CAVE MINING HANDBOOK

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Chapter 1

INTRODUCTION

The object of this handbook is to highlight factors which lead to problems in cave mining and to summarize the contents of the “Manual on Block Caving” so as to provide a reference on all facets of cave mining. The same format as the revised manual has been used so that the details of the subject can be referred to in the manual. The manual has been edited so that there is easy reference with all page numbered in sequence.

It is apparent that whilst facets of the design might be correct it is often the installation and application that is at fault. This can be due to lack of understanding of the consequences, poor supervision or lack of interest and production expediency. It is clear that there must be a complete understanding of the process by all involved. Geological and geotechnical investigations provide the data on which the design is based. For example, management’s desire for large equipment cannot be met if the design size of drifts stipulates a smaller size. Production calls are based on the fragmentation data and calculated draw rates. This will ensure correct draw control for optimum ore extraction with minimum dilution and minimum damage to the extraction horizon. The consequences of overdraw in certain areas is often not immediately apparent, but manifests itself in early dilution entry or column loading, both dire consequences. The anticipated rock mass response to the cave mining operation is of great importance in designing the operation, particularly as mining proceeds to greater depths and only high draw columns are economical to mine.

Mining sequences must be designed for the life of the operation and not for short term expediency; the recovery of capital expenditure is a long term process. Initial high support costs to ensure a smooth long term operation is far better economic sense than having high working costs and low productivity as result of cutting capital costs and poor installation procedures. Man made problems can be the major cause of mining problems it is often not the design that is at fault.

What is most apparent is that in many cases the lack of strong management leads to a lack of direction and major problems. At what stage management becomes involved will vary. On operating mines where the object is to bring in new sections mine management must be familiar with the operation from day one. In grass roots operations mine management might only become involved when it is apparent that mining will proceed and management staff are being selected. However, there has to be management of the operation from day one and it is important that those in the management position are familiar with all the investigation requirements to proceed from a mineralized zone to an operating mine so as to ensure that all the necessary data is gathered at an early stage at the lowest cost.

Drill and blast mining methods permit a degree of flexibility during the course of mining and allow for changes in techniques with technical improvements in equipment and blasting techniques. Block cave mining allows very little freedom in change once the layout in complete or virtually once the design has been approved.

During production the only control is through the drawpoint. There is no room for the philosophy that ‘well maybe it will work’, sound planning, honesty, three dimensional thinking and an open mind are required to ensure a successful operation. It has been noted over the years that people experienced in one facet of block cave mining often insist on using those techniques in a totally different environment and this can lead to problems, each new lift or deposit must be fully assessed. One of the most important aspects of cave mining is draw control, but often management only pay lip service to it and this results in abuses down the line. During the production stage, draw control plans must be adhered to and production calls decreased if necessary.
Management need to create a project philosophy conducive to work of quality and within the time frame. Policies must be set down to ensure that realistic standards are established for each phase of the operation from investigation to production. It is important that procedures are laid down for contractors so that there is no confusion on the required standards, after all the contractors are on site to do specific work. The handbook will emphasize that there cannot be departures from production schedules for short-term expediencies. Successful planning and mine design occurs when all personnel contribute and all aspects are studied and there is a response to pointers that do not conform to engineering judgement.

The object of the handbook is to highlight the important issues in:-

* investigating a potential block cave deposit,
* the planning of the operation,
* the design of the method,
* the operational aspects,
* the role of management in ensuring a successful operation.

Reference will be made to detail and figures and plates in the ‘Block Caving Manual’ by showing the chapter number and the page in that chapter, e.g. (Ch5, p 12)
Chapter 2

GEOLOGICAL INVESTIGATIONS

1.0 GENERAL

Geological investigations are ongoing for the life of the operation from grass roots to the completion of the operation. The object of the investigation is to provide the input data so as to decide on the course of action in planning the operation and how to manage the operation as well as increase one’s knowledge of block caving. It is a reiterative process.

The first stage in an investigation is to derive sufficient data to undertake a conceptual study which will result in one or more mining options. A mining option is a distinct mining method. No conceptual studies should be undertaken unless there is sufficient information available to make a serious selection of a method(s). A study of the handbook will show what aspects are required to be covered. In a large orebody with a strike length of 2000m, a width of 300m and height of 400m and an apparent uniform grade distribution, boreholes at 100m spacing will provide ample information to conduct a conceptual study. However, if the dimensions were decreased to 400m strike and 200m width and 200m height then boreholes at 50m spacing along strike would be required with holes at the extremities, if the grade distribution was erratic then more boreholes would be required.

The start of a geological investigation lays with the exploration department personnel who, often in the past had little interest in the likely mining method. Hopefully this has now changed. Because of the drilling time and drilling costs, it is important that full use is made of all available data. Small modifications to an exploration program can often lead to later significant benefits, for example, holes should be extended to cover all the likely peripheral geology and not only the orebody. This means that the mining geologist should be contributing to the exploration program at an early stage. All cores must be photographed and examined in detail with emphasis on structural geology and rock mass classification. How often have we not looked at borehole logs or geological sections and said why wasn’t that hole taken another 50m.

The geological investigation provides the regional picture with the preparation of both small and large scale plans and cross and longitudinal sections. The large scale must include surface. A 3D computer presentation is also useful and in the past 3D solid models have been used and proved to be extremely useful, particularly if the surface is shown, this makes it must easier to explain important points to an audience. Whilst it is often convenient to look at plans and sections on computer screens and to flip through them, the significance of certain features is often missed – hard copies must be available for detailed study. Data plotted on a hard copy makes a greater impact than feeding data into a computer. The object is to gather data which will be used to plan the mining method. The Geologist / Technician must always consider the end result, this means that they must be familiar with block caving planning so that the presentation is relevant to operation. Defining zones of different structural, densities and chemical patterns is equally as important as lithological changes.
Rock mass classification data is collected at this stage and it is essential that structures are classified and the classification details recorded. For mining situations the IRMR / MRMR system has proved to be suitable and is described in the rock mass classification section. The potential dilution zones must be investigated in detail and properly valued.

All relevant geological data in the peripheral zone (hangingwall, side and below) must be plotted on plans and sections and must cover the area surrounding the orebody beyond the subsidence and failure zones. Permanent infrastructure will be sited in the peripheral zone beyond the defined failure zone. The need for the geologist to have a 3D mental picture of the relevant rock mass cannot be sufficiently emphasized. It is also important to realise the role that Geological Technicians can play in gathering data as the experience on the asbestos mines of Zimbabwe has proved. This is of particular importance nowadays when Geologists only seem prepared to put in the minimum time on field work, being more concerned with computer programs - which of course are only as good as the person who designed the program and the quality of the input data. The mining geologist in particular must be involved from the time that the mineralized zone becomes an orebody throughout the planning and production stage. It is only by having continuous observations of the response of the rock mass that production programs can be adjusted, before major problems occur.

It is important that the deposit is viewed as a whole and not as a series of little windows. It is this area that mining geologist, who is familiar with mining methods can express himself in terms of his knowledge of the deposit and ability to identify areas where additional work is required in order to minimise the risk.

2.1 ROCK TYPES

This is a detailed description of the rock types in the orebody, peripheral zones and the hangingwall zone to surface, 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. Zones of different density must be identified as caving relies on gravity for material to move. Anything that will show a difference in the rock mass in its failed state must be included. Variations in modulus often result in rockbursts at the contacts in high stress areas or 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.

2.2 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 as more competent intrusives can be stress attractors and their contacts can become rock burst sites. Intrusives should have their own IRMR. Descriptions of the contacts are required, are they sheared or ‘frozen’? The hangingwall might contain sills, which could inhibit the propagation of the cave.
3.0 MINERALISATION, MINERAL AND GRADE DISTRIBUTION

In the orebody does the mineral occur in veins or is it disseminated? Do the veins have continuity and can they be classed as joints or are they fractures, do they have a bearing on the strength of the rock mass? Are the veins weak so that the mineral reports in the fines or is the mineral in weak zones that will form the fines? Is the mineralization boundary a sharp contact or gradational?

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 or weak mineralised zones. 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, are these higher density zones? This is important as the fines flow through coarse rock and therefore the mineral in the dilution zone fines could up-grade the ore. How extensive is the mineralization in the peripheral rocks and is it zoned or disseminated. This information is important as it might provide economic justification to increase the undercut area to induce caving, that is, provide sufficient revenue to pay for the operation. Will the mineral(s) form toxic or corrosive substances?

4.1 MAJOR STRUCTURES

All major structures in the orebody and peripheral zones must be identified. The IRMR of the structures to be determined and plotted on plans and sections with the break down in brackets e.g. IRMR 20(4.8.8). Major structures influence cave angles and also the angle of draw zones, particularly if the structures are shear zones.

4.2 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. Every effort should be made to distinguish between the two. Without underground exposures, 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 with gypsum filled features it might be necessary to take a ‘flyer’ and assume that one third are joints. Local techniques must be developed.

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.
4.3  STRUCTURAL ZONES

It is important that any variation in joint/fracture spacing is noted and 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 caveability assessments, in fragmentation analyses and draw control calculations where highly fractured zones might be high or low grade. 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 caveability of the hangingwall zone.

5.0  OREBODY - SHAPE, DIMENSIONS, TONNAGE, DIP AND STRIKE

This section will describe the basis for defining the orebody outlines whether it is economic, stratigraphic or structural. The economic ore body outlines are relevant to the mineral price at that particular time. It is therefore important to show outlines for decreasing value zones. For example, the current cut-off grade might be 1.0% at a dollar value; this could change so that 0.8% could have the same dollar values. By judicious zoning outlines can be changed to suit mineral prices.

There must be a laid down policy in calculating the outline. High value ore blobs in the hangingwall separated by unpay zones would become part of the orebody if the overall grade in a vertical direction exceeded the cut off value.

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. Interpret the section-to-section difference in outlines of the ‘ore’ and marginal ore zones. Check for and explain anomalies especially when data is computer generated and use hard copies for analysis for presentation of the data. Ensure that all data is checked and cross referenced.

Tonnages of high grade, medium grade, low grade and marginal mineralized zones must be calculated and updated as data becomes available. 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. Ensure that all data is correctly filed and stored.

6.0  PRESENTATION OF DATA

Identify areas requiring detailed investigation as early as possible and review the needs as data is gathered. Hard copy accurately 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.
7.0  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. This will provide all the previously mentioned 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 cores 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. Colour prints are useful as well so it will be a case of personal preference.

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 rubble” 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. They could also 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.
Chapter 3

ROCK MASS CLASSIFICATION

1.0 GENERAL

It is important to decide on which rock mass classification system is going to be used and then persist with it so as to develop expertise in the system. This is important as one cannot pay lip service to a classification system. As it not only defines the rock mass, but is also a means of communication between geological, planning, operating and managerial personnel. So it is most important that all personnel are familiar with the system. The simpler the system, the more useful it will be, for example, 0 – 100 is easier to relate to than a log scale with a range of 0.01 – 1000.

The MRMR system has a proven track record on the mines where it has been correctly employed. The IRMR ratings of 0 -100 and five classes is simple to understand and covers all variations in the rock mass from very poor to very good. IRMR data must be shown in as much detail as is practical as differences and interrelationships can be important. Do not average out ratings as this can be misleading, ratings must be zoned and the average value applied to the zone. Always highlight the dominant feature i.e. is it joint condition or fracture frequency as this might influence the orientation of drifts and what support is required.

Plans must be produced showing the IRMR as these represent what operating personnel can see. The plans showing MRMR show what is expected with the mining operation and the comparison between the two highlights potential problem areas. The object is to ensure that all personnel, both technical and operational, understand the system and nomenclature to a level (preferably higher) than they need so that they become familiar with that rock mass. The rock mass must be examined at regular intervals during all stages of mining to ensure that the MRMR adjustments are correct.

2.0 ADJUSTMENTS

The MRMR adjustments are simple to implement if the engineer thinks about the processes to which the rock mass will be subjected. A document should be produced for that particular operation showing the methodology and adjustments. Keep the MRMR data current and reviewed regularly (conditions do change), making sure that the records of the adjustments made initially and with each subsequent change are maintained and filed for posterity and review. Physical models showing a 3D presentation of structures with IRMR data are useful in conveying the significance of various structures and their influence on sequence, caving and support. Two different sets of structures might have the same rating with a difference in fl/m and joint condition; therefore the individual items must be shown.

**Only those adjustments that are relevant to that particular investigation are used.** It is not a case of multiplying out all the adjustments. The rock mass might weather with time and have a 90% adjustment and the development adjustment might be 90%, but if the caveability is being assessed these adjustment are not relevant in terms of space and time and are therefore ignored.

3.0 MRMR / IRMR PROCEDURE

The procedure to calculate the MRMR and the IRMR is summarised in the following Table:-
The details of the system are found in Chapter 2 p 1-20 in the Cave Manual.
Chapter 4

GEOTECHNICAL INVESTIGATIONS

1.0 GENERAL

Geotechnical investigations cover all rock mechanics investigations leading to a method selection and should continue during the mining operation. By observing rock mass response, the remedial strategy can be taken if 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 proposed monitoring programs once the method has been selected and here caution must be exercised in the selection of techniques. The KISS principle applies simple and cost effective procedures so that results are readily available. Also numerical modelling recommendations might be required at a later stage to assist in the planning of the sequence and to highlight potential areas of damage.

2.0 STRESS MEASURING TECHNIQUES

Regional stresses are usually obtained from in situ stress measuring techniques by overcoring if underground access is available. Hydrofracturing can be done in boreholes at depth. 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 breakouts 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.

3.0 INDUCED STRESS PREDICTION

During the conceptual planning stage 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, common sense will clearly show areas of high stress. This is obviously a reiterative 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 is abutment stresses. The magnitude and damage effects of abutment stresses are well known and seem to be accepted as part of a cave mining operation. The damage caused by abutment stresses is 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.
4.0 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. In other cases they can give rise to massive wedge failures, promote unfavourable draw so that there is early dilution entry, and 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 sufficient height so as to impose large enough caving / arching stresses to break these rock blocks. However, the mass of the overlying ground will be carried by the pillars on which the large blocks are resting, there will be a load transference until the block is broken. If 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.

5.0 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 in a large orebody can come from part of a level for six years and from the remainder of the level 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. 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 phenomenon 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.

6.0 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:

I. poor working environment
II. poor hauling conditions
III. risk of mud rush
IV. washing out of fines
V. rapid wear of roadways
VI. excess equipment wear (tyres and rust)
VII. support damage (rust)
VIII. problems in orepasses and loading bins

Dewatering programs should be designed and implemented at an early stage so as to provide good working conditions in the production area. Layouts to be designed to handle water in the most efficient manner, water problems should not come as a surprise once mining has started. 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 can be removed on the upper two levels, or the 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 (Ch3 p5). Water balance calculations (inflow / mine dewatering) should be completed on a regular basis to indicate potential water accumulation and subsequent catastrophic discharge.
7.0 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 still being extensively drilled.

8.0 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 minimize 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. **Do not ignore the need for ongoing observations to be made by all technical and operating personnel, this aspect can be missed with the tendency to mechanization.**
Chapter 5

MINING LIMITS

1.0 GENERAL

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. The mining limits could vary depending on the pay limits used and will have outlines for one or more pay limits. The mining limits are derived from data on:-

- Grade distribution
- Cut-off grade
- Orebody shape and dimensions
- Dip and strike
- Plan area
- Potential ore column heights
- Draw column heights
- Rock mass classification data
- Major structures

DRAW COLUMN HEIGHT - Draw column heights are calculated from the height of ore above the drawpoint plus an acceptable height of dilution. The columns are generally vertical, but can be inclined if there is a significant variation in the topography of the caved ground. Draw columns will angle towards the high ground as occurred on King Mine.

PAY LIMITS - Different pay limits might be defined as was done on Shabanie and Gaths mines in Zimbabwe with great success. On these mines three pay limits were used: the all-in pay limit, the working cost pay limit and the draw pay limit. The all in pay limit would define an orebody, where all the block values in a vertical column exceeded a value required to meet capital, working costs and a profit margin. The working cost limit was based on working costs only and a profit and a draw pay limit would break even. This means that at this early stage an estimate must be made of the likely operating and capital costs. The mining limits are a function of grade as well as the tonnage and potential draw angle above a drawpoint. The mining limits could be as shown on the next page.

Draw points would also be located in the footwall of the $8 –10 zone - the working cost pay limit. The cut-off grade or all -in mining limit 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 on which the conceptual planning is done. Once the viable all-in mining limits have been defined, extra drawpoints might be developed where they only have to carry development, support and maintenance costs, provided the average grade meets the planned economic returns.
MINE PLANNING - A realistic mining limit is required at an early stage in mine planning so that the study can proceed on a sound basis of outline and draw heights and there must be no doubt about the economic viability of the project. As certain items become apparent with detailed analyses there will be some variation as the design develops before the limit is finally decided. 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. This would promote caving through a more competent zone or in tight corners which might create overhangs.

A correct assessment of the grade distribution will result in the correct decisions being made on the mining limits. Bear in mind that a block cave layout lends itself to overdraw in the final stages, particularly if this will defer or reduce capital expenditure. Therefore higher grade zones in the hangingwall must be show as these could warrant overdraw in certain areas before the block is abandoned.

It is necessary 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.

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.

PERIPHERAL DRAWPOINTS – Low grade peripheral drawpoints might be necessary to improve caveability or to straighten the outline for optimum ore recovery.
Chapter 6

CAVEABILITY

1.0 GENERAL

Caveability 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 for that fragmentation. 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 caveability. Guidelines are provided to place the deposit in the correct ‘ball park’, but the final decision must be based on a close examination of the following factors:-.

