CERTIFICATE IN ROCK MECHANICS

LEARNING GUIDE FOR:
PART 3.3 : MASSIVE UNDERGROUND MINING (HARD AND SOFT ROCK)

SOUTH AFRICAN NATIONAL INSTITUTE OF ROCK ENGINEERING

CHAMBER OF MINES OF SOUTH AFRICA

MINING QUALIFICATIONS AUTHORITY

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CHAPTER 1

PAPER 3.3: MASSIVE UNDERGROUND MINING (HARD AND SOFT ROCK)

PAPER 3.3 SYLLABUS

TOPICS COVERED

This is a specific mining type paper covering rock mechanics practice applicable in massive mining environments in hard and soft rock at all depths. The rock engineering knowledge required here is thus of a specific nature, relating to the mining of massive ore bodies in hard and soft rock at shallow, moderate and great depth.

CRITICAL OUTCOMES

The examination is aimed at testing the candidate's abilities in the six cognitive levels: knowledge, comprehension, application, analysis, synthesis and evaluation. Thus, when being examined on the topics detailed in this syllabus candidates must demonstrate their capacity for :

- Comprehending and understanding the general rock engineering principles covered in this syllabus and applying these to solve real world mining problems
- Applying fundamental scientific knowledge, comprehension and understanding to predict the behaviour of rock materials in real world mining environments
- Performing creative procedural design and synthesis of mine layouts and support systems to control and influence rock behaviour and rock failure processes
- Using engineering methods and understanding of the uses of computer packages for the computation, modelling, simulation, and evaluation of mining layouts
- Communicating, explaining and discussing the reasoning, methodology, results and ramifications of all the above aspects in a professional manner at all levels.

PRIOR LEARNING

This portion of the syllabus assumes that candidates have prior learning and good understanding of :

- The field of fundamental mechanics appropriate to this part of the syllabus
- The application and manipulation of formulae appropriate to this part of the syllabus as outlined in the relevant sections of this document
- The terms, definitions and conventions appropriate to this part of the syllabus as outlined in the relevant sections of this document.

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STUDY MATERIAL

This portion of the syllabus assumes that candidates have studied widely and have good knowledge and understanding of :

- The reference material appropriate to this part of the syllabus as outlined in the relevant sections of this document
- Other texts that are appropriate to this part of the syllabus but that may not be specifically referenced in this document
- Information appropriate to this part of the syllabus published in journals, proceedings and documents of local mining, technical and research organisations.

NOTE

This syllabus is available in PDF format on the SANIRE webpage

2. GEOTECHNICAL CHARACTERISTICS

2.1. GEOLOGY

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to :

- Identify and describe the rock types associated with massive hard rock and soft rock orebodies
- Describe, explain and discuss how the rock types associated with massive orebodies were formed
- Sketch, describe and discuss geological sequences associated with massive orebodies
- Sketch, describe and discuss major geological structures associated with massive orebodies.

2.2. ROCK STRENGTH

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to :

- Describe, explain and discuss the compressive strengths of rock types associated with massive hard rock and soft rock orebodies
- Describe, explain and discuss the tensile strengths of rock types associated with massive hard rock and soft rock orebodies
- Describe, explain and discuss the relative rockmass strengths of rock types associated with massive hard rock and soft rock orebodies
- Apply the above knowledge to the design of various types of mine workings in massive orebodies.

2.3. ROCKMASS CHARACTERISTICS

2.3.1. GEOTECHNICAL ROCKMASS CLASSIFICATION

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to :

- Describe, explain and discuss the geotechnical characteristics of rock types associated with massive orebodies
- Describe, explain, discuss and apply standard rockmass classification and assessment systems to predict excavation stability
- Describe, explain, discuss and apply Barton's Q system to classify rockmasses
- Describe, explain, discuss and apply Bieniawski's RMR system to classify rockmasses
- Describe, explain, discuss and apply Laubscher's MRMR system

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to classify rockmasses

- Apply rockmass classification results to determine the stability of mining excavations
- Apply rockmass classification results to determine the stability of unsupported spans
- Apply rockmass classification results to determine the support requirements of excavations
- Apply rockmass classification results to determine the cavability of mining excavations
- Apply rockmass classification results to determine the fragmentation of caved material
- Describe, discuss and apply rockmass classification techniques for the selection of massive mining methods
- Determine, discuss and apply rockmass 'm' and 's' parameters for the Hoek and Brown criterion based upon rockmass classification results
- Determine, discuss and apply rockmass deformability from joint stiffness and rockmass classification results
- Describe, discuss and apply the rockwall condition factor (RCF) to predict tunnel stability and support requirements.

3. ROCK AND ROCKMASS BEHAVIOUR

3.1. ROCKMASS BEHAVIOUR AROUND MASSIVE STOPING EXCAVATIONS

3.1.1. GENERAL CONSIDERATIONS

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to :

- Describe, discuss and explain the nature of fracturing around massive mining excavations in low, moderate and high stress
- Sketch, describe, discuss and explain how excavation shape and size affects stress distribution and failure of the surrounding rock
- Sketch, describe, discuss and explain how excavation shapes may be optimised to suit the stress environment
- Describe, discuss and explain the factors that affect the stability of blocks of rock that are defined by geological structures
- Describe, discuss and explain the factors that affect the stability of blocks of rock that are defined by joints
- Sketch, describe, discuss and explain how excavation shapes may be optimised to suit the geological environment
- Determine and characterise the stability of massive stoping excavations using the various rockmass classification systems
- Estimate stable spans for excavations by making use of rockmass classification methods
- Estimate excavation support requirements by making use of rockmass classification methods
- Evaluate excavation stability for given mining layouts, geological conditions and stress environments.

3.2. OPEN STOPING OPERATIONS

3.2.1. FRAGMENTATION OF BLASTED ORE

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to :

- Describe, discuss, explain and apply standard rockmass

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classification techniques to determine the fragmentation of blasted ore

- Describe, discuss, explain and determine appropriate blasting practice to achieve suitable fragmentation of blasted ore.

3.2.2. FLOW BEHAVIOUR OF BROKEN ORE

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to :

- Describe, discuss and explain how broken rock flows in unchoked conditions towards a draw point
- Describe, discuss and explain how broken rock flows in choked conditions towards a draw point
- Describe, discuss and explain the draw of fine and coarse materials in the same column and its effect on dilution
- Describe, discuss and explain the effect of static pillars and temporary hang-ups on the loading of drawpoints
- Evaluate and estimate likely rock flow patterns for given rock property and drawpoint layout conditions
- Evaluate and estimate likely dilution for given rock property and drawpoint layout conditions.

3.2.3. ORE DILUTION

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to :

- Describe, discuss and explain the geotechnical factors that affect dilution in open stoping operations
- Estimate likely dilution by using rockmass classification methods for given rock property and excavation layout conditions
- Determine remedial measures to limit dilution for given rock property and excavation layout conditions
- Describe, discuss and explain methods of reducing dilution.

3.3. CAVING OPERATIONS

3.3.1. CAVING AND CAVABILITY

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to :

- Sketch, describe, discuss and explain the conditions required for successful block caving
- Sketch, describe, discuss and explain the mechanisms associated with the following types of caving :
  - Subsidence caving, Stress caving, Chimney caving
  - Determine undercut requirements and dimensions by using rockmass classification methods for given block cave situations
  - Estimate likely caving behaviour by using rockmass classification methods for given rockmass and undercut conditions.

3.3.2. FRAGMENTATION OF BROKEN ORE

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to :

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- Sketch, describe, discuss and explain the factors that affect fragmentation in cave mining operations
- Estimate likely fragmentation by using rockmass classification methods for given rockmass and undercut conditions.

3.3.3. FLOW BEHAVIOUR OF BROKEN ORE

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to :

- Sketch, describe, discuss and explain the factors that affect the flow behaviour of broken ore in cave mining operations
- Sketch, describe, discuss and explain the flow of rock towards drawpoints in sub-level caving layouts
- Sketch, describe, discuss and explain how drawpoint spacing affects dilution and recovery
- Sketch, describe, discuss and explain how the amount of rock blasted affects dilution and recovery
- Sketch, describe, discuss and explain the principles of draw control to limit dilution in block caving operations
- Describe, discuss, explain, compare and contrast the methods of draw control in panel caving and block caving situations
- List, describe and discuss the advantages and disadvantages of each of the above methods.

3.3.4. ORE DILUTION

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to :

- Sketch, describe, discuss and explain the factors that affect dilution in cave mining operations
- Sketch, describe, discuss and explain the remedial measures to limit dilution in cave mining operations
- Describe, discuss and explain the relationship between dilution and recovery for cave mining operations
- Describe, discuss and explain the causes of the above relationship.

3.4. ROCKMASS BEHAVIOUR AROUND SERVICE EXCAVATIONS

3.4.1. ROCKMASS BEHAVIOUR AROUND ACCESS AND SERVICE TUNNELS

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to :

- Describe, explain and discuss how rocks and rockmasses may be classified · Describe, explain and discuss how rockmass classification systems may be used to predict tunnel stability at shallow, intermediate and great depths
- Describe, explain and discuss the nature of fracturing around tunnels at depth
- Describe, explain and discuss the effects and consequences of fracturing on tunnel size
- Describe, explain and discuss the effects and consequences of fracturing on tunnel shape
- Describe, explain and discuss the effects and consequences of fracturing on tunnel stability

- Describe, explain and discuss the phenomenon of time-dependant fracturing and deterioration
- Describe, explain and discuss the effects and consequences of time-dependant fracturing and deterioration on tunnel stability
- Describe, explain and discuss the effects of stress regime and geological structure on rock behaviour and tunnel stability
- Describe, explain and discuss the effect of tunnel size, tunnel shape and tunnel excavation technique on rock behaviour and tunnel stability
- Describe, explain and discuss the optimisation of tunnel size, shape and orientation to suit geological considerations
- Describe, explain and discuss the optimisation of tunnel size, shape and orientation to suit field stress considerations
- Describe, explain and discuss the behaviour of rock surrounding tunnels at depth during seismic events
- Describe, explain and discuss the siting of tunnels with respect to existing, current and future mining operations
- Describe, explain and discuss the siting of tunnels with respect to geologica stratigraphy and structures
- Describe how rockwall condition factor (RCF) is determined
- Describe and discuss how the rockwall condition factor may be applied to predict tunnel stability and support requirements
- Apply the rockwall condition factor to predict tunnel stability and support requirements at depth
- Evaluate given rock conditions and field stresses to predict excavation stability
- Evaluate given rock conditions and field stresses to recommend remedial measures to improve excavation stability.

3.4.2. ROCKMASS BEHAVIOUR AROUND SERVICE EXCAVATIONS AND SHAFTS

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to :

- Describe, explain and discuss how rocks and rockmasses may be classified
- Describe, explain and discuss how rockmass classification systems may be used to predict service excavation stability at shallow, intermediate and great depths
- Describe, explain and discuss the nature of fracturing around vertical sinking shafts at depth
- Describe, explain and discuss the nature of fracturing around inclined sinking shafts at depth
- Describe, explain and discuss the nature of fracturing around rock passes at depth
- Describe, explain and discuss the effect of stress fracturing and geological structure on the final shape of vertical and inclined shafts
- Describe, explain and discuss the effect of stress fracturing and geological structure on the final shape of rock passes
- Describe, explain and discuss the conditions that may lead to the formation of large unstable rock wedges in vertical shaft sidewalls
- Describe, explain and discuss the conditions that may lead to the formation of instabilities in rock passes

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- Describe, explain and discuss the nature of fracturing around service excavations at depth
- Describe, explain and discuss the effects and consequences of fracturing on service excavation size
- Describe, explain and discuss the effects and consequences of fracturing on service excavation shape
- Describe, explain and discuss the effects and consequences of fracturing on service excavation stability
- Describe, explain and discuss the phenomenon of time-dependant fracturing and deterioration
- Describe, explain and discuss the effects and consequences of time-dependant fracturing and deterioration on service excavation stability
- Describe, explain and discuss the effects of stress regime and geological structure on excavation stability
- Describe, explain and discuss the effects of excavation size, shape and orientation on excavation stability
- Describe, explain and discuss the effects of excavation technique on fracturing and the stability of service excavations
- Describe, explain and discuss the effects of excavation sequence on stress fracturing and the stability of service excavations at depth
- Describe, explain and discuss the effects of support sequence on stress fracturing and the stability of service excavations at depth
- Describe, explain and discuss the optimisation of service excavation size, shape and orientation to suit geological considerations
- Describe, explain and discuss the optimisation of service excavation size, shape and orientation to suit field stress considerations
- Describe, explain and discuss the behaviour of rock surrounding service excavations at depth during seismic events
- Describe, explain and discuss the siting of service excavations with respect to existing, current and future mining operations
- Describe, explain and discuss the siting of service excavations with respect to geological stratigraphy and structures
- Describe, discuss and explain how rockmass characterisation may be applied to predict service excavation stability and support requirements
- Describe, discuss and explain how rockmass characterisation may be applied to predict service excavation stability and support requirements
- Apply rockmass characterisation techniques to predict service excavation stability and support requirements
- Evaluate given rock conditions, field stresses, geological conditions, excavation sizes, shapes and orientations to predict excavation stability
- Evaluate given rock conditions, field stresses, geological conditions, excavation sizes, shapes and orientations to recommend remedial measures to improve excavation stability
- Evaluate given rock conditions, field stresses and geological conditions to predict shaft stability
- Evaluate given rock conditions, field stresses and geological conditions to recommend remedial measures to improve stability.

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3.5. MINE SEISMOLOGY

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to :

- Define, describe and discuss the term seismic event
- List, describe, explain and discuss the factors that determine the nature of seismicity in massive mining
- Describe, explain and discuss the relationship between mine layout and seismicity
- Describe, explain and discuss the relationship between depth and seismicity
- Describe, explain and discuss the relationship between rock types and seismicity
- Describe, explain and discuss the relationship between local geology and seismicity
- Describe, explain and discuss the relationship between regional geology and seismicity
- List, describe, explain and discuss the measures that may be applied to control seismicity
- Describe, explain and discuss measures to reduce the maximum magnitude of seismic events
- Describe, explain and discuss measures to reduce the frequency of seismic events
- Explain the difference between Local magnitude and Richter magnitude
- Describe and explain the following common methods of seismic data analysis :
  - Gutenberg-Richter analysis
  - Magnitude-Frequency relationship
  - Energy-Moment relationship
  - Trend analysis
- Describe, explain and discuss the relationships between :
  - Event magnitude and Damage caused by seismic events
  - Peak particle velocity and Damage caused by seismic events
- Define and discuss the term 'rockburst'
- Describe, explain, discuss and contrast the relationship between :
  - Rockburst damage and Seismic source characteristics
  - Rockburst damage and Excavation support characteristics
- Describe, explain and discuss the different types of seismic emission sources in massive mining
- Demonstrate familiarity with the guideline for the compilation of a code of practice to combat rock related accidents by being able to :
  - Define, describe and explain the term 'seismically active' as defined in the guideline
  - Describe and explain the steps necessary to define emission sources in various ground control districts
  - Describe and explain the term 'reasonably practicable' in the context of seismic risk management
  - Describe and explain the various rockburst damage control

measures

- Describe and explain the use of a seismic system to monitor blasting practice, caving activity and slope stability

4. MINING LAYOUT STRATEGIES

4.1. MASSIVE MINING METHODS

4.1.1. CHOICE OF MASSIVE MINING METHOD

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to :

- Sketch, describe, explain and discuss the fundamental mining and rock engineering principles associated with the following massive mining methods :
  - Open stoping methods
  - Vertical crater retreat
  - Sub-level caving methods
  - Block caving methods
  - Cut and Fill
  - Drift and fill
  - Room and Pillar
- Sketch, describe, explain and discuss the layout of the above mining methods
- Sketch, describe, explain and discuss the location and stability of access tunnels and service excavations
- Sketch, describe, explain and discuss how the following orebody geometries and/or combinations of orebody geometries may be mined :
  - Thick ore horizons, Multiple ore horizons
  - Steeply dipping ore horizons, Flat dipping ore horizons
  - Shallow ore horizon, Deep ore horizon
  - Irregular ore blocks
  - Scattered ore blocks.

4.1.2. CONDITIONS REQUIRED FOR MINING METHODS

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to :

- Describe, explain and discuss the conditions required to successfully apply the following mining methods in terms of :
  - Rockmass characteristics:
  - Stress regime characteristics:
  - Open stoping methods
  - Vertical crater retreat
  - Sub-level caving methods
  - Block caving methods
  - Cut and Fill methods
  - Drift and fill methods
  - Bord and Pillar methods.

4.2. R EGIONAL STABILITY STRATEGIES

4.2.1. REGIONAL PILLARS

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The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to :

- Sketch, describe, explain and discuss the functions of regional stability pillars for the variety of massive mining methods at shallow, intermediate and great depth
- Sketch, describe, explain and discuss the dimensions and layout of regional stability pillars for the variety of massive mining methods at shallow, intermediate and great depth
- Design regional stability pillars for workings at shallow depths
- Design regional stability pillars for workings at intermediate depths
- Design regional stability pillars for workings at great depths.

4.2.2. CROWN PILLARS

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to :

- Sketch, describe, explain and discuss the functions of crown pillars for the variety of massive mining methods at shallow, intermediate and great depth
- Sketch, describe, explain and discuss the dimensions and layout of crown pillars for the variety of massive mining methods at shallow, intermediate and great depth
- Design crown pillars for massive orebody workings
- Describe, explain and discuss the problems associated with designing adequate crown pillars.

4.2.3. BACKFILL AS REGIONAL SUPPORT

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to :

- Describe, explain and discuss the functions of backfill as regional support for the variety of massive mining methods at shallow, intermediate and great depth
- Sketch, describe, explain and discuss typical particle size distribution, for :
- Sketch, describe, explain and discuss typical stress-strain curves for :
- Sketch, describe, explain and discuss typical SG s for :
- Sketch, describe, explain and discuss typical drainage characteristics of :
- Sketch, describe, explain and discuss typical pumping characteristics of :
  - Classified tailings backfill
  - Full plant tailings
  - Paste fill
  - Rock fill
  - Combinations of fill materials
- Sketch, describe, explain and discuss the use and effects of cementitious binders with the above types of fill
- Specify performance criteria, backfill type and placement method to ensure successful backfilling on a mine.

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4.2.4. MINE LAYOUT FOR REGIONAL SUPPORT

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to :

- Sketch, describe, explain and discuss mining layout strategies to ensure regional stability of massive workings at shallow, intermediate and great depth
- Sketch, describe, explain and discuss mining layouts to ensure regional stability of workings at shallow, intermediate and great depth
- Sketch, describe, explain and discuss mining sequences to ensure regional stability of workings at shallow, intermediate and great depth
- Design mining strategies, layouts and sequences to ensure the regional stability of massive workings at shallow, intermediate and great depth.

4.3. L OCAL STABILITY STRATEGIES

4.3.1. MINING LAYOUTS AND SEQUENCES

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to :

- Sketch, describe, explain and discuss mining strategies, layouts and sequences to ensure local stability of massive workings at shallow, intermediate and great depth
- Sketch, describe, explain and discuss layouts and sequences for the following massive mining methods :
  - Open stoping methods
  - Vertical crater retreat
  - Sub-level caving methods
  - Block caving methods
  - Cut and Fill methods
  - Drift and fill methods
  - Bord and Pillar methods
- Evaluate geotechnical data to select appropriate massive mining methods based on stability, dilution and extraction considerations
- Design appropriate massive mining layouts using rockmass classification techniques
- Design the layout of production tunnels, drawpoints and drilling tunnels for open stoping and sub-level caving operations for given geological, geotechnical and stress conditions.
- Design the layout of production tunnels and the undercuts for block caving operations for given geological, geotechnical and stress conditions.
- Design production excavation sequences for the various massive mining methods for given geological, geotechnical and stress conditions.

4.3.2. UNDERCUT LAYOUT AND DESIGN

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to :

- Determine and design appropriate massive mining undercut dimensions and layouts using rockmass classification techniques.

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4.4. SERVICE EXCAVATION LAYOUTS

4.4.1. TUNNEL LAYOUTS

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to :

- Describe, explain and discuss the rock engineering criteria used to determine the optimum size, shape and orientation of service tunnels
- Describe, explain and discuss the rock engineering criteria used to determine the optimum layout, positioning, orientation and spacing of service tunnels relative to :
  - Stopes, Pillars
  - Geological features
  - Other excavations
  - Apply these criteria to design service tunnel layouts.

4.4.2. DRAWPOINT AND LOADING LAYOUTS

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to :

- Describe, explain and discuss the rock engineering criteria used to determine the optimum layout, positioning, orientation and spacing of drawpoints and loading bays relative to :
  - Stopes, Pillars
  - Geological features
  - Other excavations
- Apply these criteria to design drawpoint, loading bay and loading tunnel layouts
- Describe, explain and discuss the practical aspects of drawpoint and loading bay layouts and sizes with respect to aspect such as machine size, haul distance, etc.

4.4.3. LARGE CHAMBER LAYOUTS

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to :

- Describe, explain and discuss the rock engineering criteria used to determine the optimum size, shape and orientation of large service chambers
- Describe, explain and discuss the rock engineering criteria used to determine the optimum excavation methodology and sequence for large service chambers
- Describe, explain and discuss the rock engineering criteria used to determine the optimum support methodology and sequence for large service chambers
- Describe, explain and discuss the rock engineering criteria used to determine the optimum layout, positioning, orientation and spacing of large service chambers relative to :
  - Stopes, Pillars
  - Geological features
  - Other excavations
- Apply these criteria to design large service chambers layouts.

4.4.4. ROCKPASS LAYOUTS

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The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to :

- Describe, explain and discuss the causes of instability in rockpasses
- Describe, explain and discuss the effects of instability in rockpasses
- Describe, explain and discuss the options available for stabilising rockpasses
- Describe, explain and discuss the options available for supporting rockpasses
- Determine appropriate rockpass layout, orientation and support measures using rockmass and stress characterisation techniques
- Assess the suitability of rockpass layout, orientation and support measures for given sets of rockmass and stress conditions
- Describe, explain and discuss the rock engineering criteria used to determine the optimum size, shape and orientation of rockpasses
- Describe, explain and discuss the rock engineering criteria used to determine the optimum layout, positioning and spacing of rockpasses relative to :
    - Stopes, Pillars
    - Geological features
    - Each other
- Apply these criteria to design rockpasses
- Describe, explain and discuss the rock engineering criteria used to rehabilitate damaged rockpasses.

4.5. SHAFT LAYOUTS

4.5.1. SHAFT PROTECTION PILLARS

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to :

- Sketch, describe, explain and discuss the rock engineering criteria used to design incline shaft protection pillars at shallow, intermediate and great depth
- Sketch, describe, explain and discuss the rock engineering criteria used to design vertical shaft protection pillars at shallow, intermediate and great depth
- Calculate stress, strain and tilt in vertical shafts using equations for circular shaft pillars in an elastic medium
- Apply the results from the above calculations to design appropriate shaft pillar dimensions
- Design shaft layouts for given depth, geology and rockmass conditions
- Explain the limitations that concrete linings and steelwork place on allowable movements in shafts.

4.5.2. LAYOUT DESIGN CRITERIA

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to :

- Describe, explain and discuss the derivation and application of layout and support design criteria for massive mining operations at shallow, intermediate and great depth
- Describe, explain, discuss and apply layout design criteria applicable to all of the massive mining situations set out above.

5. MINING SUPPORT STRATEGIES
5.1. STOPE SUPPORT STRATEGIES
5.1.1. ROCKWALL REINFORCEMENT

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to :

- Describe, explain and discuss the variation in the behaviour of rock surrounding stoping excavations at shallow, intermediate and great depth
- Describe, explain and discuss how this affects the support requirements at shallow, intermediate and great depth
- Evaluate and determine appropriate support requirements in given circumstances in terms of the following factors :
  - Rock conditions, Field stresses, Geological features
- Describe, explain and discuss appropriate support strategies to control stope hangingwall under static conditions at shallow, intermediate and great depth
- Determine the required support reaction for static conditions given typical fall of ground dimensions and loading conditions
- Describe, explain and discuss appropriate support strategies to control stope hangingwall under dynamic conditions at shallow, intermediate and great depth
- Determine the required support reaction for dynamic (rockburst) conditions given typical fall of ground dimensions and loading conditions
- Sketch describe, explain and discuss the function, use and installation of cable bolting to stabilise open stopes
- Sketch, describe, explain and discuss the function, use and installation of cable bolting to stabilise cut and fill stopes
- Sketch, describe, explain and discuss the function, use and installation of cable bolting to stabilise pillars
- Sketch, describe, explain and discuss methods of installation of cable bolts in the above situations
- Determine and design cable bolt layouts, strengths and performance characteristics for a given open stoping situations
- Determine and design cable bolt layouts, strengths and performance characteristics for a given cut and fill stoping situations
- Determine and design cable bolt layouts, strengths and performance characteristics for a given pillar extraction situations
- Evaluate rock conditions, field stresses and mining layout circumstances to determine bolting requirements for the above stoping situations
- Describe, explain and discuss the special problems of supporting wide orebodies at shallow, intermediate and great depths
- Describe, explain and discuss appropriate strategies to ensure stability under the above conditions
- Describe, explain and discuss the special problems of supporting multiple, closely spaced orebodies at shallow, intermediate and great depths
- Describe, explain and discuss appropriate strategies to ensure stability under the above conditions.

5.1.2. PILLARS AS STOPE SUPPORT

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to :

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- Describe, explain and discuss the behavioural characteristics and applications of the following types of pillar :
  - Crush pillars, Yield pillars, Rigid pillars
  - Crown pillars, Rib pillars
- Estimate the strength of hard rock pillars by using rock classification techniques and pillar strength equations
- Apply Ryder and Ozbay's procedure to derive the strength of hard rock pillars
- Describe, explain and discuss the stress-strain behaviour of square pillars at various width to height ratios
- Describe, explain and discuss the post peak behaviour in the above situations
- Describe, explain and discuss tributary area theory in respect of pillar design
- Describe, explain and discuss the limitations of tributary area theory in respect of pillar design
- Estimate pillar stresses using tributary theory
- Describe, explain and discuss the effect of joints and other weaknesses on pillar strength
- Describe, explain and discuss the role of loading system stiffness in the stability of pillar systems once pillars have exceeded their peak strength
- Describe, explain and discuss the effect of loading system stiffness on the mode of failure of pillars
- Describe, explain and discuss the mechanism of foundation failure of pillars
- Describe, explain and discuss the factors that affect foundation failure of pillars
- Describe, explain and discuss the factors that are taken into consideration when designing pillar support systems
- Design in-stope pillar support layouts for the variety of massive mining methods
- Describe, explain and discuss the functions and use of pillars as a means of ground control in steeply-dipping massive open stope
- Describe, explain and discuss the functions and use of pillars as a means of ground control in shallow-dipping massive orebodies
- Describe, explain and discuss the functions and use of post pillars as a means of ground control in cut and fill operations
- Describe, explain and discuss how the stability of sill or rib pillars may be assessed
- Design appropriate pillar systems for given rockmass characteristics, stress environments and mining situations.

5.1.3. BACKFILL AS STOPE SUPPORT

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to :

- Describe, explain and discuss the backfill characteristics and types of backfill required for filling in the following mining situations :
  - Cut and Fill operations
  - Drift and Fill operations
  - Vertical Crater Retreat operations

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    - Shallow Wide-reef operations
- Describe, explain and discuss how backfill fits into the mining cycle in the above situations
- Describe, explain and discuss how backfill assists in improving the stability of rock masses
- Determine the support resistance offered by backfill given the backfill performance characteristics
- Evaluate given rockmass and mining situations and note the potential benefits of backfilling in these situations
- Evaluate given rockmass and mining situations and recommend appropriate backfill type and placement procedures

5.2. SERVICE EXCAVATION SUPPORT STRATEGIES

5.2.1. TUNNEL SUPPORT STRATEGIES

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to :

- Sketch, describe, explain and discuss the various behaviour of rock around excavations at shallow, intermediate and great depth
- Describe, explain and discuss how this affects tunnel support requirements at shallow, intermediate and great depth
- Describe, explain and discuss the objectives primary support in tunnels
- Describe, explain and discuss the objectives secondary support in tunnels
- Describe, explain and discuss the concept of integrated support in tunnels
- Describe, explain and discuss various support strategies for :
    - Return Airways
    - Roadways, Haulages, Crosscuts
    - Travellingways
    - Winzes, Raises
    - Tunnel intersections
    - Tunnels traversing faults, Tunnels traversing dykes
    - Undercuts, Drawpoints
    - Gathering drives, Drilling drives
- Describe, explain and discuss the strategies to support tunnels under the following conditions :
    - Low stress with joint controlled behaviour
    - High stress where stress fractures dominate stability
    - Seismic and rockburst conditions
- Determine appropriate support strategies for tunnels for given layouts, rockmass conditions, stress regimes and geological circumstances.

5.2.2. SUPPORT OF DRAWPOINTS

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to :

- Sketch, describe, explain and discuss support measures for drawpoints, loading bays and similar excavations at shallow, intermediate and great depth

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- Sketch, describe, explain and discuss support measures for draw-points, loading bays and similar excavations where the stress to strength ration of the rockmass approximates unity
- Sketch, describe, explain and discuss special support measures for high wear areas of drawpoints, loading bays and similar excavations.

5.2.3. SUPPORT OF ROCKPASSES

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to :

- Sketch, describe, explain and discuss support measures for rock-passes and similar excavations at shallow, intermediate and great depth.

5.2.4. SUPPORT OF LARGE CHAMBERS

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to :

- Sketch, describe, explain and discuss the various behaviour of rock around large excavations at shallow, intermediate and great depth
- Describe, explain and discuss how this affects large excavation support requirements at shallow, intermediate and great depth
- Describe, explain and discuss rockmass classification schemes for the determination of support strategies for large chambers
- Describe, explain and discuss the application of the rockwall condition factor methodology to design support for large excavations at depth
- Apply the rockwall condition factor (RCF) methodology to design support for large excavations at depth
- Describe, explain and discuss the effects of excavation sequence on support installation
- Determine appropriate excavation sequences to facilitate support installation
- Describe, explain and discuss the effects of excavation sequence on support effectiveness
- Determine appropriate excavation sequences to facilitate support effectiveness
- Describe, explain and discuss the rules of thumb for determining the length and spacing of tendons in large excavations
- Apply these rules of thumb to the design of support in large excavations
- Describe, explain and discuss the effects of overstoping and understoping on the stability and support requirements of off-reef excavations
- Determine appropriate support strategies for large chambers for given mining and shaft layouts, rockmass conditions, stress regimes and geological circumstances.

5.2.5. SUPPORT OF SHAFTS

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to :

- Describe, explain and discuss rockmass classification schemes for the determination of support strategies for shafts

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- Determine appropriate support strategies for vertical shafts for given mining and shaft layouts, rockmass conditions, stress regimes and geological circumstances
- Determine appropriate support strategies for inclined shafts for given mining and shaft layouts, rockmass conditions, stress regimes and geological circumstances
- Determine appropriate support strategies for roadways for given mining and shaft layouts, rockmass conditions, stress regimes and geological circumstances.

5.3. SUPPORT DESIGN CRITERIA

5.3.1. STOPE SUPPORT DESIGN CRITERIA

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to :

- Describe, explain and discuss stope support requirements in terms of :
  - Initial stiffness, Yieldability
  - Areal coverage
  - Support resistance, Energy absorbtion
- Describe, explain and discuss how support resistance varies for static and dynamic conditions.

5.3.2. TUNNEL, CHAMBER AND SHAFT SUPPORT DESIGN CRITERIA

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to :

- Describe, explain and discuss excavation support and rockwall reinforcement requirements in terms of :
  - Initial stiffness, Yieldability
  - Unit length, Areal coverage
  - Support resistance, Energy absorbtion
- Describe, explain and discuss how support resistance varies for static and dynamic conditions
- Calculate required support resistances for static conditions
- Calculate required support resistances for dynamic (rockburst) conditions
- Calculate support loads for support systems made up of various units
- Calculate energy absorption for support systems made up of various units
- Describe, explain and discuss support requirements for time-dependent (creep) conditions.

5.3.3. SUPPORT AND SUPPORT SYSTEM TYPES AND CHARACTERISTICS

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to :

- Apply rockmass classification systems to select appropriate support for stopes, tunnels and chambers in massive mines
- Apply the rock condition factor (RCF) to select appropriate support for tunnels in high stresses

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- Describe and discuss the following rockwall support types :
  - Mechanically anchored bolts, Cable bolts, Friction bolts (split sets,
  - Cement grouted bolts, Resin bonded bolts
  - Full-column grouted/bonded bolts
  - Yielding bolts (cone bolts)
  - Multiple rod bolts, Multiple strand cable anchors
  - Prestressed tendons
  - Steel arches, Massive concrete linings
  - Shotcrete, Gunite, Thin sprayed linings
  - Wire mesh, Rope lacing, Tendon straps
- Characterise the following aspects of the above units :
  - Their principles of operation
  - Their technical specifications
  - Their load-deformation characteristics
  - The methods of ensuring support unit quality
  - Their installation/application procedures
  - The methods of ensuring their installed quality
- Design and evaluate the use of appropriate support units, support systems, support patterns and installation procedures for given rockmass conditions, stress regimes and mining layouts.

6. INVESTIGATION TECHNIQUES
6.1. ROCK TESTING

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to :

- Describe, explain and discuss various rock testing procedures
- Interpret and incorporate test results in analysis and design.

6.2. MONITORING
6.2.1. SUBSIDENCE MONITORING

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to :

- Sketch, describe, explain and discuss the techniques used to measure surface subsidence
- Describe, explain and discuss the equipment used to measure surface subsidence
- Describe, explain and discuss how vertical and horizontal displacements are determined
- Describe, explain and discuss how strains and tilts may be derived from these determinations
- Calculate strain and tilt from given sets of measurements
- For given sets of underground mining and surface infrastructure conditions :
  - State, describe, explain and discuss what types of measurements need to be made and monitored
  - Describe, explain and discuss required monitoring station layouts

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  - Describe, explain and discuss appropriate monitoring programs
  - Describe, explain and discuss the frequency of measurements
  - Interpret, explain and discuss given subsidence data in terms of likely surface behaviour.

6.2.2. EXCAVATION DEFORMATION MONITORING

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to :

- Sketch, describe, explain and discuss the techniques used to measure deformations in underground excavations
- Describe, explain and discuss the equipment used to measure deformations in underground excavations
- Describe, explain and discuss how displacements in the rockmass are determined
- Describe, explain and discuss how strains and tilts may be derived from these determinations
- Calculate strain and tilt from given sets of measurements
- For given sets of underground mining conditions :
  - State, describe, explain and discuss what types of measurements need to be made and monitored
  - Describe, explain and discuss required monitoring station layouts
  - Describe, explain and discuss appropriate monitoring programs
  - Describe, explain and discuss the frequency of measurements
  - Interpret, explain and discuss given monitoring data in terms of likely excavation behaviour.

6.2.3. IN-SITU STRESS MEASUREMENT AND MONITORING

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to :

- Sketch, describe, explain and discuss the techniques used to measure in-situ stress in the underground rockmass
- Describe, explain and discuss the equipment used to measure in-situ stress in the rockmass
- Interpret, explain and discuss given stress measurement data in terms of likely rockmass, pillar or excavation behaviour.

6.2.4. SEISMIC MONITORING SYSTEMS

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to :

- Describe, discuss and explain how to determine objectives for a seismic monitoring system in massive mining operations
- Explain and discuss key performance criteria for seismic networks
- Explain and discuss the evaluation of historic seismicity for network design
- Explain and discuss the requirements of spatial coverage
- Explain and discuss the needs of various stakeholders for seismic information with respect to monitoring objectives
- Describe, discuss and explain how to derive a compromise between ideal network layout and the following constraints :

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- Accessibility constraints
- Budgetary constraints

- Describe, discuss and explain the factors that need to be considered in placing seismometers
- Explain and discuss how seismometer placement could affect the event location accuracy of a seismic system
- Explain and discuss how seismometer placement could affect the sensitivity of a seismic system
- Describe, discuss and explain relevant procedures to ensure that seismic monitoring information is distributed effectively to stakeholders
- Describe, discuss and explain methods of quality assurance in seismic system management
- Describe and discuss relevant aspects of seismic system audits
- Describe and discuss methods to assess stakeholder needs
- Describe and discuss methods to identify possible shortfalls between seismic monitoring objectives and seismic system performance.

6.3. MODELLING

6.3.1. NUMERICAL MODELLING

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to :

- Describe, explain and discuss the selection of appropriate codes to tackle various problems
- Describe, explain and discuss the input of appropriate parameters to investigate various problems
- Describe, explain and discuss the interpretation of output in the investigation of various problems.

6.4. AUDITING

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to :

- Describe, explain and discuss the concept of monitoring for understanding, prediction and design.

7. ROCKBREAKING IN MASSIVE MINES

7.1. CUTTING TECHNIQUES

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to :

- Describe, explain and discuss geotechnical aspects associated with various non-explosive rock breaking procedures that include :
    - Tunnel boring, Raise boring
- Describe, explain and discuss the methodologies and applications of these techniques
- List and discuss the advantages and disadvantages of these techniques.

7.2. DRILLING TECHNIQUES

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The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to :

- Sketch, describe, explain and discuss the different rounds used in shaft sinking
- Describe, explain and discuss the different cuts used in shaft sinking
- Describe, explain and discuss the types of initiation used in the above rounds
- Describe, explain and discuss the sequence of initiation of blast holes used in the above rounds
- Sketch, describe, explain and discuss the different rounds used in tunnel development
- Describe, explain and discuss the different cuts used in tunnel development
- Describe, explain and discuss the types of initiation used in the above rounds
- Describe, explain and discuss the sequence of initiation of blast holes used in the above rounds
- Sketch, describe, explain and discuss blast hole layouts in massive stopes
- Describe, explain and discuss the drilling of blast holes in from drilling drives in massive sub-level open stopes
- Describe, explain and discuss the drilling of blast holes in from drilling drives in massive vertical crater retreat stopes
- Describe, explain and discuss the direction of drilling of blast holes in massive stopes
- Describe, explain and discuss the sequence of initiation of blast holes in massive stopes
- Describe, explain and discuss the importance of blast-hole drilling accuracy in the following applications :
    - Shaft sinking, Chamber excavation, Tunnel development
    - Massive stoping operations
    - Cushion blasting, Smooth blasting.

7.3. BLASTING PRACTICES

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to :

- List and discuss the advantages and disadvantages of these techniques
- Evaluate and determine blasting requirements for tunnels making
- Describe, explain and discuss the effect of the following parameters on blast damage :
    - Explosive type, Initiation method,
    - Initiation sequence, Hole orientation
- Describe, explain and discuss the objectives and effects of de-coupling explosives
- Describe, explain and discuss the methods by which de-coupling of explosives is achieved
- Describe, explain and discuss the following excavation cushion blasting and smooth blasting techniques :

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  - Pre-splitting
  - Concurrent smooth blasting
  - Post-splitting
- Describe, explain and discuss the methodologies and typical applications of each technique
- Evaluate and determine appropriate blasting rounds to suit given conditions in tunnels
- Evaluate and determine appropriate explosive types to suit given conditions in tunnels
- Evaluate and determine blasting requirements for stopes making use of knowledge of explosives
- Evaluate and determine appropriate blasting rounds to suit given conditions in stopes
- Evaluate and determine appropriate explosive types to suit given conditions in stopes.

8. SURFACE AND ENVIRONMENTAL EFFECTS
8.1. SURFACE EFFECTS
8.1.1. SUBSIDENCE ENGINEERING

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to :

- Describe, explain and discuss the following terms in the context of surface subsidence :
  - Angle of draw, Curvature, Tilt, Critical span
  - Horizontal strain, Vertical subsidence, Differential subsidence
- Describe, explain and discuss the following surface expressions of subsidence:
- Tension cracks, Compression humps, Ridges, Thrusts
- Describe, explain and discuss how mining height to depth ratio affects the type and severity of surface subsidence
- Describe, explain and discuss the effects of geological structures on surface subsidence.
-

8.1.2. SURFACE PROTECTION

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to :

- Describe, explain and discuss how the following surface features are affected by subsidence:
  - Roads, Buildings, Pylons, Lands, Streams, Pans
- Describe, explain and discuss possible remedial measures that may be applied to surface structures to limit subsidence damage
- Describe, explain and discuss possible changes that may be made to underground mining layouts to reduce subsidence damage
- Determine potential subsidence damage to the following types of structure for given mining depths and mining heights using published damage tables:
  - Roads, Buildings, Pylons

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8.2. ENVIRONMENTAL EFFECTS

8.2.1. LONG-TERM STABILITY AND THE ENVIRONMENT

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to :

- Describe, explain and discuss the possible effects and consequences of given mining methods on the following issues :
  - Long-term stability of the ground surface
  - Groundwater
  - Ultimate closure of the mine
- Describe, explain and discuss the possible effects and consequences of given factors of safety on the following issues :
  - Long-term stability of the ground surface
  - Groundwater
  - Ultimate closure of the mine.

9. MINING STRATEGIES IN DIFFICULT CIRCUMSTANCES

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to :

- Describe, explain and discuss the geotechnical aspects of dealing with the following difficult circumstances :
  - Mining through geological structures or disturbances
  - Mining in localised disturbed, weak or poor ground conditions
  - Mining in localised high or anomalous stress situations
  - Mining in areas of high water pressures and/or large water inflows
  - Remnant mining
  - Pillar mining

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- Rehabilitation of previously mined areas.
- Dealing with excessive overbreak situations
- Dealing with runaway caving situations

2. GEOTECHNICAL CHARACTERISTICS

2.1. GEOLOGY

- Lurie J 1987 South African Geology for Mining, Metallurgical, Hydrological and Civil Engineering Lexicon Publishers Jhb Chapter 9

2.2. ROCK STRENGTH

- Brady BHG & Brown ET 1993 Rock Mechanics for Underground MiningChapman & Hall New York Chapter 4
- Obert L & Duvall WI 1967 Rock Mechanics and the Design of Structures in Rock John Wiley & Sons New York Chapters 10, 11
- Jaeger JC & Cook NGW 1969 Fundamentals of Rock Mechanics Chapman & Hall London Chapter 4, 6
- Budavari S (ed) 1986 Rock Mechanics in Mining Practice SAIMM Jhb Chapter 2
- Ryder JA & Jager AJ 2002 Rock Mechanics for Tabular Hard Rock Mines SIMRAC Jhb Chapter 2

2.3. ROCKMASS CHARACTERISTICS

2.3.1. GEOTECHNICAL ROCKMASS CLASSIFICATION

- Hoek E & Brown ET 1980 Underground Excavations in Rock IMM London Chapter 2
- Brady BHG & Brown ET 1993 Rock Mechanics for Underground Mining Chapman & Hall New York Chapter 3
- Stacey TR 2001 Best Practice Rock Engineering Handbook for 'Other' Mines SIMRAC Jhb Chapter 2

3. ROCK AND ROCKMASS BEHAVIOUR

3.1. ROCKMASS BEHAVIOUR AROUND MASSIVE STOPING EXCAVATIONS

3.1.1. GENERAL CONSIDERATIONS

- Brady BHG & Brown ET 1993 Rock Mechanics for Underground Mining Chapman & Hall New York Chapters 7, 8, 9

3.2. OPEN STOPING OPERATIONS

3.2.1. FRAGMENTATION OF BLASTED ORE

- Brady BHG & Brown ET 1993 Rock Mechanics for Underground Mining Chapman & Hall New York Chapters 12, 13, 14, 15

3.2.2. FLOW BEHAVIOUR OF BROKEN ORE

- Brady BHG & Brown ET 1993 Rock Mechanics for Underground Mining Chapman & Hall New York Chapters 12, 13, 14, 15

3.2.3. ORE DILUTION

- Brady BHG & Brown ET 1993 Rock Mechanics for Underground Mining Chapman & Hall New York Chapters 12, 13, 14, 15

3.3. CAVING OPERATIONS

3.3.1. CAVING AND CAVABILITY

- Brady BHG & Brown ET 1993 Rock Mechanics for Underground Mining Chapman & Hall New York Chapters 12, 13, 14, 15

3.3.2. FRAGMENTATION OF BROKEN ORE

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- Brady BHG & Brown ET 1993 Rock Mechanics for Underground Mining Chapman & Hall New York Chapters 12, 13, 14, 15

3.3.3. FLOW BEHAVIOUR OF BROKEN ORE

- Brady BHG & Brown ET 1993 Rock Mechanics for Underground Mining Chapman & Hall New York Chapters 12, 13, 14, 15

3.3.4. ORE DILUTION

- Brady BHG & Brown ET 1993 Rock Mechanics for Underground Mining Chapman & Hall New York Chapters 12, 13, 14, 15

3.4. ROCKMASS BEHAVIOUR AROUND SERVICE EXCAVATIONS

3.4.1. ROCKMASS BEHAVIOUR AROUND ACCESS AND SERVICE TUNNELS

- Brady BHG & Brown ET 1993 Rock Mechanics for Underground Mining Chapman & Hall New York Chapters 7, 8, 9

3.4.2. ROCKMASS BEHAVIOUR AROUND SERVICE EXCAVATIONS AND SHAFTS

- Brady BHG & Brown ET 1993 Rock Mechanics for Underground Mining Chapman & Hall New York Chapters 7, 8, 9

3.4.3. MINE SEISMOLOGY

- Chief Inspector of Mines Latest Guideline for the Compilation of a Mandatory Code of Practice to Combat Rock Fall and Rock Burst Accidents in Massive Mining Operations - Ref. No. DME 16/3/2/1-A5 DME

- Mendecki AJ (ed) 1997 Seismic Monitoring in Mines Chapman & Hall London Chapters 1, 2, 5, 7, 8

4. MINING LAYOUT STRATEGIES

4.1. MASSIVE MINING METHODS

4.1.1. CHOICE OF MASSIVE MINING METHOD

- Brady BHG & Brown ET 1993 Rock Mechanics for Underground Mining Chapman & Hall New York Chapter 12

4.1.2. CONDITIONS REQUIRED FOR MINING METHODS

- Brady BHG & Brown ET 1993 Rock Mechanics for Underground Mining Chapman & Hall New York Chapters 12, 13, 14, 15

4.2. REGIONAL STABILITY STRATEGIES

4.2.1. REGIONAL PILLARS

- Brady BHG & Brown ET 1993 Rock Mechanics for Underground Mining Chapman & Hall New York Chapters 12, 13, 14, 15

- Stacey TR 2001 Best Practice Rock Engineering Handbook for 'Other' Mines SIMRAC Jhb Chapter 4

4.2.2. CROWN PILLARS

- Brady BHG & Brown ET 1993 Rock Mechanics for Underground Mining Chapman & Hall New York Chapters 12, 13, 14, 15

4.2.3. BACKFILL AS REGIONAL SUPPORT

- Brady BHG & Brown ET 1993 Rock Mechanics for Underground Mining Chapman & Hall New York Chapters 12, 13, 14, 15

- Stacey TR 2001 Best Practice Rock Engineering Handbook for 'Other' Mines SIMRAC Jhb Chapter 4

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4.2.4. MINE LAYOUT FOR REGIONAL SUPPORT
- Brady BHG & Brown ET 1993 Rock Mechanics for Underground Mining Chapman & Hall New York Chapters 12, 13, 14, 15

4.3. LOCAL STABILITY STRATEGIES
4.3.1. MINING LAYOUTS AND SEQUENCES
- Brady BHG & Brown ET 1993 Rock Mechanics for Underground Mining Chapman & Hall New York Chapters 12, 13, 14, 15

4.3.2. UNDERCUT LAYOUT AND DESIGN
- Brady BHG & Brown ET 1993 Rock Mechanics for Underground Mining Chapman & Hall New York Chapters 12, 13, 14, 15

4.4. SERVICE EXCAVATION LAYOUTS
4.4.1. TUNNEL LAYOUTS
- Jager AJ & Ryder JA 1999 Rock Engineering Practice for Tabular Hard Rock Mines SIMRAC Jhb Chapter 5
- Brady BHG & Brown ET 1993 Rock Mechanics for Underground Mining Chapman & Hall New York Chapters 7, 8, 9, 12, 13, 14, 15
- Budavari S (ed) 1986 Rock Mechanics in Mining Practice SAIMM Jhb Chapters 4, 9

4.4.2. DRAWPOINT AND LOADING LAYOUTS
- Brady BHG & Brown ET 1993 Rock Mechanics for Underground Mining Chapman & Hall New York Chapters 12, 13, 14, 15

4.4.3. LARGE CHAMBER LAYOUTS
- Jager AJ & Ryder JA 1999 Rock Engineering Practice for Tabular Hard Rock Mines SIMRAC Jhb Chapter 7
- Brady BHG & Brown ET 1993 Rock Mechanics for Underground Mining Chapman & Hall New York Chapters 7, 8, 9, 12, 13, 14, 15
- Budavari S (ed) 1986 Rock Mechanics in Mining Practice SAIMM Jhb Chapter 9

4.4.4. ROCKPASS LAYOUTS
- Jager AJ & Ryder JA 1999 Rock Engineering Practice for Tabular Hard Rock Mines SIMRAC Jhb Chapter 7
- Brady BHG & Brown ET 1993 Rock Mechanics for Underground Mining Chapman & Hall New York Chapters 7, 8, 9, 12, 13, 14, 15

4.5. SHAFT LAYOUTS
4.5.1. SHAFT PROTECTION PILLARS
- Jager AJ & Ryder JA 1999 Rock Engineering Practice for Tabular Hard Rock Mines SIMRAC Jhb Chapter 7
- Budavari S (ed) 1986 Rock Mechanics in Mining Practice SAIMM Jhb Chapter 10

4.5.2. LAYOUT DESIGN CRITERIA
- Brady BHG & Brown ET 1993 Rock Mechanics for Underground Mining Chapman & Hall New York Chapters 7, 8, 9
- Ryder JA & Jager AJ 2002 Rock Mechanics for Tabular Hard Rock Mines SIMRAC Jhb Chapter 6
- Budavari S (ed) 1986 Rock Mechanics in Mining Practice SAIMM Jhb Chapters 9, 12

5. MINING SUPPORT STRATEGIES
5.1. STOPE SUPPORT STRATEGIES

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5.1.1. ROCKWALL REINFORCEMENT

- Brady BHG & Brown ET 1993 Rock Mechanics for Underground Mining Chapman & Hall New York Chapter 11
- Stacey TR 2001 Best Practice Rock Engineering Handbook for 'Other' Mines SIMRAC Jhb Chapter 4

5.1.2. PILLARS AS STOPE SUPPORT

- Brady BHG & Brown ET 1993 Rock Mechanics for Underground Mining Chapman & Hall New York Chapter 13
- Stacey TR 2001 Best Practice Rock Engineering Handbook for 'Other' Mines SIMRAC Jhb Chapter 4

5.1.3. BACKFILL AS STOPE SUPPORT

- Brady BHG & Brown ET 1993 Rock Mechanics for Underground Mining Chapman & Hall New York Chapter 14
- Stacey TR 2001 Best Practice Rock Engineering Handbook for 'Other' Mines SIMRAC Jhb Chapter 4

5.2. SERVICE EXCAVATION SUPPORT STRATEGIES

5.2.1. TUNNEL SUPPORT STRATEGIES

- Jager AJ & Ryder JA 1999 Rock Engineering Practice for Tabular Hard Rock Mines SIMRAC Jhb Chapter 6
- Ryder JA & Jager AJ 2002 Rock Mechanics for Tabular Hard Rock Mines SIMRAC Jhb Chapter 7
- Budavari S (ed) 1986 Rock Mechanics in Mining Practice SAIMM Jhb Chapters 4, 9
- Brady BHG & Brown ET 1993 Rock Mechanics for Underground Mining Chapman & Hall New York Chapters 7, 8, 9

5.2.2. SUPPORT OF DRAWPOINTS

- Stacey TR 2001 Best Practice Rock Engineering Handbook for 'Other' Mines SIMRAC Jhb Chapter 4

5.2.3. SUPPORT OF ROCKPASSES

- Stacey TR 2001 Best Practice Rock Engineering Handbook for 'Other' Mines SIMRAC Jhb Chapter 4

5.2.4. SUPPORT OF LARGE CHAMBERS

- Stacey TR 2001 Best Practice Rock Engineering Handbook for 'Other' Mines SIMRAC Jhb Chapter 4

5.2.5. SUPPORT OF SHAFTS

- Jager AJ & Ryder JA 1999 Rock Engineering Practice for Tabular Hard Rock Mines SIMRAC Jhb Chapter 7
- Stacey TR 2001 Best Practice Rock Engineering Handbook for 'Other' Mines SIMRAC Jhb Chapter 4

5.3. SUPPORT DESIGN CRITERIA

5.3.1. STOPE SUPPORT DESIGN CRITERIA

- Stacey TR 2001 Best Practice Rock Engineering Handbook for 'Other' Mines SIMRAC Jhb Chapter 4

5.3.2. TUNNEL, CHAMBER AND SHAFT SUPPORT DESIGN CRITERIA

- Jager AJ & Ryder JA 1999 Rock Engineering Practice for Tabular Hard Rock Mines SIMRAC Jhb Chapter 7

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- Ryder JA & Jager AJ 2002 Rock Mechanics for Tabular Hard Rock Mines SIMRAC Jhb Chapter 7

5.4. SUPPORT AND SUPPORT SYSTEM TYPES AND CHARACTERISTICS

- Brady BHG & Brown ET 1993 Rock Mechanics for Underground Mining Chapman & Hall New York Chapter 11

6. INVESTIGATION TECHNIQUES

6.1. ROCK TESTING

- Ryder JA & Jager AJ 2002 Rock Mechanics for Tabular Hard Rock Mines SIMRAC Jhb Chapter 2

6.2. MONITORING

6.2.1. SUBSIDENCE MONITORING

- van der Merwe JN 1995 Practical Coal Mining Strata Control Sasol Coal Division Jhb Chapter 5

6.2.2. EXCAVATION DEFORMATION MONITORING

- Brady BHG & Brown ET 1993 Rock Mechanics for Underground Mining Chapman & Hall New York Chapter 18

6.3. IN-SITU STRESS MEASUREMENT AND MONITORING

6.3.1. SEISMIC MONITORING SYSTEMS

- Ryder JA & Jager AJ 2002 Rock Mechanics for Tabular Hard Rock Mines SIMRAC Jhb Chapter 5
- Jager AJ & Ryder JA 1999 Rock Engineering Practice for Tabular Hard Rock Mines SIMRAC Jhb Chapter 9

6.4. MODELLING

6.4.1. NUMERICAL MODELLING

- Ryder JA & Jager AJ 2002 Rock Mechanics for Tabular Hard Rock Mines SIMRAC Jhb Chapter 8
- Jager AJ & Ryder JA 1999 Rock Engineering Practice for Tabular Hard Rock Mines SIMRAC Jhb Chapter 11
- Lightfoot N & Maccelari MJ 1998 Numerical Modelling of Mine Workings SIMRAC Jhb Chapters 1-11

6.5. AUDITING

- Jager AJ & Ryder JA 1999 Rock Engineering Practice for Tabular Hard Rock Mines SIMRAC Jhb Chapter 10

7. ROCKBREAKING IN MASSIVE MINES

7.1. CUTTING TECHNIQUES

7.2. DRILLING TECHNIQUES

7.3. BLASTING PRACTICES

- Brady BHG & Brown ET 1993 Rock Mechanics for Underground Mining Chapman & Hall New York Chapter 17

8. SURFACE AND ENVIRONMENTAL EFFECTS

8.1. SURFACE EFFECTS

8.1.1. SUBSIDENCE ENGINEERING

- Brady BHG & Brown ET 1993 Rock Mechanics for Underground Mining Chapman & Hall New York Chapter 16
- Obert L & Duvall WI 1967 Rock Mechanics and the Design of

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Structures in Rock John Wiley & Sons New York Chapter 18

- vd Merwe JN & Madden BJ 2002 Rock Engineering for Underground Coal Mines SAIMM Special Publication Series No.7 Jhb
- van der Merwe JN 1995 Practical Coal Mining Strata Control Sasol Coal Division Jhb Chapter 5

8.1.2. SURFACE PROTECTION

- Salamon MDG & Oravecz KI 1976 Rock Mechanics in Coal Mining CoM of SA Jhb Chapter 7
- vd Merwe JN & Madden BJ 2002 Rock Engineering for Underground Coal Mines SAIMM Special Publication Series No.7 Jhb
- van der Merwe JN 1995 Practical Coal Mining Strata Control Sasol Coal Division Jhb Chapter 5

8.2. ENVIRONMENTAL EFFECTS

8.2.1. LONG-TERM STABILITY AND THE ENVIRONMENT

9. MINING STRATEGIES IN DIFFICULT CIRCUMSTANCES

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PAPER 3.3 MASSIVE UNDERGROUND MINING (HARD AND SOFT ROCK)

LEARNING OUTCOMES

2. GEOTECHNICAL CHARACTERISTICS

2.1. GEOLOGY

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

- Identify and describe the rock types associated with massive hard rock and soft rock ore bodies;
- Describe, explain and discuss how the rock types associated with massive ore bodies were formed;
- Sketch, describe and discuss geological sequences associated with massive ore bodies; and
- Sketch, describe and discuss major geological structures associated with massive ore bodies.

2.2. ROCK STRENGTH

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

- Describe, explain and discuss the compressive strengths of rock types associated with massive hard rock and soft rock ore bodies;
- Describe, explain and discuss the tensile strengths of rock types associated with massive hard rock and soft rock ore bodies;
- Describe, explain and discuss the relative rockmass strengths of rock types associated with massive hard rock and soft rock ore bodies; and
- Apply the above knowledge to the design of various types of mine workings in massive ore bodies.

2.3. ROCKMASS CHARACTERISTICS

2.3.1. GEOTECHNICAL ROCKMASS CLASSIFICATION

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

- Describe, explain and discuss the geotechnical characteristics of rock types associated with massive ore bodies;
- Describe, explain, discuss and apply standard rockmass classification and assessment systems to predict excavation stability;
- Describe, explain, discuss and apply Barton's Q system to classify rockmasses;
- Describe, explain, discuss and apply Bieniawski's RMR system to classify rockmasses;

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- Describe, explain, discuss and apply Laubscher's MRMR system to classify rockmasses;
- Apply rockmass classification results to determine the stability of mining excavations;
- Apply rockmass classification results to determine the stability of unsupported spans;
- Apply rockmass classification results to determine the support requirements of excavations;
- Apply rockmass classification results to determine the cavability of mining excavations;
- Apply rockmass classification results to determine the fragmentation of caved material;
- Describe, discuss and apply rockmass classification techniques for the selection of massive mining methods;
- Determine, discuss and apply rockmass 'm' and 's' parameters for the Hoek and Brown criterion based upon rockmass classification results;
- Determine, discuss and apply rockmass deformability from joint stiffness and rockmass classification results; and
- Describe, discuss and apply the rockwall condition factor (RCF) to predict tunnel stability and support requirements.

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2. GEOTECHNICAL CHARACTERISTICS
2.1. GEOLOGY

LEARNING OUTCOME 2.1.1

CONNECTION 1

Refer to
Paper 2, Outcome 2.1
Paper 3.1, Outcome 2.1
Paper 3.2, Outcome 2.1
Paper 3.4, Outcome 2.1

IDENTIFY AND DESCRIBE THE ROCK TYPES ASSOCIATED WITH MASSIVE HARD ROCK AND SOFT ROCK ORE BODIES

Hard rock ore bodies: Hard rock ore bodies typically contain ore such as gold, platinum, chrome, silver, diamonds, copper, iron, zinc, nickel and lead.

Soft rock ore bodies: Soft rock ore bodies typically contain coal, oil shale, tar sands, salt, talc and other minerals or geological materials.

Rock types associated with these types of ore are commonly igneous, metamorphic or massive sedimentary rock. Three basic rock types exist:

- igneous,
- sedimentary, and
- metamorphic rocks.

Within each of these basic rock types, a large number of different rock types have been formed. Typical rock types that can be found within South Africa have been summarised into Table 1.

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Table 1: Typical rock types in South African hard rock, tabular environments

Igneous rocks	Sedimentary rocks	Metamorphic rocks
Andesite – an intermediate volcanic rock Anorthosite – an igneous ultramafic rock composed predominantly of plagioclase Basalt – a volcanic rock of mafic composition Diabase or dolerite – an intrusive mafic rock forming dykes or sills Diorite – a coarse grained intermediate plutonic rock composed of plagioclase, pyroxene and/or amphibole Dunite – an ultramafic cumulate rock composed of olivine and accessories Gabbro – a coarse grained plutonic rock composed of pyroxene and plagioclase basically equivalent to basalt Granite – a coarse grained plutonic rock composed of orthoclase, plagioclase and quartz Harzburgite – a variety of peridotite; an ultramafic cumulate rock Kimberlite – a rare ultramafic, ultrapotassic volcanic rock and a source of diamonds Lamprophyre – an ultramafic, intrusive rock Norite – a hypersthene bearing gabbro Pegmatite – an igneous rock (or metamorphic rock) with giant-sized crystals Pyroxenite – a coarse grained plutonic rock composed of >90% pyroxene Syenite – a plutonic rock dominated by orthoclase feldspar; a type of granitoid Tuff – a fine-grained volcanic rock formed from volcanic ash	Argillite – a sedimentary rock composed primarily of clay-sized particles Banded iron formation – a fine-grained chemical sedimentary rock composed of iron oxide minerals Chert – a fine-grained chemical sedimentary rock composed of silica Coal – a sedimentary rock formed from organic matter Conglomerate – a sedimentary rock composed of large rounded fragments of other rocks Diamictite – a poorly sorted conglomerate Dolomite – a carbonate rock composed of the mineral dolomite +/- calcite Greywacke – an immature sandstone with quartz, feldspar and rock fragments within a clay matrix Jaspillite – an iron-rich chemical sedimentary rock similar to chert or banded iron formation Laterite – a residual sedimentary rock formed from a parent rock under tropical conditions Lignite – a sedimentary rock composed of organic material; otherwise known as Brown Coal Limestone – a sedimentary rock composed primarily of carbonate minerals Mudstone – a sedimentary rock composed of clay and muds Sandstone – a clastic sedimentary rock defined by its grain size Shale – a clastic sedimentary rock defined by its grain size Siltstone – a clastic sedimentary rock defined by its grain size	Anthracite – a type of coal Gneiss – a coarse grained metamorphic rock Marble – a metamorphosed limestone Mylonite – a metamorphic rock formed by shearing Pseudotachylite – a glass formed by melting within a fault via friction Quartzite – a metamorphosed sandstone typically composed of >95% quartz Schist – a low to medium grade metamorphic rock Serpentinite – a metamorphosed ultramafic rock dominated by serpentine minerals Slate – a low grade metamorphic rock formed from shale or silts

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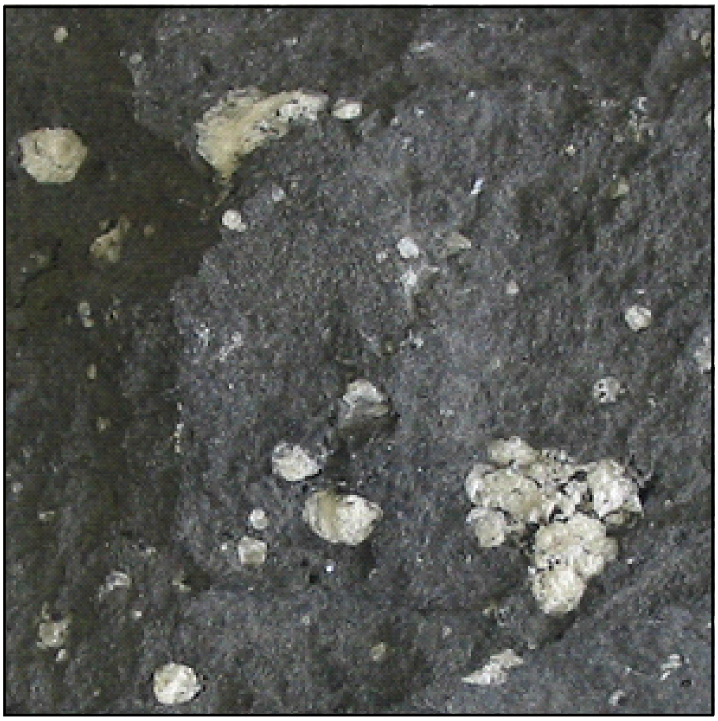

Figure 1: Andesite (igneous rock)

Figure 2: Limey shale overlain by limestone (sedimentary rocks)

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Figure 3: Banded gneiss (metamorphic rock)

LEARNING OUTCOME 2.1.2

DESCRIBE, EXPLAIN AND DISCUSS HOW THE ROCK TYPES ASSOCIATED WITH MASSIVE ORE BODIES WERE FORMED

Igneous rocks

Igneous rocks are formed by the solidification of molten lava or magma. This is an exothermic process (it loses heat) and involves a phase change from the liquid to the solid state. These rocks can be characterised as either extrusive or intrusive.

- Extrusive igneous rocks solidify from molten material that flows over the earth's surface (lava). Due to rapid cooling of the lava, crystal growth is prevented, and consequently fine grained igneous rocks are formed.
- Intrusive igneous rocks form from the solidification of molten magma that flows below the ground surface. The magma cools slowly underground, allowing crystal growth, and consequently coarse grained igneous rocks are formed.

Igneous rocks are given names based upon two things: composition (what they are made of) and texture (how big the crystals are).

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- Links (courtesy of Jersey University, Oregon)
  - Click here for more on igneous rock composition and texture
  - Click here for more on elements and minerals common in igneous rocks
  - Click here for more on magma and igneous rocks
  - Click here for more on plate tectonics and the formation of magma
  - Click here for a chart summarising the main divisions of igneous rocks
  - Click here for more on basalt and granite

Igneous rocks are typically made up of crystallized molten rock (magma) from the earth mantle. The material is made up of different chemical elements, different types of minerals, thus when it crystallize it result to different types of igneous rock. Magma may crystallize below the earth surface, forming intrusive igneous rocks; e.g. granite, pyroxenite. Some may crystallize above the earth surface, forming extrusive igneous rocks (lava), e.g. basalt, obsidian etc. Intrusive are differentiated from extrusive by grain size, the later normal characterized by very fine grain size that can barely be seen with the naked eye, compare to the intrusive.

Sedimentary rocks

Sedimentary rocks are formed at the surface of the earth, either in water or on land. Temperatures and pressures are low at the earth's surface, and sedimentary rocks show this in their appearance and the minerals they contain. Most sedimentary rocks are cemented by minerals and chemicals or by electrical attraction, while others remain loose and unconsolidated. The layers are typically parallel to the earth's surface (flat dip) and when they exist at high dip angles, some earth movement has occurred since the rock was formed.

There are three main types of sedimentary rocks:

- Clastic: Basic type. Accumulations of little pieces of broken up rock which have piled up and been "lithified" by compaction and cementation.
- Chemical: When standing water evaporates, dissolved minerals are left behind (i.e. deposits of salt and gypsum).
- Organic: Accumulation by organic processes. Many animals have calcium or silica for shells bones, and teeth. The remains of these animals fall to the seafloor and accumulate into a thick enough layer to form a sedimentary rock.(eg Limestone, chert)

- Links (courtesy of Jersey University, Oregon)
  - Click here for more on sedimentary processes and rocks (RCC).
  - Click here for more on sedimentary rocks (GPHS).

Metamorphic rocks

The metamorphic rock types get their name from "meta" (change) and "morph" (form). Any rock can become a metamorphic rock when the rock is moved into an environment where the minerals which make up the rock become unstable and change to other minerals in the new

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environmental conditions, which in most cases involves a rise in temperature and pressure.

The process of metamorphism does not melt the rocks, but instead transforms them into denser, more compact rocks. New minerals are created either by rearrangement of mineral components or by reactions with fluids that enter the rocks.

Some kinds of metamorphic rocks - gneiss and schist - are strongly banded or foliated (metamorphic minerals such as mica and feldspar begin to form and are oriented in layers by stress that gives the rock a striped appearance). Movement on a fault zone is an example of how stress may cause foliation to occur, together with great heat and pressure. The greater the stress the greater the intensity of foliation.

Common metamorphic rocks include slate, schist, gneiss, and marble.

INTERESTING INFO

Rock-forming and rock-destroying processes have been active for billions of years.

- In the hills South of the Hartebeestpoort dam in North West Province you can stand on dolomite, a chemical sedimentary rock, that was an algal reef in a tropical sea about 2.2 Billion years ago.
- In Gravellote Limpopo you can see schist, a metamorphic rock, that was once mud in a shallow sea over 3 billion years ago.
- The Halfway House dome at Midrand, Gauteng, which is now 1600 m above sea level and the bushveld rocks at Rustenburg are composed respectively of granite and anorthosites and amphibolites that solidified several thousand feet within the Earth.

LINK

- Links (courtesy of Jersey University, Oregon)
  - Click here for more on metamorphic processes and rocks (RCC).
  - Click here for more on metamorphic rocks (GPHS).

REFERENCE

- Pages 14, 25, 35, 95, Lurie J, South African Geology, 1987

LEARNING OUTCOME 2.1.3

SKETCH, DESCRIBE AND DISCUSS GEOLOGICAL SEQUENCES ASSOCIATED WITH MASSIVE ORE BODIES

NOTE

For the purpose of these learning guides, 'massive' ore bodies and 'massive' mining methods are defined as all those that do not fall into the typical tabular type ore bodies seen in gold, coal and platinum.
In South Africa, 'massive' ore bodies, as defined above, include the following:

- Zinc / Lead / Copper / Silver - Black Mountain,
- Gold - South Deep, Target, Sheba, Cons Murc (and antimony)
- Diamonds - Cullinan, Star, Sedibeng, Helam, Messina, Koffiefontein, Finch;
- Nickel / chrome - Nkomati;
- Phosphate -Phalabora;
- Platinum - Lesego, Mphathele.

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NOTE

This list and the expansion of this list into the geology and rock strengths are not necessarily complete, but are provided to indicate the variability of ore and rockmass types classified in this paper.

Black Mountain Mine

The Black Mountain Mine, an underground base-metal operation mining zinc/lead/copper/silver, is situated within a range of hills and mountains and displays a very diverse and complex geology. The Deeps ore body is the down-plunge continuation of the Broken Hill ore body. Mineralisation in the Deeps is hosted by iron formation and massive sulphide horizons. Ore minerals comprise galena (lead sulphide), sphalerite (zinc sulphide) and chalcopyrite (copper sulphide). Silver mineralisation occurs in the galena and chalcopyrite minerals.

REFERENCE

- Page 98, Lurie, J, South African Geology, 1987

South Deep

South Deep is situated on the north-western rim of the Witwatersrand Basin, comprising the Witwatersrand Supergroup, which consists of sedimentary rocks interspersed with quartz-pebble conglomerates (reefs).

LINK

- Geology at South Deep Van Biljon

REFERENCE

- Pages 14, 24, 34, Ryder and Jager, A Textbook on Rock mechanics, 2002
- Page 43, Lurie, J, South African Geology, 1987

Courtesy of Goldfields, the following:

The reefs are widely considered to represent extensive fluvial deposits into the basin where the gold is mainly of detrital origin, and are continuous as a consequence of the regional nature of the erosional surfaces. The South Deep mining right area is underlain by outliers of Karoo Supergroup shales and sandstones, followed by the Pretoria Group sediments and the Chuniespoort Group dolomites. The Chuniespoort Group overlies the Klipriviersberg Group volcanic rocks, which in turn are underlaid by the Central Rand Group that hosts the gold-bearing conglomerates (reefs) exploited by South Deep.

LINK

- Geology at South Deep Van Biljon

The reef horizons that are currently exploited at South Deep include the Ventersdorp Contact Reef (VCR) and the reef horizons that comprise the Upper Elsburgs of the Mondeor Formation. In the western half of the mining lease area the VCR occurs as a single reef horizon that overlies footwall lithologies of the Turffontein Subgroup. The Upper Elsburgs, which subcrop with the VCR in a north-northeast trend, comprise multiple stacked reef horizons that form part of an easterly-divergent clastic wedge as illustrated in the schematic section. This wedge attains a thickness of approximately 120 to 130 metres in the vicinity of the eastern boundary of the mining right area.

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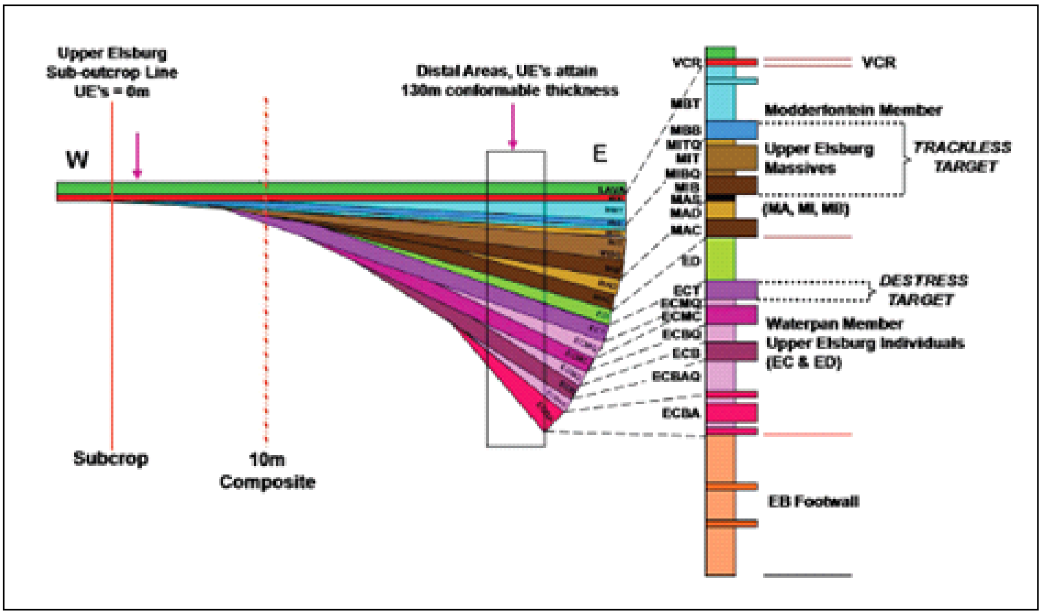

Figure 4: South deep ore bodies (Courtesy of Goldfields)

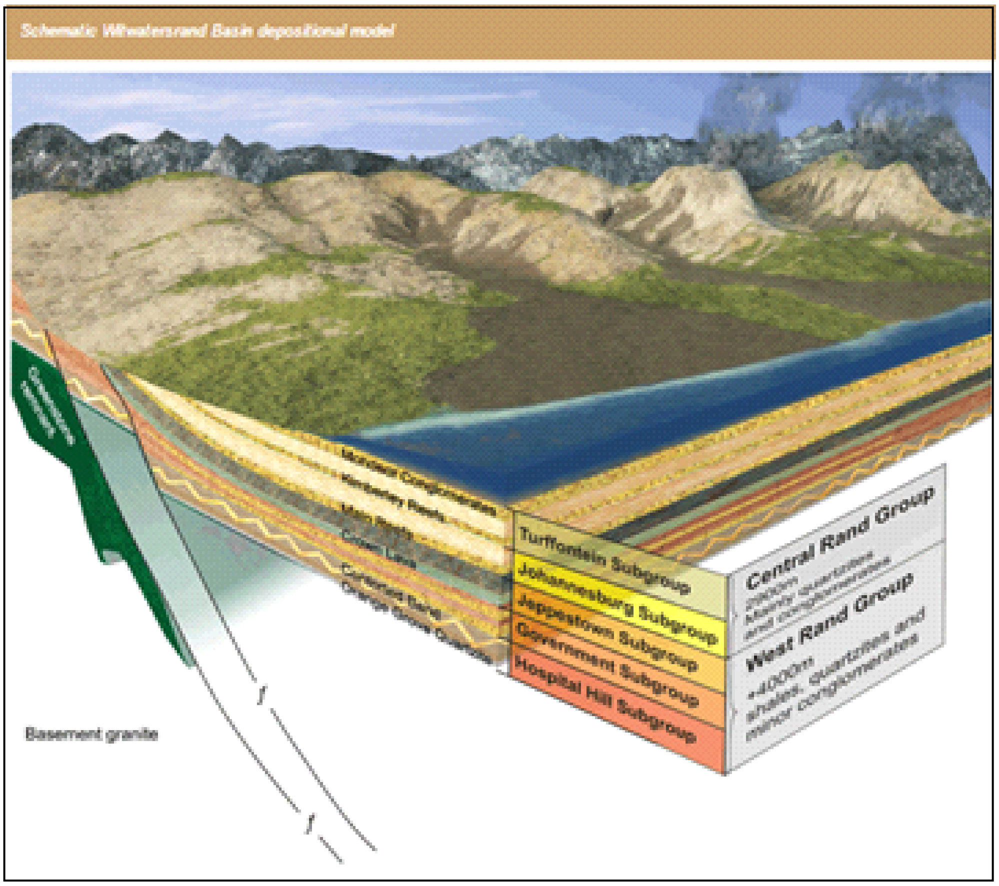

Figure 5 : Witwatersrand basin (Courtesy of Goldfields)

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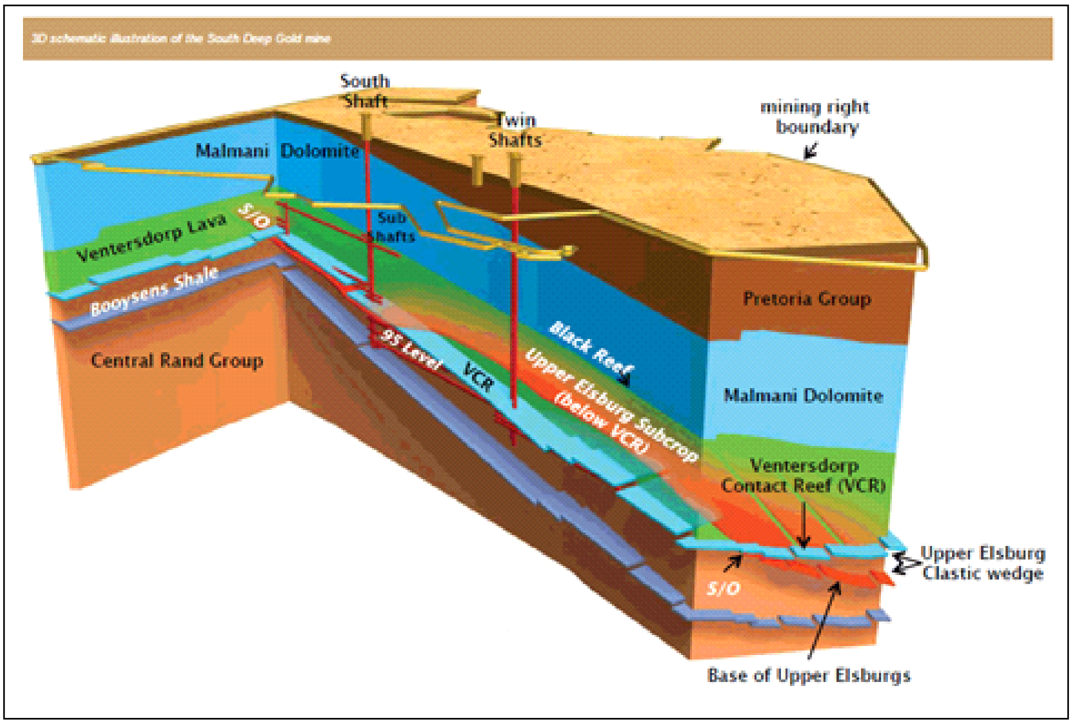

Figure 6 : South Deep layout (Courtesy of Goldfields)

- Geology at South Deep Van Biljon

Target

Target is located in the Free State and mines ore body that is complex (Figure 7), with 67 individual conglomerate (reef) bands, stacked in a fan-shaped sequence of Elsburg and Dreyerskuil sediments, all of which are approximately 150 metres thick.

