same thing will happen when the cave breaks through into a large pit.

CHARTS TO DETERMINE CAVE ANGLES

The cave angle defines the boundary or plane of active movement with drawdown. Cave angles are a function of rock mass strength as reflected by the MRMR, density of the caved material, height of the caved material and width of the cave zone in terms of arching stresses and restraint on the cave boundary. Major structures can cause the cave angle to steepen or flatten depending on their dip. There are examples of overhangs forming below prominent shear zones.



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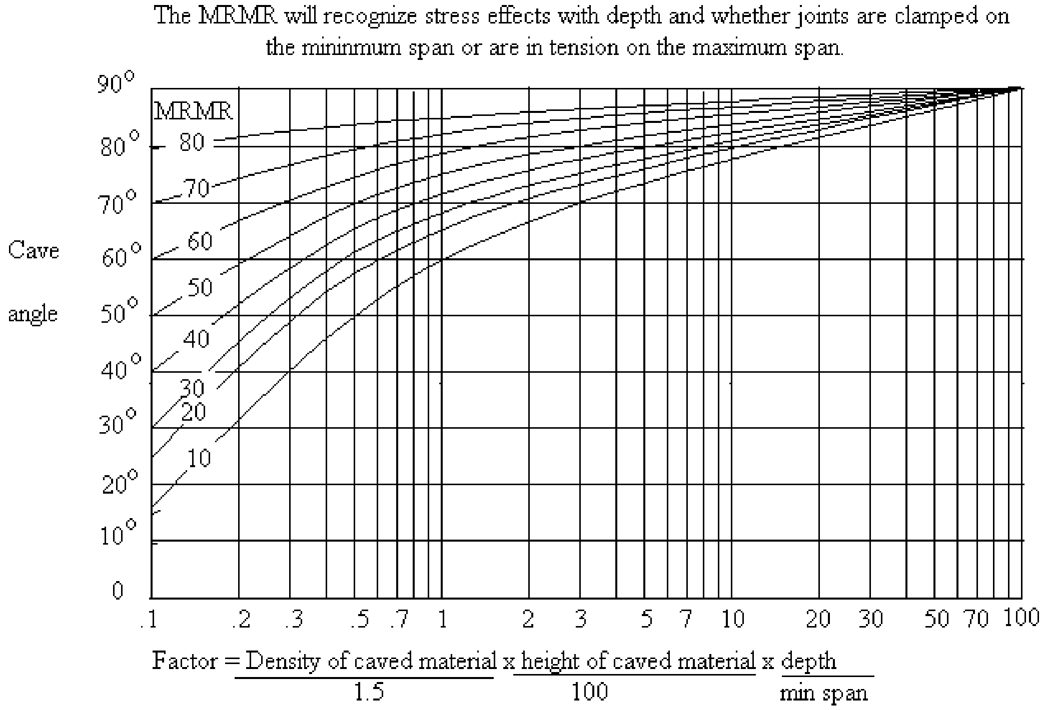
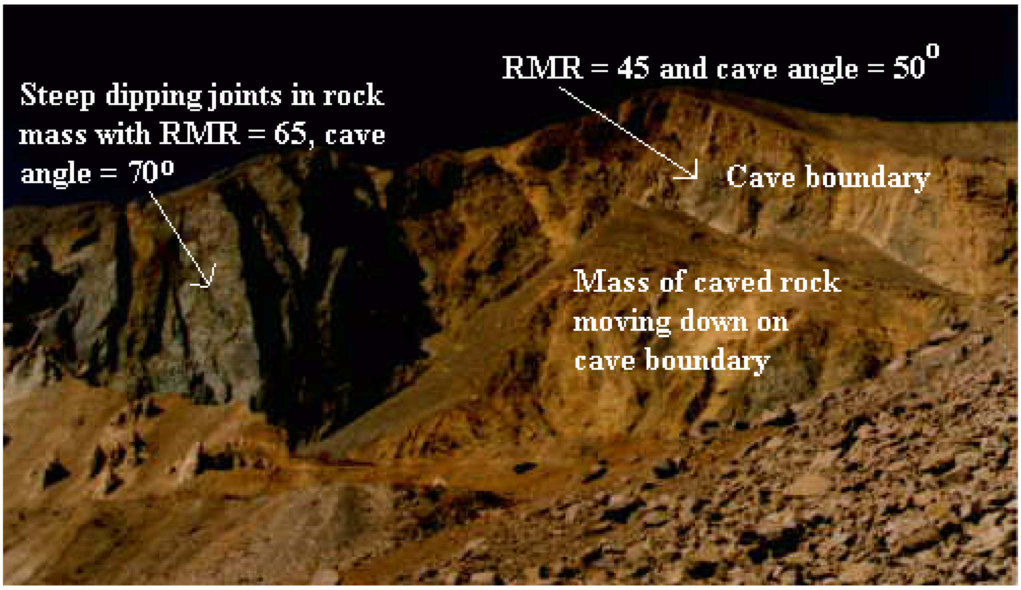
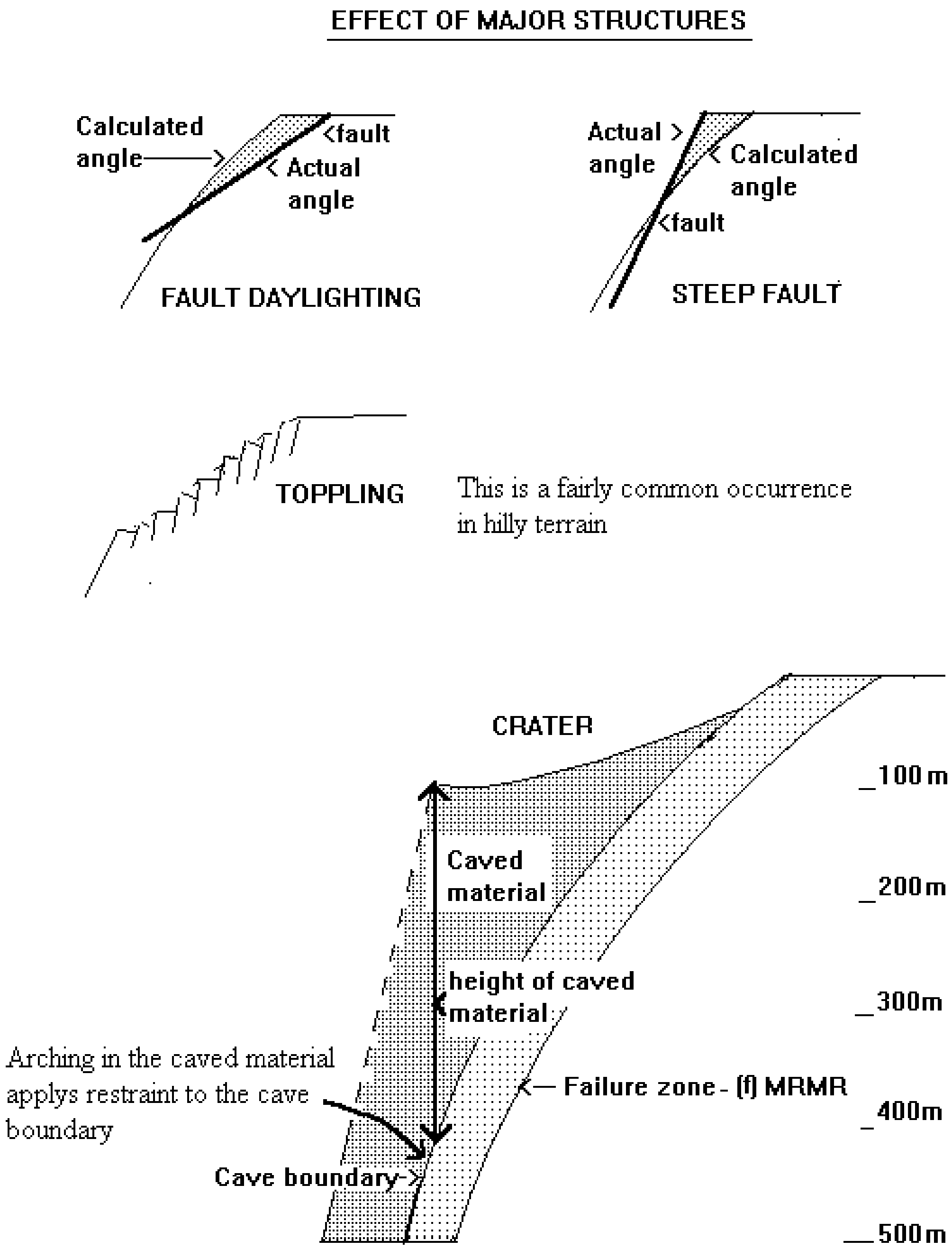
The above diagram is a conservative approach and should be used for siting important infrastructure such as shafts or plant.



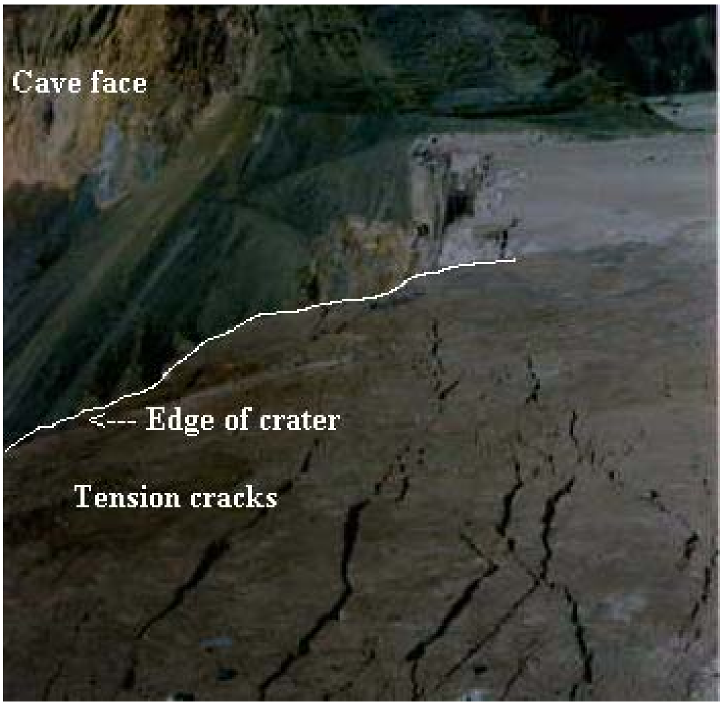
The above diagram is less conservative and would be used for draw control and water inflow calculations.





The failure zone is function of the MRMR. Ratings and zone widths are :-
0 - 10 = 100m, 11 -20 = 70m, 21 - 30 = 50m, 31 - 40 = 40m, 41 - 50 = 30m,
51 - 60m = 20m, + 61 = 10m. Figures based on empirical data.

The failure zone is a zone with limited displacements adjacent to the cave. Examples of failure zones against a crater are shown in the following photograph, where the edge of the crater is shown and the tension cracks that extend for some 50m backwards :



FAILURE ZONE AGAINST A CRATER

EFFECT OF / ON MAJOR UNDERGROUND INSTALLATIONS

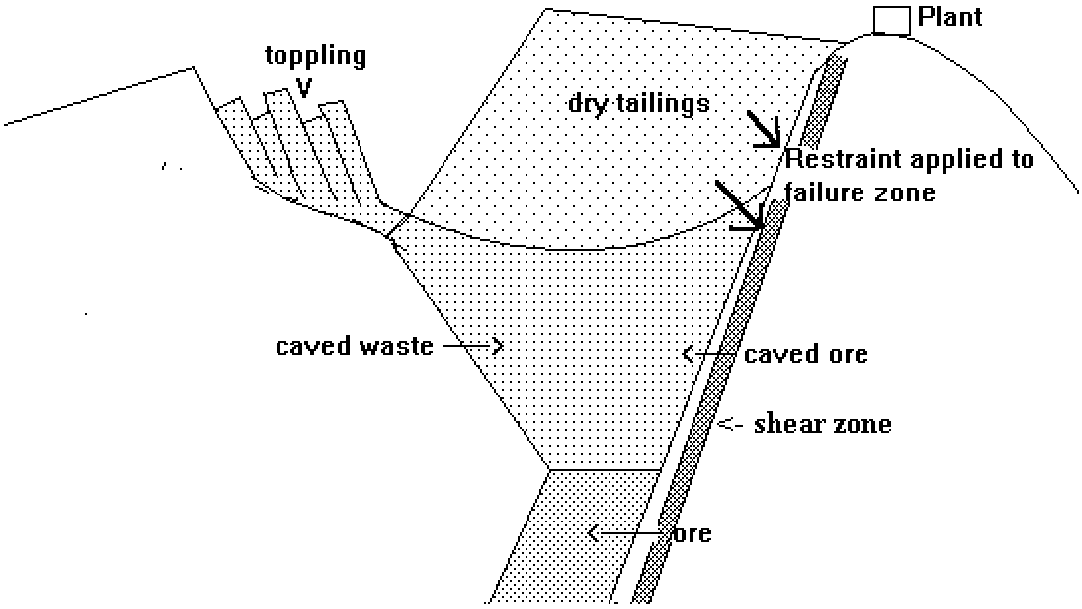
Large excavations such as crusher installations could influence the cave angle if the proximity of the calculated cave angle was such that the intervening pillar could become more highly stressed and fail. This would tend to flatten the cave angle, increase the extent of the failure zone and have an adverse effect on the major installation.

EFFECT ON SURFACE INSTALLATIONS

A conservative approach is required in calculating the cave angle and the extent of the failure zone when it comes to siting major surface installations such as shafts or plant. If sound geological information is not available, then the approach is often to take a 45° angle from the lowest possible production level.

CONTROLLING SUBSIDENCE

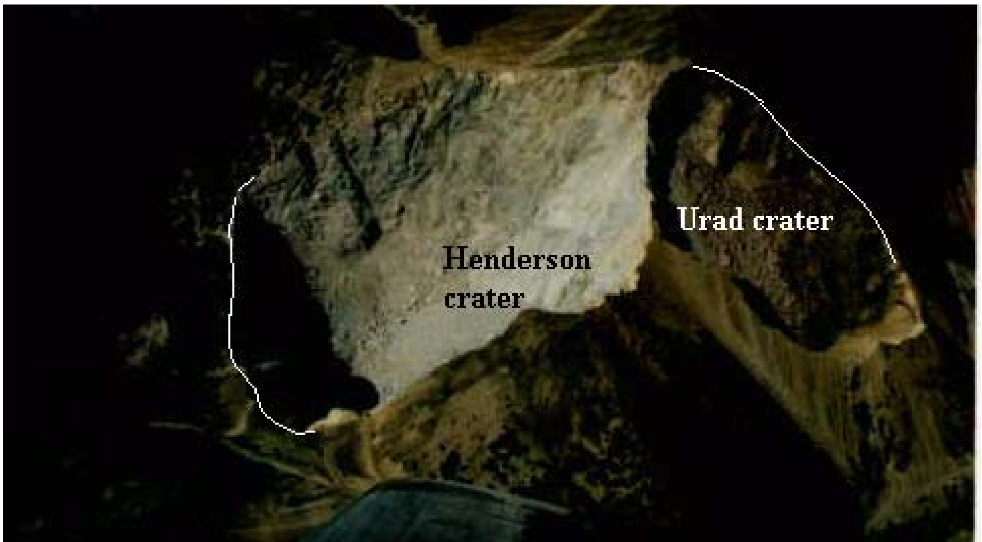
The cave angle and the extent of subsidence can be controlled by dumping fill on the subsiding slope. This was done by dumping coarse dry tailings over the cave at Havelock Mine to protect the plant which was located in the footwall on top of a hill. The confining pressure on the shear zone effectively stabilised the area.



STABILISING THE FOOTWALL BY DUMPING DRY TAILINGS

CRATERS

The Henderson mine and the Urad mine caves intersect at surface



Surface expression of the Henderson and Urad caves

THE ANDINA MINE CRATER

ENVIRONMENTAL ASPECTS

There is often an unwarranted adverse reaction to the creation of a cave crater. In fact cave craters can often be attractive if well developed cliff faces that have formed. In the case of open pit versus block cave, with a block cave the overlying sulphide bearing waste rock is contained in the crater and not dumped on surface. In large open pits, waste is dumped on surface where it could create an acid water problem, which could large areas and entire drainage systems. Low grade zones in hangingwall of an orebody can and have been successfully leached under controlled conditions.

DESIGN TOPIC

Mining Costs / Productivity

MINING COSTS

Mining costs, whether capital or working, are a function of:-

- layout
- development - access, haulages, production level, undercut level, orepasses, ventilation drifts and monitoring drifts
- undercutting - cave initiation
- ore extraction
- secondary breaking hangups and oversize
- primary support
- support repair
- .LHD costs
- haulage and hoisting costs
- .ventilation and pumping

The selection of a cave mining method requires extensive investigation as there is very little room for manoeuvre once the development is underway.