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

Boundary conditions are very important and competent zones must be viewed with suspicion whether they are internal or whether they form the boundary. A good example of how necessary it is to examine all factors is the cessation of caving at Northparkes mine. The IMRM of the Northparkes deposit based on borehole results indicated that for a footprint with a HR of 45 that caving should not be a problem. What was not available at the time was:-

- The decision by management to do a pre-break for the lower 60m, this meant that the correct sequence could not be set up to ensure maximum use of regional stresses.
- That the structural pattern on the west side was different from the east side and that there would be clamping of the steep structures on the west side from horizontal stresses.
- That the central silicified zone was a dominant feature and had a higher IRMR.
- That an open pit would be mined and that this would remove a large mass of rock required to induce caving.

2.0 FACTORS INFLUENCING CAVEABILITY

REGIONAL STRESSES - The magnitude and orientation of the regional stress plays a significant role in caving. Undercutting towards the principal stress will improve the caveability and fragmentation, but could cause squeezing damage or rockbursts. Developing away from the principal stress is advisable in the case of weak ground. The orientation of the
Principal stress on the sides or the back of the cave opening can be significant. Large horizontal stress acting on a long face would lead to failure whereas the same stress acting on a circular cave could have a stabilizing effect. There are several examples of how horizontal stresses have clamped dominant structures thereby inhibiting caving, for example Shabani Mine and Northparkes Mine (Ch 5 p3).

INDUCED STRESSES IN THE CAVE BACK - It is important that the stresses in the cave back and sides (as the cave progresses) are calculated for different heights. 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 the caved zone, variation in rock types and proximity to previously mined areas. The stresses in the sides and back of the cave zone 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.

Principal horizontal stresses clamping vertical joints will inhibit caving. These stresses do not have to be of large magnitude. 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 and lateral extent of caving. Average values are fine for initial assessments, but, can be misleading if there is large range in IRMR and there are large areas of high IRMR 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 IRMR ratings, the onset of caving will be based on the lower rating zones if these are continuous in plan and section. This data will show if there are changes in the rock mass and all 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 and orientation of the lower ratings that will determine the size of the undercut. Any abnormal features that might impact on the caveability 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.

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.

MAJOR STRUCTURES - Major structures have to have sufficient continuity so that they will influence the caveability 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 generally not as important as dipping structures. However, weak shear zones can deform setting up tensile stresses in the boundary rocks, numerous examples of this can be seen on chrysotile asbestos mines. The orientation and dip can influence the direction of undercutting. The empirical Laubscher hydraulic radius graph provides the operator with a ‘ball park’ figure on the caveability 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.
structures can be the determining factor in assessing caveability, 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 caveability, 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 caveability have been identified (Ch 5 p13, 14).

MINOR 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. 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.

3.0 HYDRAULIC RADIUS

There will be continued reference to hydraulic radius - HR - in this section and therefore a description is in order. It has been noted that the term ‘caving radius’ has been used by another author. 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, 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 for example (Ch 5 p 2):-

<table>
<thead>
<tr>
<th>Area</th>
<th>HR</th>
</tr>
</thead>
<tbody>
<tr>
<td>100m x 100m</td>
<td>10000 m² and perimeter of 400m with HR of 10000/400 = 25</td>
</tr>
<tr>
<td>200m x 50m</td>
<td>10000m² and perimeter of 500m with HR of 10000/500 = 20</td>
</tr>
<tr>
<td>400m x 25m</td>
<td>10000m² and perimeter of 850m with HR of 10000/850 = 12</td>
</tr>
</tbody>
</table>

The maximum area for the minimum perimeter will be achieved with a circle and then a square. The minimum span is a critical dimension in promoting caving and 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 a small increase in the minimum span will have a significant influence on the hydraulic radius for example, an area of 40m x 200m has a H.R. = 17, increasing the minimum span by 10m to 50m then the H.R. = 20 and caving could be ensured. 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!

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 occurred on Northparkes mine. (Ch5 p12) The hydraulic radius on the base cannot be applied to weaker rock higher up if an overhang has formed. There are many examples of permanent overhangs with continued caving to the side owing to a weaker rockmass in that area. (Ch 5 p11) 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.
GEOMETRY OF PROPOSED CAVE - The hydraulic radius recognizes variation in geometry particularly with respect to minimum span and will give the highest HR for a circle. However, a circle has ‘hoop’ stresses which increase the stability of an excavation with uniform stresses. Where there are high horizontal stresses the ‘hoop stresses’ are cancelled which results in instability. The effect of high horizontal stresses is more pronounced on the longer face of a rectangular shape, see following diagram.

An equi-dimensional shape will be more stable than a rectangular shape owing to the ‘hoop’ stresses, particularly in a horizontal stress environment. The empirical caveability diagram of MRMR vs. HR should make provision for a shape factor which will provide for overhangs that can form in the corners, thereby changing a square into a circle. 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 the hydraulic radius and would, in fact sound an immediate warning. The ‘equi-dimensional shape factor should be a ratio of $1 \pm 0.30\%$.

The following diagram shows two curves – curve A for rectangular orebodies and curve B for equi-dimensional orebodies.
4.0 CAVE PROPAGATION - VERTICAL OR LATERAL EXTENSION

At the start of a block caving operation the cave will propagate vertically, while 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.
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 advancing face of 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.

5.0 FACE SHAPE AND UNDERCUT DIRECTION

A concave face confines the rock mass behind the face whilst a convex face allows relaxation (Ch 5 p 17). It is generally good mining practice to mine from weak to strong rock in certain caving situations it might not be advisable. The rapid caving of the weaker rock might leave strong rock in the orebody boundary because the induced stresses are not high enough to induce caving. In this case it is preferable to start the undercutting in the strong rock as 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. It is worth noting that 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 from strong to weak rock, caving did occur and fragmentation improved. Advancing the undercut towards the principal stress will ensure a better caving environment.

6.0 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 response 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 - \( RD \) - in the undercut drifts. 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.

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 terms ‘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.

7.0 NUMERICAL MODELLING

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

8.0 RATIO OF DEPTH TO HYDRAULIC RADIUS

The following information shows that there is no relationship between depth and caveability. It is the environment that is important:-

<table>
<thead>
<tr>
<th>Mine</th>
<th>Ratio of depth to hydraulic radius</th>
<th>Status</th>
</tr>
</thead>
<tbody>
<tr>
<td>Cassiar Mine</td>
<td>40: 1</td>
<td>Caved</td>
</tr>
<tr>
<td>Henderson Mine</td>
<td>35: 1</td>
<td>Caved</td>
</tr>
<tr>
<td>Shabanie Mine - Bl 58</td>
<td>25: 1</td>
<td>Caved</td>
</tr>
<tr>
<td>Shabanie Mine - Bl 52</td>
<td>20: 1</td>
<td>Caved</td>
</tr>
<tr>
<td>Shabanie Mine – Bl 6</td>
<td>9: 1</td>
<td>Caved</td>
</tr>
<tr>
<td>Andina Mine 2nd panel</td>
<td>7: 1</td>
<td>Caved</td>
</tr>
</tbody>
</table>

9.0 MONITORING (Ch5 p18)

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. It has been found that the simple techniques are often the best – (CH5 p18) Visual observations are extremely important and a simple descriptive code should be set up.

10.0 BULKING / VOLUME INCREASE

The bulking factor or volume increase of the caved ground is an important number as it dictates the rate of draw to ensure an equitable rate of caving. Useful information is available from observations made on Shabanie Mine, Zimbabwe. When block 7A caved the volume increase was 7%, owing to the failure of large blocks bounded by major shears at a rate of caving of 230m in three weeks. Block 6 had a bulking factor of 20%. Block 52 had a bulking factor of 14%. In deep seated orebodies bulking can mean that the cave will not reach surface, as was the case with block 58 at a depth of 670m and a draw height of 120m. At King Mine the bulking factor was calculated at 13%. The fine fragmented secondary ore of the Chilean copper mines have bulking factors of 30%.

11.0 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. In finely fragmented material high production rates are attainable whereas in coarse fragmented rock with a slow rate of caving the production rate must be tuned to the rate of caving until there is a high enough column of caved ore. The problem arises when there is a combination of coarse and fine in the same production block and there is a tendency to overdraw the finer material. This must be avoided. (Ch 5 p 21)
12.0 CHIMNEY / LOCALISED CAVES

Chimney caves or localized rapid caving can occur where caving occurs along a major structure or a zone of weaker rock, giving the impression that the back is caving. These areas should be identified in the preliminary stages and the draw controlled so as not to allow the cave to run away in these areas (Ch5 p22).

13.0 BOUNDARY WEAKENING

The techniques mention here are to assist caving for a portion of the orebody and not to pre-break the whole orebody. 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 this will not succeed. 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.

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. 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 were not successful if there were major structures which had a greater influence on the cave. However, if the pre-split plane is orientated at an angle to the confining stress to promote shear failure then they may well be more successful.

14.0 PRE-CONDITIONING

Pre-conditioning is a possible technique(s) to increase the fracturing of the whole or part of the draw column so that it will cave or fragment more readily. This is not a pre-break operation.

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 other techniques or a mass drill and blast situation.

LAYERED FAILURE - In high horizontal stress environments a double undercut would be developed so as to protect the production level. Then at regular intervals in the draw column horizontal cuts would be mined so as to create horizontal pillars which would fail and fragment under the high stresses:

The stresses in the pillar would increase as the pillar widths decrease and this would improve the fragmentation. The draw rate would be controlled during this period to reduce the incidence of large seismic events. The actual mining operation has to be sequenced so as not to create problems in the undercut zone. Open stoping followed by mass blasting pillars would be the safest way to go.

CHOKE BLASTING FOR FRACTURE INDUCTION - A technique of inducing fractures in the rock mass by blasting large diameter widely spaced holes is currently being experimented with. The theory is that fractures will be generated in the rock mass along cemented features such as gypsum veins or through the rock itself. If successful then this technique could have an application in competent orebodies. In high stress environments maybe only the lower section needs to be blasted to set up stress spalling in the upper section.
15.0 CONSOLIDATION

Certain material such as fine Chrysotile Asbestos, soft sheared material and Kimberlites will consolidate under load and create problems in the propagation of the cave. These zones must be identified and kept on the move.
Chapter 7

AIR BLAST POTENTIAL

1.0 GENERAL

Air blasts are the result of a plunger effect from the rapid collapse of a large volume of rock into an underground void. The air is compressed and forced out through openings or through the muckpile and then through the loosening caving mass when it reaches surface. The size of the void, the volume of the collapsing rock mass and the rapidity of the collapse governs the scale of the air blast. If there is sufficient cover or sealing of the openings leading into the cave zone/void then the bulk of the air will be forced out through the surface openings as they develop. This would result in a large plume of dust emerging on surface.

Air blasts have occurred on several block caving operations. These have generally been near surface, but, recently an air blast occurred at a depth of 900m. In the near surface situation, the mass of rock causing the air blast will be the rock layer between the cave back and surface. At depth, a slab can form with sub-horizontal structures or a change in lithology or a sill-layer of rock between the cave back and a previously mined area. Air blasts can only occur if there is an air gap and a slab(s) or a coherent mass of rock which can dislodge from the cave back. These can occur if the production rate exceeds the rate of caving.

Any new caving operation must be assessed in terms of potential air blasts even if these have not occurred previously on the property. Changes in depth and rock mass could create different situations. Extreme caution must be exercised if the hydraulic radius of a static cave back equals or exceeds the caveability prediction for that MRMR based on the caveability diagram. Some unidentified factor could be present and the time dependant failure could be dramatic. Drawpoints are not the only openings into a cave and all openings must be checked to ensure that the safety precautions have been taken. Any opening that does not enter the cave but lies in the potential cave zone must be assessed as the collapse of the back can lead to side failures.

2.0 TYPES OF FAILURE

Surface sill pillar - The collapse of a sill pillar between the void and surface has been an occurrence which has lead to air blasts. At Urad mine in Colorado, USA, a surface sill pillar collapsed resulting in an air blast through the underground workings R. Kendrick (Mining Congress Journal, October 1970, Vol. 56, No. 10). At Shabanie mine, Zimbabwe, a surface sill pillar collapsed without causing an underground air blast as there was sufficient height of muckpile. Collapse of cut and fill stopes near surface at Shabanie mine led to small underground air blasts.

Major sub-horizontal structures - In a horizontal stress environment large slabs can be created by shear failure along widely spaced low angle joints. This is the air blast situation that occurred at the depth of 900m on a mine where the production demands had left a large air gap.

Potential Sill Pillar - The situation can also occur where the cave back approaches previously mined areas and a sill pillar has formed. A high production rate will result in a large void and the sudden collapse of the sill pillar will lead to air displacement and an air blast. (Ch-6 p 2)

Rapid Unravelling of Cave Back - The rapid unraveling of the cave back can occur with dramatic effects when a key item is removed. The key item could be a narrow zone of more competent rock which has failed owing to time dependant failure or extraneous methods to cause failure such as blasting or hydro-fracturing.
3.0 AIR GAP

**Height of Air Gap** - Air gaps are common in stress caving situations as stresses build up in the back prior to the intermittent caving and the draw rate exceeds the rate of caving. As long as the height of the gap of a sub-horizontal cave back is not more than an average of 10m and the height of the muckpile (caved ground) meets the specified requirements then there is no problem if production is geared to maintain this maximum height (of muckpile). However, in the case of an incline cave back as with a panel retreat operation there should be no air gap so as to stop migration of ore or dilution down the muckpile slope. It is when the block is the only or dominant source of ore that problems occur with a draw rate higher than the rate of caving and an increase in height of the air gap. **The range in height and the average height of the air gap must be known at all times and the hazards assessed.**

4.0 HEIGHT OF BROKEN GROUND ABOVE DRAWPOINTS OR OPENINGS

The fragmentation of the muckpile must be known in order to arrive at the safe height of muckpile over a drawpoint or any opening into the cave zone. There is less chance of air moving through fine fragmented material, but fine material becomes air borne more readily. However, based on practical experience of poor caving blocks, the chances are that a large percentage of the fine material has been drawn at the expense of coarse material. Thus, in deciding on the height of broken rock to be left above the drawpoint the coarse nature of the column needs to be taken into consideration. 60m of well graded material will not permit the flow of air, but, 90m of poorly graded coarse material might be required for the same situation.

In the paper by R. Kendrick, describing the large Urad air blast in October 1968, he mentions that people were knocked down by air penetrating a 60m muckpile. However the cross sections drawn in August 1968 show that the muckpile only had a **minimum height of 30m** and there were also openings into the void. Therefore, we can conclude that 30m was not sufficient to prevent air blast damage on Urad Mine where a report stated that LHD’s were moved down drifts.

In block 6, Shabanie Mine, the stress caving was intermittent with failures of 10m in the back until the final collapse of the 28m surface sill pillar. No air blasts were noted in the drawpoints with a cover of 70m of broken rock, but there was a large dust plume on surface.

**Note that the drawpoints are not the only openings into a cave. The undercut drifts and observation or monitoring drifts could connect with travelling ways creating air passages and subsequent safety problems.** Drifts overlying the top of the cave with a narrow middling also fall into this category. Bulkheads have to be designed to cater for the enormous forces generated by large air blasts.

5.0 MAGNITUDE OF AIR BLASTS

The scale of an air blast will depend on the volume of the air gap, the height that the plug will travel, the size of the plug and the coherence of the plug. The air blasts at Shabanie Mine could be classed as small, those at Urad Mine and Northparkes Mine would be classified as large.

6.0 MONITORING THE RATE OF CAVING

The rate of caving is the fundamental criteria on whether an air blast will occur. Therefore monitoring of the movement of the cave back is essential in any caving situation including Sub Level Caving operations. Boreholes are the only means of accessing the cave back and once again exploration holes should always be assessed in terms of whether they can be used for this purpose. The best and simplest device to determine the position of the cave back where there is intermittent caving is an open hole into which a ‘cave monitoring device’ is installed – see manual. The hole can also be used for photography. (Ch6 p 4)
The rate of caving is a function of the MRMR and the presence of major structures. There are also other factors to recognise, such as maintaining a low draw rate to avoid seismic events, in which case the production rate is less than the rate of caving or to avoid the inflow of dilution. Measuring seismic activity and correlating those results with physical measurements will allow for a good understanding of the rock mass behaviour.
Chapter 8

ROCKBURST POTENTIAL

1.0 GENERAL

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 the following table:-

Suggested classification of seismic event sources with respect to tunnels

<table>
<thead>
<tr>
<th>Seismic Event</th>
<th>Postulated Source</th>
<th>First Motion from Seismic Records</th>
<th>Guideline Richter Magnitude M_L</th>
</tr>
</thead>
<tbody>
<tr>
<td>Strain-bursting</td>
<td>Superficial spalling with violent ejection of fragments</td>
<td>Usually undetected; could be implosive</td>
<td>-0.2 to 0</td>
</tr>
<tr>
<td>Buckling</td>
<td>Outward expulsion of pre-existing larger slabs parallel to opening</td>
<td>Implosive</td>
<td>0 to 1.5</td>
</tr>
<tr>
<td>Face crush</td>
<td>Violent expulsion of rock from tunnel face</td>
<td>Implosive</td>
<td>1.0 to 2.5</td>
</tr>
<tr>
<td>Shear rupture</td>
<td>Violent propagation of shear fracture through intact rock mass</td>
<td>Double-couple shear</td>
<td>2.0 to 3.5</td>
</tr>
<tr>
<td>Fault-slip</td>
<td>Violent renewed movement on existing fault</td>
<td>Double-couple shear</td>
<td>2.5 to 5.0</td>
</tr>
</tbody>
</table>

It should be noted that strain bursting does not necessarily require high stress levels for their occurrence; on Shabanie Mine strain bursting occurred in a brittle aplite dyke at 100m depth. 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 10% of the uniaxial compressive strength of the rock material. 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):
Types of rockburst mechanisms have been identified to include ejection, buckling, gravity enhancement, shake-out, fall of ground associated with a large, distant seismic event, disruption and displacement, convergence and heave (Ch7 p 4).