The Eldorado Reef mineralisation is contained in a series of ancient alluvial fans, stacked sequentially within the rocks of the Central Rand Group of the Witwatersrand Supergroup.

- Page 15, Ryder and Jager, A Textbook on Rock Mechanics, 2002
- Page 47, 83, 87, Lurie, J, 1987 South African geology

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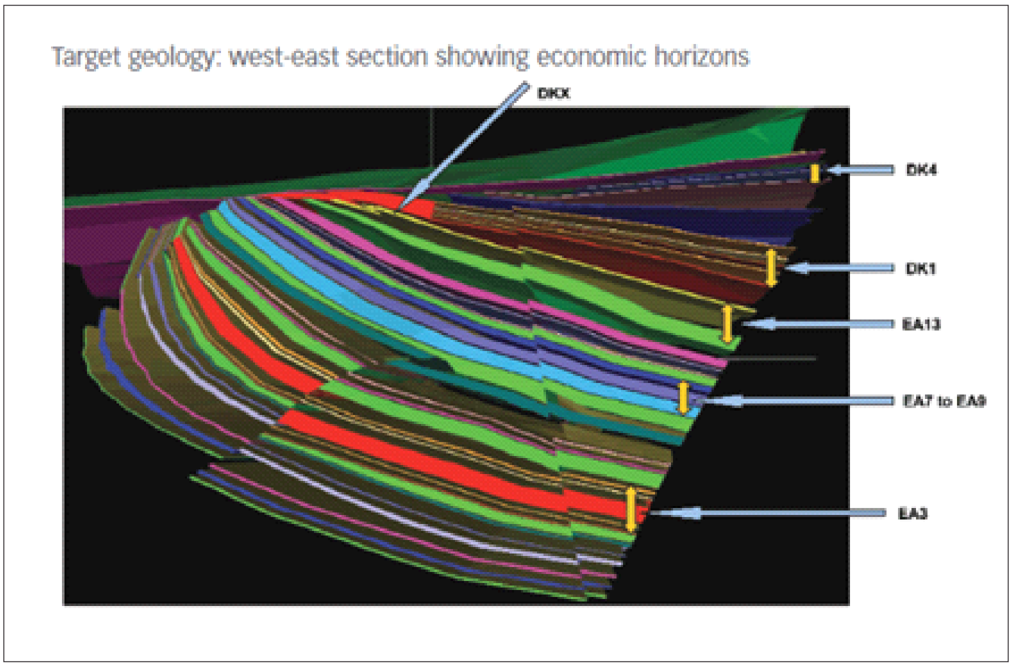

Figure 7 : Target ore bodies (Courtesy of Harmony)

INTERESTING INFO

Mining in this area is not limited to gold deposits, but also includes asbestos, iron, magnesite, barite, nickel-copper, talc, mercury, tungsten and zinc.

Sheba

The Sheba ore body is part of a hydrothermal gold deposit in the Barberton area and falls within the Swaziland Supergroup and Barberton Sequence, as shown in the table below. The deposit is part of what is known as the Greenstone belt and gold occurs in all the rock groups shown in Table 2 and Figure 8. Structures were important in the control of the mineralisation, causing the size of deposits to be associated with the inclination of the fractures.

Table 2: Barberton sequence (Lurie, 1987)

Group	Subgroup	Formation	Main rock types
Moodies		Baviaanskop	Shale, Quartzite, Conglomerate
Joe's Luck		Shale, Quartzite, Lava	Clutha
Shale, Quartzite, Conglomerate		Fig tree	Schoongezicht
Shale, Chert, Greywacke, Lava		Belvue road	Greywacke, Shale, banded ironstone
Sheba		Greywacke, Shale	Overwacht
Geluk		Swartkoppie	Chert
Kromberg	Lava, Banded Cherts	Hooggenoeg	Metamorphosed
Tjakastad	Komati	Lava, Chert	Theespruit
Lava, Chert	Sandspruit	Lava, Chert	

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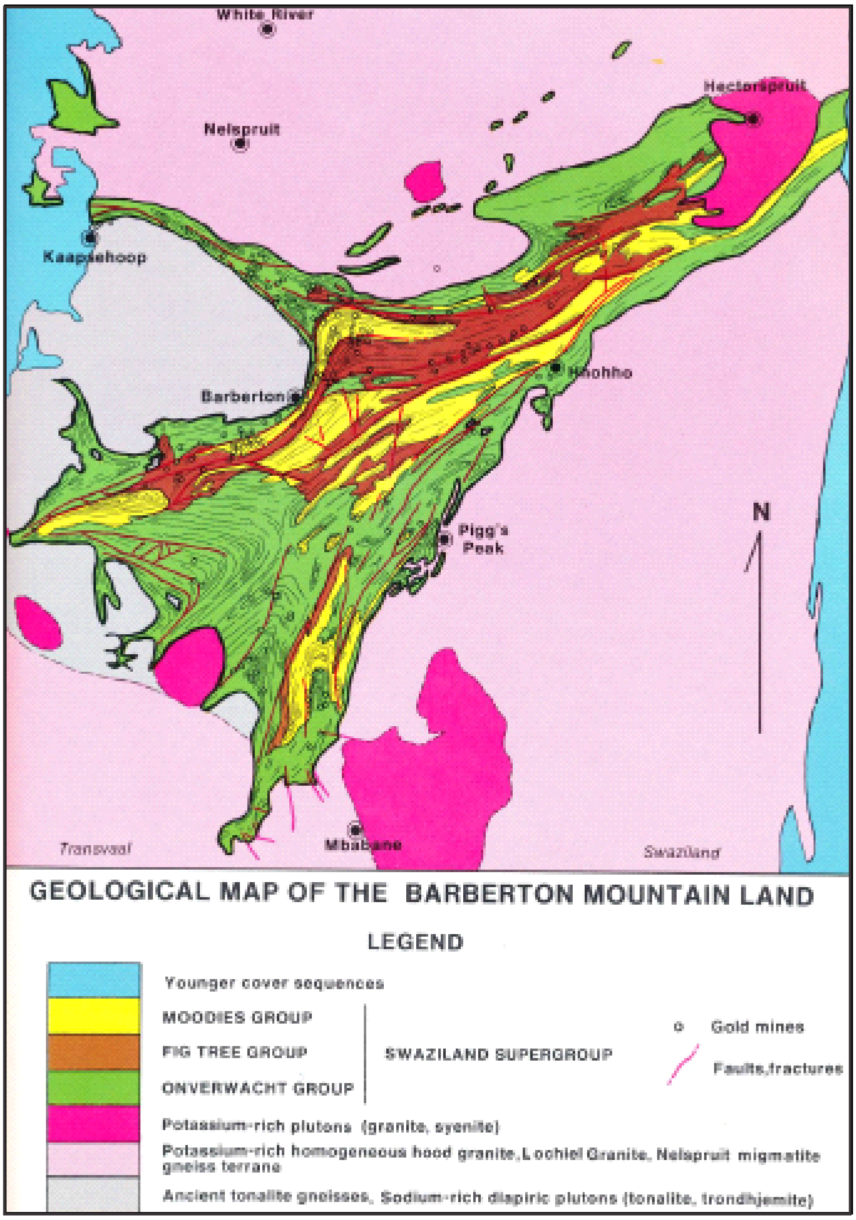

Figure 8: Barberton geology (Dannhauser, 2012)

LINK

- The history of mining Barberton Dannhauser

REFERENCE

- Page 44, Lurie, J, 1987, South African Geology

Helam Diamond Mine
The area surrounding Helam Diamond Mine (Swartruggens area) is underlain by bedrock of the Pretoria Group within the larger Transvaal Supergroup. The bedrock is generally sub-horizontal, dipping at a shallow angle of between 2 and 3° to the north. At surface, and extending to a depth of approximately 180m, lie andesitic formations of the Hekpoort Group, commonly amygdaloidal lava. Below this, to a depth of approximately 550m below surface, are shales of the Lyttleton Formation of the Chuniespoort Group. These shales are intruded by pre-Bushveld Igneous Complex diabase sills with thicknesses of up to 80m. From 550m to the mine bottom (±1000m), the shales have transitioned to increasingly arenaceous rocks, and may be more correctly described as impure

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quartzites. Three dominant kimberlite types are mined, namely the Main Fissure, Changehouse Fissure and Muil Fissure. The Main Fissure is made up of multiple intrusive phases and has a near-vertical dip.

REFERENCE

- Page 93, Lurie, J, South African Geology, 1987

Sedibeng diamond mine

Sedibeng diamond mine mines (Bellsbank area) a near vertical fissure system. The host rocks to the ore bodies are dolomitic argillites and quartzites overlying Ventersdorp lavas, which can be correlated with the lower Chuniespoort section of the Transvaal Sequence of the Pretoria System. Along the fissure system, dolomites interlayered with argillite and quartzites form a dominant westerly dipping (-5º) unit to a depth of 405 metres. A unit of quartzites containing minor shale and dolomitic lenses occurs 405 to 540 metres below surface. The base of these quartzites at 450 metres forms an unconformity with the underlying Ventersdorp sequence.

REFERENCE

- Page 93, Lurie, J, South African Geology, 1987

Koffiefontein

Koffiefontein mines a kimberlite pipe surrounded by host rock that consists of shale, dolomite and granite where the pipe was mined opencast until the presence of granite host rock allowed the creation of stable underground tunnel layouts.

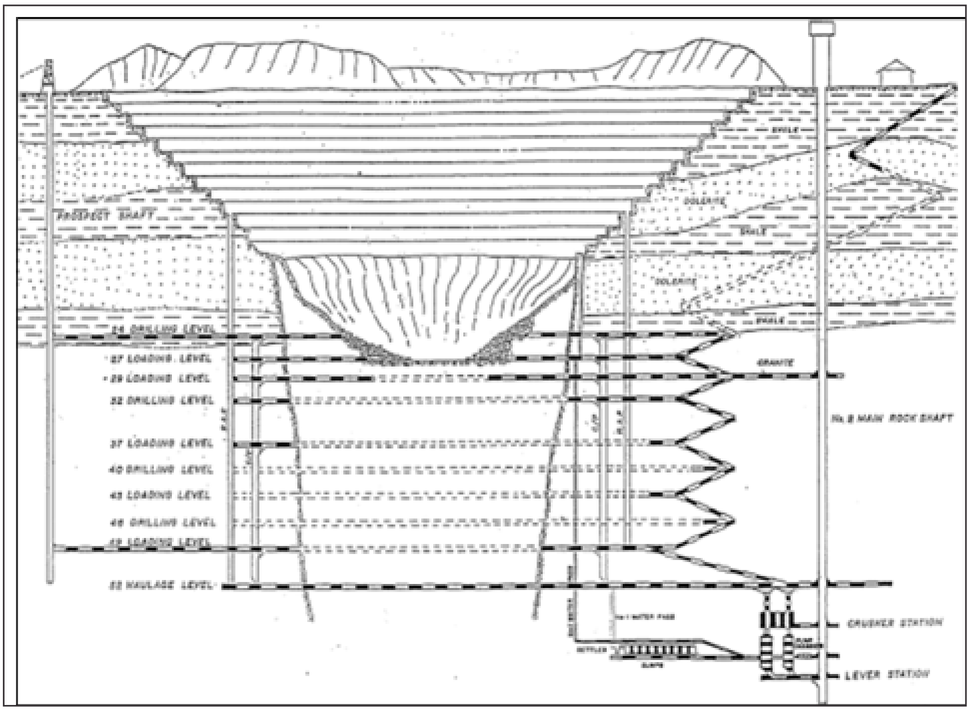

Figure 9: Koffiefontein geology (Courtesy of SRK Consulting)

REFERENCE

- Page 93, Lurie, J, South African Geology, 1987

Finch

Finch mines a classic diamondiferous kimberlite pipe within country rocks that consist of banded ironstones overlying dolomites and limestones. The pipe itself consists of weathered kimberlite (yellow ground) to a depth of approximately 100m with unweathered material (blue ground) beneath. The kimberlite has been classified into a series of different kimberlites.

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- Page 93, Lurie, J, South African Geology, 1987

Cullinan

Cullinan (previously known as Premier Diamond Mine) is located on a diamond-bearing kimberlite pipe and has three types of kimberlite, namely brown, grey and black (hyperbyssal). The pipe is surrounded by quartzite, felsite and norite in the shallower areas, with the Transvaal Supergroup forming the base.

- Page 93, Lurie, J, South African Geology, 1987

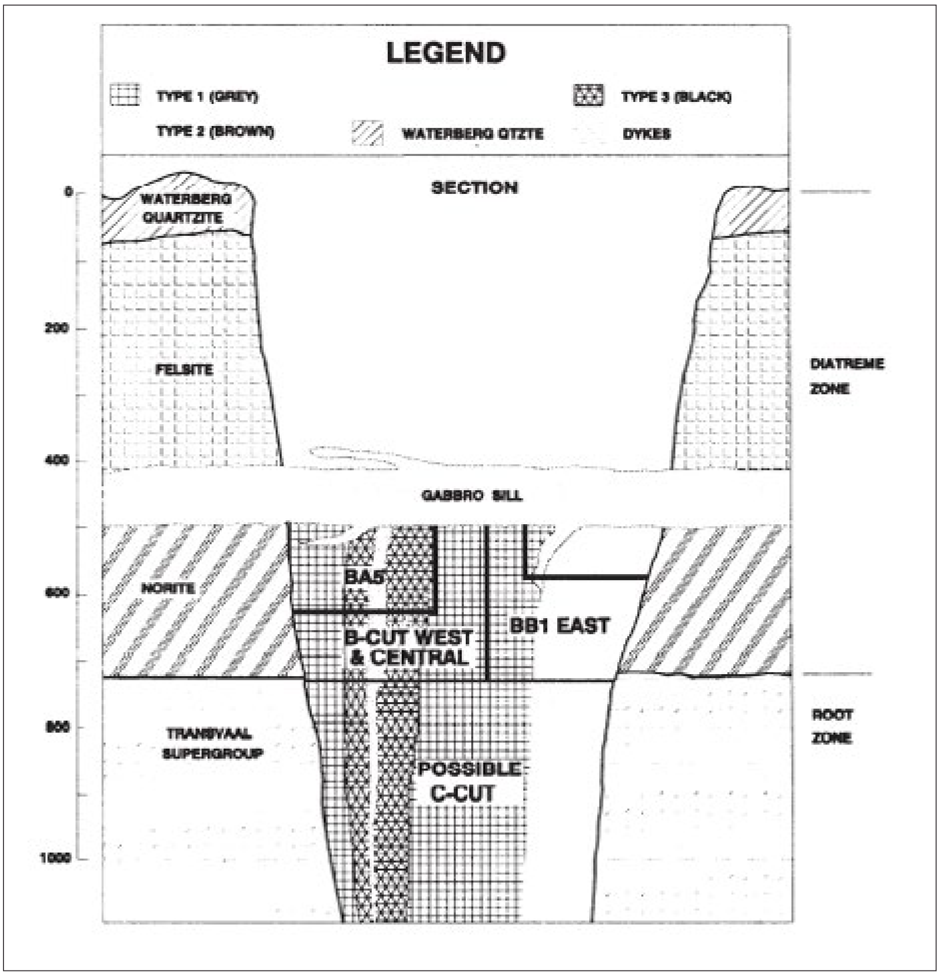

Figure 10: Cullinan geology (Bester, 2005)

Kimberlite occurs freely in different facies/types within diamond mines and is reported to have very different properties. The rock engineer should be aware of the specific kimberlite types present so that the appropriate properties can be applied to designs.

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Figure 11: Kimberlite

- Kimberlite classes Skinner
- Kimberlite types in RSA Ito
- Cullinan Kimberlite types Scott
- Kimberlite Properties Morkel

Nkomati is producing from a massive sulphide ore body situated in the footwall of the Uitkomst ultramafic complex and within the Transvaal Sequence, and consists of three lenses separated by diabase intrusions.

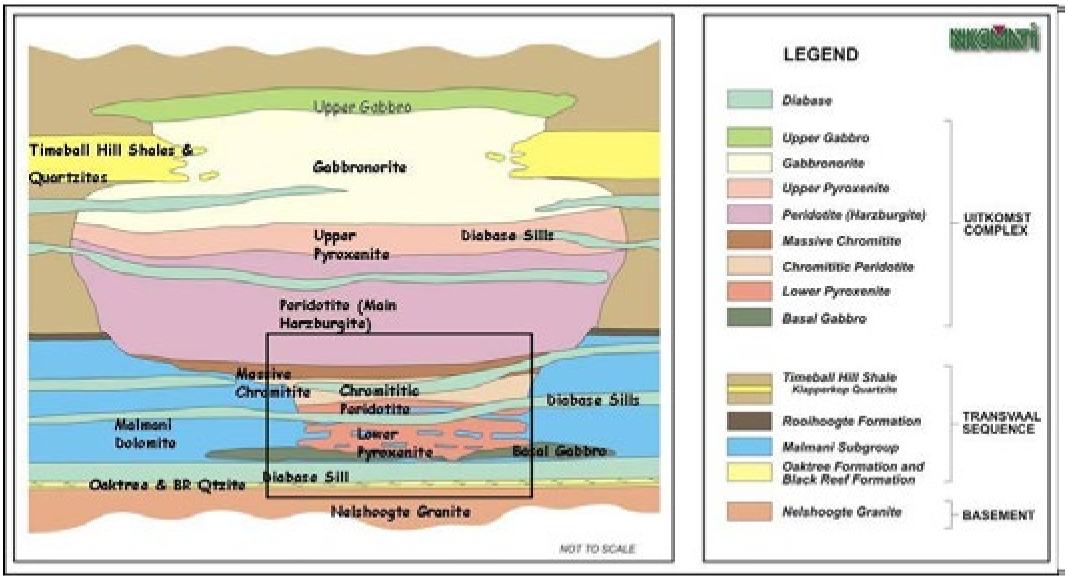

Figure 12: Nkomati geology (Courtesy of ARM)

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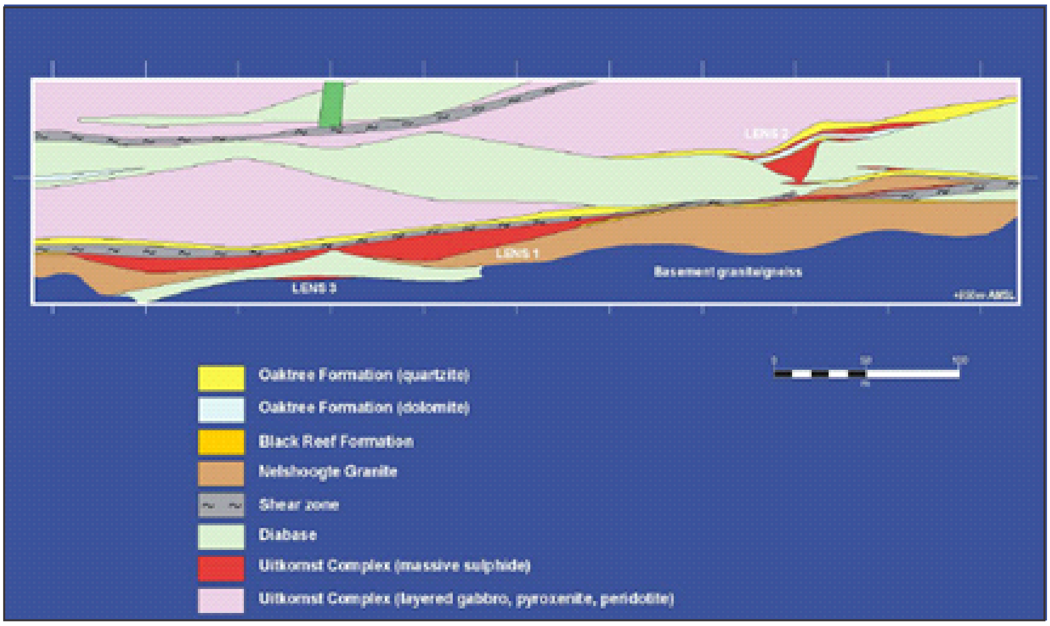

Figure 13: Geological section through Nkomati (Courtesy of ARM)

Phalaborwa

The Phalaborwa mine operates in the Archaean granite and gneiss group with a successful underground block-cave mine in a relatively competent ore body.

The Phalaborwa Igneous Complex (PIC) consists of a central backbone of ultra-basic rocks surrounded by numerous plugs of syenite. The pyroxenite body, which probably represents the remains of a volcanic vent, encloses three areas of ultra-basic pegmatoid intruded into Archaean granite gneiss country rock.

The Igneous Complex is a vertical pipe of pyroxenite, foskorite and carbonatite, roughly kidney-shaped. It is made up of three concentrically zoned lobes, each of which is a major phosphate source with its own characteristics. The Complex contains phosphate and copper with the co-products of silver, gold, magnetite (iron ore), vermiculite, zirconia and uranium (uraninite-thorianite and baddeleyite).

The ultramafic rocks of the PIC contain economic deposits of apatite and vermiculite. The Complex is unique when compared to other African alkaline complexes, because its carbonatite pipe contains economic copper minerals.

The geological sequence within the Phalaborwa area is recorded as:

- Basement, comprising Archaean granites and gneisses of the Kaapvaal Craton, currently with steeply dipping foliation striking east west.
- Intrusion of pyroxenites during the Proterozoic, and subsequently carbonatites within the core of these pyroxenites, and these comprise the PIC.
- Phosphate, copper, baddeleyite, magnetite and uranium/thorium mineralisation emplaced during these intrusions.
- Syenite intrusions into the basement regionally, including around the periphery of the PIC.
- North-east to south-west striking dolerite dykes intruded during the Mesozoic Era.
- Weathering of phlogopite developed the vermiculite mineralisation.

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REFERENCE

- Pages 51, 97, 100, 108, Lurie, J, South African Geology, 1987

Lesego

The Lesego project aims at mining the Merensky Reef and UG2 Chromitite in the eastern limb of the layered package known as the Rustenburg Layered Suite, which forms part of the Bushveld Igneous Complex. The Rustenburg Layered Suite forms three principal limbs, namely the Western Limb (Rustenburg – Brits area), the Eastern Limb (Steelpoort – Burgersfort Area) and the Northern Limb (Mokopane area).

The project is situated adjacent to the Phosiri dome, an inticlinal structure of the footwall rocks of the Bushveld Complex, consisting of Transvaal Supergroup rocks. The dome represents an uplift of footwall rocks and as a direct consequence also caused the uplift of the Critical, Man and Upper Zone lithologies, resulting in shallower deposits.

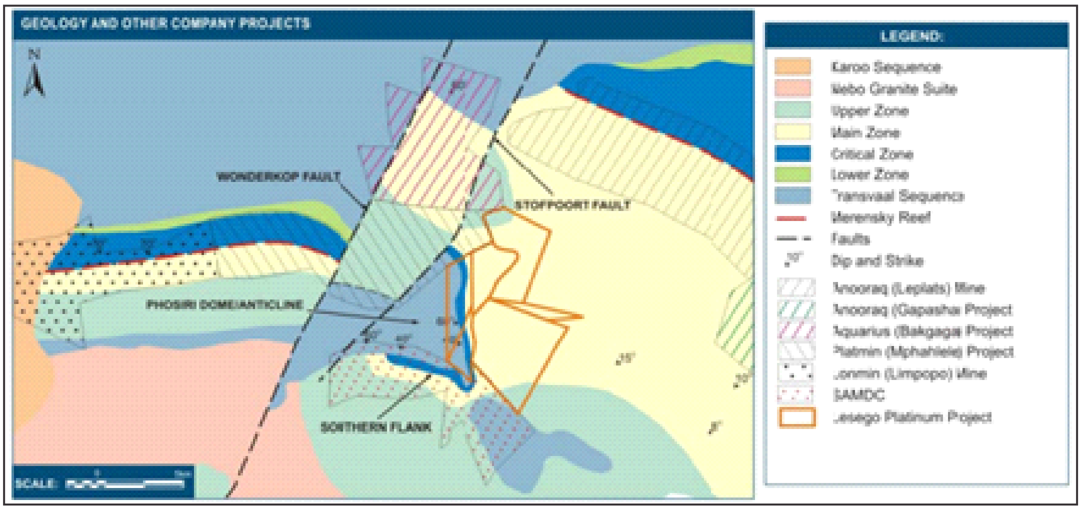

Figure 14: Lesego geology (Courtesy of Village Main Reef)

The Merensky (red) and UG2 (green) reefs are shown in Figure 15 below, indicating the increase in ore body dips as the Phosiri dome is approached.

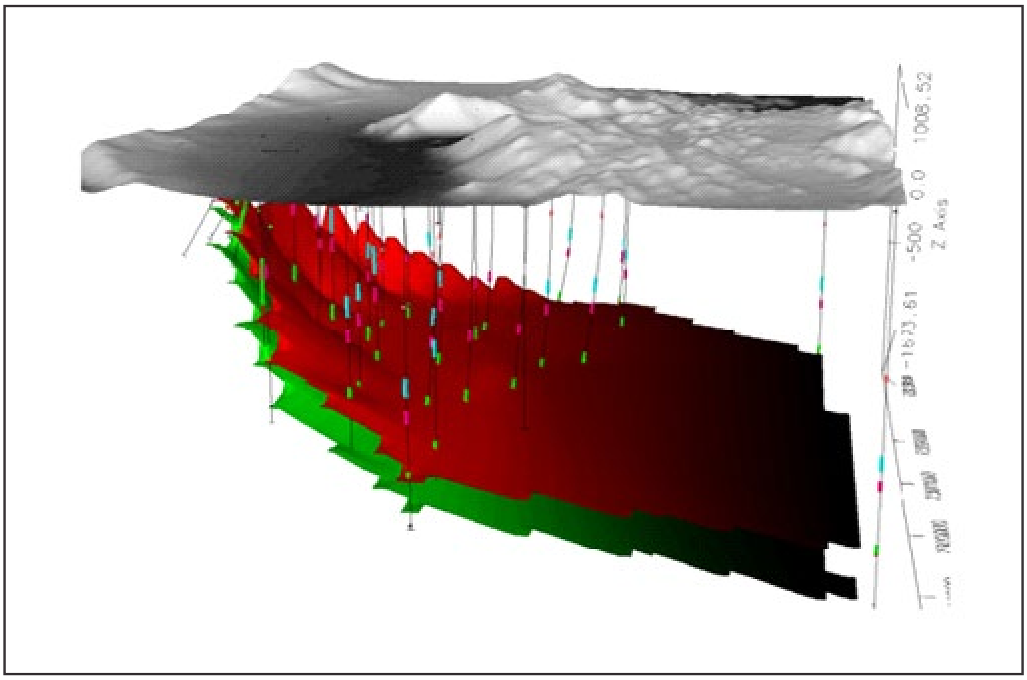

Figure 15: Ore body dips (Courtesy of Village Main Reef)

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Mphahlele

The Mphahlele deposit is situated along the east-west trending, northern part of the Eastern Limb of the Bushveld Complex, a layered igneous complex that is divided into the lower Rustenburg Layered Suite (RLS) of ultramafic to mafic rocks, the Lebowa Granite Suite (LGS) and the felsic extrusive rocks of the Rashoop Granophyre Suite (RGS).

The PGE mineralisation occurs within the UG2 Chromitite Layer (UG2) and the Merensky Reef, lying within the Upper Critical Zone (UCZ) of the RLS.

Throughout the BC, the reefs are tabular bodies, and on the Mphahlele project, both the Merensky Reef and UG2 have the form of a gentle, open plunging anticline, dipping at an average of 51° to the south-southeast.

The UG2 is a chromitite layer, approximately 1.2m thick and in this northern area of the Eastern Limb it is unique in that it is sulphide rich.

The Merensky Reef is a pyroxenite layer typically 3 to 6m thick, with mineralisation disseminated in the top metre of the pyroxenite.

REFERENCE

- Page 49, Lurie, J, 1987, South African Geology
- Page 38, Ryder and Jager, A Textbook on Rock Mechanics, 2002

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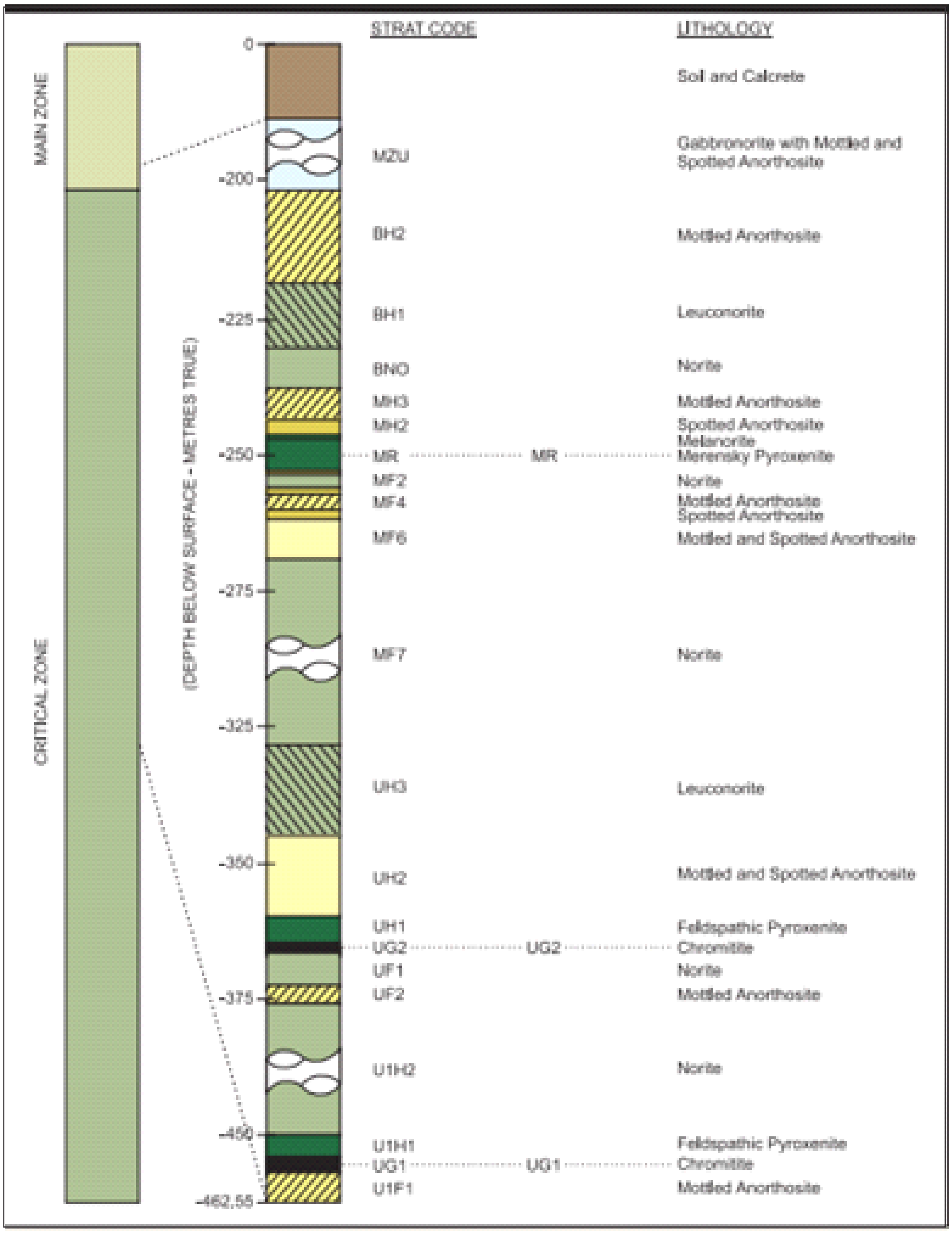

Figure 16: Mphahlele stratigraphic column (Courtesy of Platmin)

LEARNING OUTCOME 2.1.4

SKETCH, DESCRIBE AND DISCUSS MAJOR GEOLOGICAL STRUCTURES ASSOCIATED WITH MASSIVE ORE BODIES

Faults:
A fracture in the rock on which relative movement / displacement has taken place. Faulting is the tectonic process which disrupts the continuity of the regular pattern dipping strata.

In the case of a normal fault, which generally results from high vertical stresses inducing horizontal tensional forces, the hangingwall (upper side) has apparently moved down.

NOTE

From a mining point of view, this usually results in a 'loss of ground'.

High horizontal Compressive forces tend to cause reverse faults. In these

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the hangingwall (upper side) has apparently moved up.

Strike slip faults are caused by high horizontal Compressive forces in one direction inducing vertical tensional forces in the other horizontal direction

NOTE

The result is usually a 'gain of ground'.

IMPORTANT

The presence of faults in a rock mass may affect the stability of mining excavations as additional surfaces on which rock mass movement can occur is provided. At the same time, faults are often associated with a number of 'sympathetic' faulting (similar dip and strike structures), suggesting a rock mass that is of poorer quality and thus potentially has a higher potential for instability.

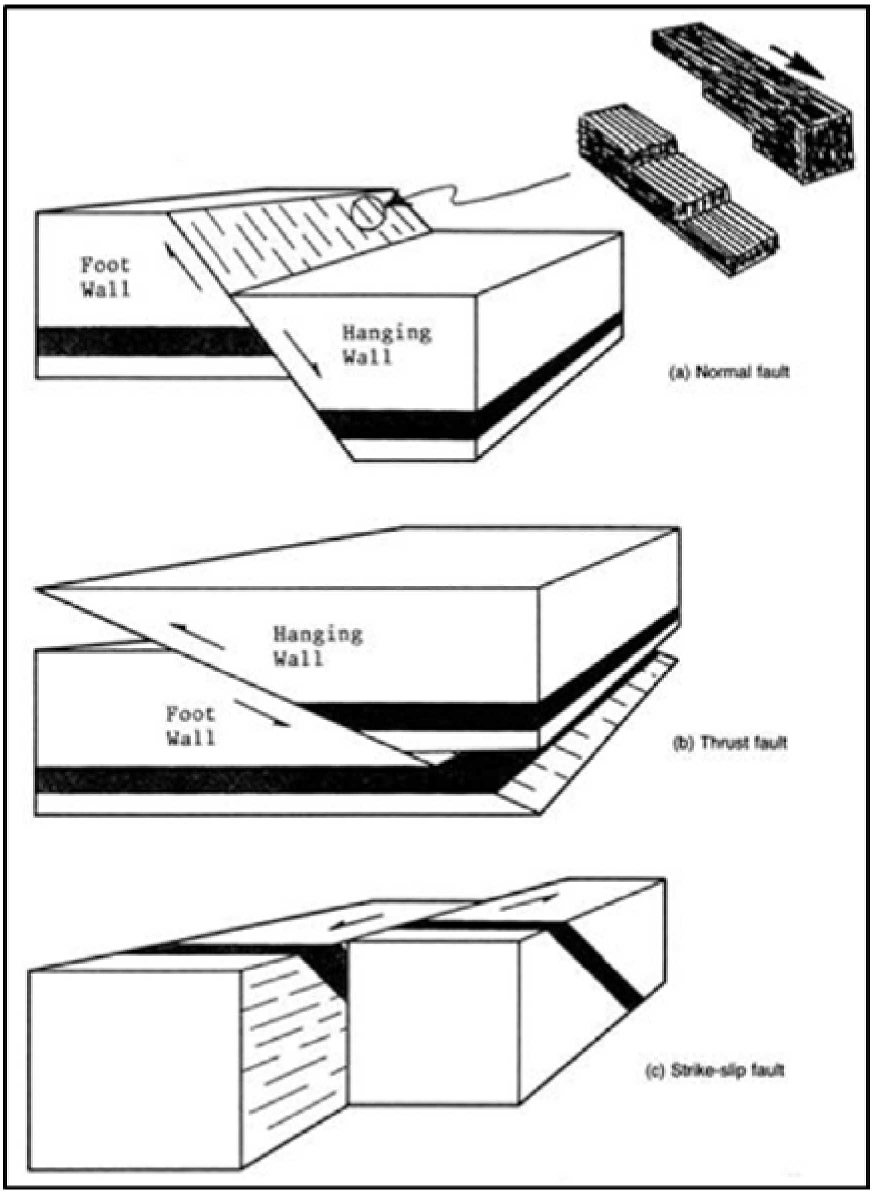

Figure 17: Diagram of fault structures

Dykes and sills:
Dykes are magmatic intrusions that cut across bedding planes or fabric of the rockmass of the country rock whilst Sills are magmatic intrusions ejected along bedding planes or fabric of the rockmass(parallel

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to sub-parallel) in the country rock (Figure 18). The presence of these structures affects rock mass around it due to metamorphism of the rock-mass and stress changes inducing jointing and may improve or reduce rock mass strengths and quality.

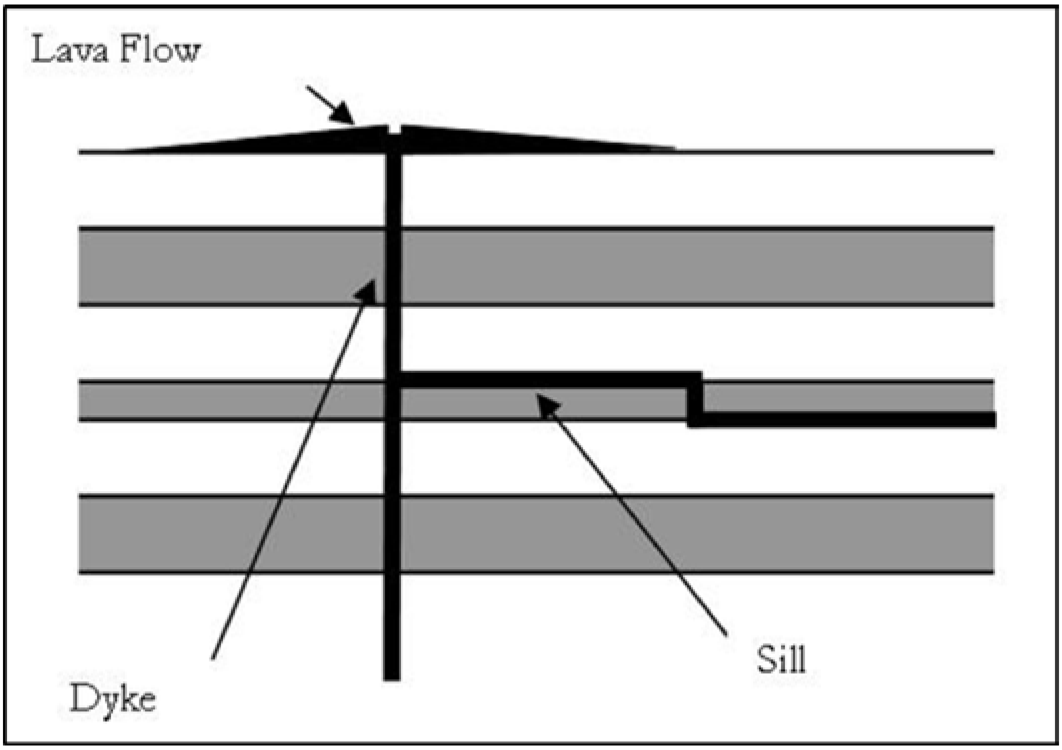

Figure 18: Diagram showing dyke and sill

Joints:
A joint is a fracture of natural origin in the rock that has no visible or measurable movement on the surface (plane) of the fracture. They most frequently occur as joint sets and systems, where a joint set is group of parallel, evenly spaced joints. A joint system consists of two or more joint sets.

Joints have a significant influence on ground conditions in that it reduces rock block sizes and thus also rock mass quality, resulting in an increase in potential for failure to occur.

NOTE

Smaller joint spacings suggest smaller block sizes and a higher potential for a rockmass to unravel and collapse.

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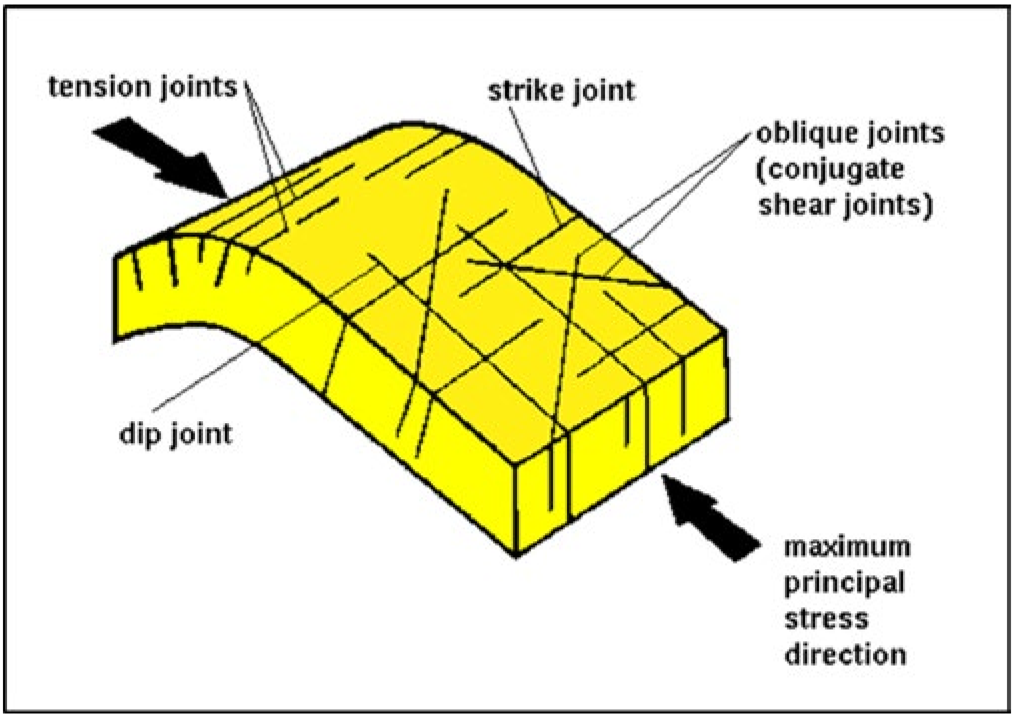

Figure 19: Diagram showing joints

Bedding planes:

The surfaces between successive layers (bedding) of a Sedimentary deposit are known as bedding partings (Figure 9). Different beds can be distinguished from one another by their composition, grain size, texture and colour or by a distinct physical break or plane of separation.

The combination of these properties defines the rock type and the important rock mechanic parameters of 'strength', elastic moduli, friction characteristics, abrasiveness and toughness.

Apart from variations in these properties, the most important features of bedding that influence rock wall stability are bed thickness (bedding plane spacing) and the quality of the contact between successive beds.

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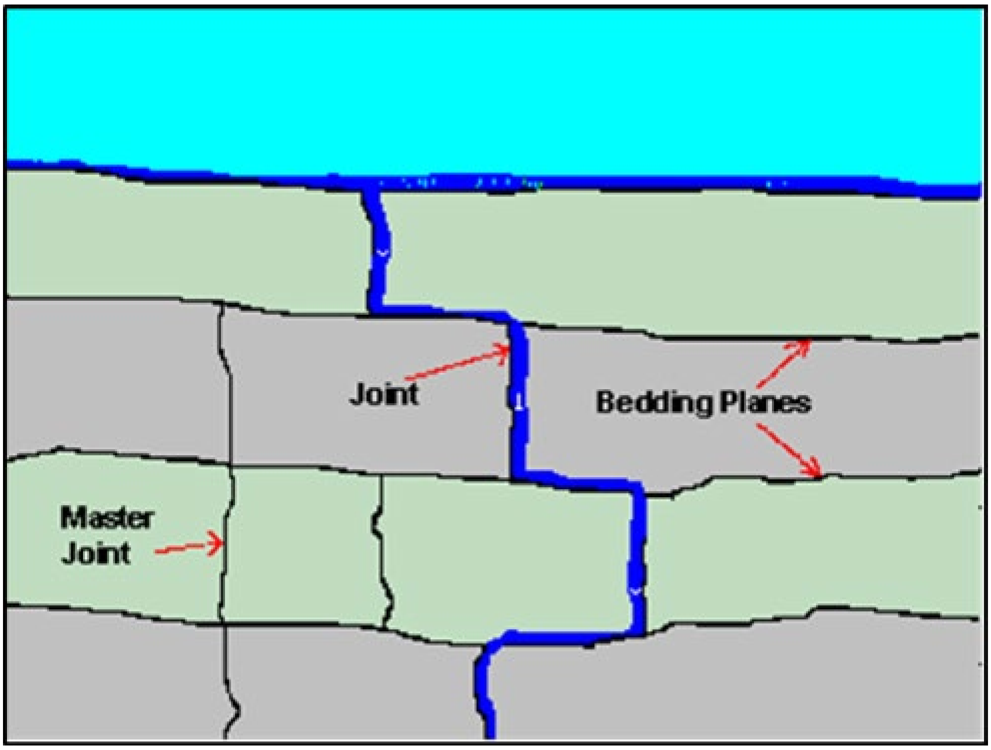

Figure 20: Diagram showing typical bedding planes

Folds:
When stress, temperature and rock properties are suitable then rocks will act plastically and deform by folding rather than fracture to create faulting As a result of this permanent deformation, a stack of originally flat stratified rocks become bent or curved (Figure 21). Folds form under various stress conditions, hydrostatic pressure, pore pressure and temperature gradients. In the process of folding this rock mass, a number of fractures are created at different orientations, reducing its inherent ability to remain stable during mining.

NOTE

Folding could affect principal stress components (magnitude and orientation) due to 'stress lock-up' after folding, while folded rockmass often creates difficult mining conditions.

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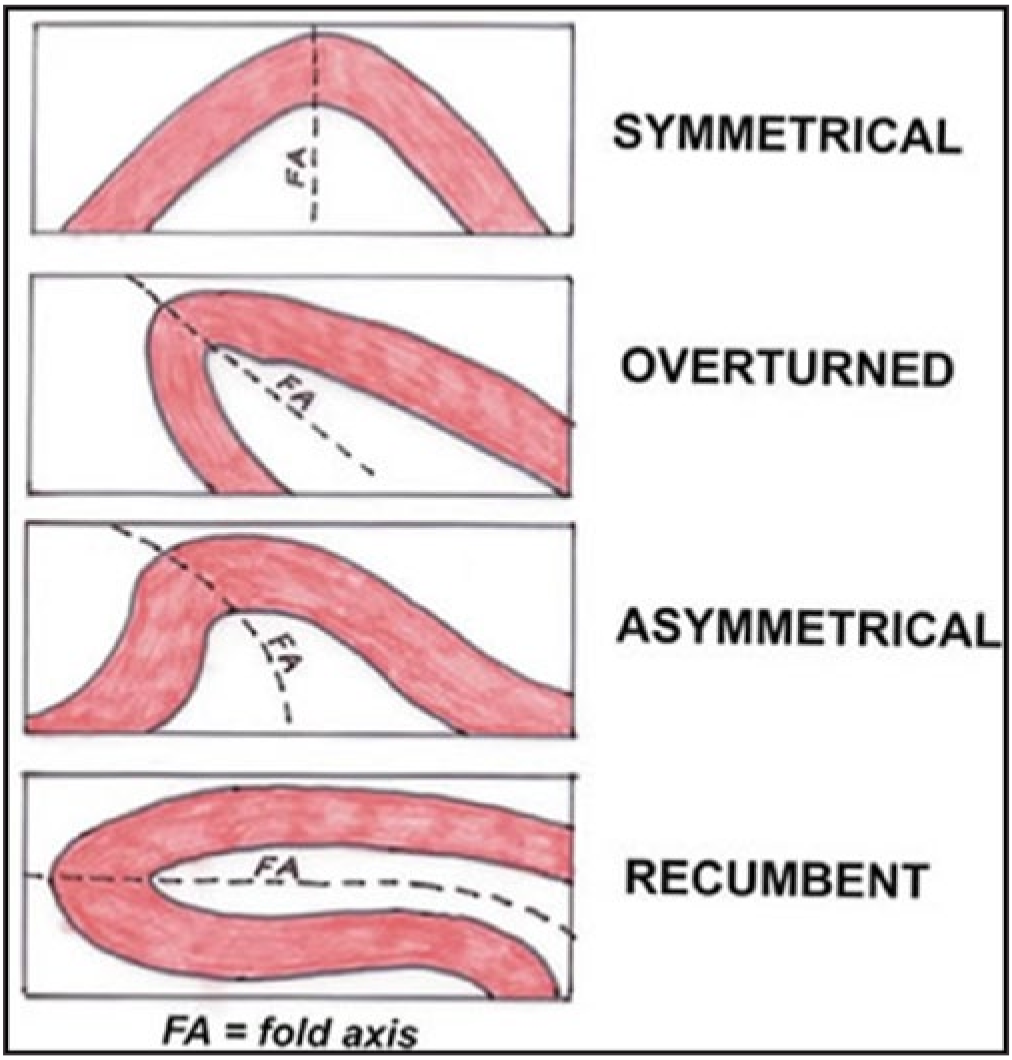

Figure 21: Diagram showing folding of strata

REFERENCE

- Page 4, Updated cave mining handbook Laubscher

2.2. ROCK STRENGTH

CONNECTION 2

Refer to
Paper 1, Outcome 4.1 and Outcome 6.1
Paper 3.1, Outcome 2.2
Paper 3.2, Outcome 2.2 and Outcome 6.1
Paper 3.4, Outcome 2.3

LEARNING OUTCOME 2.2.1

DESCRIBE, EXPLAIN AND DISCUSS THE COMPRESSIVE STRENGTHS OF ROCK TYPES ASSOCIATED WITH MASSIVE HARD ROCK AND SOFT ROCK ORE BODIES

Kimberlite

UCS tests were conducted by various authors and provided the information shown in Table 3.

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Table 3: UCS of rock material surrounding the Cullinan ore body

Material	Courtesy Owen, 1984	Courtesy Bartlett, 2001
UCS	UCS	Hypabyssal Kimberlite
176	73-1193	Grey Kimberlite
129	80-130	Norite
160	140-220	

Point load index tests conducted on the same materials yielded the presentation in the figure below (Note that the strength is given in MPa).

NOTE

The strength differences between the different Kimberlite types are shown.

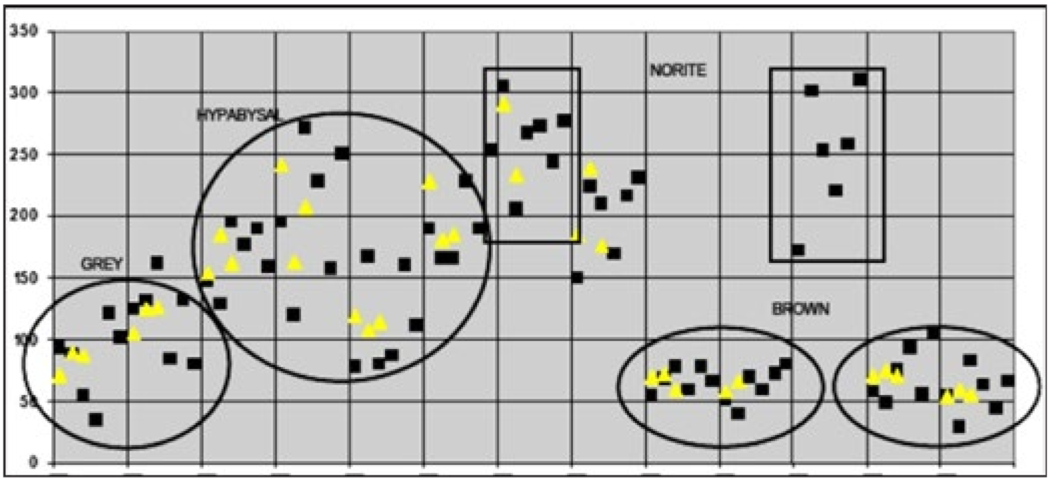

Figure 22: Point load index test results (Courtesy of Cullinan Mine)

Table 4: Sample strength tests results

Rock type	UCS (MPa)	Minimum UCS (MPa)	Maximum UCS (MPa)	E (Gpa)	Density (ton/m3)	Tensile strength (MPa)
Black Hypabyssal	155	109	190	51.3	2.79	11.7
Piebald Hypabyssal	156	82	199	42.2	2.78	12.9
All Hypabyssal	156	82	199	48.9	2.77	12.3
Grey Kimberlite	115	50	203	42.5	2.79	14.1

Gneiss, amphibolites, sulphides

Various rock strength test results for this environment are summarised in the table below.

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Table 5: Strengths and properties

Rock type	Density (kg/m3)	Young's Modulus (GPa)	Poisson's ratio	UCS (min-max) (MPa)
Hangingwall gneiss	2 767	72	0.19	161-201
Ore body (amphibolites and sulphides)	2 906	81	0.23	100-201
Footwall gneiss	2 776	67	0.19	143-249

NOTE

Not all the environments and materials present in each area have been shown in these sections and the user is advised to ensure that the material properties are known in each area where design work is being executed.

LEARNING OUTCOME 2.2.2

CONNECTION 3

Refer to "LEARNING OUTCOME 2.2.1" on page 56

DESCRIBE, EXPLAIN AND DISCUSS THE TENSILE STRENGTHS OF ROCK TYPES ASSOCIATED WITH MASSIVE HARD ROCK AND SOFT ROCK ORE BODIES

REFERENCE

- Kimberlite properties Morkel

LEARNING OUTCOME 2.2.3

CONNECTION 4

Refer to
Paper 1, Outcome 6.1
Paper 3.1, Outcome 2.2.2

DESCRIBE, EXPLAIN AND DISCUSS THE RELATIVE ROCKMASS STRENGTHS OF ROCK TYPES ASSOCIATED WITH MASSIVE HARD ROCK AND SOFT ROCK ORE BODIES

Rock mass strengths are determined by a combination of material sample strength (UCS) and the composition of the rock mass in terms of jointing (spacing, surface conditions, infilling, joint length and surface shape), groundwater, etc.

LEARNING OUTCOME 2.2.4

CONNECTION 5

Refer to "Outcome 2.3" on page 58, "Outcome 4" on page 223 and "Outcome 5" on page 288

APPLY THE ABOVE KNOWLEDGE TO THE DESIGN OF VARIOUS TYPES OF MINE WORKINGS IN MASSIVE ORE BODIES

The strength of rockmass materials surrounding massive ore bodies is critical in designing underground excavations. The focus of this knowledge during design would be to ensure stable mining spans in some methods, while spans must be sufficient in other methods to ensure caving of material.

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CONNECTION 6

Refer to
Paper 1, Outcome 6.1 and 6.2
Paper 2, Outcome 3.1
Paper 3.1, Outcome 2.3 and 3.2
Paper 3.2 Outcome 2.3

CONNECTION 7

Refer to
Paper 3.1, Outcome 2.2.2
Paper 3.2 Outcome 2.2.2
Paper 3.4 Outcome 2.2.1.1

2.3. ROCKMASS CHARACTERISTICS

2.3.1. GEOTECHNICAL ROCKMASS CLASSIFICATION

LEARNING OUTCOME 2.3.1.1

DESCRIBE, EXPLAIN AND DISCUSS THE GEOTECHNICAL CHARACTERISTICS OF ROCK TYPES ASSOCIATED WITH MASSIVE ORE BODIES

In addition to the information provided in the outcomes referred to above and discussed in Papers 3.1, 3.2 and 3.4, the following is provided (but is not deemed exhaustive).

Table 6: Geotechnical characteristics

Environment	Geotechnical characteristics	Impact on designs for total or partial extraction mining methods
Diamond pipes	Rock strengths vary depending on the host and/or ore body types where kimberlite could vary between 100MPa and 160MPa. Rock quality (RMR) of the ore body is typically <60 with slightly higher RMR values for the host rock. Within the ore body, typical structure frequencies are >1 joint/m with joint conditions typically slightly separated and weathered.	Poor rockmass strength (low RMR, high joint frequency, etc.) of the host rock can result in caving into the caved ore body and result in premature stopping of drawing from draw points and therefore a loss of ore. Excessive kimberlite rockmass strength (high UCS and RMR) can inhibit/prevent caving of the material in caving methods. Access tunnels are typically situated in host rock where higher strengths are more common. Time-related and stress-related damage to tunnels situated within the kimberlite ore body can occur and may require substantial support precautions, while alternative development methods (boring) could be investigated to reduce development time and therefor exposure to higher stress levels.
Diamond fissures	Rock strength of the host rock types varies with the material types that could consist of granite, shale, quartzite or lava. The strength of the ore body rockmass could vary due to intact strengths and/or joint properties within the ore body.	Placement of access tunnels in some materials is problematic especially in shale (scaling) and lava (strain bursting). Caving potential of the kimberlite material varies and could prevent caving methods from being implemented, requiring access into the stopes and therefore the need for in-stope support.

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Environment	Geotechnical characteristics	Impact on designs for total or partial extraction mining methods
Gold	Multi-reef mining is common due to the large number of reefs and is found in both flat and steep dipping areas. Generally good ground conditions in areas with no stress fracturing (shallow depth), widely spaced joints and strong quartzite. However, when at depth, stress fracturing reduces the rockmass strength.	The impact of various reefs on each other should be considered in determining mining method design. De-stressing at depth is to be considered when attempting to apply massive mining methods. Span control is required as stress fracturing can add a substantial impact on stable open spans and affect final mining layouts. The use of backfill should be considered to assist with span control, but also ventilation at depth and seismic vibration control.
Platinum	Rock material strengths are mostly relatively high, except for the pyroxenite in some areas, where lower strengths have been reported. Rockmass quality is mostly a function of the joint sets found in these rock types, but is generally high. The quality varies across the material types and across the mining areas and does not allow the simple listing of characteristics. Joint spacings vary, but mostly occur in at least three sets and at variable dips (although mostly steep) and dip directions. Joint infilling is often serpentinised increased potential for instability on joints, while other fillings, such as calcite and pegmatoid, result in high-strength joints. Structures present in this environment include weakness planes, faults, dykes (dolerite, diabase, syenite and lamprophyre), domes (cooling/ramp fault structures) and potholes (replacing or thinning of seams, exposing anorthositic material, increased joint density). Significant structural disturbances in some areas increase joint frequency.	Rockmass strength is high due to good material quality, affecting span and support designs, but assisting the potential for seismic activity at depth. High joint frequencies affect the application of massive mining methods and may require the use of backfill. High host rock strengths allow good access tunnel placement and stable accesses.

REFERENCE

- Page 206,221 Cable Bolting in underground mines Hutchinson
- Page 9, Updated cave mining handbook Laubscher

LEARNING OUTCOME 2.3.1.2

CONNECTION 8

Refer to
Paper 3.1, Outcome 3.2.2

DESCRIBE, EXPLAIN, DISCUSS AND APPLY STANDARD ROCKMASS CLASSIFICATION AND ASSESSMENT SYSTEMS TO PREDICT EXCAVATION STABILITY

LEARNING OUTCOME 2.3.1.3

CONNECTION 9

Refer to
Paper 3.1, Outcome 2.3.1.4
Paper 2, Outcome 3.1.2.3
Paper 3.2 Outcome 2.3.1.4

DESCRIBE, EXPLAIN, DISCUSS AND APPLY BARTON'S Q SYSTEM TO CLASSIFY ROCKMASSES

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LEARNING OUTCOME 2.3.1.4

DESCRIBE, EXPLAIN, DISCUSS AND APPLY BIENIAWSKI'S RMR SYSTEM TO CLASSIFY ROCKMASSES

CONNECTION 10

Refer to
Paper 2, Outcome 3.1.2.4
Paper 3.2 Outcome 2.3.1.5

LINK

- Rock mass characterisation for underground mines Milne

LEARNING OUTCOME 2.3.1.5

DESCRIBE, EXPLAIN, DISCUSS AND APPLY LAUBSCHER'S MRMR SYSTEM TO CLASSIFY ROCKMASSES

CONNECTION 11

Refer to
Paper 2, Outcome 3.1.2.5Paper 3.2 Outcome 2.3.1.6

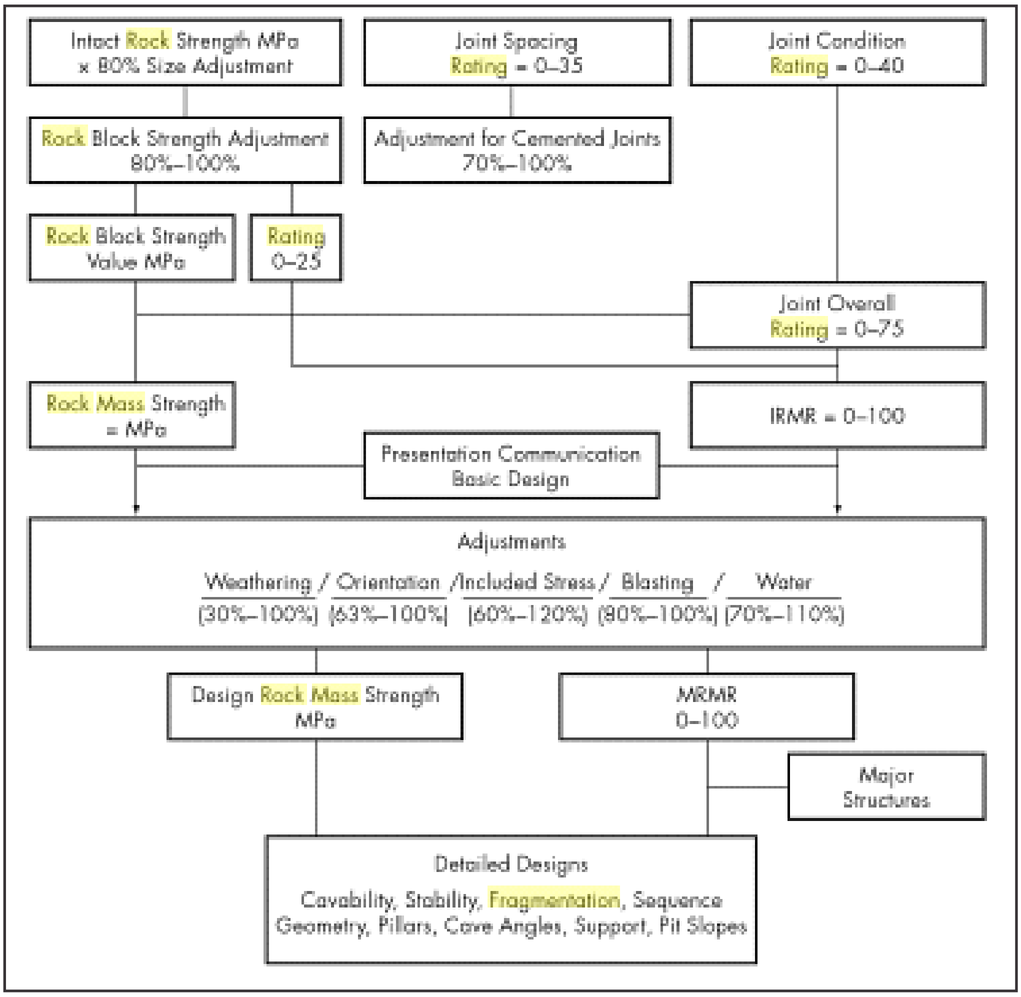

Figure 23: Laubscher's MRMR flow sheet

REFERENCE

- Page 9, Updated cave mining handbook Laubscher

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LEARNING OUTCOME 2.3.1.6

APPLY ROCKMASS CLASSIFICATION RESULTS TO DETERMINE THE STABILITY OF MINING EXCAVATIONS

CONNECTION 12

Refer to
Paper 3.1, Outcome 2.3.1
Paper 2, Outcome 3.1.2
Paper 3.2 Outcome 2.3.1

EXAMPLE

Stable span design

The design process is based on the understanding that open (supported or unsupported) mining excavations will be created. In the open stopes, one of or a combination of the ore body, hangingwall or footwall material could collapse. By drafting the relationship between the rockmass quality and mining span, the critical unsupported, stable span for a specific rockmass quality can be determined. Potvin has provided a relationship and has indicated the zones of stability, instability and transitions areas, based on the data sample used at the time. Potvin utilised a rockmass quality parameter N', called the Modified Stability Number and the Hydraulic Radius concept (stope dimensions) to determine the stability of mining spans.

The Modified Stability Number N' is determined by multiplying the Q' parameter with the Potvin factors a, b and c. The Q' parameter is taken from the rockmass quality Index Q, by replacing the 'stress'-related input parameters in the Q Index with the Potvin factors. Since the Q Index is determined from

$Q=RQD/J_n \,.J_r/J_a \,.J_w/SRF$,

the J_w/SRF ratio is removed to provide:

$Q'=RQD/J_n \,.J_r/J_a$

And $N'=a.b.c.Q'$

The stability Number N is related to Hydraulic radius through the empirical relationship of Potvin and provides maximum mining spans for unsupported stopes.

Stability number

The selection of the a, b and c factors to calculate the stability number N' was based on the following:

1. A factor: The A factor accounts for the influence of high stresses that reduce rockmass stability and is determined by the ratio of unconfined compressive strength of intact rock to maximum induced stress parallel to the opening surface. A is given by: A = 1.125R – 0.125, where 1 > A > 0.1 and where R is the ratio of the uniaxial compressive strength of the rock material to the maximum induced compressive stress, estimated by applying the calculated virgin stress levels.

Table 7: Calculation of A Factor

UCS of material		Stress		UCS / stress Ratio		A	
Hangingwall	161.87	2.78	12.53	58.15	12.92	65.29	14.41
Orebody	269.95	2.78	12.53	96.97	21.55	108.97	24.12
Footwall	161.87	2.78	12.53	58.15	12.92	65.29	14.41

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2. The B factor looks at the influence of the orientation of discontinuities with respect to the surface analysed. Discontinuity orientations have been identified in section 4.1.1 and those dipping away or towards the hangingwall and footwall or the 'roof' of the ore body were utilised to determine the B Factors. The factors for the hangingwall and footwall was selected as similar, while the ore body factor was higher due to more favourable orientations, applying the lowest factor in each case.

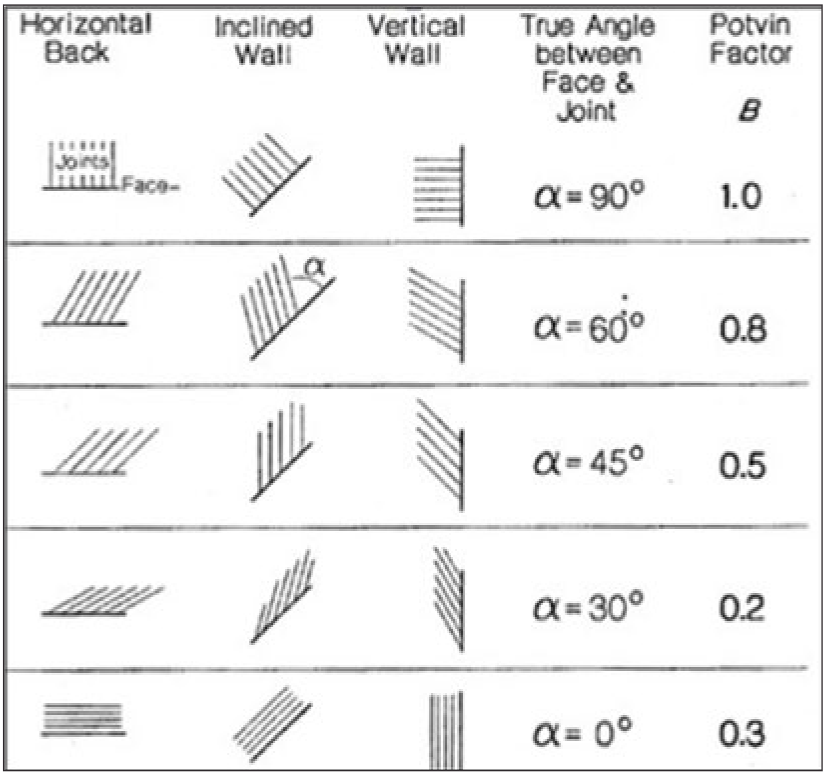

Figure 24: Potvin B Factor

Table 8: B Factor determination

	Estimated general joint orientations		Hangingwall (Back / Horizontal face)		Angle between HW and joint	B
	Dip	Dip direction	Dip	Dip direction		
Host rock	72° - 85°	160°	47° - 55°	298° - 336°	40° - 63°	0.4 – 0.85
Ore			0	NA	72° - 85°	0.9 – 0.95

3. The C factor considers the orientation of the surface being analysed and reflects the inherently more stable nature of vertical walls compared to a horizontal back. For the variation in ore body dips, the lowest C Factor was applied for the flattest dips.

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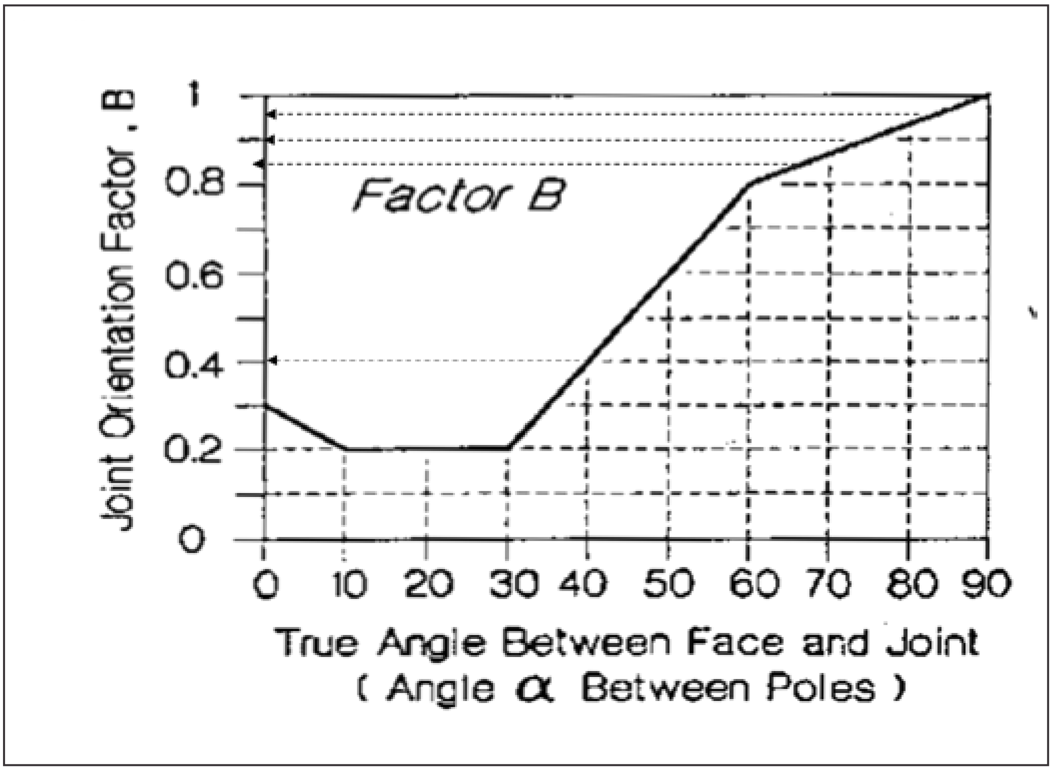

Figure 25: Potvin C Factor

Table 9: C Factor calculation

Gravity	
Ore dip	C
40°	3.40
50°	4.14
63°	5.28

Applying the Q index ratings determined during the logging and data analysis phase, the Q' value and Stability Number N' could be calculated and are provided.

Table 10: Ore body Q' and N results

BC	Hangingwall			Footwall		
	Q	Q'	N	Q	Q'	N
3.0m	7.03	10.66	14.49	6.64	10.06	13.69
6.0m	7.12	10.80	14.68	6.48	9.81	13.35
9.0m	7.02	10.63	14.46	6.65	10.08	13.71

Hydraulic radius

The Hydraulic Radius of mining blocks or stopes is calculated from as shown in Figure 26 below and represents the size and shape of a mined-out area.

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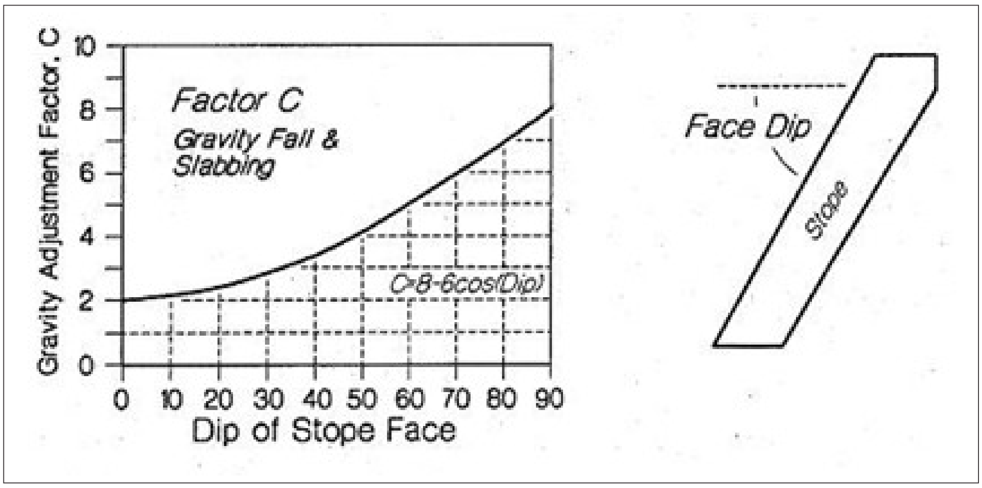

Figure 26: Hydraulic radius definition (Hutchinson et al., 1996)

Potvin related the Hydraulic Radius to Stability Number N' for a number of actual case studies and provided the relationship similar to an updated version presented in Figure 27. Even though this work is empirical and drafted in completely different rockmass environments, it has been used successfully in the past to indicate possible stable stope spans for the SLOS mining method.

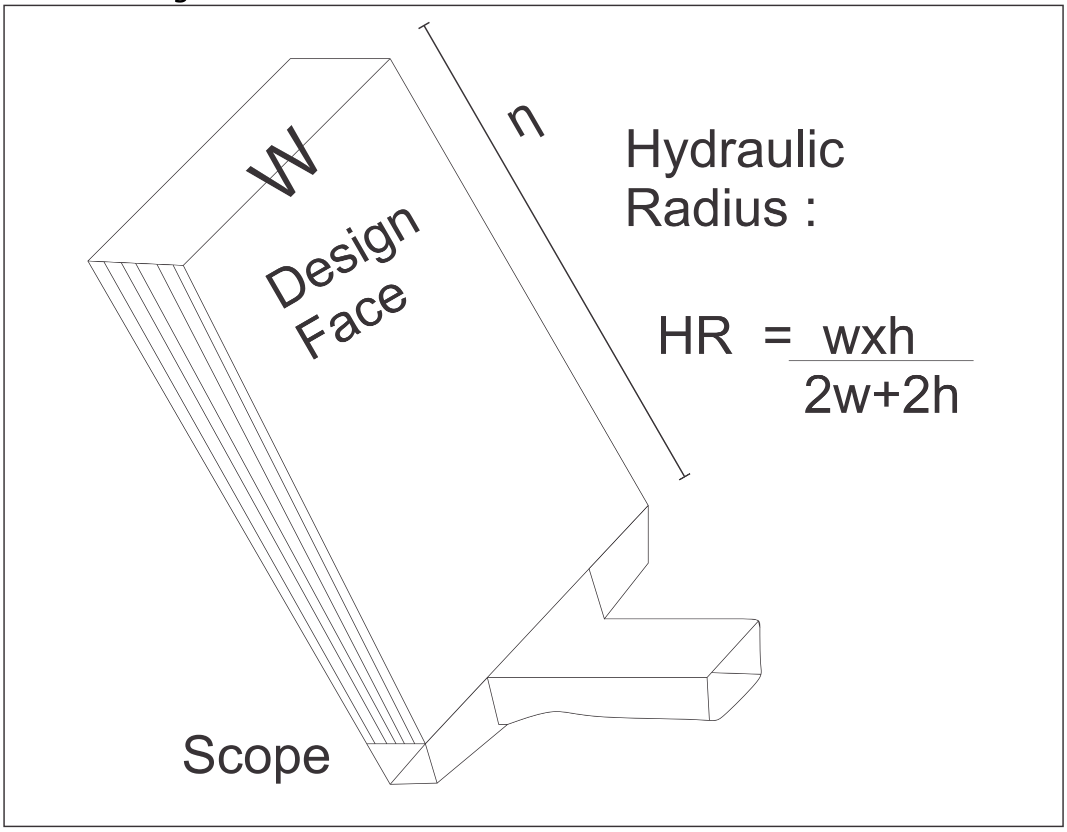

Figure 27: Hydraulic radius and stability number (Hutchinson et al., 1996)

By accepting the line between the caving zone and transition zone as appropriate to the operations, the equation that represents this hydraulic radius and stability number relationship has been derived and takes on the form of Hydraulic radius=$2.3458 \times N^{0.3695}$, where N is the calculated stability number.