What is required is a spread sheet which will show the different operations of a block cave and the cost of these items as well as repair costs if the job is not done properly. For example, an anti-socket drift (inspection drift) on the crest of the major apex in an incline advance undercut will have a cost, but, how significant is this cost if this drift will ensure that the undercutting is successful, with a draw height of 400m. On a 15m spacing the cost of this drift could be ten cents per ton. Often management are very concerned about capital costs, with not enough consideration given to the long term saving in working costs and a resultant overall low cost per ton.

Glen Heslop of Mine Geotechnics in Perth Western Australia, has developed a spreadsheet for the assessment of mine development and operation costs. This is a multi-sheet work book which is designed to enable the user to estimate the cost of various mining layouts, mining methods, draw sequences etc.

Frequently, study managers of feasibility level studies turn to contractors to provide unit cost estimates for development, support etc. While this has the advantage of being based on that contractor's latest actual costs for similar operations, it has three major disadvantages. Firstly, study engineers require to evaluate various mining options before they have sufficient detail to pass on to a mining contractor for cost estimates. When the mining plan and schedules are sufficiently developed, they can be handed to a mining contractor to provide cost estimates. These would be drawn from costs on one of the contractor's recent jobs, usually in a different mining situation, which are then arbitrarily factored up or down to suit the mining situation being studied. Finally, the study manager has no control over the quality of the cost estimates he is given by the contractor.

This spreadsheet has been developed over a 10-year period on several mine costing jobs and has been adapted to several mining methods. It has been used for comparative costing of the development of different drawpoint layouts, of different undercutting methods, different mining methods, different mining sequences, through to life of mine DCF cost analyses. The approach used in the workbook is to synthesise unit costs for each component activity, such as equipment operating unit costs, which are integrated with labour, consumable and supply costs into the cost of the drilling and charging of a blast hole or the cost an installed a rock bolt. Each activity has five cost factors: the total dollar cost per unit, broken into consumable, maintenance and labour costs, and, operator and a maintenance times per unit. These are then assembled into unit costs for different sizes or types of development including any special support or flooring requirements and tramming distances. From a development schedule or a list of development sizes and metrages, these may then assembled into larger units, such as all the development for a drawpoint horizon.

Mine production costs are developed in a similar way. When the study is sufficiently advanced the development and production schedules can be introduced into the workbook and the mining and development costs calculated for input into financial models to calculate the NPV and IRR of the project. Management, supervision, technical services and infrastructure costs are usually estimated in conventional ways and added to the calculated mining costs.

With a simple "use/don't use" flag in each input component, the number of man-hours required for operator or maintenance per period can be calculated. The equipment operating fuel or power requirements, numbers of rock bolts, etc. can also be calculated. These also allow for input cost sensitivity analyses to be undertaken easily.

It draws on the following general data bases:

- Mine consumable costs, usually up-dated from mine warehouse stock and price lists,
- Equipment capital costs, these are updated where necessary from equipment suppliers or, previous prices that are escalated where necessary.
- Equipment performance factors.

In addition the following study specific factors have to be provided by the user:

- Design criteria: hours per shift, mine operating shifts per year,
- Operator and maintenance labour wages, on costs, effective hours per shift and shifts per year.

- Local fuel and power costs.
- Cost escalation and currency conversion factors.

For each level of activity the user can alter the parameters, such as the type and length of the rock-bolt,. the type of drill used to drill the hole, etc. On the next level he can specify inter alia, the numbers of bolt per metre developed for a range of development sizes. On the final level he can specify the metres to be developed in each class of heading, or tonnes to be produced from each stope or block.

PRODUCTIVITY

Statistics on the performance of LHDs and secondary blasting breaking are extremely important and must be related to the different geological environments (MRMR, fragmentation, water, mud etc.). Overall figures do not mean much.

As regards LHD performance :The Utilisation Percentage, Mechanical Availability Percentage and Overall Availability Percentage should be readily available and should be known by the operating personnel.

Secondary blasting figure, expressed as grams per ton - g/t - must be related to the different areas of fragmentation and not an overall figure. It is amazing that very few personnel can quote these statistics.

DESIGN TOPIC

Role of Physical and Numerical Modelling in Designing Block Caving Operations

GENERAL

This section will cover in detail physical and mathematical modelling which have been and might be of assistance in designing block caving mines. Also the shortcomings of modelling techniques must be shown. For example, recently numerical modelling was used in an attempt to determine cavability at Northparkes Mine - this was found to be completely inaccurate as it was mesh dependent. Physical models have been used to explain flow behavior. The important point is how accurate is numerical modelling compared with empirical systems.

PHYSICAL MODELS

Two Dimensional (2-D) Sand Models

2-D sand models have been used for many years and were used to develop initial concepts in block caving and sub level caving A thesis ‘ Development of body motion under controlled flow of bulk solids’ by A Mansson describes a series of 2-D experiments in models which show different flow characteristics for a variety of materials used e.g. sand, magnetite, lead shot and by varying the packing density from compact to loose. These results need to be analyzed and the influence of the limited column height determined At first glance these confirm what is to be expected in practice. Two dimensional models are of no use in block caving analysis and can be used to model sub level caving situations, but even here 3-D results are better.

Three Dimensional (3-D) Sand Models

A three dimensional sand model with 50 drawpoints at a spacing of 108mm with a height of 2400mm and a base of 760mm x 760mm, was developed at Shabanie Mine in the late 1970’s . It was used to determine loading on the base of a cave, the ratio of load distribution between sides and base, and eventually to carry out draw control experiments for block caving and sub level caving methods. To measure the total loads, the base and sides were mounted independently on load cells. The model has been used for investigating a variety of aspects in sub level caving and block caving, such as loads on major apexes, effect of differing drawpoint spacings and draw with inclined footwall drawpoints. The movement of colored sand markers was studied by wetting the model material and cutting horizontal and vertical slices through it. These views were photographed and sketched using a reference grid to reconstruct what happened inside the model during drawdown. The model at the time was the only three dimensional model in use. Experiments were carried out varying the spacing of the drawpoints and are described by Marano(1980) and Heslop & Laubscher(1981).

Solid Models

Solid three dimensional models made out of polystyrene - styrofoam have been used to show the spatial relationship between different orebodies as was done on Shabanie mine, with its numerous orebodies. It was also done on Andina Mine to convey the right impression of the size and shape of the orebody and the relationship of satellite mineralized zones. The outcome of that modelling was to recommend a study of an open pit instead of a block cave. The study confirmed the first impression that an open pit was the right way to go, however management were not prepared to follow this up. Solid models are now replaced by 3-D presentations on a computer screen -not as satisfactory.

Glass / Perspex Sheet Models

By drawing sections on transparent sheets it is possible to obtain a better overall understanding of the orebody.

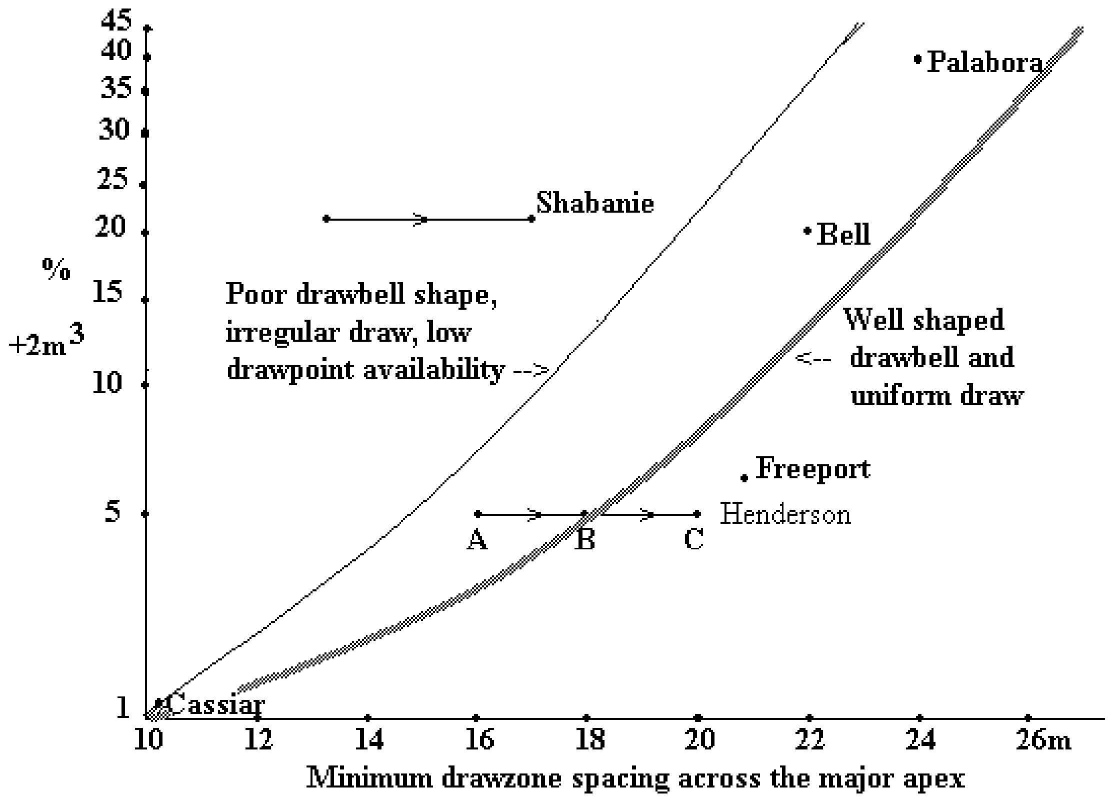
NUMERICAL MODELLING

There is a fair amount of enthusiasm for numerical modelling in the hope that it will be possible to improve on the empirical systems currently in use or to fine tune the empirical systems.

Numerical models are useful tools to provide numbers for calculations and are very dependent on the correct input data. For example, engineering judgment and empirical data at Shabanie Mine showed that the proposed large open stoping method proposed was not likely to succeed. This was clearly demonstrated when modelling was used to obtain numbers from the back analysis of another operation which had problems and that the numbers for the proposed operation were significantly higher. Care must be taken as modelling results can be suspect as was shown when Flac was used at Northparkes to determine cavability and shown to be mesh dependent.

PFC is being proposed as a possible tool to determine cavability and materials flow so as to determine drawpoint spacing. If it is to be used to determine the effect of drawbell shape and drawpoint spacing, then at least twelve drawpoints must be worked, that is, six on either side of the major apex. However, to determine the effect of shutting drawpoints then twenty four drawpoints are required. This demands a large 3-D capability and at this stage PFC is a 2-D model.

If the empirical system for drawpoint spacing is examined we have the following design diagram with plots of different operations. Henderson is an important plot as there are three different spacings. Points that have been ignored in the past have been the influence of drawbell shape, draw and drawpoint availability on the interaction between drawpoints, modelling must be able to include these.



In the case of Henderson Mine, the spacing has been increased significantly. The question is how is PFC going to determine the spacing at Henderson? Examples of modelling predicting the answers to dynamic situations are required.

Dilution entry is another case in point. The empirical approach is to recognize all the factors that influence dilution entry. Where this system has been used it has been found to be accurate. It can be seen from the dilution entry graphs that there is virtually no difference if the entry point is ± 5% i.e. at 25% to 30% to 35%.

Cavability has caused much concern where the orebody is small and no concern where the orebodies are large, provided the production area is greater than the caving hydraulic radius. It has been stated that even where the orebody is large it is important to know the exact hydraulic radius, so as to plan production. This is rather far fetched, as the rate of caving is also a prime factor. Also, how can the mining of the first few metres be important when columns of +200m are considered?. In high stress areas it has been found that the rate of caving must be controlled by a low draw so as to reduce the incidence of seismic events.

DESIGN TOPIC

Mine Planning, Planning Schedules and Stope Dossiers

GENERAL

Gone are the days when block cave mines were located in secondary zones of finely fragmented ore and one deposit was broadly similar to the other. In fact, many of those deposits that were block caved in the pre 1970's if exploited today would be mined by open cast methods.

Generally the current and planned block cave operations are at depth below open pits or previously caved areas or have some other unique characteristic that precludes mining by another method and the low cost, high productivity block cave method is selected.. This uniqueness of the orebody to be mined means that there are no direct comparisons that can be drawn on how to plan the operation, even in extensions with depth it is a case of extrapolation with recognition of the higher stress environment as a result not only of depth, but also the increasing size of the caved zone. Comprehensive monitoring of rock mass response in previous mining areas is the start point for sound predictions on possible rock mass behaviour in the new areas. Another complicating factor that comes into the equation is the desire to reduce mining costs by increasing production from larger mining areas and the use of larger equipment to hopefully reduce operating costs.

Systems that worked in the past might no longer work and modifications are required. For example, long orepasses to a major haulage level operated for many years on a large block cave mine. However, as mining progressed with depth, the induced stresses below the cave increased and these orepasses now started to fail owing to stress spalling. This obviously was not predicted and the mine had to examine other ore handling methods. Thus, whilst extrapolations are done with the best knowledge based on previous experience, no doubt backed by numerical modelling, the mining of the next block will be a unique experience and part of the learning curve. Adjustments / modifications must always be considered, that is mine management must be fully aware of the situation and respond in an appropriate manner. The experience at Shabanie Mine with projecting data down dip showed that this could present a planning problem (see Section 33 - Rock mass response).

Production schedules for 1 year, 5 years and depending on the life of the deposit a long term figure of 10, 15, 20 or 25 years is required so that geological, rock mechanics and mine planning efforts can be geared to providing the information required for the mine to operate in that time frame.

Successful mine planing depends on adherence to an agreed schedule based on the capabilities of the organisation. A schedule can be prepared to show the role of various departments and the sequence of events leading to the underground exploitation of massive orebodies. Experience has shown that production problems occur when there is a departure from these procedures and expediency is allowed to over ride previous decisions. The timing of the schedule is based on the commissioning of mining blocks in a complex geological environment in the correct production sequence. The period between exploration and production can be reduced if the geology is simple / clearly understood, or, more effort is put into obtaining the data and in increasing development rates. Whilst some shortening of the process may be achievable, allowance must be made for unexpected ground conditions or groundwater.

TECHNICAL FACTORS REQUIRED IN MINE PLANNING

The technical factors required in mine planning can be listed as:

- Geological investigations.
- Geomechanics rock mass classification of the orebody and peripheral rock mass
- Rock mass strength
- Orebody shape, dimensions, dip and depth
- Ratio of the ore/unpay interface to the contained ore
- Regional stresses
- Mineral and value distribution in the orebody and potential dilution zones
- Rock mechanics investigations
- Cavability of the orebody and hangingwall
- Muckpile fragmentation data
- Draw and grade analysis
- Rock mass stability and surrounding rock mass response
- Geographical and environmental considerations
- Location and strength of extraction horizons
- Mining sequence
- Induced stresses
- Numerical modelling of proposed layout and sequence
- Support requirements
- Production tempo
- Planning schedule
- Ore handling, ventilation and access

Check lists should be developed by the geology, rock mechanics and planning departments to ensure that all items are investigated. It is important that geology and rock mechanics involvement is maintained through planning and into production.

Ongoing rock mechanics monitoring of the operation is essential should it be necessary to adjust or modify the mine plan as production proceeds. For example, the planned rate might have been a draw down of 200mm per day, but seismic events in the cave back would mean a reduction to 100 per day until the cave was complete.

PLANNING SCHEDULE

The following table shows the planning schedule successfully used at Shabanie Mine, Zimbabwe:

BLOCK PLANNING SCHEDULE

MINE: OREBODY: BLOCK(S): DATE:

MAIN DEPARTMENTAL RESPONSIBILITIES	19	19	19	19	19	19	19	19	19	19	19	19
	WORK APPLICABLE TO AN OREBODY OR MAJOR SUB-DIVISION OF LARGE OREBODY					WORK APPLICABLE TO SPECIFIC BLOCKS						
	Widely Spaced Drilling	Closely Spaced Drilling				Specific Valuation, Layout and Rock Mechanics Drilling						CRASHING — RATE OF PRODUCTION RELATED TO MINING METHOD WITH ALLOWANCE MADE FOR CAVE INITIATION — FULL PRODUCTION IN THE FOLLOWING YEAR
GEOLOGY	Exploratory Outline				Planning Outline & Classification					Final Outline & Classification		
GEOLOGY		General Orebody Specifications for sequence			Preliminary Orebody Specifications		Intermediate Orebody Specifications				Final Orebody Specifications	
GEOLOGY	Exploratory Development Layout				Define Ore Reserve Blocks		Cube Valuation			Cube Valuation Completed		
ROCK MECHANICS		General Rock Mechanics Assessment for sequence			Preliminary Rock Mechanics Assessment		Intermediate Rock Mechanics Assessment	Rock Mechanics Monitoring Programme Design	Install		Final Rock Mechanics Assessment	
PLANNING	Access for Exploratory Development		Overall Mining Sequence			Investigate Mining Methods	Decide on Mining Method	Define Mining Blocks			Mining Programme	
PLANNING	Exploratory Development Layout	Investigate Shafts, Compressors, Pumps, etc.		Primary Access Layout	Primary Production Layout	Ensure Sequence Correct	Stope Access and Ventilation Layouts	Design Support Systems	Review Production Tramming and Loading Equipment		Draw Control Programme	
PLANNING								Stope Production Layout			Drilling Dossier	
PRODUCTION	Access for Exploratory Development	Exploratory Development			Primary Access Development	Primary Production Development	Stope Access Development	Orepass System Complete	Stope Production Development with concurrent support			
PRODUCTION												
ENGINEERING		Assess Shafts, Pumps, Compressors, etc.		Assess Hoists, Fans, Rolling Stock	Commission Loading Pocket Major Shaft	Design Electrical Reticulation		Commission Hoist in Serviceway	Review Production Tramming and Loading Equipment etc.			
ENGINEERING					Design Workshops Loco Bays etc.							