2.0 IMPLICATIONS OF SEISMICITY FOR CAVE MINING LAYOUTS

In hard rock, high stress 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 do 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 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. An example of such an event is shown in Figure 2. 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.
3.0 REGIONAL AND INDUCED STRESS

A high regional stress environment means that the induced stresses can easily exceed the rock mass strength as the rock mass respond to the large excavations 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:-

![Diagram showing rockburst and induced stress]

ROCK MASS STRENGTH
90 MPa
70 MPa
50 MPa

MINING INDUCED STRESS = REGIONAL STRESS
35 - 45 MPa

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

4.0 ROCKMASS STRENGTH, DIFFERENCES IN MODULUS

As the induced stresses increase rockbursts will not occur in weaker rocks as these will yield rather than store energy. Squeezing ground conditions apply in these cases. 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.

5.0 RATE OF CAVING

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 in a high regional stress environment where the seismic events in the back lead to rock bursts in highly stressed pillars some hundreds of metres away.

6.0 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 after the undercut has passed. 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.

7.0 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 brittle rock pillars can be loci for seismic events, as experienced by some events on high stress mines

8.0 SQUEEZING GROUND
In weaker ground the response to the high stress is for squeezing to take place and this imposes an enormous strain on the support.

9.0 SUPPORT
If a potential rockburst zone has been defined then the support must be designed accordingly. Excellent work done by Ortlepp has shown that an effective yielding system can be developed. A paper presented at Massmin 2000 by Stacey and Ortlepp describes the rockburst support systems.
Chapter 9

WATER INFLOW / MUD RUSH

1.0 WATER INFLOW

Water inflow from underground water and surface water in high rainfall areas and even in semi-arid areas where cyclones sometimes occur can be a problem. The basic sources of water are:-

Surface:
- Precipitation (surface runoff)
- Failure of “man made” structures, such as mine water reservoirs etc.
- Water accumulating in depressions on the surface of the cave

Sub-surface:
- Groundwater (aquifers)
- Water seepage
- Water from flooded underground excavations (e.g. Cassiar)
- Water from hydro-fracturing
- Water from unplugged drillholes

In the 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 arid areas the water entering the cave could be surprisingly high due to the intensity of the sporadic storms. Another aspect which has to be carefully investigated is preferential flows. For example, the tension cracks forming on the perimeter of the subsidence zone or major structures forming an incline cave boundary could cause large quantities of the surface water to be directed into caved zone without having the chance to evaporate.

2.0 SURFACE WATER CONTROL

In Canada the caves were at shallow depths and dry mill tailings were dumped on the cave primarily to stop the inflow of cold air during winter – drawpoints froze and could not be worked, however the tailings reported in the drawpoints. This practice was stopped and coarse waste from overburden stripping was dumped on the caves. The waste kept the cold out and reduced water inflow during the thaw.

In the Philippines, Philex Mine is located in a high rainfall area with a large underground water inflow. Their problem is twofold, controlling the underground water and keeping the surface water out of the cave during the monsoon conditions. The cave is kept topped up with waste and the surface inclined so that the surface water flows away into the natural drainage. A drainage drift and boreholes control the underground water.

Henderson mine, in the USA, is not in a particularly high rainfall area but does have snow during winter. The melting of the snow in the crater is a slow process owing to shade from high walls and low temperatures at those altitudes; this does not lead to a sudden increase in surface water. The caved column is high in the order of 900m so water retention is high and release into drawpoints is low. In Chile the cave mines are in the desert or in medium winter precipitation areas in high altitudes in the Andes. Whilst large quantities of snow might be dumped over a short period, the melting will be a fairly long process and generally does not generate surges of surface water. The freezing level is normally 2000m, however when during abnormal conditions the freezing level rises to say 3000m there is heavy rainfall and flash floods occur. On one occasion a diversion tunnel could not handle the flow and a low lying cave was flooded. As this cave contributed only 15% of the production there were no real problems underground, but surface damage was extensive.
In Zimbabwe the rainfall is not high and will range from 300mm to 700mm and falls from November to April usually with wet and dry spells. The caves cover a large area and have cave heights ranging from 100m to 500m and no problems have been experienced with surface water inflows. Pumping data from Shabanie mine shows that peak pumping occurs about one month after peak rainfall. This indicates that there is retention in the cave zone and with saturation a steady release of the water. The greater the depth, the slower will be the release. At Epoch mine there is no significant inflow of rain water from the crater through 300m to 400m of caved material.

The rate of water flow through the cave column will vary according to:

- height of cave column,
- percentage of the area under active draw,
- the fragmentation of the caved material, coarse fragmentation will allow for rapid flow compared with well graded material,

Large quantities of water flowing out of drawpoints leads to poor working conditions with low productivity and high tyre wear on LHD’s. In areas of high rainfall layouts must be designed for good drainage. The management of surface water is done with good surface drains and if necessary the cave is covered with waste rock to lead the water away. Underground water must be controlled with drainage holes preferably drilled from perimeter drainage drifts. One of the most important tools is an overall water balance. Comparison of the inflow and underground pumping rates could for example indicate water accumulation within the cave area. In the case of mud flow susceptible mines the “wet drawpoints are good drawpoints” indicating good draining potential of the muckpile.

3.0 MUD FORMING AND MUD CUMULATING POTENTIAL

The potential source of mud needs to be defined, in high rainfall areas it will be any finely decomposed material at or near surface. In semi-arid areas it is 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. Water is one of the fundamental parameters 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.

Part of the mud source investigation is:

I. Mine geometry
II. Surface Topography
III. Geology/Lithology
IV. Subsidence Predictions
V. Cave material properties:

Mine Geometry - The overall mine and cave geometry should be assessed. Location of old workings with respect to the potential cave failure 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 also be investigated. If the surface of the cave is irregular, with hills and valleys, 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 stripped 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 and the right composition can move rapidly under pressure. 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..

DANGER – Mud is far more dangerous than water because mud can accumulate in static parts of the cave column and then be forced out when the pressure is too high for the dam or when a cave intersects the high pressure pod of mud.

4.0 MUD FLOWS

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:
5.0 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
6.0 MUD FLOW POTENTIAL

Flow ability 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 progresses. 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 from “static” flow under gravity. There are known cases of the mud flow (under pressure) of the very stiff mud.

7.0 MUD FLOW RISK ASSESSMENT

In the mud flow risk assessment for a particular mining operation there is a likelihood of occurrence and consequences of mud forming potential. Triggering mechanisms and mud flowing/discharge have to be assessed and an overall qualitative risk assessment of mud flow potential investigated.

8.0 SAFETY AND ECONOMICAL ASPECTS

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

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

9.0 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 of the life of the block, mining against old blocks with a mud problem or with the sub level caving method owing to its irregular draw pattern. The layout must be designed to ensure a uniform draw and draw management must be practised. 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. Areas of irregular draw must be identified and remedial action taken.

Backfilling the crater - If the crater is backfilled with waste rock then surface water ingress is reduced

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 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  
VIII. identify channelways and drainage patterns

**NOTE** – When the factors influencing mud flows/pushes are identified then 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.
Chapter 10

PRIMARY FRAGMENTATION

1.0 GENERAL

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

Caving results in primary fragmentation which can be defined as the particle size that separates from the cave back and enters the draw column (Ch9 p 2). 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
* Mean joint spacing and maximum and minimum spacings
* Average joint dip and direction as well as the range in dips
* Orientation of cave front
* Induced stresses

The +2m³ content of the caved material is the standard adopted to define fragmentation on the basis that a 6yd LHD can handle 2m³ material. If the plan is to use 8yd machines then the cut-off value becomes 3m³.

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 these problems.

2.0 INPUT DATA FOR A FRAGMENTATION ANALYSIS

2.1 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.

2.2 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 diagram on page …..in the block cave manual 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.

2.3 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 with 0.5m minimum and 1.5m maximum means that the largest block is not likely to exceed 3m³, but, if the maximum were 3m then the largest block could be 24m³ - a significant result for production efficiencies.

2.4 Number And Spacing of Joint/Fracture Sets - This calls for precise structures mapping as it puts structural geology into the engineering category. It is necessary to define joint spacing or fl/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 +2m³ might = 50%, whereas by assessing two structural zones separately, the result would be Zone A at 70% of the area = 10% and zone B at 30% = 70% +2m³. Combined result = 28% +2m³. This approach could call for modifications to the layout in areas of coarse fragmentation (Ch9 p 5).

2.5 Joint Condition Ratings - Joint condition ratings must be assigned to individual joint sets as a set with lower ratings 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.:-

Rock block ~>

<p>| | |</p>
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joint condition rating = 15

<-- joint condition rating = 23

2.6 Effect of stress - The induced stress is the strike and dip stress in 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 of 0.5 to 2 MPa will only be significant if the cave face is vertical or dipping towards the cave. The primary fragmentation from vertical extension (stress) caving is generally finer than from subsidence caving owing to the failure not only along joints but often of the rock blocks under high stress. Stress spalling has been observed in cave backs and fracturing of the rock mass occurs due to modulus differences. Lateral extension (subsidence) caving in a low stress environment or in a relaxation zone adjacent to an operating cave can lead to failure along widely spaced joints resulting in large blocks.

There can be a change in primary fragmentation in the cave back as a result of stress differences or changes in rock mass properties. This also applies to the hangingwall zone as finer material could present a dilution problem. A cave front moving...
upwards towards a sill pillar below a previously mined area should enter a higher stress zone with resultant finer fragmentation. It is important to correctly assess the magnitude of the defined stresses. Numerical modelling can provide the 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 are assessed.

2.7 Cave Front - Dip and Direction - The orientation of the cave face will not change as the cave propagates. 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 orientation of the cave face with respect to joints and stresses has a significant bearing on the 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 primary fragmentation and therefore caving.

2.8 Draw Rate - The draw rate is an important factor in that it must provide space for caving; also it must not be too fast to create a large air gap and possible air-blasts. If the draw rate is too fast seismic activity. Production must be based on this value and not on 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 dependent failure; a slow draw rate will mean improved fragmentation.

3.0 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.

4.0 PRACTICAL CONSIDERATIONS

At the earliest possible stage primary fragmentation calculations must be conducted for different heights in the column. These assessments will indicate production problems and the need for additional data. Stresses change with depth, thus, if a high ore column is going to be drawn then the primary fragmentation for specified intervals e.g. 50m should be determined :-

\[
\begin{align*}
\text{Secondary (2)} & \\
100m & \text{Primary fragmentation (2)} \\
\text{Secondary (1)} & \\
0m & \text{Primary fragmentation (1)}
\end{align*}
\]

5.0 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. A histogram of the predictions will be useful.
Chapter 11
SECONDARY FRAGMENTATION

1.0 GENERAL

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 to, determines the fragmentation size distribution in the drawpoints. A well jointed material with high rock block strength can result in a stable particle shape at a low draw height. A wide range in fracture frequency readings will result in a wide range in fragment size as compared to the fragment size distribution produced by a rock mass with a narrow range of readings. The fine material produced tends to cushion the larger blocks and prevents further attrition of these blocks. Cushioning is a common occurrence on chrysotile asbestos block cave mines where the shear zones will have IRMR ratings from 10 to 24 and this material will cushion the larger primary fragments with IRMR 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. The column height is the height of the caved ground overlying the drawpoint; it must not be confused with draw height which is the column of ore and dilution that will be drawn. Fragmentation is the major factor that determines drawpoint productivity. The following are required to determine the secondary fragmentation:

- The effect of fines cushioning
- Draw strategy and draw rate
- Rock block strength - RBS
- Shape of fragments
- Frictional properties of fragments
- Workability of the fragments
- Column height - caving stresses due to arching.

2.0 VOLUME INCREASE / BULKING

The volume increase or bulking is a function of the 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 unless draw rates are increased.

3.0 COLUMN HEIGHT - CAVING AND ARCHING STRESSES

The caving stress is the load imposed on pillars and stationary particles by the arching and direct loading of 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 laterally and vertically in the draw column when the arch fails. Point loading of the blocks in the arch causes the blocks to break. 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.
4.0 ROCK MASS PROPERTIES

4.1 Frictional Properties - Low friction material will tend to flow more readily and arches in low friction material will collapse more readily than those in high friction material.

4.2 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 (Ch10 p4).

4.3 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 size, its shape and its RBS.

5.0 FINES

Fines occur as the breakdown of weak zones and also as blocks move down the cave column the corners are broken off in the process of creating more stable shapes. The BCF fragmentation program assumes that 5% fines will be generated by rounding off the corners.

6.0 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.

7.0 DRAW PROCEDURES

A low draw rate will result in time dependant failure of the rock blocks as they are subjected to the caving and arching stresses. 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.

A 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.

8.0 DRAW HEIGHTS

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.
9.0 PRODUCTIVITY

Fragmentation is the major factor that determines productivity from a drawpoint. Fine material will ensure high productivity from a 6 yd LHD (Ch10 p 5). A productivity graph (Ch10 p6) shows the relationship between fragmentation and LHD’s of different sizes.
Chapter 12

DRAWDPOINT / DRAWWZONE SPACING

1.0 GENERAL

Drawpoint spacing is an essential part of the design of any caving layout and must be debated carefully at the start of the project. This is a step in the iterative process and will have to be revisited in the cycle of planning. The decision must be based on the following factors:

- The isolated draw zone diameter – IDZ, which is a function of the fragmentation and can vary with draw.
- The fragmentation for the bulk of the draw – wide spacing and fine fragmentation means ore loss and high dilution.
- The internal friction of the material to be drawn
- The size of LHDs to be used (must match the drawpoint layout NOT dictate it)
- The size of the drawpoint’s loading area (the LHDs’ digging characteristics)
- The number of drawpoints required for the production rate
- The required recovery of ore and the dilution thereof
- The planned shape of the drawbells
- The possibility of brow wear (it will probably be asymmetric)
- The planned draw strategy
- The expected interaction and flow of ore into the drawpoints (the isolated draw-zone diameter)
- The effect of drawpoint closures
- The possibility / likelihood of poor draw control (production dictates)
- Compare the spacings for different layouts e.g. Teniente vs. Herringbone

Pay attention to the dimensions selected and the effect of these on the swell relief from undercutting.

2.0 DRAWPOINT INTERACTION

3D sand model studies (Ch11 p 3 – 7) show that the interaction between drawzones is a function of the spacing. Underground experience has shown that the spacing of the drawzone interaction increases as the fragmentation becomes coarser (Ch11 p 17).

3.0 SPACING GEOMETRY AND SELECTION

The guiding light to selecting the drawpoint spacing is that the drawpoint is the only means of controlling the caving activity once the production level is developed and the block undercut. The drawpoint spacing must be such that ore recovery and dilution are kept at optimum values and that any column loading of the apexes can be controlled by increasing draw in the affected areas. 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 an LHD has to be dead straight before the bucket / scoop enters a fine muckpile. Thus the major factor of optimum ore recovery can be prejudiced by incorrect equipment selection or mistaken ideas about loading procedure. The following diagram shows the critical distances in a layout and how these can vary across the minor apex and across the major apex if the drawpoint is increased in length :-
As can be seen from the above diagram the drawzone spacing of 8m in the drawbell is considerably less than the 22m across the major apex. Therefore, if the production drift spacing were increased to 32m then the spacing in the drawbell -A- would increase to 10m and still have good interaction in the drawbell. The diagram also shows that if the length of the drawpoint is reduced then the spacing across the major apex would reduce and improve interaction. By contrast the Teniente layout in the following diagram has a more uniform spacing of the drawzones.

4.0 SELECTION OF LAYOUTS

The selection of layouts is influenced by the following factors:

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. If in practice, this is not done then ore recovery decreases and dilution increases - (Ch 11 p24-27).

DRAWBELL GEOMETRY - The term drawbell is descriptive in that in theory the ideal 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 life of the draw. It needs to be established how much influence the shape of the drawbell has on interaction. It has always been an empirical point that shaped drawpoints improve ore recovery as the ore should have better flow characteristics than a drawbell with vertical faces and a large flat top major apex. PFC modelling might confirm the practical belief.

INCREASING DRAWPOINT SPACING - Henderson Mine have increased the drawpoint spacing over the years based on their belief that they have mass flow from their draw strategy and excellent draw control. In the first instance the increase was from closely spaced drawpoints at a nominal 12.2m to a spacing of 16.2m along the drift and the drift spacing was kept at 24.4m. Subsequently the drift spacing has been increased to 30.5m and the spacing along the drift to 20.6m. This has meant an increase in area of influence from 148m² to 190m² to 300m². The distance across the major apex has increased from 16m to 18m to 20m. The layout has been changed from a staggered herringbone to a straight through diagonal (Teniente). The spacing of the current drawzones are shown below, with the off-set herringbone for comparison:-

<table>
<thead>
<tr>
<th></th>
<th>Diagonal (Teniente)</th>
<th>Off-set Herringbone</th>
</tr>
</thead>
<tbody>
<tr>
<td>Minimum distance across major apex</td>
<td>= 20m</td>
<td>= 24m</td>
</tr>
<tr>
<td>Maximum distance across major apex</td>
<td>= 21m</td>
<td>= 24m</td>
</tr>
<tr>
<td>In the drawbell</td>
<td>= 14.7m</td>
<td>= 10m</td>
</tr>
<tr>
<td>Drawbell to drawbell along drift</td>
<td>= 20.6m</td>
<td>= 20.6m</td>
</tr>
<tr>
<td>Diagonally from drawbell to drawbell</td>
<td>= 17.3m</td>
<td>= 24m</td>
</tr>
</tbody>
</table>

Whilst the spacing across the major apex is low at 20m the other distances are large in terms of interactive draw. Henderson mine draw a line of alternate drifts on a shift with good draw control, this means that the drawzone to drawzone spacing on either side of the major apex will be 20.6m which is large for fine fragmented ore. Drawing a line of drawbells would possibly give a better result as the dimensions of 14.7m and 17.3m in and between drawbells would mean interaction. The success of these changes must be followed up.
Chapter 13

UNDERCUTTING

1.0 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 and the sequencing of the undercut is important 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 will be left. This rule, which is often ignored owing to the problems in re-drilling holes, results in the leaving of pillars resulting in the collapse of large areas and consequent high ore losses. Management 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 affect 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 undercutting 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 (Ch12 p 2).