Using this equation, the hydraulic radii that would represent the point of unstable equilibrium for the environments have been calculated and are shown below in Table 11.

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Table 11: Hydraulic radii for the ore bodies and host materials

Hydraulic radius - Unsupported							
Ore-body	**Hangingwall**			Ore-body	**Footwall**		
	3m	6m	**9m**		3m	6m	9m
BC	6.28	6.29	6.29	7.92	6.03	6.04	6.20

Unsupported mining spans

If unsupported rock walls are considered for open stopes, the ability of the rockmass to remain stable and not unravel over the created mining spans is essential. It has been shown in previous sections that the material is viewed as very blocky and that joint spacings are generally very small. These conditions normally create high potential for scaling to occur, especially when the ore bodies are relatively flat dipping, as in this case.

In an attempt to evaluate the potential for continued scaling, the data gathered was utilised using the Hydraulic radius results. By changing the vertical stope height (i.e. level spacings), open stope strike spans could be determined using the hydraulic radius concept and are presented in Table 11 and Figure 28. Indications are that to remain below the limiting hydraulic radii reported in Table 10, while applying standard vertical level spacings of 15m, 20m, and 25m, only the options shown in Table 11 appear possible when stope backs remain unsupported. As soon as 25m level spacings are applied, the strike spans of the stopes become impractically small at approximately 19m. These results indicate that open stopes will remain at very small spans if left unsupported and that the mining of single stopes, separated by sill and rib pillars, will be essential to remain stable.

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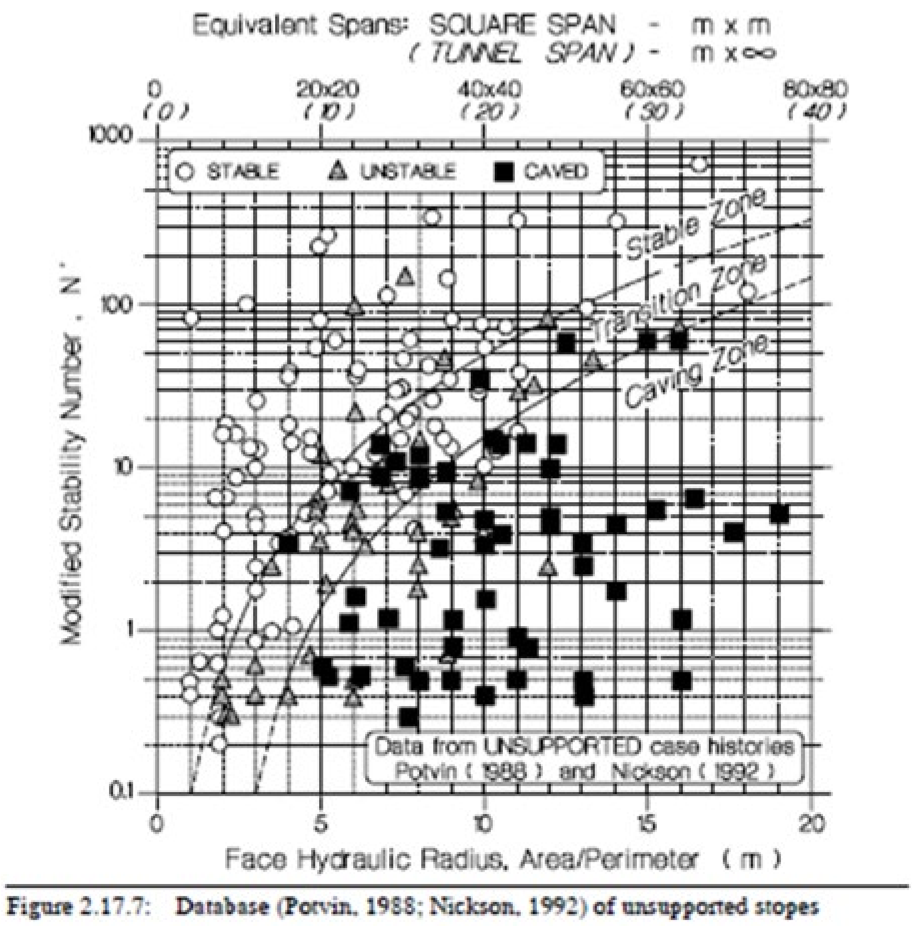

Figure 28: Stope dimensions for both the ore bodies

Table 12: Stope dimensions for ore bodies

	Ore body 1		Ore body 2	
Strike span	32	24	39	25
Level spacing	15	20	15	20
HR	5.11	5.45	5.42	5.56

LINK

- Empirical design of stope openings Brady
- Empirical open stope design Potvin
- Empirical design methods for UG Mines – Ramis Pakalnis
- Empirical design methods – UBC geomechanics
- RMR to design open stopes McGregor

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REFERENCE

- Page 206,221 Cable Bolting in underground mines Hutchinson
- Page 9, Updated cave mining handbook Laubscher

LEARNING OUTCOME 2.3.1.7

APPLY ROCKMASS CLASSIFICATION RESULTS TO DETERMINE THE STABILITY OF UNSUPPORTED SPANS

CONNECTION 13

Refer to
"LEARNING OUTCOME 2.3.1.6" on page 62
Paper 2, Outcome 3.1.2.12
Paper 3.2 Outcome 2.3.1.8

LEARNING OUTCOME 2.3.1.8

APPLY ROCKMASS CLASSIFICATION RESULTS TO DETERMINE THE SUPPORT REQUIREMENTS OF EXCAVATIONS

CONNECTION 14

Refer to
Paper 2, Outcome 3.1.2.14
Paper 3.2 Outcome 2.3.1.10

EXAMPLE

Supported mining spans

The installation of support into the stope backs is a practice that has been implemented with great success in the past. However, design methods available are based on empirical approaches that were developed based on experience on specific mines, where the host rock may be very different to that present at your specific ore body. However, the approaches available have been implemented at this stage to indicate the potential impact of support installation in this environment.

Potvin (Hutchinson *et al*., 1996) indicated that the effectiveness of support installation could be provided by the ratio where the block size is given by the factor . This ratio was calculated for the host rock materials and related to the stable hydraulic radius suggested for the material and is shown in Table 13 and Figure 29. The results indicate that, based on the suggested (Hutchinson *et al*., 1996) ratio of 0.6 as a limit for effective cable support installation, a maximum hydraulic radius of between 9.2 and 10.4 is possible to maintain stability with support installation to a regular pattern, into the stope back.

Table 13: Actual 'block size to hydraulic radius' ratios for both ore bodies

	Ore body 1			**Ore body 2**		
	Block size	HR	Block / HR	Block size	HR	Block / HR
Hang-ingwall	7.11	6.29	1.13	5.54	6.08	0.91
Footwall	6.61	6.09	1.09	6.27	6.53	0.96

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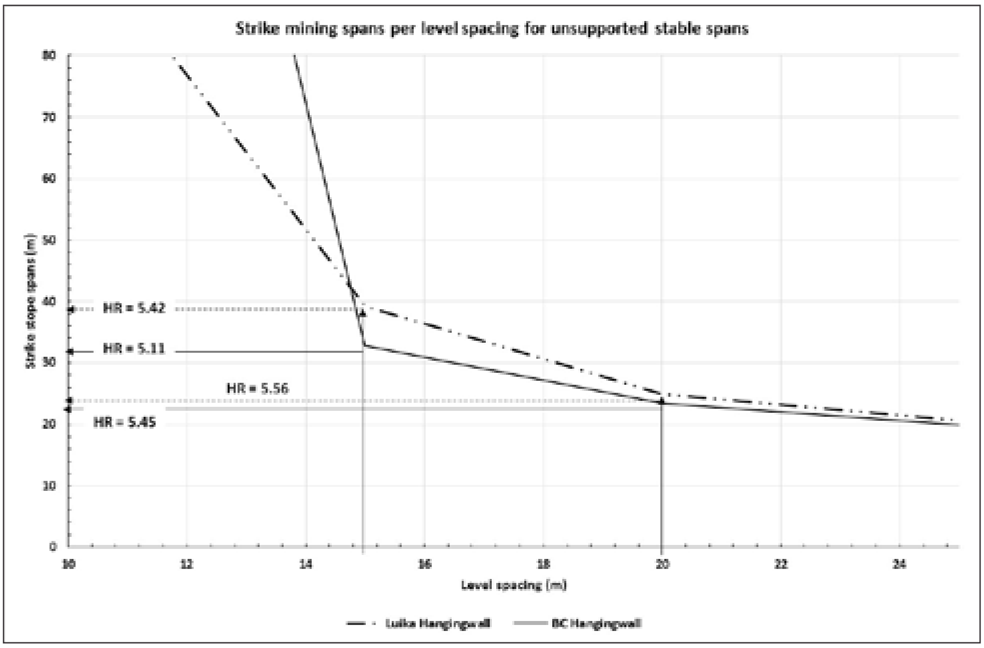

Figure 29: 'Block size and hydraulic radius' ratio

Accepting that support installation could therefore assist in increasing the stope spans, other empirical charts were applied to determine the most appropriate support patterns and potentially required cable anchor lengths. Hutchinson *et al*. (1996) provided several charts derived mostly from work by Potvin and indicated that at the stability numbers suggested for the hangingwalls of the ore bodies, support patterns that vary between 3.0m and 1.0m spacings could allow hydraulic radii of between 8 and 13.5, respectively.

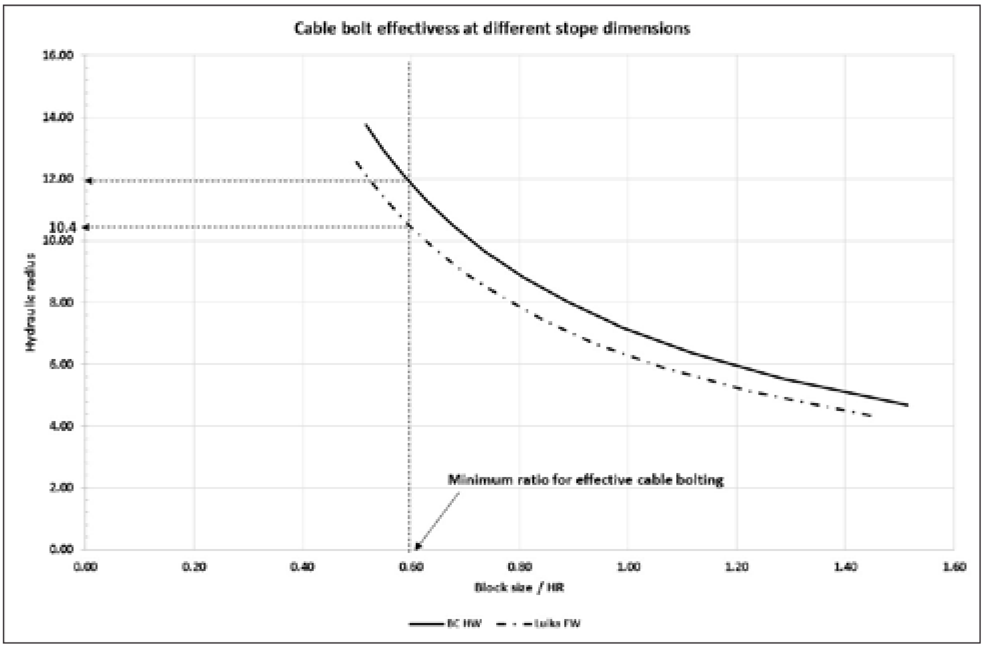

Figure 30: Support spacings impact on hydraulic radius and stable spans (Hutchinson et al.)

Support lengths required are indicated in Figure 31, based on empirical support charts provided by Hutchinson (1996) and indicate the need for 8.0m length cables at a maximum hydraulic radius of 8.0, 11m at a hydraulic of 10 and lengths of between 13 and 15m at the larger radii of 10.4 and 12.0 as suggested as upper limits for the ore bodies.

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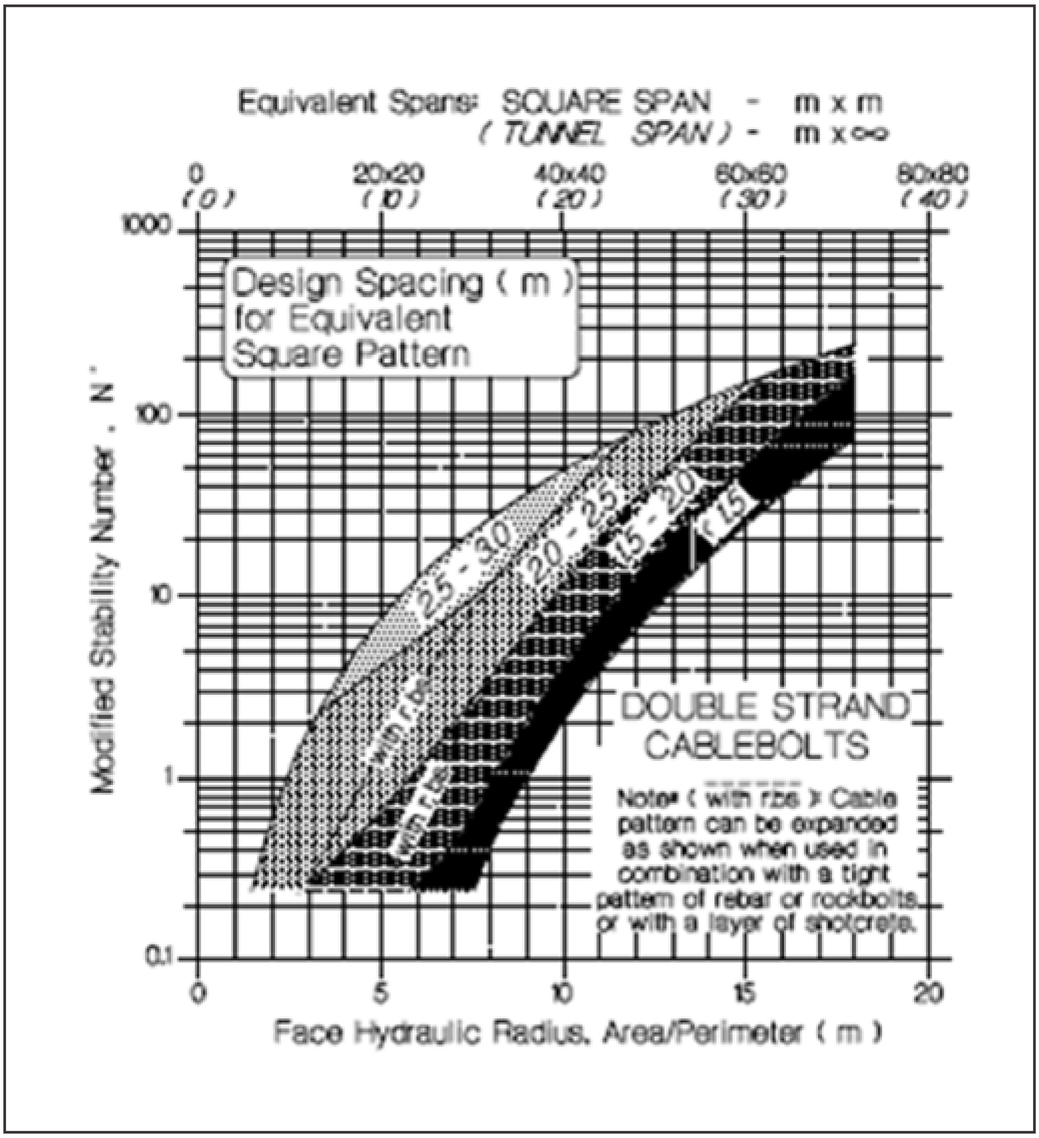

Figure 31: Support length impact on hydraulic radius and stable spans (Hutchinson et al., 1996)

LINK

- Empirical design of stope openings Brady

REFERENCE

- Page 217, 236 Cable bolting in underground mines Hutchinson

LEARNING OUTCOME 2.3.1.9

CONNECTION 15

Refer to
Paper 2, Outcome 3.1.2.12
Paper 3.2 Outcome 2.3.1.8

APPLY ROCKMASS CLASSIFICATION RESULTS TO DETERMINE THE CAVABILITY OF MINING EXCAVATIONS

The cavability of an ore body is a function of:

- rockmass strength (as indicated by the Mining Rock Mass Rating of Laubscher);

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- the structure of the rockmass (geological structures inherent to the material);
- the stress field present within the ore body before and during (induced) mining;
- the size of the ore body to be caved (as represented by the hydraulic radius); and
- the presence of water.

Within all these parameters, it is obvious that caving will be easier if:

- a weak material (UCS) with water present and significant weaknesses (quality), yielding a lower MRMR is present;
- with high levels of tensile stress caused by previous caving or mining; and
- a large ore body to cave/large stope opening.

NOTE

This also suggests that all rockmasses can cave, providing the parameters above are sufficiently present. For example, if the rockmass is strong, a larger undercut/opening would be required to allow caving to occur. The counter argument is that if openings/spans are maintained low enough, caving of a rockmass could be prevented.

Laubscher suggests that two types of caving occur, namely:

- Stress caving: When the tensile stress induced in the rockmass exceeds the tensile strength of the rockmass.
- Subsidence caving: When adjacent caving reduces the lateral restraint on the rockmass, reducing confinement and thereby reducing its strength or ability to withstand tensile stresses.

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PAPER 3.3: CHAPTER 2

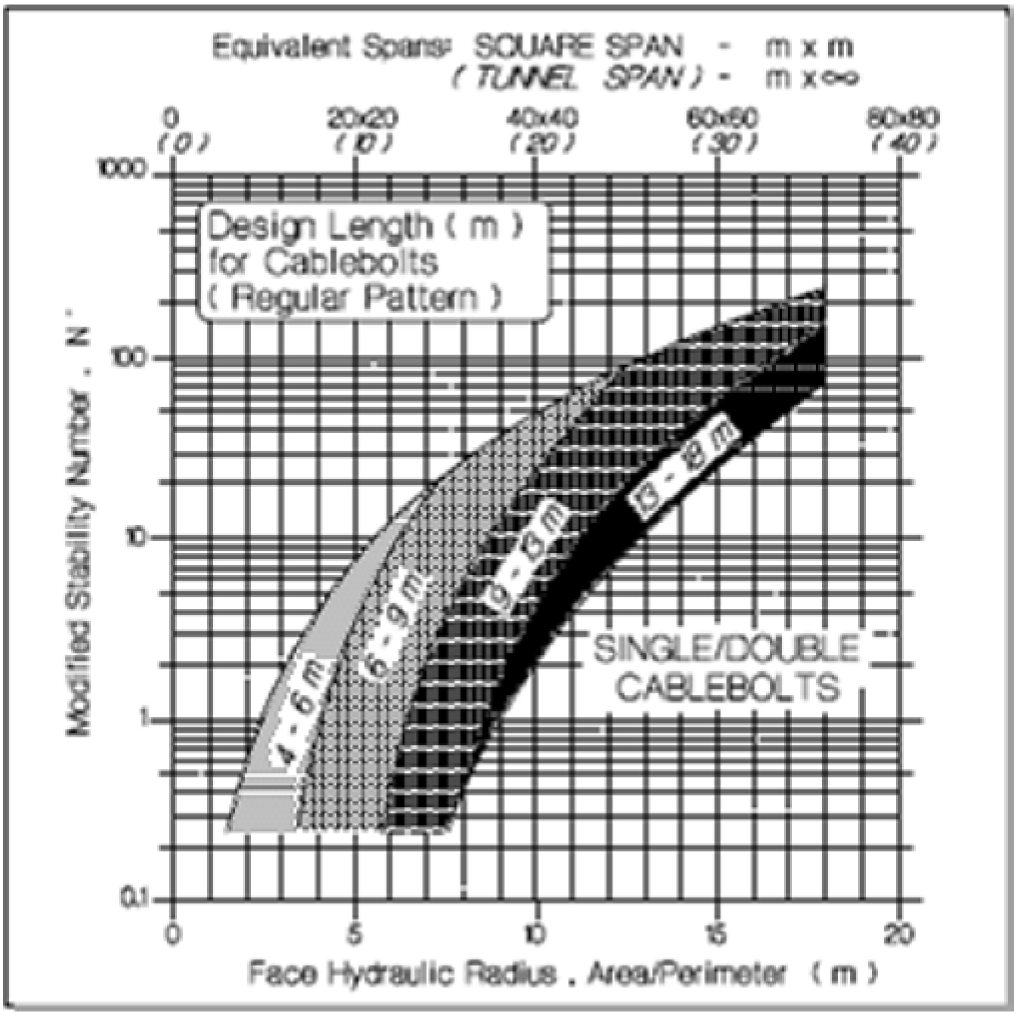

Figure 32: Stability diagram using MRMR and span for various mines (Laubscher)

NOTE

Refer to Figure 32.

Stable = Only local support is required
Transitional = Supportable in upper band, arching in middle band, intermittent caving/arching in lower band depending on outside influences such as water/blasting.
Caving = Progressive caving of back or walls.

LINK

- Empirical design of stope openings Brady
- State of the art cave mining Laubscher

REFERENCE

- Page 16 Updated cave mining handbook

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LEARNING OUTCOME 2.3.1.10

APPLY ROCKMASS CLASSIFICATION RESULTS TO DETERMINE THE FRAGMENTATION OF CAVED MATERIAL

Fragmentation is suggested to be a function of:

- rockmass strength (as given by MRMR);
- geological structures present in the rockmass;
- structure spacings;
- structure contact surface conditions;
- caving type (stress/subsidence); and
- induced stresses.

NOTE

Fragmentation is often classified as primary (size of blocks separating during caving and on entering the draw column) or secondary (size as the rock moves through the draw column).

Fragmentation caused by stress caving is often smaller than subsidence caving due to:

- *slow process during stress caving results in small portions separating from the rockmass as the tensile strength of the material is overcome and shear stresses are induced due to rockmass movements;*
- *subsidence caving can be a faster process that quickly reduces the strength of structures when the confinement is removed allowing separation on structures rather than shear processes.*

LINK

- State of the art cave mining Laubscher
- Block size measurements Palmstrom

REFERENCE

- Pages 39, 42 Updated cave mining handbook Laubscher

LEARNING OUTCOME 2.3.1.11

CONNECTION 16

Refer to
Paper 3.1, Outcome 2.3.1.1
Paper 2, Outcome 3.1.2.11

DESCRIBE, DISCUSS AND APPLY ROCKMASS CLASSIFICATION TECHNIQUES FOR THE SELECTION OF MASSIVE MINING METHODS

LINK

- Mining method selection Turner
- Guides to Underground Mining Atlas Copco
- AHP process of selecting underground mining method Gupta

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INTERESTING INFO

The UBS mining method selection process is automated using the website below. http://www.edumine.com/xtoolkit/xmethod/miningmethodgraphic.htm

Table 14: Selection of mining methods (Courtesy of Goldfields Training Academy)

Steep dip	Shallow	Narrow	High Pay	Low Tech High Tech	Sub-level stoping. Cut and Fill Vertical crater retreat mining
			Low Pay	Low Tech High Tech	As above but more selective mining As above but more selective mining
		Wide	High Pay	Low Tech High Tech	Semi,--Full Shrinkage or open stoping Nil
			Low Pay	Low Tech High Tech	As above but more selective mining Nil
	Deep	Narrow	High Pay	Low Tech High Tech	Sub-level stoping, Room and Pillar, Cement Backfill VCR with Cement Backfill TM3, Hydro Power
			Low Pay	Low Tech High Tech	As above but more selective mining Vertical crater retreat mining, Cement Backfill, HP
		Wide	High Pay	Low Tech High Tech	Open Stope, Cement Backfill, CCT/Aggregate Fill,HP Nil
			Low Pay	Low Tech High Tech	As above but more selective mining HP

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Flat dip	Shallow	Narrow	High Pay	Low Tech High Tech	Sub-level Stoping VCR with Cement Back-fill, TM3, Hydro Power
			Low Pay	Low Tech High Tech	As above but more selective mining As above but more selective mining
		Wide	High Pay	Low Tech High Tech	Face/Strike Scrapers, Up-dip mining TM3 > 1.5m, Throw Blasting
			Low Pay	Low Tech High Tech	As above but more selective mining As above but more selective mining
	Deep	Narrow	High Pay	Low Tech High Tech	Sub-level stoping, Room and Pillar, Cement Backfill VCR with Cement Back-fill, TM3, cct/ Agg, HP
			Low Pay	Low Tech High Tech	As above but more selective mining As above but more selective mining
		Wide	High Pay	Low Tech High Tech	Open Stope, Cement Back-fill, CCT/Aggregate Fill,HP As above
			Low Pay	Low Tech High Tech	As above but more selective mining As above but more selective mining

LEARNING OUTCOME 2.3.1.12

DETERMINE, DISCUSS AND APPLY ROCKMASS 'M' AND 'S' PARAMETERS FOR THE HOEK AND BROWN CRITERION BASED UPON ROCKMASS CLASSIFICATION RESULTS

CONNECTION 17

Refer to
Paper 1, Outcome 6.2.2
Paper 3.2 Outcome 2.3.1.11

LEARNING OUTCOME 2.3.1.13

DETERMINE, DISCUSS AND APPLY ROCKMASS DEFORMABILITY FROM JOINT STIFFNESS AND ROCKMASS CLASSIFICATION RESULTS

CONNECTION 18

Refer to
Paper 3.2, Outcome 2.3.1.12
Paper 2, Outcome 3.1.2.16
Paper 1, Outcome 6.2.5

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LEARNING OUTCOME 2.3.1.14

DESCRIBE, DISCUSS AND APPLY THE ROCKWALL CONDITION FACTOR (RCF) TO PREDICT TUNNEL STABILITY AND SUPPORT REQUIREMENTS.

CONNECTION 19

Refer to
Paper 3.2, Outcome 2.3.1.3
Paper 3.1, Outcome 3.2.16

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CHAPTER
3

PAPER 3.3 : MASSIVE UNDERGROUND MINING (HARD AND SOFT ROCK)

LEARNING OUTCOMES

3. ROCK AND ROCKMASS BEHAVIOUR
3.1. ROCKMASS BEHAVIOUR AROUND MASSIVE STOPING EXCAVATIONS

3.1.1. GENERAL CONSIDERATIONS

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

- Describe, discuss and explain the nature of fracturing around massive mining excavations in low, moderate and high stress;
- Sketch, describe, discuss and explain how excavation shape and size affect stress distribution and failure of the surrounding rock;
- Sketch, describe, discuss and explain how excavation shapes may be optimised to suit the stress environment;
- Describe, discuss and explain the factors that affect the stability of blocks of rock that are defined by geological structures;
- Describe, discuss and explain the factors that affect the stability of blocks of rock that are defined by joints;
- Sketch, describe, discuss and explain how excavation shapes may be optimised to suit the geological environment;
- Determine and characterise the stability of massive stoping excavations using the various rockmass classification systems;
- Estimate stable spans for excavations by making use of rockmass classification methods;
- Estimate excavation support requirements by making use of rockmass classification methods; and
- Evaluate excavation stability for given mining layouts, geological conditions and stress environments.

3.2. OPEN STOPING OPERATIONS
3.2.1. FRAGMENTATION OF BLASTED ORE

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

- Describe, discuss, explain and apply standard rockmass classification techniques to determine the fragmentation of blasted ore; and

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- Describe, discuss, explain and determine appropriate blasting practice to achieve suitable fragmentation of blasted ore.

3.2.2. FLOW BEHAVIOUR OF BROKEN ORE

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

- Describe, discuss and explain how broken rock flows in unchoked conditions towards a drawpoint;
- Describe, discuss and explain how broken rock flows in choked conditions towards a drawpoint;
- Describe, discuss and explain the draw of fine and coarse materials in the same column and its effect on dilution;
- Describe, discuss and explain the effect of static pillars and temporary hang-ups on the loading of drawpoints;
- Evaluate and estimate likely rock flow patterns for given rock property and drawpoint layout conditions; and
- Evaluate and estimate likely dilution for given rock property and drawpoint layout conditions.

3.2.3. ORE DILUTION

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

- Describe, discuss and explain the geotechnical factors that affect dilution in open stoping operations;
- Estimate likely dilution by using rockmass classification methods for given rock property and excavation layout conditions;
- Determine remedial measures to limit dilution for given rock property and excavation layout conditions; and
- Describe, discuss and explain methods to reduce dilution.

3.3. CAVING OPERATIONS

3.3.1. CAVING AND CAVABILITY

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

- Sketch, describe, discuss and explain the conditions required for successful block caving;
- Sketch, describe, discuss and explain the mechanisms associated with the following types of caving:
  - Subsidence caving, stress caving, chimney caving;
- Determine undercut requirements and dimensions by using rockmass classification methods for given block cave situations; and
- Estimate likely caving behaviour by using rockmass classification methods for given rockmass and undercut conditions.

3.3.2. FRAGMENTATION OF BROKEN ORE

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

- Sketch, describe, discuss and explain the factors that affect fragmentation in cave mining operations; and
- Estimate likely fragmentation by using rockmass classification methods for given rockmass and undercut conditions.

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3.3.3. FLOW BEHAVIOUR OF BROKEN ORE

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

- Sketch, describe, discuss and explain the factors that affect the flow behaviour of broken ore in cave mining operations;
- Sketch, describe, discuss and explain the flow of rock towards drawpoints in sub-level caving layouts;
- Sketch, describe, discuss and explain how drawpoint spacing affects dilution and recovery;
- Sketch, describe, discuss and explain how the amount of rock blasted affects dilution and recovery;
- Sketch, describe, discuss and explain the principles of draw control to limit dilution in block caving operations;
- Describe, discuss, explain, compare and contrast the methods of draw control in panel caving and block caving situations; and
- List, describe and discuss the advantages and disadvantages of each of the above methods.

3.3.4. ORE DILUTION

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

- Sketch, describe, discuss and explain the factors that affect dilution in cave mining operations;
- Sketch, describe, discuss and explain the remedial measures to limit dilution in cave mining operations;
- Describe, discuss and explain the relationship between dilution and recovery for cave mining operations; and
- Describe, discuss and explain the causes of the above relationship.

3.4. ROCKMASS BEHAVIOUR AROUND SERVICE EXCAVATIONS

3.4.1. ROCKMASS BEHAVIOUR AROUND ACCESS AND SERVICE TUNNELS

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

- Describe, explain and discuss how rocks and rockmasses may be classified;
- Describe, explain and discuss how rockmass classification systems may be used to predict tunnel stability at shallow, intermediate and great depths;
- Describe, explain and discuss the nature of fracturing around tunnels at depth;
- Describe, explain and discuss the effects and consequences of fracturing on tunnel size;
- Describe, explain and discuss the effects and consequences of fracturing on tunnel shape;
- Describe, explain and discuss the effects and consequences of fracturing on tunnel stability;
- Describe, explain and discuss the phenomenon of time-dependant fracturing and deterioration;
- Describe, explain and discuss the effects and consequences of time-dependant fracturing and deterioration on tunnel stability;
- Describe, explain and discuss the effects of stress regime and

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geological structure on rock behaviour and tunnel stability;

- Describe, explain and discuss the effect of tunnel size, tunnel shape and tunnel excavation technique on rock behaviour and tunnel stability;
- Describe, explain and discuss the optimisation of tunnel size, shape and orientation to suit geological considerations;
- Describe, explain and discuss the optimisation of tunnel size, shape and orientation to suit field stress considerations;
- Describe, explain and discuss the behaviour of rock surrounding tunnels at depth during seismic events;
- Describe, explain and discuss the siting of tunnels with respect to existing, current and future mining operations;
- Describe, explain and discuss the siting of tunnels with respect to geological stratigraphy and structures;
- Describe how the rockwall condition factor (RCF) is determined;
- Describe and discuss how the rockwall condition factor may be applied to predict tunnel stability and support requirements;
- Apply the rockwall condition factor to predict tunnel stability and support requirements at depth;
- Evaluate given rock conditions and field stresses to predict excavation stability; and
- Evaluate given rock conditions and field stresses to recommend remedial measures to improve excavation stability.

3.4.2. ROCKMASS BEHAVIOUR AROUND SERVICE EXCAVATIONS AND SHAFTS

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

- Describe, explain and discuss how rocks and rockmasses may be classified;
- Describe, explain and discuss how rockmass classification systems may be used to predict service excavation stability at shallow, intermediate and great depths;
- Describe, explain and discuss the nature of fracturing around vertical sinking shafts at depth;
- Describe, explain and discuss the nature of fracturing around inclined sinking shafts at depth;
- Describe, explain and discuss the nature of fracturing around rock passes at depth;
- Describe, explain and discuss the effect of stress fracturing and geological structure on the final shape of vertical and inclined shafts;
- Describe, explain and discuss the effect of stress fracturing and geological structure on the final shape of rock passes;
- Describe, explain and discuss the conditions that may lead to the formation of large unstable rock wedges in vertical shaft sidewalls;
- Describe, explain and discuss the conditions that may lead to the formation of instabilities in rock passes;
- Describe, explain and discuss the nature of fracturing around service excavations at depth;
- Describe, explain and discuss the effects and consequences of fracturing on service excavation size;
- Describe, explain and discuss the effects and consequences of

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fracturing on service excavation shape;

- Describe, explain and discuss the effects and consequences of fracturing on service excavation stability;
- Describe, explain and discuss the phenomenon of time-dependant fracturing and deterioration;
- Describe, explain and discuss the effects and consequences of time-dependant fracturing and deterioration on service excavation stability;
- Describe, explain and discuss the effects of stress regime and geological structure on excavation stability;
- Describe, explain and discuss the effects of excavation size, shape and orientation on excavation stability;
- Describe, explain and discuss the effects of excavation technique on fracturing and the stability of service excavations;
- Describe, explain and discuss the effects of excavation sequence on stress fracturing and the stability of service excavations at depth;
- Describe, explain and discuss the effects of support sequence on stress fracturing and the stability of service excavations at depth;
- Describe, explain and discuss the optimisation of service excavation size, shape and orientation to suit geological considerations;
- Describe, explain and discuss the optimisation of service excavation size, shape and orientation to suit field stress considerations;
- Describe, explain and discuss the behaviour of rock surrounding service excavations at depth during seismic events;
- Describe, explain and discuss the siting of service excavations with respect to existing, current and future mining operations;
- Describe, explain and discuss the siting of service excavations with respect to geological stratigraphy and structures;
- Describe, discuss and explain how rockmass characterisation may be applied to predict service excavation stability and support requirements;
- Describe, discuss and explain how rockmass characterisation may be applied to predict service excavation stability and support requirements;
- Apply rockmass characterisation techniques to predict service excavation stability and support requirements;
- Evaluate given rock conditions, field stresses, geological conditions, excavation sizes, shapes and orientations to predict excavation stability;
- Evaluate given rock conditions, field stresses, geological conditions, excavation sizes, shapes and orientations to recommend remedial measures to improve excavation stability;
- Evaluate given rock conditions, field stresses and geological conditions to predict shaft stability; and
- Evaluate given rock conditions, field stresses and geological conditions to recommend remedial measures to improve stability.

3.5. MINE SEISMOLOGY

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

- Define, describe and discuss the term seismic event;
- List, describe, explain and discuss the factors that determine the nature of seismicity in massive mining;
- Describe, explain and discuss the relationship between mine

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layout and seismicity;

- Describe, explain and discuss the relationship between depth and seismicity;
- Describe, explain and discuss the relationship between rock types and seismicity;
- Describe, explain and discuss the relationship between local geology and seismicity;
- Describe, explain and discuss the relationship between regional geology and seismicity;
- List, describe, explain and discuss the measures that may be applied to control seismicity;
- Describe, explain and discuss measures to reduce the maximum magnitude of seismic events;
- Describe, explain and discuss measures to reduce the frequency of seismic events;
- Explain the difference between local magnitude and Richter magnitude;
- Describe and explain the following common methods of seismic data analysis:
    - Gutenberg-Richter analysis,
    - Magnitude-frequency relationship,
    - Energy-moment relationship,
    - Trend analysis;
    - Describe, explain and discuss the relationships between:
    - Event magnitude and damage caused by seismic events,
    - Peak particle velocity and damage caused by seismic events;
- Define and discuss the term 'rockburst';
- Describe, explain, discuss and contrast the relationship between:
    - Rockburst damage and seismic source characteristics,
    - Rockburst damage and excavation support characteristics;
- Describe, explain and discuss the different types of seismic emission sources in massive mining;
- Demonstrate familiarity with the guidelines for the compilation of a code of practice to combat rock-related accidents by being able to:
    - Define, describe and explain the term 'seismically active' as defined in the guidelines,
    - Describe and explain the steps necessary to define emission sources in various ground control districts,
    - Describe and explain the term 'reasonably practicable' within the context of seismic risk management,
    - Describe and explain the various rockburst damage control measures,
    - Describe and explain the use of a seismic system to monitor blasting practice, caving activity and slope stability.

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3. ROCK AND ROCKMASS BEHAVIOUR

3.1. ROCKMASS BEHAVIOUR AROUND MASSIVE STOPING EXCAVATIONS

CONNECTION 1
Paper 3.1, Outcome 3.1; 3.1; 3.3

3.1.1. GENERAL CONSIDERATIONS

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

LEARNING OUTCOME 3.1.1.1

CONNECTION 2
Paper 3.1, Outcome 3.1.1

DESCRIBE, DISCUSS AND EXPLAIN THE NATURE OF FRACTURING AROUND MASSIVE MINING EXCAVATIONS IN LOW, MODERATE AND HIGH STRESS

LEARNING OUTCOME 3.1.1.2

CONNECTION 3
Paper 3.1, Outcome 3.2.10

SKETCH, DESCRIBE, DISCUSS AND EXPLAIN HOW EXCAVATION SHAPE AND SIZE AFFECT STRESS DISTRIBUTION AND FAILURE OF THE SURROUNDING ROCK

LEARNING OUTCOME 3.1.1.3

CONNECTION 4
Paper 3.1, Outcome 3.2.12; 3.3.16

SKETCH, DESCRIBE, DISCUSS AND EXPLAIN HOW EXCAVATION SHAPES MAY BE OPTIMISED TO SUIT THE STRESS ENVIRONMENT

LEARNING OUTCOME 3.1.1.4

CONNECTION 5
Paper 3.1 Outcome 2.1.2
Paper 2 Outcome 2.1.3
Paper 1 Outcome 4.2, 6.1

DESCRIBE, DISCUSS AND EXPLAIN THE FACTORS THAT AFFECT THE STABILITY OF BLOCKS OF ROCK THAT ARE DEFINED BY GEOLOGICAL STRUCTURES

Geological structures include joints, fractures, faults and other geological structures; with joints the most common discontinuity encountered in rockmasses that are often referred to as minor structures/fabric structures. Faults and folds are major yet localised geological structures and therefore are dealt with individually.

The mechanical properties of the rockmass are influenced by the presence of geological structures such as joint sets, since more joint sets provide more potential slide planes for rock wedges or blocks to slide and fall.

NOTE

The properties of geological structures/discontinuities govern the overall behaviour of the rockmasses.

These properties include:

- Persistence: The areal extent or size of a discontinuity, and can

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be crudely quantified by observing the trace lengths of discontinuities on exposed surfaces. The persistence of joint sets controls large-scale sliding

- Orientation: This is described by its dip and dip direction or its dip and strike. The orientation of a major joint set relative to an excavation wall largely controls the possibility of unstable conditions or excessive deformations developing as the mutual orientation of discontinuities will determine the shape of the individual blocks comprising the rockmass.
- The degree of fracturing of a rockmass: The number of joints in a given area where a rockmass with more joints is considered to be 'more fractured', as more joints mean that average spacing between joints is less. The spacing of joints largely controls the size of individual blocks of intact rock that is created and controls the mode of failure.

INTERESTING INFO

A closer joint spacing gives low mass cohesion and a more circular failure (soil-like materials), while large joint spacings indicate intact rock failure or failure on the singular structures. At the same time, the spacing influences the mass permeability (ability to allow water flow).

Joint surface:

A joint surface can be smooth or rough, in good contact or poorly contacted. The condition of contact also determines the aperture of the joint that can be filled with intrusive or weathered materials.

- Roughness: A measure of the inherent surface unevenness and waviness of the discontinuity and therefore influences the shear displacement and shear strength, and therefore also the stability of potentially sliding blocks.
- Contact: In a natural joint, it is very seldom that the two surfaces are in complete contact, but there is usually a gap or an opening between the two surfaces. The perpendicular distance separating the adjacent rock walls is the aperture. An open aperture also indicates high permeability and storage capacity.

Joint surface condition:

- Filling: The joint opening is either filled with air and water (open joint) or with infill materials (filled joint), where open or filled joints with large apertures have a low shear strength. However, the range of physical behaviour depends on the properties of the filling material that affects the shear strength and deformability of the discontinuities.
- When a joint is wet, it has generally a lower friction angle than a dry joint, while a joint subjected to groundwater pressure experiences a reduced normal stress (effective normal stress, i.e. total stress – water pressure).

REFERENCE

- Page 348 Ryder and Jager, A Textbook on rock mechanics, 2002
- Block size measurements Palmstrom

LEARNING OUTCOME 3.1.1.5

CONNECTION 6

Refer to "LEARNING OUTCOME 3.1.1.4" on page 84

DESCRIBE, DISCUSS AND EXPLAIN THE FACTORS THAT AFFECT THE STABILITY OF BLOCKS OF ROCK THAT ARE DEFINED BY JOINTS

LEARNING OUTCOME 3.1.1.6

CONNECTION 7

Paper 3.1, Outcome 3.2.11

SKETCH, DESCRIBE, DISCUSS AND EXPLAIN HOW EXCAVATION SHAPES MAY BE OPTIMISED TO SUIT THE GEOLOGICAL ENVIRONMENT

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LEARNING OUTCOME 3.1.1.7

CONNECTION 8

Refer to
Outcome 2.3.1

DETERMINE AND CHARACTERISE THE STABILITY OF MASSIVE STOPING EXCAVATIONS USING THE VARIOUS ROCKMASS CLASSIFICATION SYSTEMS

LEARNING OUTCOME 3.1.1.8

CONNECTION 9

Refer to
Outcome 2.3.1.7

ESTIMATE STABLE SPANS FOR EXCAVATIONS BY MAKING USE OF ROCKMASS CLASSIFICATION METHODS

LEARNING OUTCOME 3.1.1.9

CONNECTION 10

Refer to
Outcome 2.3.1.8

ESTIMATE EXCAVATION SUPPORT REQUIREMENTS BY MAKING USE OF ROCKMASS CLASSIFICATION METHODS

EXAMPLE

Observations have shown that approximately 3.0m of hangingwall material scales into the stopes and indicated that if the immediate 3.0m of the hangingwall could be supported adequately, scaling could be significantly reduced or even eliminated, especially if mining spans are also reduced based on a lower rockmass quality.

Cable bolts designed for massive stopes can be based on empirical or numerical methods that will allow the assessment of the impact of bolting patterns in overall wall behaviour. To finalise a cable bolt design, it is necessary to determine the following:

- Bolt spacing or density;
- Bolt length;
- Bolt strength;
- Bolt type; and
- Bolt layout (total hangingwall coverage, line bolts installed only).

A significant amount of empirical work has been done by various authors in an attempt to indicate whether cable bolting has been or could be sufficient to assist with hangingwall scaling. The empirical designs are all based on rockmass quality data, as well as proposed stope dimensions and, using these rockmass quality parameters, can indicate the appropriateness of cable bolting and estimated design of cable bolt layouts.

Stope dimension (hydraulic radius) data for stope reporting scaling was plotted on Potvin's (Hutchinson et al., 1995) empirical stable/unstable relationship for stopes that had been cable bolted (Figure 1). The plot indicates that some of the stopes fall within the unstable zone and that scaling would probably occur in these stopes even if cable bolting had been installed. At the same time, it suggests that cable bolting could have assisted other stopes (approximately 75% of the stopes where scaling was reported) to some extent to prevent or reduce the scaling that had been reported.

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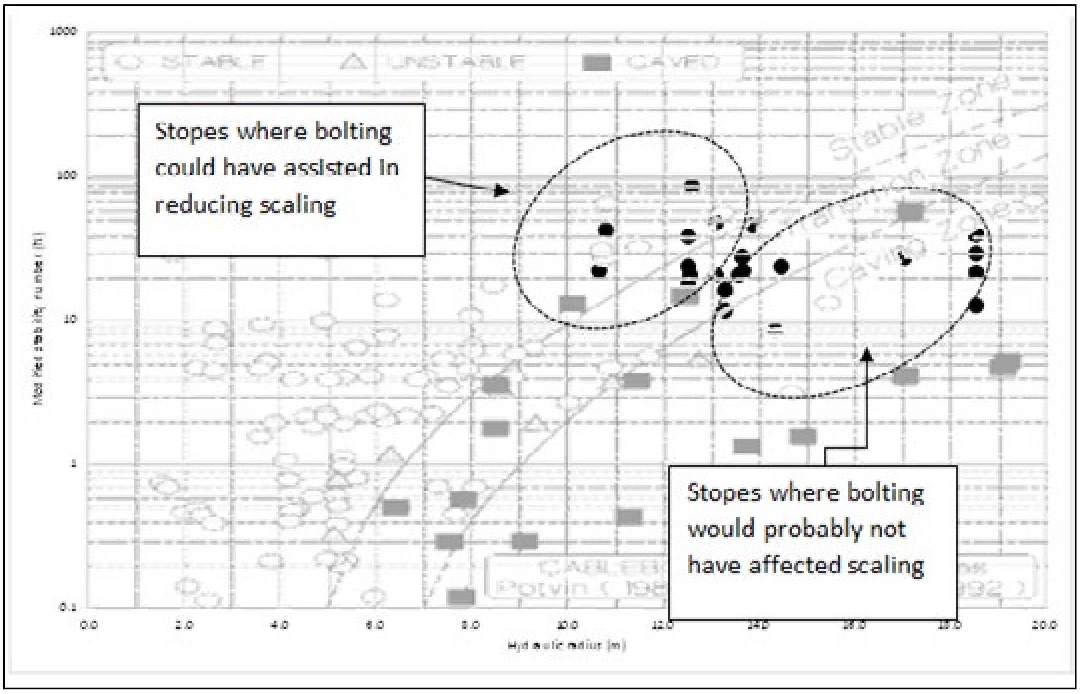

Figure 1: Rockmass quality and stability of stopes that had scaled (Hutchinson et al., 1995)

Plotting planned stope dimensions and expected rockmass quality (from borehole data) on the same empirical relationship (Figure 2), it is clear that this trend continues, namely that a number of stopes would probably experience scaling even if bolting is utilised, but that a large portion of the stopes could potentially benefit from bolting.

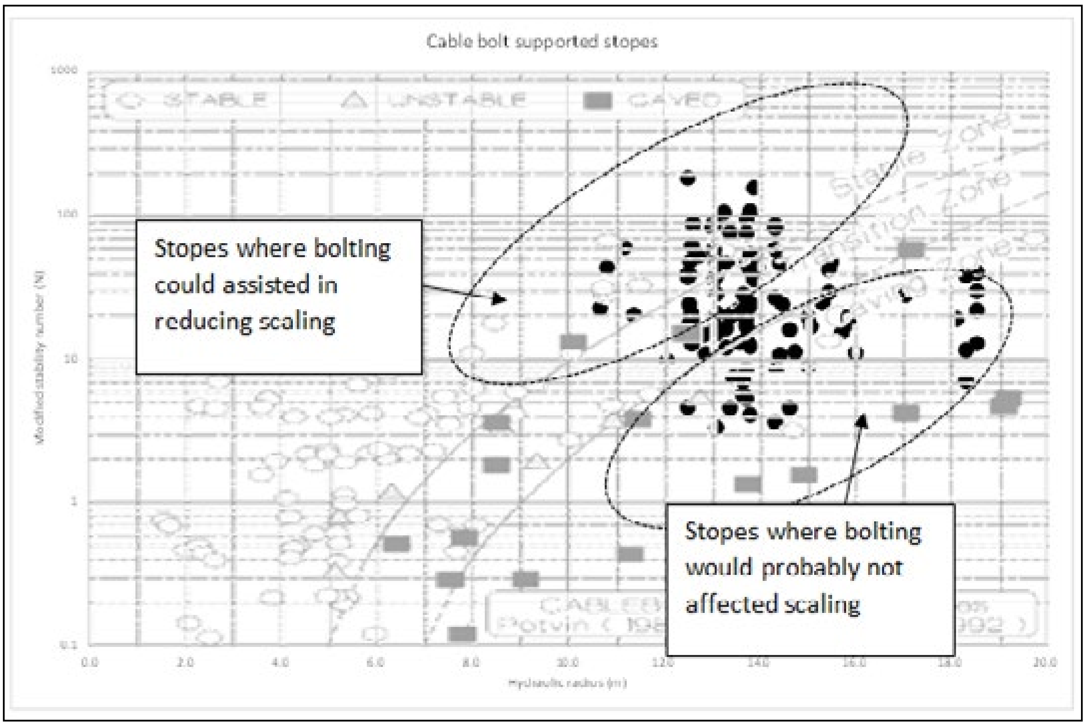

Figure 2: Rockmass quality and potential stability of cable bolt stopes (Hutchinson et al., 1995)

The same data was plotted on a relationship derived by Potvin that indicates the confidence in cable bolting to have any substantial impact on scaling (Figure 3) and indicates that:

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- Only single areas would exist where cable bolting would not be required to assist stability;
- Some stopes are present where high confidence exists that cable bolting would be beneficial;
- Many stopes exist where the confidence is low or where it is suggested that cable bolting would not impact on the scaling.

NOTE

In evaluating these results, it is critical to keep in mind that the Potvin relationships are guidelines only as they have been based on experience in other environments and conditions. However, it is clear that even if cable bolting does assist scaling reduction, some areas do exist where it would be less effective or even totally ineffective. These differences will have to be addressed through trial and error at the specific mine.

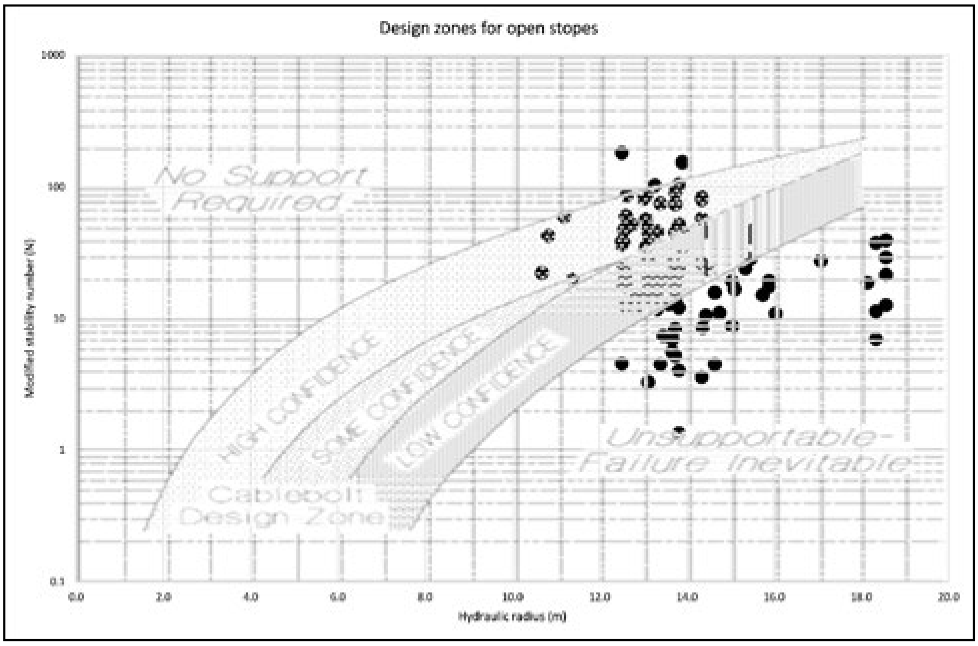

Figure 3: Confidence in impact of cable bolting on scaling (Hutchinson et al., 1995)

Figure 4 relates the cable bolt density (bolts/m2) to the relative block size (RQD/JN), which has been normalised to stope dimensions in terms of hydraulic radius.

NOTE

The smaller the value of (RQD/JN/HR) becomes, the less likely it is that stability can be maintained as the block size becomes smaller and the spans larger. At the same time, a high value would indicate near massive rockmass, i.e. large block sizes, at small spans, and therefore obvious stable conditions.

Within the range of values obtained, it was empirically indicated (based on the Potvin experience) that cable bolting could be appropriate (non-entry zone), but also inappropriate (support ineffective zone) for some of the stopes.

The data (RQD/JN/HR) for stopes already mined was plotted against an estimated bolt density of 0.25 bolts/m2 (4m2 per bolt or 2m x 2m spacing) and indicates that some stopes exist where cable colts would still not have been able to maintain stability at the dimensions plotted, but that for the largest portion of the data points, the environment suggests that cable bolts could have an impact on improving stability. At the same time, an increase in bolt spacings to 3m x 3m would plot the data along the bolt density of 0.1 bolts/m2, which would indicate a high probability

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that the bolt density would be insufficient to control hangingwall scaling.

Confidence in the suggested cable bolt spacings is presented in Figure 4, where it is shown that the rockmass quality given by the stability number N, when normalised by dividing the quality by the stope dimensions, or hydraulic radius, indicates that a 2.0m cable bolt spacing is suggested to fall within the 'conservative' design zone. Any bolt spacing substantially larger than 2.0m will place the data within the 'non-conservative' or 'caving' zones.

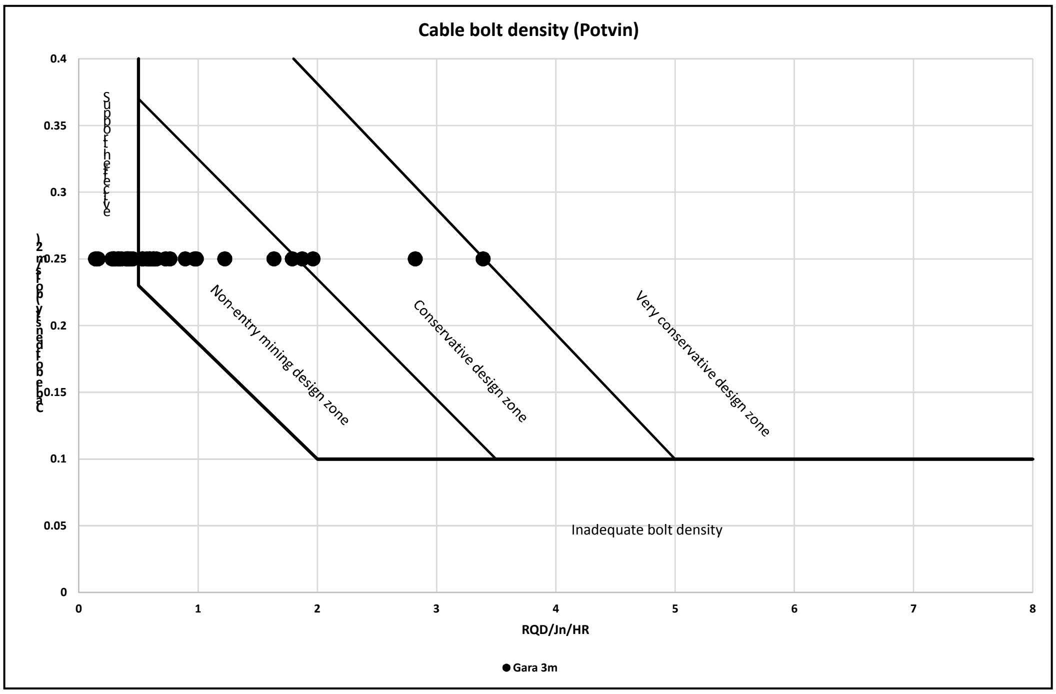

Figure 4: Appropriateness of cable bolting to Gara mine conditions (Hutchinson et al., 1995)

The relationship between rockmass quality and stope dimension was further expanded to indicate the spacings at which different bolt strand options should be installed to assist in scaling control when reinforcement of the hangingwall is required. Reinforcement suggests that the hangingwall material is inherently strong and competent, but that cable bolts are required to build stable beams across the joint affected stope back. These strands are related to strength in the following manner:

- Single strand: 20-25 tonnes or 200-250kN and
- Double strand: 40-50 tonnes or 400-500kN (Hutchinson & Diederichs, 1995)

Figure 5 indicates that single strand units required will have to be installed at spacings at and below 1.0m for most of the stopes, while a number would be effective at spacings of 1.5m to 2.0m.

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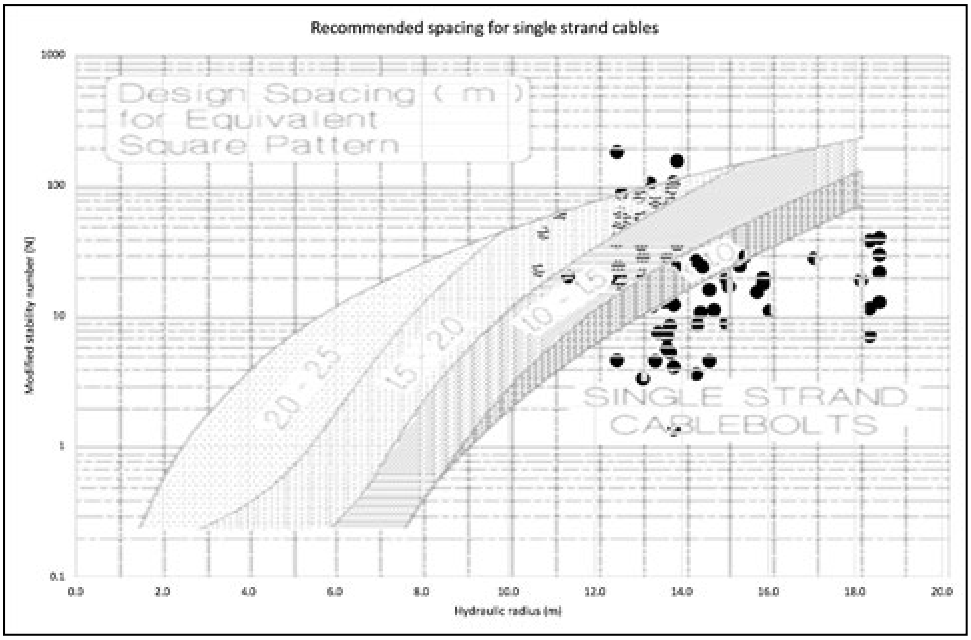

Figure 5: Single strand cable bolt spacing (Hutchinson et al., 1995)

The use of the stronger double strand bolts would allow installations at slightly larger spacings where most of the stopes would seem to suggest spacings of less than 1.5m is required, but where a large portion of stopes indicate that 2.0m spacings are adequate (Figure 6).

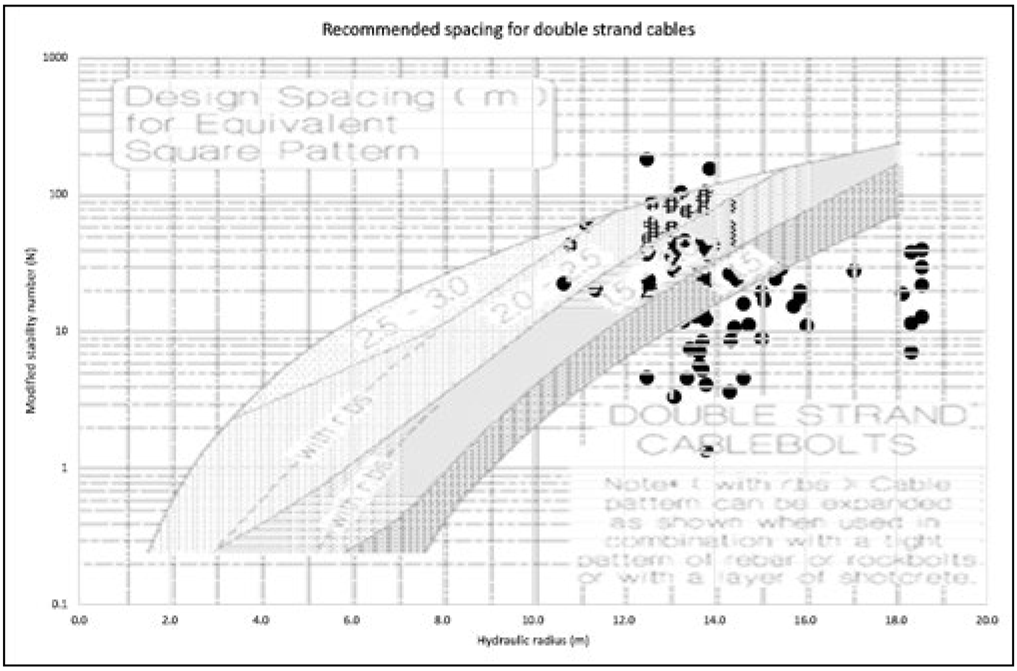

Figure 6: Double strand cable bolt spacing (Hutchinson et al., 1995)

Potvin suggests that the cable bolt length is also a function of the rock-mass quality and stope span and suggests that the relationship in Figure 7 is appropriate. Based on this relationship, bolt lengths of 6m to 18m are suggested, depending on the conditions and spans in each stope, with the greatest portion of the stopes suggesting that 18m long cable bolts are required.

A literature study indicated that, commonly, full column cement grouted cable strands with 'bubbles' or 'bird cages' are utilised when it is expected that confining stress levels will be low such as in a de-stressed open

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PAPER 3.3: CHAPTER 3

stope hangingwall environment. The shear strength of the grouted cable bolt increases when the field stress normal to the bond around the bolt increases. The presence of a bird cage arrangement increases resistance against displacement of the rockmass and therefore provides some confinement in the absence of compressive field stresses.

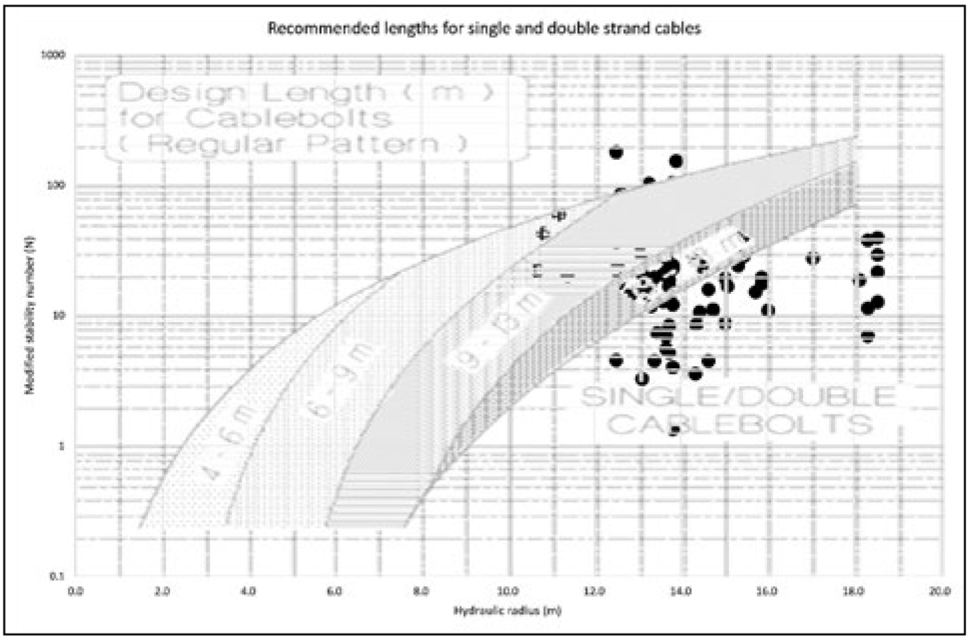

Figure 7: Cable bolt length (Hutchinson et al., 1995)

The relationships utilised above were derived for complete hangingwall coverage with support units. This means that the fully exposed hangingwall would be supported by a system of regularly spaced cable colts. For the installation of such a pattern of bolts, access into the stope hangingwall with tunnel excavations is required. If this is not in place, the design process must consider the applicability of the 'line bolting' layout.

The effectiveness of the 'line bolting' layouts was measured by Nickson and resulted in the relationship indicated in Figure 8, where Nickson correlated the 'unsupported span' to a 'normalised block size'. The hangingwall material block size is given by the ratio of RQD and JN, as derived by Barton in the Q Index formulation.

NOTE

The normalisation (division) of this block size by the stope dimension (or hydraulic radius) suggests that small ratios would indicate the existence of small blocks in a large stope and large ratios would indicate large blocks in small stopes. This indicates that 'unsupported spans' between lines of support should become smaller as the 'normalised block size' reduces.

This support method is applicable to an environment where the controlling structures in the hangingwall are parallel to the hangingwall shape.

IMPORTANT

Nickson suggested that, since this relationship is based on limited data points, it must be utilised with great care as the application of this design method may not be appropriate. For this reason, it is presented as a guideline only.

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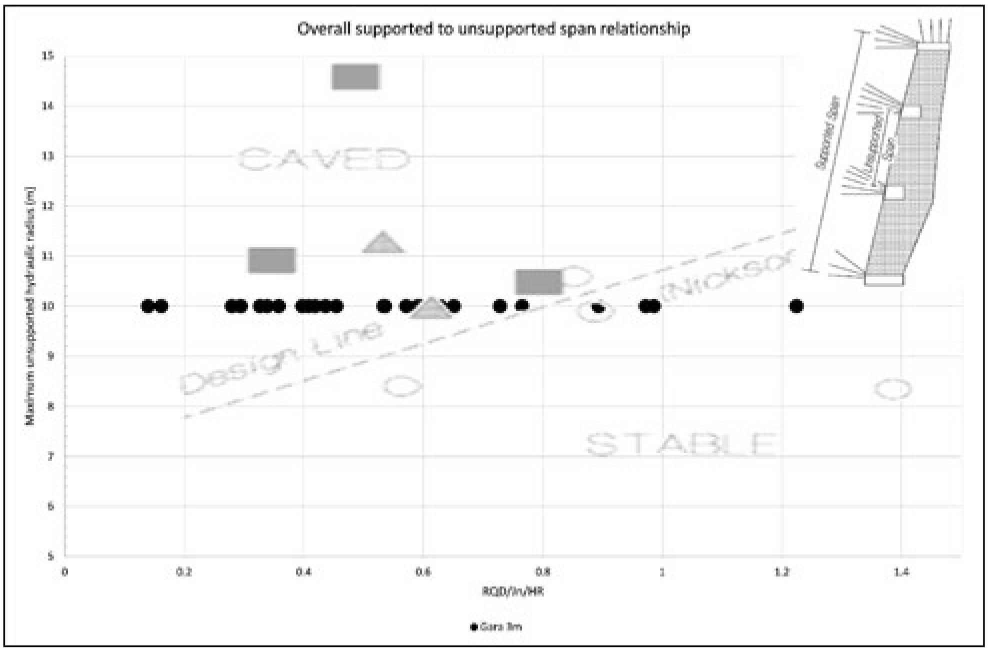

Figure 8: Line bolting potential (Hutchinson et al., 1995)

The application of the empirical methods can provide guidelines to the implementation of cable bolting in stope hangingwalls. From these empirical methods, proper guidelines could be drafted regarding the most appropriate bolt types, spacings and cable bolt strengths.

However, numerical modelling of the stopes could assist in the evaluation of the guideline precautions derived via the empirical methods. It must be noted that the models aim to test the impact of different cable bolt layouts to assist with the selection of the most appropriate support units.

NOTE

Phase 2 (Rocscience), a finite element software code, is able to suggest inelastic displacements that might arise from movement on joints/planes (as long as those remain relatively small). Since it is a two-dimensional code, it was assumed that the impact suggested by the code can be applied along the full strike length of the stopes.

The models created in Phase2 are shown in Figures 10 and 11 and included the following:

- Stopes were created between three sub-levels to result in a dip stope span that is in the region of 50m vertical and to coincide with the reports on scaling that greatly increased when the second level stope was extracted. The models included stopes at 50° and 70° ore body dips;
- A stope parallel to laminated rockmass was created to a depth of 3.0m into the stope hangingwall to allow the reported 3.0m maximum thickness of sheared material to scale. The rest of the hangingwall is suggested to be massive as its impact on the scaling is assumed to be minimal (complete hangingwall unravelling was not reported);
- Within the laminated 3.0m beam, joints that dip towards the stope were included to allow some freedom of movement and to correlate with structures reported in the stopes. The joints were opened at excavation edges to allow movement to occur and joint strength (cohesion and friction) properties were reduced to ensure that displacement would occur;
- Support was installed from the drives only and had to be installed along the 'shanty-back' created between the drive and ore body contact (Figure 10), since the model removes cables

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when portions of the cable bolts are mined out at a later stage (Figure 11b). In situ, this complete loss of support is not expected and portions of cable bolts that may remain in the hangingwall after removal of the ore body are still expected to affect the hangingwall behaviour;

- Cable bolts were installed 2.0m apart across and along the drives and included models with no support, while the supported stopes included 6m, 9m and 13m cable bolt lengths. The impact of a reduction in the number of cable bolts was also tested for a limited number of options and included a reduction in cable bolts installed in the centre drive, from five to three units per ring;

Cable bolts were fully grouted, and bird cages were placed near both ends of the bolts to increase fractional resistance, while bolt strengths of 20 tonnes were utilised, purely for comparison reasons.

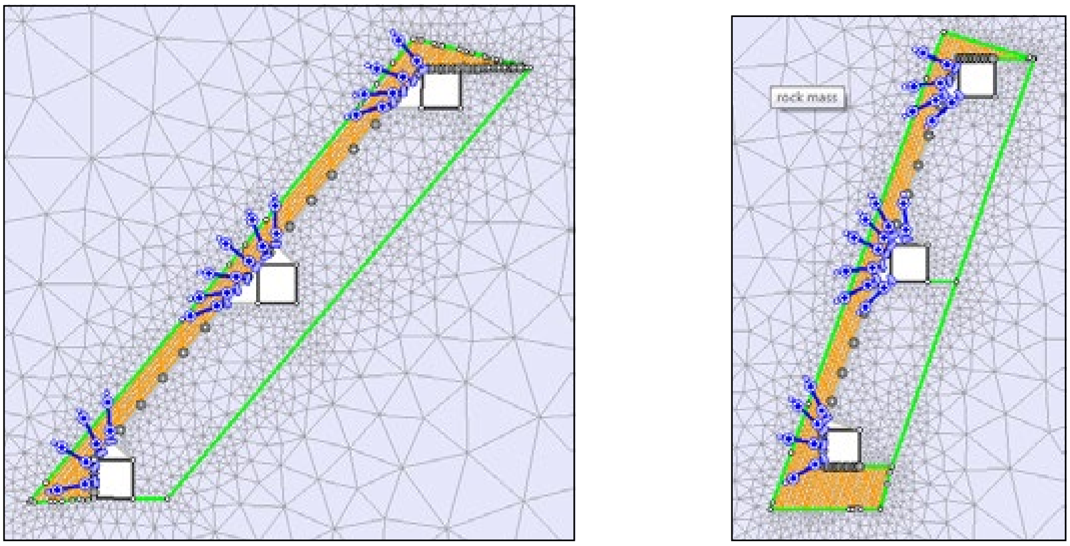

Figure 10: Model layout for (a) 50° and (b) 70° ore body dip (installation stage)

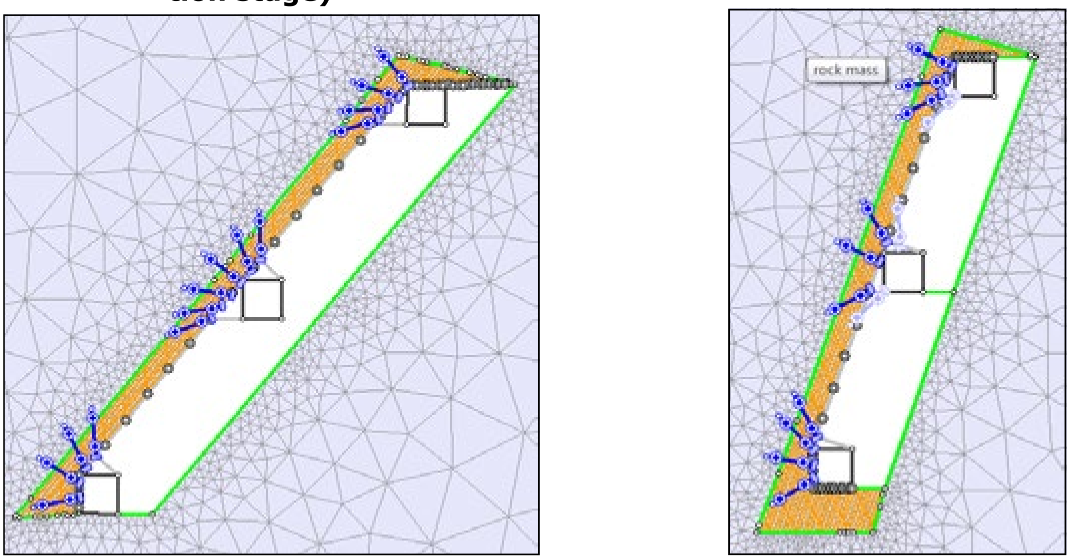

Figure 11: Model layout for (a) 50° and (b) 70° ore body dip (mining stage)

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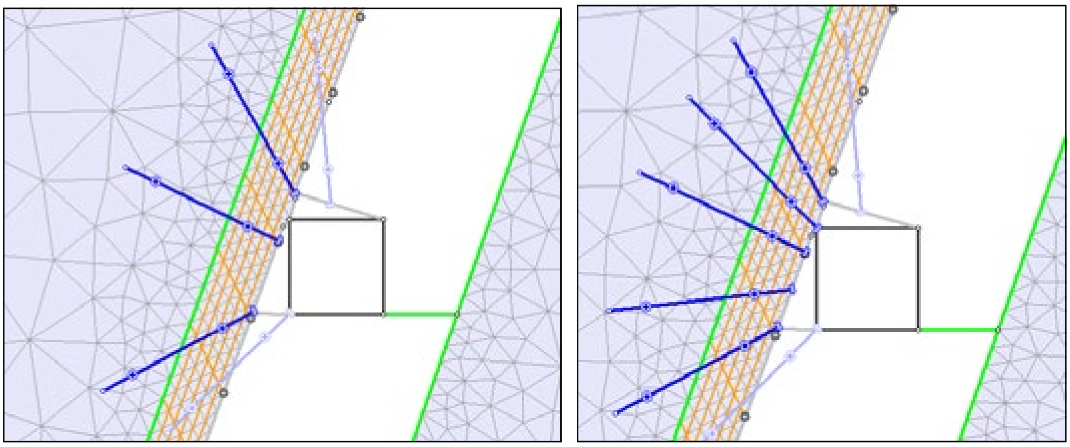

Figure 12: Number of cable bolts reduced in centre drive (a) 3 and (b) 5

Results obtained from Phase 2 and plotted in Excel allowed comparison of the different options in graphical form. These results are provided below in Figure 13 to Figure 15, and allow the following comments:

- The potential impact of cable bolting on hangingwall displacement is clear when the maximum displacements suggested for unsupported stopes are compared to any of the supported options;
- Only a small advantage can be liberated when installing cable bolts of 9m length compared to 6m, while no advantage is to be gained with a further increase to 13m cable bolt lengths;
- This reduction in displacement occurs in both the upper and the lower stope hangingwalls and suggests that cable bolting could greatly reduce displacement when the second stope is mined;
- It is also clear that the reduction in ore body dip increases the expected displacement substantially and reduces the differences obtained with the different cable bolt lengths;
- At the same time, it is suggested that at shallower dips, 13m long cable bolts start to affect hangingwall displacements;
- The reduction in numbers of cable bolts installed, i.e. at spacings greater than 2.0m, increases the expected hangingwall displacement and increases potential for scaling to continue.

In essence, the numerical modelling has shown that to control the behaviour of the potentially unstable 3.0m hangingwall material, cable bolt lengths in excess of 6.0m appear to be excessive. At the same time, the patterns suggested by the empirical design (2.0m) should be maintained as scaling risk is increased by a reduction in unit numbers, as confirmed by the numerical models.