FIGURE 8

STOPE DOSSIERS

Stope dossiers list all the items used in mine planning and are updated periodically on how the mining operation is fairing with respect to these items. Reference to the stope dossiers is done to update the design and planning procedures, that is theory versus practice. Contribution by N.J.W.BELL:

In order to strengthen the planning procedures for each block a Stope Dossier is established. This Stope Dossier, is a chronological collection of all information available on the planning and mining of the block, with a selective compilation of information filed under chosen headings.

It is emphasised that only pertinent information concerning the planning and exploitation of the block is to be retained in the Dossier and this under selected titles to maintain uniformity, and to allow ready reference.

The object of the Dossier is to be an aid to production planning by being a ready reference for information obtained and decisions made on the block being planned and a ready reference for techniques, efficiencies and costs for the planning of future blocks.

The retention of the required information falls into three categories:

1. The Planning - the section in which the Geology, the Rock Mechanics and the Planning Departments record the major work done and the decisions taken toward the exploitation of the block.

2. The Production - the section in which the above three departments together with the Production Department responsible, describe the production period with particular reference toward the planning and improvements.

3. The Reconciliation - the section in which each department reconciles the results of the production to the planning.

In order to relate the flow of planning material to the production date a block schedule will be included in the Dossier. This will serve as a memory aid to show the timing for the flow of required information to meet the required production date.

The Stope Dossier system should also be considered for major projects and exercises as well, apart from the stope planning.

UPDATE - It is suggested that the Dossier is updated every 6 months throughout the life of the block – this is in order to keep worthless information to a minimum but not to tax the memory of the compiler or contributor excessively.

The Dossier should be circulated to the departments once each half year but generally be stored in the Planning Department, where it has most application.

LAYOUT:

Major headings

1. The information will be classified under the following major headings:

   Geology
   Rock Mechanics
   Planning
   Production
   Reconciliation

2. Sub – Headings

   Emphasis on various points under the sub-headings may change with different blocks' peculiarities and it would be difficult to list every item. Notes however have been prepared on aspects that could be of interest for record under each sub-heading.

a. GEOLOGY

   i. Exploratory Development

   Brief description of general layout, amount of development, required rate of development, extra programs required, problems encountered, etc.

   ii. Geological Drilling

   Dates of the various programs, number of sites, difficulties etc.

   iii. Orebody Specifications

   The specifications are to be filed, as they become available.

   iv. Grade control

   Consideration is to be given to how grade was being controlled during production, if necessary.

   v. Memoranda

   Any pertinent memoranda should be filed here.

b. ROCK MECHANICS

   i. Rock Mechanics Assessments

   These are to be filed when available.

   ii. Monitoring Programme

Report on the proposed program and the installation of instruments. All modifications and additions to the program to be included here.

iii. Significant Observations

A progressive account of any significant observations.

iv. Ground Behaviour Interpretations

A progressive interpretation of significant observations.

v. Specific Investigations and Recommendations

All reports arising from. Observations within the block, with regard to ground behaviour, improvements to excavations and support.

c. PLANNING

i. Check Lists

A checklist has been compiled to aid in discussion. It is a checklist of material available and it will be updated.

ii. Considered Mining Methods

All methods considered, the reasons why, the reasons for discards will be filed here.

iii. Mining Layout

The mining layout for the selected method together with reasons for decisions will be described.

iv. Sequence of Mining

Sequence with regard to other blocks, direction of retreat etc. to be described

v. Draw Control

Method of draw control and reasons.

vi. Ventilation

Ventilation: layout, requirements, etc.

vii. Support Programme

Type of support required, density of support etc. (in conjunction with Rock Mechanics and Geology Departments).

viii. Equipment and Labor Requirements

Schedules and forecasts of requirements and review of availability.

d. PRODUCTION

i. Support behaviour
ii. Draw Control
iii. Mining techniques
iv. Extraction problems
v. Costs
vi. Efficiencies
vii. Recommendations

The above sub-headings are all self-explanatory and they should be written from the production angle. They will obviously involve all departments during some stages.
Costs and efficiencies should be recorded selectively and with the thought that the working costs of the block could be reconstructed at a later date, for comparative purposes with later blocks' costs.

e. SCHEDULE

A block schedule laying out the timing of the flow of progress to production will also be included in the Dossier. It can be stuck in the inside front cover and coloured in for convenience to note the progress of the information received, with a colour for each year, this will indicate any lags in the planning. (It has been found that a sepia with dates of completion inked in is more useful when planned production dates change owing to; pay limit changes or varying production rates)

f. RECONCILIATION

To investigate the inter-relationships between geological, rock mechanics, planning and production considerations in all aspects of the mining of a block.

This section should not contain descriptive matter, which rightfully belongs to other sections of the Dossier; it should be used to comment on the effectiveness of various decisions taken, the methods and techniques adopted and to analyze their effects on other aspects of the mining of the block. The effect of ground behaviour on the methods used should also be analyzed here.

The following is a list of suggested headings for this section:

i. <u>Exploration</u>

a) Valuation
b) Orebody Specifications
c) Grade control

II. <u>Rock Mechanics Assessment</u>

a) Monitoring Programme
b) Effects of ground behaviour on mining
c) Specific investigations

iii. <u>Check list comments</u>

a) Mining Methods
b) Layouts
c) Sequence and Techniques
d) Draw control, Extraction and Dilution
e) Ventilation
f) Support
g) Equipment and Labour
h) Costs and efficiencies

iv. <u>Schedule</u>

v. <u>Specific Recommendations</u>

LEACHING

If the hangingwall zone of the deposits has large zones of low grade mineralization that is amenable to leaching this must be recognised in the overall mine planning so that management can make allowance for the capital expenditure. This obviously extends the life of the resource. In very large deposits, the leaching has been done concurrently with other mining activities.

COMMENTS FROM N.J.W.BELL ON ROLE OF MANAGEMENT

Policies
Understanding of RMR and MRMR
Support ***** and Construction
Undercutting *****
Production Rates
Draw Control *****

* Probably the three most important elements to ensure success after design and commitment to a case mining option

POLICIES

Management must at the outset of the project lay down a set of policies that it will enforce throughout the life of mine.

A philosophy of **'Do It Right - Always'** is the **Aim**
The following are again some pointers that management should consider ensuring (if desired results are to be achieved):

The planning of production tonne
Under cutting philosophy
Support construction standards including timing
Suitable number and type of equipment in good well maintained order
Secondary blasting philosophy
Draw Control

UNDERSTANDING OF RMR AND MRMR

The personnel involved soon get to recognise an *insitu* class of rock RMR and when it the same as that shown on their plans they develop a confidence in the whole system and design. It is then easier to get the proper quality support installed. Particularly in better quality rock, which is going to be subjected to damage hence requires apparently too much support when first exposed in the development.

There is a tendency to use average RMR or MRMR values. This should be discouraged, as the range may be significant and zones may be overlooked. This results in misleading predictions and/or incorrect support designs e.g. block Main/5 King average 3B, but has 15% each of Class 5 and Class 2 both section can give difficulties especially if ignored and regarded all as 3B.

The use of whole classes is easy and does not clutter plans and sections with too much detail. However from experience at African Associated Mines class 3 should be split into 3A and 3B, as the 3A rock is definitely more reinforcable.

SUPPORT AND CONSTRUCTION

A very critical area affecting the success of a caving project

Initial Support

1. The minimum support in Class 4 or 5 ground is spiling bolts with every round reinforced shotcrete or other lining with rockbolts and straps post-lining.

   In class 4 or 5 ground this is likely to have to be right up to the face with every blast. Must be right to the face when a holing is about to be effected, and right up to the face within 1 week of a face being stopped, and across the holing point within 1 week of the holing being effected.

2. The minimum support in Class 3 B ground is spiling bolts with every round reinforced shotcrete or other lining with rockbolts post-lining.

3. The minimum support in Class 3A or better ground is rockbolts. (patterned to random across feature according to local conditions)

5. As apexes are vulnerable **All** development take-offs in **all classes** of ground to be pre-supported **Before** the take off development commences and **After** the first round of the new heading in accordance with a laid down standard.

Secondary Support

Is that support installed to prevent deformation of the tunnel and to support it until the production phase of support is installed.

1. This support should be installed:

   i) To within 15 metres of the advancing face, or

   ii) To within 5 metres of the face when a holing is about to be effected, or

   iii) To within 1 metre of the face within 3 weeks of the face being stopped, or

   iv) Across the holing point within 3 weeks of the holing being effected.

2. The additional minimum support for Class 3 B, 4 and 5 rock is the support of the wide spans/junctions and such additional rockbolts and cable bolts that are prescribed.

Production Support and Construction

Is that support required for the productive duty of the end and shall include **special cases**.

At least six months before production commences, this support shall be completed to within a radius of 40 m to 80m of the initial area to be crashed except that to be erected in the drawpoint after the advanced undercut. The production support shall be kept ahead of the expanding crashed area to the same advancing perimeters.

Special Cases

Where conditions differ from that predicted and planned are found, particularly if worse ground is intersected or major 25% over break or fall out occurs a special case is deemed to exist and the following is a possible procedure.

<u>Either Production or Design Personnel can initiate these.</u>

1. An underground visit to the area shall be arranged for an on-site inspection - by all personnel involved ensuring that all the relevant geological and geomechanics mapping for the area in question is to hand.

2. A critical assessment of the mapping information and the data gleaned in 1 shall be prepared.

3. Agreement shall be reached on what action should be taken and what action can be taken. Where these are different management shall decide the course to be followed and authorize its implementation.

4. The additional support shall be implemented as agreed.

5. The "Special Case" support recommendation shall be prepared and circulated.

6. The area shall be monitored until the problem is resolved.

UNDERCUTTING

With out a doubt this is the most important activity with long term consequences and is an area that is often sadly neglected until too late.

The undercut must be fully broken.

The following are some pointers that management should observe.

<u>General</u>

The effective undercutting of a production block is critical to production, efficiency and ensuring that the planned production tonne are extracted as scheduled with minimum dilution. Failure to undercut a block properly results in the following:

1. Poor caving
2. Extreme mining stresses resulting in damage or failure of the support systems.
3. Loss of production.
4. Early dilution and poor grade recovery.
5. Inability to meet production schedules.

Therefore, for all undercutting operations, no matter what cave mining method is employed, the laid out standards and procedures must be adhered to. The following are suggestions that might be considered:

1. Undercuts shall be sufficiently supported to ensure that mining proceeds smoothly and timeously without interruption.

2. Suitable number and type of props shall be stored on the undercut level to deal with any such situation that demands temporary support urgently.

3. The area shall be fully serviced, the development services intact.

4. The area shall be cleared of all extraneous material and equipment.

5. To allow for the efficient lashing, if required (it might not be part of the planing but if required will have to be done urgently), of the daily blasts the tracks, roadways and tips shall be in good order, left fully operational from the development phase.

6. A standard should be set up as to the continuity of blasting of the under cut over public holidays/weekends etc. (Regular blasting reduces the stress build up and reduces the chance of ‘seismic’ events.)

7. The area that needs to be under cut to meet the ongoing production requirements should to be established together with the face shapes and monthly aim lines which should be marked on the master plans for guidance.

8. All areas being undercut shall have two sets of overlay plans properly superimposed and kept to within 24 hours up to date by the blasting supervisor of the area.

9. One set underground the other on surface. They shall be available for inspection at all times. With particular attention being paid to sequencing to prevent any premature undercutting of the levels above when commissioning the lower level cones or overlapping undercuts.

10. Areas being undercut shall have a suitable drilling rig/s (King rig, Jackhammers Etc.) with all equipment available on the undercut level to assist in any drilling required to ensure continuity of the undercut areas when required.

11. Original production drilling shall be to a high standard.

12. Charging of the undercut shall be under the control of the blasting supervisor who shall ensure that continuity is maintained and that the blasting advance is as planned. Each and every hole drilled for undercut breaking shall be checked for length and angle before blasting. No hole shall be charged and blasted that is not open to the correct depth unless the controller is satisfied that the blasting of the particular hole will not affect the break of the whole pattern.

13. All areas of doubtful breaks to be recorded and communicated with conformation when cleared.

14. The blasted rock shall normally only be lashed;
15. To assertion continuity to adjacent crosscut/s or cones and that the last blast was to height and depth or
16. To clear the way for additional development, drilling, blasting or recharging off partly broken rings to correct the situation if the above is not achieved.

17. Careful, full and continuous monitoring by all operations staff of the undercut operation is critical and any adverse or unusual conditions arising shall be reported immediately in order that corrective action can be taken immediately.

18. Should an extremely large span be opened up with no visible sign of caving, ring drilling drives or crosscuts shall be kept full of spoil that in the event of a sudden cave, personnel are not injured by an air blast. To do this a ring is blasted and not lashed at all if the end has reached stopping position no more rings left then an additional rings should be drilled and blasted, (NOT lashed) or the 'King Rig' used to blast additional spoil off the back of the undercut area. Alternatively suitable plugs (properly hitched, pinned and robustly constructed) can be installed.

PRODUCTION RATES

The planned production rate must take into account the requirements for the caving block so as not to force the production when good cave mining practice dictates that it must be slower in order to reduce the possibilities/effects of;

Chimney Caves
Empty Drawpoints/Air Blasts
Seismic Activity
Poor Draw Control

DRAW CONTROL

Without a doubt this is a most important activity with long term consequences and again is an area that is often sadly neglected until too late.

DESIGN TOPIC

Environmental Issues

GENERAL

Mismanaged environmental issues have the potential to close the mine down or be extremely expensive to rectify. Increasingly, environmental protection authorities have the powers through the courts to close projects down. Smithen (1999) has pointed out that environmental management costs are reaching levels that are becoming material or potentially material to the financial assessment of mining operations**.** These are frequently underestimated as they focus on the easily estimated demolition, rehabilitation and related costs issues or the annual contributions to a rehabilitation or closure trust fund, while on-going environmental management and capital costs receive less attention.

The environmental risks in block caving mines are more than for cut and fill operations, but less than for open pits. To a certain extent they differ in emphasis and the time scale. Block caving mines could have substantially longer lives, but when compared with open pit mines of similar production capacity, a block cave mine produces very much less waste rock; less land is required for waste dumps and so it requires less rehabilitation.

Where the waste has an acid producing potential this could be a substantial advantage. The waste rock from the hangingwall of a cave is contained within the crater so there is no chance of an acid water problem developing unless water is allowed to discharge from the mine. In fact, in a feasibility study on a sulphide deposit the recommendation was to block cave rather than open pit and one of the reasons was that the sulphide waste would be contained in the crater and therefore, not create surface acid water as would be the case if placed on surface waste dumps

Further, the subsidence zone can be much smaller than an equivalent open pit and the effects on water catchment and disposal could be far less. Cave craters are often spectacular features, particularly if the rock mass is fairly competent and has widely spaced major joints resulting in cliff faces. They can hardly be condemned on that basis, as the reaction from laymen is invariably favourable.

However, wet, finely ground tailings cannot be partially disposed of in the crater of block caving mines, as in some cut and fill mines. While coarse development waste can be dumped into the subsidence zone, finely ground tailings should not be disposed of in the subsidence zone of a block cave under active draw or adjacent to a future block cave area, as these tailings could result in potentially life-threatening mud rushes.

Strategies for controlling the risks are similar to other mining operations. These include active measures to avoid or reduce the risk by:

- Well researched environmental impact studies by a multidisciplinary research team.
- An environmental management system that includes:
  - environmental management plans for each facet of the mining operation that could affect the environment.
  - periodic audits and reviews to check compliance with the management strategy and regulatory requirements over the life of the mine and beyond. .
- Comprehensive legal advice on exposure to environmental risks.

REFERENCE

Smithen, A A, 1999. Environmental considerations in the preparation of bankable feasibility documents *J S Afr Inst Min Metall*, vol 99, No 6, October 1999:317-320.

DESIGN TOPIC

Ancillary Development

Shafts, Ramps, Haulage's, Ventilation drifts, Crushers, Pumps / Sumps and Workshops

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GENERAL

Block caving operations are taking place and are planned to take place in high stress environments. The scale of a block cave operation is such that there are major adjustments to the regional environment and the responses need to be assessed before siting ancillary development. Peripheral geological data is required for planning purposes and would include geothermal gradient, surface / underground water, sources of 'mud'. The various sections of the manual have identified the factors that M be taken into consideration when designing a block cave operation, these must also influence the siting of ancillary development.

The scale and time interval of the operation must be recognised in determining rock ass response. It has been mentioned that rapid cave propagation in a high stress environment can result in seismic events and possible rockbursts in high stress areas remote from the event. Thus, the size and location of underground crusher stations requires careful thought.