2.0 PARAMETERS INFLUENCING THE UNDERCUTTING OPERATION

2.1 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.

2.2 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 should be designed to reduce or eliminate the rockburst potential.

2.3 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.

3.0 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 the pillars and drawpoint brows on the production level are severely damaged by the abutment stress.

(Ch12 p 4)

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 (Ch12 p 5). In fact it is surprising that no other mines have adopted this technique in preference to the conventional system. The system is not likely to work where there are squeezing ground conditions and hole closure occurs.

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. 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 (Ch12 p 6-8) or inclined over the major apex, resulting in a ‘saw tooth’ appearance (Ch12 p 9-11). 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.

DISADVANTAGES OF 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 greater.
* The flat top of the major apex and 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. Large rock blocks and high friction material will 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 (Ch12 p 9-11)

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.
* In a high stress environment the narrower the stope the lower the energy release as has been shown on high stress mines.
* It has been shown on South African gold mines that backfilling of stopes decreases the abutment stresses. 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 to 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 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.

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.

4.0 PLANNING AN UNDERCUT

The timing of production level development and undercaving 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 undercaving. 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 development would only be considered if the rock mass was competent and the stress levels acceptable. In poor ground even in the initial 30% the drawpoints would not be developed.

Various combinations are possible, for example, the Teniente layout lends itself to the development of drawpoints in the direction of undercaving. If these ends are alternated then there can be a significant saving in time.
5.0 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 3m can be used provided there is no impedance to the propagation of the cave. 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.

6.0 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 unsuccessful 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 its own is not
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.

7.0 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 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. 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.
* An undercut should not be advanced towards existing infrastructure such as haulage’s or crusher sites.

8.0 SHAPE OF UNDERCUT FACE

A concave (towards the solid) undercut face, provides better control of major structures and a more secure undercut level. The caveability of the immediate back will be controlled.

A convex face is less stable both in the cave back and on the undercut and production levels.
If possible, the cave front should not be advanced towards structures that could initiate massive wedge failures.

![Diagram showing stress and advance](image)

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

![Diagram showing structure and cave front](image)

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

9.0 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.

10.0 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.
11.0 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 drawpoint 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 level of support 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. Once experience is gained the level of support can be decreased. Delays to repair or re-support will delay the undercutting operation (often at a critical time) and can lead to undercutting stopping followed by stress levels rising, often with dramatic consequences.

Design the support particularly for:

* The access junctions and other intersections
* Areas where wide span conditions occur
* The areas where the critical caving hydraulic radius will be reached ere major poor ground, structures, dykes or zones occur

12.0 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. The question is can a slot be created by blasting large diameter holes from the undercut level so that a zone of fractured ground is left in place and drawn off when the drawbell drift is developed?

13.0 UNDERCUT LIMITS

Undercuts must protect junctions and major excavations. It might be necessary to extend the undercut to reduce the possibility of overhangs or where there might be ore losses due to tight corners.

14.0 ANTI-SOCKET DRIFT

In the case of high draw heights in competent ground with greater drawpoint spacing, consideration should be given to developing an anti-socket drift on the crest of the major apex (Ch-12 p 10).

15.0 POINTS TO BE NOTED

* Are inclined and / or canted rings to be used?
* The mechanism for the initial slotting and is re-slotting to be done when and how
* Do any areas require special consideration with regard to drilling and blasting?
* The full design of the drilling, charging and blasting patterns (calculating the void requirements):
* Verify design with a blasting engineer, explosives and accessories supplier/s
* Set minimum standards and reaction policy
* Define the lead and lag of the faces drift to drift
* The full sequence of the blasting to create the correct face shapes
* The rate of advance of the face (maximum of one ring per day in a drift) and the policy regarding weekends and public holidays
* The quantity of swell to be loaded with each blast
* How is the swell actually loaded and the tonnage to be recorded for draw control
* How is the presence of pillars to be determined when the major apex is shaped from the development just above the minor apexes?
* The way pillars are to be dealt with
* The possibility of large blocks stacking on the major apex
* Mine the undercut with full attention to detail and a high level of supervision taking particular note of drilling the holes (accuracy, direction and depth; re-drill where required) and having sufficient drilling equipment to allow for any areas that are difficult to drill or where re-drilling is required.
  Charging the holes (recheck depths).
* Good housekeeping
Chapter 14

ORE HANDLING

1.0 GENERAL

Previous standards for handling ore from block cave layouts were:-

- In a slusher layout into orepasses or directly into cars.
- In a grizzly layout through drawpoint grizzleys down an orepass onto a conveyor or into cars / trucks.
- In an LHD layout 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 developed in competent orebodies with coarse fragmentation, the ore handling process has been under review with some major changes. The LHD is now 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.

It is essential to investigate ore handling thoroughly and designs appropriately, taking into account the following:

- Match the various elements in the system; preferably each element down stream should either have more capacity than that feeding it or have surge storage preceding it.
- Don’t create bottlenecks in the planning
- Fragmentation (with time)
- Pre-loading breaking
- Loading equipment and productivity
- Grizzly sizes and oversize handling
- Size, location and security of the ore passes and breaking / crushing facilities
- Loading chutes and conveying / tramming / hauling systems
- Hoisting

2.0 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 becomes coarse to very coarse, i.e. the +2m³ is 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. This system was introduced many years ago on Bell Mine, Quebec. Here the 5yd LHD’s haul the ore to a large grizzly where three pick hammers operate. The grizzly undersize goes 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. Northparkes’s assessment is that this is a successful operation in that capital and operating costs were reduced and there was improved efficiency. 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.

Effective secondary breaking systems in the drawpoint are essential for high productivity. The tip opening and orepass diameter or the size of the grizzly on the level will determine the maximum rock block that can 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.

3.0 OREPASS CHARACTERISTICS

OREPASS 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 thumb 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) ore passes on high stress mines have spalled with the result that 2m passes become 12m. In these situations the length should be limited to within 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 - The MRMR will indicate the wear characteristics of the ore passes and the need for support.

OREPASS LININGS Ore passes generally 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.

The following chart shows the various factors that affect the performance of ore passes and their inter-relationship:-
4.0 COMPARISON OF TRUCKS, CONVEYORS AND RAILROAD

Studies on the merits of the use of trucks versus conveyors versus railroad seem to be done on a regular basis without conclusive results. 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.

5.0 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. (Ch15 p 10)
Chapter 15

LHD HORIZONTAL LAYOUTS

1.0 DESIGN REQUIREMENTS

The horizontal layout must be designed recognizing the following. Foremost are the optimum drawpoint spacing, the stability of the layout and its capability to handle large tonnages safely and efficiently:-

* The ore body geometry
* The fragmentation expected through the life of the block
* Is a secondary drilling level required for secondary blasting of large fragments?
* The tonnage to be drawn through the extraction opening
* The possibilities of brow wear and the support required for the extraction opening
* The drawpoint / drawzone spacing and the shape of the major apex and drawbells
* The development size and the subsequent pillar size and stability will give the permissible LHD size.
* The undercutting method / technique to be employed
* The logistics from start of development to full production, particularly for an advanced undercut.
* The zoning of the MRMR and structures and the orientation of the development, major apexes and brows relative to these. A solid 3D model is useful in this respect.
* The drawpoint loading area and anticipated rock flow pattern
* The ore handling systems and ventilation layout
* The number of drawpoints required to meet the production needs
* The drainage system and decide what the gradient of the development will be in both directions
* Ensure that the outer production drifts have drawpoints facing outwards.
* Consider perimeter drawpoints on the undercut level.
* Design the drawbell shape for optimum interaction within the drawbell, between drawbells over the minor apex and over the major apex.
* Irregularities in outline should not influence the layout. These areas can be left for a reclamation exercise.

The ‘manual’ lists and discusses ten horizontal layouts that are in use or have been used, in terms of strength. These range from the strong Teniente layout to the weak continuous trough. Nowadays only the Teniente and the herringbone models are considered. The drawbell design in the Teniente layout has been modified over the years but the fundamental points of facing drawpoints in line in the drawbell and the brow at right angles to the drift have not changed. Drawings and description of the layouts and modifications are in the manual. It would be appropriate to list the advantages and disadvantages of the Teniente and Herringbone systems:-

TENIENTE / DIAGONAL (Ch14 p 6-8)

Advantages:-

- There is a more uniform spacing of drawzones and the spacing across the major apex is less than the same layout dimensions in a herringbone layout.
- The drawpoint and drawbell are on line ensuring better loading and draw and better drawpoint and brow support. The LHD can back into the opposite drawpoint if there is brow wear.
- Modelling showed it to be the stronger structure when compared with the staggered and opposite herringbone layouts
One of the original advantages was the ease of development; however that has fallen away with advance undercutting and the Henderson approach to leave a pillar in the drawbell drift.

Disadvantages:-

- The Teniente layout is not favoured for electric LHD’s.
- If the ore passes are only at one end of the drift then the LHD travels with bucket forward when loading drawpoints on one side of the drift – vision impaired and dust from travel against air flow?

HERRINGBONE (Ch14 p 3):

- The herringbone lends itself to the use of electric LHD’s and the LHD’s have the same orientation in travel when loading from opposite drawpoints.
- The herringbone layout lends itself to a mirror image layout as was introduced on Northparkes mine. The central pillar can be used for an orepass and ventilation system.
- The mirror image layout is suitable for very steep dipping orebodies up to 200m wide. With central ore passes the haul distances are reduced for high LHD efficiency.
- With the drawpoint and drawbell at an angle this means that draw is not uniform across the drawpoint.
- The brow support is not at parallel to the brow unless the brow is modified as done at Northparkes

2.0 ADVANCE UNDERCUTTING

With the emphasis on advance undercutting, it is important to select a layout which is best suited to advance undercutting. With the Teniente diagonal system the drawbell drift can be developed from the rear and advanced towards the advancing undercut. The following two diagrams show the situation with the Teniente layout in the one case where only the drift is developed ahead of the undercut and in the other case where the drawpoint take-offs are developed as well. Another adaptation would be to only develop the take-offs on one side of the drift, in this case it should be those facing backwards.
In the case of the staggered herringbone the development of the drawbell drift is not as simple as in the diagonal layout. It appears that the drawbell drift can only be started when the undercut is further ahead. Furthermore, the drawbell drift is developed from both sides meaning there are two ends operating from each drift with more congestion. As the object is to develop and commission the drawbells as quickly as possible the layout that has ease of development and support installation must have advantages:-
3.0 OTHER HORIZONTAL LAYOUTS

None of the other layouts that have been tried over the years have any advantages over the herringbone and the Teniente diagonal / parallelogram. The continuous trough has a distinct disadvantage, namely that the minor apex has been removed. There is no support to the major apex or the brow allowing the brow and major apex to fail as was the case in block 7 AB, Shabanie Mine.
Chapter 16

INCLINE DRAWPOINT LAYOUT / FRONT CAVE

1.0 INCLINE DRAWPOINT – 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](image)

There are two incline drawpoint layouts, namely, the truncated ‘SLC’ and the inclined ‘egg-box’ and these have been described in the ‘Block Cave manual’. There is no reason why the ‘false footwall’ technique should not be applied to other mining operations.

2.0 SPACING OF DRAWZONES

One of the major issues that have 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. However, because of the straight and long drawpoints the widths can be greater than with a horizontal layout which means that larger equipment can be used. Also, as the loading is straight on and across the full width of the drawpoint an optimum dig is obtained leading to a larger drawzone and better interaction than would be the case with a horizontal layout.
### Dip of Incline Plane

<table>
<thead>
<tr>
<th>Dip of Incline Plane</th>
<th>Fragmentation</th>
<th>Vertical</th>
<th>Along Strike</th>
<th>Down Dip</th>
</tr>
</thead>
<tbody>
<tr>
<td>35°</td>
<td>Medium</td>
<td>12m</td>
<td>12m</td>
<td>17m</td>
</tr>
<tr>
<td>35°</td>
<td>Coarse</td>
<td>15m</td>
<td>15m</td>
<td>22m</td>
</tr>
<tr>
<td>40°</td>
<td>Fine</td>
<td>10m</td>
<td>10m</td>
<td>12m</td>
</tr>
<tr>
<td>40°</td>
<td>Medium</td>
<td>12m</td>
<td>12m</td>
<td>14m</td>
</tr>
<tr>
<td>40°</td>
<td>Coarse</td>
<td>15m</td>
<td>15m</td>
<td>18m</td>
</tr>
<tr>
<td>40°</td>
<td>Coarse</td>
<td>18m</td>
<td>15m</td>
<td>22m</td>
</tr>
</tbody>
</table>

#### 3.0 Stability of the Layout

The overall stability of the layout is of paramount importance. Ideally the layout is located in a strong dipping footwall, however as this is not always the case it is essential that the layout is overcut so that there is no chance of wedge loading. The undercutting sequence will also ensure a stable layout. It has been suggested that the undercut is mined away from the drawpoint brow towards the orebody so that the abutment stress is thrown away from the underlying drawpoint.
4.0 TYPES OF INCLINE DRAWPOINT LAYOUTS

4.1 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. This layout was used on Cassiar and King Mines. The details of the Cassiar Mine layout are shown in the manual (Ch15 p4).

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.
4.2 INCLINE EGG BOX

The incline egg box layout was used on Bell Mine Canada with success in some drawpoints and problems in others. Where the footwall was sound over 100,000 tons was drawn, however in areas of wedge failures and well developed shear zones drawpoints were subjected to pressure and calls were not achieved. A good example of why regional stability is essential. The advantage of the incline egg-box is the continuity of the pillars surrounding the drawbell leading to a strong structure. Details can be found in the manual.

3-D VIEW OF EGGBOX LAYOUT

This layout lends itself to an advance undercut either as a narrow slot or as rings.

5.0 DESIGN CRITERIA

All the design criteria recorded in the horizontal layout section apply to the incline layout. Of particular note is the security and stability of the area containing the extraction levels and the infrastructure relating to the drawpoint layout? Some of the advantages of the incline layout are:-

- The simplicity of the layout means long straight stable drawpoints so that larger equipment can be used.
- Drainage is superior to the horizontal layout
- Drilling of hang-ups and secondary blasting can be undertaken without interfering with production
- Loss of brow is not a serious problem.

Disadvantages of the incline layout are:-

- Far more development
- Increased ventilation costs
6.0 FRONT CAVE

The front cave is basically a retreating SLC with static drawpoints at regular intervals. A minimum of two levels is recommended if caving follows the advance to the next line of drawpoints. The spacing of drawpoints will follow the same rules as for other layouts. The interaction will be along the line of drawpoints for that series of drawpoints, thus ore losses will be higher than for layouts where all drawpoints are operational.

Rock mass stability is essential, particularly with respect to wedge failures. If necessary an overcut level must be developed and mined. The front cave is a ‘make do’ method where the quantity of ore does not justify the infrastructure required for more conventional layouts. Caving following undercutting is a requirement. Face shape and orientation must be such so as to avoid large scale rockmass failures.
Chapter 17
GRIZZLY / SLUSHER LAYOUTS

A) GRIZZLY LAYOUTS

1.0 GENERAL

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. In secondary ore 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.

![Diagram of Grizzly Layout](image-url)
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 the close spacing of the drawpoints results in good ore recovery and the dilution is low.

At Bell Mine, Quebec the grizzly layout was used 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. See manual for diagrams. (Ch16 p3)

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. Unfortunately with brow wear the drawzone spacing increases. See Diagrams in the manual page. (Ch16 p5)

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

<table>
<thead>
<tr>
<th>Mine</th>
<th>Layout</th>
<th>D/P spacing</th>
<th>Tons per day - tpd</th>
</tr>
</thead>
<tbody>
<tr>
<td>Andina</td>
<td>Diagonal</td>
<td>9.0m</td>
<td>20000</td>
</tr>
<tr>
<td>Teniente</td>
<td>Opposite</td>
<td>7.5m</td>
<td>15000</td>
</tr>
<tr>
<td>San Manuel</td>
<td>Opposite</td>
<td>5.0m</td>
<td>45000</td>
</tr>
</tbody>
</table>

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.

### SLUSHER LAYOUTS

Slusher layouts were and are still in use on some 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. Furthermore, attending to hung up drawpoints means that the operation must stop. In dusty operations it is difficult to see into the drift and the tendency is for high production from the nearest drawpoints. Various controls have been introduced to record the various distances that the slusher has been moved so as to improve the draw control without great success.
Chapter 18

PRE-BREAK / INDUCED CAVING / FORCED CAVE / PRE-CONDITIONING
AND BOUNDARY WEAKENING

1.0 GENERAL

A pre-break or a forced cave or induced caving can only apply to certain sections of an orebody planned as a caving operation. Pre-breaking the whole orebody is a shrinkage operation and not a block caving operation, other techniques apart from blasting the orebody to create a broken rock mass need to be considered.

The pre-break would apply to areas where there was doubt about the caveability or that there was a need to reduce the size of the fragmentation. There might be areas where there were large pods of competent rock in material that would cave. It might be necessary to break up potential large wedges. Boundary weakening is often considered for caving operations were there is doubt about the start of caving or to ensure that caving takes place in a specified area. The decision to pre-break or to force a cave must be made considering available techniques and costs.

When only the lower section of the orebody is pre-broken the reason often given for doing this is to make a large tonnage 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 Northparkes pre-break example clearly shows that the method was costly and time consuming and achieved very little in the long run.

A pre-break is applicable if the production level is in very good rock, overlain by a high column of good caving ground, as is the case at the contact between primary and secondary rock in Chilean mines. In these situations a relatively thin but, large slab of competent rock can be left between the top of the undercut and the weaker secondary rock. The collapse of a large slab could lead to an air blast and therefore the slab should be pre-broken.

There is merit in pre-conditioning a competent orebody by a simple cost effective method. This will increase the fractures so that caving propagates and fragmentation is acceptable.