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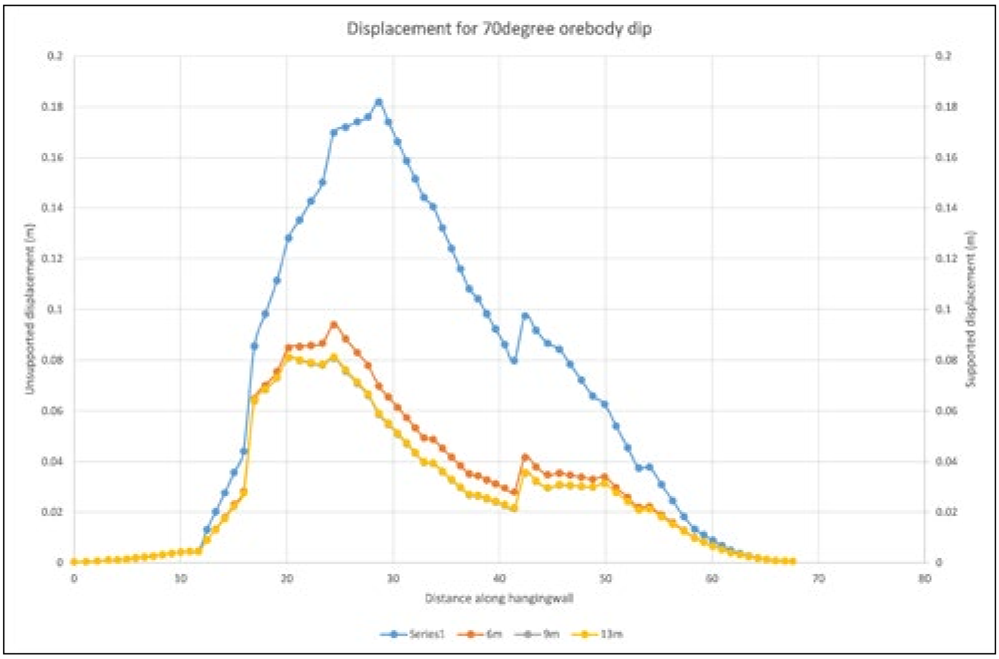

Figure 13: Hangingwall displacement for 70° ore body (5 centre drive bolts)

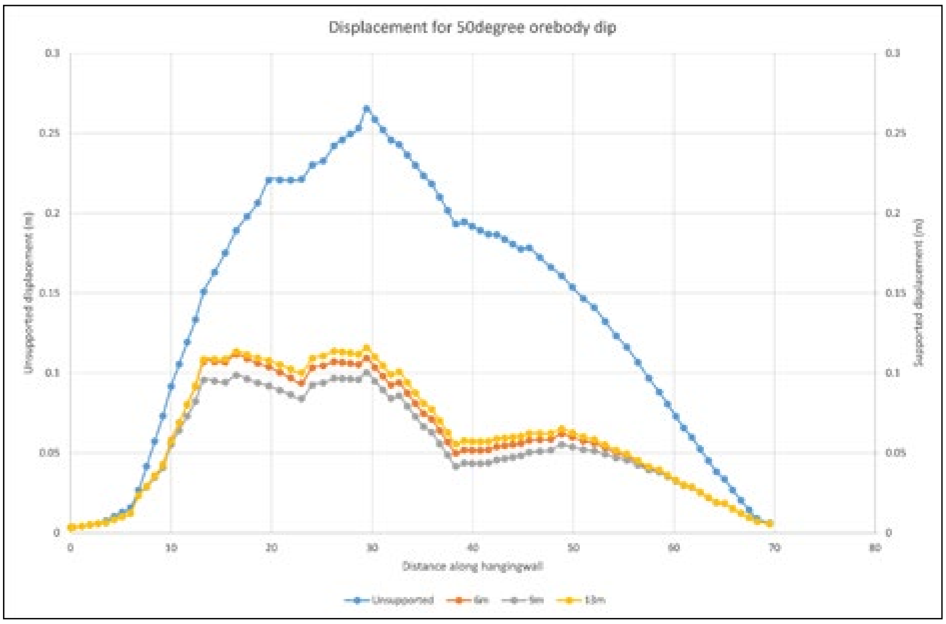

Figure 14: Hangingwall displacement for 50° ore body (5 centre drive bolts)

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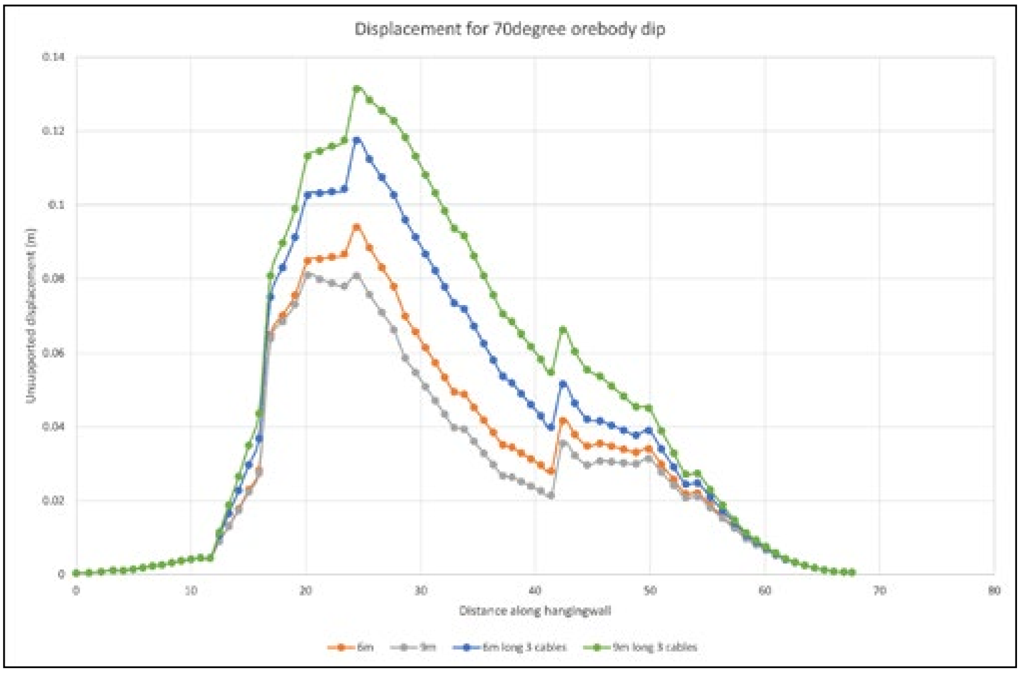

Figure 15: Hangingwall displacement for 70° ore body (3 centre drive bolts)

The results obtained from Phase 2 are not expected to indicate the real level of displacement expected, but only to assist in selecting the most appropriate support units for the initial implementation. The installation should be monitored and its effectiveness recorded, so that the empirical results could be adapted with time. In other words, should scaling continue to occur in supported stopes, the recorded data should be utilised to increase lengths, change bolts strengths or types, or reduce spacings until the system is effective.

REFERENCE

- Chapter 63, Rock bolting for underground excavations, Underground Mining Methods, SME, 2001
- Chapter 64, Cable bolting, Underground Mining Methods, SME, 2001
- Chapter 70, Mining dilution in moderate to narrow width deposits, Underground Mining Methods, SME, 2001

LEARNING OUTCOME 3.1.1.9

CONNECTION 11
Refer to
Paper 3.1 Outcome 3.3.29

EVALUATE EXCAVATION STABILITY FOR GIVEN MINING LAYOUTS, GEOLOGICAL CONDITIONS AND STRESS ENVIRONMENTS.

3.2. OPEN STOPING OPERATIONS
3.2.1. FRAGMENTATION OF BLASTED ORE

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

LEARNING OUTCOME 3.2.1.1

CONNECTION 12
Refer to
Outcome 2.3.1.10

DESCRIBE, DISCUSS, EXPLAIN AND APPLY STANDARD ROCKMASS CLASSIFICATION TECHNIQUES TO DETERMINE THE FRAGMENTATION OF BLASTED ORE

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INTERESTING INFO

There is also a link between rockmass condition, blast fragmentation and mill fragmentation as proposed by the process optimisation and integration methodology. The methodology involves a number of steps, i.e. benchmarking, rock characterisation, measurements, modelling/ simulation and, where required, material tracking. The first site visit establishes current operating practices and allows the collection of rock characterisation and blast fragmentation data as well as process performance information. This is followed by modelling and simulation studies to determine how to best exploit hidden inefficiencies.

A fragmentation analysis can be executed if the following information is available:

- Rockmass quality ratings of the ore body and hangingwall material for twice the ore body height;
- The intact rock strength as provided by the average of the intact sample tests;
- Geological structure and joint set spacings or fracture frequency (the range expected is critical as maximum spacings of 2m and 3m indicate a substantial difference in block volume of 8m3 to 27m3;
- Joint condition ratings where the weaker joints will fail first creating primary blocks that could only later fail along other structures;
- Effect of stress where primary fragmentation from stress caving will create finer fragmentation due to failure of large blocks due to the stress levels;
- Cave front orientation with respect to the joints and stress orientations has a great impact on primary fragmentation;
- Draw rates that are slow may create finer fragmentation, but they have to be fast enough to create gaps to allow caving.

REFERENCE

- Page 39, 42 Updated cave mining handbook Laubscher
- Blasting and fragmentation Esen

LEARNING OUTCOME 3.2.1.2

CONNECTION 13

Refer to "Chapter 7: ROCK BREAKING IN MASSIVE MINES" on page 353

DESCRIBE, DISCUSS, EXPLAIN AND DETERMINE APPROPRIATE BLASTING PRACTICE TO ACHIEVE SUITABLE FRAGMENTATION OF BLASTED ORE

3.2.2. FLOW BEHAVIOUR OF BROKEN ORE

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

LEARNING OUTCOME 3.2.2.1

DESCRIBE, DISCUSS AND EXPLAIN HOW BROKEN ROCK FLOWS IN UNCHOKED CONDITIONS TOWARDS A DRAWPOINT

DEFINITION

Choked conditions: Area through which material will flow is completely choked with broken ore, with the only air gaps at the top of the flowing ore where typically the caving of the ore body is progressing.

Unchoked conditions: Area through which material flows could have voids to one or both sides of the flowing ore and flow is regulated by the broken material angle of repose.

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CONNECTION 14

Refer to "LEARNING OUTCOME 3.2.2.1" on page 97

The flow of ore, while the open stope is choked with broken ore, will exhibit the same behaviour as that discussed in Outcome 3.2.2.2.

Once the broken ore has been drawn to such an extent that voids in the open stope have been created, the remaining ore flows towards the drawpoint based on its angle of repose.

LEARNING OUTCOME 3.2.2.2

DESCRIBE, DISCUSS AND EXPLAIN HOW BROKEN ROCK FLOWS IN CHOKED CONDITIONS TOWARDS A DRAWPOINT

DEFINITION

Choked conditions: Area through which material will flow is completely choked with broken ore, with the only air gaps at the top of the flowing ore where typically caving of the ore body is progressing.

Unchoked conditions: Area through which material flows could have voids to one or both sides of the flowing ore and flow is regulated by the broken material angle of repose.

NOTE

Major issues involving all caving methods are the flow or broken/caved ore, the design of layouts to maximise recovery and minimise dilution, while ensuring that all access excavations remain stable.

Material flow is impacted on by the following:

- Differential fragmentation: Finer fragmented material flows more readily than coarse material with fine material that can easily flow 'through' coarse material. Dilution material fragmentation should be coarser than the ore.
- Compaction: Compacted material does not flow as readily as freshly blasted material. The blast should compact the 'waste' in front of the fresh ore so that it does not flow as readily as the freshly blasted ore.
- Temporary arching: Coarser material at the top of the ring impedes the flow of waste from above, while coarser material draws over a much wider 'arch' than fine material does. Coarse material can temporarily 'hang up', while the finer material below the arch is drawn (but very coarse or unblasted material will allow waste to flow around the ore and reduce recovery).
- Interactive draw: Drawpoints drawn together result in a much wider zone of moving material where the increase in recovery can be significant and the material in the ring can be drawn to much flatter angles.
- Ground support: Support is primarily for the brow as the brow has to stay stable and support intensity will be much more than required for normal tunnel stability as the brow has to accept the blast damage and loading by broken material.

INTERESTING INFO

The flow of broken ore is not well understood (Brady & Brown, 2006) but has been studied using experiments:

- Granular materials in bins (Kvapil 1965, Jenike 1966);
- Numerical modelling using the theory of plasticity (Pariseau & Pfeider, 1968);
- Numerical modelling using the theory of probability (Jolley 1968; Gustaffson 1998);
- Full-scale studies based on marker recovery (Janelid & Kvapil 1966; Rustan 2000).

Issues that make the understanding of the flow of ore difficult include:

- The particle size of broken ore is different to that of sand, and therefore studies based on other granular materials may not be appropriate;
- The progressive breakdown of ore during flow may also mean that studies based on other granular materials may not be appropriate;
- The degree of fragmentation is not constant and cannot be repeated, while a uniform fragmentation is impossible, again making the use of studies in uniform granular materials inappropriate.

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The theory behind the Janelid and Kvapil (1966) flow assumes that a certain void has been filled with broken material and that the material flow would be governed by two ellipsoids, namely the limit ellipsoid and the ellipsoid of flow/motion (Figure 1).
These ellipsoid shapes are not pure ellipsoids in practice, but the essence of material flow suggested by this method remains applicable. Within the ellipsoid of flow/motion, material flows towards the drawpoint and forms a draw cone, while material within the limit ellipsoid will loosen and eventually move towards the ellipsoid of motion.

NOTE

The designs usually ensure overlapping ellipsoids from different draw-points to ensure material flow, minimising ore loss and dilution.

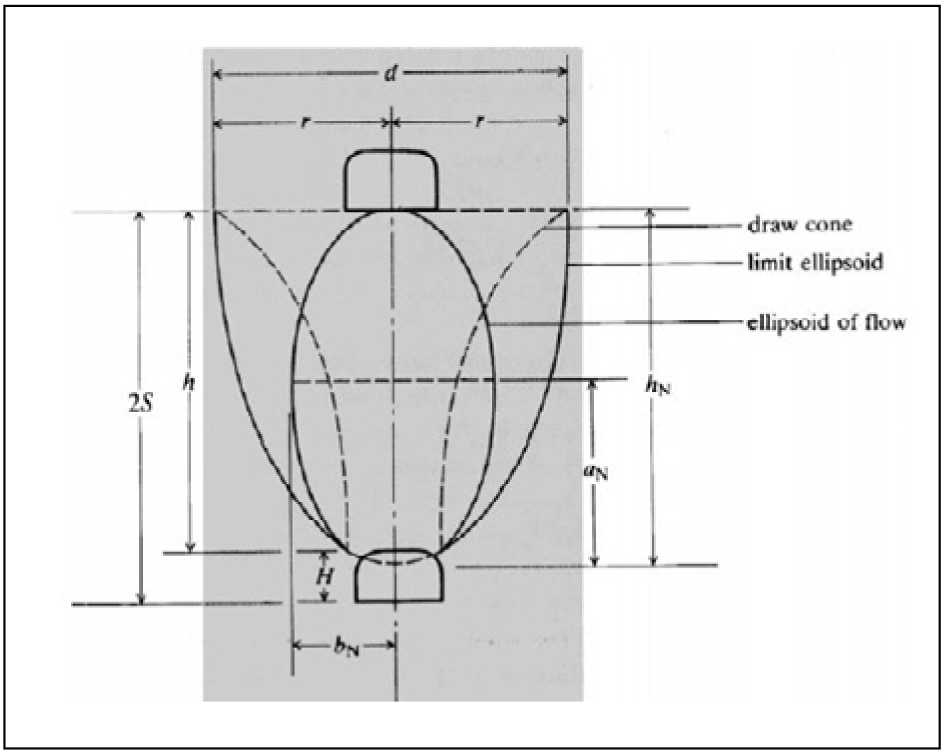

Figure 16: Flow ellipsoid concept after Janelid and Kvapil (Brady & Brown, 2005)

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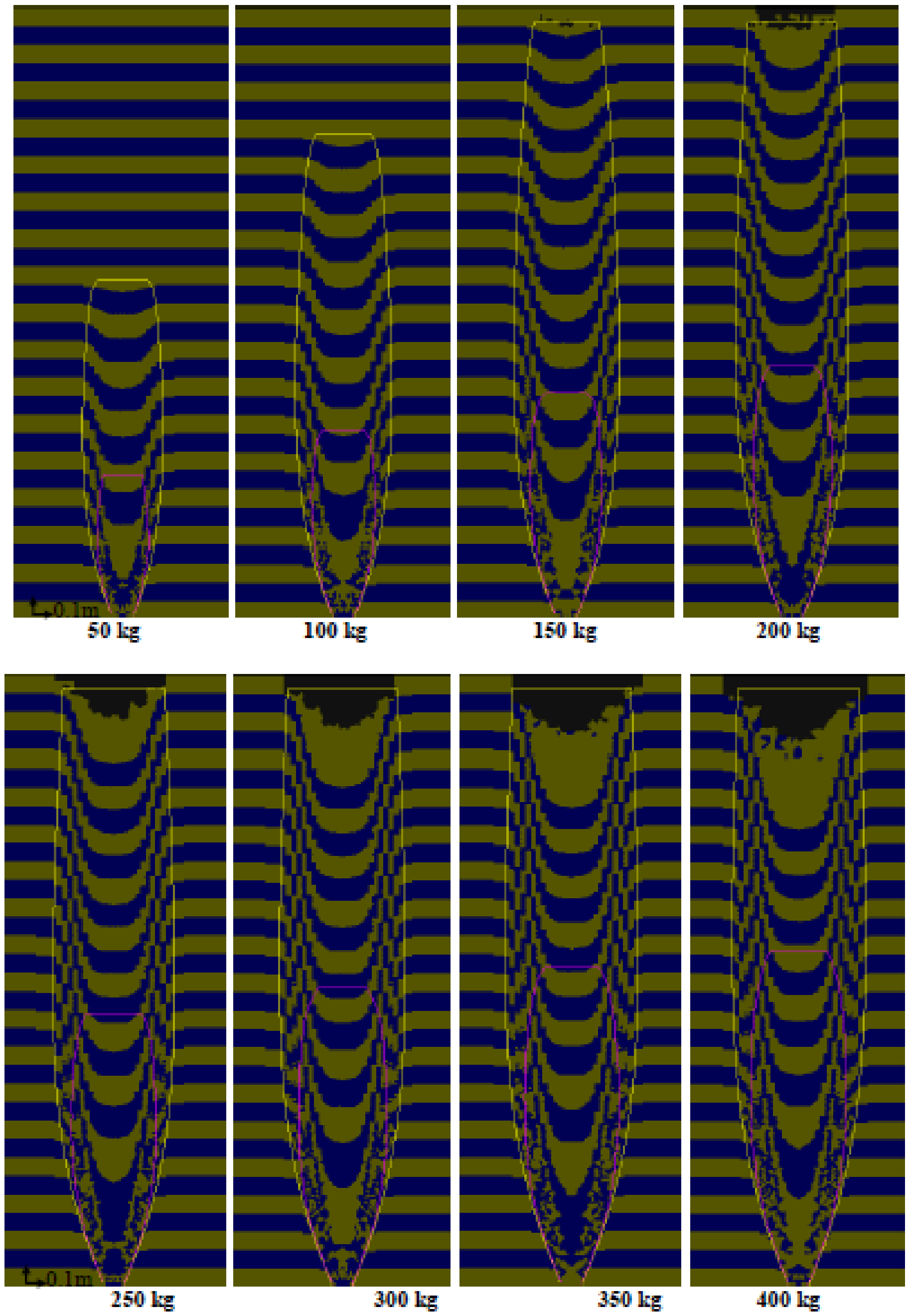

Figure 17: Drawdown profile for single drawpoint simulated (Pierce, 2010)

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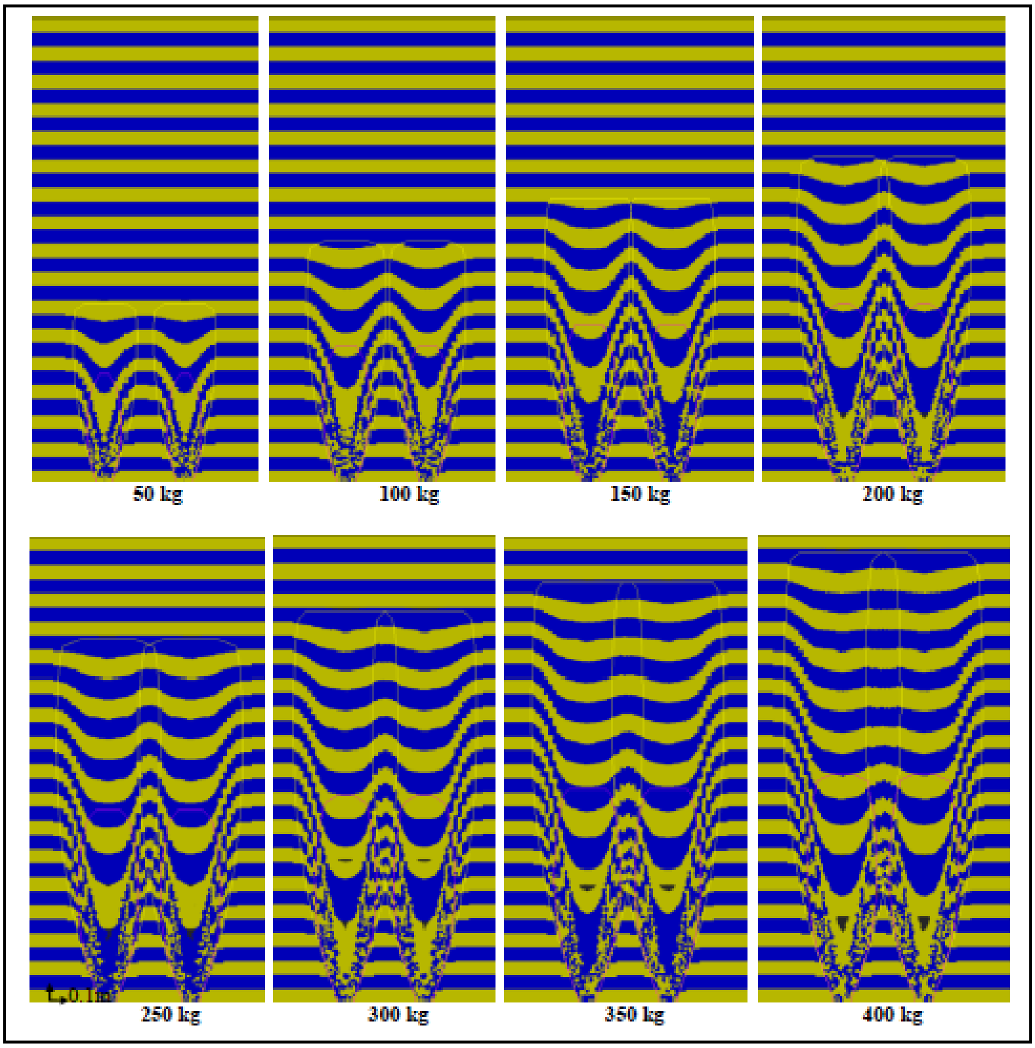

Figure 18: Drawdown profile for twin drawpoints simulated (Pierce, 2010)

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Figure 19: Multiple drawpoint flows analysed by Kvapil (Pierce, 2010)

REFERENCE

- Page 454, Brady and Brown, Rock mechanics for underground mining, 2006
- Design of cave mining issues Kosowan
- Gravity flow of fragmented ore Pierce
- Study of drawzone interaction Trueman
- Draw control Smith

EXAMPLE

Drawpoint spacing design based on material flow

The design methods considered included:

- Janelid and Kvapil's ellipsoid concept (Brady & Brown, 2006): The drawpoint spacing can be determined from the limit ellipsoid radius; and
- Laubscher's isolated drawzone diameter (Laubscher, 1994): The drawpoint spacing can be determined from a table of values relating fragmentation size to loading width and drawpoint spacing.

The following issues were considered during the design process:

- Interactive draw: Interactive draw requires planned ore extraction by drawing from adjacent drawpoints on a rotational basis to increase the area subjected to ore movement and improving ore flow. At the same time, interactive draw suggests that draw zones within the broken ore overlap and reduce the potential for ore loss should a single drawpoint become unusable.
- Arching of coarse material: Coarse material impedes material flow and will, if the drawpoint extraction layout is too narrow, result in arching across the drawpoint and the potential loss of ore above the bridge.

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- Blasting and fragmentation: Fragmentation is critical to ore flow success. Finer particles will flow easier, while coarser particles will increase the potential for bridging.
- Brow stability: The stability of the brow between the drawpoint and the ore body contact is critical as a collapse of the brow will block a drawpoint and require additional blasting to attempt to restore the access to the ore.
- Shape of drawpoint: The shape of the drawpoint will be determined by the angle of the footwall ore body contact, the width of the ore body or the distance to the hangingwall contact and the spacing between the drawpoints. The angles created within the drawpoint will affect ore flow and may result in ore loss if too flat or bridging if too steep.
- Friction angle of the broken material: The friction angle of the broken material determines the theoretical digging depth by a loader in a drawpoint.
- Drawpoint size: The drawpoint size assists material flow with small openings limiting flow and large openings allowing near parallel flow into the drawpoint.
- Drawpoint pillar stability: In caving layouts, pillars are created between drawpoints and are subjected to vertical loading either by the overburden or the broken ore. In the SLOS layout, these pillars will not be formed, while loading by broken ore will be minimal (weight of a single blast only).
- Drawpoint life: The SLOS mining method will require the services of any single drawpoint for a limited period of time, as the production faces retreat away from the drawpoint and it is abandoned. The long-term stability of the drawpoint is therefore not a serious requirement.

Janelid and Kvapil's ellipsoid concept

In this design, it was assumed that the eccentricity of the ellipsoid of motion is 0.9, which is the minimum of the suggested range of 0.9 to 0.98 (Brady & Brown, 2006) and is based on the presumption that the shape of the ellipsoid of motion will be flatter in the SLOS ore pile than what is suggested for caving ore.

Estimating the heights of the ellipsoid of motion (h_N) and limit ellipsoid ($2.5*h_N$) from basic volume calculations performed for a single blast in the SLOS layout, the radius of the limit ellipsoid could be determined from:

$$r \approx \sqrt{[h_N(h_G-h_N)(1-\varepsilon^2)]}$$

This allowed the estimation of the drawpoint spacing A using A < 2r (Table 1).

Table 1: Drawpoint spacing after Janelid and Kvapil's ellipsoid concept

Radius of limit ellipsoid (m)	hN (m)	HG (m)	Eccentricity	Drawpoint spacing (m)
12.3	23	57.5	0.9	24.6

- Page 455-459, Brady and Brown, Rock mechanics for underground mining, 2006

Laubscher's isolated drawzone diameter

Laubscher developed Figure 20 based on rockmass class, fragmentation size of the ore, fracture frequency and loading width. Laubscher

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indicated that drawzone diameter and isolated drawzone spacing can be estimated from the input parameters shown above.

From the isolated drawzone spacing (IDZ), the drawpoint spacing to allow interactive drawing between drawpoints can be estimated at 1.5*IDZ (Table 2).

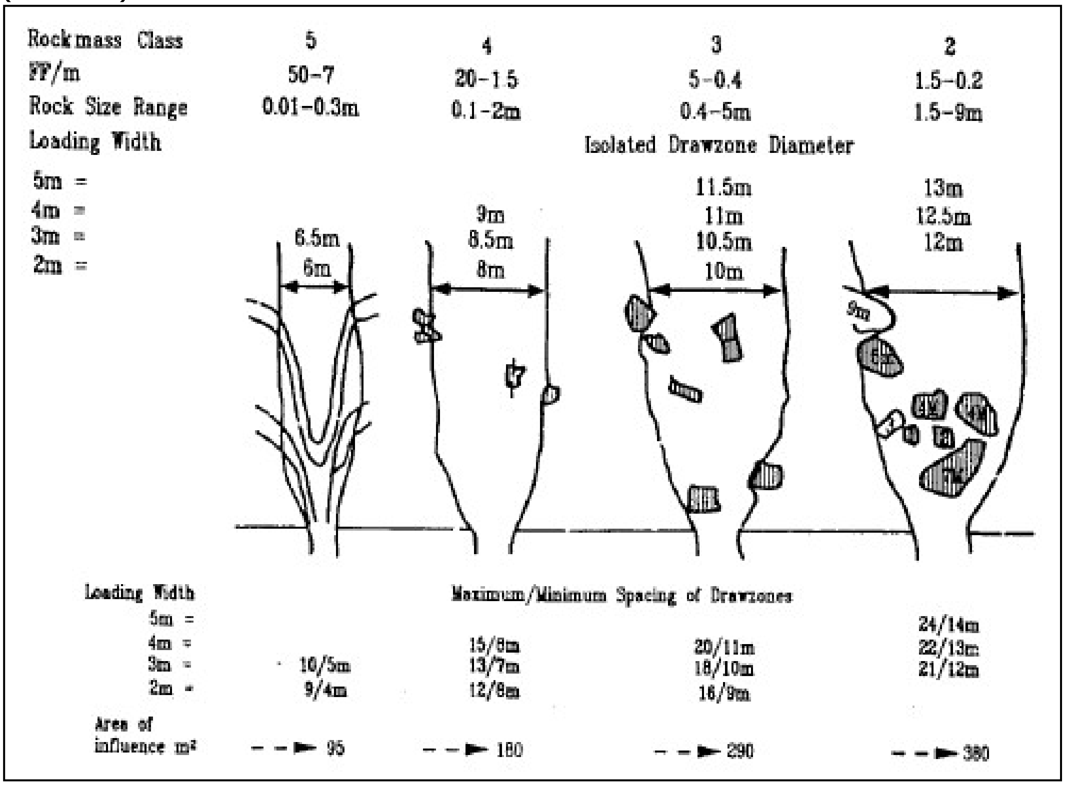

Figure 20: Drawpoint spacing based on Laubscher's IDZ (Laubscher, 1994)

Table 2: Drawpoint spacing results

Fragmentation	Loading width	IDZ (maximum)	Drawpoint spacing
0.6m Average	5m	>24m	36m

- State of the art cave mining Laubscher
- Drawpoint design Castro
- OTH303 Investigation into Drawpoints

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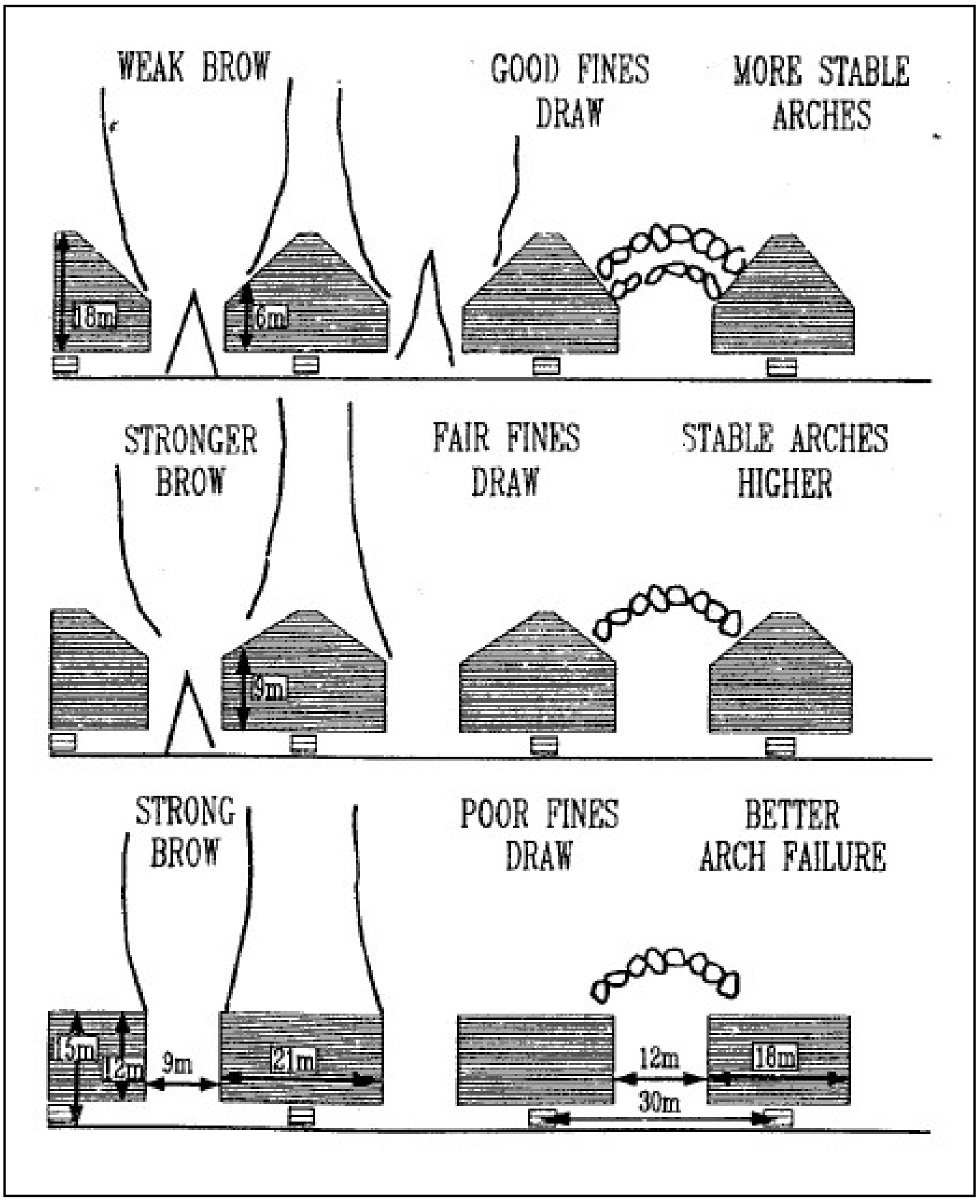

Figure 21: Drawpoint shape (Laubscher, 1994)

INTERESTING INFO

- Blasting and fragmentation: All efforts should be made to achieve fragmentation that will prevent large rocks from forming that could lead to bridging.
- Brow stability: To protect drawpoints, support installation into the drawpoint brow should be considered. Secondary blasting of rocks that may collapse into the drawpoints will serve to further destabilise the footwall material and should be prevented.
- Drawpoint shape: The shape of the drawpoint (Figure 3) is critical in ensuring good ore flow into the drawpoint. Flatter drawpoint side slopes will limit bridging, but will negatively affect fine material flow.

- Interactive draw: Interactive draw is ensured by applying the drawpoint spacing as derived though the methods indicated above. A maximum spacing of 36m and a minimum spacing of 24m are suggested. The planned 30m drawpoint spacings are viewed as appropriate.
- Arching of coarse material: By maintaining a sufficient drawpoint width, the potential for bridging is shown by Laubscher (Figure 21) to be limited, as the material is generally of small dimension and the loading width across the total drawpoint access substantial.
- Digging depth: The friction angle of the broken material is estimated at approximately the angle of repose of 38° (appropriate average loose ore value). The relationship between digging depth (x) and the drawpoint height and friction angle is suggested by Janelid and Kvapil (Brady & Brown, 2006) to be

  $x=H[\cot\varphi-\tan(45^\circ-\varphi/2)]$,

  allowing the calculation of a digging depth of 4m for the planned drawpoint layout (Table 3, Figure 4).

Table 3: Digging depth

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Loading depth (m)	Friction angle (degrees)	Drawpoint height (m)
3.96	38	5

This digging depth of 4m indicates that loaders should not approach the brow in the drawpoint closer than 2.4m and should therefore not be directly subjected to any brow instability during the bulk of the loading periods. Loading practices can therefore continue at low risk to equipment and personnel up to the final loading stage, at which time the drawpoint will have to be entered.

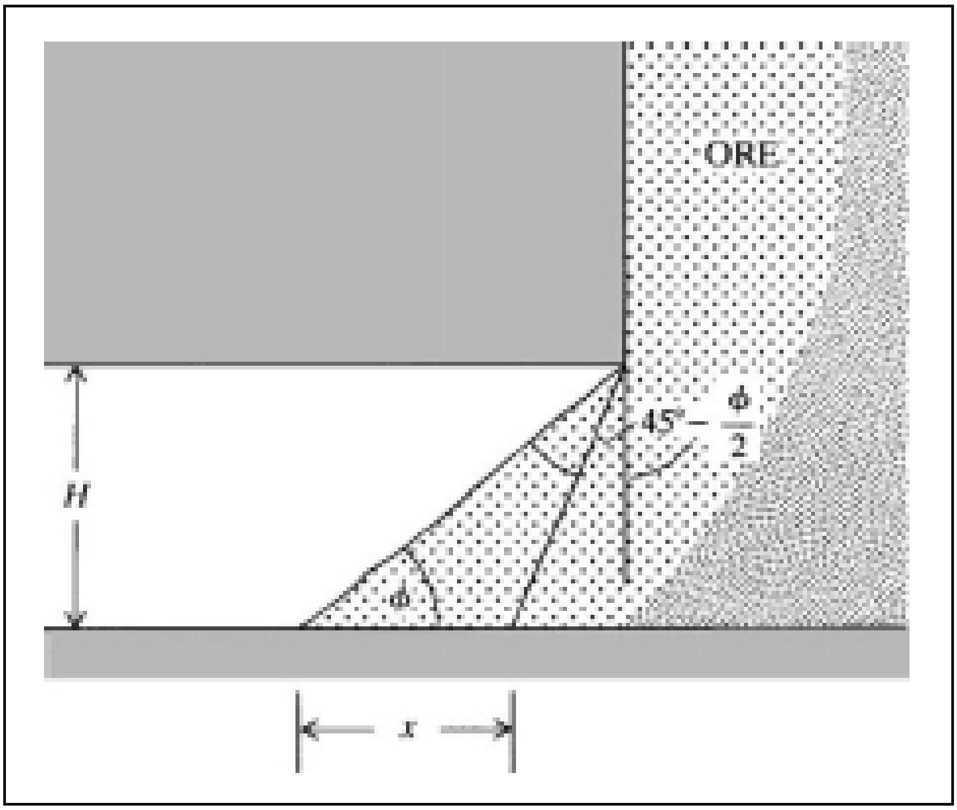

Figure 22: Digging depth in the drawpoint (Brady & Brown, 2006)

REFERENCE

- Page 461, Brady and Brown, Rock mechanics for underground mining, 2006

LEARNING OUTCOME 3.2.2.3

CONNECTION 15

Refer to "LEARNING OUTCOME 3.2.2.2" on page 98

DESCRIBE, DISCUSS AND EXPLAIN THE DRAW OF FINE AND COARSE MATERIALS IN THE SAME COLUMN AND ITS EFFECT ON DILUTION

NOTE

The primary objective should be delayed dilution entry. The later the dilution enters the muck pile, the greater the proportion of 'clean' ore and the higher the grade factor for the same extraction.
Fragmentation of the ore and surrounding waste material is a critical issue. The fragmentation from blasting and the natural fragmentation of

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the material that caves (joint frequency, joint condition and joint direction) should be considered in estimating its impact on flow through the broken material.

A competent rockmass is usually the most suitable, as larger, widely spaced blast holes will still result in good fragmentation, but the caving of the waste will be very coarse. This results in blasted material that is finer than the cave and a preference to flow towards the drawpoint, delaying the entrance of the coarser waste material and consequently dilution.

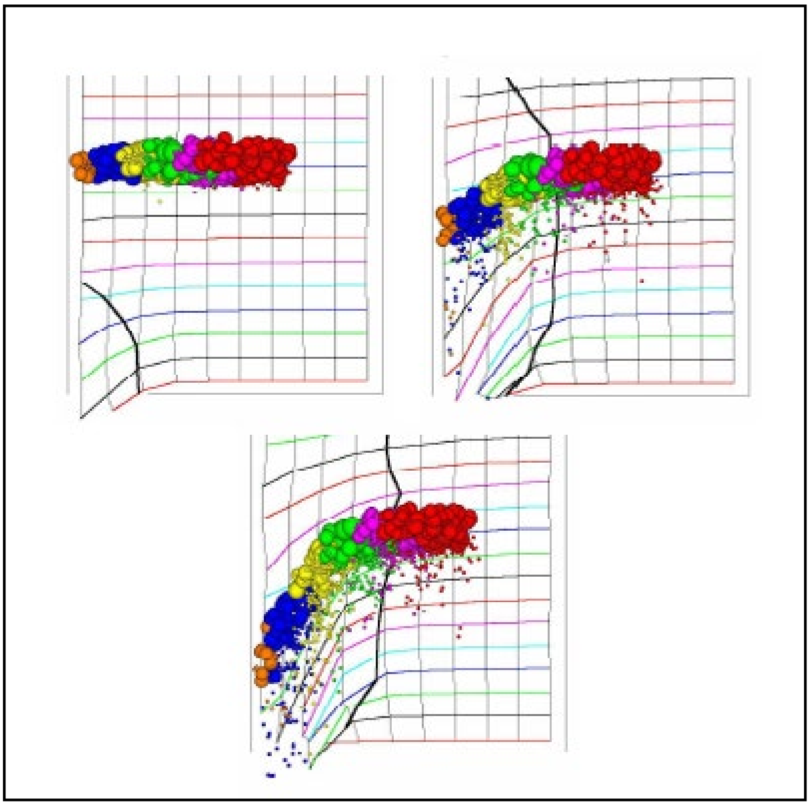

Figure 23: Fines migration simulations (Pierce, 2010)

REFERENCE

- Gravity flow of fragmented rock Pierce

LEARNING OUTCOME 3.2.2.4

CONNECTION 16

Refer to
"LEARNING OUTCOME 3.2.2.3" on page 106

DESCRIBE, DISCUSS AND EXPLAIN THE EFFECT OF STATIC PILLARS AND TEMPORARY HANG-UPS ON THE LOADING OF DRAWPOINTS

NOTE

In block caves, the 'pillars' are viewed as the 'higher stressed areas' due

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to an uneven cave linea as well as 'unpulled pillars' due to the non-overlapping of draw cones between drawpoints, while rib-and-sill pillars are left intact in other mehods to support the back.

Impact of 'pillars' and abutments:

1. Block caves:

INTERESTING INFO

The drifts can be further damaged by the relaxation of the fractured rockmass that occurs after undercutting and the magnitude of this damage is normally proportional to the depth below surface, the undercut height, the distance between the undercut and the draw horizon, the strength of the rockmass, and the pillar size.

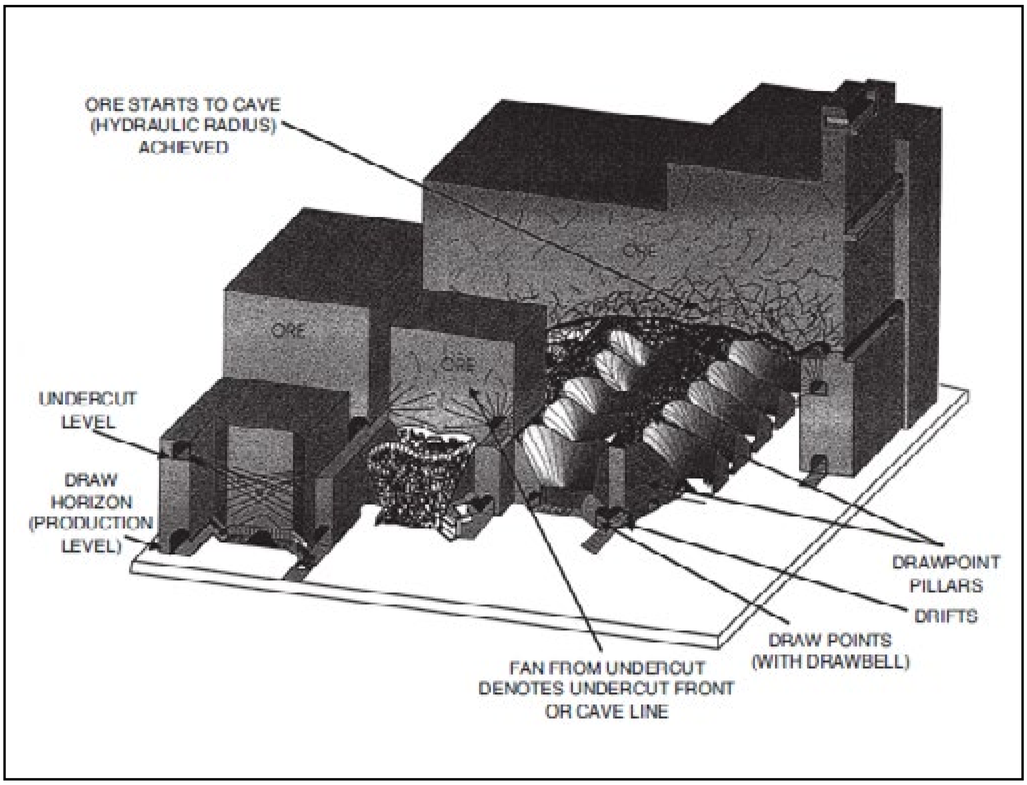

Figure 24: Block cave layout and draw horizon (Butcher, 1999)

The increase in stress during undercutting is considered to be the major factor that affects the stability of excavations on the draw horizon in block caves. The undercut abutment stresses are concentrated in the pillars between the drifts and drawpoints, resulting in possible scaling and failure of the pillars and drift damage.

NOTE

Where draw horizon development accounts for more than 60% of the level (i.e. only 40% of the area is left as pillars), stress-induced damage becomes unmanageable.

The magnitude of damage is increased if drawpoint and drift development is left unsupported for considerable periods.

Large irregularities in the horizontal configuration of the undercut front (uneven cave line) cause an increase in stress concentration on the draw horizon, which may result in serious damage.

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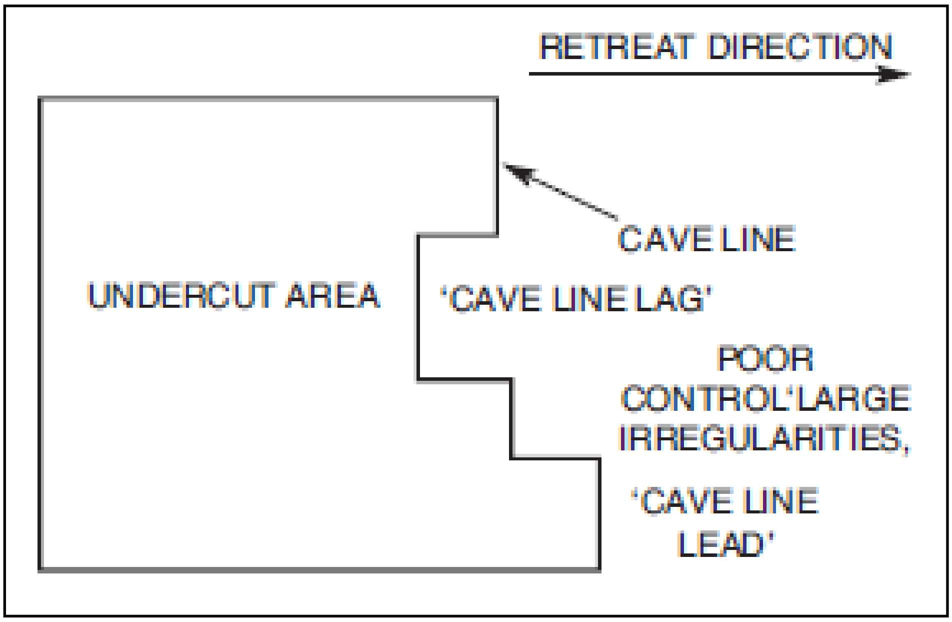

Figure 25: Uneven cave line creating 'pillars' (Butcher)

Stress-induced damage increases with a reduction in the rate of advance of the undercut front (cave line) where the abutment stresses reach the maximum magnitude when the undercut area equates to the required hydraulic radius of the block.

As the undercut starts to advance, an arch is formed above this area. The span of the arch increases until caving is achieved at the specific hydraulic radius. At this point, caving of the ore body will naturally continue to propagate if drawing commences and a reduction in undercut front stresses occurs. However, if drawing does not commence, and the undercut continues to advance, the undercut abutment stresses increase as the span of the arch increases until crushing of the undercut front occurs.

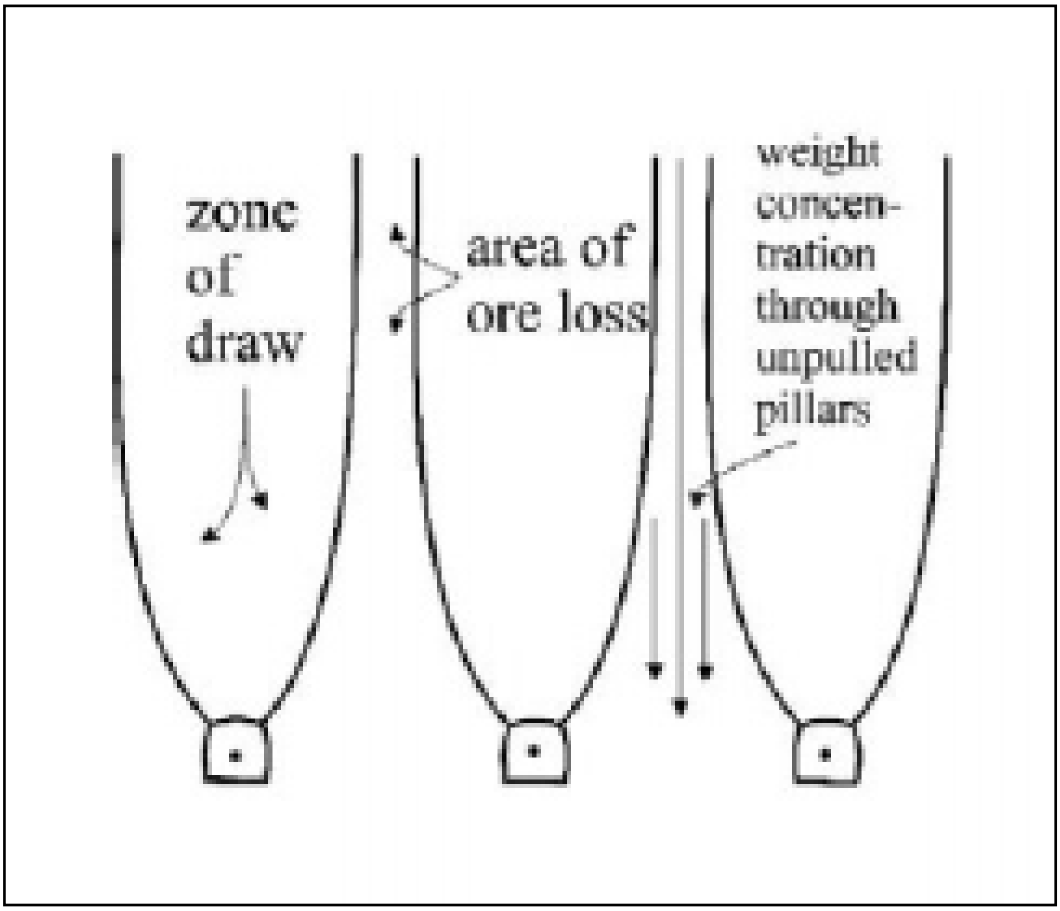

Figure 26: Unpulled "pillars" (Brady and Brwon, 2006)

In the case shown above, the draw zones of adjacent drawpoints do not overlap, isolated draw develops and pillars of un-drawn ore are left between the drawpoints, resulting in a potential loss of ore and in the application of dead weight loads to the pillars. The added weight to drawpoints requires more support installation to maintain stability.

2. Other methods:

Stress damage to drawpoints in methods where steep dipping, massive ore bodies are mined is limited to the impact of abutment stresses around the mined-out voids on the pillars between drawpoints. These abutment stresses are usually affected by the leaving of rib-and-sill pillars to support the open stopes.

The impact of the pillars on drawpoint stress levels is depicted by the following example:
The sub-level open stoping (SLOS) with backfill has been suggested for the thick ore body areas within the high grade areas where the ore body dip is practically vertical (90°), while the ore body thickness varies between 15m and 40m.

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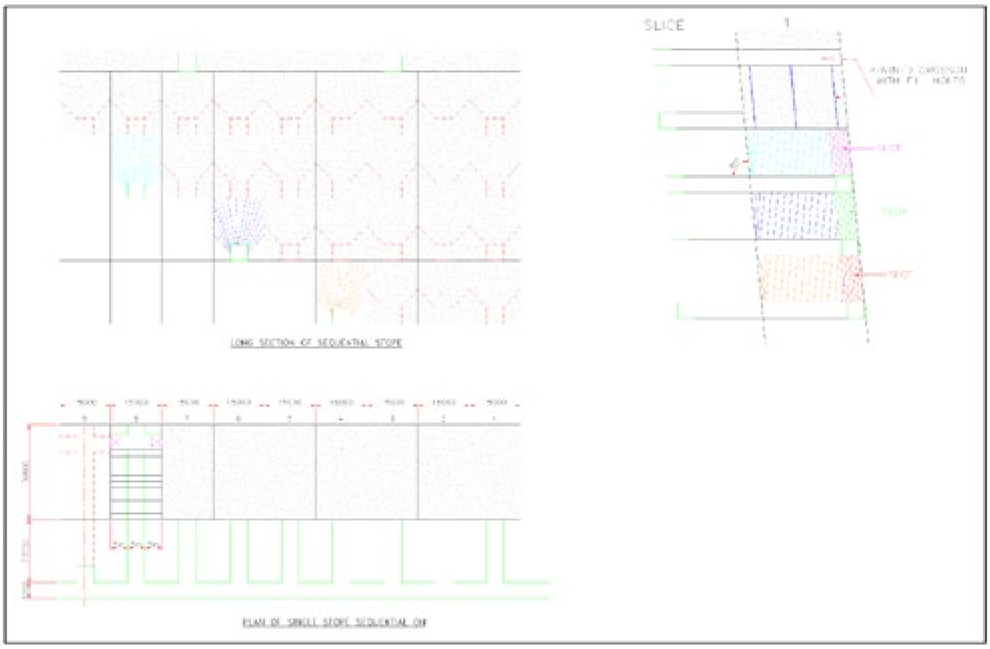

Figure 27: Sequential transverse SLOS layout

The concern was raised that this mining method, especially if a sequence where primary and secondary extractions are implemented in a multi-transverse stoping layout (Figure 8), could jeopardise the stability of the footwall excavations, due to stress concentrations on blocks still to be extracted (pillars).

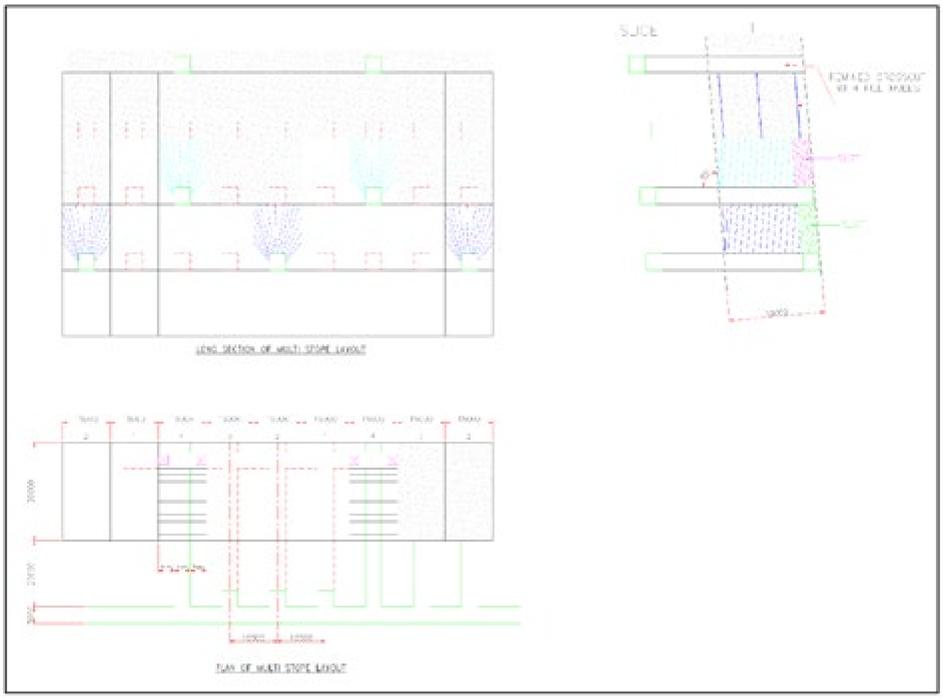

Figure 28: Multi-transverse SLOS layout

Numerical modelling and Map3D, a fully three-dimensional boundary element code, were utilised. Models were constructed to represent the current mine layout, placing an already mined-out area above the transverse stoping zone of 1 200m on strike and 120m on dip to allow a more accurate estimation of stress levels in the transverse stoping area when mining initially commences in this area.

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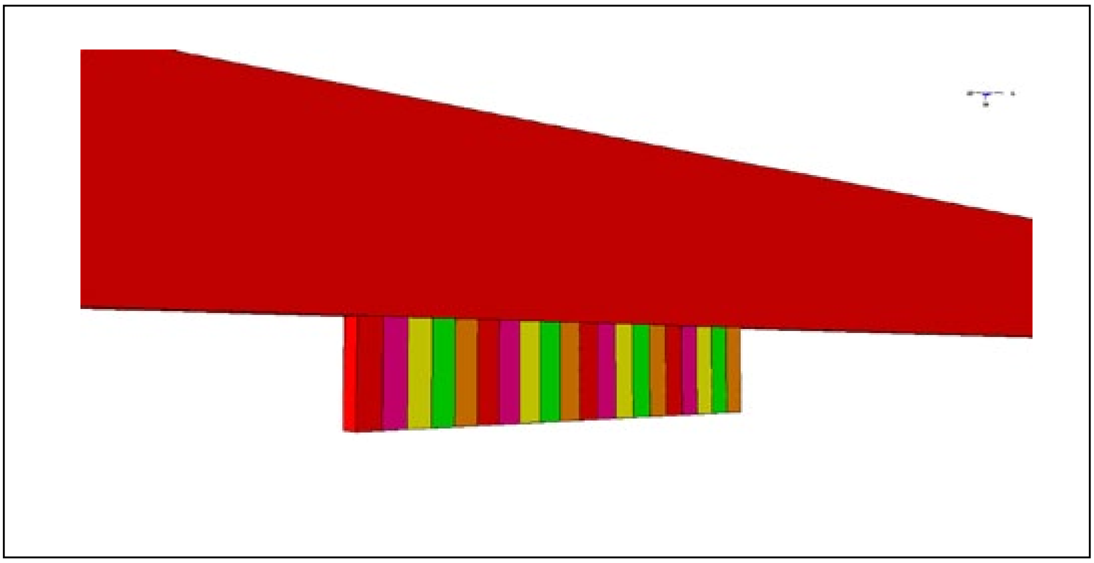

Figure 29: Ore body layout

Stopes (20 blocks) of 15m width, 60m height and thickness equal to the particular ore body width selected were created along a 300m strike distance (Figure 10) and at various mining depths and were mined in the most probable sequence (Table 3). This allowed the creation of 'pillars' and concentration of stress levels along the length of the footwall excavations, as mining progresses.

Strings were placed at regular spacings in the footwall (Figure 10) of the ore body (20m, 30m and 40m) on both major loading level and sub-level elevations, along which the induced stress levels could be determined to indicate the trend of stress changes as the ore body position is approached.

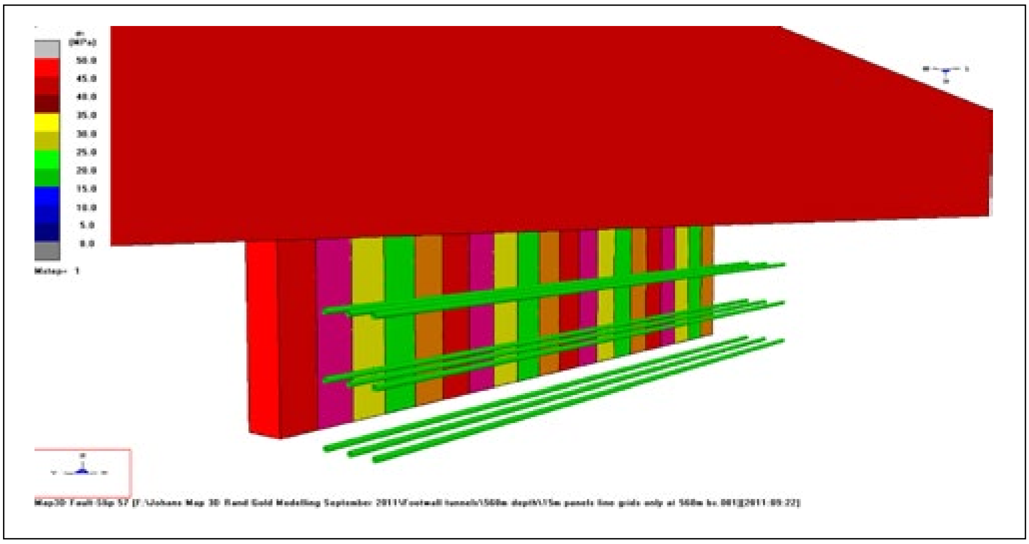

Figure 30: Strings placed in the ore body footwall

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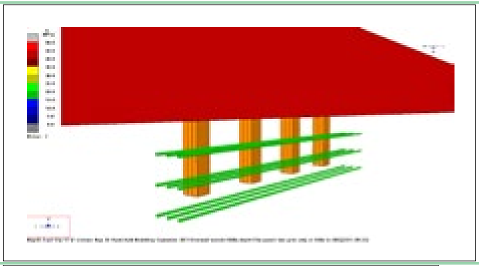
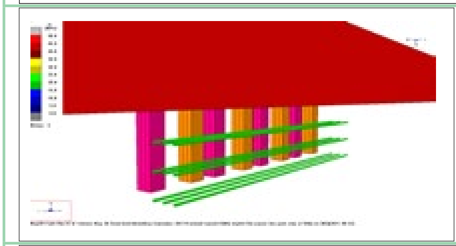
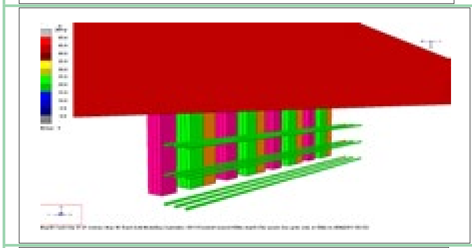
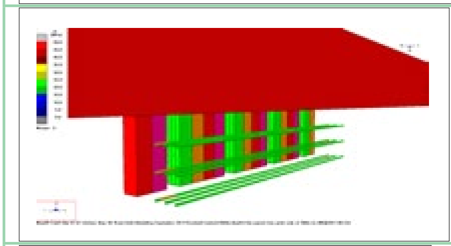
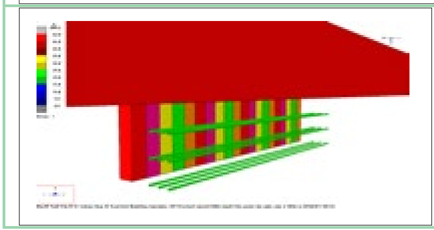
Table 4: Mining sequence simulated

	Step 1: All mining above the transverse stoping area (red). Step 2: Mining the 1st blocks at various positions in the multi-transverse layout (yellow).
	Step 3: Extraction of 2nd blocks (pink).
	Step 4: Extraction of 3rd blocks (green).
	Step 5: Extraction of 4th blocks (red).
	Step 6: Final extraction of last block in the multi-transverse layout (yellow).

Input parameters for the assessment were taken from historical the design reports and included the following:

Table 5: Input parameters

Parameter	Value	Parameter	Value
Young's Modulus (host rock)	50GPa	UCS (footwall)	117MPa
Poisson's ratio (host rock)	0.18	Vertical stress variation	-0.028MPa/m
K-ratio (horizontal to vertical parallel to ore body) – based on regional stress measurements	1.5	Horizontal stress variation (parallel to ore body)	-0.042MPa/m
K –ratio (horizontal to vertical perpendicular to ore body) – based on regional stress measurements	1.0	Horizontal stress variation (perpendicular to ore body)	-0.028MPa/m
Hoek-Brown strength parameters (mi,m)	5.09 , 0.16	Hoek-Brown strength parameters (s,a)	0.0003, 0.505

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PAPER 3.3: CHAPTER 3

Some of the results from a total of 12 models (to cater for the varying mining depth and ore body thickness) are presented below.
Maximum field stress level changes during the extraction of the different blocks in the multi-transverse SLOS layout are represented in Figure 31 for the loading level at 560m below surface and 20m from the ore body. Significant stress changes and/or concentrations commence when the 3rd block in the transverse SLOS sequence is mined and increases during the extraction of the 4th block with maximum impact during the final extraction. The major principal stress (horizontal to semi-horizontal) does appear to increase substantially until the extraction of the 4th and 5th (final) blocks, and it is also clear that the presence of mined-out stopes affects the minor principal stress levels, reducing it by nearly 50%, suggesting the de-stressing of the excavations in the vertical to semi-vertical direction.

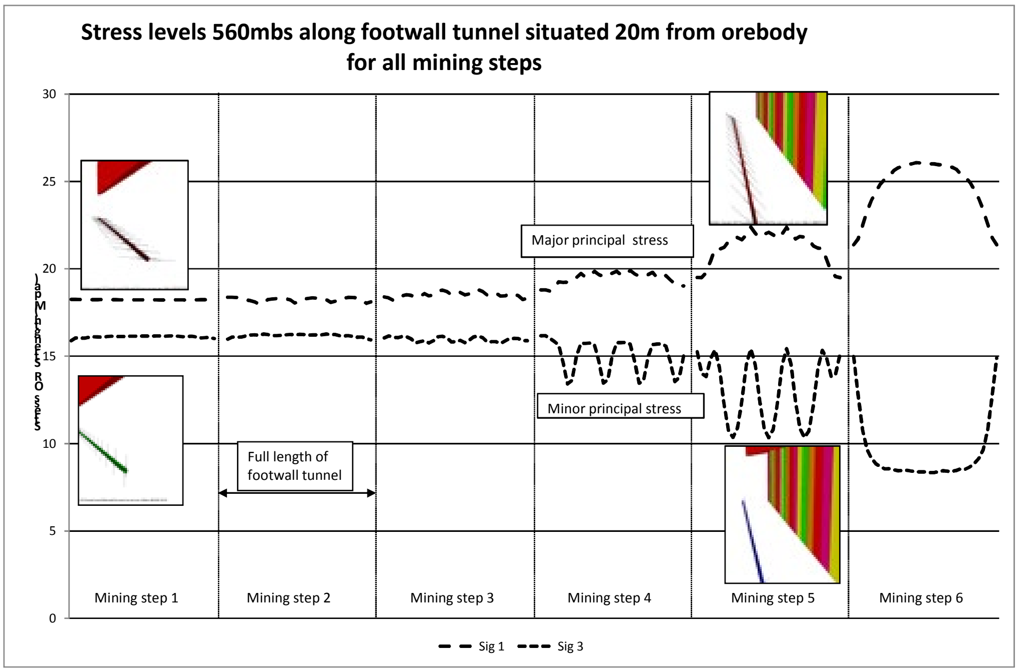

Figure 31: Field stress levels 20m from ore body at 560mbs for all mining steps

Stress levels induced by the mining around the ore body on the loading level are presented in Figure 12 at various distances from the ore body and represent the highest induced stress levels calculated for the different footwall excavation levels during all mining steps.
Figure 32 suggests that ore body thickness has very little impact on the induced and therefore also the final field stress states around the proposed footwall excavations (different colour lines at each simulated depth) and that depth below surface and distance from the ore body are the major contributors to the final field stress levels.

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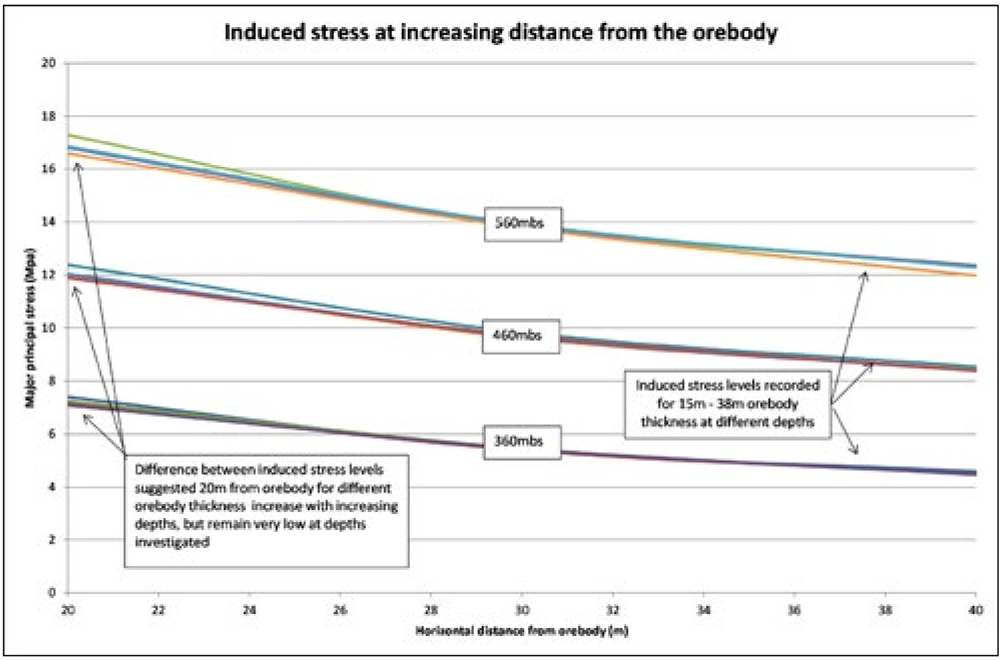

Figure 32: Induced stress levels at increasing loading level distance from ore body

NOTE

Since the modelling concentrated on stress levels near the footwall dives, it is clear that the major principal stress increases non-linearly as the ore body is approached, suggesting that stress levels around the lower abutment of the mined-out zones, where the drawpoints will be situated, will be significantly higher than those indicated at 20m from the ore body.

In the absence of modelling results around the abutments, the same results were extrapolated towards the ore body abutment and indicate the expected higher stress levels that drawpoints will potentially be subjected to.

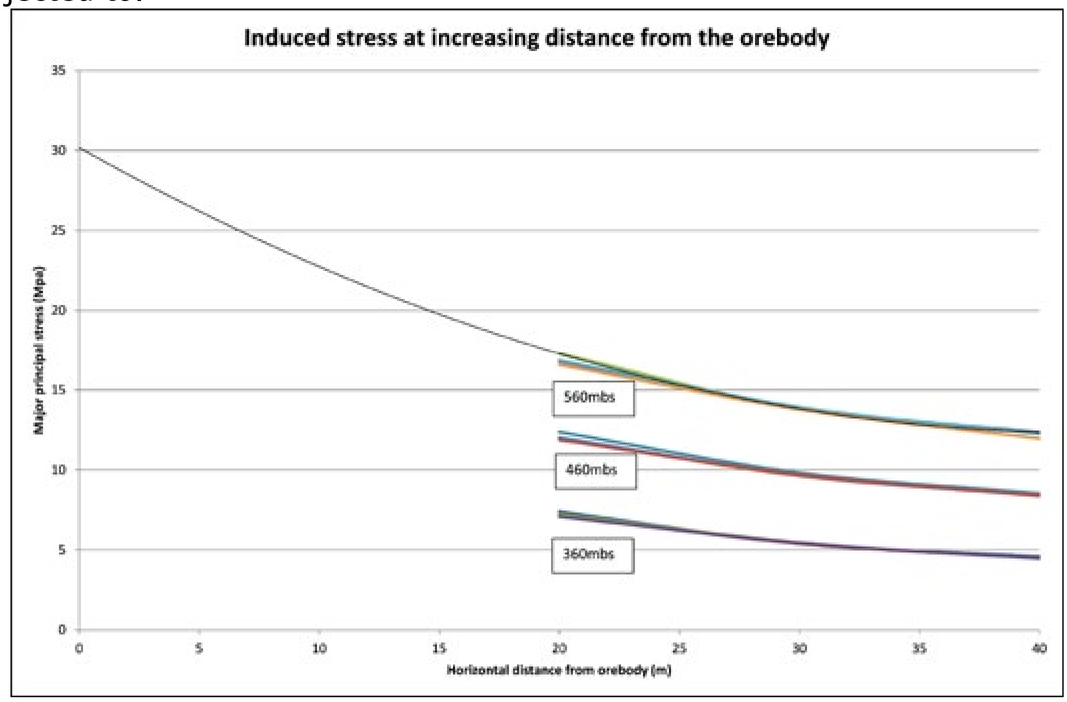

Figure 33: Extrapolated stress levels towards the drawpoint position

Impact of temporary hang-ups

Stress abutment level stresses reach the maximum magnitude when the undercut area equates to the required hydraulic radius of the block. This

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area is in essence an arch that forms above the undercut until it is large enough to induce failure and commence caving. The abutment stresses reach a maximum since the weight of the overburden is concentrated along the abutments of this stable arch. As soon as the material caves, the overburden above the undercut is no longer supported along the abutments and stress levels reduce.

A temporary hang-up, as shown in Figure 34, due to arching, has the same impact in that the weight of the failed material, due to the arch, is now transferred to the abutments of the arch/hang-up and the stress levels along these abutments increase. With a hang-up, this abutment is situated on the drawpoint and this increase in stress increases the potential for stress-induced damage to drawpoints.

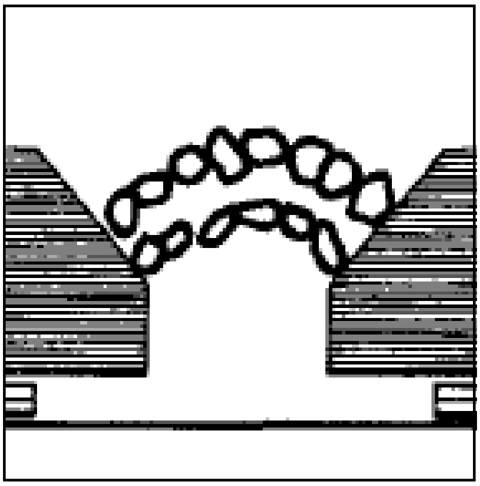

Figure 34: Stable arch formation/hang-ups (Laubscher, 1994)

REFERENCE

- Design rules to avoid draw horizon damage Butcher
- Stress analysis of block cave layouts Esterhuizen
- Study of draw zone interactions Trueman
- Gravity flow of fragmented rock Pierce

LEARNING OUTCOME 3.2.2.5

CONNECTION 17

Refer to
"LEARNING OUTCOME 3.2.2.1" on page 97

"LEARNING OUTCOME 3.2.2.2" on page 98

"LEARNING OUTCOME 3.2.2.3" on page 106

EVALUATE AND ESTIMATE LIKELY ROCK FLOW PATTERNS FOR GIVEN ROCK PROPERTY AND DRAWPOINT LAYOUT CONDITIONS

REFERENCE

- Study of draw zone interactions Trueman
- Gravity flow of fragmented rock Pierce
- Drawpoint design Castro

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LEARNING OUTCOME 3.2.2.6

EVALUATE AND ESTIMATE LIKELY DILUTION FOR GIVEN ROCK PROPERTY AND DRAWPOINT LAYOUT CONDITIONS.

One way to quantify dilution is ELOS (Equivalent Linear Overbreak Sloughing), which indicates the volume of material from outside the stope boundaries from FW and HW divided by the area of particular wall, seen in Figure 35.

This is not dependent on the volume of the stope, only the surface area of the walls. ELOS can be calculated using the following formula:

$$\text{ELOS (m)} = \frac{\text{Volume of overbreak from certain side}(m^3)}{[\text{stope length(m) x stope height(m)}]}$$

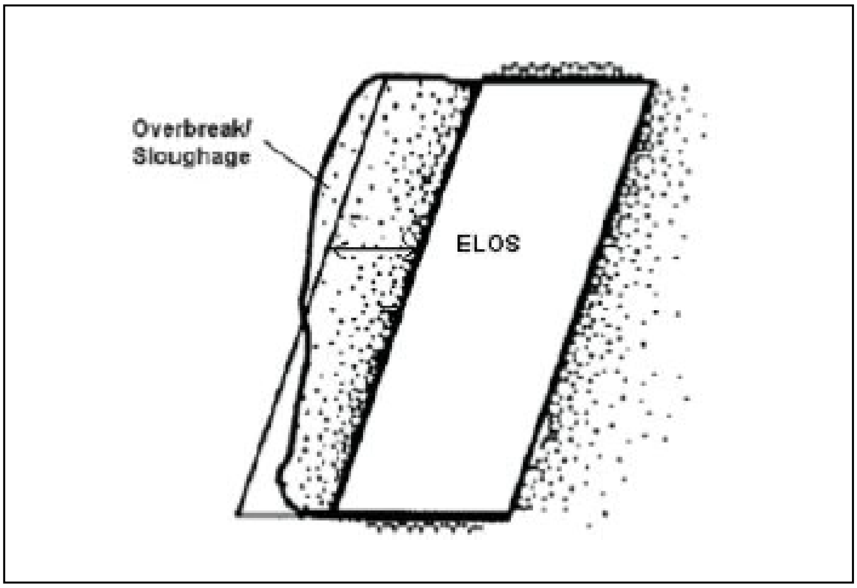

Figure 35: ELOS (Tommila, 2014)

There are many different possible reasons for dilution and can basically be divided into three categories:

- Drill and blast issues: Drilling accuracy is a result of errors in location of collar, wrong inclination, hole deviation and incorrect hole length. The effect of drill and blast issues to dilution is difficult to predict if there is not enough data to compare drilling accuracy and ELOS.

NOTE

Poor performance in drilling can lead to a variety of other results, including poor fragmentation, loss of blast holes, overbreak or underbreak.