Is the undercut advancing towards a haulage. The upper portion of a ramp and a workshop, for example might be opposite the highly stressed sill pillar which has formed as the cave back approaches the bottom of a pit or a previously mined area.

As block caving operations have longer lives, it is commonly seen as desirable to locate mine service facilities as close to the block cave as practical to minimise travelling and communication times and also the initial development. However, most caving blocks are large and thus a large volume of ground around them will be affected by the stress changes produced by a block cave. Presently, many block caving operations are located in regions of high lateral stresses, and more are planned.
The potential effects of a block cave on installations located in the peripheries of the block include:

- Increased stress levels leading to stress-related damage in "toe" and lateral abutment areas, and strain bursts or rock bursts in severe cases.

- Shear displacements on faults and shear zones. These could produce rockbursts, or result in damage to openings where the shear displacement destabilises large blocks of rock in the back or in pillars

- Relaxation of stress or dilation of structures can also destabilise blocks of rock in the back or pillars.

- Shear displacements and dilation can change the groundwater regime and allow water inflows along opened structures.

Clearly, these facilities need to be located in positions that are safe from the gross effects of the caving process operation. Thus they will need to be located in the best available ground in the vicinity of the block cave and sufficiently removed to be clear of damaging stress concentrations. This requires:

- The diamond drill holes used for evaluation and geotechnical purposes should be drilled through the orebody and into the peripheral ground for 100 m or more if the hole is steep. These holes should be fully geotechnically logged.

- When the position of the facilities have been selected from a rock mass quality point of view, the situation should be examined to determine whether there are any major structures that have excess shear strain and could produce rockbursts.

- Finally the induced stress concentrations should be examined by three-dimensional stress analyses (using such programs as FLAC^{3D}, EXAMINE^{3D} or MAP3D). Two -dimensional stress analyses (using such programs as FLAC^{2D} and PHASE^{2}) can also be used, but being two-dimensional usually give much higher stress concentrations than the three-dimensional models. They are, however, easier to set up and quicker to run than the tree dimensional models. They and could be used to demonstrate that the stress concentrations are not a problem, or, that there could be a potential problem requiring three dimensional modelling.

DESIGN TOPIC

Underground Research Projects

GENERAL

The new mining projects or many of the extensions with depth of current operations are all taking place in virgin territory. That is, no mining has taken place there and there is no experience of those conditions. New techniques have to be developed and new concepts have been presented, but often rejected by management, because nobody has tried that method. Well, someone has to experiment. The opportunities exist on the large operations, but seldom is anything done. When one considers the amount of mining research done on the relatively small operations of Shabanie and King mines in Zimbabwe then the large mining companies need to reflect on their reluctance to test other techniques or concepts.

NEW MINING METHODS

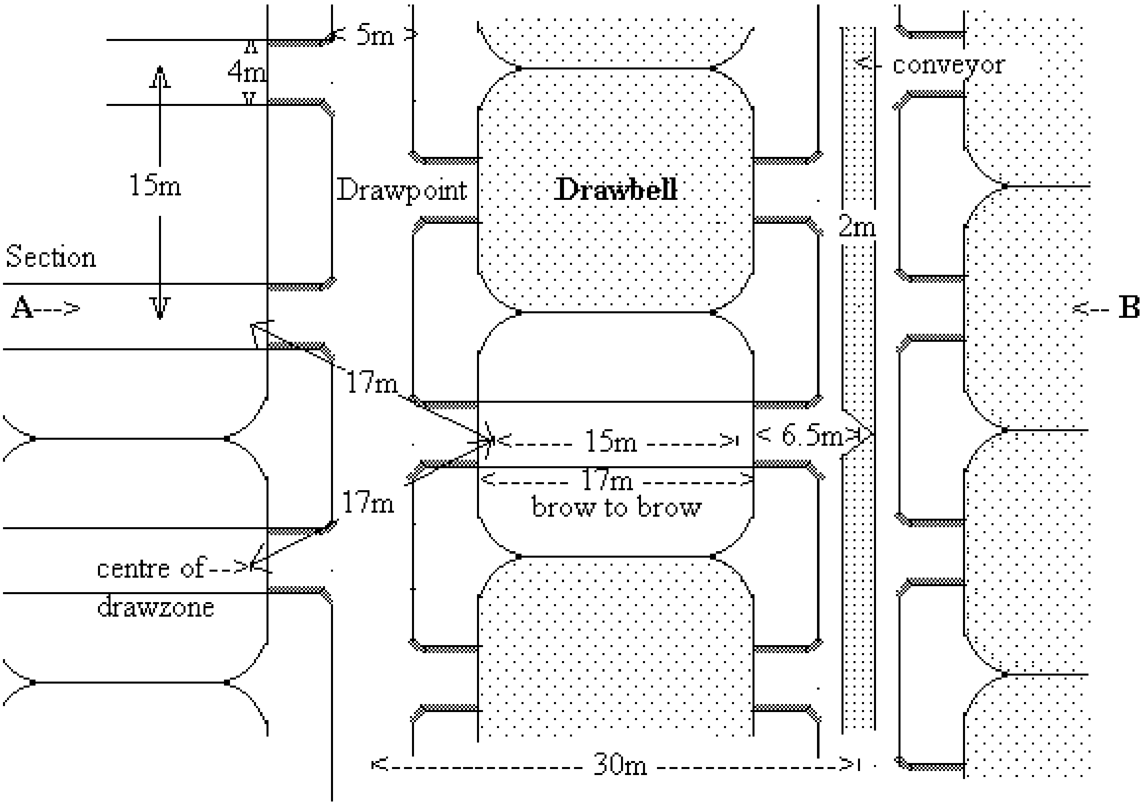
Requirements

- Reduce drawzone spacing across the major apex
- Increase strength of the extraction level
- Efficient handling of ore from drawpoint to drift and along drift to tip point.
- Is it possible to move ore laterally on production level or below, but, in stress shadow.

Strength of extraction level

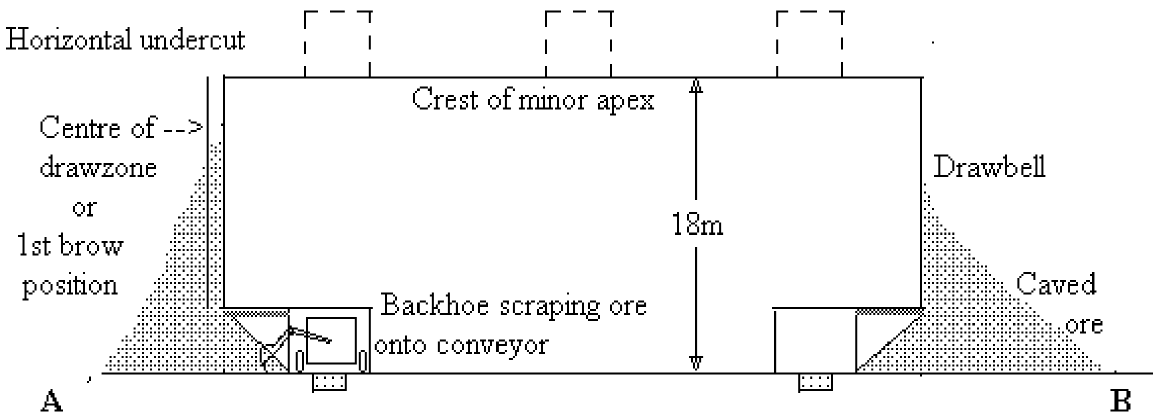
The strongest structure is a staggered layout with drawpoints at right angles to the production drift.

The following diagram illustrates a possible layout where the concept is to move the caved ore is scraped onto a conveyor with the drawpoints at right angles to the production drift.



OFF-SET BACKHOE LAYOUT - DRAWZONE SPACING = 15m, 15m, 17m

The principle is that the backhoe is very efficient in moving material and a conveyor is efficient in transporting material. Whilst the drawpoint is short it is backed by solid ground and at right angles to the drift. A strong structure.



Section A - B through drawpoint across minor apex

Secondary breaking level

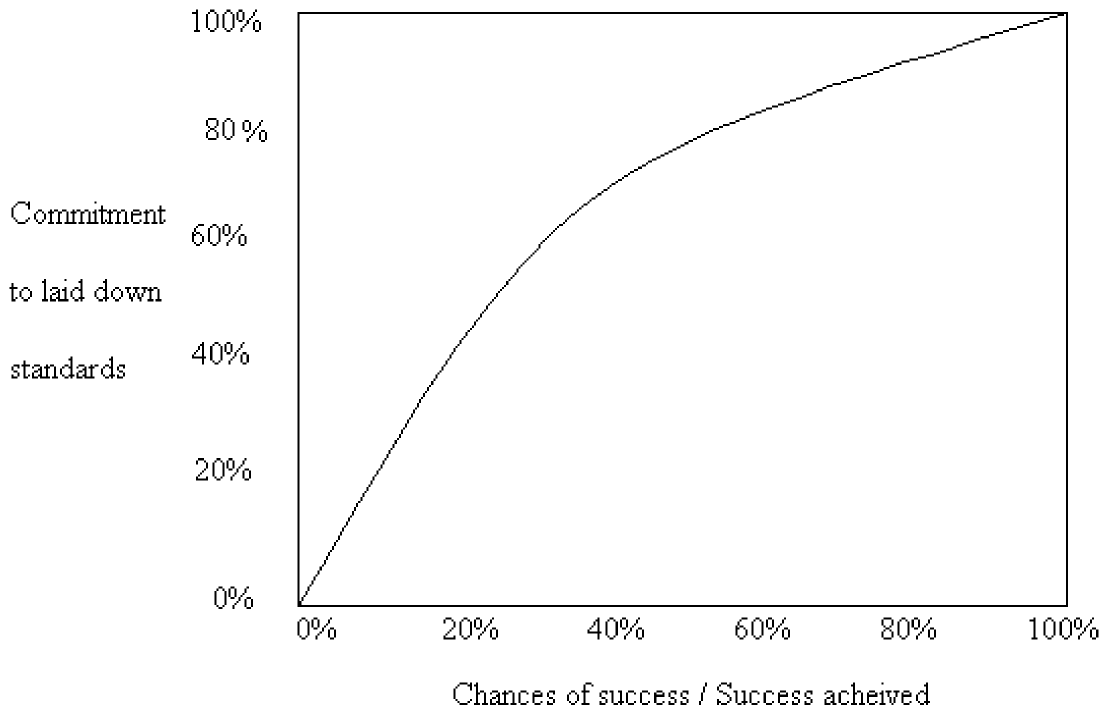
The secondary breaking or mezzanine level in the major apex has been referred to in the section on secondary breaking.   Here is a classic example of a large mine having all the facilities to try it out but not prepared to do so, they would rather be content to complain about the low productivity from that area.

DESIGN TOPIC

Role of Management - N.J.W.Bell

POLICIES

Management must at the outset of the project lay down a set of policies that it will enforce throughout the life of the project. These should include all the items detailed below and remembering the following graph :-



A philosophy of **'Do It Right'** needs to be established.

UNDERSTANDING OF RMR AND MRMR

Contractors and some mine employees will not understand the concept of adjustments from IRMR to MRMR and when faced with obviously “competent rock underground cannot understand a plan that shows weak rock and a demand that high quality timeous support is to be installed.

It is imperative therefor that this be fully explained. It has been found at AA Mines (Zimbabwe) that plans showing the IRMR in Classes 1, 2, 3A, 3B, 4 and 5 are easier understood and by then insisting on the correct support being installed the desired results are achieved.

It will be noted that 3A and 3B are separated as it has been found at AA Mines that the difference is marked in terms of the level of support required, 3A and better the rock can readily be reinforced but 3B and worse requires a lining (Shotcrete).

The use of average RMR or MRMR unless they have a range of less than 20 is to be discouraged as vital inferences could be lost.

SUPPORT AND CONSTRUCTION

This is one of the most vital areas for management to ‘get it right’.

The following is a guideline as to areas to be covered:

The support shall be designed in conjunction with the Production Officials and shall use the MRMR.

The overall policy of support installation through the initial, secondary and production phases below is one of progressive steps such that, ultimately, there is an integrated support system for production. Where possible, the type of support installed in the initial and secondary phases shall not preclude the effective installation of production support.

If at any time, in the opinion of an underground operator, additional support is required either for safety reasons or for overall ease of construction logistics, then the next higher phase(s) of support can be installed when he deems necessary.

The person in charge of the area shall have plans of all the development support work showing the in-situ rock mechanics classification and recommended support for the **initial, secondary and production** support for his area of the mine. The plans shall be updated at least monthly indicating what has actually been installed and making particular note of deviations from the specifications.

Any ground conditions seen in underground exposures which differ from those on the geomechanics plan shall be recorded and notified within twenty-four hours.

Shotcrete shall <u>always</u> be reinforced.

Initial Support

1 Is that support installed to hold the tunnel open until the next phase of support is added and is an integral part of the final support system.

2 This support shall be installed :

i to within 5 metres of the advancing face, in class 4 or 5 ground this might have to be right up to the face with every blast or

ii right to the face when a holing is about to be effected, or

iii right up to the face within 1 week of the face being stopped, or

iv across the holing point within 1 week of the holing being effected.

3 The minimum support in Class 4 or 5 ground is spiling bolts with every round, reinforced shotcrete or other lining with rockbolts and straps post-lining.

4 The minimum support in Class 3 B ground is spiling bolts with every round, reinforced shotcrete or other lining with rockbolts post-lining.

5 The minimum support in Class 2 or 3 A ground is rockbolts.

6 **All** development take-offs in **all classes** of ground shall be pre-supported **Before** the take off development commences and **After** the first round of the new heading in accordance with a laid down standard.

Secondary Support

1 Is that support installed to prevent deformation of the tunnel and to support it until the production phase of support is installed.

2 This support shall be installed :

i to within 15 metres of the advancing face, or

ii to within 5 metres of the face when a holing is about to be effected, or

iii to within 1 metre of the face within 3 weeks of the face being stopped, or

iv across the holing point within 3 weeks of the holing being effected.

3 The additional minimum support for Class 3 B, 4 and 5 rock is the support of the wide spans/junctions and such additional rockbolts and cable bolts that are prescribed.

Production Support and Construction

1 Is that support required for the productive duty of the drift and shall include **special cases**.

2 At least six months before production commences, this support shall be completed to at least a 40 m periphery of the initial area to be crashed except that to be erected in the drawpoint after the advanced undercut. The production support shall be kept ahead of the expanding crashed area to the same parameters.

Special Cases

These can be initiated by either Production or Design Personnel.

1 An underground visit to the area shall be arranged for an on-site inspection - by all personnel involved ensuring that all the relevant geological and geomechanics mapping for the area in question is to hand.

2 A critical assessment of the mapping information and the data gleaned in 1 shall be prepared.

3 Agreement shall be reached on what action should be taken and what action can be taken. Where these are different management shall decide the course to be followed and authorize its implementation.

4 The additional support shall be implemented as agreed.

5 The area shall be monitored until the problem is resolved.

UNDERCUTTING

This is a vital aspect of the cave mining process and must be carried out diligently.

The undercut must be continuous.

The effective undercutting of a production block is critical for production, efficiency and to ensure that the planned production tonne are extracted as scheduled with minimum dilution.

Failure to undercut a block properly results in the following:

a) Poor caving

b) Extreme mining stresses resulting in damage or failure of the support systems.
c) Loss of production.
d) Early dilution and poor grade recovery.
e) Inability to meet production schedules.

Therefore a procedure must be laid out and adhered to and should cover the following:

1 Undercuts shall be sufficiently supported to ensure that mining proceeds smoothly and timeously without interruption.

2 The area shall be prepared to allow for the efficient lashing of the daily blasts, should this be required (It may be planned that no lashing will be required) however if the need arises it will need to be quickly.

3 Drilling of the undercut must be to a high standard.

4 The blasting must in the full control of the blasting supervisor who will decide if redrilling is required or if multiple blasts of the same ring are required.

5 The shape, direction, rate and agreed stop positions of the undercut must be clearly communicated to all concerned.

6 After each blast, it must be ascertained if the ring has broken to full depth and height. To this end it may be required to lash the broken spoil.

7 If the ring has not broken to full depth and height corrective measures must be taken immediately e.g. Lash to expose sockets for recharging or redrill (remotely) and charge. It is vital that **No** pillars are left, they will **Not** crush and collapse, they **Will** punch through and even hold the entire caving process.

Sufficient suitable props need to be available at short notice to effect temporary support if required.

8 Careful monitoring of the undercut operation is critical and any adverse or unusual conditions arising shall be reported immediately in order that corrective action can be taken.

9 During the undercutting process, if a large open area is created and at the end of under cutting or at a stop position the undercut drifts must be filled with spoil as a precaution against air blast damage. To achieve this depending on the plan blast additional rings without lashing or drill ‘King Rig’ holes into the back of the undercut area and blast them.