2.0 FORCED CAVE TECHNIQUES

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, that no verandas form and that there is continuity of mining. When pre-break drifts are developed far apart to keep development costs down, large volumes of rock are broken per blast.

If the same approach is used on the undercut level 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.
3.0 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 caveability 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, the logical place to start is in the high grade zone.

4.0 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 under void conditions at the start, once the undercut area approaches the hydraulic radius and back failure has not occurred the ends have to be left full of broken rock and any other openings sealed off. Therefore, if the drifts 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 them.

5.0 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.
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 would be obtained if the mining commenced in the centre and worked outwards 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.

The cave back can be supported by the broken rock and a stable arch can form. Drawing off of the broken material will remove the support over then whole area and can result in a sudden collapse with air blast consequences.

6.0 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.

7.0 PLANNING REQUIREMENTS

If the decision is to follow this route then it must be meticulously planned and must consider the following points:

- The area to be mined needs to be carefully defined and the operation planned so as not to prejudice the overall caving operation.
- The inconvenience of coarse fragmentation and extensive secondary blasting is often less costly than doing a pre-break.
- Boundary weakening is done by cutting a slot or by blasting a pre-split, note that a pre-split is not better than a major structure.
• Hydraulic fracturing has been successful in competent ground, but has yet to be proved in very competent ground. The development required providing the necessary drilling sites; the drilling and actual operation is costly and must be measured against secondary breaking costs on the production level.
• As the possibility of air blasts exists, there must be monitoring of the back and height of the air gap.
• Blasting of large tonnages must be controlled to prevent damage to the major apex as was the case at Northparkes.
• Rapid advance of the pre-break can create areas for massive wedge failures.
• The pre-break must be linked to the eventual cave.
• A good record of tonnages drawn from the pre-broken ore and the coarsely fragmented cave material

8.0 PRE-CONDITIONING

Pre-conditioning techniques are discussed in the section on caveability.
Chapter 19
SECONDARY BREAKING

1.0 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.

Jumbos are used effectively to drill and blast large rocks in drawpoints with low powder consumption as the rock have free faces. A hang-up drill was developed in Zimbabwe some 30 years ago to effectively drill and bring down hangups up to 10m 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. An effective secondary breaking program is designed to break rocks in the drawpoint, in low hangups, high hangups and large rocks lying over apexes. This cannot be done with one machine. Palabora Mine are experiencing major problems with oversize such that production is 11000 tpd and not the planned 30000 tpd.

2.0 SECONDARY FRAGMENTATION DATA

Secondary fragmentation data is prepared for increasing draw heights. These will show whether there is a progressive decrease in coarse fragmentation or whether the +2m³ material will reach a constant level for the remaining draw life. It is important to establish from the start whether high hangups are 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 correct equipment can be on site and the production potential of the block can be determined. It also means that a sound secondary breaking program can be designed and the necessary equipment purchased and development done.

3.0 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 grizzlies 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 grizzles 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 is worth noting. 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 getting 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.

### 4.0 PERCENTAGE OF HANGUPS

The size distribution from the BCF program will give a good indication of the number of hang-ups (see manual). 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. If 15% oversize = 600 tons and a high hangup is estimated at 200 tons then the number of hangups per day = 3.

In terms of bringing down a hangup, the important question is the stability of the hang-up. Hang-ups 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.

### 5.0 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 dependent on the location of the legs of the arch or the key block.

High hang-ups have been successfully drilled with a hangup rig developed on King Mine, Zimbabwe some twenty years ago. The rig is described in the manual and works on the principle of locating the drill rods in flush jointed diamond drill casing (Ch18 p 4, 5). 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.

### 6.0 BLASTING TECHNIQUES

Drill and blast is the most efficient technique as the hole can usually be sited to break effectively - after the entire 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 than homogeneous rock blocks.

Conical charges are effective when properly placed. Conical charges - made from funnels – are placed on large rocks which have been hauled to 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.

7.0 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 hang-ups. Northparkes mine have persevered with experiments on non explosive techniques and have a wealth of experience.

8.0 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. 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. It is extremely difficult to understand why some mines have not experimented with this concept. One of the arguments that it will weaken the pillar ( major apex ), is misleading as the weakest part of the major apex is on the extraction level where 50% of the rock has been removed. Strategically sited pre-drilled holes from the secondary drilling level would be used to drill large hangups above the major apex. Flush jointed casing inserted through the pre-drilled hole and against the hangup would be used to drill through and to charge the hole.
Section A – B:-

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

The standards and procedures for secondary breaking must be laid down and adhered to with definite time frames, e.g. how long a hang-up can be left without being blasted and when, in the shift, should blasting ideally be done. They must be designed to cover the breaking of large rocks from the drawpoint through to the high hang-ups and this will mean the use of different techniques and equipment. The standards must cover:-

Conical lay-on charges
Placing and blasting explosives in porous bags
Drilling and blasting
Pre-drilled holes from the production drift or the secondary drilling level
Non-explosive breaking
ANCILLARY DEVELOPMENT

1.0 GENERAL

Ancillary development covers a broad range of activities: shafts, ramps, haulage’s, ventilation drifts, orepasses, crusher sites, sumps and workshops. All the items required to ensure a smooth ore extraction and ore handling operation. Each item must be subjected to the same geological and geotechnical investigation and assessment as the actual caving operation.

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 must 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 recognized in determining rock mass 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 the 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 long lives, it is commonly seen as desirable to locate mine service facilities as close to the block cave as practical to minimize traveling 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. 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 destabilizes large blocks of rock in the back or in pillars.
- Relaxation of stress or dilation of structures can also destabilize 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. This will provide sound geotechnical data to cover all ancillary development in the vicinity of the extraction workings.
- The location of shafts usually follows requirements: namely near the existing plant or in the footwall of the orebody. The site must be examined in detail as it and the haulage’s are the main artery of the operation.
• 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 FLAC3D, EXAMINE3D or MAP3D). Two -dimensional stress analyses (using such programs as FLAC 2D and PHASE 2) can also be used, but being two-dimensional usually gives 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 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.

2.0 PASSES

Ore passes located on the extraction level must be well supported as repairs have an adverse effect on production, even though LHD’s can be directed to nearby passes. In high stress areas the extension of these passes below destressed zones have been subjected to extensive stress spalling resulting in decisions to avoid these layouts and to move ore laterally. Orepasses outside the production area, but close to it must be carefully sited to avoid abutment stresses and weak ground.

3.0 INTERNAL VENTILATION DRIFTS

Internal ventilation drifts are located immediately (15m) below the extraction level in stress free areas. However they do require full support to last the life of the operation.

4.0 HAULAGES, VENTILATION DRIFTS, WORKSHOPS, CRUSHERS

These excavations are located outside the area of influence of the mining operation except for haulage’s connecting these facilities with the production areas. Support design will be unique to that particular excavation and must follow the basic principles. In complex layouts such as workshops, pillar design must follow basic rules.

5.0 GENERAL NOTES

* Site and design the ancillary development, employing the same sound design parameters used for the actual caving layout and support.
* Do not site the ancillary development too close to the cave area where potential damage is higher.
* Consider numerical modelling to determine the level of induced stresses likely in the vicinity of the planned ancillary development. Consider the problem in 3 D.
* Remember all the aspects of the design for all ancillary development: Accesses, drainage, ventilation (intake and exhaust), ore-handling (density and bulking factors), hauling, hoisting
* Logistics for Men, Materials and Machinery (routes and storage)
* Sand and Stone, complete with batching bays for shotcrete and concrete
* Services: Power (substations, distribution boards, LHD plugs if electric) Compressed Air, Water (machine and drinking), Sanitation, Fuel, Workshops (maintenance), Lunch-rooms, Escape routes (second egress) and refuges
Chapter 21

ROADWAYS

1.0 GENERAL

The design of roadways is still a little controversial in terms of whether rails should be embedded in the concrete floor as wearing protection. Whether there should be two layers of concrete of different qualities, whether gravel on the concrete is the right route or whether gravel only 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. Cassiar mine experienced heaving floors and did extensive floor support, which did help prolong the roadway, but was an expensive operation. Northparkes used various combinations to produce hard wearing floors as described in the manual. Abutting slabs and interlocking slabs have been used. Both systems require a good base. The abutting slabs are not successful and the interlocking slabs were successful for smaller LHD’s. Roller compacted concrete has not been used even though it has been recommended and has many merits. Premier Mine has experimented with various systems and none really that successful even though they brought in road specialists.

At Henderson mine the roadways are basically very simple. The floor is not cleaned to bed rock and a 30MPa 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.

It is apparent that where floor conditions are good then roadways do not present a problem. The problems arise when floor conditions are poor due to swelling ground from the effects of water or heaving ground due to stress or a combination of the both. The decision that has to be made is, is it feasible and economical to support the floor to stop the floor heave? (Ch-20 p 2) The effects of water can be eliminated or reduced by a comprehensive drainage program and good housekeeping.

2.0 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. here 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 roadway practice of massive concrete with embedded rails (Ch20 p 5). Objections to rails can only be based on experience of poor installation. The measure of success of roadways is the floor conditions and the tonnage hauled before repairs and the ease of repairs as well as time in terms of loss of production.

Bell Mine spends 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.
Decide on what roadways are to be used on the project as different materials and standards might be selected for different areas / duties. Look at roadways as a total cost concept:

- Installation costs
- Maintenance costs
- Operating costs for equipment
- Savings on support maintenance
- Improved safety / working environment

Poor roadways are an insidious user of expenditure in operating costs. The floor area of the drawpoint is subjected to an extensive ‘work load’ from the tyres during loading, the bucket on the floor, impact from rocks and secondary blasting and must be designed accordingly.

3.0 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.**

4.0 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 and the holes form again. As regards heaving floors it is standard practice to dig out the excess material. This however does reduce the foundation support to the sidewalls and can have an overall major effect. Therefore is prevention not better than questionable cures.

5.0 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.
6.0 GOOD HOUSEKEEPING

Good housekeeping and good drainage result in:

* Great improvement in safety
* LHD costs are reduced by as much as 50% and production efficiencies are increased owing to:
  • reduction in tyre wear,
  • reduction in wear on the center articulation,
  • increased tramming speeds,
  • reduction of sidewall impact (affects machine and support).
* Improved working environment, resulting in improved efficiencies.

7.0 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 shown in the table at the end of the text.
<table>
<thead>
<tr>
<th>ROADWAY TYPE</th>
<th>ADVANTAGES</th>
<th>DISADVANTAGES</th>
</tr>
</thead>
<tbody>
<tr>
<td>Conventional concrete</td>
<td>Good finished surfaces 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.</td>
<td>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 preferably 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.</td>
</tr>
<tr>
<td>Interlocking Roadway Bricks</td>
<td>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.</td>
<td>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.</td>
</tr>
<tr>
<td>Compressed Bricks – e.g. G pattern, cubic or other varieties.</td>
<td>Same as the interlocking brick, except easier and cheaper to make.</td>
<td>Same as the interlocking brick, but as laborious if not more so to place and tend to tear out on corners.</td>
</tr>
<tr>
<td>Reinforced Concrete with rails/RSJ’s</td>
<td>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.</td>
<td>If footwall heave occurs the whole floor lifts and access into the area can be prevented or huge damage to tyres results.</td>
</tr>
<tr>
<td>Roller compacted concrete</td>
<td>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.</td>
<td>Underground this has proved difficult to install as the controls have to be exceptional to make the concrete to the required strength.</td>
</tr>
<tr>
<td>Steel Plate in the loading area</td>
<td>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.</td>
<td></td>
</tr>
<tr>
<td>Run of Mine Gravel</td>
<td>Easy to lay</td>
<td>High rolling resistance to vehicles. Maintenance costs ongoing and higher. Material must be suitable.</td>
</tr>
</tbody>
</table>
Chapter 22

INDUCED STRESS

1.0 GENERAL

The object is to identify all the mining induced stresses. This means mining the deposit on paper in order to have a ‘feel’ – a 3D visualization for all facets of the operation. 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. Support design must be based on the eventual effect the induced stress will have on the rock mass and not the appearance of the rock mass in the development end.

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. Variations in regional stress not only occur with increases in depth but also with topography such as under valleys in mountainous country.

2.0 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.

3.0 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 prior to the propagation of the cave 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 on 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 (Ch21 p 2).

4.0 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. The high induced stresses are on the production level where the stress relief occurs with rock mass failure. However, in the case of an advance undercut, the development on the undercut level would 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. This factor must be recognized in size of development and support design.
5.0 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 column 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.

6.0 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.

7.0 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.

At Shabanie Mine numerical 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. The toe stresses develop with time as the cave moves upwards therefore there is time for the rock mass to adjust (Ch21 p 3).

8.0 CAVE BACK STRESSES

The orientation and magnitude of the induced stresses in the cave back have a major influence on fragmentation and caveability. 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. In high stress environments cave back stresses become significant and result in stress spalling – improved caving and fragmentation – and seismic events. The rate of caving must be controlled to eliminate the seismic events.
Chapter 23

ROCK MASS RESPONSE

1.0 INTRODUCTION

The response of the immediate surrounding rock mass to block caving is referred to in several of the previous sections as:-

* Caveability
* Air Blasts
* Rockbursts
* Massive wedge failures
* Induced stresses
* Subsidence and cave angles

What also needs to be considered are:-

- The influence of changes in size of excavation,
- the influence of several distinct caving operations,
- the role of major and minor structures in determining the extent of the failure zone,
- the geometry of the excavations on the remote surrounding rock mass.

It is also worthwhile to summarize the response in the immediate surrounds and how to predict response in subsequent operations and its influence on support and sequence. There are cases when the effect of greater depths and the subsequent changes in the stresses have not been recognized in designing the next lift and serious problems have occurred. In certain rock types such as kimberlite the response to drift development can be significant with fracture zones extending 2m into the sidewall. Introduction of water into certain rock types will result in deteriorating conditions. If response can be measured up to 300m away from the cave then it is reasonable to predict that the failure zone in weaker rock masses will be extensive.

2.0 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. The larger the area mined, the greater the response.

3.0 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 (Ch2 p 2). 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. In the case of Henderson mine the first production level was at 8100’ and the next on 7700’ level an elevation difference of only 400 feet or 123m which is not significant in mountainous terrain where the range in elevation is from 584m to 1285m. The bulk of the 7700 level is overcut by the 8100 layout (Ch22 p 3)

In mountainous country increase in depth could have a significant impact if the increase in depth coincided with major topographical changes such as from the side of a mountain to below a valley resulting in a totally different stress environment:

4.0 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.
Chapter 24

SUBSIDENCE AND FAILURE ZONES

1.0 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 drifts, declines, orepasses, shafts, surface installations 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 summarizes the many factors that have been discussed in detail in the cave mining manual:-

* 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, topping of the upper slopes often occurs. The same thing will happen when the cave breaks through into a large pit.

2.0 CHARTS TO DETERMINE SUBSIDENCE ZONE

The cave angle defines the boundary of the subsidence zone / crater, which is the 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. In small equi-dimensional subsidence zones there are examples of overhangs forming below prominent shear zones.
The above diagram is a conservative approach and should be used for siting important infrastructure such as shafts or plant, whereas the following diagram is less conservative and would be used for draw control and water inflow calculations.

\[
\text{Factor} = \frac{\text{Density of caved material} \times \text{height of caved material} \times \text{depth}}{1.5 \times 100}\]

The MRMR will recognize stress effects and whether joints are clamped as on the minimum span or in tension on the maximum span side.
3.0 FAILURE ZONES

A failure zone can be described as the envelope of failed rock surrounding a cave zone in which the fracture and joints have failed with limited movement. Often the limiting factor in the movement is the confining stress from the caved mass. As the caved column is drawn this confinement decreases and more pronounced movements take place with lateral extension of the cave. Failure zones occur below the cave on the production level as the rock mass responds to the removal of the vertical stress. Good support on the production level nullifies this effect. Where orebodies are separated from one another, the pillar between the two will have a well defined failure zone. Failure zones must be defined so that important excavations are not developed in these areas. An empirical guide is given at the bottom of the following diagram.

A failure zone against the cave boundary at Andina Mine is shown in the manual (Ch22 p 5).
EFFECT OF MAJOR STRUCTURES

FAULT DAYLIGHTING

STEEP FAULT

TOPPLING

This is a fairly common occurrence in hilly terrain.

CRATER

Arching in the caved material applies restraint to the cave boundary.

Caved material

Failure zone - [1] MRMR

The failure zone is a 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 - 60 = 20m, + 61 = 10m. Figures based on empirical data.
Chapter 25

EXCAVATION STABILITY

1.0 GENERAL

The stability of the various excavations must be assessed by recognizing their different spatial and geological environments and the life of the excavation. Also to be taken into consideration is the stress changes to which the rock masses surrounding the excavations might be subjected and whether the rock masses will weather. Sound looking ground during the development stage will deteriorate when subjected to high stress changes unless adequately supported. The support can only be designed if the excavation stability is determined. As with so many planning requirements in cave mining the following factors need to be recognised:-

* The geological environment in terms of IRMR and MRMR and structural geology.
* The induced stresses and the height to base ratio.
* Abutment zones on all levels, which occur during the initial undercutting where the undercut area is between 60 to 120 percent of the anticipated caving hydraulic radius.
* The abutment zone on all levels, at the final undercut limit or where the undercut is stopped between panels or blocks is a result of production demands.
* Areas where wedge failures are likely to occur.
* Areas of potentially squeezing ground.
* Areas of potential rockbursts.
* Rock masses that weather / deteriorates – effects of water.
* Excavation size and orientation (3D)
* Drilling and blasting of the excavation
* Mining sequence
* Poor draw control
* Safety
* The required life of the excavation
* Support – design, installation and maintenance based on above.

In other words the planning engineer must develop a ‘feel’ for how that excavation will behave for its planned life. Correct assessment of the mining sequence can mean the difference between a sound excavation with no production problems and excavations under continuous repair.