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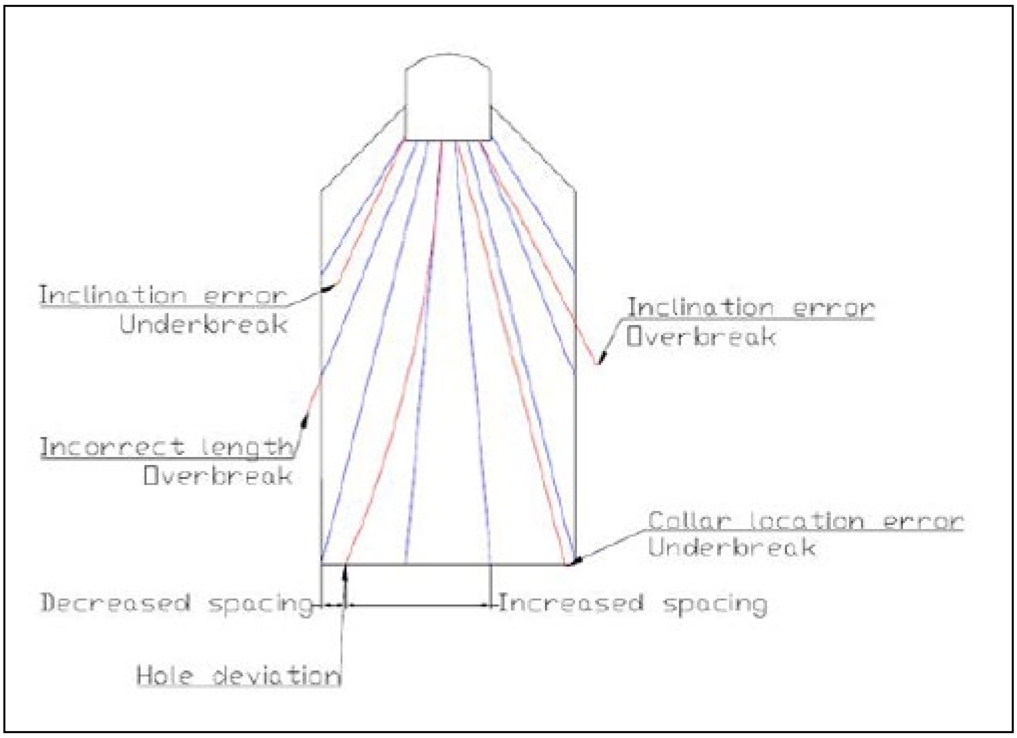

Figure 36: Drilling causing dilution

- Errors in planning: Dilution due to planning means that the stope boundaries are designed so that dilution is almost inevitable. This may include poor ore body geometry data, poor geological information, poor grade information, poor drilling layout planning and design etc.
- Geotechnical issues: Include the influence of stresses and the condition of the surrounding rockmass. Failures are usually caused by interaction of these two factors such as the relaxation of poor quality/strength rock. Normally, stope designs are based on the quality of the rockmass material, leading to the implementation of a specific length, width and thickness of an open stope. It has been indicated (Tommila, 2014) that for a specific operation, dilution increases for higher and longer stopes.

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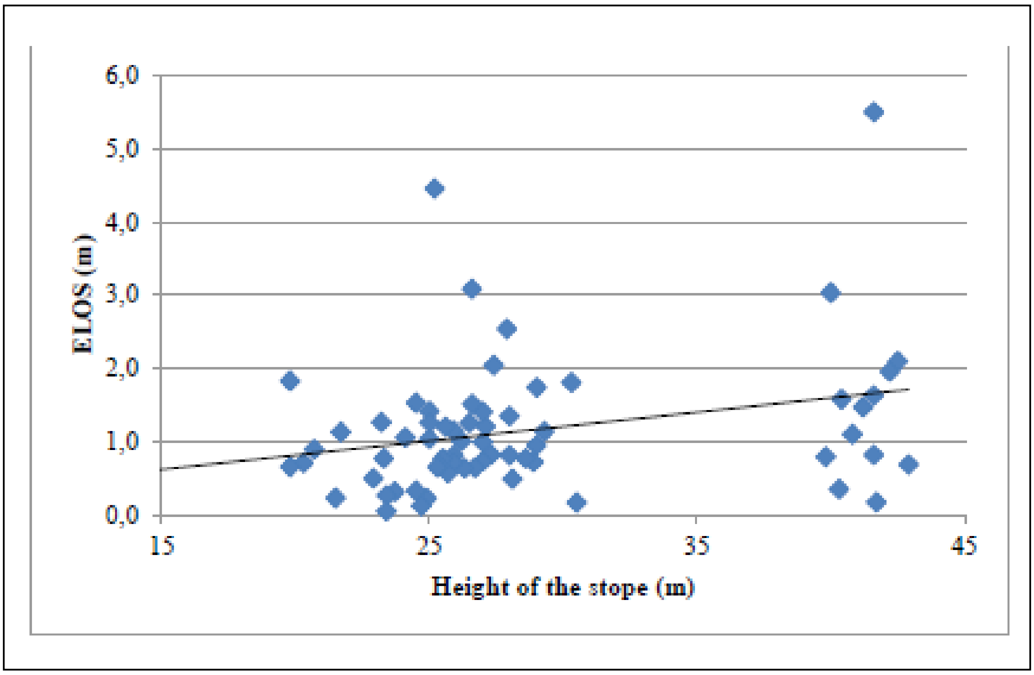

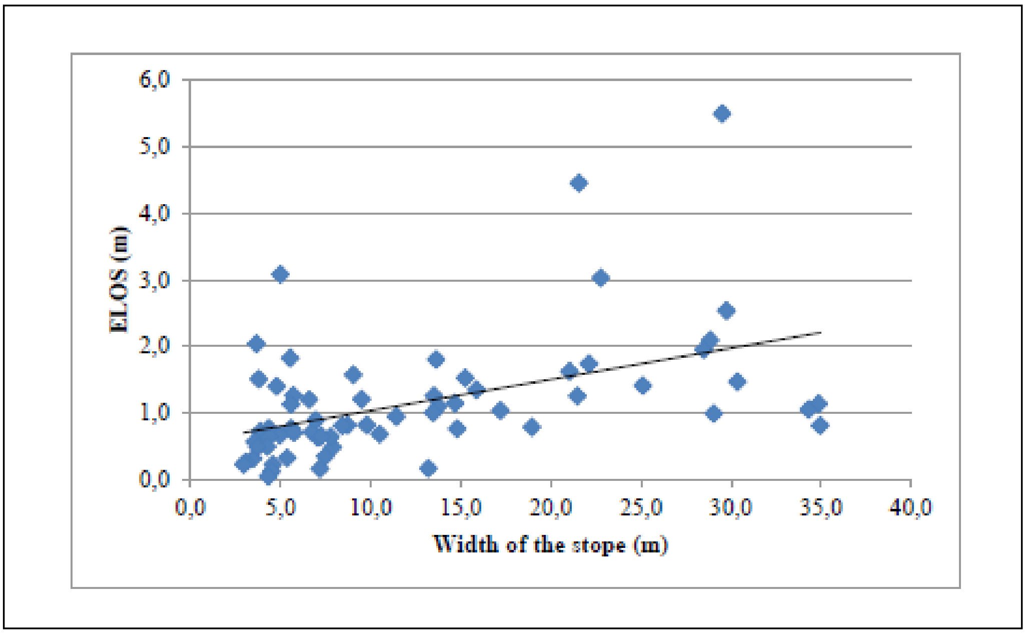

Figure 37: Stope height and width impact on dilution (ELOS) (Tommila, 2014)

Research has enabled the quantification of the amount of wall slough using a parameter called the equivalent linear overbreak/slough (ELOS), which is used to express volumetric measurements of overbreak as an average depth over the entire stope surface.

ELOS is defined as the volume of slough from the stope surface divided by the product of stope height times wall strike length known as the hydraulic radius (HR).

ELOS = Volume of slough / HR

The stability graph was adapted to relate hydraulic radius (HR) of the stope wall to empirical estimates of overbreak slough and allows the estimation of expected dilution, based on the rockmass quality and stope dimension.

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REFERENCE

- Minimising and predicting dilution Stewart

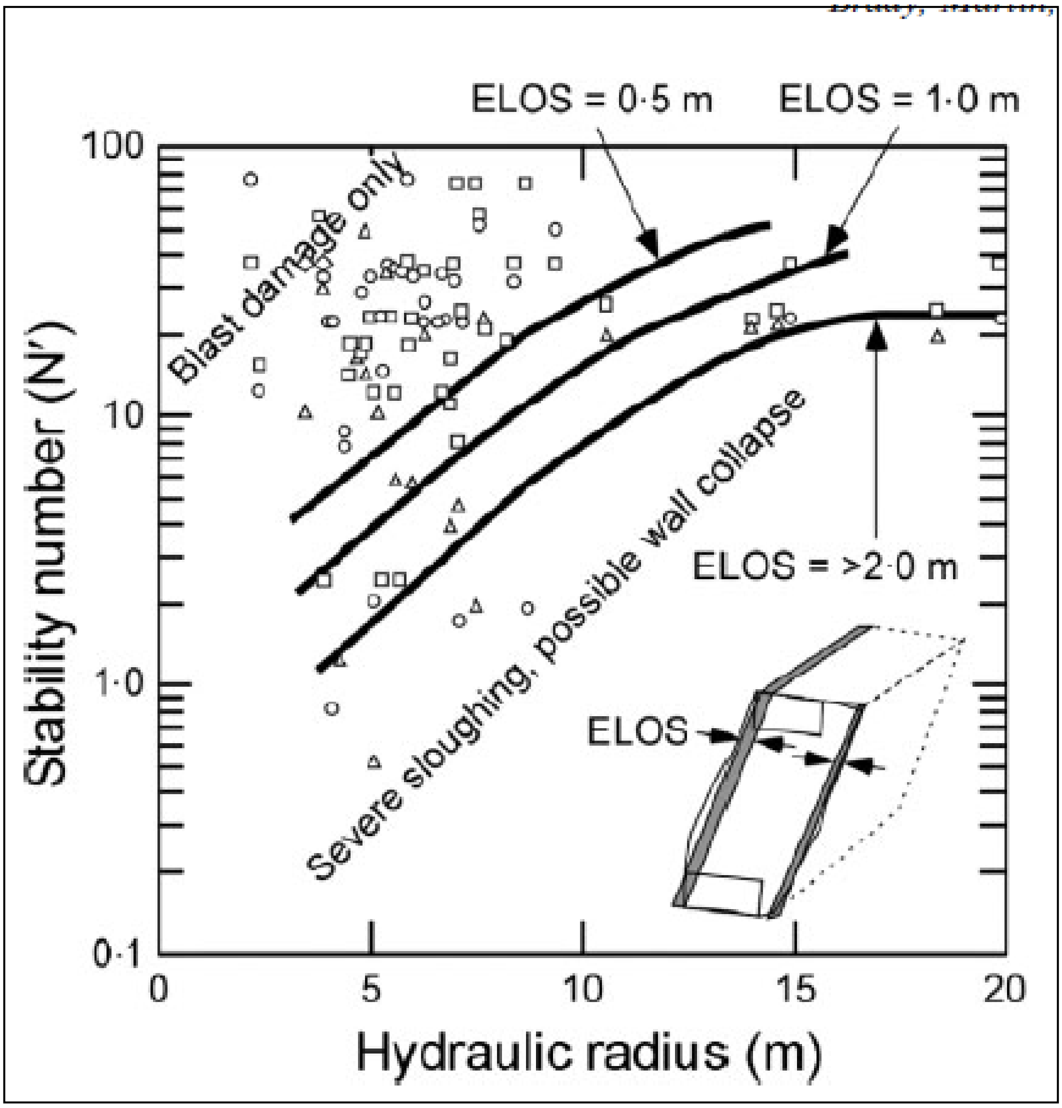

Figure 38: Estimating ELOS using the stability graph

INTERESTING INFO

This adaptation included:

- the addition to the Matthews cases, to the 400 case histories, which include open stoping experiences for a broad range of rockmass conditions in Australia, specifically at the Mount Charlotte mine were added
- much larger stopes that allowed the extension of the modified stability graph to a hydraulic radius of 55m as compared to the previous maximum value of 20m
- but, it was largely qualitative because surveys were not available to measure the wall slough

Hydraulic radius (HR) is defined as the surface area of an opening divided by the perimeter of the exposed wall.

$$N' = Q' \times A \times B \times C \quad (2)$$

where
N′ is the modified stability number;
Q′ is the Norwegian Geological Institute (NGI) rock quality index with the stress reduction factor and Jw ratio = 1.0
A is high stress reduction;
B is orientation of discontinuities; and
C is the orientation of surface.

NOTE

For the ELOS graph points, the database for the stability graph was derived from mining operations that are generally dry.

The A factor accounts for the influence of high stresses that reduce rockmass stability and is determined by the ratio of unconfined compressive

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strength of intact rock to maximum induced stress parallel to the opening surface.

The B factor looks at the influence of the orientation of discontinuities with respect to the surface analysed.

The C factor considers orientation of the surface being analysed and reflects the inherently more stable nature of vertical walls compared to a horizontal back.

- Drawpoint design Castro
- Minimising and predicting dilution Stewart
- Empirical design of stope openings Brady
- Empirical design methods for underground mines – Rimas Pakalnis
- Mining method and dilution control Tommila

3.2.3. ORE DILUTION

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

LEARNING OUTCOME 3.2.3.1

DESCRIBE, DISCUSS AND EXPLAIN THE GEOTECHNICAL FACTORS THAT AFFECT DILUTION IN OPEN STOPING OPERATIONS

Geotechnical issues that affect the potential to experience dilution include:

- Rockmass quality: Poor rockmass will not be able to maintain stability and will tend to cave. In open stopes, the intersection of areas with poorer quality could increase dilution;
- State of weathering: Weathering affects the tensile strength of the material and, when mining spans increase, leads to failure, increasing dilution;
- Stress levels: High stress levels could add structures to the rockmass that reduce its quality and strength, while low stress levels (tensile) assist separation on any structures and could also induce tensile failure of material, increasing dilution;
- Geological structures: Structures not only affect the rockmass quality and therefore its ability to remain stable, but also provide planes on which shear movement under gravity (or stress) could occur, leading to dilution;
- Water: Water affects both the rockmass quality and the tensile and shear strength of structures and thereby the rockmass, increasing the potential for dilution;
- Stope dimensions: Increased spans increase the potential for dilution as the ability of the material to remain stable in larger spans is lower;
- Blasting: Not only do poor blasting practices affect the quality of the rockmass (blasting fractures), but drilling practices can cause damage to the rockmass and result in poorer rockmass quality and increased potential for scaling;
- Ore body dip: Flatter dips increase the potential of dilution from the hangingwall as the impact of gravitation increases, but also from the footwall due to the shearing of flowing ore against the footwall material;
- Support practices: If specific structures are known around the

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ore body and support can be installed to control the movement of this rock, dilution can be reduced. However, for this to be effective, the rock quality must be good and structure positions well known.

EXAMPLE

This example is provided to indicate possible factors that affect dilution in open stopes and should be used for learning purposes only, as large portions of data were excluded in the interest of learning.

The major concerns with the scaling currently occurring into the stopes are that:

- it is occurring while mining is still continuing in some of the stopes where blasting of bridges and benches that had not been removed by the first blasting process was still being done;
- the large pieces of waste falling to the footwall of the stopes affect the possibility of mining the ore below the collapsing stopes; and
- the current stope dimension design could be negatively affected by having to mine at smaller stope spans if the rockmass requires small spans to maintain stability, affecting the long-term production plan.

The following general comments with regard to the reported scaling are provided:

- scaling appeared to have started after completion of a two-level dip span and then in most cases approximately two months after mining had been completed (except where additional blasting had to be done);
- the ore body dip in the region of the stopes that had experienced scaling is anything from ± 40° to 72°, increasing the back spans substantially, even though the vertical distance may be limited to two levels;
- the hangingwall has a clear contact with the ore body that has been sheared and folded during its formation;
- Logging results do not show constant poorer RQD levels (indication of broken up nature of the rock material) in the SQR (schistose/laminated quartz rose) hangingwall material, i.e. the hangingwall does not appear to disintegrate or break into significant pieces, even though it is laminated.

Table 6 combined the blocks mined on each level (from the plans above), showing the position of the scaling blocks in relation to each other. It indicates that scaling occurred in general where two (2) or more levels had been stoped out.

Table 6: Approximate position of scaling stopes

	Blocks situated above each other forming the same stope between levels.				
North coordinate	1 447 500	1 447 550	1 447 600	1 447 700	1 447 550
65 Level	Block 5	Block 1	Block 2	Block 3	
85 Level		Block 4	Block 3	Block 1	Block 2
110 Level				Block 1	Block 2

NOTE

Yellow highlights indicate stopes in which scaling was reported

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- Observations in the affected stopes:
- Scaling occurred after increasing the vertical stope spans to beyond two levels, i.e. beyond approximately 45m vertical (Table 6);
- Scaling occurred from the hangingwall only, with no signs of crown pillar or dip pillar instability, but with some indications that footwall scaling (mostly ore body material) also occurred;
- The ore body dip in this area is flatter than in most areas, but even flatter dipping areas were mined between 40L and 65L only, which shows no scaling at all ;
- The hangingwall consist of SQR material that has an approximately 2.0m thick zone that has been severely sheared. It is this zone that appears to be scaling into the stopes;
- Reports of sounds emanating from the stopes just prior to or during the scaling were received, but no signs of dynamic failures could be found.
- Waste scaling had occurred from hangingwalls but some ore could still be seen in some places, suggesting that the drill pattern and blasting did not succeed in removing all the ore;
- Scaled rocks included very large blocks, ± 3m x 2m x 1.5m, but also some very broken up, fine materials, suggesting that the sheared hangingwall and ore material may sometimes disintegrate when collapsing;
- In at least three of the areas that had scaled, blasting operations were not as successful as they should be, creating bridges and benches that required additional drilling and blasting. In at least two cases, drilling had to be done from the footwall drive into the ore bridge/bench to allow blasting and its removal;
- No sockets could be seen in the stope hangingwalls and although it is possible that scaling removed the evidence in areas where they did exist, no sockets were evident in areas where scaling did not occur.
- No signs of any crown or rib pillar instability were observed;
- The presence of smooth planes, starting on the ore body hangingwall contact and striking into the hangingwall bounded the scaling rocks, while other boundaries to the scaled rocks, allowing enough freedom for movement, were joints striking perpendicular to the ore body direction. The smooth planes had curved shapes, not unlike the 'domes' reported in other sheared environments. The curved shape provided a plane on which shear displacement could occur when the weight of the 'wedge' overcame the strength of the plane (Figure 39);
- Scaling of ore material from the hangingwall also appears to be a function of the curved ore/hangingwall contact and the possible drill pattern (Figure 40);
- Even though no signs of any dynamic failures were evident, failure along a plane such as that described above may have created the reported sounds.

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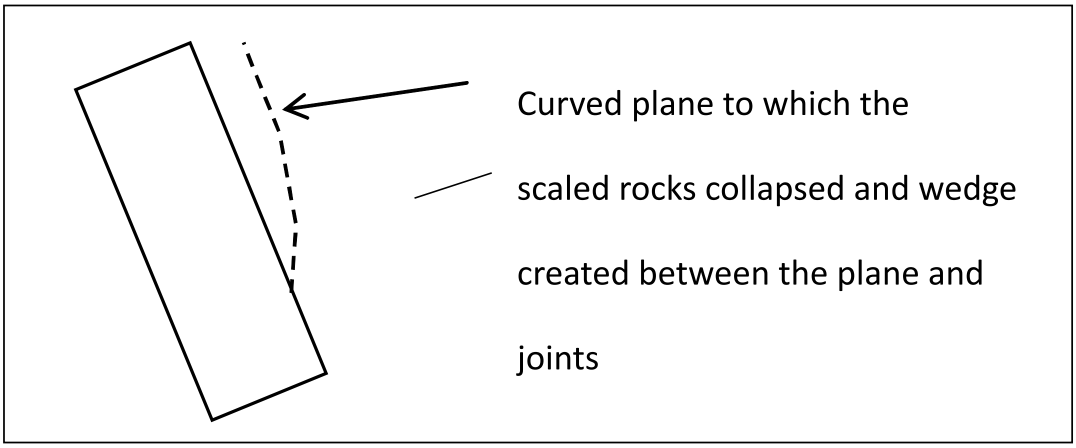

Figure 39: Estimated shape of scaled areas

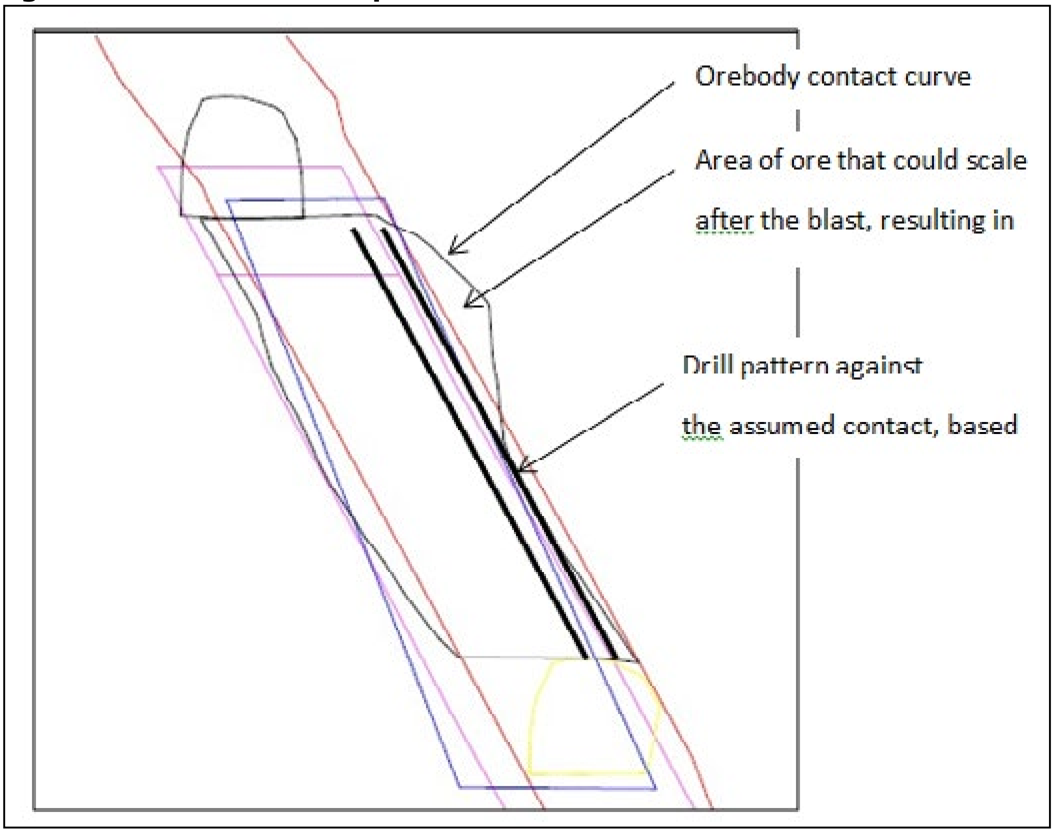

Figure 40: Wavy nature of the ore body contact

The scans completed of the scaled areas by the Survey department were provided for scrutiny. Table 7 shows that the scans completed after the scaling confirm the presence of smooth, curved planes that could be the 'ramp' structures mentioned above.

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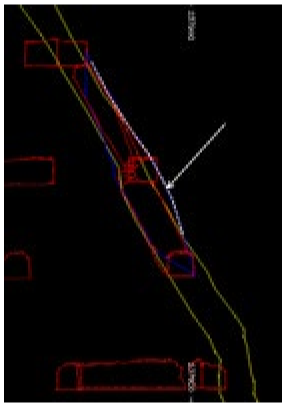
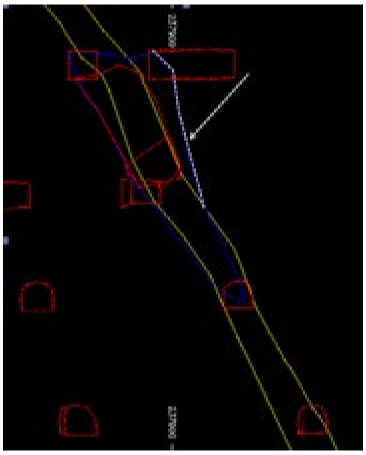
Table 7: Presence of a 'ramp' structure in the hangingwall

	
	Gold = Ore body wireframe Red = Scan after blasting stope Blue = Scan after scaling (white line to highlight hangingwall shape) All three pictures indicate that the blue/white line followed a smooth plane situated within the stope hangingwall, and that the 'daylighting' of this structure in the hanging-wall only occurred after mining of the lower level was done. This confirms the structures observed underground and appears to be of larger scale than the 'wavy' nature seen earlier. This scaling included the collapse of waste material and not only ore.

The following geology related issues are noted as having some impact on the occurrence of scaling in these stopes:

- A clear contact exists between the ore body and the SQR hanging-wall and should allow any ore left intact against the hangingwall to collapse easily to the contact. However, the presence of ore in some areas may suggest that some cohesion may exist along this contact that prevents initial collapse during blasting, but allows scaling at a later stage. This scaling should result in a hanging-wall shape such as shown in Figure 39;
- On the footwall, the contact is not clear due to a gradation in material properties towards the footwall waste. This, together with the fact that gravity is less effective against the footwall, could indicate why ore left intact along the footwall in most cases remains in place;
- Contacts between the ore body and the surrounding waste mate-rial are 'wavy' due to the folding that has occurred in and around the ore body. Folding processes create shearing on contacts be-tween layers and also the creation of 'ramp structures' between the prominent planes in which shear occurs.
- The geology department drills holes, for sampling and ore body delineation, on approximate 20m spacings, but the wavy nature of the ore body (due to folding) may create curves between these holes, resulting in straight (interpolated between data points) ore body contact wireframes, as shown in Figure 40. It was sug-gested at a meeting with all role players that a 10m grid may be required to pick up the waves in the ore body contact, to allow better management of dilution and drill pattern design;

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- Internal waste is reported to sometimes be present in the ore body, but is not shown on the ore body wireframes. The presence of this material may lead to blast damage if the drill pattern is not adapted to suite this material.

Discussions with the personnel responsible for the drilling and blasting yielded the following comments:

- The forming of bridges and benches during the blasts is a function of the blasting process. Currently, Anfo is being used, an explosive type known for its ability to negatively affect rock wall conditions, especially if incorrectly used. The current explosive management system also allowed the Anfo to age to a point where diesel seeped from the product, affecting its explosive properties;
- At the same time, charging up of the complete hole is sometimes affected by fines blocking the drilled holes, then affecting the final blast results and even contributing to the creation of benches;
- The potential for harder waste material within the ore body (interval waste) has been discussed in previous sections and the drill round implemented may not cater for this material, resulting in large burdens and poor breaking, but also damage to surrounding material, especially when the back of the holes are against the hangingwall;
- The ore body wireframe information made available to lay out the drill patterns is followed and yields the shapes shown in Table 7, where the blasted stope shape follows the original ore body wireframe. However, if the original drill layout was designed based on 'insufficient information' due to unknown curves in the hangingwall contact, drilling into the hangingwall is possible and scaling can be induced even though the design 'appears' to be correct. This concern indicates the need for a more stringent pattern of geological drilling to ensure the information is detailed enough before the drill pattern is drafted;
- The quality control on drilled holes is not yet in place and even though they are inspected with the borehole camera, methods such as the Boretrack equipment may assist in ensuring that the implemented pattern is also the designed pattern;
- Although planning to utilise emulsions instead of Anfo is underway, it has not yet been implemented. This change can bring about better management of explosive properties when the hangingwall should be protected;
- The position of the ore drive within the ore body has a huge impact on the drill round and the potential to induce blasting-related damage to the hangingwall. However, a meandering hangingwall contact makes it nearly impossible to manage drill patterns in such a way that holes are always drilled parallel to the hangingwall contact.

While some scans showed that the stope shape after the blast conformed very well to the indicated ore body wireframe, Figure 41 indicates that this conformation does not always occur and that some concerns with the blasting process do exist.

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Figure 41: Blasting and the indicated ore body position

Discussions with various personnel indicate a variety of reasons for this, and include claims of:

- Ore body wireframes are changed after the drill rounds have been designed and drilled, based on 'new' information. Figure 42 indicates the different stages of geological information received and indicates that uncertainty in detailed geological information may exist.
- Poor drilling and non-adherence to the designed drill round. The absence of a quality assurance system that can indicate adherence or not to a designed drill round makes it possible that some blast rounds may not be as designed.

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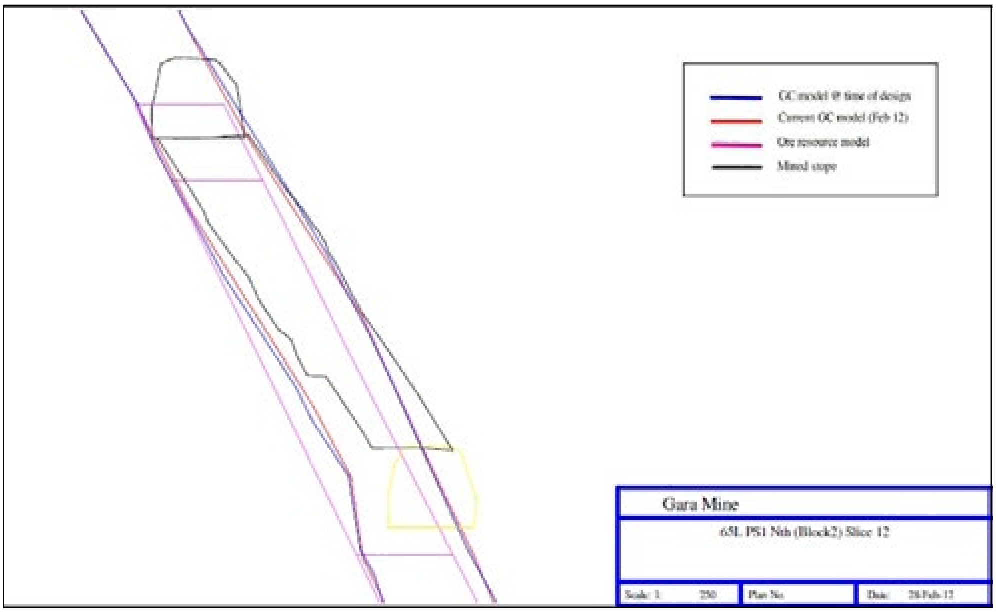

Figure 42: Ore body wireframes at different dates

Observations have shown that the scaling is not dynamic in nature and that the source of the scaling is not the crown or rib pillars, but the hangingwall of the ore body. Based on the information gathered, the scaling that occurred is a function of:

- the presence of the ramp structures that are exposed when the spans increase to two levels,
- dip back lengths created over two vertical levels that might be excessive for the rockmass in this area,
- the shallow ore body dips in this area creating the larger back lengths,
- difficulty to protect the hangingwall material with drill rounds due to the ore body and drive positions,
- the use of poor quality explosives and poor charging-up practices,
- the additional blasting to remove bridges that occurred at a stage when mining in the stope should already have been completed, increasing the time of exposure of mining operations within a stope;
- insufficiently detailed geological information on the presence of internal waste and the exact shape of the hangingwall contact within a stope prior to the design of the drill round.

In different degrees, all of the above contributed to the scaling that had occurred and all issues should be revisited and improved upon if this scaling is to be limited. Table 8 shows a summary of possible solutions to minimise the occurrences of scaling in order of importance.

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Table 8: Causes and solutions

Issue	Cause	Solution
Dip spans	Dip spans not only expose the ramp structures, but also increase potential for displacement. Note that spans of 60m back length will not remain stable indefinitely, especially in the shallower dipping areas.	**Reduce dip spans to maintain the 60m back length design**. This means that in shallow dipping areas, only 2 levels could be mined at any one time. This affects the mining of at least one 110L stope, but is possible to implement. The **implementation of backfill (paste or CAF)** might be critical at Gara to maintain small spans as single-level stopes will be excavated only before filling. **The stope span design must be revisited** using this information and the limited available geotechnical logs, while new holes will have to be logged to ensure continued adaptation of designs if required.
Shallow ore body dip	The impact of gravity in shallower dipped areas increases and requires a reduction in mining spans to maintain stability. Shallow dip means that separation of waste material from the ramp structure and other structures is prevented by cohesion on those planes only. As the dip increases, the stability is limited by the shear strength of those planes, which is cohesion plus friction. Improved stability is therefore expected at steeper dips.	**Refrain from mining shallow dip areas at maximum back lengths** (note stable stope above 65L at very flat dip).
Explosives and charging up	Poor explosive types and charging-up practices increase potential for damage and poor blasting results (bridges etc.).	**Change the explosive type** immediately to emulsion and ensure **good charging-up quality**. This is in the pipeline, but it should be fast tracked if at all possible.
Geological information	Insufficient geological information on the exact position of the hangingwall contact in the 'wavy' Gara ore body increases potential for blast-induced damage and scaling.	**Increase the geological drill pattern density** to allow better management of expected dilution and improve information to design better drill rounds. **Increase the distance ahead of the stopes that holes are being drilled**, so that detailed information is made available timeously.
Drill rounds	Drill rounds that penetrate the hangingwall material or result in excessive burdens in harder material, increase potential for blast-induced damage and poor blasting results (bridges etc.).	**Drill rounds should be based on sufficient (and the latest) geological information** only and must be **quality controlled** so that it can be ensured that the holes are according to the design.
Additional blasting	The need for additional blasting is mainly a time concern in that increased time spent in a stope increases the exposure to potential instabilities.	**Improved charging up, drilled burdens and explosive types** should remove this concern completely.
Ramp structures	Provides a plane on which displacement into the stope is possible and which allows easy scaling of hangingwall waste material.	**Support installation (cables) is a possible solution,** but requires long anchor support installed in all ore body drives prior to mining occurring. Cable anchor patterns will have to be designed to maintain stability. In this case, cable bolts should be sufficient to control the portion of the hangingwall that is currently scaling; approximately 2-3m.

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Issue	Cause	Solution
Ore drive position	The position of the drive determines the drilling round to reach the extremities of the ore body. Optimum placement against the hangingwall contact will ensure that holes are drilled parallel to the contact and reduce damage.	**Where possible, position the drives such that hangingwall parallel holes can be drilled**. This position is determined by several factors, none of which are controllable and include ore body width, ore body folding (wavy nature), and drill machine manoeuvrability. However, if other issues are resolved, the drive position concern can be managed.

LEARNING OUTCOME 3.2.3.2

ESTIMATE LIKELY DILUTION BY USING ROCKMASS CLASSIFICATION METHODS FOR GIVEN ROCK PROPERTY AND EXCAVATION LAYOUT CONDITIONS

CONNECTION 18
Refer to
"LEARNING OUTCOME 3.2.2.6" on page 117

LEARNING OUTCOME 3.2.3.3

DETERMINE REMEDIAL MEASURES TO LIMIT DILUTION FOR GIVEN ROCK PROPERTY AND EXCAVATION LAYOUT CONDITIONS

- Drilling accuracy can be increased by systematically surveying the drill holes and re-drilling the holes that do not meet required standards. Increasing the diameter will also reduce the deviation, but this might be a problem in narrow stopes due to an increase in spacing and burden.
- Stope designs could be improved by taking geology and rock mechanics information into account during the process of defining the stope boundaries. This requires systematic mapping/logging of joints, faults and shears near the stope boundaries and then either adjusting the stope limits, modifying drill plans or design reinforcement if needed. Decreasing the stope size should effectively result in lower dilution. However, lower level height will result in more development, which will be assessed in the financial appraisal. Dividing wide stopes into two separate stopes might have a positive effect on dilution, but would result in slower cycle times due to more slot raises required and delays due to increases in curing time of paste needed for two stopes instead of one. This design could also expose backfill from the first stope when the second one is being mined, causing extra dilution from backfill, which could neglect the benefits of having lower dilution from HW and FW.

The dilution related to over- and undercuts could be reduced by cable bolting, but the cost and effort should be weighed against the reduction in ELOS that might occur.

LEARNING OUTCOME 3.2.3.4

DESCRIBE, DISCUSS AND EXPLAIN METHODS OF REDUCING DILUTION

CONNECTION 19
Refer to
"3.2.3 ORE DILUTION" on page 121

A large number of precautions can be implemented to limit/reduce dilution. Some of these include:

- Improved data for design purposes to allow more accurate span design;

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- Improved design methods that are more appropriate to the designed-for environment;
- Improved mining method selection to ensure spans are limited when required;
- Improved mining method selection also changes excavation orientation, sidewall heights, drilling and blasting methods or application;
- Support installation to control rockmass movement.

EXAMPLE

Many of the issues discussed in Outcome 3.3.3 are taken up in the example provided below.

NOTE

It is suggested that this example is read to ascertain as much knowledge as possible on how to deal with dilution and not to evaluate the results, since a great deal of data had to be removed to shorten the example for the purpose of learning.

Dilution of the ore during mining has been reported as the single most problematic issue for the mine. This dilution of the ore manifests in the low gold grades recovered from working place drawpoints compared to the high in situ sampled grades and has been explained by the collapse of the waste rock surrounding the vein into the broken material during the mining of a block or drawing of ore from the block (shrinkage). Volumes of waste collapsing into the ore are being estimated from estimated tonnes drawn as no visual inspection is possible (dangerous to enter drawpoints or collapsed areas).

Mine information was studied in an attempt to identify trends in the occurrence of dilution within each mining block (i.e. whether dilution increases with time or with extraction rate, etc.) to identify processes driving the dilution. However, Figure 43 shows the only success and gives the total dilution calculated for each mining block over its life of operation as well as the average vertical height (vertical span) mined in each mining block. From this, it appears that all working places at dilution levels of less than 52% had mined to an average vertical height of less than 15m, while blocks with extraction heights of between 25 and 45, all yielded dilution levels greater than 55%. Only three outliers are visible in this ‘higher extraction height results in increased dilution’ trend:

- Block 401: No apparent explanation could be identified;
- Block 102: Appears to be situated in an area where the November ore body branches into several veins and dilution could be the result of a very wide, incompetent, altered rockmass, unstable even at small spans;
- Block SC800: This block has only been mined for one month and is apparently drawing material from block 100 above it, skewing the dilution figures.

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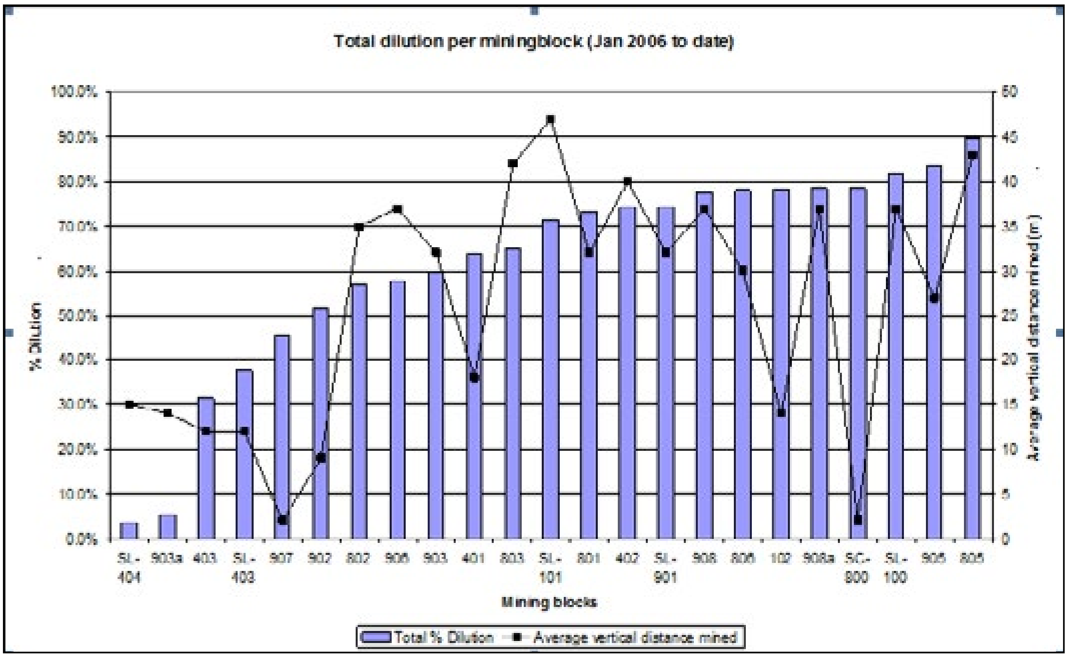

Figure 43: Total dilution per mining block (2006 to date)

Except for stating the obvious, the following factors could also assist in understanding the significance and seriousness of the dilution problem:

- Every tonne treated in the gold plant has a fixed cost. Increasing the amount of tonnes due to dilution increases this cost significantly. Table 9 below shows an example calculation and indicates that a 75% dilution (the dilution experienced on the mine since 2006) increases the tonnes to be treated in a 4:1 ratio, or a 400% increase in tonnes treated and therefore also plant treatment costs above the planned;

Table 9: Example of tonnes treated calculation

	In situ	Drawpoint	Unit
Grade (example)	20	5	g/t
Dilution percentage	75		%
Tonnes required to produce 1kg gold	50	200	Tonnes
Ratio of tonnes to be treated	1	4	

- A 400% increase in tonnes transported underground from working places to the adit entrances and delivered to the plant stockpiles via trucks is another result of the high dilution levels. In essence, all transport costs and man-time costs allocated to the transportation of these additional tonnes (with no added income benefit) impact on the mine's financial status;
- Drawing additional tonnes from a working place postpones the delivery of the total amount of gold to be gained from the same mining block and has a significant influence on the monthly income (gold produced) of the mine;
- With only a fixed number of man hours available to transport tonnes to the adit entrances and plant, the amount of gold that can be delivered in a day is indirectly proportional to the dilution percentage. This necessitates an increase in staffing levels to transport more tonnes to increase the gold delivered to the plant within any specific time span, increasing production costs (expansion of accommodation facilities at Aginsky mine is already being planned to facilitate an increase in staff levels).

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The issues mentioned above are depicted in Figures 44 and 45 where examples of the impacts of dilution on gold produced as well as production tonnes are shown. If the current production level of approximately 12 000 tonnes per month is maintained, it is possible to increase gold production by 50% through an average dilution reduction of 27%. It is therefore possible to achieve gold targets by winning small battles against dilution through small changes in the mining methods and practices.

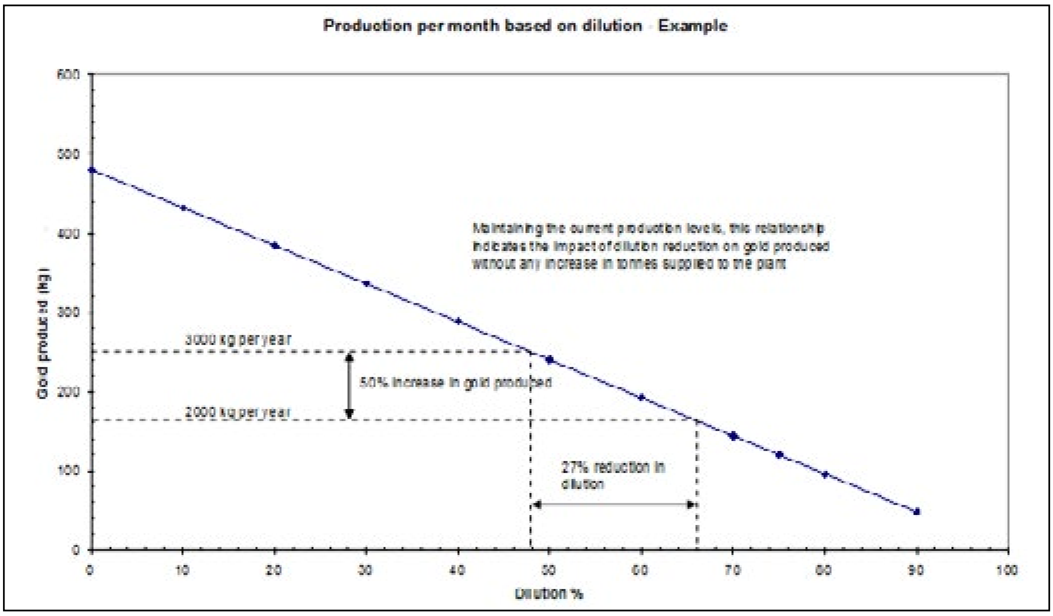

Figure 44: Impact of dilution reduction on gold production – example

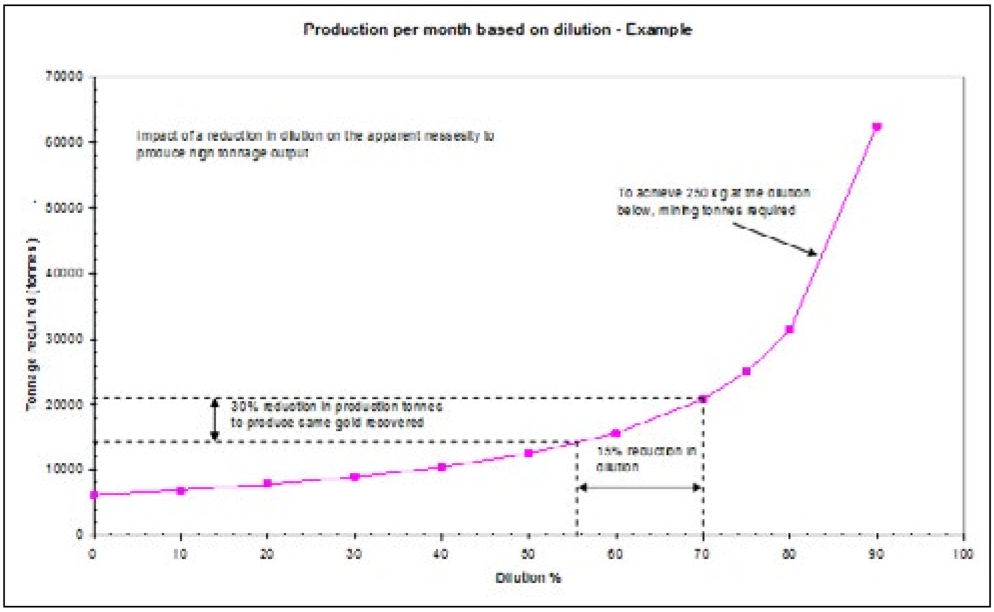

Figure 45: Impact of dilution on production tonnages - example

Figure 45 indicates the relationship between the percentage dilution and production tonnes required to produce 250kg per month (assumed target). This relationship indicates that the 'chasing' of high tonnages only may not be the best solution for Aginsky and that actual production tonnes can be reduced if small dilution improvements are made, reducing the number of employees required.

This analysis was not intended to explain the implications of high dilution levels, but to highlight potential additional benefits to be obtained

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for further investigations and implementation of the solutions to reduce dilution, as suggested later in this document.
Based on data gathered, Table 10 summarises the major issues and findings on the environment at the mine.

Table 10: Conclusions on the environment and observations

Issue	Summary with concluding remarks
Gold	On whole mine, 34% of the gold is mined from the 13.6% of the veins that are wider than 2.0m. Mining of the wider areas is essential to the success of this operation.
Ore	Thickness, dip and trend change over short distance. Low clay percentage is present, but ore compacts when left in situ for long periods. Variations provide difficult environment for a single mining solution.
Rock strength	Ore body is in most cases incompetent and clay content is in general low. The surrounding rockmass is weathered close to surface and competent in footwall, but incompetent in the immediate hangingwall and even appears less competent in the immediate footwall (some areas). Strengths vary between 122MPa for fresh andesite to ± 40MPa for the ore and altered rock material. The rockmass surrounding the veins is in most cases incompetent and should not be mined at large mining spans or subjected to raised stress levels.
Water	Areas vary between wet and dry, with large water inflow in areas close to surface and sometimes when major veins are approached. Water reduces the competency of the altered rockmass and should be avoided if possible. Good drainage of water away from the mining blocks is important.
Joints	Joint spacing of 8-17cm appears to be constant in all rock types indicating a very broken-up rockmass. Broken rockmass environments usually require more support installed or smaller span mining.
Quality	Rockmass quality confirms poor rockmass with Qore = 0.22, MRMRore = 6.9, MRMRhw = 5.2, MRMRfw = 19.5. Low rockmass quality confirms need for smaller span mining.
Stress	Some veins may be shielded from higher tectonic affected horizontal stress by the topography reducing the ability of the rockmass to self-maintain stability. Confirmation of need for more support or small span mining.
Seismicity	Should not have an impact on operation.
Secondary support	Large amounts of timber-based secondary support are installed in access excavations, with reduced-to-no support in good ground conditions and increased support installation near adit entrances and vein intersections. Huge labour-intensive timber support in poor condition areas suggests investigation into placing accesses in the vein footwall to reduce support need.
Off-vein excavations	Off-vein excavation conditions deteriorate when approaching veins, but are mostly very stable. Confirms the need for accesses to be placed in the footwall.
Hangingwall rock	Very altered hangingwall rock with smooth vein contacts in most cases. An incompetent hangingwall is present in all veins and ore bodies.
Separation and stress	Separation on some weakness planes suggests block movement of the hangingwall into mined-out areas. Limited sidewall deformation may indicate some stress effect on the on-vein drive sidewall. Stress appears to be impacting on the excavations, but may change from compression to tensile.
Separation	Separation on weakness planes also visible on veins indicating loosening of rockmass in stopes. Confirms possible tensile stress environment.
Vein intersections	Very poor conditions near vein intersections. Confirms the need for stable accesses placed in the vein footwall.

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Issue	Summary with concluding remarks
Alteration	The vein hangingwall rockmass is very incompetent. The amount of hangingwall alteration is directly related to vein thickness due to the mode of ore body creation and extremely poor conditions are encountered in vein intersection areas. Vein intersection areas will require special investigation and remedial action.
Large rocks	Large boulders blocking drawpoints. Instability does not only provide small rock pieces during instability, as observed in some drifts, but also creates large boulders blocking drawpoints. Confirms low (even tensile) stress environment when large boulders collapse uncontrolled into mined-out voids.
Small spans	Small span areas appear stable. Confirming that small spans may assist stability.
Tendons	Tendons are not assisting to create stable conditions as the rockmass is deteriorating around them and in such a way that they are totally ineffective. Bolting of the hangingwall is not perceived as an answer to the conditions. If a tensile stress environment is present, the tendons need to anchor beyond this zone and excessively long and strong tendons will then be required. Considered as impractical in this environment.

CURRENT MINING PRACTICE

1. On-vein excavations

On-vein tunnels were developed during the exploration phase of this operation and are being maintained where required and where rehabilitation is possible. Conditions in the approximately 3.5 x 3.5m tunnels are mostly poor and require substantial support installation in terms of timber sets. Set-type support is appropriate for dead-weight conditions as those found on these on-vein tunnels where the ore body and its surrounding rockmass deteriorate to an extent that the sets must carry the weight of the loose rock. These sets are very well constructed and of a high standard such as not often seen in mines that apply this support method. Set legs are linked to each other with steel pins and spragged against the rock walls for stability (Figure 46, Nr 1,2). Although this support method is effective and appropriate for the on-vein tunnels, it is extremely labour intensive.

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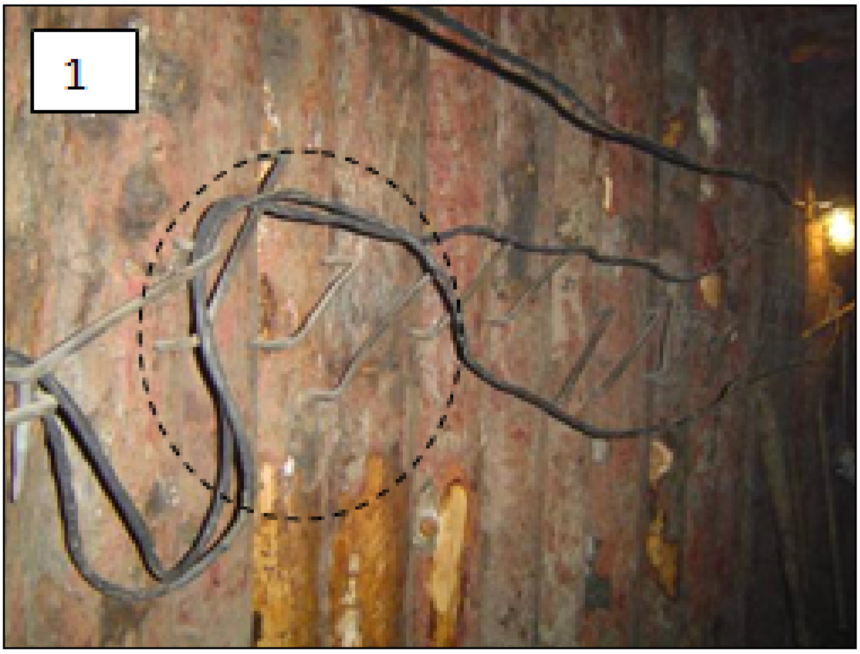

Figure 46: On-vein development

Other on-vein excavations visited consist of flanking raises and scraper drifts. Flanking raises are positioned to both sides of the planned mining block, are 3.8m x 1.5m in dimension, and are completely setted out with timber logs. This support method is extremely labour intensive, but effective in this environment, especially if considered that the hangingwall above the raises also appears to deteriorate during mining of the blocks. Set construction, as well as ladder construction, is very good and stable and did not indicate any signs of failure in the areas visited, even in blocks that have already been mined out.

Ground conditions in scraper drifts vary significantly and appear to be affected by the local rockmass quality and positioning of the excavation. In most cases where scraper drifts are being used, crown pillars are left between the mining blocks on successive levels, a practice that will assist stable stope conditions, especially when the lower block is mined after completion of the top block. On-vein drives/tunnels on the main levels also appear to be in better condition when scraper drifts are used

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instead of drawing ore through chutes directly into the tunnel, as in the 'chinaman box' area on 9 level. Scraper drifts appear to be supported with tendons (new 903a drift) and timber sets. On-vein scraper drifts are exposed to deteriorating hangingwall and ore body conditions and the utilisation of timber set support is appropriate. The use of off-reef (footwall) scraper drifts appears to have ensured more stable conditions and these ends are not expected to deteriorate as mining continues or require substantial support installation, but should remain stable with only resin grouted tendons to support the jointed rockmass.

2. Off-vein footwall excavations

Off-vein footwall excavations are limited to areas where the current on-vein tunnels' condition had deteriorated beyond effective rehabilitation. New tunnels are then developed around the problem area and back to the vein positions to access mining blocks. These tunnels are approximately 3.5m high and 3.5m wide and appear to be developed with square profiles. Limited scaling takes place during development, due to excessive jointing in the rockmass, but conditions vary from good to very good and support requirements are usually limited to areas where smaller veins or shear zones are intersected. This appears to be the preferred position for longer-term access development and should become the norm.

MINING METHODS

The feasibility studies suggested three mining methods and versions of these methods are being used on the mine. The suggested mining methods are:

- Sub-level stoping with narrow gauge drilling (using airleg rock drills);
- Shrinkage;
- Horizontal drifts with backfill (cut-and-fill): *This method never came into practice and was not completely investigated in the feasibility study.*

After the completion of a report in 2008, the slice-and-cave mining method is in the process of being implemented (Blocks 800 and 807), but progress is very slow due to administrative issues regarding 'permission to continue' with a new mining method.

Selection criteria include the gold grade, thickness, continuation and persistence of the ore bodies and the competency or stability of the host rock:

- Shrinkage is most suitable when:
  - the vein width is less than 3.0m (average vein width 1.5m);
  - the grade is between 20g/t and 50g/t;
  - the host rock is fairly stable; and
  - the veins are persistent and steeply dipping.
- Sub-level stoping is most suitable when:
  - the vein width is continuous and narrow (average vein width 0.9m);
  - the grade is lower than 20g/t;
  - the host rock is fairly good; and
  - the veins are persistent.

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- Horizontal drift-and-fill or the more recent slice-and-cave methods are most suitable when:
  - the veins vary in width between 2 and 5m;
  - the grade is above 50g/t; and
  - the host rock is less competent due to severe alteration in thick vein areas.

These criteria appear to have been simplified recently (due to dilution problems experienced) to:

- Slice-and-cave when:
  - the vein is > 3.0m thick;
  - the grade is very high;
  - the vein dip is flat; and
  - the rockmass is extremely incompetent.
- Sub-level stoping: All other cases, i.e. narrow vein areas;
- Shrinkage: No mining blocks appear to have been allocated for shrinkage.

The existence of selection criteria to allocate the most suitable mining method is a sound practice and should be continued and even refined based on successes and failures with the implemented mining methods.

A planned mining block at the mine typically has a length of 50m along the ore body strike and 50m along dip. The descriptions of the mining methods currently practiced at the mine and given in the sections below will describe the mining of such a typical mining block.

Shrinkage (refer to Figure 47)

- Flanking raises are developed with dimensions 3.8m wide by 1.5m high and are placed in the vein from one level to the next (50m);
- The equipping of each raise with timber set support, creating three compartments for men travelling, equipment transport and a development ore chute, is done as the raise is developed;
- Access ways at 5m intervals are made from the raise into the mining block being accessed;
- An ore transfer way is developed from the bottom level up to the scraper drift elevation and a scraper winch chamber is created;
- A scraper drift (2.5m by 2.2m), situated 4.5m above the bottom level, is advanced between the two flanking raises in the vein footwall and is supported with timber sets;
- Ore drawpoints are then made from the scraper drift into the bottom of the mining block;
- From the closest access way in the raises, these drawpoints are connected to create a working platform within the vein;
- The vein is mined by blasting the hangingwall of the on-vein working platform and the blasted ore is left in the vein to create the next working platform, with just enough ore being drawn from the drawpoints to create sufficient working space on the platform;
- The mining block is extracted by repeating the drill, blast and draw process described above cleaning the ore via the scraper drift into the ore pass;

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- Drilling is being done using airleg rock drills and since drilling straight up into the hangingwall above the drill operator is problematic (drilling rods get stuck), a method of ramping is utilised. This method entails the creation of two vertical faces, approximately 1.8m high, at the flanking raise positions. Working from blasted ore platforms, horizontal drilling is then practiced into the vertical faces;
- The block is mined by the upwards advancing of the working platform of blasted material to its upper limits, after which the majority of the ore is released by drawing the ore from the drawpoints.

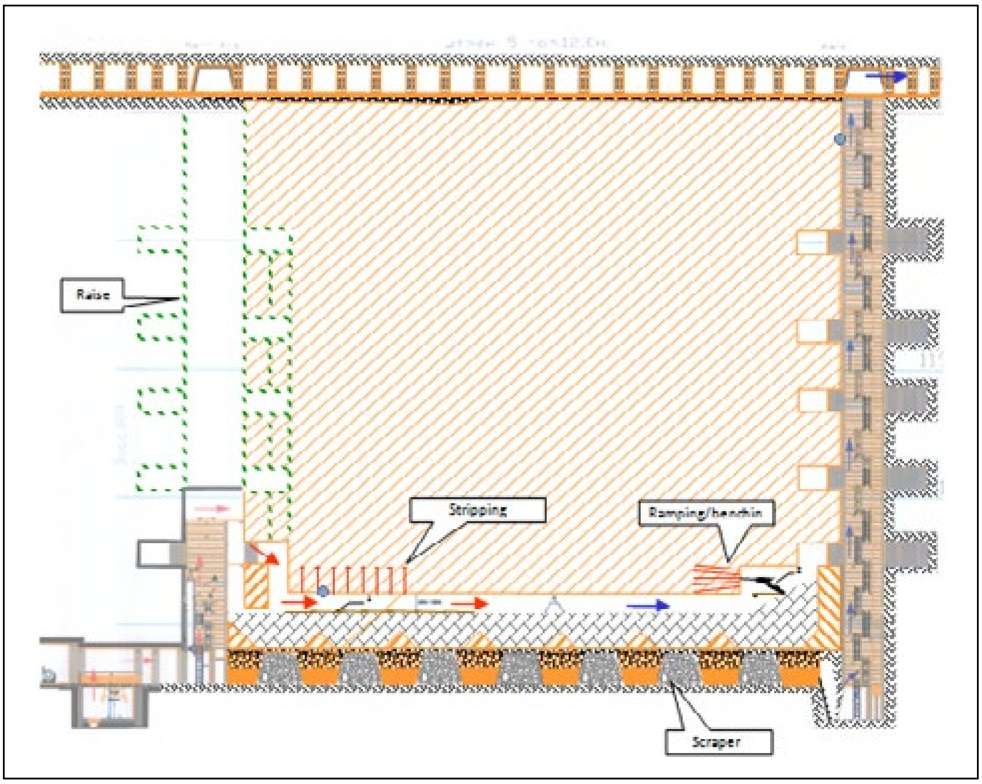

Figure 47: Shrinkage

Shrinkage is a well-known mining method implemented on several mines in the mining industry with great success, but is hampered in its success on the mine by the following issues:

- The incompetent surrounding rockmass: During drawing of the blasted ore in the shrinkage method, the sliding ore material creates significant friction against the surrounding rockmass, causing damage and pulling (sucking) the broken waste material into the mined-out space and mixing the waste with the ore;
- Undulating vein walls: The undulating vein walls increase sidewall friction and increase the potential for scaling and dilution;
- Clay particles: The presence of clay is not reported to be substantial, but cases have been noted where the clay presence had resulted in compaction of the ore in a shrinkage stope and secondary blasting was required to loosen the material, probably causing more damage to the surrounding waste rock and an increase in dilution.

1. Sub-level stoping (refer to Figure 48)
    - The blocks are developed in the same way as for the shrinkage method

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- The timbered and equipped service raises are put up on either side of the block (50m apart);
- The access ways are made out of the raise at regular intervals;
- An additional unequipped raise is developed on the vein and in the centre of the mining block;

- The scraper drift is developed with its drawpoints into the vein;
- Horizontal drifts (dimensions 2m by 2m) spaced 10m apart vertically along the dip of the vein are then developed from one flanking raise to the other;
- From the sub-levels, 110mm diameter holes are drilled upwards into the vein. The number of holes drilled depends on the thickness of the vein and they are charged-up and blasted;
- The centre raise is used as the blasting breaking point from where mining commences, while blasting occurs from the bottom sub-levels upwards in the vein, creating open spaces within the vein into which blasted ore from the top sub-levels is thrown, eventually sliding into the drawpoints to the scraper drift;
- The ore can be drawn soon after each blast is completed and therefore prevents the lock-up of high percentages of ore (during shrinkage approximately 66% of the ore remains in the stope).

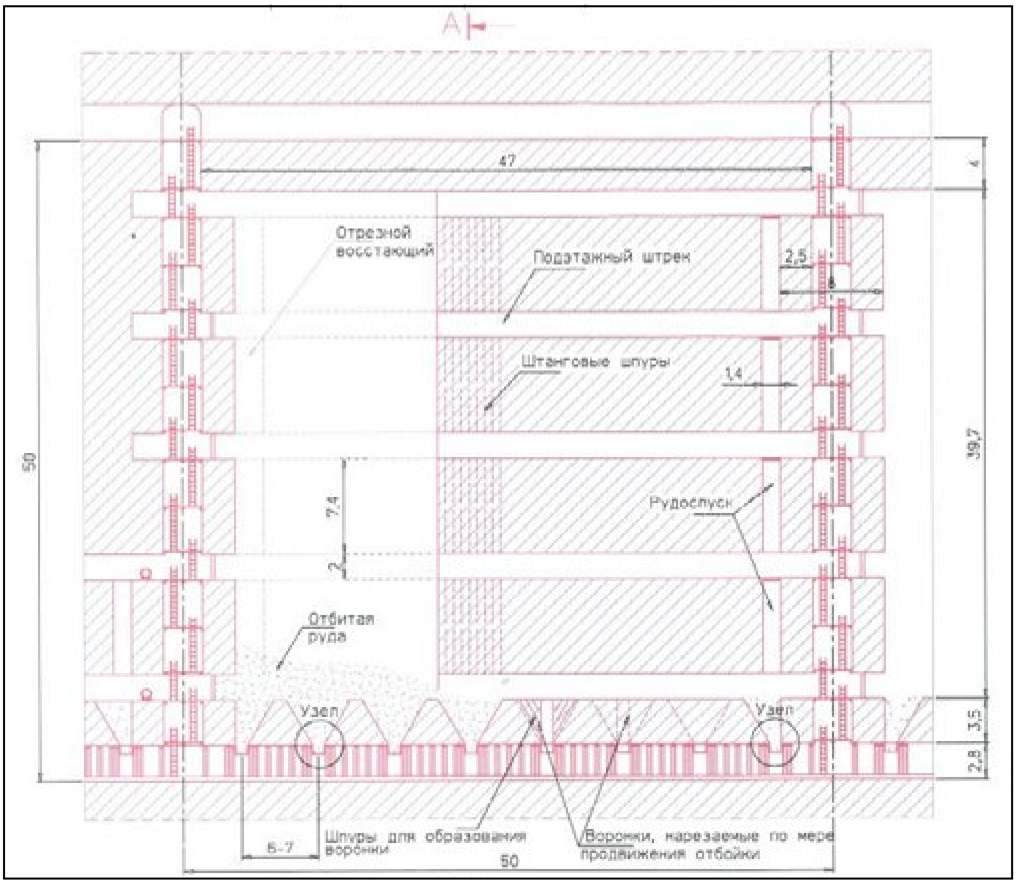

Figure 48: Sub-level stoping

2. Slice and cave (refer to Figure 49)

   - Currently, this method is still in the experimental stage and is planned for application to conditions for which the drift-and-fill method was originally devised. The backfill method is aimed at keeping the surrounding rockmass (below the area being mined) in place after the ore had been extracted (mining upwards in the vein), whereas the slice-and-cave method also proposes to support the collapsing rockmass around the vein, but in this case, the rockmass above the area being mined (mining downwards in the vein), thereby preventing dilution;

- On one side of the mining block, an inclined access is developed below the vein and in the footwall rock. This access is used for ventilation, timber transport and the movement of rock;
- From this access, the vein is intersected with closely spaced-off vein drifts;
- Starting from the uppermost drift, a horizontal drive is mined along the strike of the mining block and is completely supported with timber sets. If the vein is wider than 2.5m, drives can be mined next to each other, to extract the full width of the vein;
- Once the full width has been extracted on a particular level (slice), a timber mat is constructed along the entire footwall of the drive;
- Holes are drilled into the hangingwall and sidewall of the drive and blasted to cause the collapse of the surrounding rockmass and timber sets onto the timber mat on the floor;
- The whole process is then repeated underneath the timber mat floor, using the base of the timber mats as hangingwall to the new slice and installing the timber sets against this mat;
- Two or three slices would be done from each access drift before working from the next drift situated below the first.

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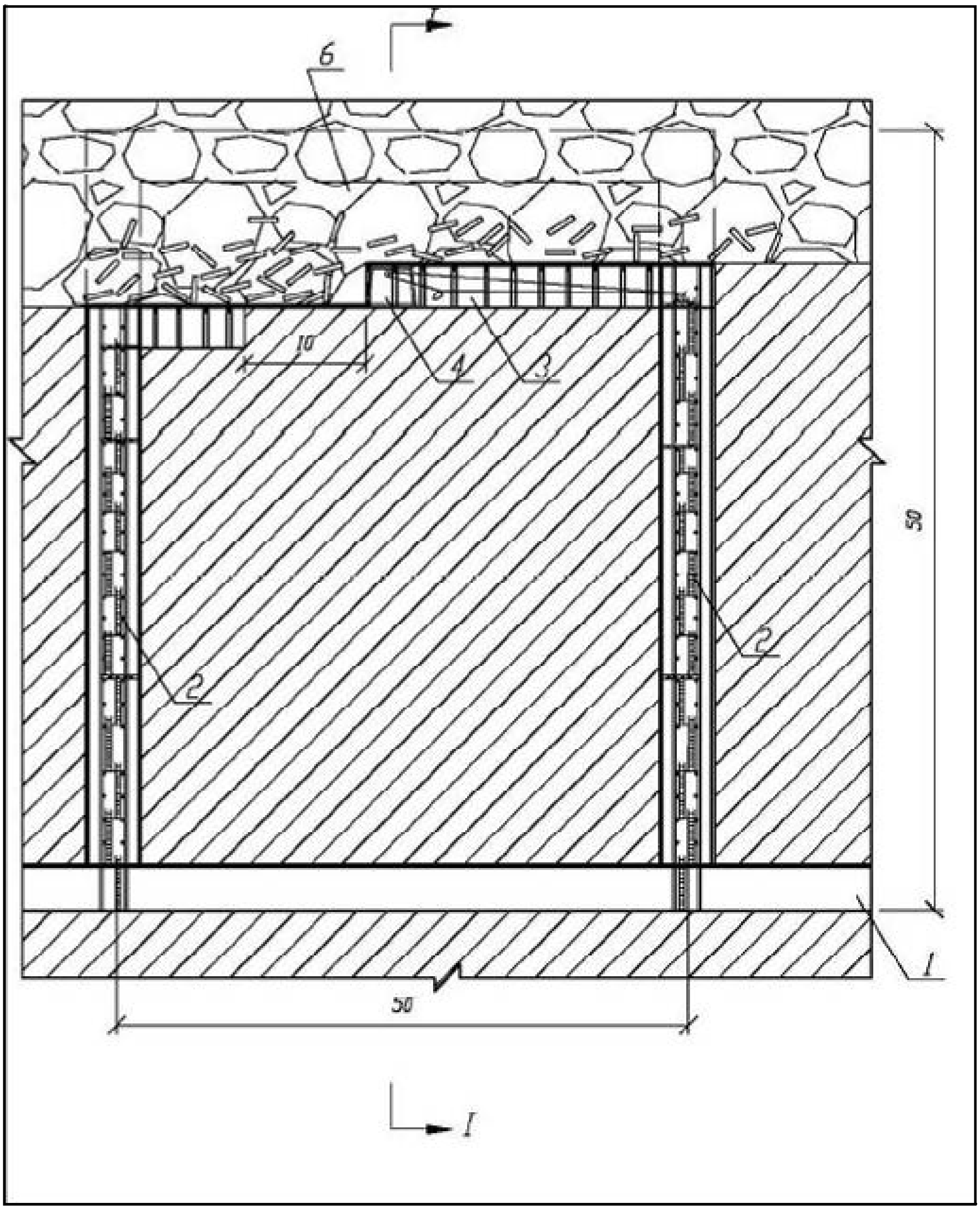

Figure 49: Slice-and-cave

EFFECTIVENESS OF MINING METHODS IMPLEMENTED

The effectiveness of the mining methods utilised to extract the mining blocks to date was analysed to evaluate the appropriateness of the mining methods to the conditions surrounding the ore body in each specific block. It is not being argued at this stage whether other methods would have been more successful, but the perceived success of the applied methods is merely stated. In the analysis, the following issues were evaluated for each mining block:

- Extraction percentage of the mining block/unplanned pillars left, which indicates presence of mining problems due to poor rockmass conditions;
- The need for additional development to facilitate mining of the block, which indicates the loss of access to the block, blocking of drawpoints, etc.;
- Completion of extraction following re-development, which indicates whether additional development solved the problem and

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whether

- Crown pillars are in place which, if not in place, indicates the possibility that ground conditions in blocks above this specific block are impacting on the mining results.

The data is shown in Table 11 and indicates that for:

- Shrinkage mining:
  - Average estimated percentage extraction: 82%
  - Re-development was required in 36% of the cases and was successful in only 40% of these and then only when new drawpoints were developed;
  - Crown pillars were not left in 43% of the cases, increasing the possibility that older and previously mined blocks could have an effect on mining results in the new blocks.
- Sub-level mining:
  - Average estimated percentage extraction: 70%
  - Re-development was required in 16% of the cases and was not successful;
  - Crown pillars were not left in 40% of the cases, increasing the possibility that older and previously mined blocks could have an effect on mining results in the new blocks.

At estimated 82% and 70% actual extraction ratios of the mining blocks mined by shrinkage and sub-level methods, compared to a planned 90% extraction of the ore body, indicates that the methods may not be appropriate for this mining environment. Additional development is of limited use to allow complete extraction of the ore body and in most cases does not contribute to higher extraction percentages. Normal practice in poor rockmass conditions is to separate mining blocks from each other with ore pillars (in this case crown pillars), thereby ensuring that poor conditions in one area do not affect mining of the other. This practice is not implemented well (crown pillars).
All investigations have shown that shrinkage of ore material is inappropriate for this ore body and its surrounding rockmass and is most probably contributing to the dilution problem. The shrinkage method was originally developed to utilise the ore to support relatively weak walls (walls that would collapse if exposed over large spans). In this rockmass environment, it is suggested that the ore's movement, as it is drawn from the bottom of the stope is allowing and assisting the breaking up of the walls and mixing of waste with ore. The process is briefly shown below in Figure 50:

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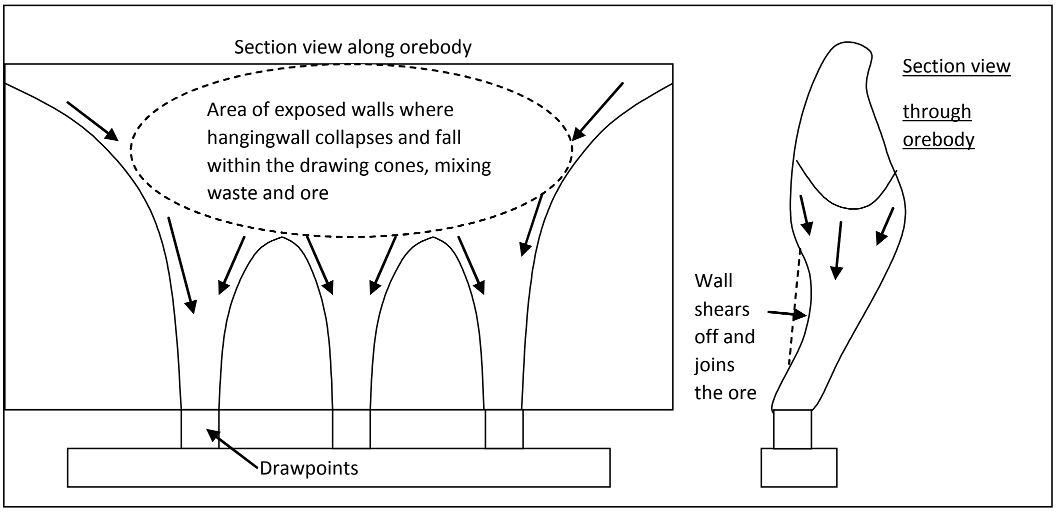

Figure 50: De-stabilisation during drawing of shrinkage material

This process appears to be worse for mining blocks that applied the shrinkage method than for the sub-level method, purely since ore is drawn from the sub-level stopes very quickly, reducing the 'assistance' to dilution given by the shearing of walls as blasted material falls rather than slides down the open stope. However, dilution levels still remain high due to stope walls that remain unsupported for long periods of time, allowing collapses of the incompetent and loosening hangingwall material into the ore during and after sub-level blasts.

To be able to completely understand the conditions and problems on the mine with regard to the ground stability, it was necessary to also investigate other special circumstances identified during the evaluations and which were anticipated to play a role in the dilution problem. These circumstances include mining in areas where different ore bodies intersect each other and the large variability in ore body thickness.

MINING IN OR CLOSE TO VEIN INTERSECTION AREAS

Since the ore bodies were created by a tectonic injection of fluids into fractures in the host rock, ore bodies converge on each other or branch into other ore bodies (Figure 28) at varying strike and dip angles. The consequences of this phenomenon are twofold:

- In the formation of these ore bodies, alteration of the host rock occurred. When ore bodies are situated at small middlings, this alteration appears to be exaggerated and ground competency is very poor and
- Mining at small inter-vein middlings (distances) normally creates mining difficulties as excavations interact with each other.

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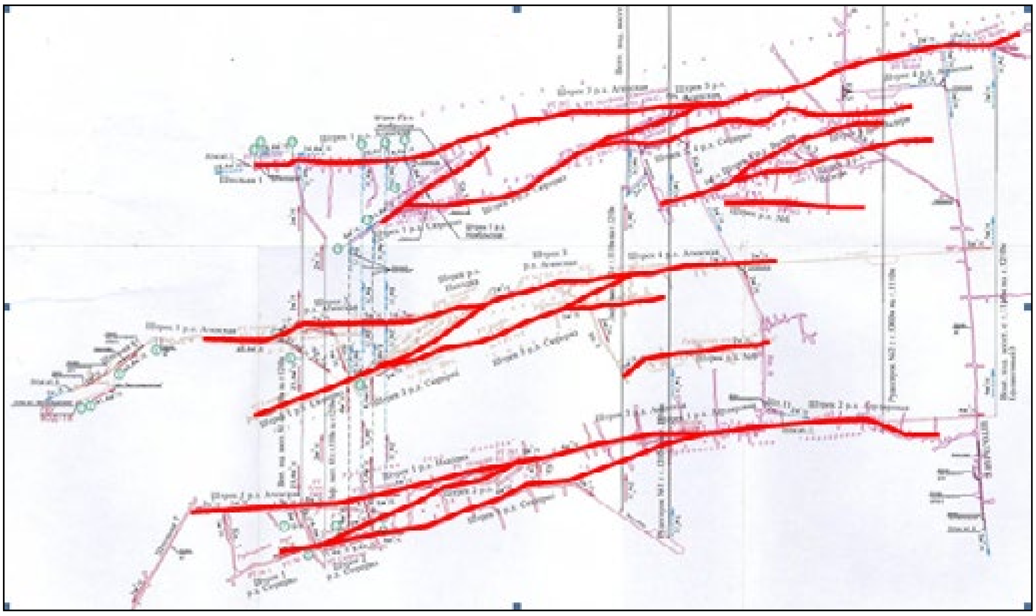

Figure 51: Ore body positions and intersections (plan view)

GROUND CONDITIONS AND ROCK INCOMPETENCE

From reports received, as well as through observations made (Figure 52) at various working places, it appeared that rockmass alteration appears worse at vein intersections and affects stope stability, mining difficulties and dilution directly.