Production Rates

The planned production rate must take into account the requirements for the caving block so as not to force the production when good cave mining practice dictates that it must be slower in order to reduce the possibilities/effects of:

Chimney Caves
Empty Drawpoints/Air Blasts
Seismic Activity
Poor Draw Control

Draw Control

The draw control philosophy needs to be clearly understood by all. In general there is no shame in the truth.

1. The draw from drawpoints shall be from those detailed by the control instructions and at the rate prescribed.

2. If any other drawpoints are worked they shall be indicated with tonnage and the reason stated.

3. Drawpoint closures need to be effectively handled.

APPENDIX I

Support Appropriate For Dynamic Loading and Large Static Loading In Block Cave Mining Openings

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ABSTRACT

Seismicity associated with deep block cave mining operations in hard rock can give rise to dynamic loading of the mining openings. A description is given of the source and damage mechanisms associated with such seismic events, and the possible demands placed on the rock support by the resulting dynamic loading. In addition, production level drifts are often subjected to large static deformations during mining.

Appropriate support, able to sustain large static deformations, and capable of absorbing the energy released during dynamic loading, is described. The performance of these support types has been determined by means of large scale dynamic loading tests, and large deformation static laboratory tests.

Finally, a support system which is considered to be appropriate for extraction drifts in block cave layouts subjected to dynamic loading is described.

1 INTRODUCTION

Cave mining of deep, hard rock orebodies, involving removal of large volumes of rock, will inevitably lead to the generation of mining-induced seismicity, which may lead to rockbursts. A rockburst may be understood to be “a seismic event which causes violent and significant damage to tunnels and other excavations in the mine.” There are no constraints on the magnitude of the seismic event. Thus, the event can range from a strainburst, in which superficial surface spalling with violent ejection of fragments occurs, to a mining-induced “earthquake” involving slip along a fault plane. The range in Richter magnitudes for these two limits is from about -0.2 to 5.0. This seismicity can lead to dynamic loading of the rock surrounding mining openings and may also cause rockbursts. The main types of rockburst source mechanism which have been identified (Ortlepp and Stacey, 1994) are summarised in Table 1.

TABLE 1. Suggested classification of seismic event sources with respect to tunnels

Seismic Event	Postulated Source	First Motion from Seismic Records	Guideline Richter Magnitude M_L
Strain-bursting	Superficial spalling with violent ejection of fragments	Usually undetected; could be implosive	-0.2 to 0
Buckling	Outward expulsion of pre-existing larger slabs parallel to opening	Implosive	0 to 1.5
Face crush	Violent expulsion of rock from tunnel face	Implosive	1.0 to 2.5
Shear rupture	Violent propagation of shear fracture through intact rock mass	Double-couple shear	2.0 to 3.5
Fault-slip	Violent renewed movement on existing fault	Double-couple shear	2.5 to 5.0

It should be noted that some types of seismicity do not necessarily require high stress levels for their occurrence. Stacey (1989) found that, from a review of the occurrence of this type of seismicity, particularly with massive rock conditions, strain bursts can occur in tunnel development when the field stress is as low as 15% of the uniaxial compressive strength of the rock material. More recent information suggests that this figure could be as low as 10% for very brittle rocks.

The location of the source of the seismicity and the location of the rockburst damage may or may not be coincident. In the larger magnitude events, the separation of the two locations may be hundreds of metres. The factors determining the intensity of the seismic impulse include the following (Ortlepp, 1997):

- the amount of energy available;
- the rate of liberation of energy;
- the source distance and dimension;
- the peak particle motion (shear, compression or other);
- the ray path properties, influenced by the geological structure, or major excavations, intervening to cause reflection, shielding, channelling etc of the seismic waves.

Factors which influence the response of the excavations to the seismicity include excavation geometry (size and shape); site amplification factors (stress intensity, stress distribution); characteristics of the surrounding rock (strength, brittleness, fabric, structure, intensity of induced fracturing); characteristics of existing support (length, strength, density, yieldability, quality of installation, quality of containment support).

Types of rockburst mechanisms which have been identified include ejection, buckling, gravity enhancement, shake-out, fall of ground associated with a large, distant seismic events, disruption and displacement, and convergence and heave.

2 IMPLICATIONS OF SEISMICITY FOR CAVE MINING LAYOUTS

In hard rock, deep cave mining conditions, it can be expected that the following types of seismicity will be experienced:

- strain bursting from the walls and faces of tunnels could be expected to occur intermittently on any level during primary development. This is hazardous, since sharp edged fragments, often plate sized, are ejected violently, but does not result in major stability problems. During undercutting, strain bursting could be expected to occur with greater frequency in all undercut development, but particularly in the region of the abutments and advancing undercut front. This behaviour should reduce substantially or even stop once the cave has propagated through to surface;
- owing to the major changes in the stress distribution in the cave back and surrounding rock mass during the development of the cave, and subsequently in the surrounding rock mass as a result of the creation of the "destressed cave void", stress conditions may be conducive for the generation of buckling, fault slip, and possibly shear rupture types of seismic events. These events could involve large amounts of energy, and their effects could be manifested in the production level excavations, and openings such as ventilation, rock breaker, crusher and other service excavations adjacent to or below the production level. The damage associated with such events has involved roof, sidewalls and floor of excavations, in many cases resulting in complete closure of the tunnel. It has typically been observed that approximately a metre thickness of rock from the walls of the excavation is violently ejected. From back analyses of the ejection velocities of several of these occurrences, it was suggested that an appropriate velocity for support design purposes could be 10 m/s (Ortlepp and Stacey, 1994). The undercut level will be substantially protected from these types of events.

There are three potential approaches to the alleviation of problems due to rockbursts. These are:

- prevention of seismicity, and hence rockbursts;
- prediction of rockbursts, and timely evacuation of personnel and equipment;
- containment of rockburst damage with appropriate support.

It is considered that the only practical option of these three is that of containment of damage. Success with prevention and prediction is likely to be only partial, and the implication is that there will always remain some risk of rockbursts which will have to be addressed by means of appropriate rock support. In the following sections, the results of dynamic testing of different types of rock support are described.

In addition to potential dynamic loading, deep caving layouts will be subjected to significant static stresses and stress changes, and support will be required to contain large static deformations. The results of some large deformation static testing of reinforced shotcrete will also be given.

3. PERFORMANCE OF SUPPORT UNDER SEVERE LOADING

Several programmes of dynamic support testing have been carried out over the past 8 years, involving both retainment support such as rockbolts and cables, and containment support, including various types of wire mesh, wire rope lacing, and shotcrete. Retainment support testing, containment support testing and static load testing will be dealt with separately in the sections below.

3.1 Dynamic testing of retainment support

Using explosives as the impulse force, Ortlepp (1994) carried out tests on rebar and smooth bar rockbolts, and special yielding rockbolts. The results of these tests are summarised in Table 2.

TABLE 2. Results of impulse load testing of rockbolts

Rockbolt Type	Total Resistance (kN)	Amount of Explosive (kg)	Max. Height (m)	Max. Velocity m/s at distance S mm	Damage to Tendons
16mm cone	290	0.5	0.33	6.5/180	Nil
16mm cone	290	1.0	1.80	8.6/80	All pulled out but totally undamaged
16mm smooth	700	0.5	0.38	7.3/20	Nil
16mm re-bar	725	0.5	2.03	8.0/50	All broken
22mm cone	1035	1.0	0.50	12.8/55	Nil
25mm re-bar	1350	1.0	4.65	10.2/500	Two pulled out three broken
Calibration tests	-	0.5	1.8	6.5/20	-
	-	1.0	5.20	10.2/100	-

From these results it can be seen that the rebar rockbolts were not able to contain the energy, and this type of support failed in all cases. In contrast, the yielding cone bolts performed well, and none of the bolts was broken. The 16mm cone bolts failed as a system under the impulse of a large amount of explosive, but the bolts were undamaged. The smooth bar bolts also performed well. Thus, in spite of the fact that the rebar rockbolts had the greatest strength capacity, they were unable to resist the dynamic loading.

Using a drop weight impact system for loading, dynamic testing has been carried out on the following (Stacey and Ortlepp, 1999):

- 16mm diameter rebar (fully grouted, rigid);
- 16mm diameter smooth bar (partially bonded, semi-yielding);
- destranded hoist rope strand (lubricated, helix);
- 18mm compact strand cable (fully grouted, prestressed, very stiff);
- 39mm diameter split sets (percussively driven, friction contact);
- standard Swellex bolts (inflated, friction contact).

The elements were grouted into thick-walled steel tubes, or installed in boreholes formed in simulated rock in steel tubes. The dimensions of the tubes were chosen so that the tubes provided confinement of the same order as that provided by the rock mass. Each test specimen consisted of two lengths of tube, butted together. The butt provided the joint at which failure of the rockbolt element could take place.

Large input energies, sufficient to fail all of the element types, were achievable by means of the drop weight impacting on a swing beam. Measurements of the impact velocities showed that velocities in excess of 20 m/s were achieved in some cases.

In all of the tests, the behaviour of the rockbolts was characterised by one, or a combination of two or all, of the following mechanisms:

- if the rockbolt was strongly bonded, or prevented from slipping, necking of the bolt commenced at the separation surface. Owing to the small volume of steel involved in the yield process, the total energy consumed before failure takes place is small. This energy component is the rupture energy;
- if failure of the grout/steel bond takes place, then yield of the steel with concurrent work hardening also occurs. This takes the form of incipient necking, which advances away from the initiation point. The energy component developed in this progressive elongation of the rockbolt is termed the elongation energy;
- a small decrease in the diameter of the rockbolt accompanies the progressive elongation of the steel. This causes progressive debonding to occur and, for smooth rockbolts, allows sliding to take place. In the case of friction rockbolts, the hollow steel tube makes it relatively easy for the pressure normal to the interface to relax sufficiently to allow sliding to take place. The energy component in this mechanism is termed the sliding energy.

The results of the testing are shown in Table 3. These represent a summary of tests on 58 specimens, often involving multiple drops on each specimen.

TABLE 3. Results of drop weight testing of rockbolt elements

Rockbolt Type	Rupture Energy kJ	Elongation Energy kJ/100mm	Sliding Energy kJ/100mm
Rebar 16mm	4.8 ± 0.7	-	-
Smooth bar 16mm	-	13.9	3.0 x l_e – 0.15*
Vaal Reefs hoist rope strand	<2.4	-	0.8 to 8.2
18mm compact strand cable	17.5 ± 2.0	-	-
39mm diameter splitsets	11.6 to 15	11.6 4.6 ± 1.3	3.0 to 6.3
Standard Swellex	4.6 ± 0.5	9.9	3.0 to 5.8
* l_e is initial embedded length in metres			

In initial tests, in which input energies were small, failure of the rebar rockbolts could not be achieved. It was found that the softness of the testing system enhanced the occurrence of the second mechanism described above, and allowed energy absorption to occur due to the progressive elongation of the steel. This phenomenon also occurred with the other rockbolt types and it was necessary to stiffen up the testing system.

Rupture of the rebar rockbolts was similar to that commonly seen underground, with fairly abrupt necking. Rupture of smooth bar did not occur, and all energy was usually consumed in elongation of the steel. On some occasions bars pulled out of the grout, and sliding also absorbed energy. The hoist rope strand could also not be ruptured since, in all cases, slip of the strand occurred and most of the energy was absorbed in sliding. The 400kN prestressed cables ruptured in a brittle fashion, with classical cup and cone surfaces formed on each strand. This confirms that strength is not important in dynamic performance of support.

The tests on friction rockbolts were not very satisfactory. In a straight hole, slip always occurred and it was not possible to rupture the bolts. Rupture was achieved when the bolts had been artificially “fixed”, to simulate the clamping that would occur due to partial dislocation of the hole.

The testing programme yielded several interesting and important insights into the likely behaviour of tendons under real dynamic “in-mine” conditions:

- where strong rock inhibits the development of a softer shell of fractured rock around the tunnel, stiff rockbolts will be vulnerable to acute stress concentrations across joint separation surfaces, resulting in their rupture at low energy levels;

- if a dynamic impulse is experienced, which causes some separation at a joint or fracture, but which is of insufficient magnitude to rupture the rockbolt, then, due to the debonding experienced by the slightly elongated portion of the bar across the joint, a greater length of bar is available to absorb more energy if it is subjected to further impulses later. The rockbolt effectively acquires extra energy-absorbing capacity;

- in uniform straight holes, hollow friction dependent rockbolts will not break in tension, but will slide even if the fit is very tight in the hole. The full potential tensile capacity of these rockbolts will only be realised if the hole is sufficiently distorted or dislocated to clamp the rockbolt without shearing it;

- in practice, reduced support system stiffness will result from:

- the prior existence of significant fracturing in tunnel walls;
- commencement of debonding as a result of earlier static yield or sub-critical dynamic loading;
- the presence of softer elements in the support system such as non-stiff face plates, poor quality grout, incomplete grouting etc;

- inadequacies in terminating arrangements on the rockbolts (face plates, the welded ring on splitsets, the sealing weld and soft ferrule on Swellex, and the swaged sleeves on the loops of destranded hoist rope strand), often result in premature failure.

Although no testing of the shearing resistance of rockbolts was carried out, it could be expected that yielding rockbolts would perform much better in shear than rigid, fully grouted rockbolts. For softer rocks (with the strength of good concrete), Gillerstedt (2000) has shown that this is the case. His tests showed that shear displacements, normal to the rockbolt axis, of more than 200 mm could be achieved with yielding bolts, compared with about 40 mm for the rigid bolts.

Finally, it is ironical that poor installation quality, including weak grout and incomplete filling of holes with grout, is likely to improve the yield capability of rigid rockbolts such as rebar bolts. This is by mistake, not by design, and cannot be relied on in the application of a formal support design procedure.

3.2 Dynamic testing of containment support

Dynamic tests have been carried out on a range of containment support types, including welded wire mesh, chain link wire (diamond) mesh, wire mesh with wire rope lacing, fibre reinforced shotcrete and fibre reinforced shotcrete with wire rope lacing (Ortlepp et al, 1999). The aperture and strand diameters of the weld mesh used were 100mm x 3.5mm and 100mm x 4mm. The corresponding values for the diamond meshes were 75mm x 3.2mm, 100mm x 3.2mm and 100mm x 4mm. 8mm, 10mm and 12mm wire rope lacing was used. Shotcrete was reinforced with 100mm x 4mm weld mesh, with 30mm long Dramix fibres, and 50mm long monofilament polypropylene fibres. A few of the tests incorporated special mesh, and lacing with yielding capabilities.

In a rockburst situation, the dynamic loading imposed on containment support is in the form of a violent impact on the support by the rock mass, distributed across the surface of the support. To simulate this type of loading, a drop weight test method consisting of the following was used. The panel of the containment support was suspended by means of four yielding rockbolts, spaced 1 m apart, with an artificial "rock mass" in contact with and on top of the panel. Above this a load distribution pyramid was formed. Tie-back cables, acting in the plane of the panel, provided extended boundary conditions. Dynamic loading was imposed by a drop weight, which could provide an input energy to the system of up to 70 kJ/m^2, and a maximum impact velocity of about 8 m/s. The aim was to provide a method which could apply a series of repeatable loading conditions, and thus allow comparative performance of different support types to be determined.

The results of the tests are plotted in terms of centre deflection of the panel against the total input energy. It can be seen that unreinforced shotcrete has the poorest performance, and that weld mesh also occupies the lower energy area of the plot. Strands of the weld mesh broke in almost all of the tests, and in some cases welds broke as well. It was found that the sharp-edged steel face plate contributed to the failure of the strands. From the results of tests on the weldmesh, it is estimated that the practical total energy input limit for weld mesh is 10 kJ/m^2.

Diamond mesh performed better, and the practical limit for total energy input is estimated to be 15 kJ/m^2. However, there was a tendency for this mesh to unravel once a single strand had failed, and this allowed the rock mass to spill through.

The performance of fibre reinforced shotcrete is approximately equivalent to that of diamond mesh. Shotcrete reinforced with Dramix steel fibres performed slightly better than that with monofilament polypropylene fibres, the former having an estimated energy absorbing capability of 20 kJ/m^2. A concern from the testing is that, after the initial weight drop, which was contained by the panel, a second drop on the same panel destroyed the support. The implication is that fibre reinforced shotcrete on its own may be suspect if subjected to repeated dynamic loading, or to dynamic loading after it has been cracked significantly by static deformation. This potential weakness can be reduced if rope lacing is added.

The dynamic testing showed that, with the addition of wire rope lacing to the containment support system, the capability of the support system to absorb energy was considerably increased. Compared with the mesh only behaviour of the two types of mesh tested, the performance characteristics were reversed with the addition of lacing – the weld mesh with lacing performed better than the diamond mesh with lacing. Failure of a strand of the diamond mesh generally allowed the rock mass to spill through. In contrast, the weld mesh contained the rock mass even though some of the strands failed. The maximum capacity achieved by the weld mesh and lacing was 51 kJ/m^2. Diamond mesh and lacing support failed catastrophically with energy inputs of 34 and 48 kJ/m^2.

A monfilament polypropylene fibre reinforced shotcrete panel was tested with the addition of wire rope lacing. It absorbed a substantial amount of energy, this occurred with cracking, but not complete failure, of the shotcrete and the capacity is regarded as being conservative.