2.0 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’ (Ch24 p 4). Column loading at one of the Kimberley mines led to the failure of the slusher drift with its one metre thick, 60MPa concrete lining and an underlying drift 15m below.
3.0 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 column heights and a relatively small base, the height to base ratio is large and the bulk of the ‘weight’ is carried by the sides through internal arching. And vice versa, if the height to base ratio is low then the stresses on the base will increase and the stresses on the base will be higher in the centre of the mining area than on the sides:-

4.0 INDUCED STRESSES

The dominant induced stress is the abutment stress ahead of the advancing cave front. In conventional layouts the abutment stress in high regional stress environments would be high enough to cause extensive fracturing and failure in spite of the area being supported. The object must be to reduce the stress values and thereby reduce the damage. 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 cave front.
5.0 STRUCTURES AND SQUEEZING GROUND

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 owing to the mining sequence. Shear zones or zones of weak rock can deform resulting in squeezing ground. Squeezing ground has two problems, one is that to control the squeeze into the drift requires a high level of support and the other is that the movement of the ground sets up tensile conditions at the contact causing strong rock to fail.

6.0 ROCKBURSTS

The rockburst potential is a function of the stress distribution, 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. Any excavations that can be defined as having rockburst potential should have a yielding support system.

7.0 WEATHERING

Certain rock masses weather in contact with moist air and also with water inflow. Some kimberlites are particular prone to extensive weather when water is present resulting in a rock mass at a fraction of the original strength. The solution is to seal the rock and to control the inflow, but it is amazing how often this is neglected creating major problems. Dunites weather on exposure to air and need to be sealed if the weathering is going to be extensive.

8.0 MINING FACTORS

**Excavation size** - The size of the excavation must be based on good geotechnical investigation and the determination of the excavation stability not on the largest LHD available.

**Grade** – There are examples of lack of attention being paid to grade during development with the result that drainage becomes a major problem. This not only results in rock mass deterioration, but also in creating production problems due to wear of roadways and poor hauling conditions.

**Overbreak** of up to 35% has been measured which means potential excavation instability as the support would have been designed for the planned size.

**Drilling and blasting** - Sound drilling and blasting techniques are required to ensure minimum damage to the rockmass. The blast fracture zone around the drift does permit a certain amount of stress relief. Roadheaders or tunnel borers will give a smooth finish and an apparently stable excavation near the face. However, high stresses in the periphery often lead to failure back from the face.

**Pillars and brows** – The production level drifts and drawpoints are located in pillars (major apex) and their stability is dependant to a large extent on the stability / strength of the pillar(s). 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 then 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; rock reinforcement and a strong lining are required. Wear on the pillars will increase the role of the drift support.
Support - The support has to be designed for the life of the operation, which is a function of draw height and rate of draw. Support must be installed as early as possible and not when it is convenient to do so. Development advance at the expenses of the correct support can only lead to high repair costs and production problems as well as higher working costs.

Mining sequence - The mining sequence must recognize the above factors so as to minimize 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.

9.0 MAJOR COLLAPSES

Large areas at Teniente Mine have been affected by collapses which cover up to twenty drawpoints, representing a significant tonnage loss (Ch24 p 4). It is not possible to design excavation stability for major collapses; the mining sequence must avoid these situations. 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.

10.0 WEDGE FAILURES

Massive wedge failures occur when a cave front is advanced on a line towards an unfavourably dipping major structure. This is particularly the case in a panel retreat operation where there is slow rate of advance. The removal of lateral restraint to the caving face allows structures to open and for the wedge to settle and to load the pillars on the production level. This situation has led to major collapses and loss of production. If the wedges are large then rehabilitation of these areas is very difficult, often the only solution is developing a new production level on the underlying ventilation level. Where there is no alternative to advancing towards a major structure then the advance must be at an angle through the structure and the undercut ends are advanced through the structure one at a time. The object is to allow the wedge to break up over the undercut. One of the disadvantages of a pre-undercut is that massive wedges can settle and load the potential major and minor apexes and there are no drawpoints to relieve the ‘weight’.
11.0 FAILURE ZONES

A failure zone can be described as the envelope of failed rock surrounding a cave zone in which the fracture and joints have failed with limited movement. Often the limiting factor in the movement is the confining stress from the caved mass. As the caved column is drawn this confinement decreases and more pronounced movements take place with lateral extension of the cave. Failure zones occur below the cave on the production level as the rock mass responds to the removal of the vertical stress. Good support on the production level nullifies this effect.

Failure zones must be defined so that important excavations are not developed in these areas. Where orebodies are separated from one another the pillar between the two will have a well defined failure zone.

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 - 60 = 20m,
- 61 - 70 = 10m.

Figures based on empirical data.

12.0 SIZE OF MINING AREA

It is often the case that rock mass response only becomes significant, once the cave zone 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.

13.0 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.
Chapter 26

SUPPORT

1.0 GENERAL

Support is installed in excavations to ensure rock mass stability during the course of the mining operation. The support system is designed to cater for all potential events and this means good installation in properly developed excavations. However, support is often installed in excavations that have been poorly developed with excessive overbreak, poor blasting, incorrect grades and poor alignment and the support is then expected to rectify these errors. The correct support for a 4m x 4m excavation with smooth wall blasting can hardly be expected to work in a 4.6m x 4.6m end with a 1m blast fracture zone. Overbreak can mean 32% extra rock removed and far more expensive support to cater for the poor conditions that have been created. Unfortunately this is a common occurrence when developers have no concern with the finished product. It is management’s responsibility to ensure that development standards are implemented and maintained.

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 design the basic support to cover the general situation and then to come back and install additional support where required. This does not apply to those situations where spiling is required, here spiling must be done as designed, there are no short cuts. Support is designed for the life of the operation and therefore there must be a correct assessment of potential instability in time and place.

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. The drawpoint drift needs to have rock reinforcement and a strong lining. The details of support design are recorded in the manual; it has many facets and details that must be considered. Rock reinforcement systems are integrated systems and each component must be correctly installed (Ch25 p 5,7).

2.0 INSTABILITY

This section summarises the factors that can affect the stability of the excavations. The Rockmass strength is known or should be known, not as an average classification rating, but for distinct rock mass rating zones.

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. The direction of undercutting with respect to the principal stress can have a significant effect on the induced stresses. Abutment stresses in conventional layouts cause extensive fracturing of the rock mass ahead of the undercut, often prejudicing the already installed support and adversely affecting the stability of the brows.
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.

ROCKBURSTS - The rockburst potential is a function of a high stress environment with stress concentrations in abutments of conventional layouts manifesting in pillar failure. 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. 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.

SQUEEZING GROUND - In softer rocks or weak shear zones in more competent rocks high stresses manifest themselves in squeezing ground conditions. Weathering of certain rocks, such as Kimberlites, also create potential squeezing ground. Squeezing ground is difficult to control and the mining sequence must always be examined in an effort to reduce these effects. It is essential to have the right equipment and personnel available to cope with the squeezing ground. Yielding steel arches are used in squeezing ground, but only if the arches are removed and replaced when they have fully yielded.

MINING FACTORS: -

- The excavation size must be based on good geotechnical investigation and not a design based on the largest LHD available. Man made errors leading to overbreak, excessive water and incorrect grade, nullify good support design.
- Sound drilling and blasting techniques are required to ensure minimum damage to the rockmass.
- The mining sequence must recognise the above factors so as to minimise damage to the extraction level. 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.
- 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’
- In the conventional layout it is common for the pillars to show a high degree of fracturing after the undercut has passed, as these ‘pillars’ are only 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.
3.0 SUPPORT DESIGN NOTES

* The lateral restraint on the rockmass is vital and the use of deep anchoring, particularly of the sidewalls, is essential.
* Design the support systems as a complete system covering the various phases well before the excavations are developed. There must be full awareness of the installation difficulties that might be created and match the various elements in the system in terms of capacity and suitability. The support must have load bearing capabilities and yield capability to maintain support in the event of deformation. Also it must have capability to withstand shear displacements, protection from vehicular damage and the ability not to corrode or deteriorate with time.
* The operating personnel must buy into the design/s selected and ensure that they are correctly installed. Poor, incorrect (less than the design) support installed is expensive. Operating personnel must be aware that the level of support can be increased by them, in the interests of safety, but must be reported.
* The support design must include the floor if it is not stable and to link the rest of the system into it. Floor support is often ignored because it is inconvenient for the developer, but leads to many production problems.
* Have procedures, to cater for special cases, which are fast and fully proactive.
* Set standards for every element, the sequence, the timing, a monitoring system and procedure when non compliance occurs.
* Have a standard procedure on how contractors are to be managed.
* Avoid having bolts of different lengths as the tendency is to install shorter bolts where long bolts are required
* Support failures have a ripple effect and insidious hidden cost implications, taking excessive time and resources to rectify had they not failed if installed correctly at the start. Always use good support materials. Particular care must be taken in chemically active or corrosive environments.
* Make a model/s either of the system in stages or of the complete integrated system to aid the understanding of both design and operating personnel.
* The sidewalls on the extraction horizon are potential weak areas as the pillars are small and vulnerable. This, together with the wide-spans created by the drawpoint takeoffs, makes the correct well-designed and installed support vital.

4.0 UNDERCUT

Whilst the undercut is a temporary phase of the caving operation it is the most important and therefore its stability must be ensured. Doing repair work on the undercut level which delays the commissioning of drawpoints is not acceptable. In the advance undercut situation there are more ends than in the conventional layout and support must be designed for high stress situations.

5.0 NOTES ON SUPPORT ELEMENTS

GROUTING - Grouting is a vital component of many of the following support elements and is often neglected. Holes should be fully grouted with stiff, durable grout. Low pressure grouting of open fractures or loose ground conditions has been successful.

BOLTS - The bolt length is designed for the worst conditions, formulas are available (Ch25 p 8). Avoid having short and long bolts in the system, deep anchorage is done with cables or jointed bolts. Ensure that the bolts apply constraint (fitted with well designed washers) on the sidewall, operating on the lining where one is to be used. Design the directions for the bolts to achieve maximum effectiveness, coupled with ease of installation.
STRAPS – Straps improve the constraint on the sidewall particularly if in intimate contact with the lining. Keep the straps as straight as possible, blocking from sidewall to strap to increase the effectiveness. Radial straps tend to buckle under load and longitudinal straps are more effective. Overlap the straps to make a continuous interacting restraint along the drift.

SPILING - Plan the spiling operation meticulously with suitable standards and controls taking particular care in selecting the length, material and spacing for the spiling bars. Ensure that the exposed end of the spiling bars is secured. Place restraints and controls on development loading where spiling is being installed.

CABLES, CABLE CLAMPS, ROPES - Cables are used to give deep seated anchorage to the integrated support system. Cable slings have been effectively used on bullnoses and camelbacks (Ch25 p 28, 29). High strength ropes (hoist) are used on bullnoses and camels and provide excellent yielding support.

SHOTCRETE – Shotcrete must be reinforced, preferably with chain link mesh (Ch25 p 6). Fibres are used but are not as effective as chain link. Whilst laboratory test might show good results underground failure is by cracking of the shotcrete which reduces the effectiveness of the fibres (Ch25 p 19, 20). Weld mesh might be easier to install, but it fails at low loads at the welds (Ch25 p 16). Prepare the surface well before shotcreting and monitor the application closely; it is easy to camouflage poor workmanship. Shotcrete should take place right to the footwall of the drift.

ARCHES (steel, reinforced concrete, lattice girder) - Brace the arches one to the other and reinforce the concrete, spreading any load that might impact on the structure and protect the arches from damage, vehicular and blasting in particular (Ch25 p 37 – 45).

SUPPORT PROTECTION - Protect the support from vehicular strikes, particularly: on corners, at takeoffs, where drifts narrow by design, poor mining or increased support (arches), in reloading bays and at truck loading areas (Ch25 p31).

6.0 HIGH STRESS SUPPORT REQUIREMENTS (Ortlepp & Stacey)

Seismicity associated with deep block cave mining operations in hard rock can give rise to dynamic loading of the mining openings. In addition, production level drifts are often subjected to large static deformations during mining. 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.

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, yield ability, quality of installation, quality of containment support).

7.0 SUPPORT SYSTEMS FOR LARGE STATIC AND DYNAMIC DEFORMATIONS

The results of the testing programmes described in the manual 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.

Bear in mind support can corrode and might not be visible to cursory examination. Many mine atmospheres are extremely corrosive and mine water can be significantly acidic. Very rapid corrosion of steel fibres can occur to the extent that the fibres no longer contributed to the integrity of the support system. Use of monofilament polypropylene fibres can be very successful in these corrosive conditions.

It is also necessary to consider what the requirements for support of caving layouts are. 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.
- retention 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 retention and containment elements must be compatible, and all support elements must be matched in terms of capacity.
### 8.0 SUPPORT TECHNIQUES

<table>
<thead>
<tr>
<th></th>
<th>Low stress</th>
<th>High stress</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>Bolts length</strong></td>
<td>(= 1m + (0.33 \times W \times F))</td>
<td>(= 1m + (0.5W \times F))</td>
</tr>
<tr>
<td><strong>Spacing</strong></td>
<td>1m</td>
<td>1m</td>
</tr>
<tr>
<td><strong>Type</strong></td>
<td>Rigid</td>
<td>Yielding cone</td>
</tr>
<tr>
<td><strong>W</strong> = width of drift</td>
<td>(F = \text{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)</td>
<td></td>
</tr>
<tr>
<td><strong>Deep seated</strong></td>
<td>Cables = (1m + 1.5W)</td>
<td>Ropes = (1m + 1.5W)</td>
</tr>
<tr>
<td><strong>Mesh</strong></td>
<td>0.5mm x 100mm</td>
<td>0.5mm x 75mm</td>
</tr>
<tr>
<td><strong>Linings</strong></td>
<td>Mesh reinforced Shotcrete</td>
<td>Mesh reinforced Shotcrete</td>
</tr>
<tr>
<td></td>
<td>Rigid steel arches</td>
<td>Yielding steel arches</td>
</tr>
<tr>
<td></td>
<td>Massive concrete</td>
<td>Reinforced concrete</td>
</tr>
<tr>
<td><strong>Surface restraint</strong></td>
<td>Large plates - triangular</td>
<td>Large plates - triangular</td>
</tr>
<tr>
<td></td>
<td>Open straps</td>
<td>Open straps</td>
</tr>
<tr>
<td><strong>Corners</strong></td>
<td>25mm rope / cable slings over shotcrete at 1m vert.</td>
<td>25mm rope / cable slings over shotcrete at 0.7m vert.</td>
</tr>
<tr>
<td><strong>Brows</strong></td>
<td>Birdcage cables into brow</td>
<td>Birdcage cables into brow</td>
</tr>
<tr>
<td></td>
<td>(&lt;--- 75mm pipes inclined towards brow from d/p-----&gt;)</td>
<td>(&lt;--- 75mm pipes inclined towards brow from d/p-----&gt;)</td>
</tr>
<tr>
<td><strong>Repairs</strong></td>
<td>(&lt;---------- Grouting, mesh reinf. Shotcrete, extra bolts, Plates, straps and cables-----------&gt;)</td>
<td>(&lt;---------- Grouting, mesh reinf. Shotcrete, extra bolts, Plates, straps and cables-----------&gt;)</td>
</tr>
<tr>
<td><strong>Arches</strong></td>
<td>Yielding arches</td>
<td>Yielding arches</td>
</tr>
</tbody>
</table>
Chapter 27

EXTRACTION LEVEL AND DRAWPOINT REPAIR

1.0 GENERAL

Extraction levels and drawpoints in very poor ground conditions such as squeezing ground might need repairs as part of the operation. In these cases a monitoring system must be set up and repairs effected immediately. In other cases repairs are needed when abutment stresses have damaged the rock mass prior to production, or when the original development and support was not of the right standard – these repairs are avoidable. It is obvious that maintaining a high standard of development and 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 and in some cases a loss of ore.

2.0 REASONS FOR FAILURE

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. Repair in these situations is difficult particularly if the rock mass is not strong. The only available procedure is to reinforce the rock mass, install a substantial drawpoint lining and draw at a higher rate in the hope that the pillar will be destroyed.

TONNAGE DRAWN - 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.

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, the support techniques described in the support section can be used. However, where there has been a major collapse, with little chance of successful 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 the brow and pillars necessitating extensive repairs of an oversize drawpoint. The repairs would consist of rock reinforcement and massive linings.
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 of the drawpoint lining 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. A reinforced concrete brow was placed in a drawpoint at bell mine with great success (Ch25 p22).

3.0 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 (Ch25 p 28, 29, 30). Badly damaged corners have been rebuilt with reinforced concrete. There is no excuse, however to let them deteriorate to that extent.

4.0 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 steel 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.

5.0 MONITORING AND RESPONSE

It cannot be too highly stressed that all areas must be monitored on a regular basis and damage reported immediately. A standard scale of damage is required and plotted on plans. Immediate response by support crews to accelerating damage is essential. Knowledge of rock mass response to different situations and the degree and rate of damage must be obtained by having a monitoring system in place to detect and report damage as soon as it occurs and to track its progress. The extent of damage recorded must be defined. As soon as the damage is seen, determine the reason and rectify the root cause. Determine if repairs are warranted for example tons remaining for draw. Early maintenance / repairs are cost effective so do not skimp on the re-support or support repairs. The re-support or repair must overlap well into undisturbed ground and if anything tend to ‘over support’. The overlap must be thoroughly cleaned and prepared before the re-support / repair.
Chapter 28
DRAW COLUMN HEIGHTS

1.0 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.