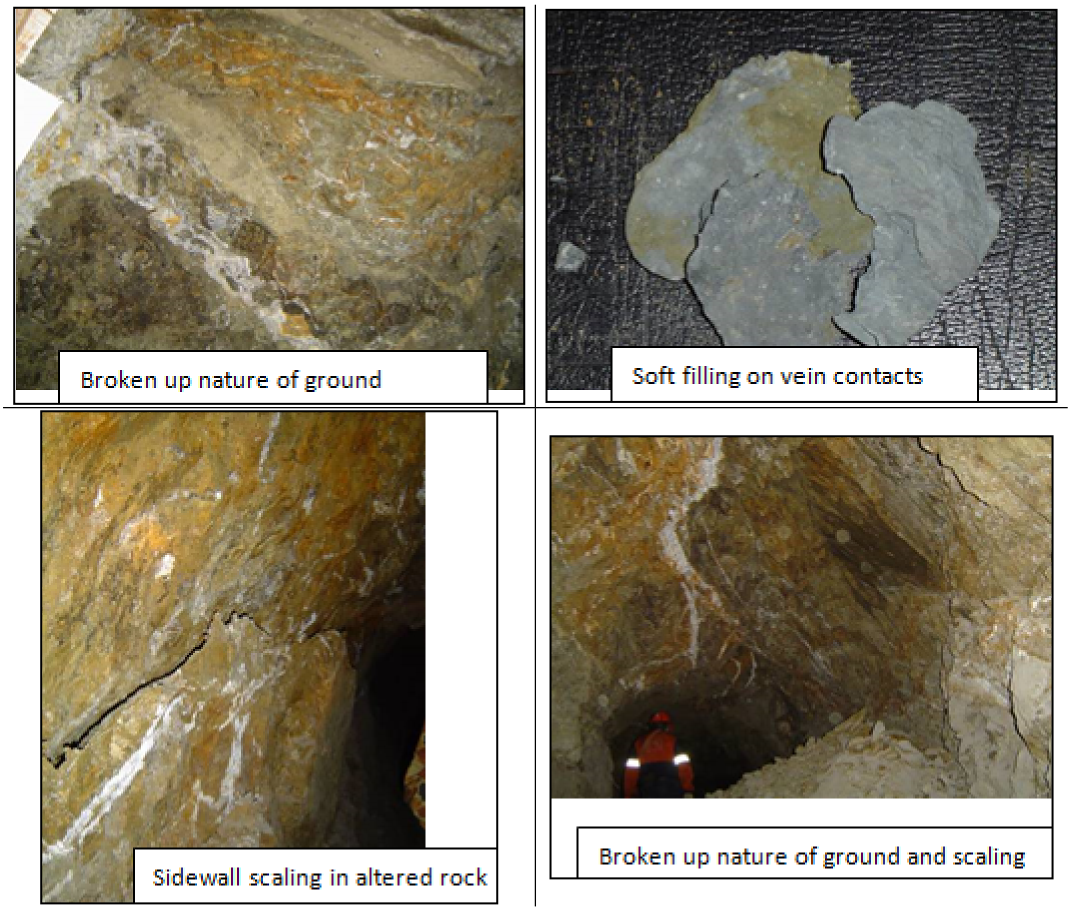

Figure 52: Ground conditions in vein intersection areas

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Mining excavation interaction is a major issue in most multi-seam environments and some ore bodies have been known to be sterilised by the mining of an economically more viable ore body situated at a small middling to the first.
The following factors are important to understand the impact of the high number of ore body intersections (10 evaluated) on the stability of stopes:

- the variability in ore body dip and dip direction of the ore bodies in relation to each other;
- the angle at which the ore body branches converge/diverge towards/from each other;
- the difference in dip direction between the converging/diverging ore body branches and the resultant position of the ore bodies with regard to each other;
- the mining sequence of mining within the two ore bodies (Figure 53).

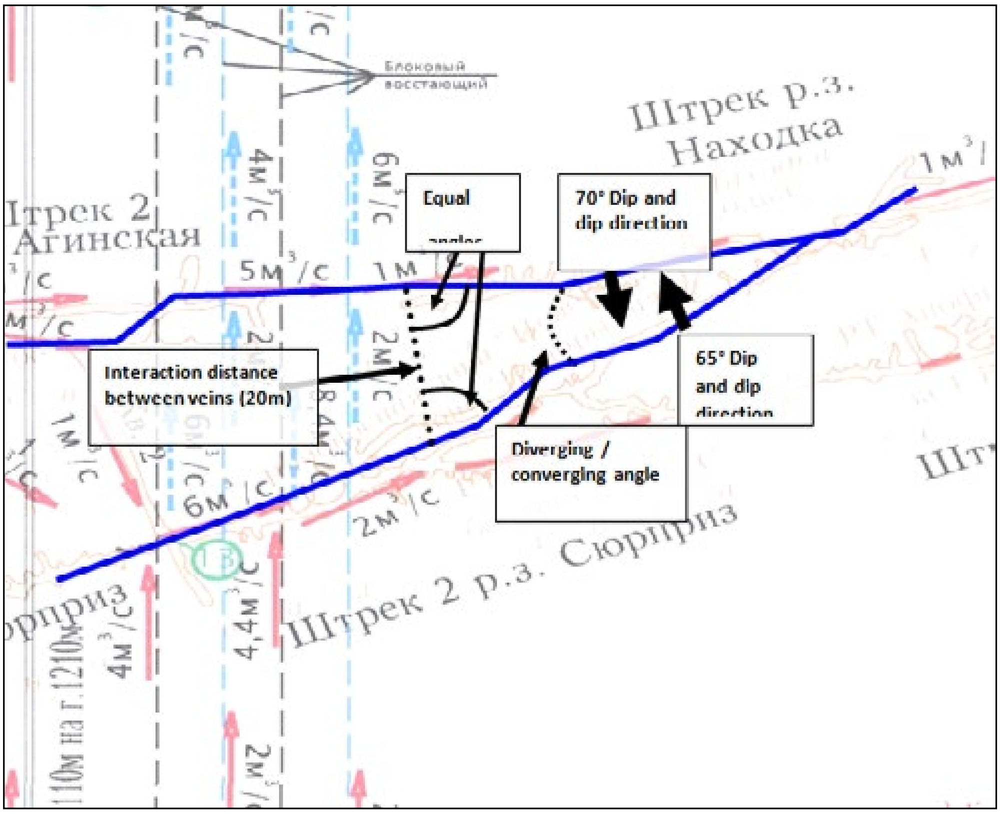

Figure 53: Measurement of angles for analysis of inter-vein conditions

Each of the vein intersection areas were evaluated using the factors given above and results are summarised below:

- Four out of five of the mining blocks where instability has been experienced:
  - had the other vein situated in its hangingwall;
  - were dipping in the same direction (east or west) as the other vein, i.e. the veins were orientated semi-parallel to each other;
  - had a converging/diverging angle between the branching veins of less than 25°, i.e. were exposed to each other over long distances;
- Of those intersections where the branching veins were dipping in

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the same direction (east), all the instability cases had the veins dipping within 0-15°of each other, i.e. veins are situated parallel to each other;

- All influence of the blocks on each other seems to terminate when the inter-vein distance reaches approximately 20m.
- The three blocks that have experienced no apparent instability showed that:
  - the blocks with opposite vein dip directions (east vs west) and a relatively low angle between the converging/diverging veins of 35° had a (large) 40° difference in their dip angles (distance between veins on dip increase quickly) and
  - the blocks with similar vein dip directions and a low difference in vein dip of 5°, had an angle between the converging/diverging veins of 60° (distance between veins on strike increase quickly).

This comparison allowed conclusions on the interaction of mining activities in the intersection area of the veins, suggesting that mining difficulties (and therefore also dilution increases) will most probably be experienced when:

- the other vein is situated in the hangingwall of the planned vein;
- the veins dip in different directions (east and west) and the convergence/divergence (strike) angle between the veins is ≤ 0°;
- the veins dip in similar directions (east) and the
  - convergence/divergence (strike) angle between the veins is ≤ 25°; and the
  - dip angles differ ≤ 15°;
- the distance of interaction will vary as the orientation of the veins vary, but should be a maximum of 20m along each vein (on the level) and could therefore affect a maximum of 40% of a standard mining block (50m wide).

Optimum mining sequencing (which vein to mine first) between the veins will mostly be determined by the support methods available:

- Independent of support method, mine the mining blocks where veins intersect first as exposure of this area to additional tensile stress levels due to other mining in the area will increase the instability;
- If backfill is available, mine the vein in the footwall first and ensure good fill quality is placed, where after the other vein in the hangingwall can be mined;
- If no backfill is available, mine the highest grade ore body first and then extract the other vein or consider forfeiting (writing off) the part of the vein situated within 20m of the other vein.

FAILURE MECHANISM

The dilution problem is the result of uncontrolled wall failure during the mining process and the subsequent mixing of the waste material with ore. In an attempt to solve this problem, it is essential to have a good understanding of the process or factors that lead to the failure.
All failures of the rock environment around mining excavations are driven by one of, or combinations of the stress field around the ore body and the rockmass strength. The relationship between these two parameters takes on different proportions or shapes in different mining environments. A high strength rockmass requires either very high stress levels

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to induce failure (e.g. deep level mine) or in low stress environments experience failure on joint planes within the rockmass due to gravitation only (e.g. shallow level mine). A low strength rockmass, broken up by weakness planes, can either fail under high stress levels (e.g. shale in deep-level mine) or disintegrate under the absence of compressive stress levels (i.e. pulled apart by tensile stresses).

The mining environment is unique to this operation in that it is situated within a very mountainous area, with (possibly) high horizontal tectonic stress levels and a very incompetent rockmass surrounding an incompetent ore body. At this shallow depth, the immediate response regarding stress levels would be that the advancing face stress levels are too low to induce rockmass failure. However, as shown previously, stress levels around the advancing faces increase with increasing spans and could cause some deformation around the advancing face. However, the dilution problem is not caused by a collapse of the advancing face, i.e. collapse of the vein itself in the upwards advancing stope, but rather of the collapse of the hangingwall and footwall into the mined-out area during mining or drawing of ore.

The following mechanisms of failure were postulated for the different areas:

- In a low horizontal stress field (highest veins), stress levels would be inadequate to assist with maintaining stability and allow the destruction of the broken-up and incompetent (weathered, icing cycled, water affected) rockmass around the ore body under pure gravitational forces;
- In a higher horizontal stress field (lower veins), mining of large open spans creates tensile stresses in the immediate surrounding rockmass that would effectively pull the rockmass apart and allow it to collapse into the mined-out areas.

The proposed mechanisms given above are based on the understanding of stress fields, but also on the following observations and discussions noted in other sections of this report:

- Extremely poor ground conditions, mining difficulty, presence of water and high dilution at blocks 100 and 800, where block 100 is wholly situated within 30m of the surface and block 800 partially. It has been reported that weathering close to surface probably extends to a 30m vertical depth below surface. The impact of surface conditions on the underground rockmass conditions is also confirmed by the deterioration of ground conditions in tunnels on all levels as the portal is approached.
- High dilution levels at blocks where shrinkage is performed and where sufficient spans have been created to allow tensile driven failure of the surrounding walls);
- Large boulders (from the vein hangingwall) within the drawpoints at 903a and 805 that do not indicate high stress failure, but rather unravelling of the hangingwall;
- Significant mining problems when two veins approach each other and where the creation of the tensile zone would theoretically be more pronounced due to stress shielding, and where ground conditions are much worse due to extreme alteration, such as in blocks 805, 903a, 901 and 101 ;
- Reports that dilution occurs as mining progresses and the shrinkage ore is being drawn and not only after the block is already extracted, indicating possible creation and growth of tensile environment;
- Several observations of opening-up or separation on joints or

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other weakness planes in the hangingwall of raises, drifts and stopes, especially the observation of significant separation on a vein parallel plane in the ore body hangingwall (General comment for Adit 9 Level, 1260 m.a.s.l.) indicating a possible tensile environment;

- The impact of vein thickness on ore and rockmass interaction during drawing of the shrinkage ore that provides additional loading on a loose rockmass;
- The impact of blasting practices on assisting loosening or breaking up of the surrounding rockmass, thereby aggravating failure and dilution.

Utilising Map3D models, minimum principal stress levels in the vein hangingwall were contoured as mining progresses in a shrinkage layout to gain an understanding of confining stress levels present within the hangingwall. These models were created to confirm or negate observations noted to date and should not be utilised for any further analyses, due to some critical assumptions made during the simulation.

Results for mining in the upper sections of the mine and therefore in the proposed stress shielded areas indicate that stress levels are only slightly affected by the mining, but are low and close to tensile and are therefore not assisting in maintaining stability (Figure 31)

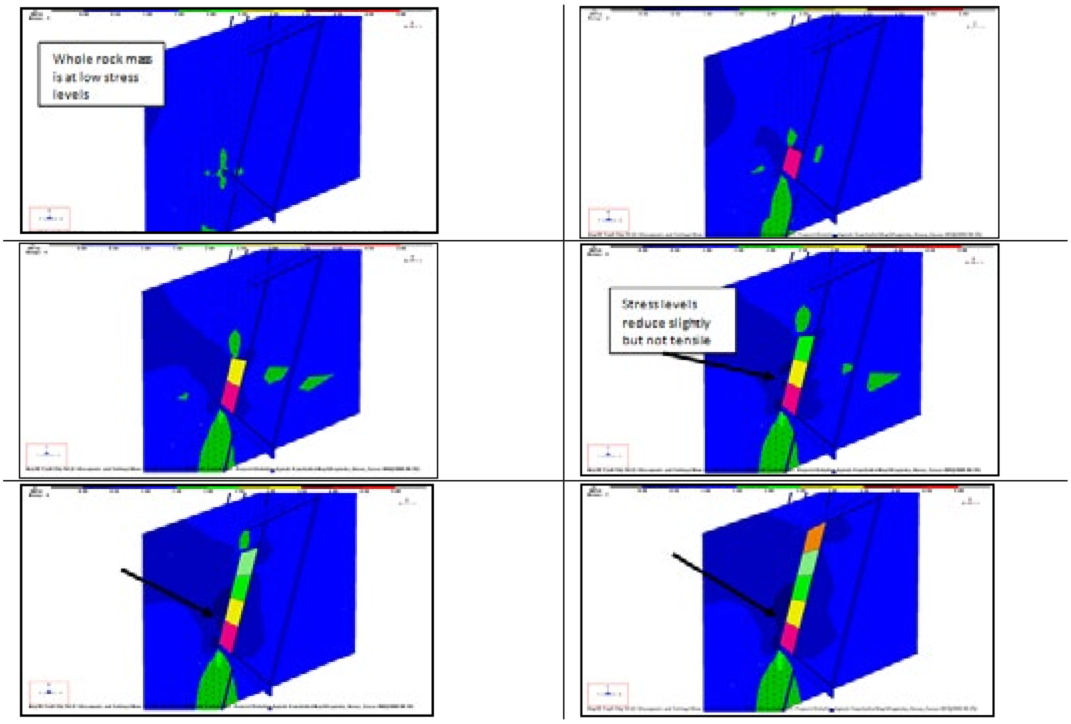

Figure 54: Stress changes as mining dimensions increase in 'shielded' areas

Mining in the lower sections of the mine (following the shrinkage and sub-level stoping methods of creating large open areas) creates significant potential for stress levels in the stope hangingwall to become tensile (Figure 55).

This means that the walls adjacent to the upwards advancing stope, as mining spans increase, experience an increase in the dimension of a zone of rock that is subjected to tensile stress, i.e. stress that pulls the rockmass apart. At the same time, the large number of joint and shear planes that exist around this 'tectonic shear zone ore body' and the altered nature of the ore body hangingwall (also the footwall rock, but to a lesser extent) present a rockmass that will be 'weak' under tension and that would easily collapse into the mined-out void.

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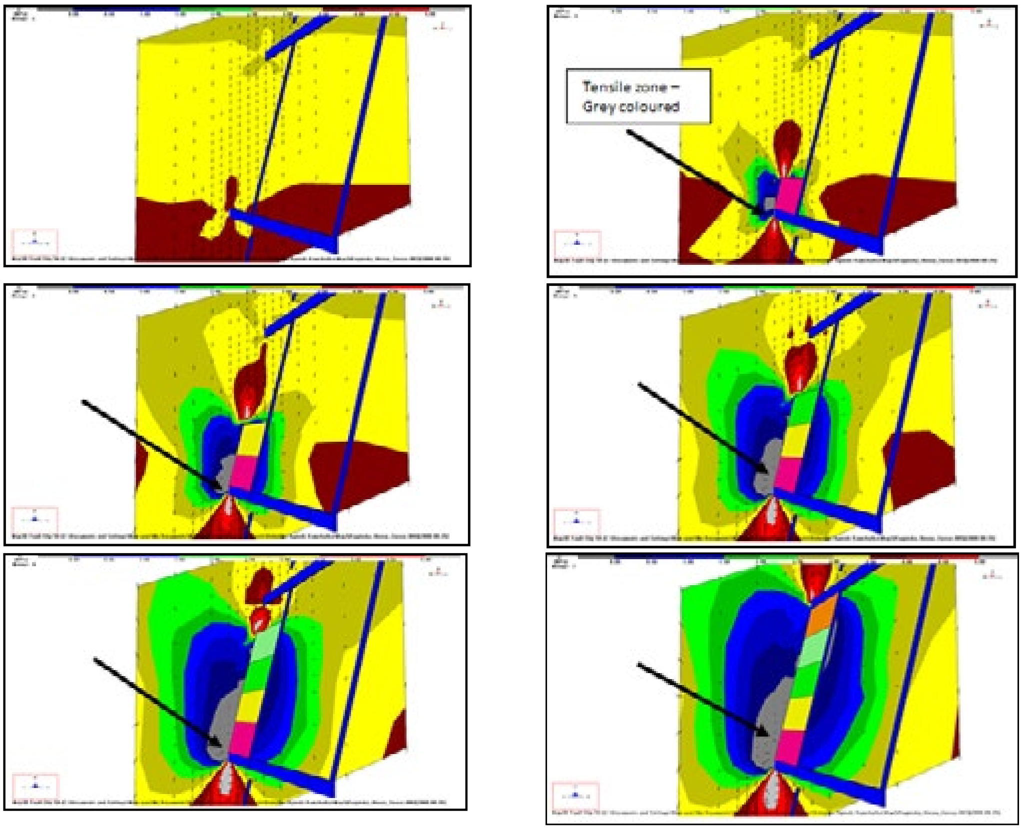

Figure 55: Increase in tensile zone dimensions as mining spans increase

The implications of these proposed mechanisms are:

- Mining spans must be kept small enough to ensure smaller, possibly manageable tensile zones by leaving vein pillars in place, i.e. follow a partial extraction strategy;
- Support installed in stopes should be able to keep the loosening rockmass in place or protect workings and ore material against probable collapses.

SOLUTIONS TO THE DILUTION PROBLEMS

It has been suggested that the dilution problem is driven by the instability of the incompetent rockmass surrounding the ore body in response to the mining practice followed, which in this case created a stress field within which the incompetent rockmass easily destructed. Since the geological environment cannot be changed, the rockmass' response to the mining can only be affected by changing the mining method or layout, thereby reducing dilution.

Short-term solutions

1. Ore control with 'grizzlies': Major ore passes in scraper drifts should be equipped with temporary 'grizzlies' (steel grid of specific grid size) to ensure that large rocks do not enter the ore pass system, especially since the larger rocks are usually waste material and should not be allowed into the ore transport system. This would also ensure that personnel are not exposed to 'open' ore passes. The creation of waste passes could also assist in handling these larger waste rocks so that mixing of waste is prevented.
2. Water control: Water control in tunnels is not a huge concern with regard to rail stability, but mention has to be made of the impact

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that water has on rockmass stability and that all efforts should be made to route water flow away from mining areas. Water flowing in main tunnels should be gathered into sumps and fed to other levels and out of the mine in positions away from mining blocks.

3. Drilling practice: Although the drilling pattern observed in the development end in the upper adit appears to work very well in the incompetent, weathered rock material, some concern exists that in more competent footwall rock, the break would not be optimal. The drilling of a more pronounced wedge cut in the centre of the tunnel would assist with breaking if advances seem to reduce in more competent rock.

4. Blasting practice:
    - Blasting practice should be adapted immediately to suite the incompetent rockmass around the vein. The use of a qualified explosives engineer (explosive supplier companies) to perform a full blasting audit, evaluate conditions and recommend optimum drilling patterns and explosive types is critical.
    - Secondary blasting practice on large rocks in drawpoints and scraper drifts to be changed to reduce the impact on surrounding conditions. One of the following practices should be followed:
        - The use of 'pillow pack' products (similar local product to be found) that direct explosive shock wave propagation in specific directions should be investigated (explosives engineer or manufacturing company);
        - Drill holes into the rock and utilise non-explosive products (local non-explosive swelling products) for large boulders within the drawpoints;
        - 'Hammer' technology is available to break larger rocks in the scraper drift.
    - Blasting of on-vein drifts or slices for both the sub-level stoping and slice-and-cave methods to implement the following practice in an effort to combat damage to the vein hangingwall:
        - Do not at any stage mine through the smooth hangingwall contact;
        - Wedge creation against the smooth hangingwall contact is possible when following this layout and potentially unstable wedges should be identified and removed by scaling/barring on a daily basis;
        - Commence with a development control system that would provide the drilling miner with an exact layout of the round to be drilled so that the contact plane is not breached by drill holes or explosive action. A possible system could be:
            a. Arrange for a survey generated direction line in the drift that will be painted on the hangingwall and extended daily by the miner;
            b. Instruct the miner to sketch the position of the smooth hangingwall contact onto a scaled template at the end of each shift;
            c. At the daily shift meeting, the supervisor will draw the position of the next drill round on the template and the miner will drill according to the drawn template layout on a daily basis.
    - Non-explosive technology:
        - Non-explosive techniques for blasting of on-vein production stopes should be investigated in the short term for implementation as soon as possible. These products are not similar to the swelling products mentioned earlier in this report, but are 'implosives' used with great success in areas where blast

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vibrations or entry periods are a concern, but where fragmentation, advance and throw are still important. It has been used with great success in several mining applications where specific needs existed. The aim with using this technology is to limit explosive action in breaking the rock by simply dropping it into the sub-level drift with minimal shock transfer to the surrounding rockmass. Velocity of detonation, which is directly related to the explosion shock and which is considered by the project team to enhance the dilution problem, is limited to $800 m.s^{-1}$ using this technology compared to $2800 m.s^{-1}$ to $4500 m.s^{-1}$ for explosives such as Grammonite and Ammonite.

4. Vein intersections: Mining of vein intersection areas should be planned in more detail and with more care as these areas are more prone to scaling and therefore also dilution. It is suggested that a specific process be implemented to handle these areas.
5. Planning and control protocol: Planning of mining activities in potentially unstable rock conditions should follow more stringent rules or systems than normal mining practices. The value of such a planning protocol includes the fact that areas where excessive dilution is possible are identified before mining commences and highlighted, ensuring that all practices available are implemented pro-actively to reduce the dilution to the absolute minimum. It allows lessons learnt under specific conditions within this complicated ore body to become easily noticeable during the analysis of information in these files and will assist in adapting mining or decision-making guidelines until optimum guidelines have been developed, thereby optimising the manageable dilution and accepting the unmanageable as a given for specific conditions within the ore body.
6. Mining sequencing: Mining sequencing is usually a major concern in environments where inducing stress levels are of a damaging nature, which normally refers to high stress levels. However, it has been shown that continued mining reduces stress levels, creating tensile zones and thereby increasing potential for failure of the incompetent rockmass into the mined-out areas.
7. Access development: Major access development that will serve to transport material and men to and from mining blocks to be developed within the footwall of the veins and a minimum of 10m (horizontally) from the vein intersection area, while the scraper drift should also be placed in the vein footwall. This will ensure stable access to a mining block and not jeopardise mining of a block due to deteriorating or collapsing access. It would also ensure that, should a mining block become impossible to mine, the re-development of the block can be conducted from stable footwall-based accesses, providing a higher probability of success with re-development.
8. Mining methods: Shrinkage stoping must be terminated immediately (except for drawing of ore from the already mined shrinkage blocks) based on the negative impact it has on dilution. Thick, high-grade areas should continue to utilise the slice-and-cave method. Thin vein areas (<2m) should continue using the sub-level stoping method, but it should be altered to provide lower probability for significant dilution during mining.

Medium-term solutions

1. Backfill application.
   - The application of backfill as a solution to the mining of this ore body has already been realised and suggested in the feasibility study This suggestion was probably based on information on the conditions provided in the exploration report and original feasibility document and indicates an understanding that the rockmass will not allow easy mining, but hoping that the methods recommended will provide adequate results. After completion of this

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evaluation, the same conclusion has been reached, but different to the feasibility document, more emphasis will be placed on this method, due to the now existing proof that the other methods have not succeeded in reducing or controlling the dilution of the ore body. It is also our opinion that the methods suggested in the project report in addition to the feasibility report do not address the critical issues affecting stability discussed in this report and will therefore not provide the required results.

- If the failure mechanism for the stoping environment is accepted as appropriate, the rockmass around the advancing mining will be exposed to an increasing zone of tensile stress and the rockmass would lose its ability to keep itself in place and would collapse into the stope. Other processes assist this failure, such as the poor rockmass environment, blasting and the drawing of the ore from the stope (as described in this report). One method to keep the loosening rock around the mining stope in place and prevent any deformation that could lead to unravelling is a backfill-driven method.
- The easiest backfill-driven mining method to implement in this environment is that of drift-and-fill, since the access layout would resemble that of the slice-and-cave, except that the vein would be mined from the bottom to the top.

2. Pillar systems (partial extraction):
   - Pillars are usually used in environments where low stress levels require the pillars to maintain rockmass stability by reducing tensile zone dimensions and providing active support to the surrounding rockmass. The same principle is suggested for this mine. The potential impact of pillars on the reduction of tensile and low stress zone dimensions in the vein hangingwall is shown in Figure 56, indicating that a reduction in the tendency for significant scaling into the stope is possible through a small-span layout, i.e. leaving ore in the form of support pillars.

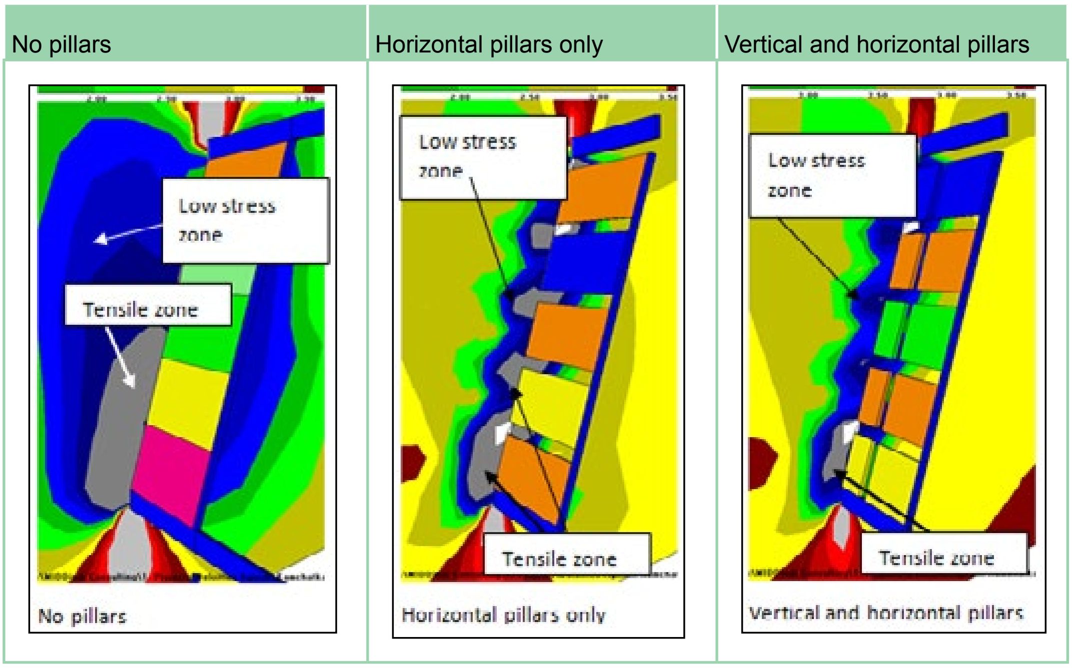

Figure 56: Impact of pillars on tensile stress development

- In operations where sufficient geotechnical data is available on the quality of the rockmass, standard methods exist whereby proper estimations of the stable spans between pillars can be determined. Since this type of information is not readily available at this mine, the short-term solution of small-span sub-level stoping can be developed once this data has been sourced, following standard open stope-span design methods.

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CONNECTION 20

Refer to
Paper 3.2 Outcomes
3.2.2, 3.2.3

3.3. CAVING OPERATIONS
3.3.1. CAVING AND CAVABILITY

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

LEARNING OUTCOME 3.3.1.1

SKETCH, DESCRIBE, DISCUSS AND EXPLAIN THE CONDITIONS REQUIRED FOR SUCCESSFUL BLOCK CAVING

In block caving, an undercut horizon is mined below an ore body with the intention of inducing the rockmass to fail, collapse and cave without further drilling or blasting. Drawbells are developed up from an extraction level to intersect the undercut, and the caved ground is drawn off via drawpoints at the base of these. As the ore is drawn off, more collapses from above and an ongoing cave are developed.

The method is best suited to massive ore bodies in weaker or more jointed rockmasses, where wide open-mined spans do not remain stable.

A typical block cave mining geometry is illustrated in Figure 57. Note that some different terminology is used for some excavations.

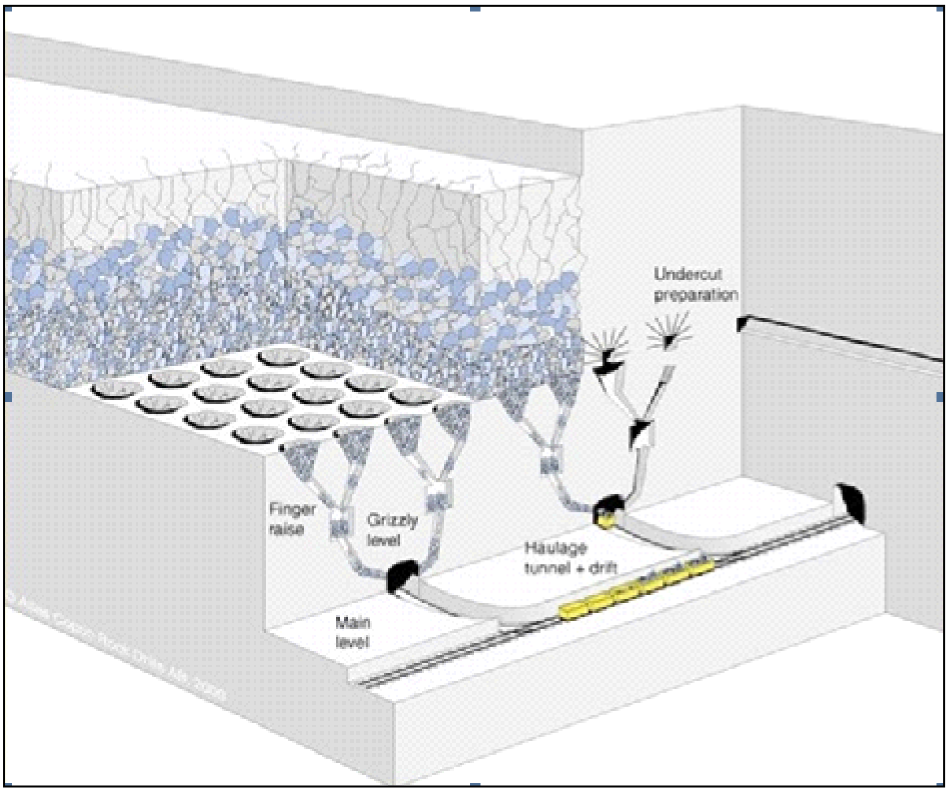

Figure 57: Schematic diagram illustrating block cave mining (after Infomine)

For successful block caving, the following conditions are important (and are indicated in Figure 58).

- For ongoing caving to occur, the width of the ore body must be greater than the maximum unsupported stable span that can be tolerated in the rockmass (see Outcome 3.1.3.1.3, below). It should not be possible for a stable arch to develop over the width of the ore body.

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- Caving is assisted by the *in situ* stress regime as well as gravity. A high stress field may drive failure and slabbing around the perimeter of, and above, the cave. However, the *in situ* stress field can also cause a cave to hang up if it results in clamping of joints or structures – for example, if caving up below an existing open pit where stresses below the pit are elevated.
- The size distribution of the caved material must be sufficiently small to pass through the drawpoint, without hang-ups. The method relies on primary fragmentation of the ore being achieved by natural mechanical processes. The initial sizes of blocks that break away from the cave back are controlled by geological structure (joints, bedding) as well as any stress-induced failure surfaces. These blocks break up further through abrasion and comminution (grinding) as they are drawn down in the cave. If geological structures are too widely spaced, the final block size may result in drawpoint blockages.
- Mudrushes are a hazard in block cave operations. If the height of the rockmass to be caved is great, there may be a large degree of comminution and production of fines in the cave. The total height of the block cave is therefore important.
- Rate of draw can determine whether caved ground re-compacts, or whether large voids open at the top of the cave, which can lead to sudden large collapses from the cave back, potentially resulting in air blasts. Re-compaction of caved rock occurs where drawpoint blockages are not cleared quickly, or where the undercut is completed too far ahead of getting drawbells operational.
- At some point, the cave will propagate upwards and interact with the ground surface, resulting in subsidence. It is important that any open pit on the outcrop is no longer operating, and no surface infrastructure will be damaged.

Laubscher lists 25 parameters that are critical for successful cave mining. His table is reproduced in Figure 59 (Laubscher D H, Cave mining – the state of the art, SAIMM Journal, October 1994, p 279-293).

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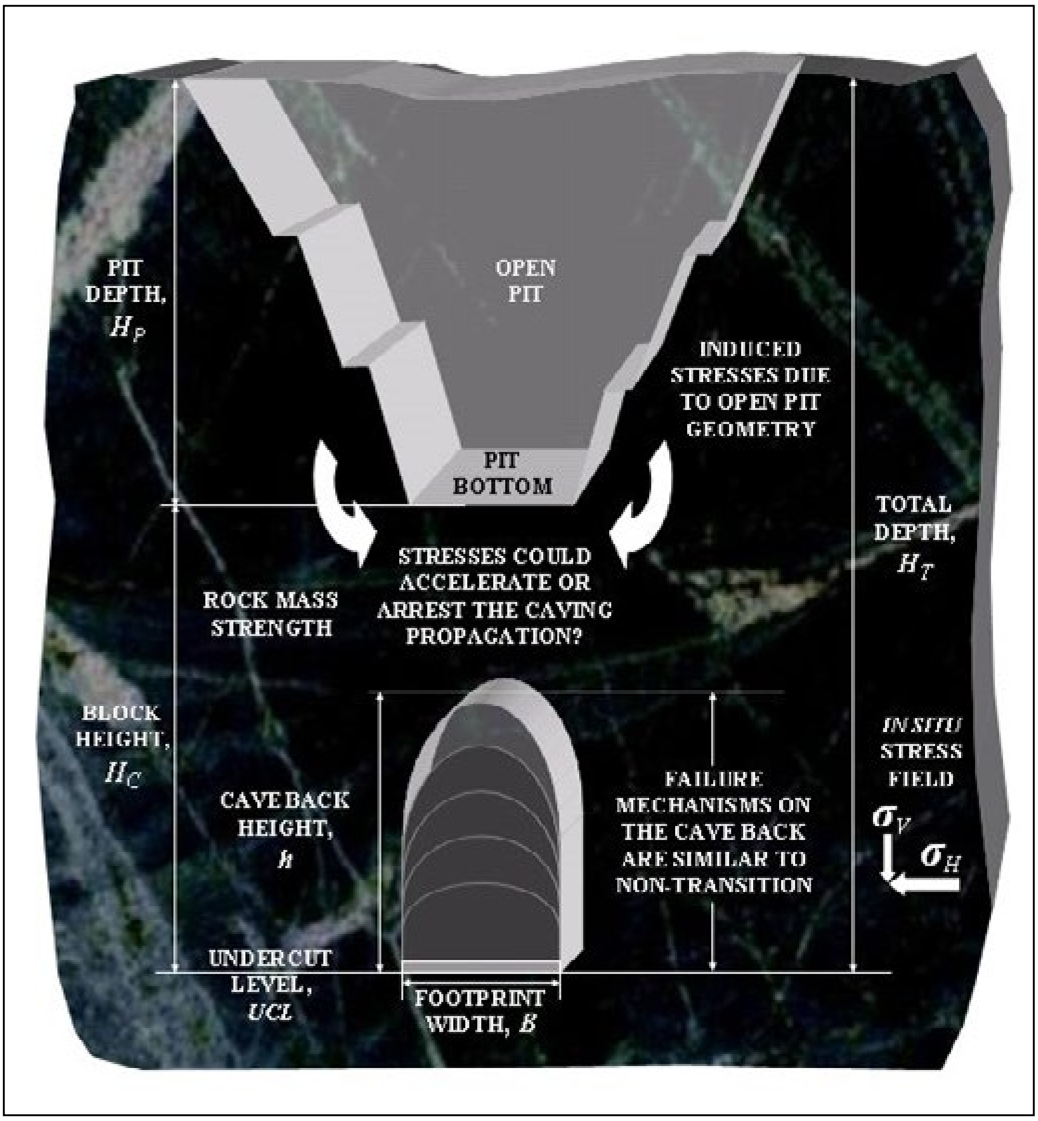
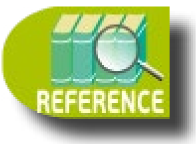


Figure 58: Key elements relating to a successful block cave (after De Beers Caving School)



- Laubscher D H, 1994, Cave mining – the state of the art, SAIMM Journal, October 1994, p 279-293
- Brown E T, 2003, Block Caving Geomechanics. JKMRC Monograph Series in Mining and Mineral Processing 3. Published by Julius Kruttschnitt Mineral Research Centre, The University of Queensland, Australia.
- http://www.jkmrc.uq.edu.au/Publications/BlockCavingGeomechanicsSecondEdition.aspx

Figure 59: Laubscher's (1994) table identifying 25 parameters to consider for cave mining

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LEARNING OUTCOME 3.3.1.2

SKETCH, DESCRIBE, DISCUSS AND EXPLAIN THE MECHANISMS ASSOCIATED WITH THE FOLLOWING TYPES OF CAVING:

- subsidence caving,
- stress caving,
- chimney caving

Subsidence caving

Surface subsidence is an almost inevitable consequence of block cave mining. In most instances, the cave propagates upwards and breaks through to surface, or into the base of an overlying open pit. As ore is drawn from the block cave below, so the depth of the subsidence trough on surface will increase. Even where a block cave is mining an ore body of limited height (e.g. Nchanga mine on the Zambian Copperbelt), the strata overlying the ore body may still collapse and have an effect on the surface.

Because of the occurrence of subsidence over block cave operations, careful consideration must be given to siting surface infrastructure as well as shafts sunk to access the mine. Some large ore bodies that are amenable to block caving cannot apply this method due to proximity to existing towns or infrastructure. An example of simple subsidence is shown in Figure 60. The effects are similar to those observed over long-wall coal operations in South Africa. The typical characteristics of the subsidence zone over a block cave are shown in Figure 61. The subsidence zone extends a significant distance outside the ore body boundary.

Figure 60: Surface subsidence due to block caving at Ridgeway Mine in Australia (public domain image), resulting in tension cracks, perimeter scarps, a subsidence basin and collapse of roads (http://www.groundtruthtrekking.org/static/uploads/photos/subsidence-from-block-caving.350x350.png)

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PAPER 3.3: CHAPTER 3

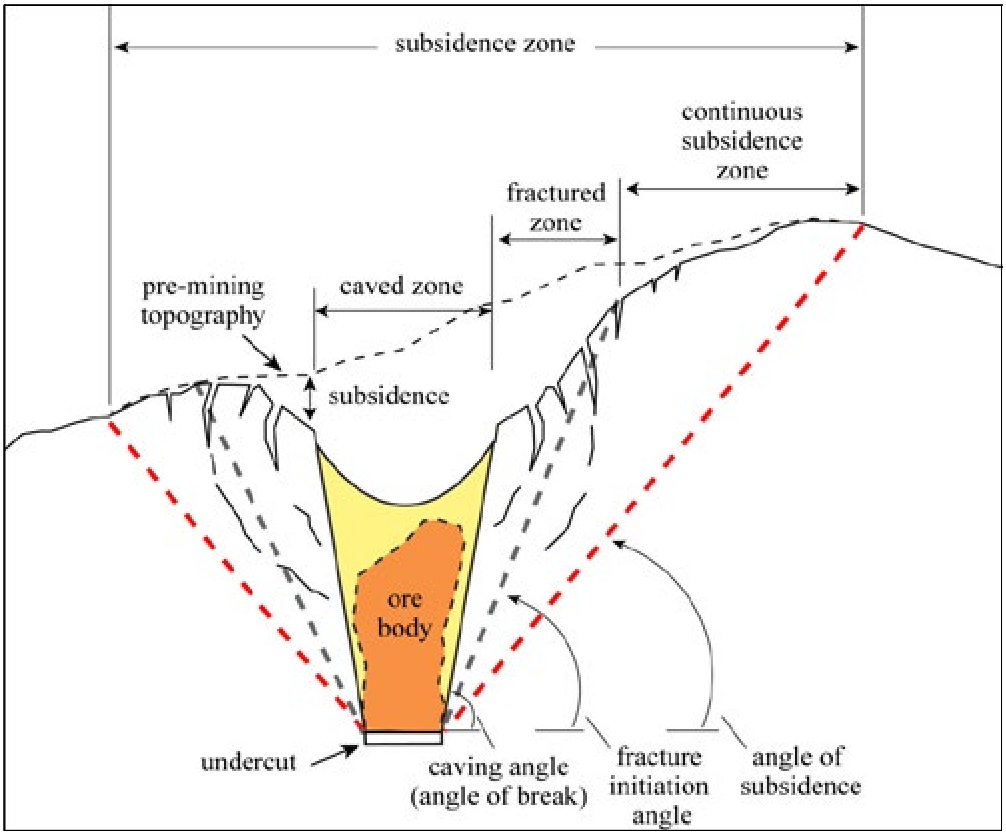
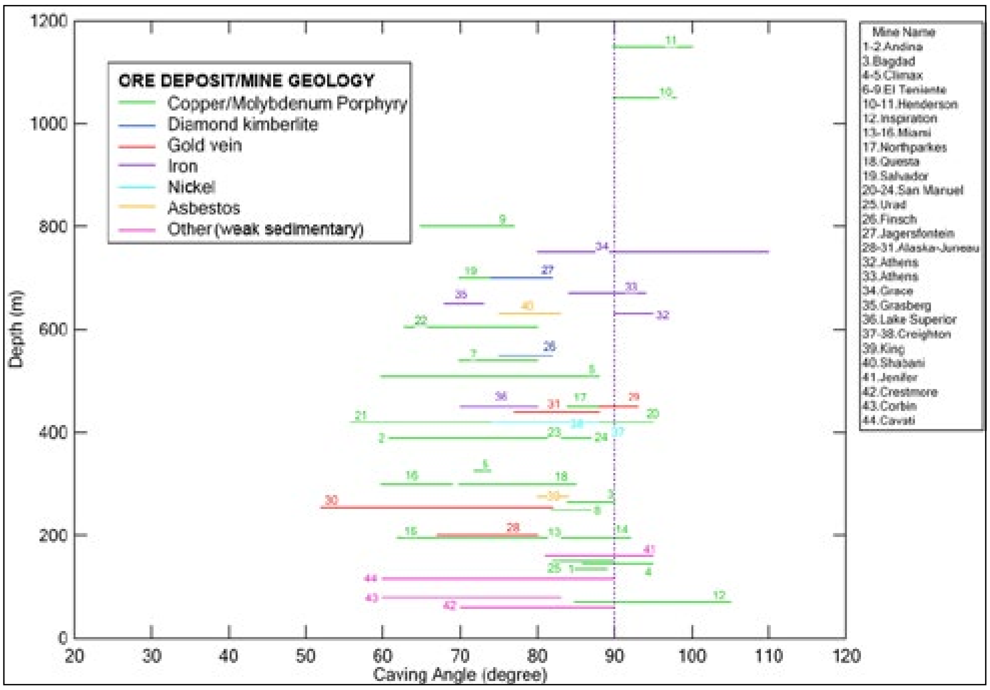


Figure 61: Block cave induced subsidence zones as defined by Australian Mining Consultants (Van As A 2003, "Subsidence definitions for block caving mines", Technical Report, Rio Tinto Technical Services, 59 pp.)



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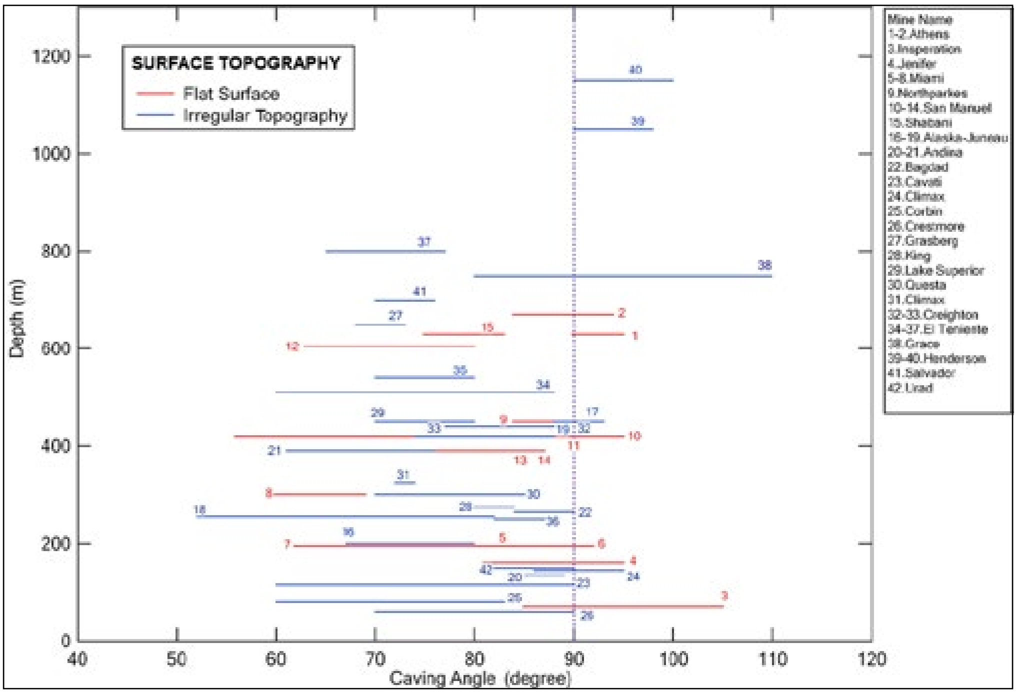

Figure 62: Recorded cave angles associated with actual block cave mines, plotted as a function of depth, and related to ore body type (top) and nature of surface topography (below) (After AMC)

Stress caving

Stress caving refers to the process by which the induced stresses in the perimeter or crown of the cave can induce failure and fracturing that assist in propagation of the caving process.

The development of a block cave results in the redistribution of the pre-mining in situ stresses. Typically, this comprises increased stress concentrations around the plan perimeter of the undercut, plus initial tensile stresses centrally in the back of the undercut, which assist in the initial development of the cave. After failure initiates, the cave front advances upwards with a typically arched shape. Increased tangential stresses are developed around the perimeter of this arch. A simple example is illustrated in Figure 63 and Figure 64.

Figure 63 shows the problem geometry for a 200m wide cave at 500m depth below surface. Two in situ field stress conditions are compared with K ratios of 1.25 and 1.75.

Figure 8 shows graphs of the tangential stress in the crown versus height of cave. For stress caving to occur, this tangential stress must exceed the rockmass strength.

Note that there is a cave height at which the tangential stresses reach a peak – partly because as the surface is approached the in situ field stress becomes less. This may result in tangential stresses being less than rockmass strength, in which case cave propagation will halt.

Note that when caving below an open pit there may be elevated stress concentrations below the pit that can assist stress caving. Figure 65 provides a chart to assist in predicting whether cave propagation will proceed or halt. The left axis plots Principal Stresses (S1-S3) on the cave surface normalised against horizontal in situ field stress at the depth of the cave (KγHT). The horizontal axis plots the cave 'aspect ratio' geometry (height h over width B). The path followed is a function of cave block

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height (Hc) below an open pit, and it can be seen that depending on both the in situ stress field as well as cave geometry, paths may be followed on which the cave may be expected to 'hang up'.

Note also that as the cave extends higher, it may tend to become narrower. As cavability is a function of span (correctly hydraulic radius, see below), a point may be reached at which the cave is narrow enough that a stable arch is formed, independently of considerations of stress.

REFERENCE

- Flores G and Karzulovic A (2002) Geotechnical guidelines for a transition from open cut to underground mining: Benchmark report. Report to International Caving Study II. JKMRC. Brisbane.
- Brown E T, 2003, Block Caving Geomechanics. JKMRC Monograph Series in Mining and Mineral Processing 3. Published by Julius Kruttschnitt Mineral Research Centre, The University of Queensland, Australia.

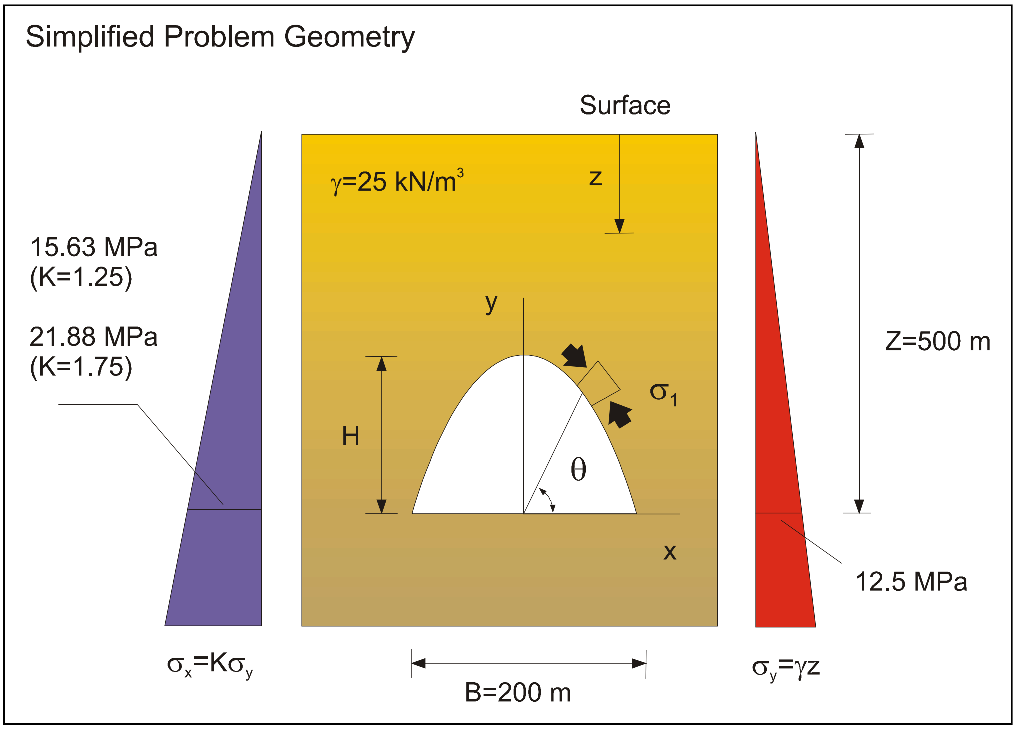

Figure 63: Problem geometry to examine tangential stress development in the back of a block cave (after De Beers Block Cave School)

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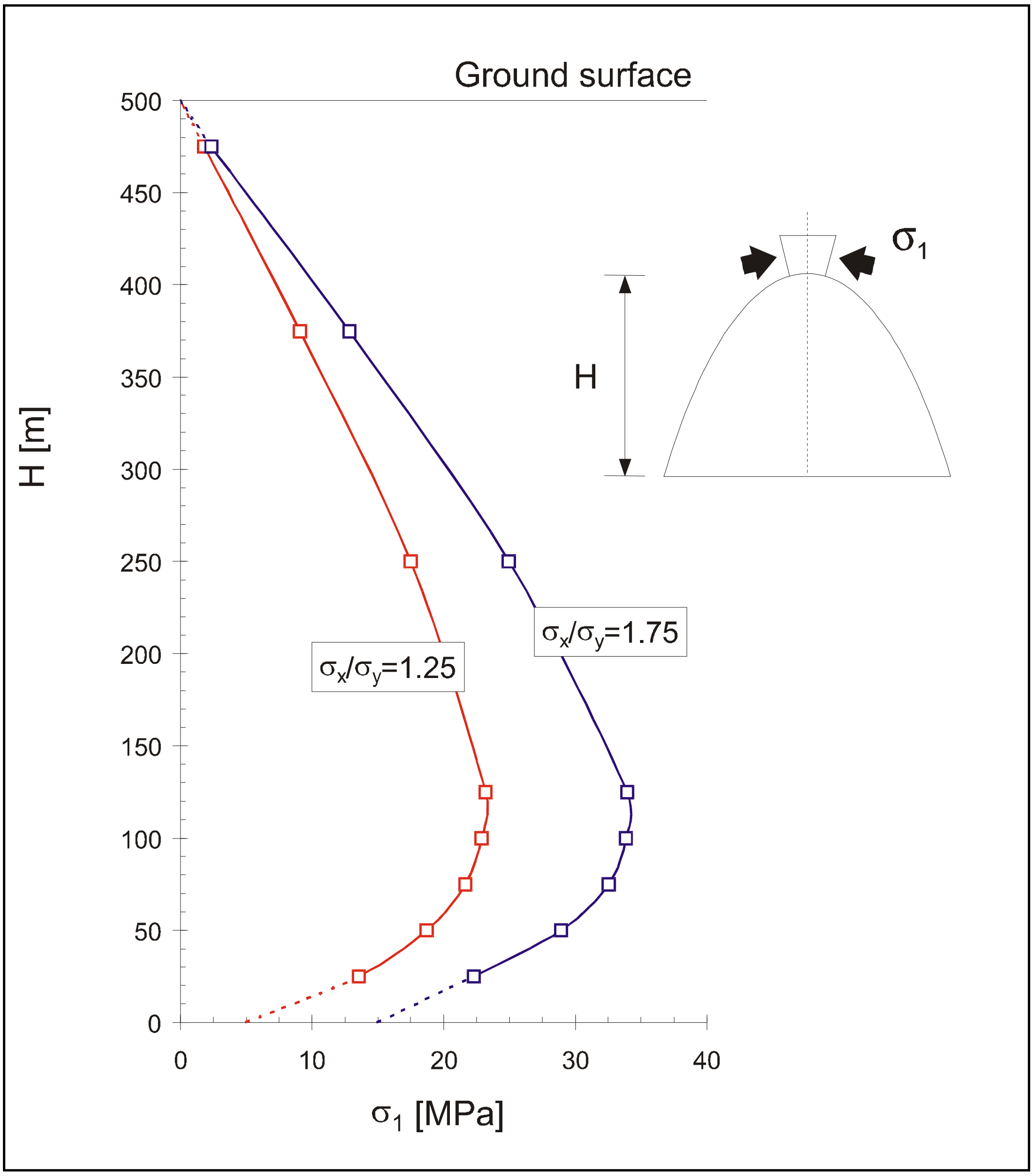

Figure 64: Calculated tangential stress in the crown of a block cave (geometry in Figure 63) as a function of cave height for two field stress cases (after De Beers Block Cave School)

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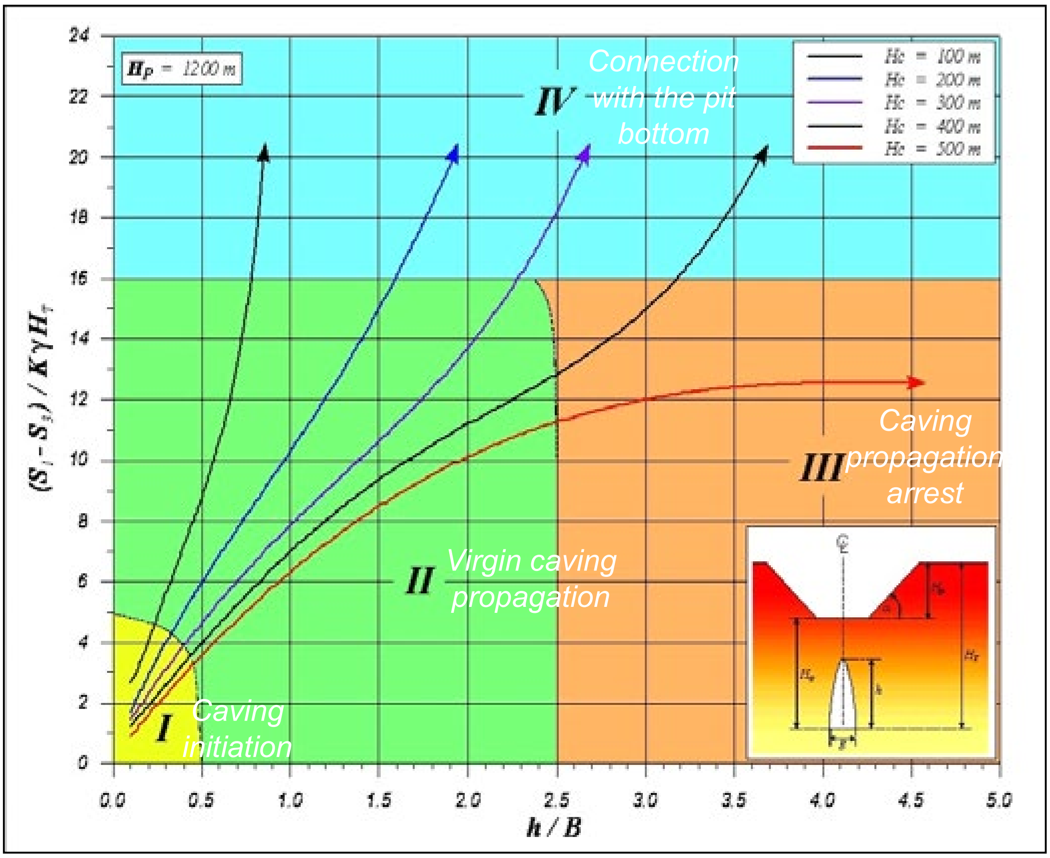

Figure 65: Prediction of on-going stress caving (after Flores & Karsulovic, 2003)

Chimney caving

Chimney caving, piping or funnelling is a description of the way a narrow cave path may be followed upwards from an underground excavation where caving or collapse occurs. It is illustrated (with other caving/subsidence types) in Figure 66.

Chimney caves may form in weak overburden material, often preferentially following (or drawing material from) particularly weak zones such as along faults or shears, where weathering has been intensified. They have been known to propagate upwards to over several hundred metres. Chimney caving has been experienced in coal mining as well as in sub-level caving and block caving operations.

A disastrous example of chimney caving occurred at Mufulira Mine on the Zambian Copperbelt in 1970. The chimney cave propagated upwards for 500m from an area of sub-level caving to daylight beneath a tailings pond. Tailings mud flooded a section of the mine resulting in 89 deaths. Further information on this case study can be found at:
http://www.tailings.info/casestudies/mufulira.htm, and

http://www.imwa.info/bibliographie/14_14_107-130.pdf

In a block cave, chimney caving is often associated with poor draw control – when too much ore is locally drawn from one drawpoint, or a group of close drawpoints.

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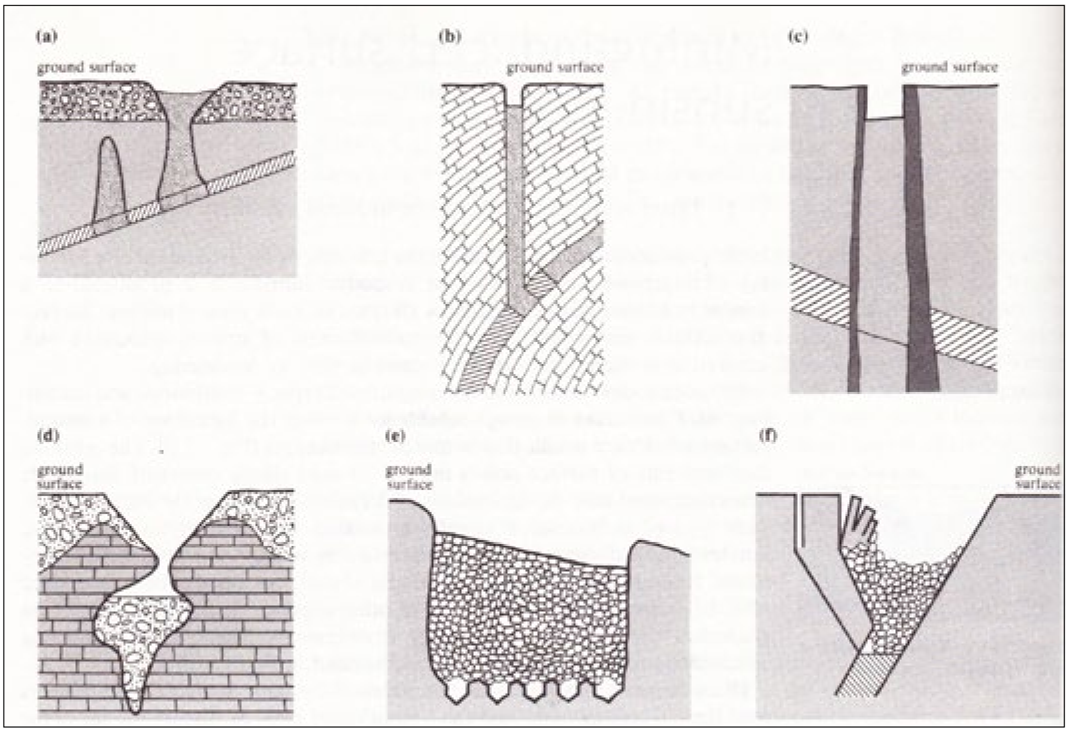

Figure 66: Types of discontinuous subsidence: (a) crown hole, (b) chimney caving, (c) plug subsidence, (d) solution cavities, (e) block caving, (f) progressive hangingwall caving (after Brady & Brown, 2005)

REFERENCE

- Brady B H G, Brown E T (2006) Rock Mechanics for Underground Mining. 3rd ed. Springer Science and Business Media, Inc. Downloadable via: https://miningandblasting.files.wordpress.com/2009/09/rock-mechanics_for-underground-mining.pdf

LEARNING OUTCOME 3.3.1.3

DETERMINE UNDERCUT REQUIREMENTS AND DIMENSIONS BY USING ROCKMASS CLASSIFICATION METHODS FOR GIVEN BLOCK CAVE SITUATIONS

A block cave is initiated by mining an undercut until its hydraulic radius exceeds a critical value. This critical value is a function of several factors, primarily the rockmass characteristics, but also in situ stress.

Hydraulic radius provides a better measure of the size and shape of the area required to be mined than simple span. Hydraulic radius is the ratio of area to perimeter length of the mined undercut area.

Undercut geometry may vary, but essentially comprises a horizontal cut through the ore body, mined using a series of parallel tunnels on one or more levels, typically on a 15m to 30m spacing. Undercutting may start at one side of an ore body and advance across it, or start centrally and retreat back to the ore body perimeter.

Broken ore is removed from the undercut as it is advanced; the result being that the ore above collapses to fill the void. Vertical propagation of the cave will then occur in response to continued removal of broken ore via drawpoints. Horizontal propagation occurs in response to undercut advance coupled with opening more drawpoints below the undercut area.

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Undercut requirements include:

- The area of undercut (hydraulic radius) required to initiate caving.
- The direction the undercut should advance.
- The shape of the undercut front.
- Choice of pre-, post- or advanced undercut sequences.

All of these are influenced by rockmass conditions and *in situ* stress.

Required undercut dimensions

The major factors that influence caving are discontinuity geometry and strength, rockmass strength, ore body geometry, the stresses induced in the crown of the undercut or cave, boundary weakening (e.g. shears around a kimberlite pipe), and the undercut geometry itself.

The area of undercut required to induce caving can be estimated using rockmass ratings. There are two similar methods, both making use of empirical stability charts derived from data from various mines worldwide.

Method 1: Laubscher's caving chart

Laubscher's chart is the most widely used method to assess cavability (Laubscher, 1990). It is based on Laubscher's Mining Rockmass Rating (MRMR) rating system, which is described in Learning Outcome 2.3.1.5. Note that Laubscher's RMR can be significantly different to Bieniawski's RMR, although the approach is similar. The final MRMR includes modification factors to account for weathering, orientation of jointing, induced stresses and blasting.

Laubscher's chart is shown in Figure 67. MRMR is plotted against the hydraulic radius of the undercut. Data from a range of mines permits stable, caving and transitional zones to be defined on the chart.

To identify the required size of undercut required to initiate caving, the rock engineer first assesses the range in MRMR for the rockmass comprising the ore body and country rock. This range is then plotted on the vertical axis on the chart and the appropriate hydraulic radius is read from the horizontal axis. Actual dimension ranges can then be calculated.

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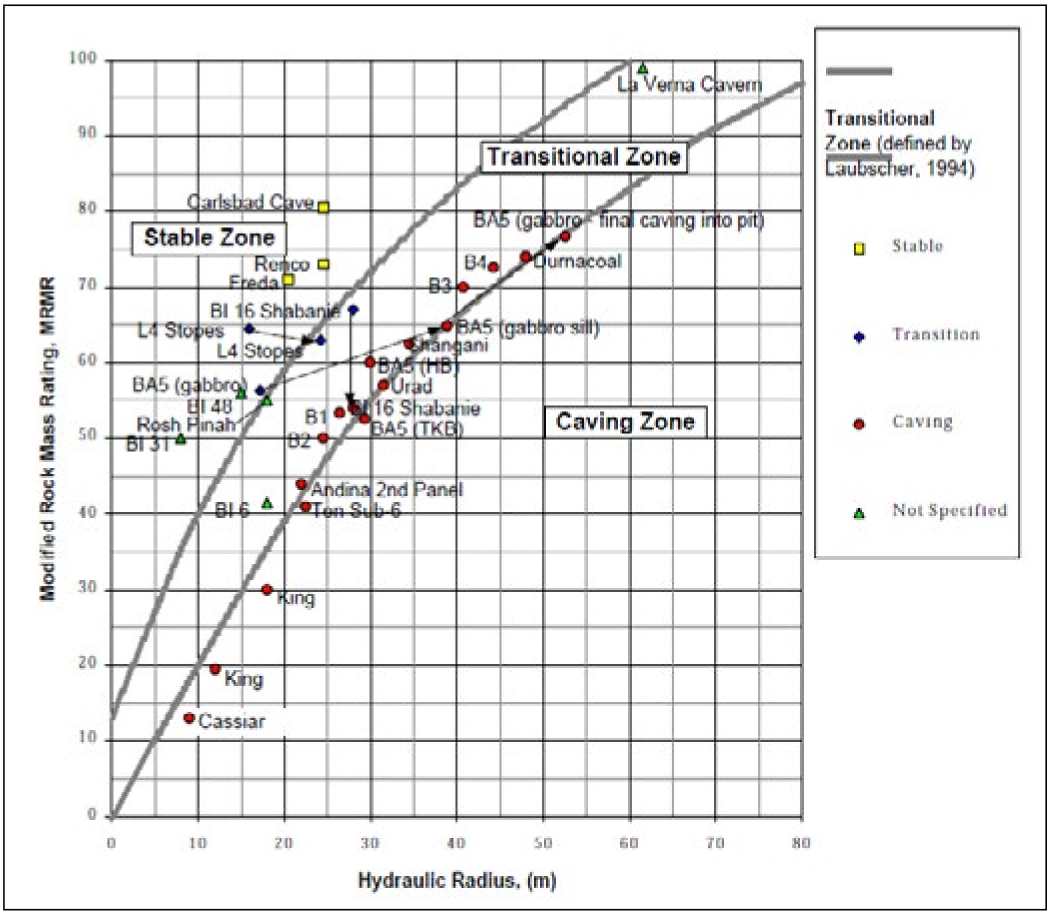

Figure 67: Laubscher's Caving Chart (after Bartlett, 1998)

Method 2: Mathew's stability graph
This is similar in concept to Laubscher's chart and is based on a stability number, N, which is derived from rockmass ratings calculated using Barton's Q'. Note that Q' assumes that both the joint water reduction factor and stress reduction factor are set to equal one in the standard Q calculation (see Outcome 2.3.1.3).

The stability number is defined as:

$$N = Q' \cdot A \cdot B \cdot C$$

Factors A, B, and C can be derived using the chart in Figure 68, and account for:

A – Represents the ratio of rock strength to stress in the back of the cave.

B – Accounts for the angle of dip of the excavation surface which stability is being assessed (typically zero dip over a cave, but could be inclined when applied in open stoping operations).

C – Accounts for the angle of jointing that intersects the cave back or undercut.

Stability number N is then plotted against hydraulic radius (sometimes known as shape factor) to create the stability graph. There have been several versions of this, as more data has been gathered by various researchers. An often-used version of the stability graph published by Hoek, Kaiser and Bawden (1995) after Potvin (1988) and Nickson (1992) is shown in Figure 69. A later version plotted on log-log axes by Mawdesley

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(2001) is shown in Figure 70.

To design an undercut, a rock engineer would take Q' ratings for the ore body, estimate modification factors to calculate N, then plot this on one of the two charts to identify a hydraulic radius at which caving would occur.

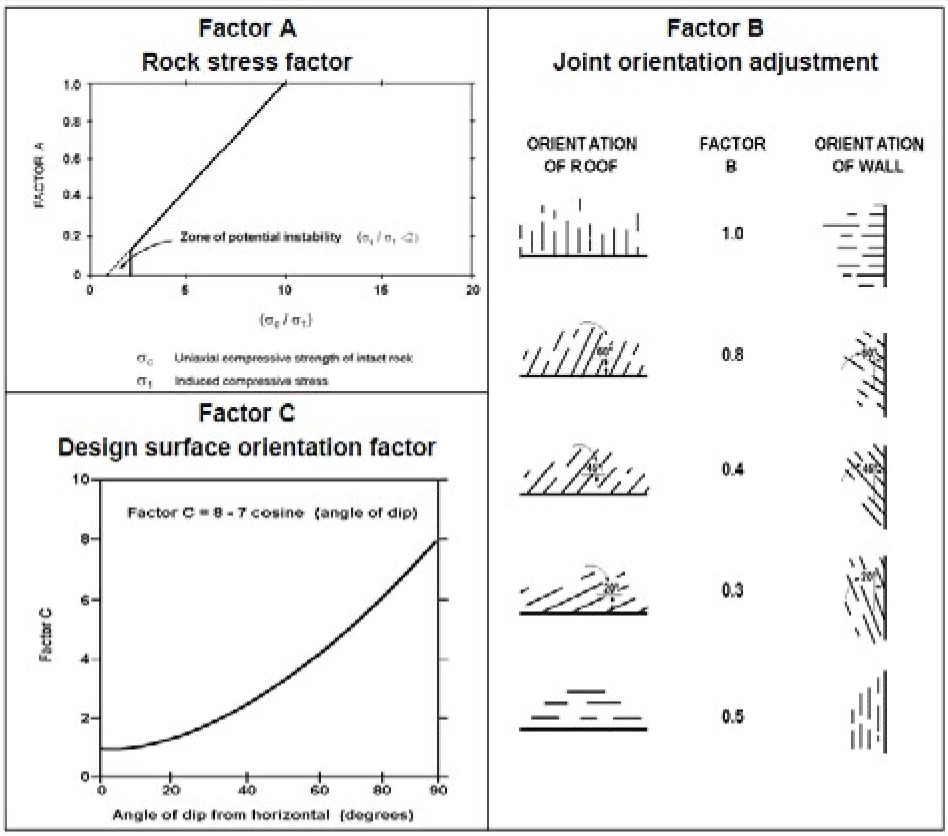

Figure 68: Assessment of stability number adjustment factors (after Mathews, 1980)

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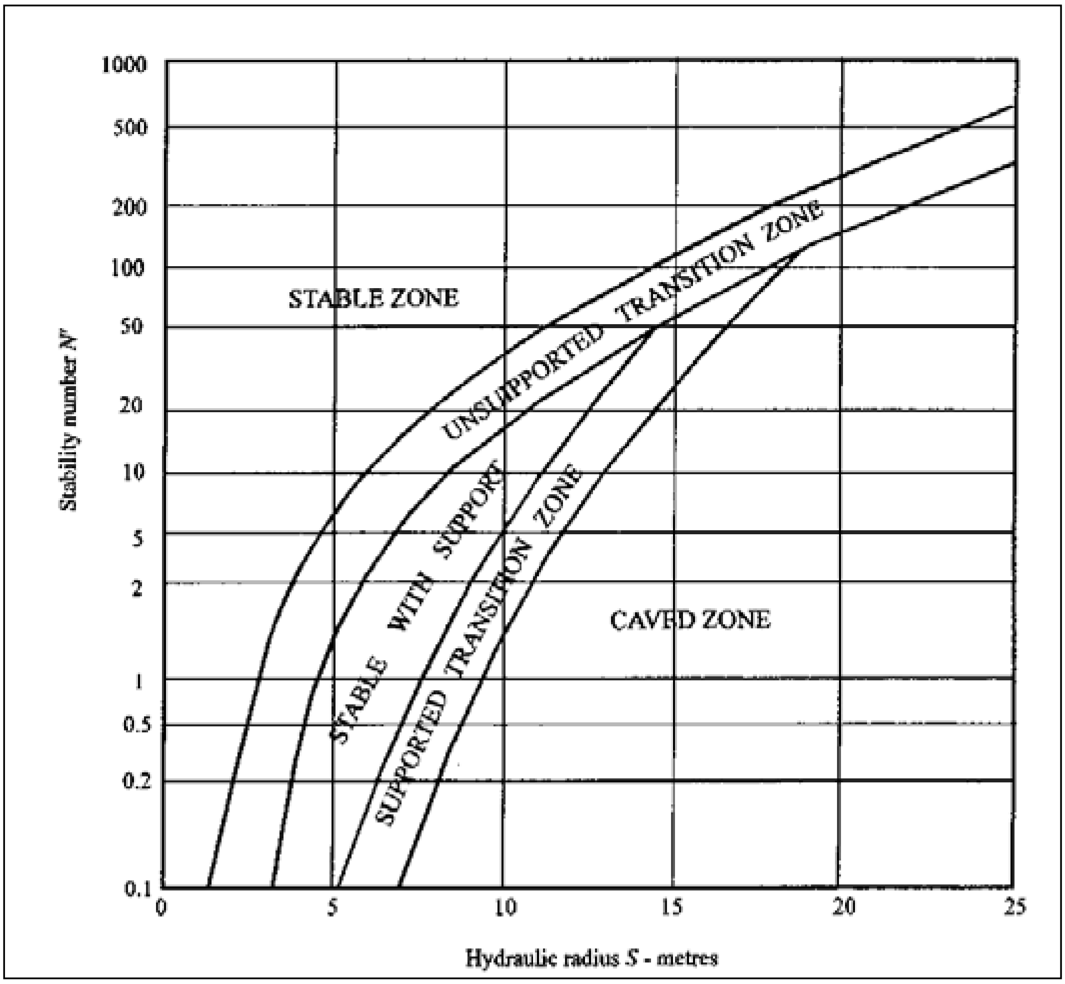

Figure 69: Stability graph published by Hoek, Kaiser and Bawden (1995) after Potvin (1988) and Nickson (1992)

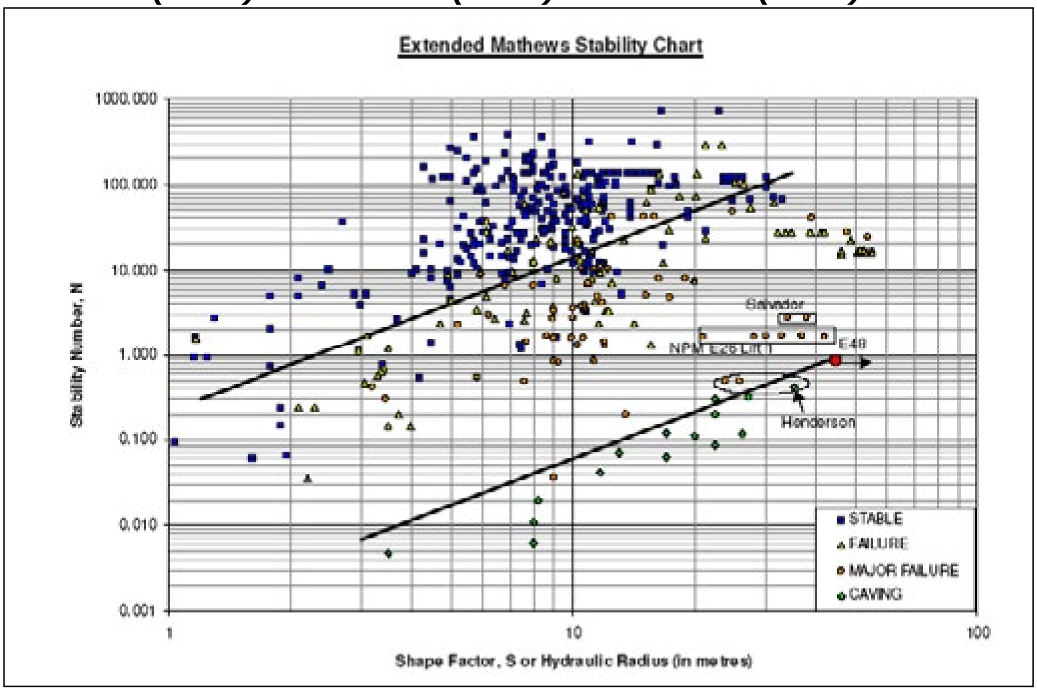

Figure 70: Stability graph after Mawdesley et al. (2001)

Other points to consider when laying out an undercut include the following:

Direction of undercut advance

The following points should be considered when deciding on direction,

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and point of commencement, of undercut advance:

Where there is an anisotropic in situ field stress, with large differences in horizontal components, the undercut should be advanced towards the direction of the major horizontal stress.

Where rockmass conditions vary, it is good practice to commence the undercut in weaker ground where the critical hydraulic radius for caving is reached earlier and advances towards the stronger ground. This also helps preserve undercut development in the stronger ground. However, undercutting should be commenced in the stronger ground if it is anticipated that stresses required to induce caving in the stronger ground would not be developed if the weak ground was caved first – i.e. that caving would only proceed in the weak ground. This level of complexity requires assessment using numerical models.

Some general points are indicated schematically in Figure 71.

Shape of the undercut front
Long leads and lags between sections of the undercut should be minimised.

An overall undercut front shape that is convex with respect to the cave (i.e. the centre of the undercut lags slightly) should cave more readily than one that is concave.