Several tests were carried out with special yielding mesh, and with such mesh and wire rope lacing in which a yield capability was introduced. This support withstood, without failing, the maximum amount of energy of 71 kJ that the testing system could apply. This amount of energy is considered to be representative of a severe rockburst, and the results demonstrate the possibility of containing this damage satisfactorily.

3.3 Performance of reinforced shotcrete under large static deformations

The addition of fibres to shotcrete considerably increases its yield capacity. Testing of the yield capacity of shotcrete panels, specifically aimed at large deformation capability, has been carried out over the past 10 years (Kirsten et al, 1997). These tests have shown that fibre reinforced shotcrete, between rockbolts spaced 1m apart, can perform as well as diamond mesh reinforced shotcrete to a central deflection of at least 150 mm. Cracking of the shotcrete occurs early in the deformation process, but the fibres and mesh bridge the cracks, and provide the on-going yield capability. In the tests it was found that the performance of weld mesh as shotcrete reinforcement was poor due to the good mechanical bonding between the mesh structure and the shotcrete. In contrast, diamond mesh is flexible and is able to pull through the shotcrete. The application of shotcrete to diamond mesh prevents unravelling of the strands after local failure has occurred.

The specifications for shotcrete and the fibre type, length and content are important, and it is important to note that they can be very different from those required for civil engineering applications. In civil engineering applications it is often the occurrence of the first crack which serves as a design criterion. In mining applications this criterion is irrelevant. In cave mining it can be expected that cracking of the shotcrete will definitely occur, and the requirement is to ensure that the fibres maintain the integrity of the shotcrete “mass”. Since the shotcrete is likely to crack in any case, it is not important that the shotcrete itself has any high strength specification. It needs to have sufficient strength to provide a competent medium for the fibres. The yield performance of fibre reinforced shotcrete is sensitive to the types of fibres used, the quantity of fibres in the mix, and the length of the fibres. In the testing that has been carried out, good yield capability was achieved with 40 mm long Dramix fibres with a fibre content exceeding 2.5% by mass, and with monfilament polypropylene fibres 40 mm and 50 mm long with a fibre content exceeding 0.35% by mass.

4. Support Systems To Withstand Large Static And Dynamic Deformations In Block Cave Layouts

The results of the testing programmes described above have shown that available support elements and systems are capable of withstanding large static and dynamic deformations without failing. With relevance to the support of caving layouts, some of the most important points to come out of these results are considered to be:

- strong, rigid elements such as fully grouted rebars, can fail in a brittle fashion after very small deformations;

- diamond mesh performs better than weld mesh in a situation with rockbolts and mesh only;
- shotcrete, suitably reinforced with fibres, performs as well as wire mesh in initial loading, but is suspect under repeated loading. Shotcrete, even with fibre or mesh reinforcement, cracks after a small amount of deformation;
- the addition of wire rope lacing greatly increases the energy absorbing capability of all support types, and both fibre reinforced shotcrete and mesh performed well in this case. Weld mesh performed better than diamond mesh;
- the incorporation of special yield capabilities in mesh and lacing elements allows large deformations and massive amounts of energy to be absorbed without failure of the support.

One of the factors which affects support performance, which has not been dealt with above is that of corrosion of support. It has been found in practice that installed support may appear to be in excellent condition, but that, under dynamic loading conditions, it fails completely (Durrheim et al, 1998). In such cases, examination often shows that the rockbolts have been corroded behind the surface containment support or where they intersect joints. This will weaken, and may totally destroy, the support. Many mine atmospheres are extremely corrosive and mine water can be significantly acidic. The rusting of steel support elements can be seen in many operations. Very rapid corrosion of steel fibres was observed by Venter and Gardner (1998), to the extent that the fibres no longer contributed to the integrity of the support system. Subsequent use of monofilament polypropylene fibres proved to be very successful.

It is also necessary to consider what are the requirements for support of caving layouts. These include:

- the support system must have stiff load bearing capability to limit static deformations under "squeezing" conditions;
- the support system must have yield capability so that it continues to provide support after significant deformation, without failing;
- the support system must have the capability of yielding rapidly, without failing, in the event that seismicity occurs;
- rockbolts must have the capability of withstanding shear displacements on joints between rock blocks;
- the support must be capable of withstanding mechanical damage due to movement of LHD's and other equipment;
- the support must not deteriorate due to corrosion or other time dependent factors such as grout weakening.

With regard to a recommended support system appropriate for deep block cave production layouts in a hard rock environment, the first point that must be made is that, whatever support system is used, **all elements must be matched** in terms of capacity. There is little to be gained from the use of high quality mesh and shotcrete with rigid rebar rockbolts. In addition, connecting elements must also be compatible. Small stiff face plates can easily pull through shotcrete, and sharp edged face plates can cause premature failure of mesh.

It is recommended that retainment elements have a significant yielding capability, of the order of 200mm as a minimum. This will enhance their performance and life under both axial and shear deformations. Rebar rockbolts and friction rockbolts are not recommended. The retainment elements should be stiff in their initial deformation behaviour. The designed length, spacing and yield capacity of the elements will vary from one block cave to another, and will have to be determined during the planning of each operation.

Containment support will require the use of shotcrete for protection of the support against mechanical damage. It would be advantageous, therefore, if fibre reinforced shotcrete could be used as the containment support, or part of it. At this stage, however, its performance has not been sufficiently proven under large, on-going deformations. Therefore, until in situ testing has proved its effectiveness, it is not the recommended containment support. It is recommended, however, that it should be used in trial sections to investigate its performance. It is also recommended that trial use of “super skins” should be carried out. These will not replace shotcrete with regard to mechanical damage protection, but may be particularly useful as the initial support layer, particularly with non-durable rocks.

Diamond mesh, with shotcrete, is the recommended containment support. This has the proven toughness and yield capacity. It should be noted that it is difficult to apply good quality shotcrete over diamond mesh, in particular mesh with an aperture of less than 100mm. This type of mesh tends to sag, and vibrates during shotcrete application.

Wire rope lacing has been proved to be very effective in situations requiring yielding support, and this is recommended in addition to the mesh and shotcrete. The wire rope should be lightly tensioned, to take up the slack, but not to stress the rope. The aperture and strand diameter of the wire mesh and the diameter and strength of the wire rope will require design for each block caving layout. As an alternative to wire rope lacing, tendon straps may be very effective. They have been proved to be so in large static deformation situations (Wilson, 1991), but have not yet been proven under dynamic loading conditions. Tendon straps are likely to provide stiffer resistance than wire ropes in the early stages of deformation. It is recommended that trials should be carried out with tendon straps to investigate their effectiveness.

The connecting elements must ensure satisfactory load transfer between containment and retainment support elements. Face plates, or other connection devices, must be of sufficient size and capacity to prevent local failure of the containment support. Again this must be taken into account in the design of each particular block caving layout.

In addition to the above “routine support”, special support may be required for drawpoints and brows, including concrete, steel or fabricated arches, long cables or wire ropes, and wire rope wrapping of bull noses. These are considered to be normal requirements, not related to dynamic loading, and will be specific to each block caving layout. They are not dealt with here.

5. CONCLUSIONS

The results of dynamic testing of retainment and containment support, and large deformation static testing of shotcrete, have been described in this paper. The following are the conclusions regarding the support of deep block cave layouts, possibly subjected to mining-induced seismic loading:

- retainment support should consist of yielding elements, with a stiff early deformation characteristic, that will perform well for large tensile and shear deformations;
- shotcrete will be required to protect support against mechanical damage;
- whilst fibre reinforced shotcrete appears to have potential, at this stage it is recommended that diamond mesh and shotcrete, with wire rope lacing, is used as the containment support. Trial sections with fibre reinforced shotcrete and with tendon straps should be implemented to investigate their performances;
- connecting elements between retainment and containment elements must be compatible, and all support elements must be matched in terms of capacity.

APPENDIX II

Rock Mass Classification IRMR/MRMR

D.H. Laubscher and J. Jakubec

INTRODUCTION

The competency and engineering properties of jointed rock masses can vary greatly. There is a demonstrated need for a systematic numerical system of describing rock masses. Such a system is used:
for communication between geologists, engineers and operating personnel to provide the basis for comparison of rock behaviour from project to project and over time, and to quantify experience for the development of empirical relationships with rock mass properties and for guidelines for method selection, cavability, stability, support design etc.

The history of the MRMR system needs to be recorded. In 1973 D. H. Laubscher met with Z.T. Bieniawski to discuss the rock mass classification system that Bieniawski was developing - RMR - for geotechnical investigations of civil engineering projects and to overcome communication problems (Bieniawski 1973). His approach was better than the system being developed in Zimbabwe by Heslop and Laubscher at that time (Heslop 1973). However, Laubscher decided that a lot more flexibility was required for the different mining situations and used the RMR **concept** for in-situ ratings and brought in adjustments for mining situations, thus the MRMR system was developed (Laubscher 1975) (Laubscher and Taylor 1976) (Laubscher 1990). Over the years changes have been made to the value of the ratings as the relative importance of the items became apparent. For some time there has been concern that the role of fractures/ veins and cemented joints were not properly included. The techniques that have been developed to cater for these items have been included here.

In order to avoid confusion with the Bieniawski RMR, the term IRMR is now used to indicate the rating of the *in situ* rock mass.

The overall objectives of this paper are (1) to show how the MRMR classification system can be applied to jointed rocks and (2) to indicate the changes made to the system over the years. Figure 1 is a flow sheet to assist the reader in following the different parts of the system.

DEFINITIONS

The competency of jointed rock is heavily dependent on the nature, orientation and continuity of the discontinuities in the rock mass. Figure 2 is a diagrammatic presentation of the different structural features.

Faults and shear zones - Major features, large scale continuity and frequently very weak zones. Must be classified separately.

Open joints - An easily identified structural discontinuity that defines a rock block.

Cemented joints - A structural feature that has continuity with the walls cemented with minerals of different cementing strength. In high stress environments cemented joints can impact on the strength of the rock mass, therefore, the frequency and hardness of the cementing material must be recorded.

Fractures and veins - Low continuities and occur within a rock block. The hardness number defines the fill material and open fractures have a hardness of 1.

Mapping and core logging - It is essential in scan line mapping to log the continuities of structures and to distinguish between fractures and joints and other structural defects. In drill core logging the geologist should attempt to classify the defects being logged. It should be noted that joint may have several partings in close proximity, these will behave as a single joint and should be logged as such.

INTACT ROCK STRENGTH (IRS) TO ROCK BLOCK STRENGTH (RBS)

Intact rock strength

The unconfined compressive strength - UCS, is the value derived from testing cores and is the value is assigned to the intact rock strength - IRS. The intact rock specimen may be homogeneous or have intercalation's of weaker material, in which case the procedure shown in Figure 3 should be adopted. Care must be taken in determining this value as often the cores that are selected represent the stronger material in the rock mass. To help the reader in this regard an example is presented. As shown in Figure 3, the UCS values for the strong and weak rock are 100 MPa and 20 MPa respectively. It is estimated that of the total, 45% is made up of weak rock. Using figure 3 one locates this value on the Y axis, moves horizontally to the curve representing the strength of the weak rock, and then drops down to the horizontal axis. In this case the appropriate "corrected" IRS is 37 MPa.

Rock Block Strength (RBS)

To obtain the rock block strength (RBS) from the "corrected" IRS, various factors are applied dependent upon whether the rock blocks are homogeneous or contain fractures and/or veins.

Homogeneous Rock Blocks

If the rock block does not contain fractures or veins then the rock block strength **-** RBS **-** is the IRS value reduced to 80% to cater for small to large specimen effect. Thus RBS = 0.8 x "corrected" IRS.

Rock blocks with fractures and veins - Fractures and veins reduce the strength of the rock block in terms of the number and frictional properties of the features (see figure 2). The Moh's hardness number is used to define the frictional properties of the vein and fracture filling. The standard hardness table is used, since values greater than 5 are not likely to be significant. Open fractures / veins would be given a value of 1.The vein and fracture filling must be weaker than the host rock:

Index	1 = Talc, Molybd.	2 = Gypsum, Chlorite	3 = Calcite, Anhydrite	4 = Fluorite, Chalcopy.	5 = Apatite
Inverse	1.0 0.5	0.33	0.25	0.2	

The procedure is to take the inverse of the hardness index and multiply that by fracture / vein frequency per meter, so as to arrive at a number which reflects the relative weakness between different rock masses. This number can then be used in Figure 4 to determine the percentage adjustment to the IRS value.

To obtain the RBS, the corrected IRS is adjusted by the size factor of 80% and then by the fracture/vein frequency and hardness adjustment i.e.

RBS = IRS x 0.8 x Fracture/vein adjustment (F/V) = MPa.

To illustrate this consider the following example:

IRS = 100 MPa
gypsum veins: Moh's hardness = 2,
ff/m = 8.0

The product of the inverse hardness and the fracture frequency is

Inverse of hardness index x fracture frequency = 0.5 x 8 = 4.0

Using Figure 4, one finds that the adjustment is 0.75. Therefore

RBS = 100 x 0.8 x .75 = 60 MPa.

The rating for the Rock Block Strength (RBS) can be read from Figure 5. The slope of the curve is steeper for the lower RBS values as small changes are significant.
In this case it is seen that the RBS rating is

RBS = 17.5

JOINTING

Open Joint Spacing

In previous papers, one had the option of using the RQD and joint spacing or ff/m. However, the fracture/vein frequency and their condition is part of the rock block strength calculation and therefore cannot be counted twice. It is for this reason that the joint spacing rating is reduced to 35 and refers only to open joints. Whilst there are situations where there are more than three joint sets, for simplicity they should be reduced to three sets. The chart in fig. 6 is slightly different to the previous ratings chart in that the ratings for the one and two sets are proportionately higher.

Cemented Joints

The cemented joints will influence the strength of the rock mass when the strength of the cement is less than the strength of the host rock. If the cemented joints form a distinct set then the rating for the open joints is adjusted down according to Figure 7.

For example, if the rating for two open joints at 0.5m spacing was 23, an additional cemented joint with a spacing of 0.85m would have an adjustment of 90%, so that the final rating would be 21, equivalent to a three joint set with an average spacing of 0.65m. The slope of the curve is increased to cater for the significant influence of the closer joint spacing. Failure can often occur at the cemented joint contact under high stress conditions or with poor blasting

JOINT CONDITION

Single joints

The IRMR system is revised to cater for cemented joints and to have water as a mining adjustment, however the joint condition rating remains at 40, but, the joint condition adjustments have been changed to those given in Table 1.

Table 1 - Joint condition adjustments

		Adjustment % of 40
A.	Large scale joint expression	
	Wavy - multi directional	100
	Wavy - uni - directional	95
	Curved	90
	Straight, Slight undulation	85
B	Small scale joint expression (200mm x 200mm)	
	Rough stepped / irregular	95
	Smooth stepped	90
	Slickensided stepped	85
	Rough undulating	80
	Smooth undulating	75
	Slickensided undulating	70
	Rough planar	65
	Smooth planar	60
	Polished	55
C	Joint wall alteration weaker than sidewall and filling	75
D	Gouge	
	thickness < amplitudes	60
	thickness > amplitudes	30
E	Cemented / filled joints - cement weaker than wall rock. The percentage in the column is the adjustment to obtain the cemented filled joint condition rating	

Hardness	Adjustment
5	95%
4	90%
3	85%
2	80%
1	75%

Multiple joints

Average joint condition ratings are required for IRMR values, however, a significant variation in joint condition ratings could be due to trying to force dissimilar areas into one rating. **It is preferable to use the classification system to show variations in the rock mass as this zoning could influence planning decisions**. A weighted average of joint condition ratings can give the wrong result particularly if the rating of one set is high. For example, a single joint set with 3 joints/m has a joint spacing rating of 22 and a joint condition rating of 20 = 42.

If this set is combined with another set with a joint condition rating of 38 and 7 joints /m then the weighted average of the joint conditions is 3 x 20 + 7 x 38 / 10 = 33. The joint spacing rating for 10 joints (two sets) is 13. Combining the joint condition and joint spacing ratings, one gets a total (combined) rating of 46. This is too high when compared with the 42 for one joint set. The addition of 7 joints must weaken the rock mass. Various procedures were tried to obtain a realistic average joint condition and it was found that the chart in Figure 8 gave the best results by using the highest and lowest ratings. Therefore, if the diagram in Figure 8 is used to average the joint condition ratings this results in 25 (JC) plus 13(JS) = 38 a more likely result when compared with 42 for one joint set.

ROCK MASS VALUES

In situ Rock Mass Rating

The *in situ* rock mass rating is defined as

IRMR = RBS rating plus Overall Joint rating - see Figure 1

Rock Mass Strength

The rock mass strength (RMS) in MPa is derived from the RBS - MPa after provision has been made for the effect of the overall joint rating, because, the strength of the rock mass must recognize the role of the joint spacing and the joint condition. This is shown in the flow sheet in Figure 1 and diagrammatically in Figure 9. The formula is based on the overall joint rating as a percentage of 75 times the RBS in MPa.