2.0 EFFECT OF OREBODY GEOMETRY

The effect of orebody geometry is most noticeable in dipping orebodies

![Diagram of orebody geometry](image)

Extraction level

3.0 POTENTIAL DRAW HEIGHTS AND DILUTION

If the interactive theory is accepted, then the higher the draw column, the lower the dilution will be. 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 draw the quantity of dilution in the draw column increases as the column height increases. If the entry percentage is 60%, then that figure would apply to the ore column regardless of height. Compared with the height of interaction zone calculation this means a significant difference in the dilution tonnage.
4.0 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 model base was 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 materials- the vertically placed core having a 15% higher base load than the aggregate:-
Chapter 29

MINING SEQUENCE AND METHOD

1.0 GENERAL

The mining sequence must be investigated in detail and all options to be considered. It is important to think through the long term planning for the whole mine and not only for the initial panel / block / project. The following factors must be considered:-

* Induced stresses for the different scenarios contemplated
* Does the sequence influence fragmentation?
* The production and economic requirements with respect to the grade distribution
* The geological environment in terms of zoning of IRMR and MRMR
* The ongoing mining and future dilution
* The mud rush potential
* The rockburst potential
* The potential for structural failure of the rock mass or massive wedge failures
* The influence of adjacent mining operations

2.0 MAJOR FACTORS

CAVEABILITY - In a caving situation where there are no caveability problems and there is uniform fragmentation then there is certain latitude in deciding on the sequence and economic aspects might be the deciding factor. However, if there is a significant variation in caveability and fragmentation then the sequence must be examined with these factors in mind. For example, the accepted practice is to retreat from weak ground to good ground; however it might be better to retreat from the good ground to the weaker ground in order to ensure better caving and better fragmentation- this was the case on San Manuel mine. The logic is that the good ground is subjected to caving stress for a longer period and when the cave propagates to an optimum height the weak ground will not be subjected to high abutment stresses.

OREBODY SHAPE – If the orebody has regular and semi parallel outlines few problems are going to be expected as there will be no reduction in the minimum span to inhibit caving. If the outline is irregular then a decision must be made whether to start in the narrow section so as to provide time for caving or in the wider section and take a chance that the narrow section will cave.

INDUCED STRESSES – The major induced stress is the abutment stress and its value will depend on whether the cave front advances towards or away from the principal stress. Advance away from the principal stress in weak ground and towards the principal stress in strong ground.

FACE SHAPE – A concave face is more stable than a convex face, thus it is important that a decision is made whether face stability is required. For example, in free caving ground control of caving might be a requirement when the front caving method is used (Ch5 p 17).

FACE ORIENTATION - A face orientated parallel to major structures dipping towards the face can result in mass failure of the rock mass and loading of the extraction level before undercutting has been done. To be avoided in front caving situations.
3.0 BLOCK OR PANEL RETREAT SYSTEM

In the case of large orebodies a major decision has to be made whether to cave the orebody as a series of blocks or as a panel retreat. The draw height, production potential and production requirements play an important role in this decision. In the case of the panel retreat layout a high draw height and a low production will mean a slow advance in commissioning drawpoints and a steep angle to the ore waste interface (Ch28 p 4). The following diagram illustrates the difference between a block cave and a panel retreat layout:

![Diagram of Block Cave and Panel Retreat](image)

The blocks are brought into production in sequence so that as a block goes out of production a new block is producing. The panel retreat is a continuous operation with depleted drawpoints being replaced by new drawpoints. One of the important factors with a panel retreat cave is that the ore / dilution interface is shorter than with a block cave. In the above diagram the difference is 33%. With a low angle interface the dilution direction is basically vertically downwards and easier to control and ore does not move across the interface. However, in the case of the block cave where there is vertical and horizontal interface, dilution can move across the vertical face into the ore zone and ore can move into the dilution zone and be lost. Ore loss with a panel retreat is less than a block cave, where ore can move sideways across the vertical interface during the drawdown and be lost in the dilution zone.

Another disadvantage of the block cave is that as the cave of the new block propagates the cave back will be against the dilution zone of the previous block and if there is an air gap dilution inflow can be significant. In the case of the panel retreat an air gap will allow dilution to migrate down cave back, however because of the larger surface area of the cave back, caving should proceed more satisfactorily than with caving of the smaller individual blocks in a block cave.

4.0 METHOD SELECTION

Because of high labour costs mechanization and the depletion of secondary finely fragmented ore deposits the automatic choice is to select a method involving mechanization. This could also be due to the younger generation not being familiar with grizzly and slusher systems.

GRIZZLY LAYOUTS – Grizzly layouts were extensively used on the finely fragmented secondary ore copper mines where productivity could be as high as 700 tons per man shift. In finely fragmented ores a grizzly layout must be considered.
SLUSHER LAYOUTS – Slusher layouts were a step towards mechanization but were not that productive and the method did not lend itself to good draw control, in spite of theoretical efforts to improve the control. The method was open to abuse in preferential drawing of fine ore.

HORIZONTAL LHD LAYOUTS – Horizontal LHD layouts have been in operation for many years and there are numerous examples of good and mediocre layouts. Horizontal layouts are particularly suited to steep dipping orebodies. Draw heights of +400m are possible if the rock mass on the production level can withstand the wear and tear. Improvements in undercutting techniques such as advance undercutting will reduce the abutment stress damage to the major apex and brow meaning that the rock mass can be properly supported and will withstand the attrition from the draw. Proper draw control will ensure that there is no column loading of the apexes.

There is a tendency to plan a layout for large equipment resulting in drawpoint spacing not entirely suited to the fragmentation resulting in ore loss and higher dilution. This is particularly the case when there is a decision that the LHD must be straight before entering the muckpile. On a mine this resulted in drawzone spacing across the major apex on 25m, when it should not have exceeded 21m. In the past the LHD was regarded as a unit to load and dump with a minimum haul so that orepasses were kept close – 90m. There is a tendency now days in use are the staggered herringbone and the diagonal Teniente layout. The advantages and disadvantages of these layouts are discussed in the layout section.

INCLINE DRAWPOINT LAYOUTS – Incline drawpoint layouts have been successfully used in dipping orebodies and in steep dipping orebodies where there are rock mass problems with a horizontal layout.. There are advantages and disadvantages with an incline layout compared to a horizontal layout:-

If the incline layout is properly overcut so that there is no chance of wedges loading the layout, then the advantages are:

- The structure is stronger than a horizontal layout because of the single drifts.
- Brow wear is not critical – a brow loss of 2m will not affect the strength of the structure or the draw.
- Because of the longer and straight drawpoints larger equipment can be used and loading is straight into the muckpile.
- Space for secondary blasting activities which do not interfere with production
- Good drainage
- Support of the drift and brow is simpler and more effective
- External ore passes

Disadvantages:

- More development than with a horizontal layout, this must be compared with height of draw, productivity, improved ore recovery, lower operating costs.
- Compared to an horizontal layout with internal orepasses then haul distances are greater.
Chapter 30

OVERALL DRAW STRATEGY AND DILUTION PRINCIPLES

1.0 GENERAL

Draw programs and dilution calculations are often designed in good faith by the planning departments and production schedules prepared from this data. However, all it needs is for on site expediency to occur in terms of overdraw of easy tonnage resulting in ore loss and dilution problems. The draw strategy must be approved by management and must have management backing in its implementation. Too often do we see lip service paid to well designed programs!

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.

All operating personnel must be aware of the draw strategy and the need for it to work. Do not change the strategy for expedience of short-term production; it can only be done after experience with the behavior of the orebody has been gained.

2.0 DRAW STRATEGY

The draw strategy will determine the draw control program. Develop a draw strategy that is practical, achievable and based on the best available information. As the object is to remove caved rock, the production call to achieve the right economic goals is of paramount importance. The studies leading up to the production stage 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 as planned. 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. This can only be achieved if all operating personnel are fully aware of the logic behind the strategy.

DRAWPOINT SPACING - The drawpoint and the subsequent drawzone spacing is a function of fragmentation and the competence of the rock mass on the production level. It is not based on a desire to use the largest equipment made. The details of the selection process are described in a separate section (Ch11). 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 - The planned strategy would be based on calculated fragmentation data for different areas and draw heights. These figures must be checked during the production stage as 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. The effectiveness of secondary breaking must be taken into consideration. In some more competent orebodies production is very dependant on secondary breaking.
DRAW ANGLES (Ch29) – The angle of draw is generally vertical with local variations; such as draw moving up shear zones or zones of finer fragmentation. The mining limit could be influenced by well developed shear zones on the ore boundaries (Ch29 p 3). Where there is a pronounced variation in the topography of the cave then draw angles can be inclined towards the highest areas (Ch29 p 2, 5). This phenomenon was used on King Mine to draw ore in the hangingwall.

DRAW RATE (Ch30) - The potential draw rate for the life of the block needs to be determined from all the geological and geotechnical data. As the cave matures the potential draw rate will increase as fragmentation improves. The planned draw rate would have been determined to allow for cave propagation, to avoid seismic events, to control dilution entry or the possibility of an air blast resulting from the development of a large air gap. The requirements for a safe propagation of the cave (start slowly and build up line by line of drawpoints in connection with the undercutting and drawbell development progress) must be calculated as the whole viability could rely on these numbers being reliable (Ch29 p2). The following factors are pertinent in arriving at the planned draw rate:

* Adhering to the draw control programme
* Fragmentation with time (location and frequency of secondary breaking requirements)
* The pre loading breaking (within the drawbells and drawpoint throat)
* The flow ability of the material
* The production potential of the LHD’s in that layout.
* The number of drawpoints available to each LHD (potential interference in the production cycle)
* The tramming distance and the complexity of the route.
* The draw program in terms of drawbells or drawpoints
* The logistics of ore handling is it a bottleneck?
* Ensure that that the draw rate is achievable.
* A monitoring program must be set up

UNIFORMITY OF DRAW – There must be sound records of tonnages from drawpoints as tons per shift and these records must be readily available. The procedure to sample drawpoints must be established both for grade and in terms of rock type to assess dilution entry.

DRAW PATTERNS - 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.

Panel Retreat Mining – A continuous panel retreat offers the best approach to controlling dilution.

Strip Mining – Strip mining offers a solution in the situation where a large orebody has to be undercut, but only a portion is required for production once caving has started. Or, if the orebody is overlain by a fluid dilution such as a fine shale then the ore can be removed in narrow strips moving across the orebody so the dilution is kept in the upper zone and not brought down to the bottom as would be the case with a block cave.
INTERACTIVE DRAW - In the manual in the section on drawpoint spacing, the 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 and uniform draw according to the draw strategy must be implemented to control dilution.

3.0 DILUTION PRINCIPLES

AREAS OF ORE TO ORE/WASTE INTERFACES - The higher the ratio of ore volume to the surface area of the ore/waste interface the lower the overall percentage dilution. In a massive orebody side dilution is negligible, whereas in a narrow orebody side dilution is significant.

ATTITUDE OF ORE/WASTE INTERFACE- Ore loss can occur with steep dipping contacts as ore can move into the dilution zone. Dilution control is more effective with uniform low angle interfaces.

FRAGMENTATION RANGE OF ORE AND UNPAY – Coarsely fragmented ore and finely fragmented 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 (Ch11 p 19 – 27). 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 (Ch 11). 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.

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 zone is readily released and will migrate more readily than the host rock then the value assigned to the ‘dilution’ in the drawpoint can be higher than its in situ grade. The converse applies if in the dilution zone the mineral is disseminated in competent blocks separated by fine unpay zones.
HEIGHT OF INTERACTION ZONE - DEGREE OF MIXING - The term ‘height of the interaction zone’ was based on observations 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 (Ch-11 p 5, 6, 7). 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. The higher the height of the interaction zones the earlier the dilution entry and the greater 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.
1.0 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 larger tonnage and to recover a larger mineral tonnage than is available from the ore reserve. The draw strategy will have been designed to maximize ore recovery and minimize dilution within the limits of a sound sustainable mining operation. Dilution entry is controlled by good draw tactics and maintaining a high standard of draw control. Monitoring the muck in the drawpoint is also necessary to control dilution. Details of dilution calculations are contained in the manual (Ch32 p 4, 5, 6). The draw strategy is designed to minimize dilution with the optimum ore recovery. There will be some repetition of comments made in the draw strategy chapter.

Dilution will originate from the hangingwall of the orebody and from the sides either from previously mined areas or from failed waste rock. The dilution zone must be analysed in the same manner as the orebody and defined in terms of fragmentation, mineral distribution and grade. Internal dilution refers to large blocks of unpay within an orebody which if easily identified can be trammed to waste.

2.0 FACTORS AFFECTING DILUTION ENTRY

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 diagram, the outline on the left shows that a large percentage of the ore is liable to dilution compared with the one on the right. This is indicated by the significant difference between the ratios of the ore area to the ore/waste contact area. 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 and coarse ore 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 result in appreciable overdraw owing to erratic sample values or visual assessment. In some asbestos deposits zones 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. If the waste is finer than the ore then dilution will be high unless the layout and draw strategy have been designed accordingly.

DRAWZONE INTERACTION AND DIRECTION OF FLOW - Good drawzone 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. The wider the drawpoint spacing the greater the likelihood of high dilution with associated ore loss unless strict draw control is practiced.

DIFFERENCES IN DENSITY - High density ore and low density waste leads to low dilution and vice versa.
BLOCK / PANEL / STRIP MINING - These draw strategies were discussed in the previous section. Panel mining with an angled ore/waste interface is the optimum shape for dilution control. Block mining with small blocks leads to high side dilution unless broken ore pillars are left at the contacts and mined in conjunction with the new block. Strip mining was introduced on kimberlite mines to control overlying fine shale dilution (Ch31 p 2).

3.0 DILUTION ENTRY

The values of the material surrounding and above the ore must be known. Consider when, how and from where the dilution will enter the draw column and how it will then behave. Calculate the flow of values through the drawpoints with time. To do this bear in mind the changing shape of the ore/waste interface with time, from the original outlines until it passes below the height of interaction zone.

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

4.0 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 an interactive zone followed by a zone of mass flow with even drawdown. For closely spaced drawpoints the contact between the two was clearly defined and its height above the drawpoints called the ‘height of the interaction zone. As the drawpoint spacing was increased, the boundary between the interactive zone and the upper even draw zone became less distinct, with pronounced peaks and troughs. 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. A large figure for ‘the height of interaction zone’ indicates a high degree of mixing. Thus, for low draw heights a high height of interaction means that the peaks could be in the waste zone with early dilution entry:-
Chapter 32

DRAW CONTROL

1.0 GENERAL

Draw control is what block caving is about, but you would not think so when some mines are visited. Expediency of maintaining a tonnage call regardless of the long term consequences is often the standard procedure and then when the grade drops or the planned tonnage is not drawn then all kinds of excuses are made. The reasons for and the principles of draw control must be clearly understood by all operating personnel. It has been repeatedly stated that the preparation of the orebody must be done in a sound way so that preventable problems do not hamper the planned draw. Poorly installed support, resulting from the lack of supervision/cutting costs - lead to repairs and unavailability of drawpoints. This has an adverse effect on the draw control program. The details of draw control procedure and calculation diagrams are to be found in the manual. 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 with minimum dilution.
- Avoiding damaging load concentrations on the extraction horizon.
- Avoiding the creation of conditions that could lead to air blasts or mud-rushes etc.

2.0 POTENTIAL DRAW RATE

The whole viability of the project could rely on the draw rate figures being correct. The potential draw rate will be calculated for the various stages in the drawing of the ore from the cave. The actual draw rate used in the draw control will be based on the build up to planned tonnages, prevention of seismic events, prevention of air blasts, number of drawpoints, size of LHD’s, ore handling facilities, secondary breaking requirements and the draw strategy – (drawbells or drawzones). Once the cave has matured then the draw rate will be standardized.

3.0 DRAW MECHANISMS

A prerequisite for designing a draw control program is an understanding of the drawdown mechanisms that occur in the block. The principles of what goes on in a block cave have been established over the years, by marker experiments, sand and gravel scale model experiments while more recently numerical modelling. The basis for good draw control is to establish a system of interactive draw. If drawpoints are drawn in isolation then rat holes occur with rapid introduction of dilution as shown in the following diagram:
The above diagram shows isolated draw and interactive draw with the intermixing that occurs in the interactive zone. The height of the interaction zone is also the height of the intermixing as previously described. With good interaction the subsidence in the upper portion of the cave is orderly and uniform and generally not influenced by small variation in the rate of draw from individual drawpoints. Large variations can result in steps. This mass flow is underlain by a zone of interaction and mixing 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 hang-ups 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.

4.0 PROCEDURE

By the time the orebody has fully caved the following need to have been examined:

- Any factors observed during the start of caving that will influence the planned caving and drawdown processes.
- Control the draw from the first tonnage into the drawpoint.
- Define the potential tonnages and grades that will be available from each drawpoint.
- The draw control system must be fully operational and would be tuned as new practical data becomes available during the early stages of draw.
- Confirmed that the planned draw strategy is correct.
- The recording and analysis of the tonnages drawn, this important aspect is often not treated with the required respect.
- Managing the draw by following the adopted draw strategy, any changes must be fully justified.
• Define how the control is to be monitored, maintained and audited. Set standards including corrective actions if and when required.
• It is important that personnel have a 3-D impression of the draw column. Think through how the draw column will behave with time; it can change.
• An estimation of the remaining tonnages and grade for future production scheduling and planning.
• Define the shape of the ore/waste interface with time from the original outlines until it passes below the height of interaction zone.
• Personnel must be aware of the definition of an isolated drawpoint.
• Decide how the horizontal redistribution of tonnages drawn is to be handled and establish the necessary programmes to handle it.
• Carry out reconciliation/s of value from actual tonne drawn (after mixing) against actual recovery and underground sampling and observations.
• Ensure the drawpoints are clearly and correctly identified underground.
• There must be a reporting system to record and describe why allocated drawpoints have NOT been drawn (particularly priority drawpoints).
• Report hung up drawpoints and action taken (including current status)
• Ensure secondary breaking can be and is done effectively and efficiently and standards are set and met.
• Always report the truth. Don’t falsify the records to please the draw control personnel and avoid the wrath of the production supervisors (this is fraud).
• Set up a liaison system for interchange of ideas, information, gripes, etc. between controllers and operators (break the barriers). Communications.
• Develop standard procedures for closed drawpoints.
1.0 GENERAL

This section is a prediction and reconciliation of what the recovery will be and has been 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.

2.0 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.

3.0 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.