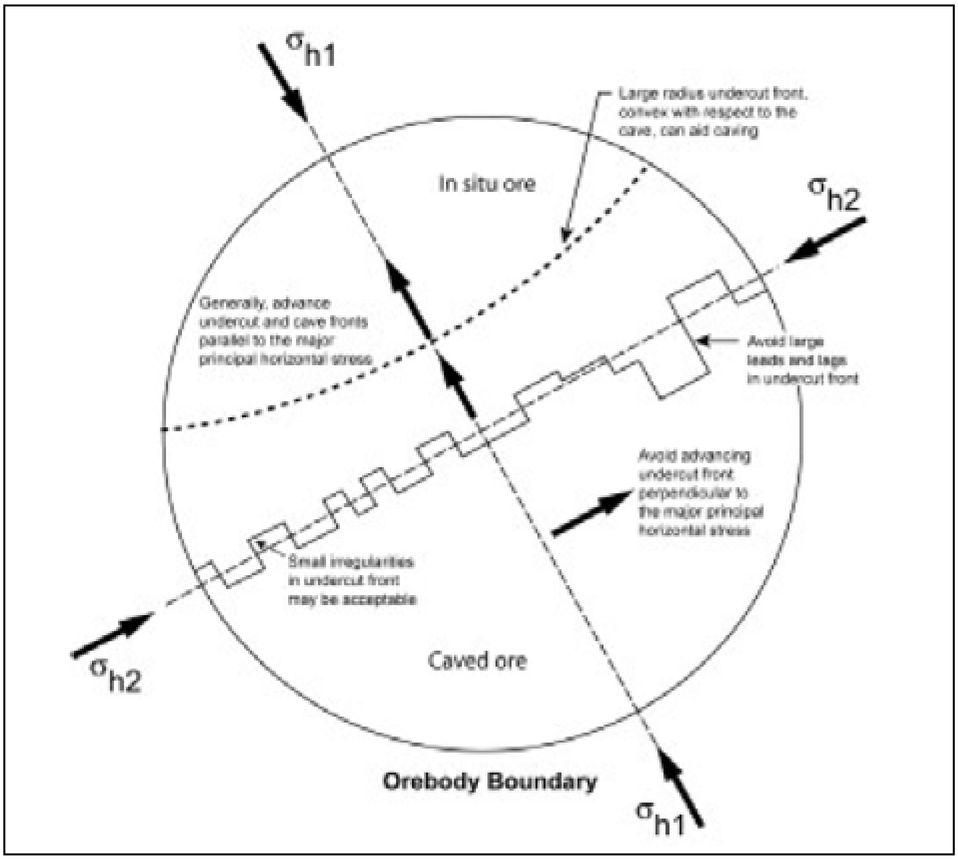

Figure 71: Factors to consider when orienting an undercut

Choice of pre-, post- or advanced undercut sequences
The two essential parts of a block cave mine are the undercut, which permits the cave to occur, and the extraction level, used in conjunction with drawbells and drawpoints to draw rock from the undercut. In general, if

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a rockmass will cave, then it is not exceptionally strong. At even moderate depth, there may be sufficient levels of increased stress ahead of an undercut to cause damage to tunnels in a weak rockmass. To minimise damage to extraction level development, it would be ideal to create it all in a de-stressed location below the undercut. However, to prevent re-consolidation of broken or caved rock on the undercut requires that drawbells are broken through and draw is initiated as soon as possible after an undercut advances. A trade-off therefore has to be made in terms of minimising extraction level development in the stressed area ahead of the undercut and getting draw initiated timeously. The mining sequences for opening up the extraction level in relation to undercut advance fall into three types:

- **Pre-undercut:** Extraction level development, including drawbells, is developed before advancing the undercut.
- **Post-undercut:** Extraction level development, including drawbells, is developed after advancing the undercut.
- **Advanced undercut:** A limited amount of extraction level development, e.g. main drives and possibly the connecting crosscut, is completed prior to undercut advance. The drawbells are completed afterwards.

REFERENCE

- Laubscher D H, 1994, Cave mining – The state of the art, SAIMM Journal, October 1994, p 279-293
- Brown E T, 2003, Block Caving Geomechanics. JKMRC Monograph Series in Mining and Mineral Processing 3. Published by Julius Kruttschnitt Mineral Research Centre, The University of Queensland, Australia.
- Hoek E, Kaiser P K, Bawden W F, 1995, Support of Underground Excavations in Hard Rock. AA Balkema, Rotterdam.

LEARNING OUTCOME 3.3.1.4

ESTIMATE LIKELY CAVING BEHAVIOUR BY USING ROCKMASS CLASSIFICATION METHODS FOR GIVEN ROCKMASS AND UNDERCUT CONDITIONS

For a given rockmass and undercut geometry, an assessment of likely caving behaviour is again made using the caving chart (Figure 67) or stability graph (Figure 70), depending on the type of rockmass rating (RMR or Q) that is available.

The available data is plotted on the graph and an assessment is made of whether the undercut geometry and ore body size result in a situation that will cave freely or may freeze.

It is also useful to examine what may happen higher above the undercut, where the cave may taper as it extends higher, or a weaker ore body may become narrower within a stronger host rock. Horizontal geological sections across the ore body at various elevations may be required to complete a thorough analysis of the caving behaviour – identifying stronger units where the cave may freeze, as well as weaker units where chimney caves could propagate.

3.3.2. FRAGMENTATION OF BROKEN ORE

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

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LEARNING OUTCOME 3.3.2.1

SKETCH, DESCRIBE, DISCUSS AND EXPLAIN THE FACTORS THAT AFFECT FRAGMENTATION IN CAVE MINING OPERATIONS

Knowledge of the likely fragmentation resulting in a caving operation is essential to the design of numerous factors, ranging from drawpoint sizing and spacing, to equipment selection and requirements for secondary breakage, and consequently has a great impact on the economics of a caving mine. Generally, it is useful to develop a fragmentation size distribution for the material that reports to the drawpoints out of the cave.

It is generally accepted that there are three levels of fragmentation:

1. ***In situ* fragmentation:** Blocks naturally present in the rockmass due to geological factors, prior to mining.
2. **Primary fragmentation:** Blocks and slabs that break from the cave back, partially controlled by *in situ* fragmentation, and partially by stress-induced fractures around the cave back.
3. **Secondary fragmentation:** Subsequent comminution of caved material as it moves down through the draw column to the drawpoints.

These are indicated in Figure 72.

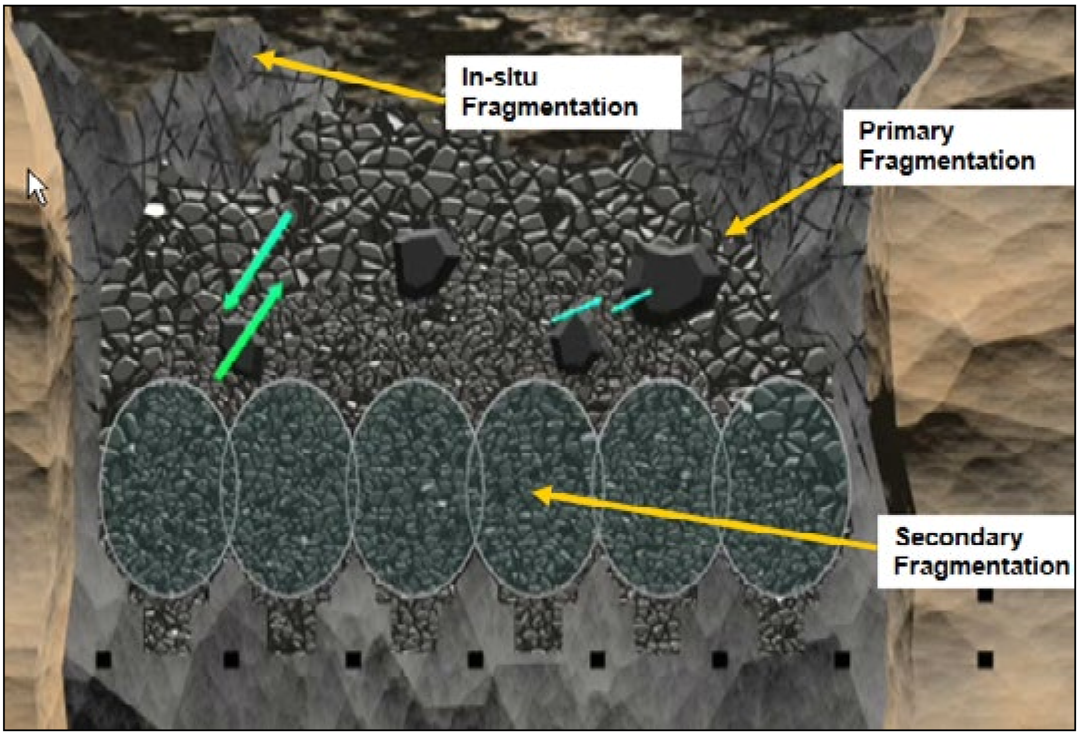

Figure 72: Types of fragmentation in a cave (after De Beers Block Cave School)

Following from the types of fragmentation, the factors that affect fragmentation include:

Geological discontinuities

These include bedding, joints, foliation, faults, shears and any weak stratigraphic horizons.

The orientation, length, termination, spacing and strength of these combine to create a network of discontinuities that define the size and shape

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of in situ blocks within the rockmass. Two very different cases that have identical individual fracture geometry statistics are shown in Figure 73. In the two cases, differences in the placement of the fractures relative to each other result in a completely different in situ fragmentation.

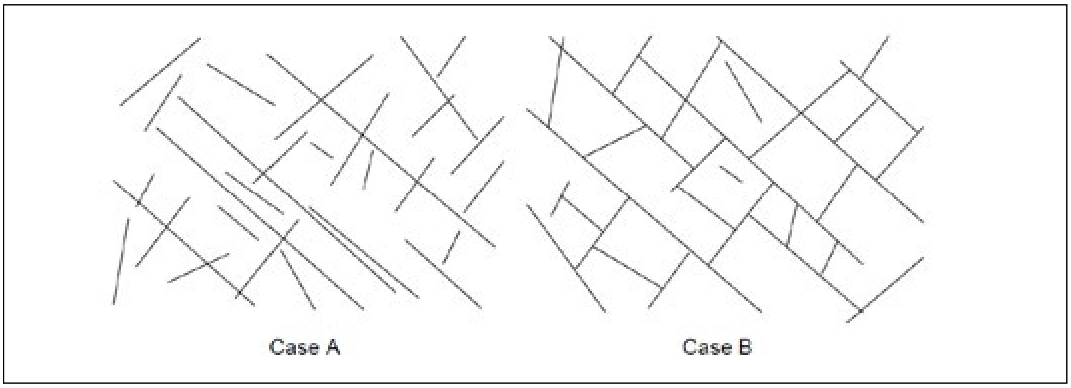

Figure 73: Influence of fracture network parameters on in situ fragmentation (after Brown, 2003)

Details of the methods for collecting, reporting and displaying information relating to geological structure information are covered in section 2.3 of this study material.

Note that many geological discontinuities may not be opened and contribute to primary caving. Joints may terminate in solid, and therefore not fully subdivide a block, but may contribute a weakness that permits failure during secondary fragmentation. Similarly, in some rockmasses, discontinuities may be strong, or healed, and not contribute to primary fragmentation.

Stress regime around the cave

Most primary fragmentation results from existing geological planes of weakness. Under high stress, or stress caving, condition fractures of intact rock may also occur. This depends upon the strength of discontinuities (i.e. a preference to fail on discontinuities) and the strength of the rock materials relative to the magnitude and orientation of imposed stresses. Where stress-induced failure occurs, the primary fragmentation size distribution is likely to be smaller than where blocks merely detach from the cave back under gravity (stress caving versus subsidence caving).

Secondary fragmentation factors

Secondary fragmentation is also a function of rock strength and stronger or healed discontinuities, and the degree of comminution that occurs as the material is drawn in the cave is a function of:

- The stress regime within the cave
- The composition and mechanical properties of the ore
- The rate of draw
- The height of draw
- The time that material is resident in the draw column

The factors than need to be considered when assessing fragmentation are illustrated, after Laubscher, in Figure 74.

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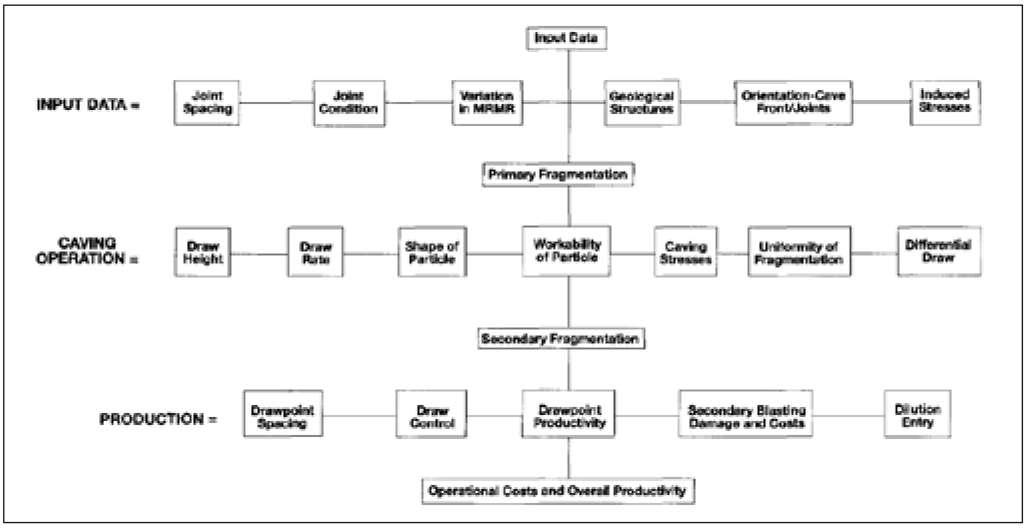

Figure 74: Factor to be included when assessing fragmentation after Laubscher DH, 1994, Cave mining – The state of the art, SAIMM Journal, October 1994, p 279-293

LEARNING OUTCOME 3.3.2.2

ESTIMATE LIKELY FRAGMENTATION BY USING ROCKMASS CLASSIFICATION METHODS FOR GIVEN ROCKMASS AND UNDERCUT CONDITIONS

The most widely used method to predict fragmentation is the expert system program, Block Cave Fragmentation (or BCF), where data can be entered to cover most of the points noted in Figure 75, and a range of assumptions and empirical rules are applied to produce an estimate of the likely fragmentation distribution. The initial BCF concept was developed by Dr DH Laubscher in collaboration with AR Guest and PJ Bartlett for application at the Premier Diamond Mine. Programming and later development were carried out by Dr GS Esterhuizen (1994, 1999).

A simplified set of hand calculations, similar to those captured in BCF, can be used to make estimates of fragmentation. BCF makes use of detailed inputs relating to joint populations; however, Heslop and Laubscher (1981) produced a table (see Figure 75 below) that permits a quick estimation of in situ block sizes for primary fragmentation. This is based on Laubscher's MRMR rating and the derived rockmass classes.

Note that two size ranges are provided: for subsidence caving, where blocks detach from the cave back under gravity, and stress caving, where stress-induced slabbing occurs and results in smaller block sizes. When selecting the appropriate rockmass ratings to represent the ground that will be caved, the following points need to be considered:

- The intact RMR (IRMR - Laubscher) of the ore body and hangingwall zone needs to be known for at least twice the ore body height, and plotted out on sections as zones, if there is a range in IRMR values greater than 10.
- Low rating zones in a high rating zone can lead to failure of the more competent rock at lower stress than if it was a uniform stronger zone.
- A realistic average for the rockmass is required.
- Using Laubscher's method, an intact rating, IRMR, is first calculated, and the adjustments are provided to determine the MRMR.

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The choice of subsidence caving versus stress caving is a function of the depth or likely in situ stress around the cave and the intact rock strength. If the stress field is likely to induce failure around an excavation in the rockmass at undercut elevation, then the stress caving scenario is likely to apply.

		Fragment size (m)	
Class	Rating	Subsidence caving	Stress caving
1	80–100	8–12	4–9
2	60–79	3–9	3–6
3A	50–59	1.4–4	0.7–3
3B	40–49	0.9–1.5	0.3–1
4A	30–39	0.6–1.0	0.2–0.5
4B	20–29	0.6	0.4
5A	10–19	0.3	0.3
5B	0–9	0.1	0.1

Figure 75: Primary fragmentation expected from block caving ore bodies based on MRMR classes (after Heslop & Laubscher, 1981)

REFERENCE

- Heslop TG, Laubscher DH, 1981. Draw control in caving operations on Southern African chrysotile asbestos mines. In Design and operation of caving and sublevel stoping mines, D R Stewart (ed.), pp 755-774. New York: Soc. Min. Engrs, Am. Inst. Min. Metall. Petrolm Engrs.

3.3.3. FLOW BEHAVIOUR OF BROKEN ORE

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

LEARNING OUTCOME 3.3.3.1

SKETCH, DESCRIBE, DISCUSS AND EXPLAIN THE FACTORS THAT AFFECT THE FLOW BEHAVIOUR OF BROKEN ORE IN CAVE MINING OPERATIONS

The flow of broken ore in cave mining operations is determined through the mine's draw control strategy, which, in turn, makes use of the extraction level drawpoint size and spacing.

Factors that affect the flow of broken ore in a cave mining operation include:

1. The flow ellipsoid associated with a single drawpoint (a function of particle size and shape, width of drawpoint, rate of draw, ore mechanical properties and moisture content).
2. Interaction of rate of draw between multiple drawpoints (drawpoint spacing).
3. The presence of more frequent larger blocks during the early stages of caving, leading to a void diffusion effect, where 'fingers' of movement occur rather than the traditional flow ellipsoid.

These are discussed in greater detail below.

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PAPER 3.3: CHAPTER 3

The flow ellipsoid and draw from a single drawpoint
This is established as the 'classical' way of describing the gravity flow of broken ore in a cave. If a drawpoint is opened beneath a large volume of broken ore, the material that will have discharged after a given time will all have originated within an approximately ellipsoidal zone over the drawpoint, known as the **ellipsoid of draw (or motion, or extraction)**. Material between this ellipsoid and a larger one, the **limit ellipsoid**, will have loosened and displaced, but not have reached the discharge point. See Figure 76. This is the basis for the flow ellipsoid concept. Draw of ore develops a draw cone within the ellipsoids.

It is assumed that a horizontal section through the ellipsoid is circular – this may not be correct in all cases.

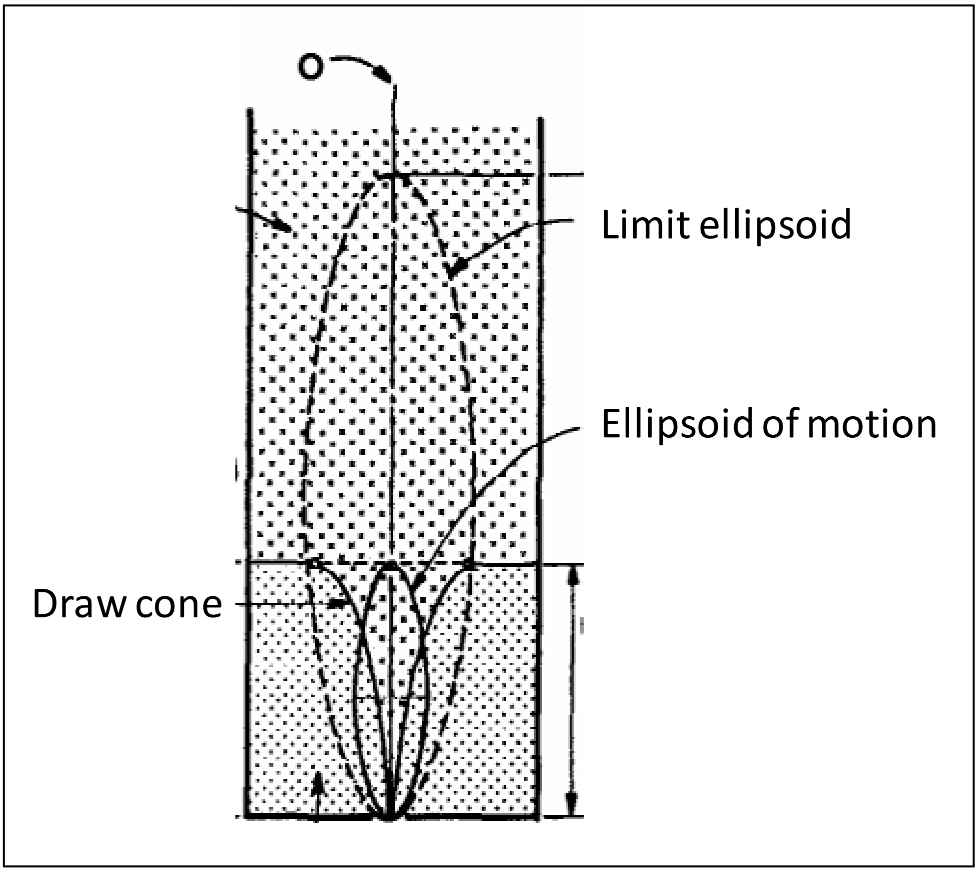

Figure 76: The flow ellipsoid concept for gravity flow of broken ore (after Kvapil, 1992)

The shape of the ellipsoids is influenced by several factors:

- Particle size: The smaller the particle size, the more vertically elongate the ellipsoid of motion is (greater eccentricity of ellipsoid).
- Width of the discharge opening, or drawpoint.
- Rate of draw: Higher draw rates result in greater eccentricity (elongation) of the ellipsoid.
- Shape and mechanical properties of particles.
- Moisture content of the broken ore.

Interaction of draw between multiple drawpoints
Ideally, drawpoints are sized and spaced on the extraction level such that ellipsoids of draw for individual drawpoints overlap. See Figure 77.

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If draw zones for adjacent drawpoints do not overlap, isolated draw zones develop and pillars of un-drawn ore are left, resulting in potential loss of ore and an application of load to pillars between excavations on the extraction level.

If draw zones overlap, no ore is lost, and no additional pillar loading is incurred. If one drawpoint is drawn more rapidly than its neighbours, there is potential to draw waste down and into the draw zones of the neighbouring drawpoints.

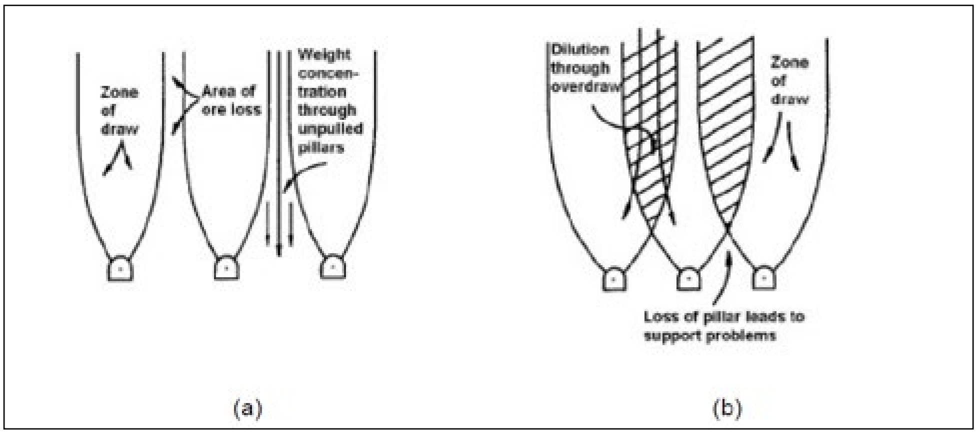

Figure 77: Idealised vertical sections showing (a) excessive drawpoint spacing with non-overlapping draw zones, and (b) closer spacing with overlapping zones (after Richardson, 1981)

Void diffusion effect

If the caved material is coarser, more angular, or poorly graded (such as in the early stages of caving), the classic flow ellipsoid may not develop. Irregular flow tends to occur in the form of 'fingers' that migrate sideways and upwards. This may lead to runs of finer material between larger fragments. This is problematic if the finer material is waste. This is illustrated in Figure 78.

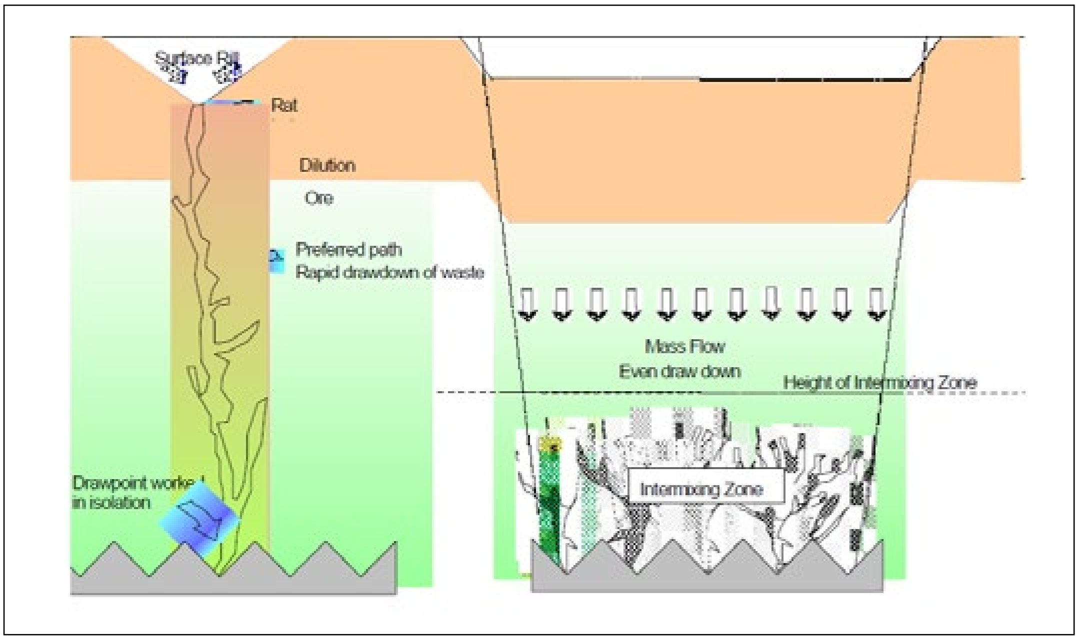

Figure 78: Void diffusion flow (after Laubscher, 2000)

In most caves, a mix of ellipsoidal draw and void diffusion draw may

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occur. If the caved material is made up of particles that are predominantly equi-dimensional and rounded with a well-graded size range, then ellipsoidal draw is favoured.

If the material is gap graded or has a wide range in sizes or a high proportion of larger angular or slabbed blocks, void diffusion behaviour is favoured.

REFERENCE

- Brown ET, 2003, Block Caving Geomechanics. JKMRC Monograph Series in Mining and Mineral Processing 3. Published by Julius Kruttschnitt Mineral Research Centre, The University of Queensland, Australia.

LEARNING OUTCOME 3.3.3.2

SKETCH, DESCRIBE, DISCUSS AND EXPLAIN THE FLOW OF ROCK TOWARDS DRAWPOINTS IN SUB-LEVEL CAVING LAYOUTS

Much of this question can be answered by referring to the material presented in section 3.1.3.3.1. The same diagrams apply.

In addition, the following points apply:

- An ellipsoidal draw zone is associated with each drawpoint. Within this zone, granular or gravity flow takes place.
- Within each draw zone, there is a draw cone, where material is drawn down more rapidly in the centre of the zone and less rapidly towards the edges.
- Particles in the centre of the draw cone move only vertically, while particles towards the edges are drawn in from the sides, towards the centre.
- If rapid draw occurs in a single drawpoint, the ellipsoid of draw can funnel upwards and draw in waste.
- If similar volumes of ore are drawn from all drawpoints, the interface between ore and waste is drawn down evenly and waste is not drawn into any single draw ellipsoid until all are exhausted simultaneously.
- Above the level where draw ellipsoids associated with each drawpoint interact, the broken ore is drawn down evenly (provided drawpoints are drawn evenly). Mass flow is considered to occur in this region, where particle trajectories are nearly vertical.
- Within the interaction zone, differences in draw between adjacent drawpoints may permit particles to move laterally as well as vertically – provided draw from one single drawpoint is not excessive, this assists in lowering the mass flow region in a uniform way.

Figure 79 illustrates these points.

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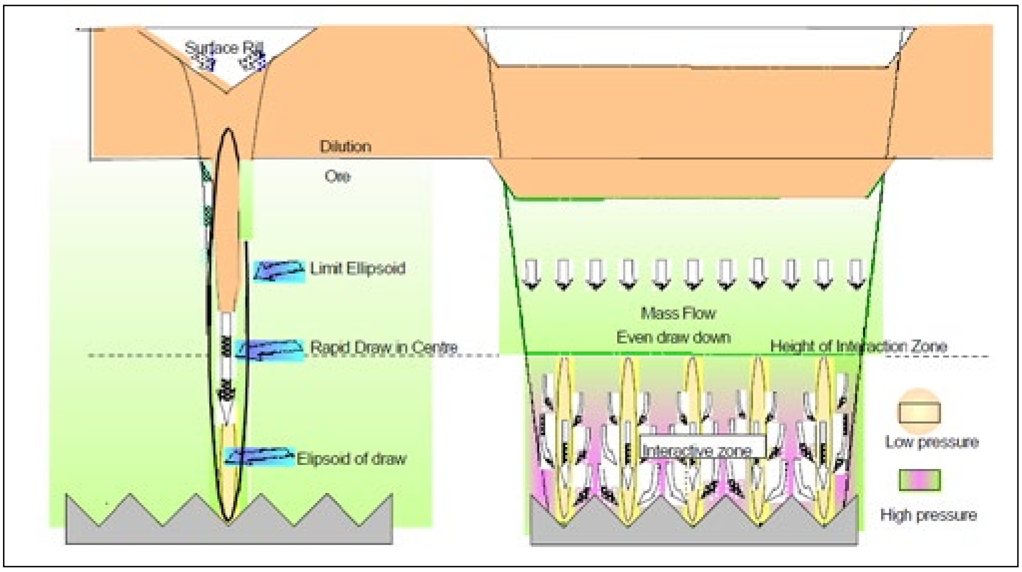

Figure 79: Schematic diagram illustrating flow paths for (a) an isolated drawpoint, and (b) several drawpoints working concurrently (after Laubscher, 2000)

LEARNING OUTCOME 3.3.3.3

SKETCH, DESCRIBE, DISCUSS AND EXPLAIN HOW DRAWPOINT SPACING AFFECTS DILUTION AND RECOVERY

This question can be answered by referring to the material presented in sections 3.1.3.3.1 and 3.1.3.3.2. The same diagrams apply.

Key points are:

- An isolated drawpoint has an ellipsoidal draw zone associated with it.
- If drawpoints are far apart, draw zones remain independent and do not interact, potentially leaving a column of ore *in situ* between zones. This may result in ore loss.
- If drawpoints are close together, draw zones overlap and no ore loss results. However, if ore is removed from one drawpoint at a faster rate than the neighbouring drawpoints, then there is a risk of drawing down waste, which may enter the draw zones for the neighbouring drawpoints. The effect is made worse if the waste comminutes readily and can flow more easily than ore.

Figure 80 broadly illustrates the effect of increasing drawpoint spacing on the height to the drawpoint interaction zone (the height over which adjacent draw ellipsoids overlap) and consequently on dilution. As spacing increases, the draw height increases. However, as spacing increases, the height to the draw interaction zone gets closer to the draw height.

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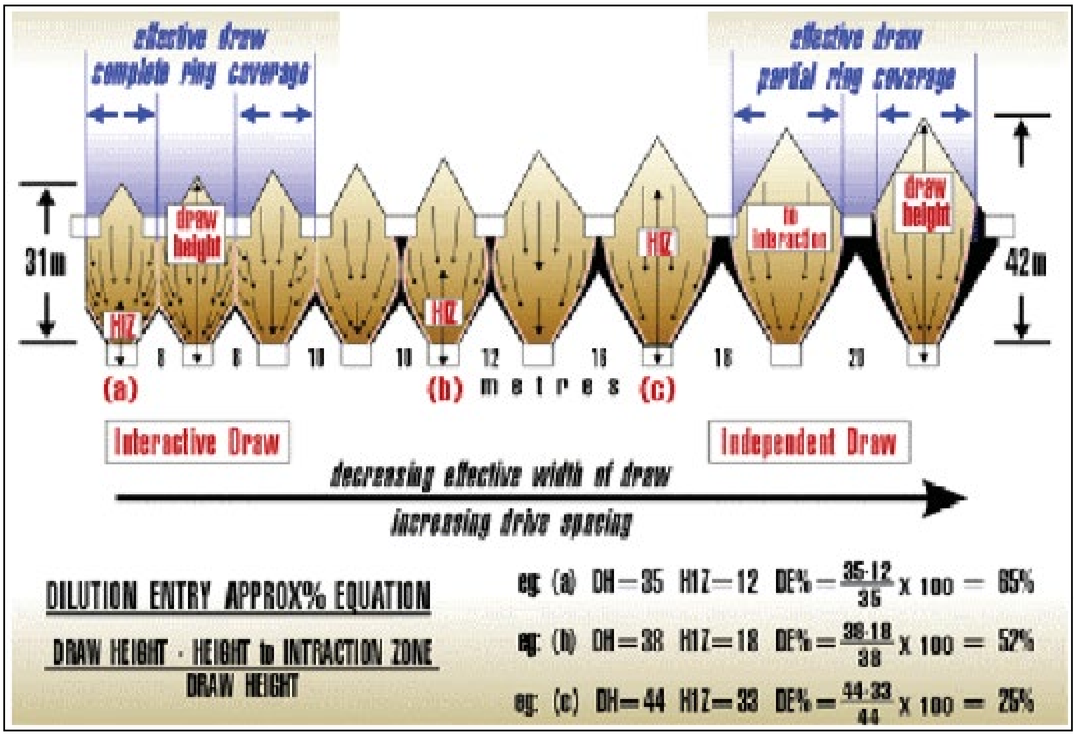

Figure 80: Effect of drawpoint spacing on draw height (DH), the height to the draw ellipsoid interaction zone (HIZ) and the percentage dilution entry

LEARNING OUTCOME 3.3.3.4

SKETCH, DESCRIBE, DISCUSS AND EXPLAIN HOW THE AMOUNT OF ROCK BLASTED AFFECTS DILUTION AND RECOVERY

Blasting is carried out regularly in sub-level caving (SLC) and front caving. This does not apply to block caving operations, except when considering initial undercutting.

When blasting an undercut (and this is also the case when ring blasting in SLC or front caving), the objective is to:

- Achieve optimum breakage and fragmentation.
- Avoid the formation of pillars.
- Produce a fragmentation size that is easily loaded and cleared to avoid blasting subsequent rings under severe choked conditions.

When advancing an undercut or SLC, the holes in a drill ring are close to the cave and in this position may become lost or closed for a number of reasons, including (1) blast damage from the previous ring blast, (2) stress effects in the abutment, (3) relaxation of the rockmass.

If rings are not drilled correctly, or are only partially detonated, the following consequences may result:

1. Creation of large blocks that may choke the end and require secondary blasting to remove. In the case of an SLC, large blocks may not be removable and may result in the loss of a drawpoint, leading to poor recovery.
2. Pillars may be left in the undercut, and this may have two consequences: (1) loading applied to extraction level development below the undercut, and (2) loss of interactive draw between drawpoints, and consequently a disruption of the draw control plan (see below).

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When blasting close to the contact between ore and waste (e.g. towards the edges of an SLC, or the edge of a pipe in a block cave), over-drilling may result in damage to the waste country rock, resulting in early waste entry to drawpoints near the edge of the ore body.

LEARNING OUTCOME 3.3.3.5

SKETCH, DESCRIBE, DISCUSS AND EXPLAIN THE PRINCIPLES OF DRAW CONTROL TO LIMIT DILUTION IN BLOCK CAVING OPERATIONS

The objectives of draw control are:

- Minimise ore dilution and maintain the planned grade of ore sent to the plant.
- Maximise ore recovery.
- Minimise any damaging load concentrations imposed on extraction level development.
- Avoid creating conditions that could lead to mudrushes or air blasts.
- Optimise secondary fragmentation.
- Minimise the incidence of ore recompaction.

To meet these objectives, the principles of draw control include:

- Conduct draw scheduling
- Ensure interactive draw between drawpoints
- Control rates of draw
- Monitor and record volumes of draw
- Perform cave modelling

These can be explained further:

Referring back to Figure 78, if draw is carried out evenly from a series of drawpoints, a region develops where there is interactive draw, where ellipsoids of draw overlap. Above the height of the interaction zone, a region of mass flow and uniform subsidence occurs. Within this draw column is an interface between ore below and waste above. The objective with draw control is to lower this interface as uniformly as possible, avoiding locally pulling waste down into one or more drawpoints, and as far as possibly reducing the potential for mixing ore and waste.

As undercutting generally starts in one part of a block, and progressively advances across it, drawpoints are progressively opened up. As a result, draw cannot be completely even across all drawpoints in the entire block, but is retreated across the block at a controlled rate, with drawpoints being exhausted in sequence (creating a depletion surface). An example showing a partially drawn block, with the location of the ore waste interface, is shown in Figure 81.

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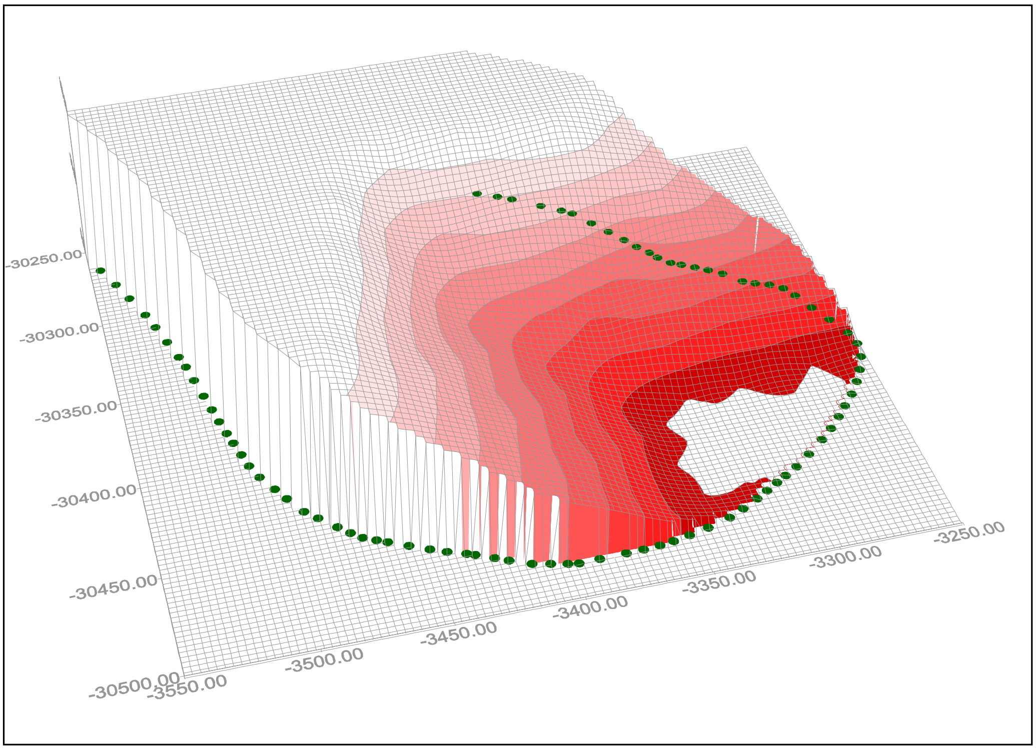

Figure 81: Example showing the idealised partial draw-down of the ore waste interface in a block cave in a kimberlite pipe, where undercutting commenced in the lower right corner

Referring back to the key points of draw control:

Draw scheduling ensures that a plan is developed to maintain the ideal depletion surface, which minimises waste tons by keeping the ore waste interface intact. This is influenced by rate of undercutting, drawpoint availability and desired overall production rates. Draw scheduling is often carried out using computer software such as PC-BC. The schedule identifies the time schedule of tonnages to be pulled from each drawpoint over the life of the cave.

Ensure interactive draw by estimating the size of an independent draw zone (IDZ) that would occur above a single, isolated drawpoint, and positioning drawpoints on the extraction level such that these zones overlap.

Loading from drawpoints is then sequenced such that alternating drawpoints across the extraction level are drawn on successive days, ensuring that draw zones associated with each drawpoint interact. The principle is illustrated in Figure 82.

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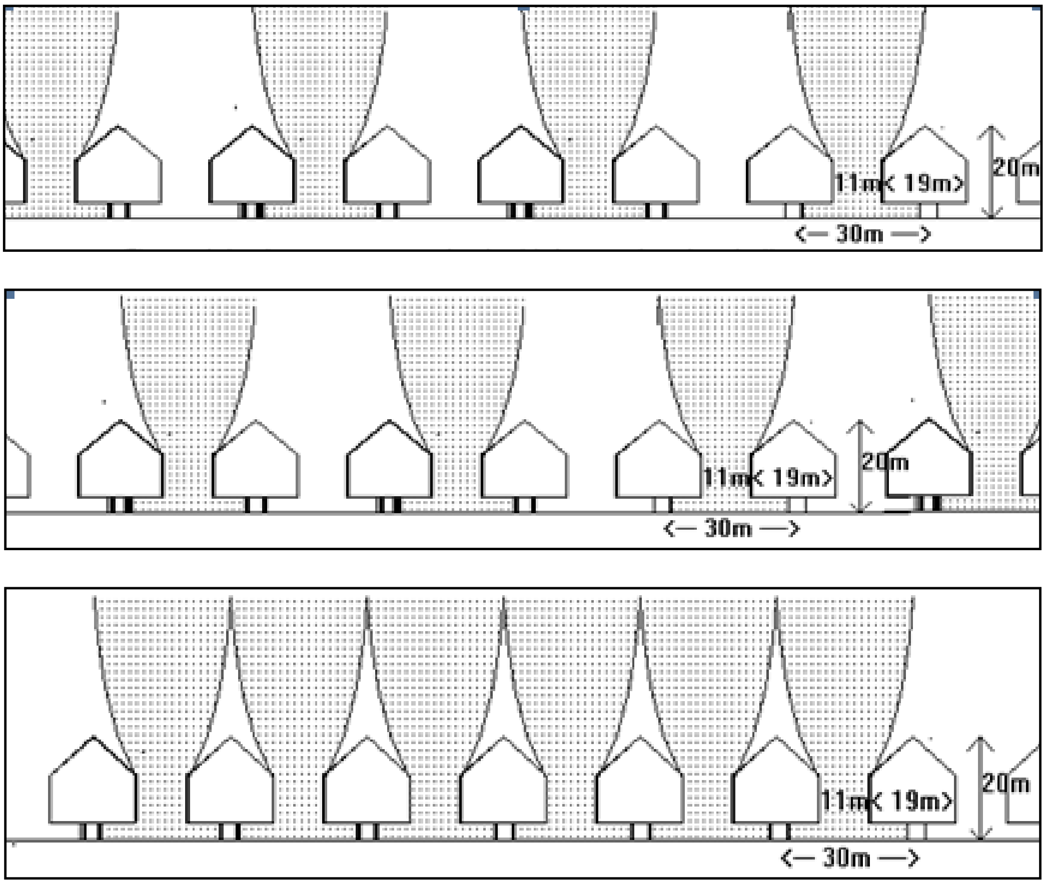

Figure 82: Principal of drawpoint sequencing to ensure interactive draw (after Laubscher, 2000)

Control of draw rates is also illustrated in Figure 82. In addition to this, the rate of draw also needs to be matched to the caving rate. If the draw rate is too great, it is possible to develop a void above the caved material. This is illustrated in Figure 83. In addition to leading to a situation in which a sudden collapse could result in an air blast, this also permits waste to enter caved ore by running down the side (in the example shown), causing dilution. In this example, the relative percentage draw for optimal control is illustrated in the lower diagram.

High rates of draw may also lead to lower comminution with larger slabs reporting to drawpoints, and possible hang-ups. Very low draw rates may lead to excessive fines, and, in turn, increase mudrush risk.

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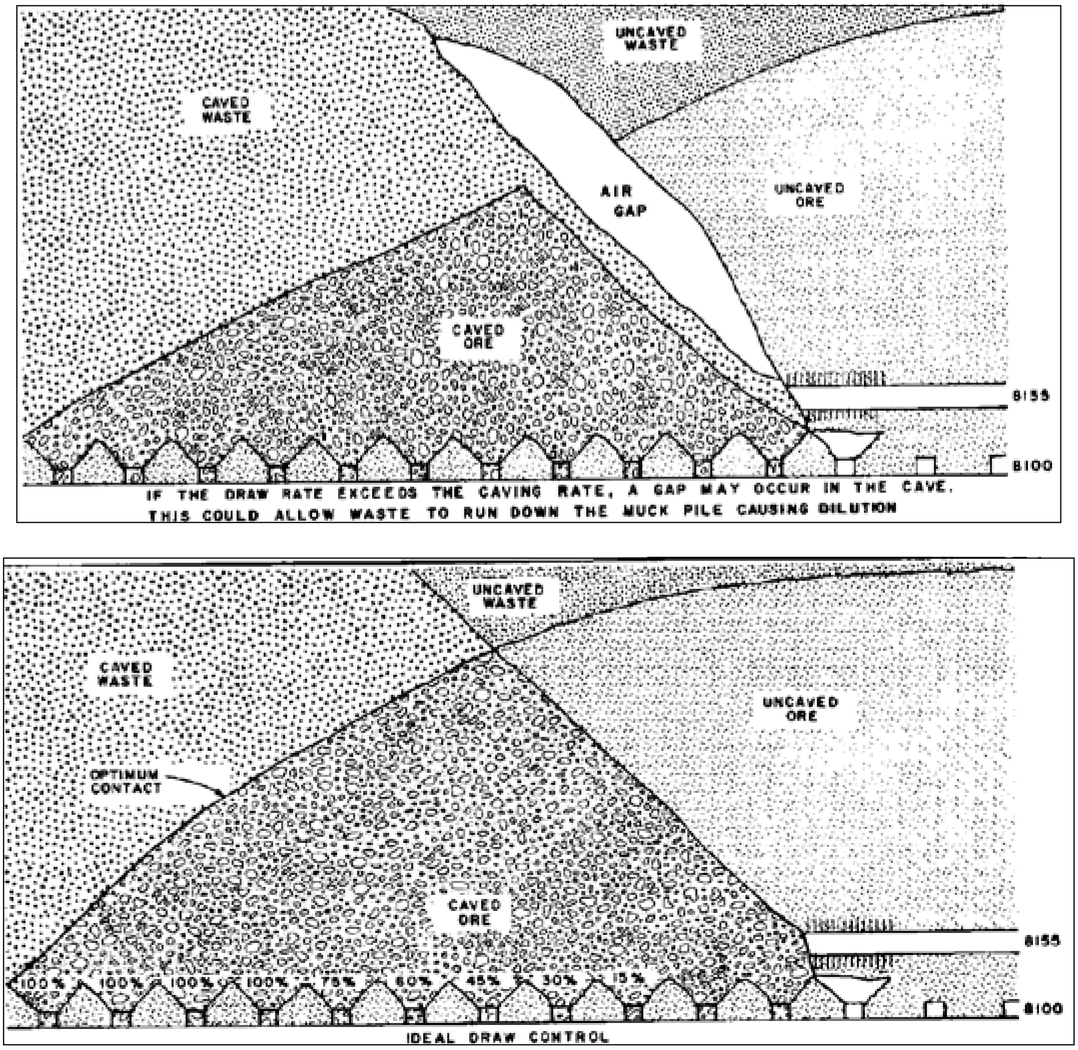

Figure 83: The effect of excessively high draw rates (top) versus optimal draw (below). (After Henderson Mine, DeWolfe, 1981)

Monitor and record volumes of draw control of draw rates:
This is an essential part of maintaining and checking the success of the draw control plan. The data that should be recorded includes the loads taken from each drawpoint for a given size of LHD, permitting volumes and tonnages to be calculated. This is then assessed in combination with other data recorded for drawpoints, which includes: fragmentation, drawpoint status (operational, stopped, blocked, etc.), ore/waste percentage at each drawpoint, and water/moisture present.

This data is then used for comparison and back analysis against the draw control plan, permitting modification if required.

It also permits correlation and checking of the geological model, calibration of cave simulations, and calibration of fragmentation predictions.

Perform cave modelling using a computer software package such as PC-BC or REBOP. This feeds back into the draw control plan and helps identify caving limits and rates and can be calibrated and re-examined as the cave is developed.

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LEARNING OUTCOME 3.3.3.6

DESCRIBE, DISCUSS, EXPLAIN, COMPARE AND CONTRAST THE METHODS OF DRAW CONTROL IN PANEL CAVING AND BLOCK CAVING SITUATIONS

To explain the differences in draw control in block and panel caving situations, the differences between the two methods first need to be understood.

A typical block cave mining layout is shown in Figure 84, and a typical panel cave layout is shown in Figure 85. Both have an undercut level, drawbells and drawpoints on the extraction level below. Both may use various (and similar) layouts for excavations.

The difference is that, in a block cave, the undercut and consequently the cave are developed as a single operation over the full extent of the ore body or mining block. In a panel caving operation, the ore body or mining block is initially not fully undercut, but is divided into strips that are undercut and allowed to cave one by one. After one panel is started, successive panels follow some distance behind. As a result, the cave front moves across the ore body at an angle to the direction of advance of the undercut.

In both cases, vertical propagation of the cave will occur in response to the continued removal of broken ore through the active drawpoints, and horizontal propagation of the cave will occur as more drawpoints are brought into operation under the undercut area.

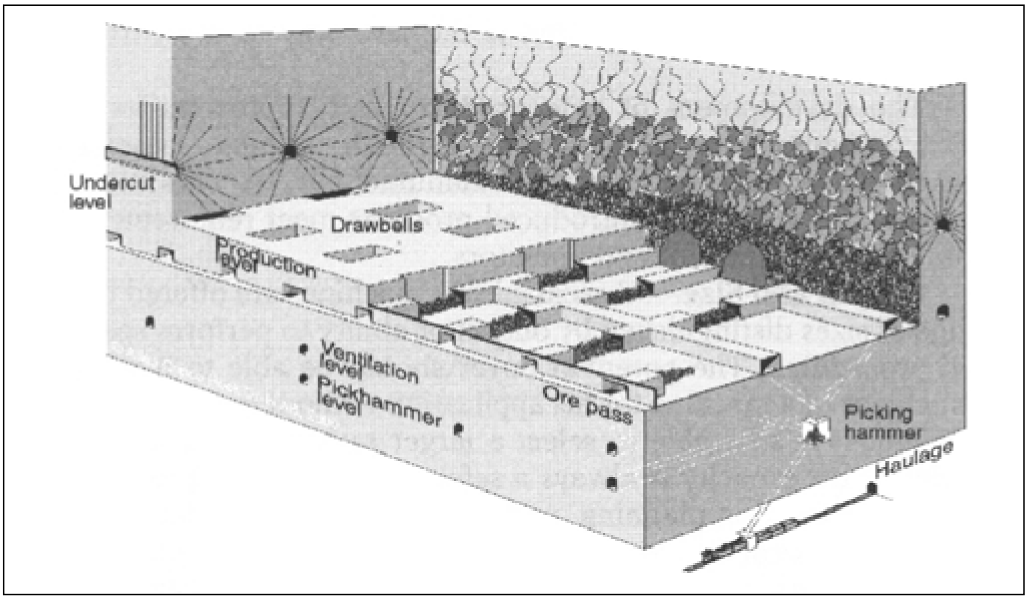

Figure 84: Example of block cave mining at El Teniente mine

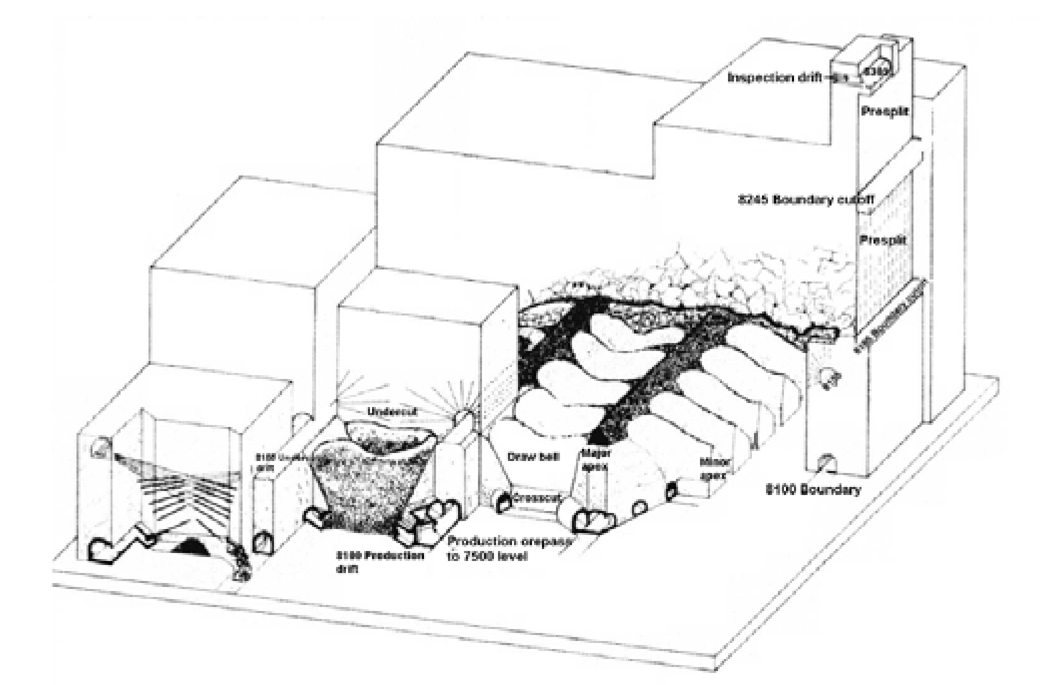

Figure 85: Example of mechanised panel caving at Henderson mine. The key feature is the step in the undercut between successive panels.

The general principles of draw control outlined in the previous section apply equally to standard block caving, as well as panel caving. However, in panel caving, the presence of leading and lagging corners between panels introduces additional concerns:

Stress concentrations: In panel caving, there are low stress concentrations around leading panel corners and high stress concentrations in the lagging corners. This can influence the initiation of caving, and the size of initial fragmentation that is created – which can vary across the front of the panel.

Caving in panel caving occurs readily on the lagging corners and less readily on the leading panel corners.

Fragmentation in panel caving may be smaller in the lagging corners and larger towards the leading panel corners.

By comparison, if a straight undercut front is applied across a block or ore body in a block cave, there are no stress concentrations and similar caving and fragmentation conditions throughout – the cave front is parallel to the undercut front.

Draw control in a panel cave operation must take account of these differences across the panel front. Areas where equal draw is required from drawpoints tend to form diagonal lines across panels (parallel to a line drawn through the lagging corners of successive panels).
If a first panel leads the next be a long distance, care must be taken to ensure that there are no large differences in elevation of the ore/waste interface between the two panels, in which case there is a risk of early waste ingress in the drawpoints in the second panel that lies closest to the edge of the first panel.

In a block cave with a straight undercut front across the block or ore body, areas of equal draw requirements tend to parallel the undercut front.

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Boundary dilution effects
It is often the case in a block cave that the block forms only a part of the overall ore body. In this case, a new block may be mined adjacent to a previously mined block and there is a risk of drawing waste from the depleted block into the ore column around the edges of the new block. Ore pillars are required at the edges of the block to limit this waste ingress.

LEARNING OUTCOME 3.3.3.7

LIST, DESCRIBE AND DISCUSS THE ADVANTAGES AND DISADVANTAGES OF EACH OF THE ABOVE METHODS

Bearing in mind that a panel cave has a series of leading and lagging undercut panels, whereas a traditional block cave attempts to have a straight undercut front across the full block or ore body, the advantages and disadvantages of block caving compared to panel caving are summarised as follows.

Stress concentrations
High stress concentrations occur in the lagging corners between panels in panel caving. These promote caving in these corners, but may also result in damage to undercut tunnels in this area, as well as any extraction level development below.

Low stress concentrations occur on the leading panel corners in panel caving. Caving may occur less readily on these corners.

This has implications for fragmentation and draw control in panel caving.

With a straight undercut block cave, there are no differing stress concentrations, and conditions are similar across the undercut front, with similar caving tendencies and similar risk of damage to excavations.

Mining flexibility
In a block cave where the undercut is to be kept in line, there is very little mining flexibility. If mining problems are experienced and a blast is not taken in one part of the undercut, the advance of the entire undercut can be delayed.

In the panel caving method, the block is divided into a series of shorter undercut panel sections, which can advance with some independence of each other. Problems in one panel do not necessarily delay other panels from advancing.

3.3.4. ORE DILUTION
The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

LEARNING OUTCOME 3.3.4.1

SKETCH, DESCRIBE, DISCUSS AND EXPLAIN THE FACTORS THAT AFFECT DILUTION IN CAVE MINING OPERATIONS

Many of the factors that influence ore dilution in cave mining operations have been indicated under the outcomes dealing with draw control, above.

Dilution is an integral part of cave mining, and there is invariably a dilution zone around the ore/waste interface, rather than there being a sharp cut off.

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In summary, the main factors are:

- Draw control: The ore/waste interface should be drawn down evenly across a cave. If one drawpoint is loaded more than its neighbours, increased draw down of this interface will occur and waste may be drawn into the ellipsoids of draw of the neighbouring drawpoints, resulting in dilution.
- Undercutting: This must be effective and complete, leaving no pillars. Pillars result in ore left in situ, as well as potentially resulting in poor caving, an uneven ore/waste interface, and inevitable waste draw.
- Geology: The location, attitude, orientation and condition of the boundary between country rock and the ore body are critical to identify and develop a specific draw control plan for those drawpoints close to the boundary.
- Grade distribution: This may vary within the ore body and include zones that are below the normal ore cut-off grade. It may be necessary to cave and draw these zones to get to higher grade ore higher up the ore column. In other cases, local small patches of high grade ore in waste may result in irregular sampling at drawpoints and a tendency to keep drawing when grade is really exhausted.
- Country rock strength: If the waste is weaker than the ore, and comminutes more finely, it may be drawn into the ore column between stronger and larger ore particles, resulting in a diffuse ore/waste contact, and ends up reporting early at drawpoints.
- Differences in density: If ore is higher density than waste, there is low dilution, and high dilution if the waste is denser.

Even when the ore/waste interface is drawn down evenly, some mixing of ore and waste will still occur. This mixing is a function of:

- Draw column height
- Range in fragmentation
- Draw zone spacing
- Range in tonnages drawn from working drawpoints

These last two factors give the height of the interaction zone between draw ellipsoids.

LEARNING OUTCOME 3.3.4.2

SKETCH, DESCRIBE, DISCUSS AND EXPLAIN THE REMEDIAL MEASURES TO LIMIT DILUTION IN CAVE MINING OPERATIONS

Once significant dilution occurs in a caving operation, it is difficult to correct. It generally results from having prematurely drawn waste into the ore column. Remedial measures may include:

- Identify the cause of dilution.
- Stop or reduce draw from drawpoints where high percentages of waste are reported.
- Reassess the draw control schedule using software such as PC-BC, and modify draw planning accordingly.

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LEARNING OUTCOME 3.3.4.3

DESCRIBE, DISCUSS AND EXPLAIN THE RELATIONSHIP BETWEEN DILUTION AND RECOVERY FOR CAVE MINING OPERATIONS

Dilution is an integral part of cave mining. There is mixing along the ore/waste interface, and waste will be loaded with the ore at drawpoints well before ore is exhausted.

The relationship between waste dilution and ore recovery is shown in Figure 86. This plots the composition (0 to 100% ore) versus the percentage of the vertical height of the ore column that has been drawn via the drawpoints. Several curved lines are drawn on the diagram – each of these represents the percentage draw at which waste starts to enter the ore column and reports at the drawpoint, e.g. after 20, 40, 60 and 80% draw of the total ore column.

For example, if waste first reports to the drawpoints after drawing 40% of the ore column height, then by the time 80% of the column height is drawn, there will be 50% ore and 50% waste reporting to the drawpoints.

The point at which waste is first reported at the drawpoint (e.g. after 20%, 40%, etc. draw of the ore column) will be a function of the factors listed in the previous sections.

$$\frac{A-B}{A} \times C \times 100 = \text{Dilution Entry}$$

A = Ore Draw Column Height x Swell Factor = 168m
B = Height of Interaction Zone = 90m
C = Draw-control Factor = 0.6

$$\frac{168-90}{168} \times 0.6 \times 100 = 28\%$$

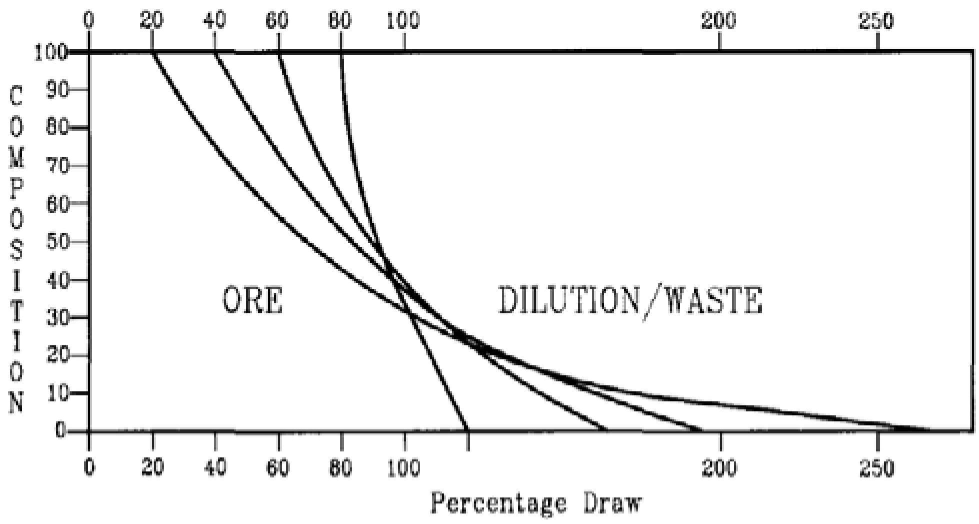

Figure 86: Example calculation of dilution entry into the ore at the drawpoints (after Laubscher, 1994)

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LEARNING OUTCOME 3.3.4.4

DESCRIBE, DISCUSS AND EXPLAIN THE CAUSES OF THE ABOVE RELATIONSHIP

The causes of the above dilution versus draw relationship have been identified in earlier sections. In summary, it is the result of the mixing of ore and waste along the ore/waste boundary. This is a function of:

- Ore draw height
- Range in fragmentation and density of ore and waste
- Height of the interaction zone between adjacent draw ellipsoids (a function of drawpoint spacing and range in tonnages taken from drawpoints)

The relationship is explained as follows. As more ore (as a percentage of the vertical height of the ore column) is drawn from the caved ore body, the overlying waste is also drawn down and enters the ore column. Mixing mostly occurs within the interaction zone, where draw zones between drawpoints overlap, and there is both horizontal and vertical movement of particles. Above this, inter-mixing relies on waste fines moving downwards in voids between ore particles, or denser waste moving down more rapidly than lighter ore.

As the ore/waste interface approaches, the percentage of waste in the mix increases.

If waste appears at the drawpoints very early (e.g. as very fine material that moves easily between ore particles), then ore may still be present in the drawpoints if approximately 200% of the ore column height is drawn – i.e. twice the expected tonnage of the ore is drawn.

If waste appears late, then possibly only 120% of the original ore column height needs to be drawn to get 100% recovery of ore, i.e. 20% waste dilution in total.

3.4. ROCKMASS BEHAVIOUR AROUND SERVICE EXCAVATIONS

3.4.1. ROCKMASS BEHAVIOUR AROUND ACCESS AND SERVICE TUNNELS

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

CONNECTION 21
Refer to
Paper 3.1, Outcome 3.2

LEARNING OUTCOME 3.4.1.1

DESCRIBE, EXPLAIN AND DISCUSS HOW ROCKS AND ROCKMASSES MAY BE CLASSIFIED

CONNECTION 22
Refer to
Paper 3.1, Outcome 3.2.1

LEARNING OUTCOME 3.4.1.2

DESCRIBE, EXPLAIN AND DISCUSS HOW ROCKMASS CLASSIFICATION SYSTEMS MAY BE USED TO PREDICT TUNNEL STABILITY AT SHALLOW, INTERMEDIATE AND GREAT DEPTHS

CONNECTION 23
Refer to
Paper 3.1, Outcome 3.2.2

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LEARNING OUTCOME 3.4.1.3

CONNECTION 24

Refer to Paper 3.1, Outcome 3.2.3

DESCRIBE, EXPLAIN AND DISCUSS THE NATURE OF FRACTURING AROUND TUNNELS AT DEPTH

LEARNING OUTCOME 3.4.1.4

CONNECTION 25

Refer to Paper 3.1, Outcome 3.2.4

DESCRIBE, EXPLAIN AND DISCUSS THE EFFECTS AND CONSEQUENCES OF FRACTURING ON TUNNEL SIZE

LEARNING OUTCOME 3.4.1.5

CONNECTION 26

Refer to Paper 3.1, Outcome 3.2.5

DESCRIBE, EXPLAIN AND DISCUSS THE EFFECTS AND CONSEQUENCES OF FRACTURING ON TUNNEL SHAPE

LEARNING OUTCOME 3.4.1.6

CONNECTION 27

Refer to Paper 3.1, Outcome 3.26

DESCRIBE, EXPLAIN AND DISCUSS THE EFFECTS AND CONSEQUENCES OF FRACTURING ON TUNNEL STABILITY

LEARNING OUTCOME 3.4.1.7

CONNECTION 28

Refer to Paper 3.1, Outcome 3.2.7

DESCRIBE, EXPLAIN AND DISCUSS THE PHENOMENON OF TIME-DEPENDANT FRACTURING AND DETERIORATION

LEARNING OUTCOME 3.4.1.8

CONNECTION 29

Refer to Paper 3.1, Outcome 3.2.8

DESCRIBE, EXPLAIN AND DISCUSS THE EFFECTS AND CONSEQUENCES OF TIME-DEPENDANT FRACTURING AND DETERIORATION ON TUNNEL STABILITY

LEARNING OUTCOME 3.4.1.9

CONNECTION 30

Refer to Paper 3.1, Outcome 3.2.9

DESCRIBE, EXPLAIN AND DISCUSS THE EFFECTS OF STRESS RÉGIME AND GEOLOGICAL STRUCTURE ON ROCK BEHAVIOUR AND TUNNEL STABILITY

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LEARNING OUTCOME 3.4.1.10

CONNECTION 31

Refer to Paper 3.1, Outcome 3.2.10

DESCRIBE, EXPLAIN AND DISCUSS THE EFFECT OF TUNNEL SIZE, TUNNEL SHAPE AND TUNNEL EXCAVATION TECHNIQUE ON ROCK BEHAVIOUR AND TUNNEL STABILITY

LEARNING OUTCOME 3.4.1.11

CONNECTION 32

Refer to Paper 3.1, Outcome 3.2.11

DESCRIBE, EXPLAIN AND DISCUSS THE OPTIMISATION OF TUNNEL SIZE, SHAPE AND ORIENTATION TO SUIT GEOLOGICAL CONSIDERATIONS

LEARNING OUTCOME 3.4.1.12

CONNECTION 33

Refer to Paper 3.1, Outcome 3.2.12

DESCRIBE, EXPLAIN AND DISCUSS THE OPTIMISATION OF TUNNEL SIZE, SHAPE AND ORIENTATION TO SUIT FIELD STRESS CONSIDERATIONS

LEARNING OUTCOME 3.4.1.13

CONNECTION 34

Refer to Paper 3.1, Outcome 3.2.1

DESCRIBE, EXPLAIN AND DISCUSS THE BEHAVIOUR OF ROCK SURROUNDING TUNNELS AT DEPTH DURING SEISMIC EVENTS

LEARNING OUTCOME 3.4.1.14

CONNECTION 35

Refer to Paper 3.1, Outcome 3.2.14

DESCRIBE, EXPLAIN AND DISCUSS THE SITING OF TUNNELS WITH RESPECT TO EXISTING, CURRENT AND FUTURE MINING OPERATIONS

LEARNING OUTCOME 3.4.1.15

CONNECTION 36

Refer to Paper 3.1, Outcome 3.2.15

DESCRIBE, EXPLAIN AND DISCUSS THE SITING OF TUNNELS WITH RESPECT TO GEOLOGICAL STRATIGRAPHY AND STRUCTURES

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LEARNING OUTCOME 3.4.1.16

CONNECTION 37

Refer to
Paper 3.1, Outcome 3.2.16

DESCRIBE HOW THE ROCKWALL CONDITION FACTOR (RCF) IS DETERMINED

LEARNING OUTCOME 3.4.1.17

CONNECTION 38

Refer to
Paper 3.1, Outcome 3.2.17

DESCRIBE AND DISCUSS HOW THE ROCKWALL CONDITION FACTOR MAY BE APPLIED TO PREDICT TUNNEL STABILITY AND SUPPORT REQUIREMENTS

LEARNING OUTCOME 3.4.1.18

CONNECTION 39

Refer to
Paper 3.1, Outcome 3.2.18

APPLY THE ROCKWALL CONDITION FACTOR TO PREDICT TUNNEL STABILITY AND SUPPORT REQUIREMENTS AT DEPTH

LEARNING OUTCOME 3.4.1.19

CONNECTION 40

Refer to
Paper 3.1, Outcome 3.2.19

EVALUATE GIVEN ROCK CONDITIONS AND FIELD STRESSES TO PREDICT EXCAVATION STABILITY

LEARNING OUTCOME 3.4.1.20

CONNECTION 41

Refer to
Paper 3.1, Outcome 3.2.20

EVALUATE GIVEN ROCK CONDITIONS AND FIELD STRESSES TO RECOMMEND REMEDIAL MEASURES TO IMPROVE EXCAVATION STABILITY.

3.4.2. ROCKMASS BEHAVIOUR AROUND SERVICE EXCAVATIONS AND SHAFTS

CONNECTION 42

Refer to
Paper 3.1, Outcome 3.3

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

LEARNING OUTCOME 3.4.2.1

CONNECTION 43

Refer to
Paper 3.1, Outcome 3.3.1

DESCRIBE, EXPLAIN AND DISCUSS HOW ROCKS AND ROCKMASSES MAY BE CLASSIFIED

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LEARNING OUTCOME 3.4.2.2

CONNECTION 44
Refer to
Paper 3.1, Outcome 3.3.2

DESCRIBE, EXPLAIN AND DISCUSS HOW ROCKMASS CLASSIFICATION SYSTEMS MAY BE USED TO PREDICT SERVICE EXCAVATION STABILITY AT SHALLOW, INTERMEDIATE AND GREAT DEPTHS

LEARNING OUTCOME 3.4.2.3

CONNECTION 45
Refer to
Paper 3.1, Outcome 3.3.3

DESCRIBE, EXPLAIN AND DISCUSS THE NATURE OF FRACTURING AROUND VERTICAL SINKING SHAFTS AT DEPTH

LEARNING OUTCOME 3.4.2.4

CONNECTION 46
Refer to
Paper 3.1, Outcome 3.3.4

DESCRIBE, EXPLAIN AND DISCUSS THE NATURE OF FRACTURING AROUND INCLINED SINKING SHAFTS AT DEPTH

LEARNING OUTCOME 3.4.2.5

CONNECTION 47
Refer to
Paper 3.1, Outcome 3.3.5

DESCRIBE, EXPLAIN AND DISCUSS THE NATURE OF FRACTURING AROUND ROCK PASSES AT DEPTH

LEARNING OUTCOME 3.4.2.6

CONNECTION 48
Refer to
Paper 3.1, Outcome 3.3.6

DESCRIBE, EXPLAIN AND DISCUSS THE EFFECT OF STRESS FRACTURING AND GEOLOGICAL STRUCTURE ON THE FINAL SHAPE OF VERTICAL AND INCLINED SHAFTS

LEARNING OUTCOME 3.4.2.7

CONNECTION 49
Refer to
Paper 3.1, Outcome 3.3.7

DESCRIBE, EXPLAIN AND DISCUSS THE EFFECT OF STRESS FRACTURING AND GEOLOGICAL STRUCTURE ON THE FINAL SHAPE OF ROCK PASSES

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LEARNING OUTCOME 3.4.2.8

CONNECTION 50
Refer to
Paper 3.1, Outcome 3.3.8

DESCRIBE, EXPLAIN AND DISCUSS THE CONDITIONS THAT MAY LEAD TO THE FORMATION OF LARGE UNSTABLE ROCK WEDGES IN VERTICAL SHAFT SIDEWALLS

LEARNING OUTCOME 3.4.2.9

CONNECTION 51
Refer to
Paper 3.1, Outcome 3.3.9

DESCRIBE, EXPLAIN AND DISCUSS THE CONDITIONS THAT MAY LEAD TO THE FORMATION OF INSTABILITIES IN ROCK PASSES

LEARNING OUTCOME 3.4.2.10

CONNECTION 52
Refer to
Paper 3.1, Outcome 3.3.10

DESCRIBE, EXPLAIN AND DISCUSS THE NATURE OF FRACTURING AROUND SERVICE EXCAVATIONS AT DEPTH

LEARNING OUTCOME 3.4.2.11

CONNECTION 53
Refer to
Paper 3.1, Outcome 3.3.11

DESCRIBE, EXPLAIN AND DISCUSS THE EFFECTS AND CONSEQUENCES OF FRACTURING ON SERVICE EXCAVATION SIZE

LEARNING OUTCOME 3.4.2.12

CONNECTION 54
Refer to
Paper 3.1, Outcome 3.3.12

DESCRIBE, EXPLAIN AND DISCUSS THE EFFECTS AND CONSEQUENCES OF FRACTURING ON SERVICE EXCAVATION SHAPE

LEARNING OUTCOME 3.4.2.13

CONNECTION 55
Refer to
Paper 3.1, Outcome 3.3.13

DESCRIBE, EXPLAIN AND DISCUSS THE EFFECTS AND CONSEQUENCES OF FRACTURING ON SERVICE EXCAVATION STABILITY

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LEARNING OUTCOME 3.4.2.14

CONNECTION 56

Refer to
Paper 3.1, Outcome 3.3.14

DESCRIBE, EXPLAIN AND DISCUSS THE PHENOMENON OF TIME-DEPENDANT FRACTURING AND DETERIORATION

LEARNING OUTCOME 3.4.2.15

CONNECTION 57

Refer to
Paper 3.1, Outcome 3.3.15

DESCRIBE, EXPLAIN AND DISCUSS THE EFFECTS AND CONSEQUENCES OF TIME-DEPENDANT FRACTURING AND DETERIORATION ON SERVICE EXCAVATION STABILITY

LEARNING OUTCOME 3.4.2.16

CONNECTION 58

Refer to
Paper 3.1, Outcome 3.3.16

DESCRIBE, EXPLAIN AND DISCUSS THE EFFECTS OF STRESS REGIME AND GEOLOGICAL STRUCTURE ON EXCAVATION STABILITY

LEARNING OUTCOME 3.4.2.17

CONNECTION 59

Refer to
Paper 3.1, Outcome 3.3.17

DESCRIBE, EXPLAIN AND DISCUSS THE EFFECTS OF EXCAVATION SIZE, SHAPE AND ORIENTATION ON EXCAVATION STABILITY

LEARNING OUTCOME 3.4.2.18

CONNECTION 60

Refer to
Paper 3.1, Outcome 3.3.18

DESCRIBE, EXPLAIN AND DISCUSS THE EFFECTS OF EXCAVATION TECHNIQUE ON FRACTURING AND THE STABILITY OF SERVICE EXCAVATIONS

LEARNING OUTCOME 3.4.2.19

CONNECTION 61

Refer to
Paper 3.1, Outcome 3.3.19

DESCRIBE, EXPLAIN AND DISCUSS THE EFFECTS OF EXCAVATION SEQUENCE ON STRESS FRACTURING AND THE STABILITY OF SERVICE EXCAVATIONS AT DEPTH

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LEARNING OUTCOME 3.4.2.20

CONNECTION 62

Refer to Paper 3.1, Outcome 3.3.20

DESCRIBE, EXPLAIN AND DISCUSS THE EFFECTS OF SUPPORT SEQUENCE ON STRESS FRACTURING AND THE STABILITY OF SERVICE EXCAVATIONS AT DEPTH

LEARNING OUTCOME 3.4.2.21

CONNECTION 63

Refer to Paper 3.1, Outcome 3.3.21

DESCRIBE, EXPLAIN AND DISCUSS THE OPTIMISATION OF SERVICE EXCAVATION SIZE, SHAPE AND ORIENTATION TO SUIT GEOLOGICAL CONSIDERATIONS

LEARNING OUTCOME 3.4.2.22

CONNECTION 64

Refer to Paper 3.1, Outcome 3.3.22

DESCRIBE, EXPLAIN AND DISCUSS THE OPTIMISATION OF SERVICE EXCAVATION SIZE, SHAPE AND ORIENTATION TO SUIT FIELD STRESS CONSIDERATIONS

LEARNING OUTCOME 3.4.2.23

CONNECTION 65

Refer to Paper 3.1, Outcome 3.3.23

DESCRIBE, EXPLAIN AND DISCUSS THE BEHAVIOUR OF ROCK SURROUNDING SERVICE EXCAVATIONS AT DEPTH DURING SEISMIC EVENTS

LEARNING OUTCOME 3.4.2.24

CONNECTION 66

Refer to Paper 3.1, Outcome 3.3.24

DESCRIBE, EXPLAIN AND DISCUSS THE SITING OF SERVICE EXCAVATIONS WITH RESPECT TO EXISTING, CURRENT AND FUTURE MINING OPERATIONS

LEARNING OUTCOME 3.4.2.25

CONNECTION 67

Refer to Paper 3.1, Outcome 3.3.25

DESCRIBE, EXPLAIN AND DISCUSS THE SITING OF SERVICE EXCAVATIONS WITH RESPECT TO GEOLOGICAL STRATIGRAPHY AND STRUCTURES

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LEARNING OUTCOME 3.4.2.26

CONNECTION 68

Refer to
Paper 3.1, Outcome 3.3.26

DESCRIBE, DISCUSS AND EXPLAIN HOW ROCKMASS CHARACTERISATION MAY BE APPLIED TO PREDICT SERVICE EXCAVATION STABILITY AND SUPPORT REQUIREMENTS

LEARNING OUTCOME 3.4.2.27

CONNECTION 69

Refer to
Paper 3.1, Outcome 3.3.27

DESCRIBE, DISCUSS AND EXPLAIN HOW ROCKMASS CHARACTERISATION MAY BE APPLIED TO PREDICT SERVICE EXCAVATION STABILITY AND SUPPORT REQUIREMENTS

LEARNING OUTCOME 3.4.2.28

CONNECTION 70

Refer to
Paper 3.1, Outcome 3.3.28

APPLY ROCKMASS CHARACTERISATION TECHNIQUES TO PREDICT SERVICE EXCAVATION STABILITY AND SUPPORT REQUIREMENTS

LEARNING OUTCOME 3.4.2.29

CONNECTION 71

Refer to
Paper 3.1, Outcome 3.3.29

EVALUATE GIVEN ROCK CONDITIONS, FIELD STRESSES, GEOLOGICAL CONDITIONS, EXCAVATION SIZES, SHAPES AND ORIENTATIONS TO PREDICT EXCAVATION STABILITY

LEARNING OUTCOME 3.4.2.30

CONNECTION 72

Refer to
Paper 3.1, Outcome 3.3.30

EVALUATE GIVEN ROCK CONDITIONS, FIELD STRESSES, GEOLOGICAL CONDITIONS, EXCAVATION SIZES, SHAPES AND ORIENTATIONS TO RECOMMEND REMEDIAL MEASURES TO IMPROVE EXCAVATION STABILITY

LEARNING OUTCOME 3.4.2.31

CONNECTION 73

Refer to
Paper 3.1, Outcome 3.3.31

EVALUATE GIVEN ROCK CONDITIONS, FIELD STRESSES AND GEOLOGICAL CONDITIONS TO PREDICT SHAFT STABILITY

LEARNING OUTCOME 3.4.2.32

CONNECTION 74

Refer to
Paper 3.1, Outcome 3.3.32

EVALUATE GIVEN ROCK CONDITIONS, FIELD STRESSES AND GEOLOGICAL CONDITIONS TO RECOMMEND REMEDIAL MEASURES TO IMPROVE STABILITY

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CONNECTION 75
Refer to
Paper 2, Outcome 3.2
Paper 3.1, Outcome 3.4

3.5. MINE SEISMOLOGY
Outcomes 3.5.1-3.5.10

NOTE

CONCEPTUAL APPROACH TO OUTCOMES 3.3.1-3.3.10:
Short summary of SOs already covered in Part II (SOs 4 & 5); massive mine design elements with seismic hazard; main drivers of seismicity in massive mines: stress and stress change on pillars, structures, roof, crown pillar, but also caving process; link to improved mining methods; conclusions in terms of seismic risk reduction.