RMS = RBS MPa x Overall Joint Rating / 75

For example , assume that

Overall joint rating = 47
RBS value = 30 MPa

then

RMS = 30 x 47/75 = 19 MPa

MRMR ADJUSTMENT PROCEDURE

Introduction

The IRMR rating is multiplied by an adjustment factor to give the MRMR or Mining Rock Mass Rating. The adjustment procedure has been described in previous papers, where it was stated that the adjustment should not exceed two classes, but, what was not made clear is that that one adjustment can supersede another and that the total adjustment is not likely to be a multiplication of all the adjustments. For example, a bad blasting adjustment would apply in a low stress area but in a high stress area the damage from the stresses would exceed that of the blasting and the only adjustment would be the mining induced stress. The MRMR for a cavability assessment would not have blasting as an adjustment, nor would it have weathering as an adjustment unless the weathering effects were so rapid so as to exceed the rate of cave propagation as a result of the structural and stress effects. The joint orientation and mining induced stress adjustments tend to compliment each other. The object of the adjustments is for the geologist, rock mechanics engineer and planning engineer to adjust the IRMR so that the MRMR is a realistic number reflecting the rock mass strength for that mining situation. Whilst expert systems are useful the wide variety of features that have to be recognized in mine planning requires a degree of flexibility in assessing the situation. The complete dedication to computer generated results has lead to some major errors in the past, one must not remove the human thought process. It has been found that there is a better appreciation of the planning process and operation when personnel have to think in terms of adjustments.

Weathering

Certain rock types weather readily and this must be taken into consideration in terms of life and size of opening and the support design. In the case of fast weathering kimberlites, for example is it necessary to seal the rock surface. The weathering adjustment refers to the anticipated change in rock mass strength as the exposed surfaces and joint fillings are altered by the weathering process, it does not refer to the existing weathered state of the rock as that would be catered for by the IRS and then the RBS. The two items that are affected by weathering are the rock block strength - RBS and the joint condition - JC. The RBS is affected by weathering of fractures and veins and penetrative weathering of the intact rock. Borehole cores give a good indication of the weathering process, but, the results are conservative as the surface area of the core is high with respect to the volume of core. The weathering adjustment factors given in Table 2 cover known situations.

Table 2- Adjustments for weathering

Description	Potential weathering and % adjustments				
	6 months	1 year	2 years	3 years	4 + years
Fresh	100	100	100	100	100
Slightly	88	90	92	94	96
Moderately	82	84	86	88	90
Highly	70	72	74	76	78
Completely	54	56	58	60	62
Residual soil	30	32	34	36	38

Joint Orientation Adjustment

The shape, size and orientation of the excavation will influence the behaviour of the rock mass in terms rock block stability. The attitude of the joints with respect to the vertical axis of the block, the frictional properties of the joints and whether the bases of rock blocks are exposed have a considerable influence on stability and the RMR value must be adjusted accordingly. The magnitude of the adjustment is a function of the number of joints that dip away from the vertical and their frictional properties. Obviously a block with joints that dip at 60 ° is more likely to fail than one where the joints dip at 80°. Also the joint adjustment cannot be looked at in isolation as a low angle joint is liable to shear failure whereas the steep angle joint could be clamped. A computer program could be developed to cater for the variety of situations, but, would only be valid if there were sufficient checks along the way. The joint orientation adjustments in Table 3 has now been changed so as to reflect the influence of low friction surfaces as defined by the joint condition rating.

Table 3 - Joint adjustment factors

Number of joints defining the block	Number of faces inclined from vertical	Orientation % adjustments for ranges in joint condition		
		0- 15	**16 - 30**	**31 - 40**
3	3	70	80	95
	2	80	90	95
4	4	70	80	90
	3	75	80	95
	2	85	90	95
5	5	70	75	80
	4	75	80	85
	3	80	85	90
	2	85	90	95
	1	90	95	

The adjustment for the orientation of shear zones at an angle to the development is :-

0 - 15° = 76%,
16 - 45° = 84%,
46 - 75° = 92%.

Advance of ends in the direction of dip is preferable to against the dip as it is easier to support blocks with joints dipping in the direction of advance. An adjustment of 90% should be made when the advance is into the dip of a set (s) of joints.

Mining induced stresses.

Mining induced stresses are the redistribution of field or regional stresses as a result of the geometry and orientation of the excavations. The orientation, magnitude and ratio of the field stresses should be known either from stress measurements and /or stress analyses. If sufficiently high the maximum principle stress can cause spalling, the crushing of pillars, the deformation and plastic flow of soft zones and result in cave propagation. The deformation of soft zones leads to failure of hard zones at low stress levels. A compressive stress at a large angle to structures will increase the stability of the rock mass and inhibit caving and have an adjustment of 120%; this was the situation in a caving operation where the back was stable and caving only occurred when adjacent mining removed the high horizontal stress. Stresses at a low angle will result in shear failure and have an adjustment of 70%. The adjustment for high stresses that cause rock failure can be as low as 60%. A classic example of this was on a mine where the IRMR was 60 in the low stress area, but, the same rock mass in a high stress area was classified as having a IRMR of 40. The 40 is not the IRMR but the MRMR and the adjustment in this case is 40/60 = 67%.

The following factors should be considered in assessing the mining induced stresses: (a) drift induced stresses; (b) interaction of closely spaced drifts; (c) location of drifts / tunnels close to large stopes/excavations; (d) abutment stresses, particularly with respect to the direction of advance and orientation of field stresses - an undercut advancing towards the maximum stress ensures good caving but creates high abutment stresses and *vice versa;* (e) uplift as the undercut advances;(f) column loading from caved ground caused by poor draw control; (g) removal of restraint to sidewalls and apexes;(h) increase in mining area and changes in geometry; (i) massive wedge failures; (j)influence of structures not exposed in the excavation but creating the probability of high toe stresses or failures in the back and (k) presence of intrusives which might retain high stresses or shed stress into surrounding more competent rock. The total adjustment is from 60% to 120%

Blasting

Blasting creates new fractures and opens up existing fractures/joints generally decreasing the rock mass strength. Boring is considered to be the 100% standard in terms of the quality of the wall rock, but, experience on several mines has shown that whilst the rock mass might be stable at the face deterioration occurs ± 25m back and this is a stress relief adjustment. Good blasting can have the effect to allow for some stress relief thereby improving the stability. The adjustments given in Table 4 are recommended:-

Table 4 - Blasting adjustment factors

Technique	**Adjustment (%)**
Boring	100
Smooth wall blasting	97
Good conventional blasting	94
Poor blasting	80

Water/ice adjustment

Water will generally reduce the strength of the rock mass by reducing the RBS and friction across structures and reducing effective stress. The adjustment factors for water are given in Table 5.

Table 5 - Water adjustment factors

Moist	Moderate pressure - 1 - 5 MPa 25 - 125 l/m	High pressure - > 5 MPa > 125 l/m
95 - 90%	90 - 80%	80 - 70%

In the presence of ice in the permafrost areas the rock mass could be strengthened. This will depend on the amount of ice and on the temperature of the ice. Because of creep behaviour of ice the strength usually decreases with time. Adjustments will range from 100% to 120%

DESIGN RATINGS AND STRENGTHS

Design Rating MRMR

The mining rock mass rating (MRMR) is used for design. Thus the IRMR value as adjusted for weathering, orientation, induced stresses, blasting and water.

MRMR = IRMR x adjustment factors

Design Rock Mass Strength

The design rock mass strength (DRMS) is the RMS reduced by the same adjustment factor relating the IRMR to MRMR. In the case where the

RMS	=	40 MPa
IRMR	=	50
MRMR	=	40

then the design rock mass strength would be

DRMS = RMS x MRMR/IRMR = 40/50 x 40 = 32 MPa.

PRESENTATION

The IRMR data should be plotted on plans and sections. The range of 0 - 100 covers all variations in jointed rock masses from very poor to very good. The classification is divided in to five classes of 20 rating with A and B sub-divisions of 10 points. A colour scheme is used to denote the classes on plan with full colour for the A sub-division and cross-hatched for the B sub-division. The colours are:

Class 1	**Class 2**	**Class 3**	**Class 4**	**Class 5**
100 – 81	80 - 61	60 - 41	40 - 21	20 - 0
Blue	Green	Yellow	Red	Brown

It is essential that classification data is made available at an early stage so that the correct decisions can be made on mining method, layout and support design.

It must be stressed that every attempt should be made to zone the rock mass, averaging a large range in ratings leads to planning and eventually production problems.

SCALE EFFECTS

Scale is a very important factor in considering the behaviour of the rock mass particularly when examining mass mining methods. For example, the stability/cavability of a deposit cannot be based on the MRMR from a drift assessment alone, as, widely spaced major structures play a significant role. That is, structures at 10m spacing would have a marginal effect on the overall IRMR value obtained from drift mapping, but, have a large influence on the cavability of an orebody by providing planes along displacements can occur. The MRMR value gives a hydraulic radius (HR) for assuring cavability This figure should be adjusted for the influence of major structures by using the following procedure to obtain an 'influence' number. The various factors that contribute to the 'weakness' of a major structure have been ranked as follows :

Rankings

A - **Dip** : 0º - 20º = 6, 21º - 40º = 4, 31º - 40º = 2, 41º - 60º = 1, > 61º = 0

B - **Spacing**: 0 - 9m = 6, 10 - 15m = 4, 16 - 21m = 3, 22 - 27m = 1 > 27m = 0

C - **Joint Condition**: 0-10 = 6, 10 - 15 = 4, 15 - 20 = 2, 20 - 25 = 1, > 25 = 0

D - **Stress / structure orientation**: 0º - 20º = 7, 21º - 30º = 9, 31º - 40º = 6, 41º - 50º = 3, 51º - 60º = 2, 61º - 70º = 1, > 71º = 0

E - **Distance of major structures from undercut boundaries** : 0 - 9m = 12; 10 - 20m = 8; 21 - 30m = 2; > 31m = 0.

F - **Stress values - Sigma 1 as % of RMS**: > 100% = 14, 80% - 99% = 12, 60 - 79% = 8, 40- 59% = 4, 20 - 39% = 2, < 20% = 0

The rankings are plotted in Table 7. - the highest likely ranking from the three sets is in the order of 100.

Table 7 - Form for determining the major structure influence number

Major Structures	A	B	C	D	E	F	Total
Set 1							
Set 2							
Set 3							
Total							

The total of the rankings when plotted in Figure 10 will indicate whether the HR is acceptable or whether it should be adjusted up or down.

If there are other features, such as internal silicified zones, that might contribute to stability then a deduction should be made. The magnitude of the deduction should be 15% to 40% of the hydraulic radius.

In the case of pit slopes the MRMR of the bench and the overall slope will vary as shown in Figure 11. The MRMR of the zone in which the benches are cut could be affected by joint orientation, blasting, induced stresses and even weathering. The overall slope angle would be based mainly on the IRMR / RMS values with the MRMR adjustments based on induced stresses as a result of depth and shape of the pit. The presence of major structures could be the significant factor.

PRACTICAL APPLICATIONS

The details of the practical applications can be found in the paper " Planning Mass Mining Operations" (Laubscher 1993). A summary of the applications are: (a) support design (Laubscher 1984); (b) cavability diagrams; (c) stability of open stopes; (d) pillar design; (e) determining cavability; (f) extent of cave and failure zones; (g) caving fragmentation; (h) mining sequence; (i) potential massive wedge failure.

CONCLUSIONS

The object of this paper is to show the changes that have been made to the original MRMR classification system. It must be stressed that where the system is properly applied the results are good. Unfortunately, rock masses do not conform to an ideal pattern and therefore a certain amount of judgment/interpretation is required. A classification system can give the guidelines, but the geologist/engineer must interpret the finer details as has been indicated. It is important that the rock mass be divided into zones in which there is not a great range in ratings and this can only be done if the geologist has an overall understanding of the rock mass.

As it is not possible to precisely define every mining situation, the engineer must use his judgment in arriving at the adjustment percentage. Where the system has been properly applied it has proved to be successful in planning and as a communications tool. However, what has been found on several mines is that the geology departments dabble in different systems and at the end of the day are masters of none. The numbers produced are incorrect because the personnel do not have a feel for the rock mass. The modern tendency, unfortunately, is to have programs that do the thinking for the operator.

ACKNOWLEDGMENTS

A classification system can only develop if it is used and tuned to cater for the many different situations that are encountered in designing mining operations. The writers wish to acknowledge the constructive suggestions from those people who are using the MRMR system and in particular the constructive suggestions from T.G.Heslop.

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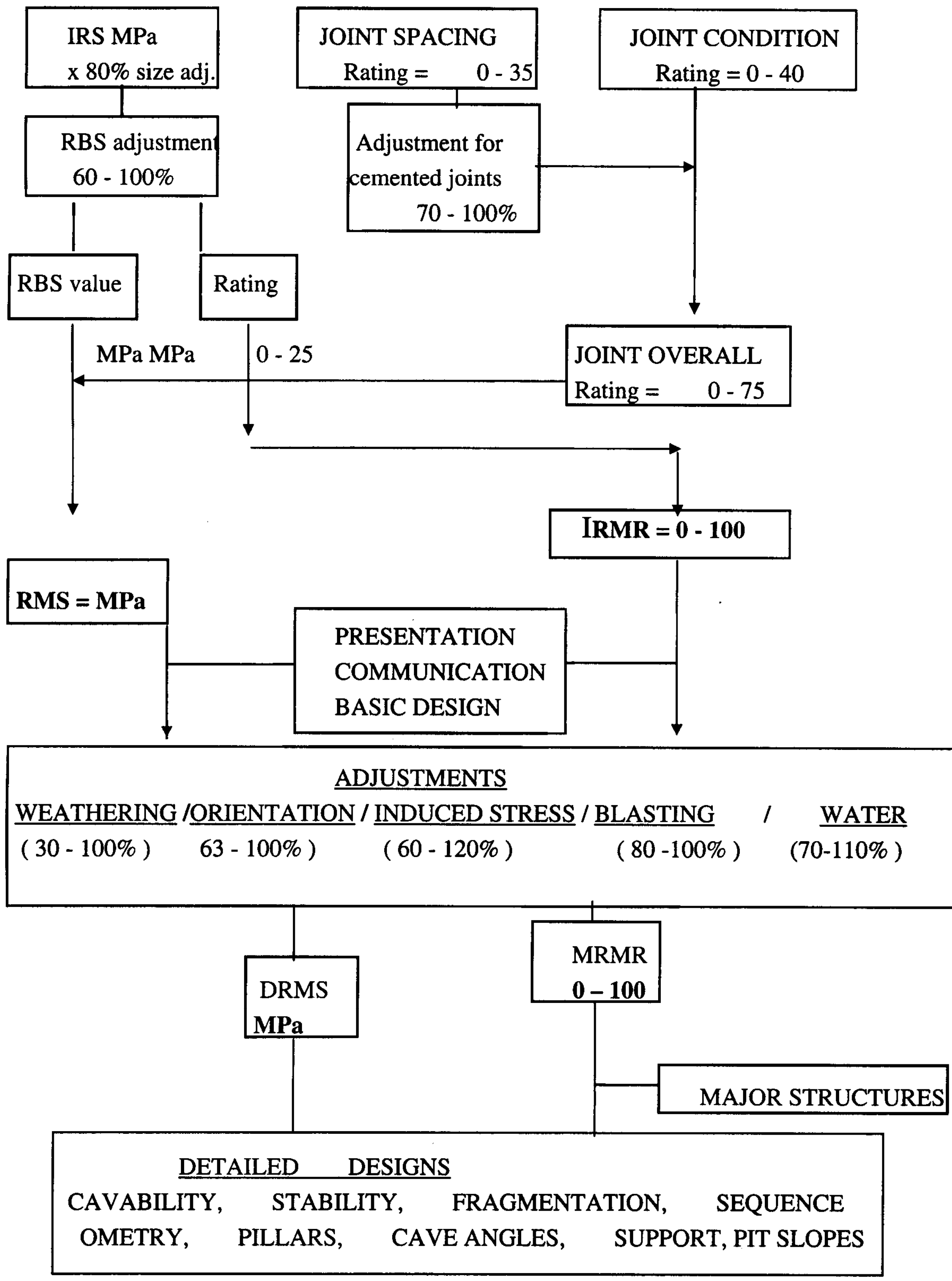
Figure 1 - Flow sheet of the MRMR procedure with recent modifications