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.

4.0 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 Mineralx100

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.

5.0 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.
<table>
<thead>
<tr>
<th>ITEMS</th>
<th>GOOD</th>
<th>RATINGS FAIR</th>
<th>POOR</th>
</tr>
</thead>
<tbody>
<tr>
<td>Draw column height dilution / fragmentation</td>
<td>+ 200m = 10%</td>
<td>80 - 200m = 7%</td>
<td>-80m = 5%</td>
</tr>
<tr>
<td>Mining area - m² large = less side dilution</td>
<td>+90 000 = 7%</td>
<td>± 40 000 = 5%</td>
<td>± 15000 = 3%</td>
</tr>
<tr>
<td>D/P spacing optimum - good interaction</td>
<td>= 10%</td>
<td>= 7%</td>
<td>= 5%</td>
</tr>
<tr>
<td>Drawbell shape, well shaped - good flow</td>
<td>= 10%</td>
<td>= 8%</td>
<td>= 6%</td>
</tr>
<tr>
<td>D/P availability, high, good interaction</td>
<td>= 10%</td>
<td>= 7%</td>
<td>= 5%</td>
</tr>
<tr>
<td>Interactive draw area</td>
<td>&gt; 100m = 8%</td>
<td>50m - 100m = 6%</td>
<td>&lt; 50m = 4%</td>
</tr>
<tr>
<td>Ore fragmentation</td>
<td>uniform, fine to medium = 10%</td>
<td>uniform, medium to coarse = 7%</td>
<td>large range = 5%</td>
</tr>
<tr>
<td>Dil. Fragmentation</td>
<td>Coarse frag. = 10%</td>
<td>Medium frag. = 7%</td>
<td>Fine frag. = 5%</td>
</tr>
<tr>
<td>Mineral distribution</td>
<td>In fines = 10%</td>
<td>disseminated = 6%</td>
<td>in coarse = 5%</td>
</tr>
<tr>
<td>Geometry – drawzones</td>
<td>Contiguous = 10%</td>
<td>Stepped = 7%</td>
<td>Scattered = 2%</td>
</tr>
<tr>
<td>Draw strategy</td>
<td>Lines of drawbells per shift = 10%</td>
<td>Lines of drawpoints = 7%</td>
<td>Random. irregular = 5%</td>
</tr>
<tr>
<td>Total</td>
<td>95%</td>
<td>75%</td>
<td>47%</td>
</tr>
</tbody>
</table>

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, but an ore column with a MRMR of 45. The orebody dimensions on the level, are 400m x 600m and 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, the ore column from 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 would be reviewed periodically based on field data.
Chapter 34

MINING COSTS, PRODUCTIVITY

1.0 GENERAL

Mining costs, whether capital or working, are a function of:

- layout
- development - access, haulage’s, production level, undercut level, ore passes, 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 spreadsheet, 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 30m spacing the cost of this drift could be five cents per ton, whereas problems associated with incomplete undercutting could be 20 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, managers of feasibility 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, engineers require evaluating 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 synthesize 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 per 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 be 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 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 databases:

- Mine consumable costs, usually updated 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 tons to be produced from each stope or block.

2.0 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.

Regards LHD performance: the ‘utilization’ percentage, ‘mechanical availability’ percentage and ‘Overall availability’ percentage should be readily available and known by the operating personnel.

Secondary blasting figures 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.

3.0 GENERAL NOTES

* Investigate all the mining cost elements of a project, both capital and working (including abnormal – ongoing replacements), and carry out “what ifs”.
* Constantly review and update this data as the project progresses.
* Decide for a project what currency and what breakdown of cost elements is appropriate.
* Have a breakdown of all the elements considered in the costing, so that changes in one element are easily altered. i.e.: 
Materials
Consumables (including maintenance spares)
Maintenance Labour
Operating Labour
Power

* Carry out a sensitivity analysis on the data and provide a probability factor on the costs and which way they are likely to deviate.
* Consider the productivity of the labour and equipment to be used in the project environment.
* Record all the factors used in the calculations with annotations.

4.0 MINING COSTS

Mining costs, whether capital or working, are a function of:

- layout
- development - access, haulage’s, production level, undercut level, orepasses, ventilation drifts and monitoring drifts
- undercutting - cave initiation
- ore extraction
- secondary breaking hang-ups and oversize
- primary support
- support repair
- LHD costs
- haulage and hoisting costs
- ventilation and pumping
Chapter 35

SCHEDULES AND DOSSIERS

1.0 GENERAL

Gone are the days when block cave mines were located in secondary ore 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. They could 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.

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 planning 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.
2.0 SCHEDULES

Prepare check lists / schedules for all the inputs into:

* The initial investigations, the design stages, the development of the layout, the
  establishing of the cave and the production stages.
* Refer to the manual when in doubt and make the schedules realistic and within the
  capabilities of the organization.
* Prepare full proposals for the logistics throughout the process, from initial
  development to final full production, with particular concern for the advanced
  undercutting and drawpoint/drawbell preparation.
* Make the schedules for development, support, construction, drilling, blasting,
  production, manpower and equipment, etc. meaningful and achievable. Where the
  various elements interplay as on an advanced undercut do allow for suitable diversity
  or interference factors.
* Do not blindly accept the norm if it feels wrong and doesn’t explain the observations
  or engineering judgement.
* It is important to update schedules from observations and to record, interpret, report
  and discuss. Query any anomalies
* Always consider the problem in 3D, think laterally and try to explain /understand
  anomalies in the original design..
* Remember the design is an iterative process and the planning routine must follow a
  sequence until the design is finalized, with all sections relying on the same data.

3.0 STOPE DOSSIERS

Stope dossiers have proved to be extremely useful records of the operations. The object is to
record, with designs and discussions, the progression from start to final layout in a stope
 dossier and keep it updated throughout the life of the project. Subsequent planning and
production will benefit from carefully kept records and interpretations of previous operations.
The rock mass classification system – MRMR – will define the conditions under which the
mining took place and the conditions of the future mining operation. This is essential when
dropping down to lower levels as changes in design might be called for. Dossiers will show
how modelling predictions and actual results compare. Risk predictions are included and
compared against actual. Standards must be clearly defined and must be attainable by the
personnel. Standards are then enforced and if not carried then management must demand an
explanation. It is important to be constantly aware of changes from the predicted and consider
adjustments and modifications in response to new situations.

4.0 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
* Caveability 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.

5.0 POTENTIAL PLANNING PROBLEMS

UPDATING MINING SEQUENCE PLANS - Often mining sequence plans are produced in the early stages of planning to give a broad picture of the proposed operation. It has been noted that on some mines these plans are not changed according to the more accurate interpretations and become the accepted procedure even though there are blatant errors. A classic example is a sequence showing 50m wide block in class 2 ground against a previously mined area and the hope that it will cave instantaneously so that no dilution is drawn. Management were informed of the dangers of an overhang, but have chosen to ignore this advice.

On a mine a diamond shape sequence was designed for a central internal orepass system so that the faces retreated away from the orepasses and production and development did not interfere. However, the orepasses were located outside the orebody, but the sequence was not changed, the result is interference in activities and also the undercut face and abutment stresses were now directed towards the major infrastructure. Management were notified of these problems but did not respond.

In another situation mining blocks were drawn up for a large orebody where mining started in the centre and worked north and south. The mining sequence and block size were drawn up at an early stage in the planning. In spite of warnings of potential caving and dilution problems management has not changed the sequence or block size.
Chapter 36
ENVIRONMENTAL ISSUES

1.0 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. In arid areas the crater could be used for water storage provided acid waters are not generated.

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. However, in the clean up operation these tailings could be relocated in the crater.

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.
Chapter 37
UNDERGROUND RESEARCH PROJECTS

1.0 GENERAL

New mining projects or the extensions with depth of many current operations are taking place in virgin territory. That is, no mining has taken place there and there is limited experience of those conditions. New techniques might have to be developed and new concepts presented, these are often rejected by management, because nobody has tried that method before. 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.

Small scale experiments underground can be misleading as the stress environment might not be the same as for the actual operation.

The interpretation of results is important. Experiments were set up on a mine to test yielding support. There were two objectives namely to simulate a rockburst and to test the selected support. The simulation of the rockburst was successful, but the support did not perform as well as expected. The researchers were happy with the result, but mine personnel could not see the benefit of simulating the rockburst and as the support had not performed as hoped the project was abandoned. What a waste, as some time later the same people designed effective yielding support.

2.0 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?

SECONDARY BREAKING LEVEL - The secondary breaking or mezzanine level in the major apex has been referred to in the section on secondary breaking (Ch14 p 23). The benefits that would be obtained from a level such as this on a mine with coarse fragmentation are significant. After all coarse fragmentation is a result of caving competent rock and therefore the major apex will be in competent ground with large pillars (major Apex). This layout was proposed for a mine with a secondary breaking problem and having all the facilities to try it out. However, they were not prepared to do so, a poor reflection on the mining industry.
3.0 GENERAL NOTES

* Always think in terms of ongoing research both in the ‘laboratory’ and underground in actual operations.
* Monitor and interpret the changing conditions in new areas and at depth.
* Communicate with other cave mining operations; exchange ideas and experiences.
* Be open to suggestions and try them if they have merit.
* Have a list of objectives to improve cave mining and plan for research projects to meet these.
Chapter 38

PHYSICAL, NUMERICAL MODELLING

1.0 GENERAL

3-D solid physical models of orebodies have in the past been of great assistance in understanding orebodies, but are now superseded by 3-D presentations on a computer screen which are of doubtful benefit! 3-D physical model to simulate caving operations by drawing sand and gravel through a drawpoint layout have been of great benefit in understanding draw behaviour. Numerical modelling of induced stresses and the comparison of different layouts has been useful. Also the shortcomings of modelling techniques must be shown. For example, recently numerical modelling was used in an attempt to determine caveability at Northparkes Mine - this was found to be completely inaccurate as it was mesh dependent. The important point is how accurate is numerical modelling compared with empirical systems.

2.0 DRAW CONTROL 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 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 – The 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 provided invaluable data on the behaviour of material drawn. The concept of isolated draw zones and interactive draw were developed from the results of these experiments.

3.0 OREBODY SOLID MODELS

Solid three dimensional models made from 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.

4.0 GLASS / PERSPEX SHEET MODELS

By drawing sections on transparent sheets it is possible to obtain a better overall understanding of the orebody.

5.0 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. Care must be taken as modelling results can be suspect as was shown when Flac was used at Northparkes to determine caveability and shown to be mesh dependent.
Numerical modelling has been used successfully to determine stress concentrations and to assess the differences between different layouts.

PFC is being proposed as a possible tool to determine caveability 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.

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%.
Chapter 39

ROLE OF MANAGEMENT AND RISKS

1.0 MANAGEMENT

It is always a surprise how little time senior management spend underground ensuring that everything is being done as planned. It is evident that many of the problems experienced in mining operations have been the result of adhering to:

- Preconceived ideas.
- Incorrect sequences put on plans in the feasibility stages and not changed in spite of advice recommending changes
- Enthusiasm clouding mature thinking and like many other facets of life ignoring the basics or should we say good mining practice.

The work done to set up a large scale cave mining operation is enormous and forms the basis for future decisions. Management should have been familiar with all facets of the planning and must now ensure that these are implemented and that no short cuts are taken. The presence of senior underground on a regular basis ensures a smooth operation. It must be remembered that personnel can develop ‘tunnel vision’ and therefore outside opinion should be encouraged.

Management must create a project philosophy conducive to quality, timely work, set down policies to ensure that realistic standards are established. These must be endorsed and enforced for each phase of the process, from investigation to full production and in particular for: support, undercutting and draw control. Management of contractors is a very important aspect.

It is essential to ensure that training of all participating personnel is done and that there is complete understanding of the functions. Do not allow departure from planned schedules to be made for short-term expediency. Management must be constantly aware of deviations from the predicted and consider adjustments and modifications in response to new situations. Management must at the outset of the project lay down a set of policies that it will enforce throughout the life of mine.

2.0 RISK ANALYSIS

A risk analysis that is done as a separate exercise at the end of the project and immediately prior to a final decision being made, serves little purpose if the proper checks have not been carried out from the start of the investigation. Planning schedules are designed to ensure that all aspects are investigated and assessed to ensure that difficult matters are not swept under the carpet. It is a check list to ensure that the correct procedure has been followed. Therefore, the risk analysis starts on day one and not at the end of the investigation.

A risk analysis is not a magic formula and is totally dependant on what has been done previously. There is a classic example of a risk analysis being done on a major mine where the mining sequence had been designed for a central layout of orepasses and crushers. This was changed, but the sequence was not changed and this lead to later problems, however the risk analysis did not pick this up.

Adherence to all the recommended procedures will ensure a low risk operation as potential problems would have been identified at an early stage and remedies instituted before there is a risk to safety, production, environment and costs.
Chapter 40

**SUMMARY**

Various parameters must be considered in planning a cave mining operation. When problems are experienced it is often asked why we had not thought of that. When the available data is examined the answer is there. A decision might have been made at an early stage that the production drifts should be east – west. Geological data might favour at north south direction, but the arguments are not strong enough to effect a change. The following table summarizes the parameters to be considered in planning a cave mining operation:-

<table>
<thead>
<tr>
<th>Parameters to be considered before the implementation of cave mining</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>CAVEABILITY</strong></td>
</tr>
<tr>
<td>Rockmass strength (RMR / MRMR)</td>
</tr>
<tr>
<td>Rockmass structure-condition geometry</td>
</tr>
<tr>
<td>In situ stress</td>
</tr>
<tr>
<td>Induced stress</td>
</tr>
<tr>
<td>Hydraulic radius of orebody</td>
</tr>
<tr>
<td>Water</td>
</tr>
<tr>
<td><strong>DRAW HEIGHTS</strong></td>
</tr>
<tr>
<td>Capital</td>
</tr>
<tr>
<td>Orebody geometry</td>
</tr>
<tr>
<td>Excavation stability</td>
</tr>
<tr>
<td>Effect on ore minerals</td>
</tr>
<tr>
<td>Method of draw</td>
</tr>
<tr>
<td><strong>SEQUENCE</strong></td>
</tr>
<tr>
<td>Caveability - poor to good or vice versa</td>
</tr>
<tr>
<td>Orebody geometry</td>
</tr>
<tr>
<td>Induced stresses</td>
</tr>
<tr>
<td>Geological environment</td>
</tr>
<tr>
<td>Rockburst potential</td>
</tr>
<tr>
<td>Production requirements</td>
</tr>
<tr>
<td>Influence on adjacent operations</td>
</tr>
<tr>
<td>Water inflow</td>
</tr>
<tr>
<td><strong>DRILLING AND BLASTING</strong></td>
</tr>
<tr>
<td>Rockmass strength</td>
</tr>
<tr>
<td>Rockmass stability (drill hole closure)</td>
</tr>
<tr>
<td>Required fragmentation</td>
</tr>
<tr>
<td>Hole diameter., lengths, rigs</td>
</tr>
<tr>
<td>Patterns and directions</td>
</tr>
<tr>
<td>Powder factor</td>
</tr>
<tr>
<td>Swell relief</td>
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</table>
# CAVE MINING HANDBOOK

<table>
<thead>
<tr>
<th>SUPPORT</th>
<th>PRACTICAL EXCAVATION SIZE</th>
<th>METHOD OF DRAW</th>
</tr>
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<tbody>
<tr>
<td>Excavation stability</td>
<td>Excavation stability</td>
<td>Fragmentation</td>
</tr>
<tr>
<td>Rockburst potential</td>
<td>Induced stress</td>
<td>Practical drawpoint spacing</td>
</tr>
<tr>
<td>Brow stability</td>
<td>Caving stresses</td>
<td>Practical size of excavation</td>
</tr>
<tr>
<td>Timing of support - initial, secondary and production</td>
<td>Secondary blasting</td>
<td>Gravity or mechanical loading</td>
</tr>
<tr>
<td></td>
<td>Equipment size</td>
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<table>
<thead>
<tr>
<th>RATE OF DRAW</th>
<th>DRAWPOINT INTERACTION</th>
<th>DRAW COLUMN STRESSES</th>
</tr>
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<tbody>
<tr>
<td>Fragmentation</td>
<td>Drawzone spacing</td>
<td>Draw-column height</td>
</tr>
<tr>
<td>Method of draw</td>
<td>Critical distance across major apex</td>
<td>Fragmentation</td>
</tr>
<tr>
<td>Percentage hangups</td>
<td>Fragmentation</td>
<td>Homogeneity of ore fragmentation</td>
</tr>
<tr>
<td>Secondary breaking / blasting</td>
<td>Time frame of working drawpoints</td>
<td>Draw control</td>
</tr>
<tr>
<td>Seismic events</td>
<td></td>
<td>Draw-height interaction</td>
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<tr>
<td>Air blasts - drawpoint cover</td>
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<td>Height-to-short axis base ratio</td>
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<td>Direction of draw</td>
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<table>
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<tr>
<th>SECONDARY FRAGMENTATION</th>
<th>SECONDARY BLASTING/BREAKING</th>
<th>DILUTION</th>
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<td>Secondary fragmentation</td>
<td>Orebody geometry</td>
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<th>SUPPORT REPAIR</th>
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| SUBSIDENCE                                   |                                                               |                                                              |
|----------------------------------------------|                                                               |                                                              |
| RMR / MRMR                                   | Minimum and maximum spans                                     | Depth of mining                                               |
| Height of caved column                       | Major geological structures                                    | Topography                                                    |

**Plans versus computer screen viewing** - There is a tendency nowadays to present data on a computer screen for viewing by interested parties. The control of the time that the section or plan is on display is often in the hands of a disinterested junior engineer. Trying to fathom things out on a computer screen with a scale is very difficult. With hand drawn plans the engineer / geologist / surveyor have a feel for what was going on and are not likely to plot errors, particularly as he knew his chief would examine the finished work. No major planning decisions can be made looking at a computer screen, plans, sections and 3D physical models must be studied.