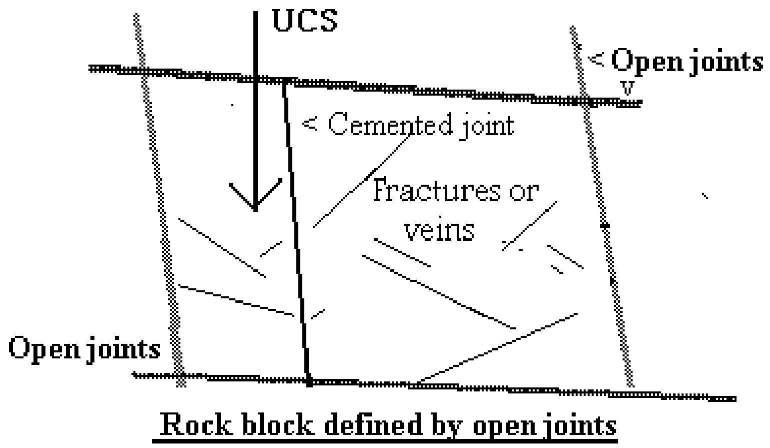


Figure 2. Definition of structural terms

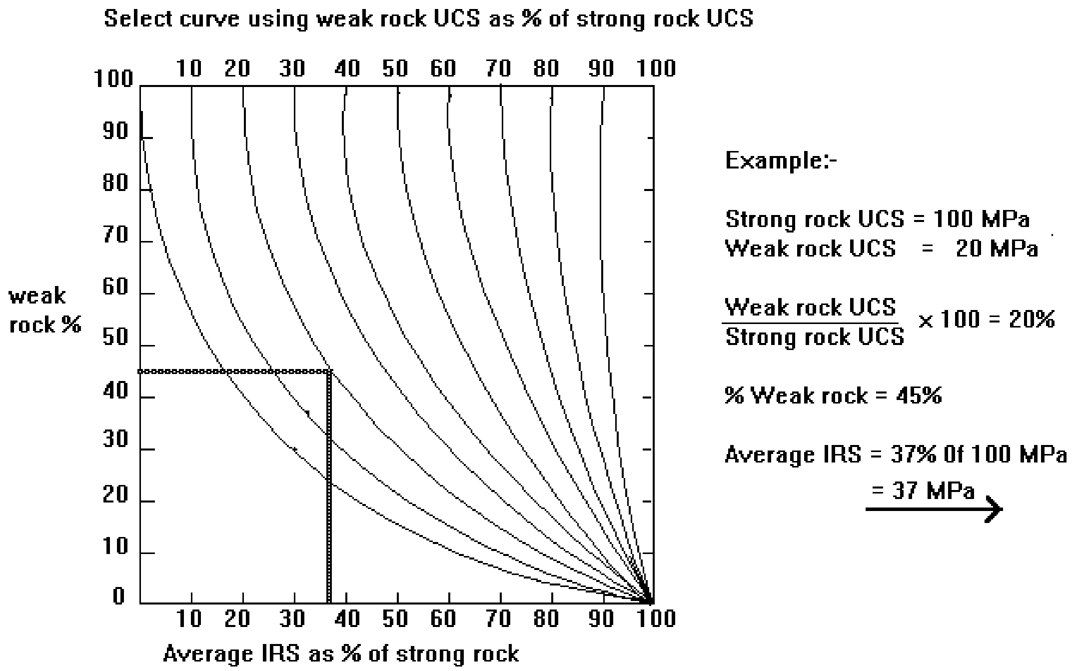


Figure 3. Nomogram for determining the "corrected" IRS value

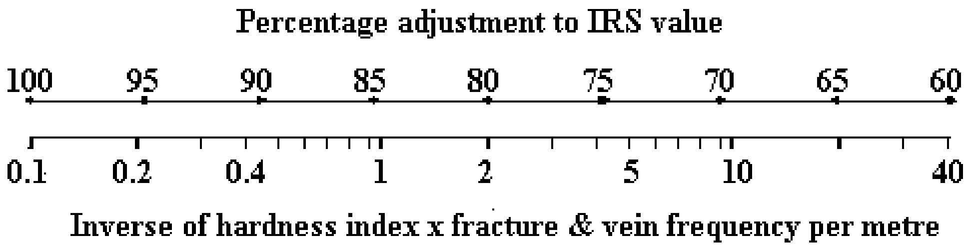


Figure 4. Nomogram for relating the IRS adjustment factor to the hardness index and fracture / vein frequency.

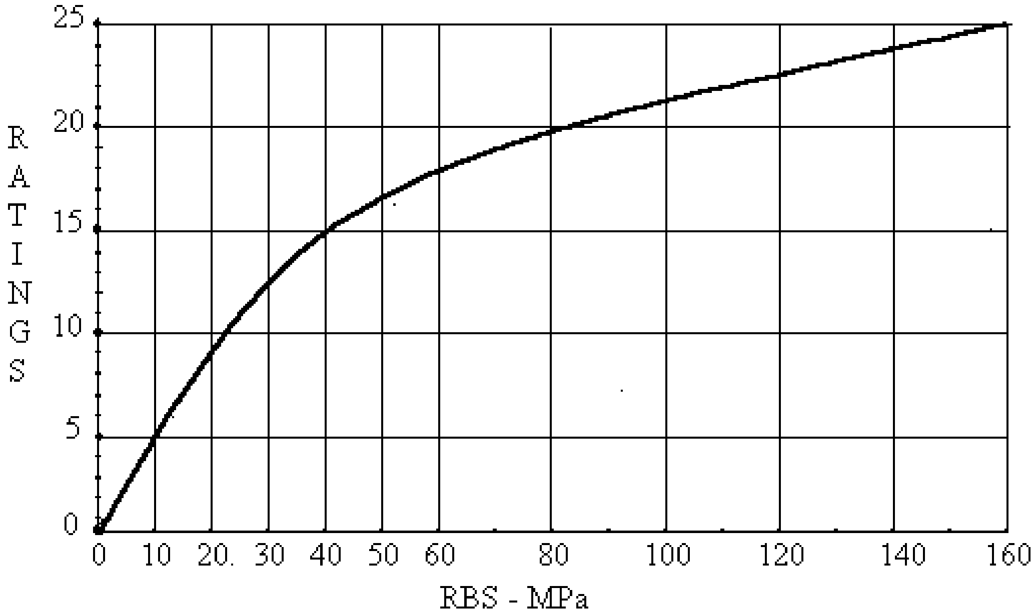


Figure 5. Rock block strength rating as a function of rock block strength.

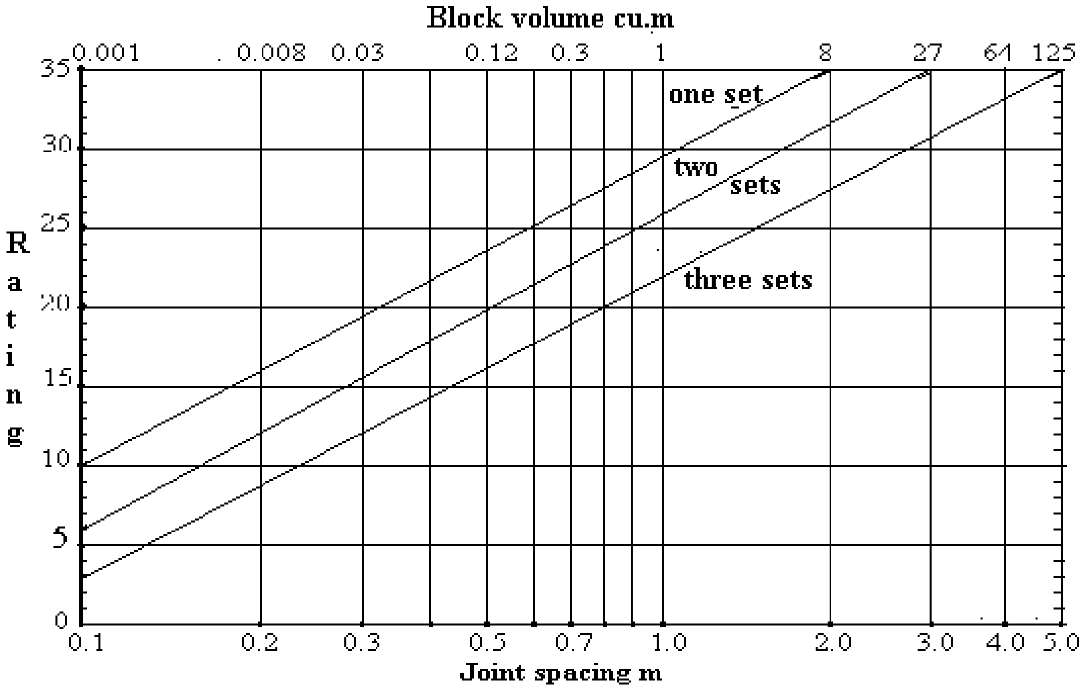


Figure 6. Joint spacing ratings

% Adjustment to the open joint spacing rating for one or two filled joints in a three joint set

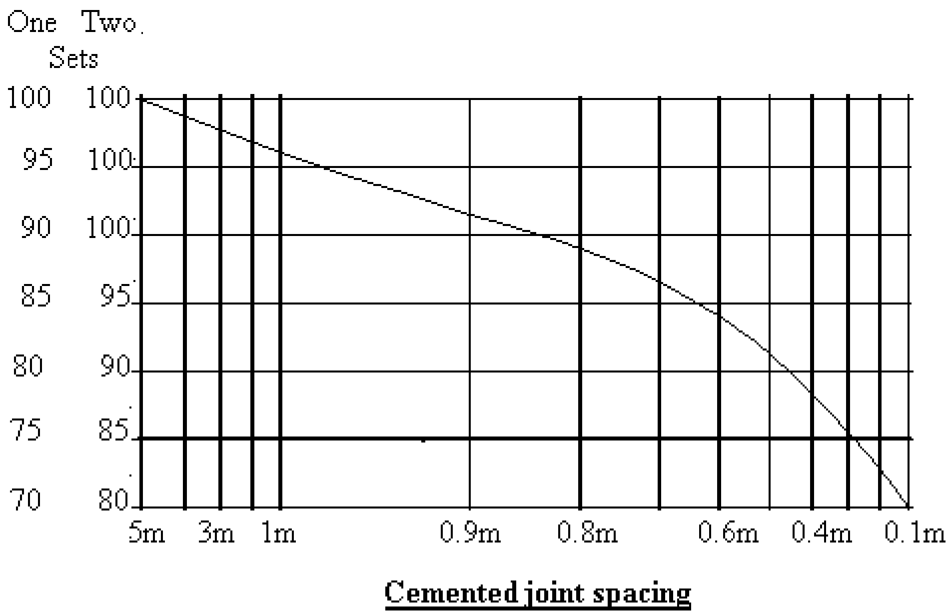


Figure 7. Adjustment factor for cemented joints

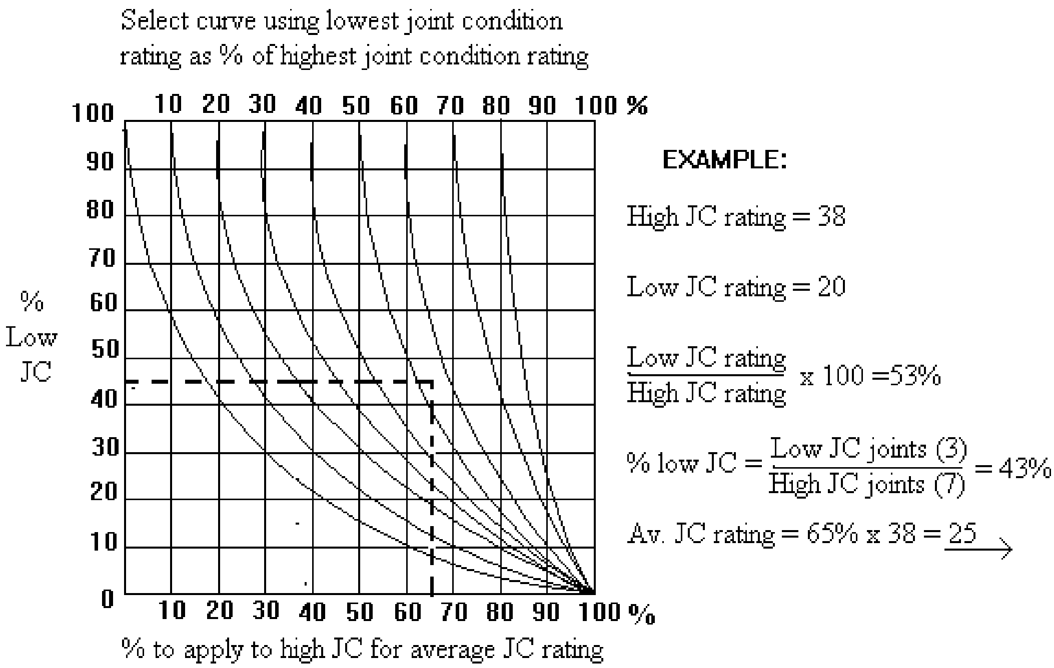


Figure 8. Chart for averaging the joint condition for multiple joint sets.

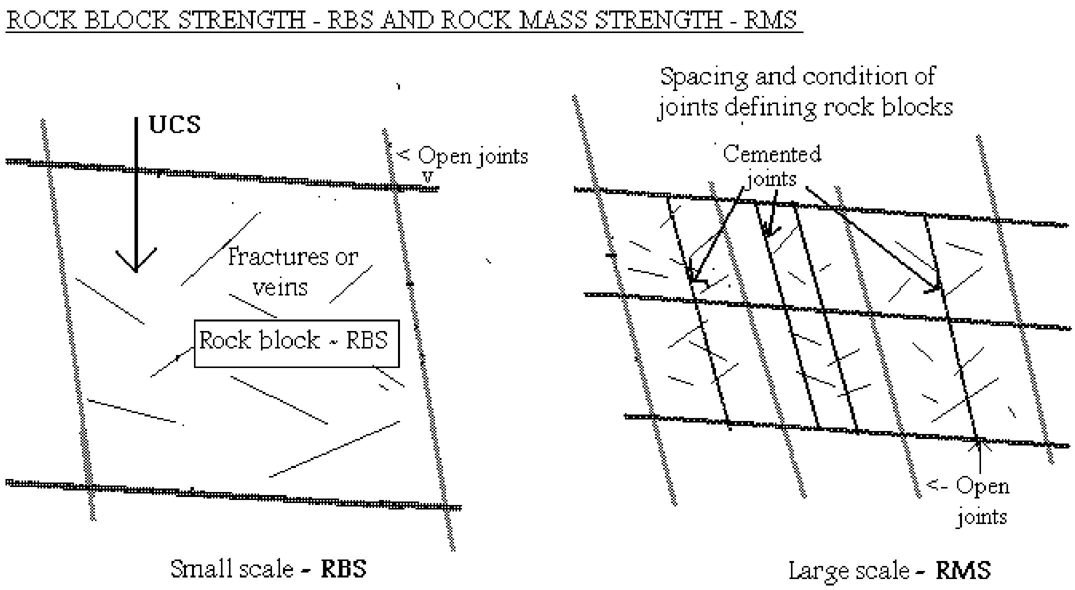


Figure 9. Diagrammatic representation of small scale RBS and large scale RMS

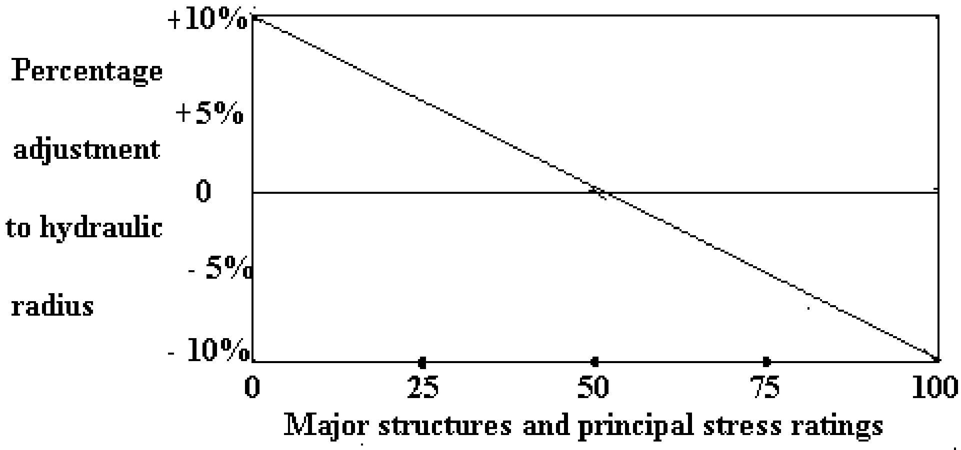


Figure 10. Adjustment factors for the effect of major structures on cavability

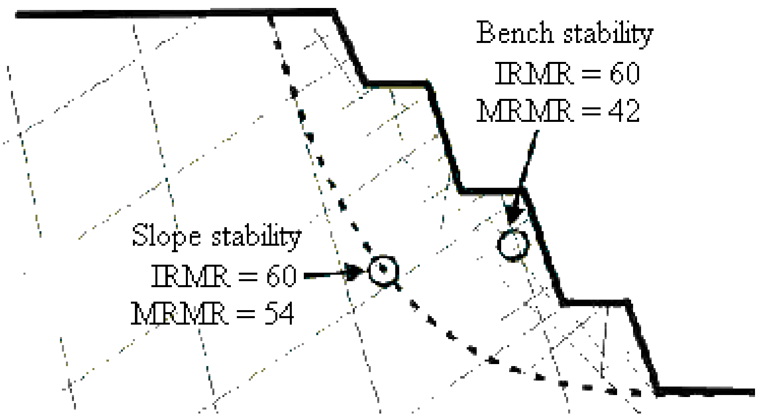


Figure 11. The difference in MRMR values used for bench and slope design.