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

LEARNING OUTCOME 3.5.1

CONNECTION 76
Refer to
Paper 3.1, Outcome 3.4.1.1

DEFINE, DESCRIBE AND DISCUSS THE TERM SEISMIC EVENT

LEARNING OUTCOME 3.5.2

LIST, DESCRIBE, EXPLAIN AND DISCUSS THE FACTORS THAT DETERMINE THE NATURE OF SEISMICITY IN MASSIVE MINING

LEARNING OUTCOME 3.5.3

CONNECTION 77
Refer to
Paper 3.1, Outcome 3.4.2.4

DESCRIBE, EXPLAIN AND DISCUSS THE RELATIONSHIP BETWEEN MINE LAYOUT AND SEISMICITY

LEARNING OUTCOME 3.5.4

CONNECTION 78
Refer to
Paper 3.1, Outcome 3.4.2.1.5

DESCRIBE, EXPLAIN AND DISCUSS THE RELATIONSHIP BETWEEN DEPTH AND SEISMICITY

LEARNING OUTCOME 3.5.5

CONNECTION 79
Refer to
Paper 3.1, Outcome 3.4.2.1.6

DESCRIBE, EXPLAIN AND DISCUSS THE RELATIONSHIP BETWEEN ROCK TYPES AND SEISMICITY

LEARNING OUTCOME 3.5.6

CONNECTION 80
Refer to
Paper 3.1, Outcome 3.4.2.1.7

DESCRIBE, EXPLAIN AND DISCUSS THE RELATIONSHIP BETWEEN LOCAL GEOLOGY AND SEISMICITY

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LEARNING OUTCOME 3.5.7

DESCRIBE, EXPLAIN AND DISCUSS THE RELATIONSHIP BETWEEN REGIONAL GEOLOGY AND SEISMICITY

CONNECTION 81

Refer to Paper 3.1, Outcome 3.4.2.1.8

LEARNING OUTCOME 3.5.8

LIST, DESCRIBE, EXPLAIN AND DISCUSS THE MEASURES THAT MAY BE APPLIED TO CONTROL SEISMICITY

CONNECTION 82

Refer to Paper 3.1, Outcome 3.4.2.1.9

LEARNING OUTCOME 3.5.9

DESCRIBE, EXPLAIN AND DISCUSS MEASURES TO REDUCE THE MAXIMUM MAGNITUDE OF SEISMIC EVENTS

CONNECTION 83

Refer to Paper 3.1, Outcome 3.4.2.1.10

LEARNING OUTCOME 3.5.10

DESCRIBE, EXPLAIN AND DISCUSS MEASURES TO REDUCE THE FREQUENCY OF SEISMIC EVENTS

CONNECTION 84

Refer to Paper 3.1, Outcome 3.4.2.1.11

INTERESTING INFO

Examples of mines extracting massive deposits in South Africa include: The diamond mines in the Northern Cape (kimberlite pipes), copper mines near Phalaborwa and Messina, gold bearing quartzites occasionally reach thicknesses that also require massive mining methods, for example at the Target mine near Welkom (Free State) and at South Deep Mine in the West Rand.

Drivers of seismicity

In a mining environment, instead of tectonic processes driving stress accumulation and strain release, the creation of excavations, their position and their shapes are seen as critical factors with respect to causes of seismic failure. Apart from excavations, there is a range of mining strategies and geological conditions known to be associated with seismic events in mines. These conditions occur in different geotechnical areas and under varying states of stress. It is not surprising therefore that seismic response to mining activity varies greatly and that seismic events bear characteristics of the conditions under which they occur.

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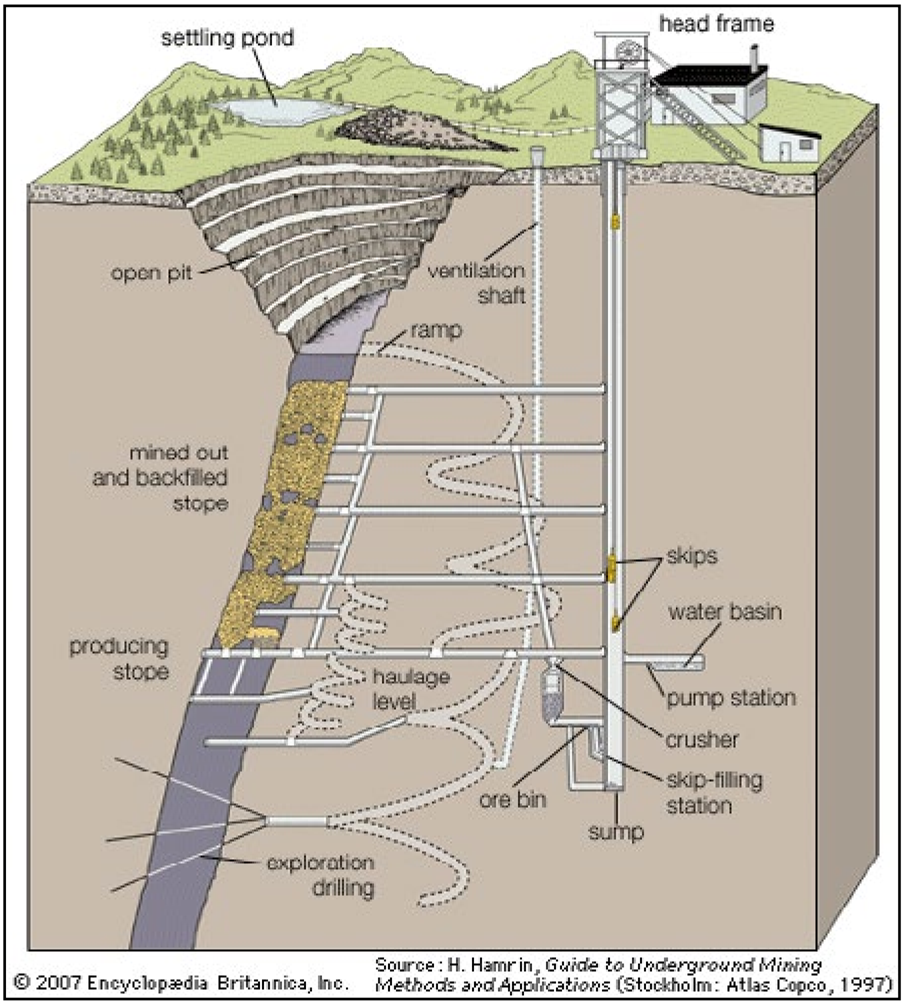

Figure 87: Layout of an underground mine beneath an open pit (Hamrin, 1997; source: Encyclopaedia Britannica, 2007)

Learn more about base metals and minerals mined in South Africa from the Chamber of Mines.
http://www.bullion.org.za

The excavations created to extract massive ore bodies are marked by much greater height than those common in tabular mining. Deposit shapes vary, but are usually significantly taller and longer than thick (see Figure 87). Frequently, a massive underground operation is the successor to an earlier open pit that reached the end of its life. Both De Beer's diamond mine in Kimberley and the Phalaborwa copper mine are examples of such a development. The result is an underground operation overlain by a deep open pit. The two are separated by a horizontal pillar of solid rock that forms the stable bottom of the pit.

Massive mining environments are marked by shallow to intermediate stress levels and mostly strong rock types with confined strengths in the range of several 10 to well above 100MPa. The resultant mining environment is marked by a low potential for sudden, violent release of strain energy, i.e. is exposed to low levels of seismic hazard.

In principle, seismicity is induced and triggered by either high stress levels (in excess of rockmass strength), or high rates of stress change. The purpose of this section is to describe in detail the specific circumstances of mining-related seismicity in South African massive, hard rock mines in order to derive an understanding of the underlying causes and

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the contributing factors, all of which are essential to the management of seismic risk.

Seismic events and rock bursts
A seismic event (called earthquake when caused by tectonic forces in the earth's crust) is the result of strain accumulation in rock due to stress. When stresses exceed the rock strength (or the cohesion and friction on a discontinuity plane), sudden failure occurs accompanied by the release of strain energy. By definition, the energy release has to happen rapidly, in fractions of a second, in order to be called 'seismic'. A rockmass failure incident that happens slowly or in distinct stages and does not emit seismic waves is by definition not a seismic failure.

Shown below are typical seismic failure mechanisms as they are observed in and around mining excavations:

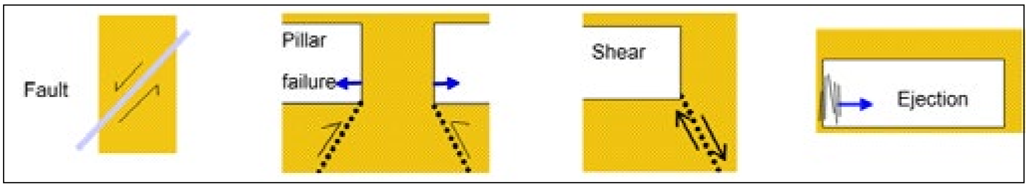

Figure 88: Seismic failure types typical for intermediate depth massive mines

The sidewall ejections are observed where high face stresses are combined with brittle rock types and high wall heights. Where the failure occurs close to the skin of a void, it may result in the ejection of face rock into the stope. Where the failure locates deeper in the rockmass, it may constitute a pillar failure where the source of strain energy release is within the pillar volume. In both cases, sidewall rock is ejected into the excavation.

DEFINITION

Seismic event = Sudden release of strain energy due to stresses exceeding the rockmass strength

Rockburst = A seismic event that results in damage to excavations

Shake-down, crushing, footwall heave and side wall ejections are examples of damage mechanisms.

'Seismic' is any physical process generating seismic waves.

When investigating rock bursts, the separation of failure and damage mechanism is essential as the same failure mechanism can lead to a range of different damage types. The photos in Figure 89 illustrate two of these damage forms.

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Figure 89: Sidewall ejection after pillar failure (left) and complete collapse of a tunnel as a result of seismic energy release

LINK

- Find useful publications on open pit and underground mining practice at the Australian Centre for Geomechanics web site

http://www.acg.uwa.edu.au/publications

The following factors are seen as governing seismicity in massive underground mines:

1. Absolute stress levels on structures and near excavations being a function of mainly depth, pillar shape, excavation size and geotechnical setting (including k-ratio);
2. The rate of stress change, which is closely related to the volume of

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rock being removed by undercut blasting or production, draws in a caving operation;

3. The rock strength in relation to the stress peaks and stress changes;
4. The orientation of major and minor stress components with regard to excavations;
5. Existence of pillars, abutments and seismically-active geological structures;

Absence of mitigating strategies, such as bracket and stabilising pillars, backfill and optimised mining sequences to reduce stress accumulations.

The list is in no particular order. Which of these factors are dominant in a certain mine depends on local conditions. While location, damage mechanism and intensity of damage are functions of seismic source characteristics, two other factors also play an important role, namely size and shape of the excavation, and the characteristics of the installed ground support.

Obviously, larger seismic sources directly affect larger rock volumes. They may result in extensive crushing, large-scale falls of ground, and significant stope closure. For a given magnitude, high moment-low energy sources have larger source dimensions than the opposite. Similarly, high energy-low moment sources radiate waves with greater ground motion velocities and acceleration increasing the likelihood of violent shakedown.

Events of which the main failure mechanism is slip typically have ES/EP ratios well above 10, with an average for a mixed mechanism being in the region of 3-10. Figure 90 shows that for several thousands of events recorded on a mine, this ratio lies well below 10. Slip events radiate strong S-waves, which, under certain conditions, are considered more damaging than P-waves.

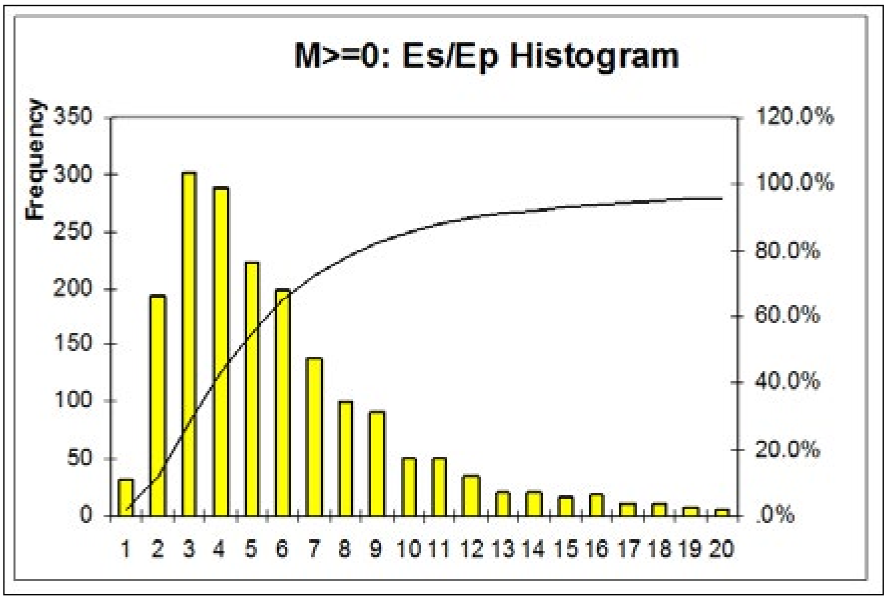

Figure 90: Es / Ep ratio

Relevant in terms of support performance characteristics is the ability to withstand dynamic loading at high ground velocities and the areal support provided to prevent hangingwall collapse between units. The specific combination of depth, mining strategy and geotechnical setting together determine the seismic emission sources relevant to a mining

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operation. In massive underground mining environments, we find field stresses exceeding the in situ rock strength, crown pillars of which the sizes are reduced by cave expansion, excavation spans much larger than the height of excavations, and strong, brittle rock types.

One or more of the following seismic emission sources are found on shallow to intermediate depth massive mines:

- small scale fracturing ahead and above the undercut zone;
- cave failure;
- crown pillar failure;
- failure of stability pillars and abutments;
- structural failure in the form of slip on faults and bursting of intrusions;
- combination of geology-related failure with pillars and abutments.

It should be noted that there is no simple relationship between seismic energy emission and the way in which excavations are affected other than to state that the largest failures usually occur on geological features. Therefore, geology-related shear sources are likely to generate the highest ground velocity values in their vicinity.

It is evident from the above that high magnitude events involve large rock volumes and release large amounts of seismic energy, which raises the likelihood of damage to existing excavations with subsequent material losses. Extensive source volumes may directly include excavations, while strong ground motion is known to be associated with high seismic energy release.

In practice, large magnitude events are often located away from tunnels and stopes resulting in a greater distance between source and excavations, which mitigates the effects of wave interaction. Generally, one can expect a positive correlation between peak particle velocity and damage caused by an event because the extent of violent shaking, especially high ground acceleration, raises the probability of induced falls of ground or even failure of support elements.

Principal contributing factors

The mining layout plays a critical role in the level of seismic response that can be expected during the extraction of a massive deposit. Since seismic response is driven by stresses and stress changes over time, all design elements that attract high stress levels are prone to seismic failure, provided stresses also exceed the in loco rock strength. Examples of design elements with potential for seismic failure are stabilising pillars, crown pillars, corners and abutments, and isolated blocks of ground in a remnant-mining situation.

Depth is the prevailing factor for increases in the (virgin) major principal stress, unless horizontal tectonic forces dominate the stress field. It is therefore also likely to be an important factor ruling the levels of seismicity. In shallow mines, a high k-ratio can be of even higher importance than depth when it comes to field stresses that may initiate seismic failure. An increase in mining depth over time usually increases horizontal stress levels, which results in higher confinement, a factor that possibly contributes to confinement and rock strength.

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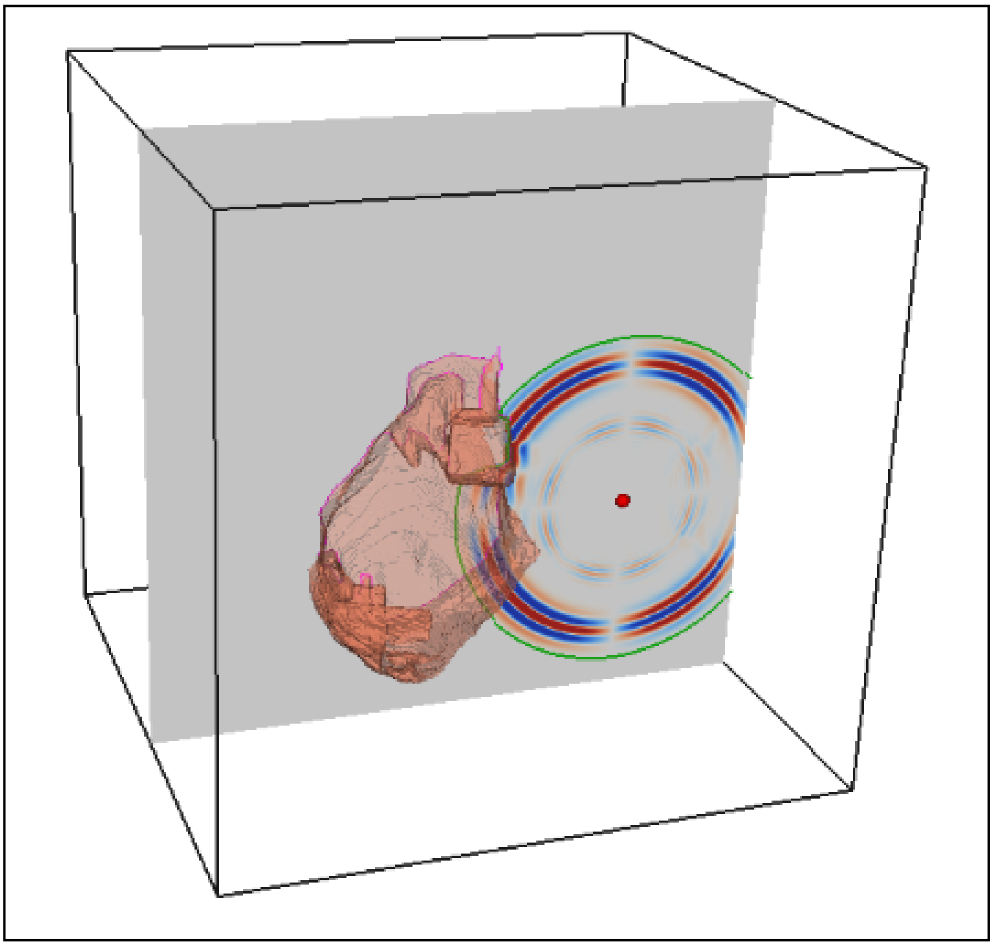

Figure 91: 3D-model of seismic waves interacting with an ore body (Vavryčuk & Kühn, 2011)

Seismicity is the result of a sudden failure of rock due to stress levels exceeding rock strength. Prior to its release in the form of seismic energy, strain energy is being stored in the rockmass, which is more likely to take place in hard and brittle rocks and less likely in ductile rocks. Brittle rock types store higher amounts of strain energy than ductile rocks do, and therefore release relatively higher quantities of seismic energy per unit of rock volume. Softer rock types, on the other hand, tend to produce seismic events with larger source volumes and greater amounts of deformation.

The local geology, in particular the presence of faults and dykes with traceable lengths of tens to hundreds of metres, is likely to contribute to seismicity in the medium magnitude range, which is equivalent to source dimensions up to 100-150m. Steeper dipping structures are more likely to slip than flatter dipping ones where the major principal stress is vertical; the opposite is true where significant horizontal forces exist.

Sills and other intrusions are known to concentrate stresses because of their material strength often being higher than the host rock, resulting in seismic failure with significant energy release. Jointing, where it exists with a common orientation over larger areas, can be a determining factor with regard to the slip direction of geology-related failures.

APPLICATION

Divide the major geological features on your mine into these categories:

1. Safe (no seismicity, low hazard)
2. Active when mined

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3. Active when approached (high hazard)

The regional geology, i.e. fault systems and dykes of regional importance are known sources of the very large events that are occasionally recorded in gold mining districts. These regional failures make up the top end of the seismic data catalogue reaching magnitudes above M_L=5. The following examples are quoted from Durrheim (2006):

Date	Magnitude	Region
8-Dec-1976	5.1	Welkom: Dagbreek Fault
28-Oct-1986	4.8	Klerksdorp: No.2 Shaft fault
23-Apr-1999	5.1	Welkom-Mathjabeng Mine: Dagbreek Fault
9-Mar-2005	5.3	Klerksdorp – DRD 5 Shaft & Stilfontein Town: 5 Shaft Fault

Events up to local magnitude 2.9 were reported during production caving in a massive mining operation (Hudyma et al., 2008). These events were located within a geologically complex zone some distance from the cave and outside the crown pillar. Other authors associate large seismic events in a massive mine with failure of the crown pillar itself (Glazer et al., 2005).

Control of seismicity

Any measure that reduces field stresses and slows down the factors that give rise to changes in stress is bound to reduce the level of seismicity. The frequency of potentially harmful events could be reduced by spreading mining activity over larger volumes and by reducing the rate of mining. The severity of failures, i.e. Mmax, is reduced by eliminating potential sources of large events such as pillars.

Admittedly, these measures are more limited in massive mines than in tabular mines, because of the reduced flexibility compared to scattered or sequential grid mining, for instance. However, seismicity-inducing factors such as blasting can be reduced; the cave shape and size can be controlled to some extent; and pillars can be designed to mitigate uncontrolled rockmass failure.

For research on the link between mine design and dynamic rockmass response refer to SIMRAC project GAP612.

REFERENCE

- gap612 Link between mine design and seismic response

Specifically, the elimination of large potential seismic sources can limit maximum magnitude, since magnitude is closely related to seismic source size. Among those measures are:

- bracketing of unstable geological structures where possible;
- limiting the length of abutments;
- reducing pillar sizes;
- limiting the cave size;
- ensuring the stability of stabilising pillars; and
- optimising sequence and angle of approach when negotiating structures.

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The largest magnitude of events occur on objects and in zones that accumulate high stresses and allow large amounts of deformation, such as pillars and fault systems. Preventing these from failing reduces the maximum event size.

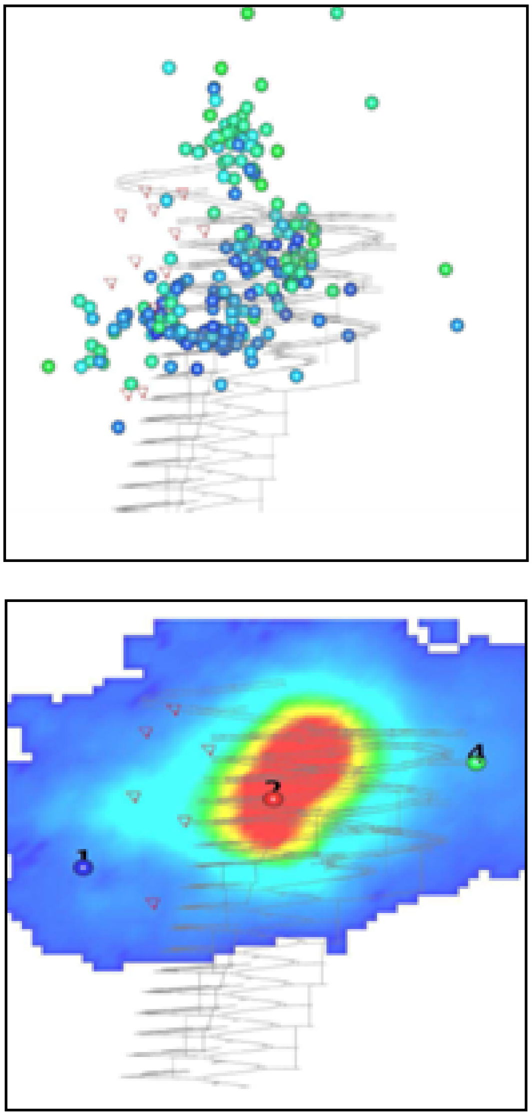

Figure 92: (a) Seismic events in the hangingwall block of a steeply dipping gold deposit (=seismic sensors), (b) seismicity-related deformation based on seismic data shown in (a)

The predominant factor driving seismic event frequency is the amount of development and undercut blasting on a mine. Removing stressed rock from the excavation surface by means of blasting initiates a process of stress change that exposes fresh, previously intact rock to high face stresses. It leads to fracturing on various scales, from micro-fracturing to shear fracturing extending several metres into the hanging- and footwall. This is observed as seismic response to blasting and causes well-known activity peaks at blasting time. The typical decay in activity rate after blasts can be used to establish a safe re-entry period, i.e. the time after a production blast that marks with near certainty the beginning of a period of minimal seismic risk for workers to enter working places.

In caving operations, the advance rate of undercut blasting and the cave propagation because of production draws determining the seismic event frequency.

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Outcomes 3.3.11- 3.3.16

NOTE

Conceptual approach to Outcomes 3.3.11-3.3.16:
Short summary of SOs already covered in Part II (SOs 4 & 5); mine design elements with seismic hazard; main drivers of seismicity in mines: stress and stress change; link to improved mining methods; and conclusions in terms of seismic risk reduction;
Motivate rockburst investigations, main aspects, and main outcomes; explain rockburst risk and methods to manage it.

LEARNING OUTCOME 3.5.11

CONNECTION 86

Refer to
Paper 3.1, Outcome 3.4.2.1.13, 3.4.1.32

EXPLAIN THE DIFFERENCE BETWEEN LOCAL MAGNITUDE AND RICHTER MAGNITUDE

LEARNING OUTCOME 3.5.12

CONNECTION 87

Refer to
Paper 3.1, Outcome 3.4.2.1.16, 3.4.1.32

DESCRIBE AND EXPLAIN THE FOLLOWING COMMON METHODS OF SEISMIC DATA ANALYSIS

- Gutenberg-Richter analysis
- Magnitude-frequency relationship
- Energy-moment relationship
- Trend analysis

LEARNING OUTCOME 3.5.13

CONNECTION 88

Refer to
Paper 3.1 Outcome 3.4.2.1.18

DESCRIBE, EXPLAIN AND DISCUSS THE RELATIONSHIPS BETWEEN:

- Event magnitude and damage caused by seismic events
- Peak particle velocity and damage caused by seismic events

LEARNING OUTCOME 3.5.14

CONNECTION 89

Refer to
Paper 3.1, Outcome 3.4.1.6

DEFINE AND DISCUSS THE TERM ‘ROCKBURST’

LEARNING OUTCOME 3.5.15

CONNECTION 90

Refer to
Paper 3.1, Outcome 3.4.2.1.24

DESCRIBE, EXPLAIN, DISCUSS AND CONTRAST THE RELATIONSHIP BETWEEN:

- Rockburst damage and seismic source characteristics
- Rockburst damage and excavation support characteristics

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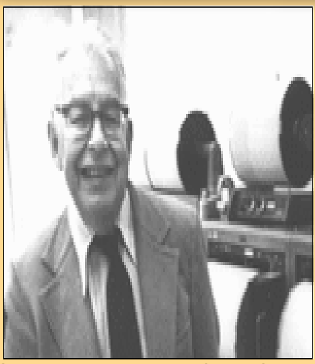
LEARNING OUTCOME 3.5.16

DESCRIBE, EXPLAIN AND DISCUSS THE DIFFERENT TYPES OF SEISMIC EMISSION SOURCES IN MASSIVE MINING

Data analysis

INTERESTING INFO



Charles F Richter of the California Institute of Technology

Seismic data analysis rests on the prior collection of seismic event information, both in time and in space. It may be instructive to reiterate the complexity of the task of quantifying a seismic event, which consists of a description of a remote, physical process of which no direct observations exist, simply on the basis of sound recordings of this process. The simplest requirements would consist of determining at least the accurate time, 3D-location and a measure of size of the event that would allow the recipient to identify the area on the mine that may have been affected and some initial estimate of the possible consequences. All of this detail must be gained from recordings of ground motion that were taken hundreds or thousands of metres away from this process by means of the application of suitable analysis methods.

Magnitude especially can be defined and calculated in a number of different ways, which results in a range of different interpretations of this parameter. As examples, consider Richter magnitude, moment magnitude, duration and body wave magnitude and others. Of central importance in mine seismology is the site-dependant 'local magnitude' ML introduced earlier.

DEFINITION

Richter Magnitude M_L
Defined in 1935 by Charles Richter as the logarithm of the maximum amplitude of ground motion, measured in micro-metres by a standard Wood-Andersen seismograph at a distance of 100km from the epicentre. The unit of ML is metres/second [m/s].
Hanks-Kanamori Magnitude
*Defined by Hanks and Kanamori in 1979 as MHK= 2/3 log(Mo)-6.0 with the unit Newton*metre [Nm].*

For users of ISS networks, local mine magnitude MM, a general earthquake strength parameter, is based on both seismic energy and moment and is strictly speaking not a primary source parameter. MM combines the two main aspects – deformation and energy – into one parameter that can provide an initial indication of the overall strength of an event:

Typical values are approximately 0.5 for A, 0.3 for B and -5 for C. By modifying the three coefficients, the local magnitude scale may be adjusted to match other magnitude scales, such as the one used by the Council for Geo-science to report large tremors.

It is clear from the magnitude relation that events can have the same magnitude whether their respective energy is high and moment is low or vice versa. This reduces the usefulness of magnitude somewhat as it leaves doubts as to what contributed to the source strength: slip area or slip velocity. ML should be seen as a general measure of size, with other parameters providing clarity on the details of source.

Users of other mine seismic systems may receive mine magnitude values based on energy or on moment, which are all equally 'correct', as there is no right or wrong way to calculate this parameter. What should be kept in mind is the methodology of calculation and that the various methods can be adapted and respective magnitudes converted from one system to another without too much effort.

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PAPER 3.3: CHAPTER 3

What is of relevance to the analysis and interpretation of these parameters are their trends over time and in space. Common analysis procedures focus either on the characteristics of an event group recorded in a specific location and over a pre-defined time period and how these characteristics may change as time passes. 'Passing time' within this context means a change in mining practice, rockmass conditions and stress field as tunnels are developed and stopes expanded.

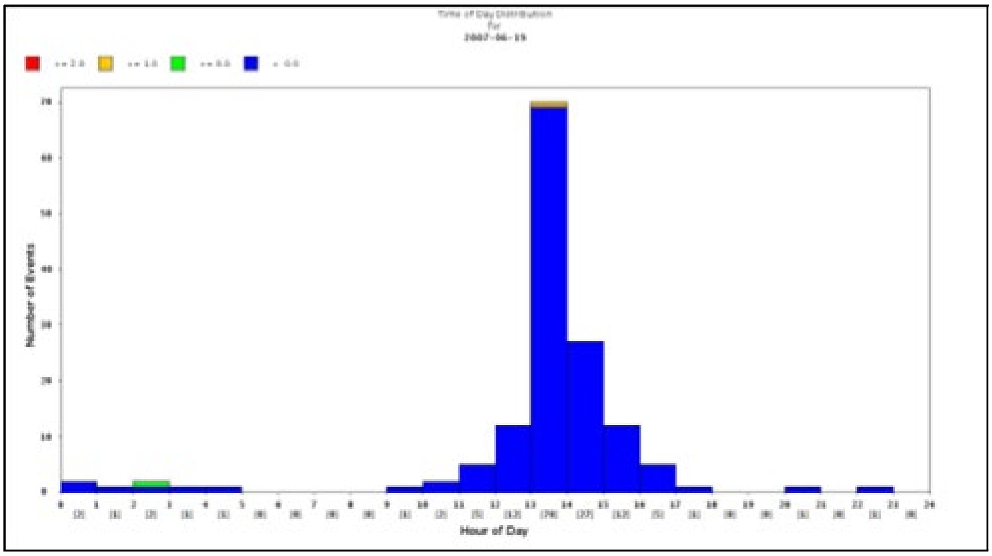

Figure 93: Typical time-of-day distribution with high activity around the time of the blast

A range of data analysis tools are used, which are based on the common set of seismic parameters describing source location in time and space, aspects of energy and moment release, stress drop and source dimension. These tools have their roots either in global tectonic seismology or were developed specifically for mining applications in South Africa.

Gutenberg-Richter analysis

The Gutenberg-Richter analysis belongs to the first group. It describes the cumulative frequency-magnitude distribution of a seismic data catalogue. It shows, in the form of a histogram, the number of events having occurred with magnitude equal to or larger than a given magnitude threshold (blue bars in Figure 94).

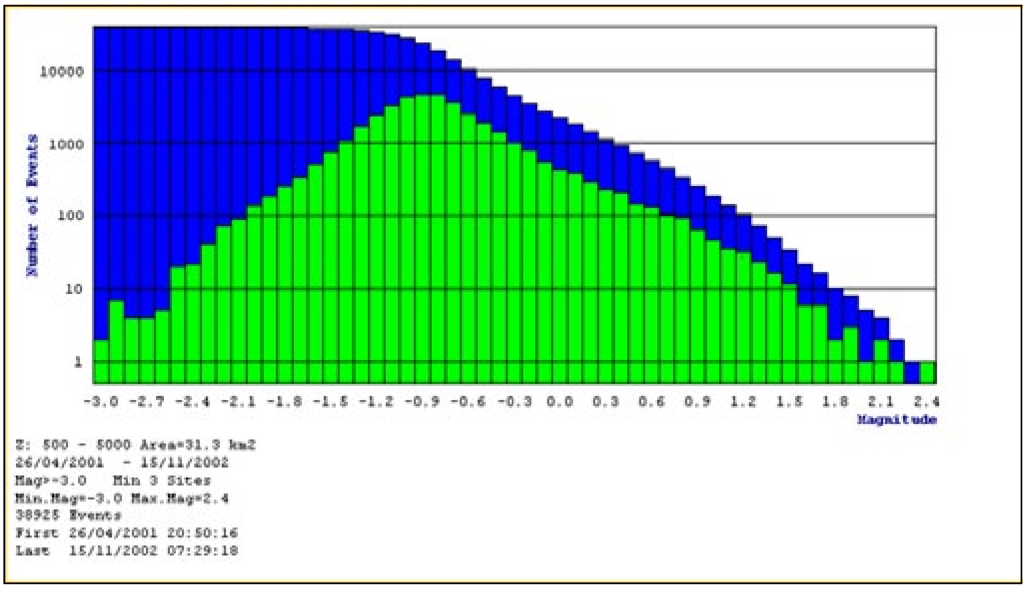

Figure 94: Gutenberg-Richter graph characterising properties of a seismic event group and the network that recorded the events

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In Figure 94, green bars reflect the frequency of events in each magnitude bin, i.e. a standard histogram of magnitude, while blue bars provide cumulative values. Axes are generally in a logarithmic scale to accommodate the large number of events in most datasets (the dataset in the illustration below contains approximately 39 000 events of which roughly 2 000 are ML≥0.0 and approximately 200 events ML≥1.0.

NOTE

In some incidents of earthquake sequences, the b-value decreased prior to the main shock, which means that temporarily more medium-sized events occurred (Mendecki, 1997)

The Gutenberg-Richter analysis yields a measure of the ratio of small to large events, which is known as the 'b-slope'. The effect of different b-values on the distribution is such that for an increase in magnitude by 1, the number of events decreases by the following factor:

- for b = 1   10 times
- for b = 1.5 30 times
- for b = 0.5   3 times

The Gutenberg-Richter graph also allows an estimate of the largest potential magnitude Mmax and of the sensitivity of the network, i.e. the magnitude above which all events are reliably recorded. This threshold is called the minimum magnitude or Mmin.

For static seismic hazard quantification based on GR analysis, refer to SIMRAC project GAP517:

LINK

- GAP517 Quantification of seismic hazards

The steeper the slope of the graph, the fewer large events occur for a given number of small events, which equates lower hazard. The shape of the graph is also of significance as it may indicate either a single or a range of different failure processes. In mine seismic data, GR graphs often contain more than one event population, which results in two or more different b-slopes visible in different parts of the graphs. The example in Figure 95 exhibits a bi-modal shape: A shallow slope in the lower magnitude range approximately up to M=-1.1 and a steeper slope above this magnitude.

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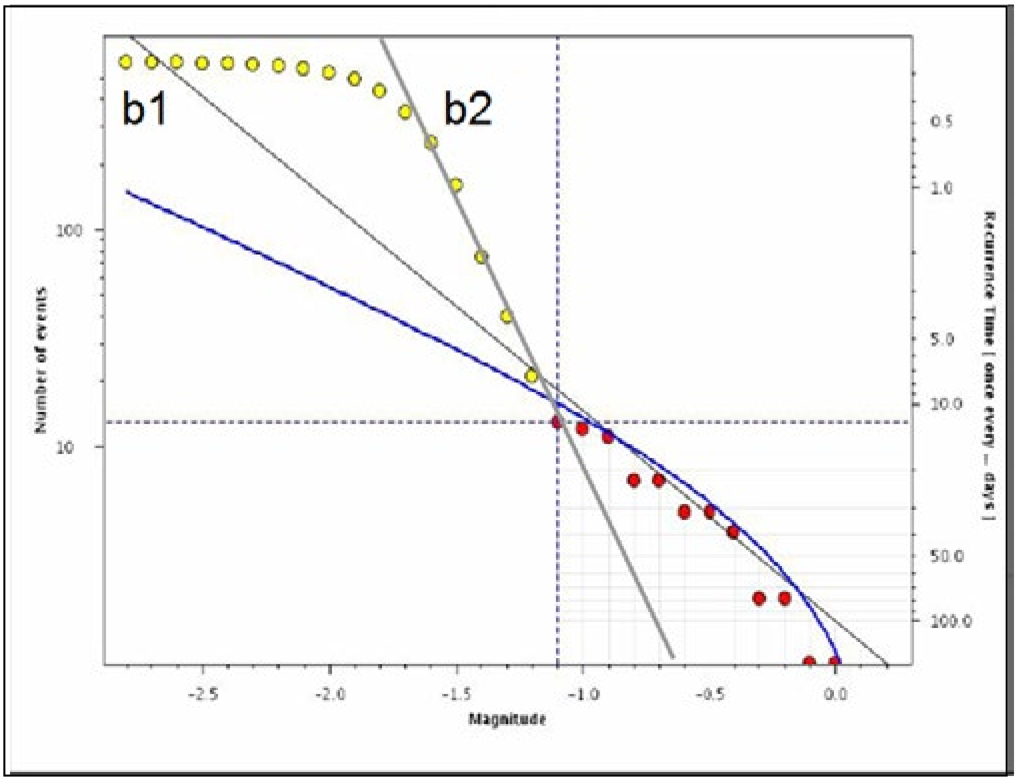

Figure 95: GR graph with two distinct event populations

Magnitude-frequency relationship

The Gutenberg-Richter relation mathematically describes the distribution shown graphically in the example above. Theoretical considerations suggest that for each increase in magnitude, there should be a decrease in event numbers by approximately 90%: For each N events between ML=1 and ML=2, there should be roughly 10N events between ML=0 and ML=1. This is expressed by the Gutenberg-Richter relation:

$$N(M_1) = 10^{a-bML}$$

with magnitude M_L, a and b constants.

N is the number of events with size equal to or larger than ML, where ML could also be log of moment, energy, or a magnitude based on these two parameters. Note that 'a' merely expresses the overall level of activity, while 'b' refers to the steepness of the graph, which is a measure of the ratio of small to large events. By convention, the value of b is always taken as a positive number, even though the slope of the linear fit is negative (e.g. where the slope is -1.3 for example, the relevant b value to be inserted into above equation would be 1.3).

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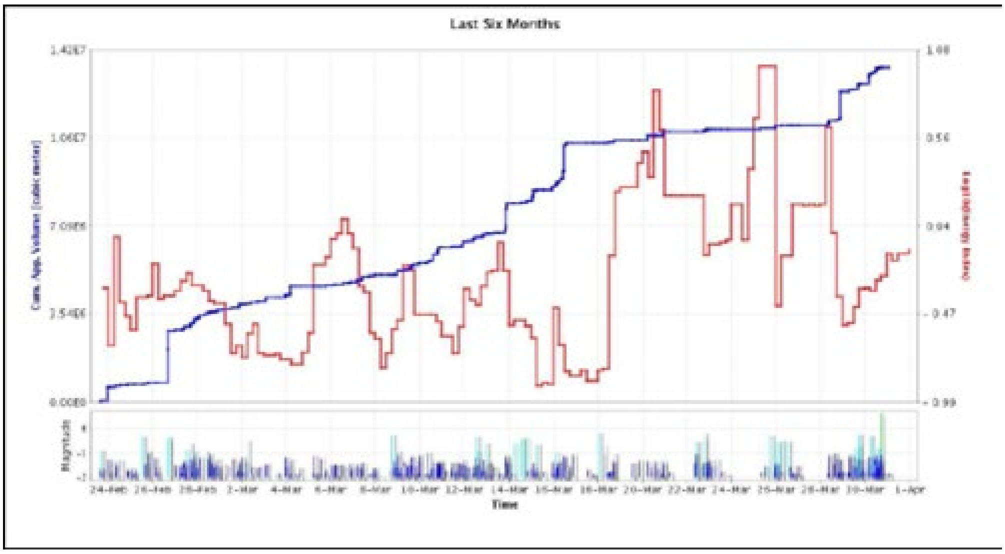

Figure 96: Energy index and cumulative seismic deformation plot with time

Cumulative seismic deformation and log(Energy Index) traced over a six-month period reflect strain rates and stress changes associated with seismic processes. The use of averaging time windows is common in trend analysis.

NOTE

'Smoothing' of time series can significantly change the shape of a graph!

Energy-moment relationship

The energy-moment graph and its associated EM relation can reveal unique features and qualities of a cluster of events. It is based on a scatter plot of log10 (seismic moment) versus log10 (seismic energy) with energy being the dependant variable (Figure 97).

In a routine analysis, the data is represented by a linear fit that describes the positive correlation between the two parameters. In this case, the linear regression suggests a best fit with slope 1.146 and offset -7.073. The corresponding EM relation is log(En)=1.5+log(Mo)-10.6.

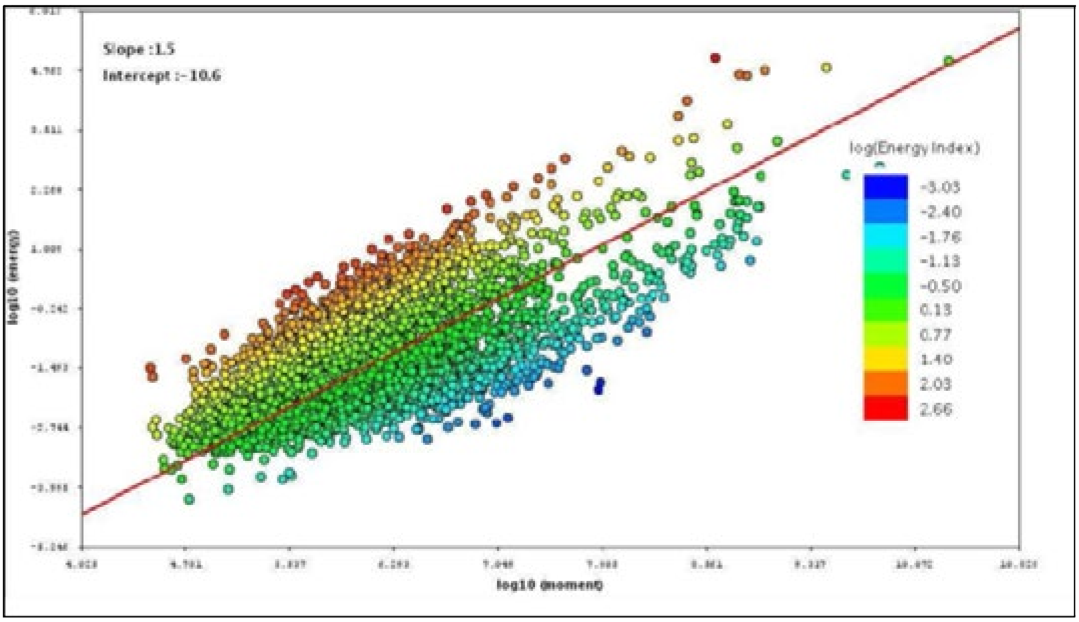

Figure 97: Seismic moment vs seismic energy scatter plot

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Obviously, the steeper the slope of the line, the higher the energy released by large events for an event of the same moment is. It may be surprising that the slope not only differs between mining areas, but varies over time when analysing data from the same area. Within this context, it should be noted that apparent stress does NOT vary with magnitude over larger ranges of magnitudes.

The most pertinent use of the EM relation is for the calculation of the energy index (EI), which was explained in detail in an earlier chapter. The slope of the EM relation has been shown to correlate with loading system stiffness, which relates to mining layout and rockmass properties. Large span excavations in a soft, ductile rockmass make for a soft loading system, while the opposite is true for a mining layout incorporating limited spans in a hard-rock environment.

Trend analysis

Trend analysis is a widely-used tool in qualitative seismic hazard analysis, both in short- and medium-term intervals. The underlying idea is that seismicity parameters, such as activity rate and cumulative seismic deformation, correlate with seismic hazard in the way that upward trends in recorded seismicity indicate a higher likelihood of seismic damage in future.

This may be useful where the inputs into the 'seismic response equation' remain constant, for instance when production rate and mining configuration do not change significantly. Then, a steepening of the slope would be an expression of a more intense and potentially more hazardous rockmass response to the mining process.

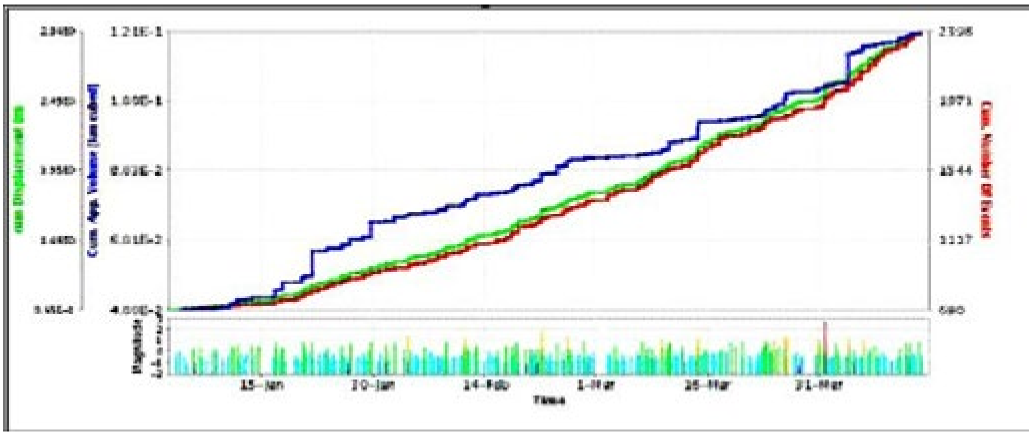

Figure 98: Seismic history plot of cumulative Vapp (blue), co-seismic displacement (green) and activity rate (red)

Graphs of cumulative parameters are interpreted by evaluating the slope of the graph rather than the actual parameter values. Periods of steepening in cumulative Vapp for instance indicate faster deformation compared to flatter periods. This could be caused by an average increase in source size or more frequent large events. Unusually, large events are recognisable by the steps they cause in the cumulative deformation graph, as seen during mid-January in Figure 98.

The detection of time-dependant features in seismicity is of central importance to the management of seismic risk. Of equal importance are spatial variations, for instance for the control of caving processes in mines with massive ore bodies. Monitoring of the caving behaviour is an important objective for such mines, as it delivers information on a process that is generally inaccessible and difficult to trace. The example from an Australian Gold mine (Figure 99) illustrates the potential benefit

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and the accuracy of event locations achieved with 3D network layouts.

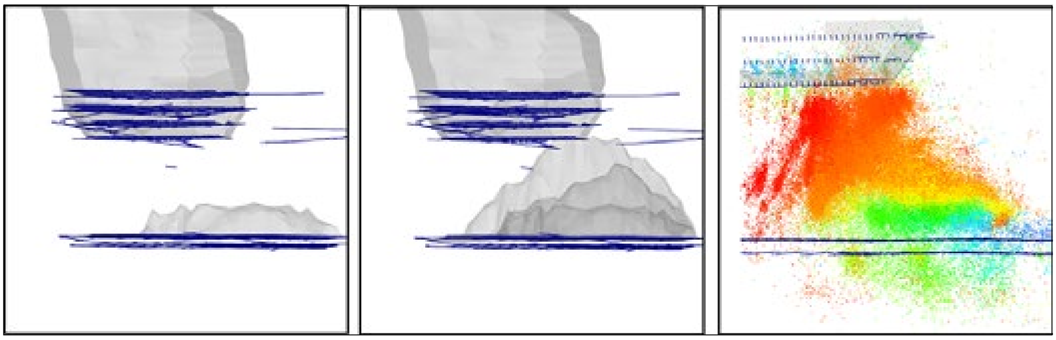

Figure 99: Caving progress and resulting seismic response (right) at Ridgeway Gold Mine, AUS (G Capes, 2010)

LEARNING OUTCOME 3.5.17

NOTE

Conceptual approach to Outcome 3.5.17:
Explain intent of regulator and purpose of seismic section of CoP to control risk; how to compile various sections; responsibilities of operator; CoPs are not approved by DMR.

LEARNING OUTCOME 3.5.17

CONNECTION 91
Refer to Paper 3.1, Outcome 3.4.2.1.26

DEMONSTRATE FAMILIARITY WITH THE GUIDELINE FOR THE COMPILATION OF A CODE OF PRACTICE TO COMBAT ROCK-RELATED ACCIDENTS BY BEING ABLE TO:

- Define, describe and explain the term 'seismically active' as defined in the guideline
- Describe and explain the steps necessary to define emission sources in various ground control districts
- Describe and explain the term 'reasonably practicable' within the context of seismic risk management
- Describe and explain the various rockburst damage control measures
- Describe and explain the use of a seismic system to monitor blasting practice, caving activity and slope stability

Code of Practice (seismic section)
Although the guidelines for the mandatory Code of Practice for massive mines do not dedicate much detail to seismicity issues, seismic hazard is one of the threats to a mining operation and its employees that has to be taken seriously. Just as is the case with any other hazard, it has to be monitored, analysed and mitigated where possible.

LINK

- COP Guidelines_in_massive_mining_operations

The main components of the seismic section of a CoP are discussed below, starting with the definition of 'seismically active' borrowed from the CoP for hard-rock tabular mines:

DEFINITION

Seismically active mine means a mine that sustains losses to persons and/or property, underground or on surface, caused by the dynamic response to a seismic event induced when creating or enlarging an excavation (from DME 7/4/118 AB1, AUGUST 2001).

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INTERESTING INFO

The 'Leon Commission' of inquiry into safety and health in the mining industry recommended (May 1994) that:

- Employees have the right to remove themselves from any place where they have reasonable cause to think that their health or safety is endangered.
- A methodology to be developed, e.g. risk management, to target the most pressing hazards.
- Mine manager is responsible for suitable and sufficient assessments of the risks to which persons are exposed.
- A workmen's representative has the right to relevant training, be informed of accidents, have access to all information in health and safety matters, and to be consulted.

The identification of relevant emission sources in a mining area requires careful analysis of seismic information with regard to location, time of occurrence and failure mechanism. Data analysis with regard to emission sources requires the seismic network location accuracy to be in the same range as the seismic source dimensions, i.e. a few metres for face-related events, a few tens of metres around pillars and abutments and approximately 100m along geological structures and in the back area. A further requirement is good spatial configuration to obtain reliable estimates of stress drop, ES/EP and, where possible, moment tensor inversions.

Obviously, terminating mining operations and removing equipment and personnel from underground would effectively eliminate the rockburst risk. However, it would also eliminate the possibility to mine the reserve profitably.

DEFINITION

Reasonably practicable means that a compromise must be found that allows mining operations to continue while at the same time ensuring the safety of workers underground.

Mitigating measures must be implemented that are affordable and effective to reduce the level of seismic hazard and the exposure to loss. In order to be reasonably practicable, an intervention should be cost-benefit balanced.

Contrary to seismic emission control, which aims to reduce the frequency and severity of seismicity, rockburst damage control aims to alleviate the negative effects that seismicity may have on the mining operation. The primary control measures are:

- installation of slope and excavation support types that are designed by competent persons for dynamic loading conditions at high ground velocities;
- effective rockburst hazard assessment methods and relevant procedures to communicate and act on the results of such assessments;
- identification of a mine's typical rockburst scenario (where it exists) and the systematic elimination of the defining elements of this scenario; and
- PPE issued to and worn by the workforce at all times.

It is also a requirement stipulated by the guidelines that "the COP must set out a description of how processed seismic data is to be used in the mine design methodology and operating procedures". Designs must take into account the stress field and any changes therein, available techniques such as rockmass classification, numerical modelling, seismic data analysis, support design criteria, and the influence that the size of the excavation and blasting technique have on the stability of excavations. The support strategy for tunnels and service excavations must be

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described (Annex 1 of the CoP Guidelines).

In its entirety, the code of practice must detail a mine's efforts to monitor seismic rockmass response, describe the analysis and reporting procedures, outline measures to mitigate seismicity and plans to reduce the negative impacts of seismicity.

REFERENCE

- Guideline for the compilation of a mandatory code of practice to combat rockmass failure and accidents in massive mining operations, DME Mine Health & Safety Inspectorate, Dec. 2006

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CHAPTER

4

PAPER 3.3: MASSIVE UNDERGROUND MINING (HARD AND SOFT ROCK)

LEARNING OUTCOMES

4. MINING LAYOUT STRATEGIES

4.1. MASSIVE MINING METHODS

4.1.1. CHOICE OF MASSIVE MINING METHOD

The candidate must demonstrate knowledge and understanding of the above subject area by being able to:

- Sketch, describe, explain and discuss the fundamental mining and rock engineering principles associated with the following massive mining methods:
  - Open stoping methods,
  - Vertical crater retreat,
  - Sub-level caving methods,
  - Block caving methods,
  - Cut and fill,
  - Drift and fill,
  - Room and pillar;
- Sketch, describe, explain and discuss the layout of the above mining methods;
- Sketch, describe, explain and discuss the location and stability of access tunnels and service excavations; and
- Sketch, describe, explain and discuss how the following ore body geometries and/or combinations of ore body geometries may be mined:
  - Thick ore horizons, multiple ore horizons,
  - Steeply dipping ore horizons, flat dipping ore horizons,
  - Shallow ore horizon, deep ore horizon,
  - Irregular ore blocks,
  - Scattered ore blocks.

4.1.2. CONDITIONS REQUIRED FOR MINING METHODS

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

- Describe, explain and discuss the conditions required to successfully apply the following mining methods in terms of:

- Rockmass characteristics:
- Stress regime characteristics:
- Open stoping methods
- Vertical crater retreat
- Sub-level caving methods
- Block caving methods
- Cut and fill methods
- Drift and fill methods
- Bord and pillar methods.

4.2. REGIONAL STABILITY STRATEGIES

4.2.1. REGIONAL PILLARS

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

- Sketch, describe, explain and discuss the functions of regional stability pillars for the variety of massive mining methods at shallow, intermediate and great depth;
- Sketch, describe, explain and discuss the dimensions and layout of regional stability pillars for the variety of massive mining methods at shallow, intermediate and great depth;
- Design regional stability pillars for workings at shallow depths;
- Design regional stability pillars for workings at intermediate depths; and
- Design regional stability pillars for workings at great depths.

4.2.2. CROWN PILLARS

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

- Sketch, describe, explain and discuss the functions of crown pillars for the variety of massive mining methods at shallow, intermediate and great depth;
- Sketch, describe, explain and discuss the dimensions and layout of crown pillars for the variety of massive mining methods at shallow, intermediate and great depth;
- Design crown pillars for massive ore body workings; and
- Describe, explain and discuss the problems associated with designing adequate crown pillars.

4.2.3. BACKFILL AS REGIONAL SUPPORT

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

- Describe, explain and discuss the functions of backfill as regional support for the variety of massive mining methods at shallow, intermediate and great depth;
- Sketch, describe, explain and discuss typical particle size distribution, for:
- Sketch, describe, explain and discuss typical stress-strain curves for:
- Sketch, describe, explain and discuss typical SGs for:

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- Sketch, describe, explain and discuss typical drainage characteristics of:
- Sketch, describe, explain and discuss typical pumping characteristics of:
    - Classified tailings backfill,
    - Full plant tailings,
    - Paste fill,
    - Rock fill,
    - Combinations of fill materials;
- Sketch, describe, explain and discuss the use and effects of cementitious binders with the above types of fill; and
- Specify performance criteria, backfill type and placement method to ensure successful backfilling on a mine.

4.2.4. MINE LAYOUT FOR REGIONAL SUPPORT

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

- Sketch, describe, explain and discuss mining layout strategies to ensure regional stability of massive workings at shallow, intermediate and great depth;
- Sketch, describe, explain and discuss mining layouts to ensure regional stability of workings at shallow, intermediate and great depth;
- Sketch, describe, explain and discuss mining sequences to ensure regional stability of workings at shallow, intermediate and great depth; and
- Design mining strategies, layouts and sequences to ensure the regional stability of massive workings at shallow, intermediate and great depth.

4.3. LOCAL STABILITY STRATEGIES

4.3.1. MINING LAYOUTS AND SEQUENCES

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

- Sketch, describe, explain and discuss mining strategies, layouts and sequences to ensure local stability of massive workings at shallow, intermediate and great depth;
- Sketch, describe, explain and discuss layouts and sequences for the following massive mining methods:
    - Open stoping methods,
    - Vertical crater retreat,
    - Sub-level caving methods,
    - Block caving methods,
    - Cut and fill methods,
    - Drift and fill methods,
    - Bord and pillar methods;
- Evaluate geotechnical data to select appropriate massive mining methods based on stability, dilution and extraction considerations;
- Design appropriate massive mining layouts using rockmass classification techniques;

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- Design the layout of production tunnels, drawpoints and drilling tunnels for open stoping and sub-level caving operations for given geological, geotechnical and stress conditions;
- Design the layout of production tunnels and the undercuts for block caving operations for given geological, geotechnical and stress conditions; and
- Design production excavation sequences for the various massive mining methods for given geological, geotechnical and stress conditions.

4.3.2. UNDERCUT LAYOUT AND DESIGN

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

- Determine and design appropriate massive mining undercut dimensions and layouts using rockmass classification techniques.

4.4. SERVICE EXCAVATION LAYOUTS

4.4.1. TUNNEL LAYOUTS

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

- Describe, explain and discuss the rock engineering criteria used to determine the optimum size, shape and orientation of service tunnels;
- Describe, explain and discuss the rock engineering criteria used to determine the optimum layout, positioning, orientation and spacing of service tunnels relative to:
  - Stopes, pillars,
  - Geological features,
  - Other excavations,
  - Apply these criteria to design service tunnel layouts.

4.4.2. DRAWPOINT AND LOADING LAYOUTS

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

- Describe, explain and discuss the rock engineering criteria used to determine the optimum layout, positioning, orientation and spacing of drawpoints and loading bays relative to:
  - Stopes, pillars,
  - Geological features,
  - Other excavations;
- Apply these criteria to design drawpoint, loading bay and loading tunnel layouts; and
- Describe, explain and discuss the practical aspects of drawpoint and loading bay layouts and sizes with respect to aspect such as machine size, haul distance, etc.

4.4.3. LARGE CHAMBER LAYOUTS

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

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- Describe, explain and discuss the rock engineering criteria used to determine the optimum size, shape and orientation of large service chambers;
- Describe, explain and discuss the rock engineering criteria used to determine the optimum excavation methodology and sequence for large service chambers;
- Describe, explain and discuss the rock engineering criteria used to determine the optimum support methodology and sequence for large service chambers;
- Describe, explain and discuss the rock engineering criteria used to determine the optimum layout, positioning, orientation and spacing of large service chambers relative to:
  - Stopes, pillars,
  - Geological features,
  - Other excavations; and
- Apply these criteria to design large service chambers layouts.

4.4.4. ROCK PASS LAYOUTS

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

- Describe, explain and discuss the causes of instability in rock passes;
- Describe, explain and discuss the effects of instability in rock passes;
- Describe, explain and discuss the options available to stabilise rock passes;
- Describe, explain and discuss the options available to support rock passes;
- Determine appropriate rock pass layout, orientation and support measures using rockmass and stress characterisation techniques;
- Assess the suitability of rock pass layout, orientation and support measures for given sets of rockmass and stress conditions;
- Describe, explain and discuss the rock engineering criteria used to determine the optimum size, shape and orientation of rock passes;
- Describe, explain and discuss the rock engineering criteria used to determine the optimum layout, positioning and spacing of rock passes relative to:
  - Stopes, pillars,
  - Geological features,
  - Each other;
- Apply these criteria to design rock passes; and
- Describe, explain and discuss the rock engineering criteria used to rehabilitate damaged rock passes.

4.5. SHAFT LAYOUTS

4.5.1. SHAFT PROTECTION PILLARS

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

- Sketch, describe, explain and discuss the rock engineering criteria

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used to design incline shaft protection pillars at shallow, intermediate and great depth;

- Sketch, describe, explain and discuss the rock engineering criteria used to design vertical shaft protection pillars at shallow, intermediate and great depth;
- Calculate stress, strain and tilt in vertical shafts using equations for circular shaft pillars in an elastic medium;
- Apply the results from the above calculations to design appropriate shaft pillar dimensions;
- Design shaft layouts for given depth, geology and rockmass conditions; and
- Explain the limitations that concrete linings and steelwork place on allowable movements in shafts.

4.5.2. LAYOUT DESIGN CRITERIA

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

- Describe, explain and discuss the derivation and application of layout and support design criteria for massive mining operations at shallow, intermediate and great depth; and
- Describe, explain, discuss and apply layout design criteria applicable to all of the massive mining situations set out above.

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4. MINING LAYOUT STRATEGIES
4.1. MASSIVE MINING METHODS
4.1.1. CHOICE OF MASSIVE MINING METHOD

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

LEARNING OUTCOME 4.1.1.1

SKETCH, DESCRIBE, EXPLAIN AND DISCUSS THE FUNDAMENTAL MINING AND ROCK ENGINEERING PRINCIPLES ASSOCIATED WITH THE FOLLOWING MASSIVE MINING METHODS :

- Open stoping methods
- Vertical crater retreat
- Sub-level caving methods
- Block caving methods
- Cut and fill
- Drift and fill
- Room and pillar

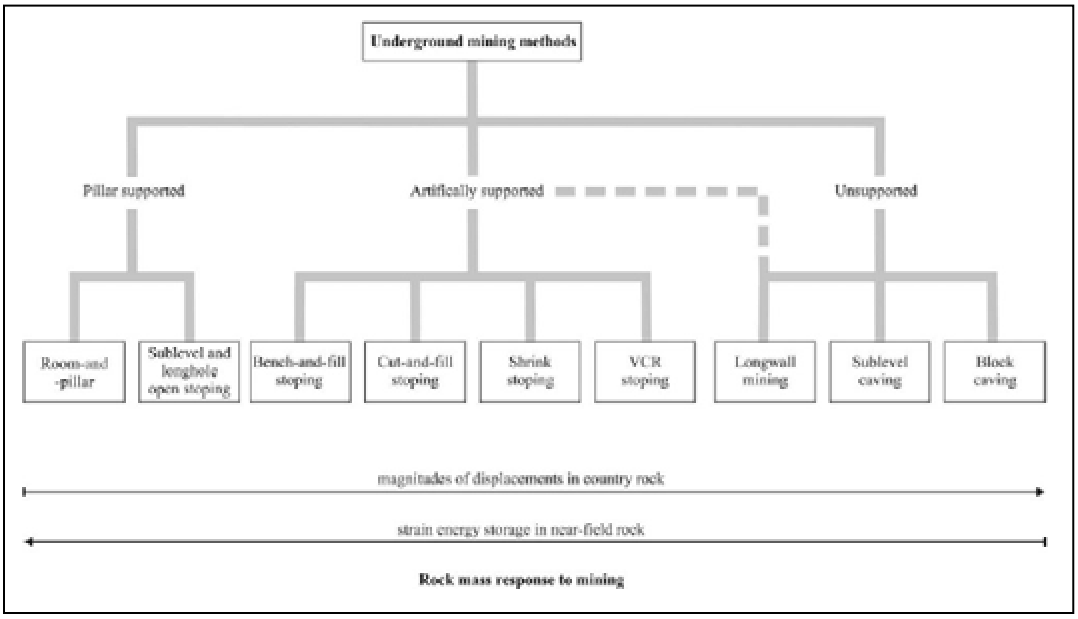

Figure 1: Underground mining methods (Brady & Brown, 2006)

The diagram in Figure 1 relates the most common underground methods with each other and suggests the first criterion for the indication of appropriate mining methods to specific envinronments, that of the response of the rockmass to the mining method.

- Naturally supported – pillar;
- Artifically supported – backfill;
- Unsupported – caving and
- Rockmass behaviour:
  - Rockmass displacement: Caving method allows substantial movement, bord-and-pillar prevents rockmass movement;
  - Strain energy storage levels: Substantial energy is realeased during caving (even seismically) with all energy stored in

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pillars and the overburden in bord-and-pillar layouts.

- Ore body properties that influence the mining method include:
  - Geometric configuration
  - Disposition and orientation
  - Size
  - Geomechanical setting
  - Grade and distribution of grade
  - Engineering environment

REFERENCE

- Page 352, Brady and Brown, Rock mechanics for underground mining, 2006

Open stoping methods

- Mining:
  - Sub-level openstoping: Stoping from a number of sub-levels leaving the stope unfilled and supported by pillars to limit spans;
  - Footwall development is executed to access and mine the ore body and consists typically of decline, level drive/transport drift, loading cross cut/access drift, drill drifts, slot raises and return and water handling tunnels;
  - Slot raise is expanded across the ore body;
  - Ring or parallel drilled blast holes and brken ore report to the drawpoints;
  - Walls are mostly left unsupported;
  - Long hole/big hole stopes use larger diameter holes drilled from a smaller number of drill drifts;
  - Blast hole penetration of walls lead to dilution;
  - Pillar recovery is possible using backfill;

- Rock engineering:
  - Applied in massive or steeply dipping ore bodies;
  - Free standing ability of the walls and ore body is important as the open areas remain mostly unsupported, requiring a good quality material;
  - Pillars control spans and therefore control wall stability by preventing displacement of walls into the void;
  - High horizontal stress perpendicular to ore body induced high stress levels in rib and sill pillars and substantial tensile stresses in backs, leading to potential caving of de-stressed materials;
  - High abutment stress can damage the loading level;
  - High stresses in the back may assist caving (through spalling) and fragmentation;

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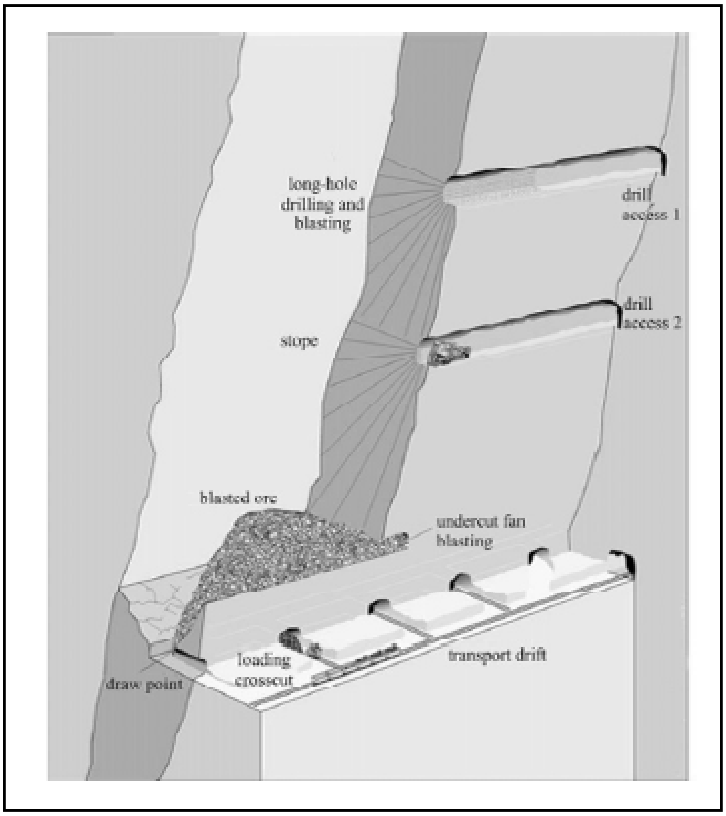

Figure 2: Open stoping with sub-levels (Brady & Brown, 2006)

NOTE

Sub-level open stoping layouts and long-hole open stoping methods vary in layout. For example:

- *Longhole open stopes:*
  - *load ore on the bottom ore drive only, gaining access to the drawpoints from a footwall drive;*
  - *driling is executed from drill drives placed within the orebody;*
- *Sub-level open stopes:*
  - *have sub-levels from which drilling (ore drives) occurs;*
  - *loading, occoationally occurs in the ore drives but can also be done from drawpoints developed from footwall dives on each of the sub-levels.*

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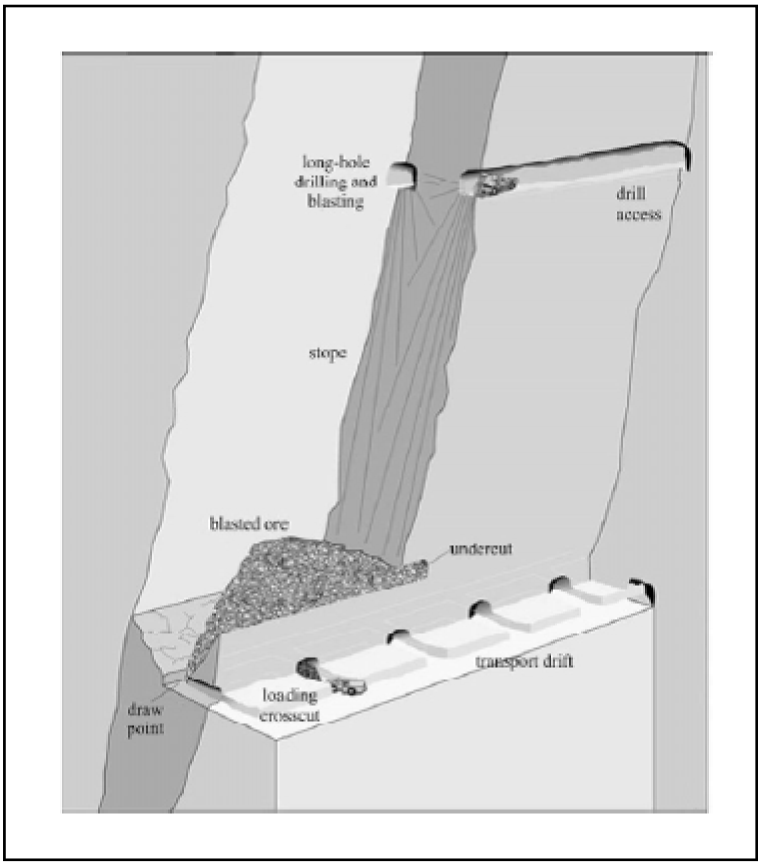

Figure 3: Long hole open stoping (Brady & Brown, 2006)

Design criteria:

- Sub-level stope with pillar extraction:
  - Size: Fairly wide
  - Shape: Tabular and regular
  - Dip: Thick or steep vertical
  - Structures: Influence sequence and spans
  - Stope spans: Based on hangingwall competency, usually large
  - Pillar size: Based on ore body competency, self supporting and dilution control
  - Cavability: Caving after pillar extraction
  - Stress: Depends on depth and caving – if hanging up, stress levels could increase
  - Drift size: To allow drilling and loading
  - Drift spacing: Depend on drill pattern
  - Drawpoint: Primary at angle of repose, pillar recovery under choked conditions
  - Draw control: Not during primary recivery but is important during pillar recovery
  - Dilution: Limited durign primary, crucial during pillar recovery
  - Selectivity: Unoay can be left against haningwall or footwall or in pillars.

- Sub-level stoping with pillars:
  - Size: Not important, will determine mechanisation

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- Shape: Tabular
- Dip: Steep at > 50°, else footwall must be blasted to allow ore flow
- Structures: Affect pillar and back stability
- Stope spans: Can be large, control dilution, use cable bolts
- Competency: Competent haningwall, footwall and orebody
- Selectivity: Limited, unpay in pillars

Vertical crater retreat

- Mining:
  - Utilises similar layout to shrinkage and is non-entry where long holes are drilled between levels;
  - The ore is blasted in an upwards sequence by a series of short, concentrated charges;
  - After each blast, just enough ore is drawn from the stope to create a suitable void for the bulking of the next blast material;
  - Steep dip ore bodies that wil allow ore flow;
  - Very narrow ore bodies are not feasible;
  - All ore is removed only after the whole stope has been blasted;

- Rock engineering:
  - Ore body rock must be competent and resistant to crushing and degradation during the long-term draw process;
  - Usually applied when host rock is not competent enough to remain free standing, but competent enough not to collapse into the stope during drawing;

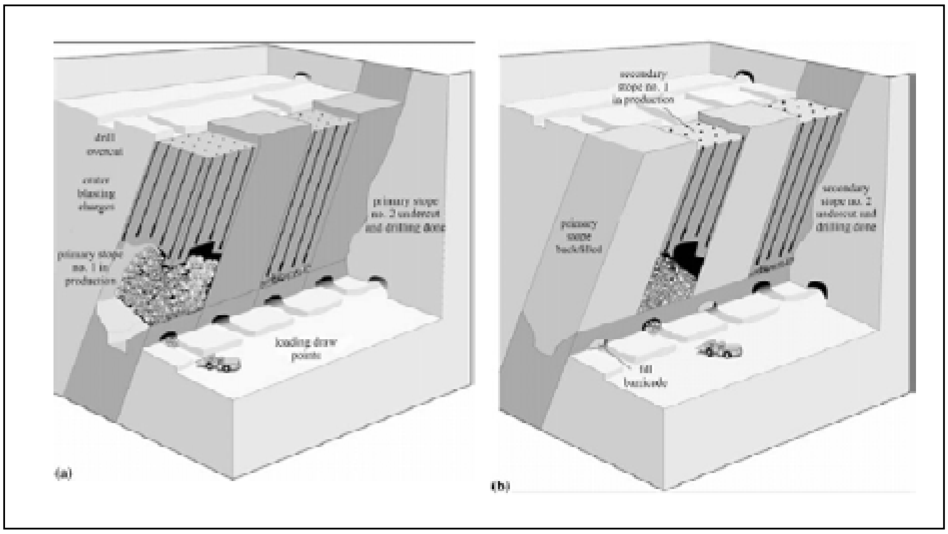

Figure 4: Vertical crater retreat (Brady & Brown, 2006)

Sub-level caving methods

- Mining:
  - Mining progresses downwards in the ore body, allowing the waste material to cave;
  - Blasting compacts the waste and assists in creating a volume for the bulking of the blasted ore

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- Ore is drawn from drawpoints until the waste is encountered, so close control of the draw is critical;
- Headings are both drill drifts and access/transport tunnels.

- Rock engineering:
  - Steep dipping ore body;
  - Competent ore and weaker/incompetent waste material to cave;
  - Grade must be sufficient to allow some dilution;
  - Surface impacts due to the overburden caving must be evaluated.

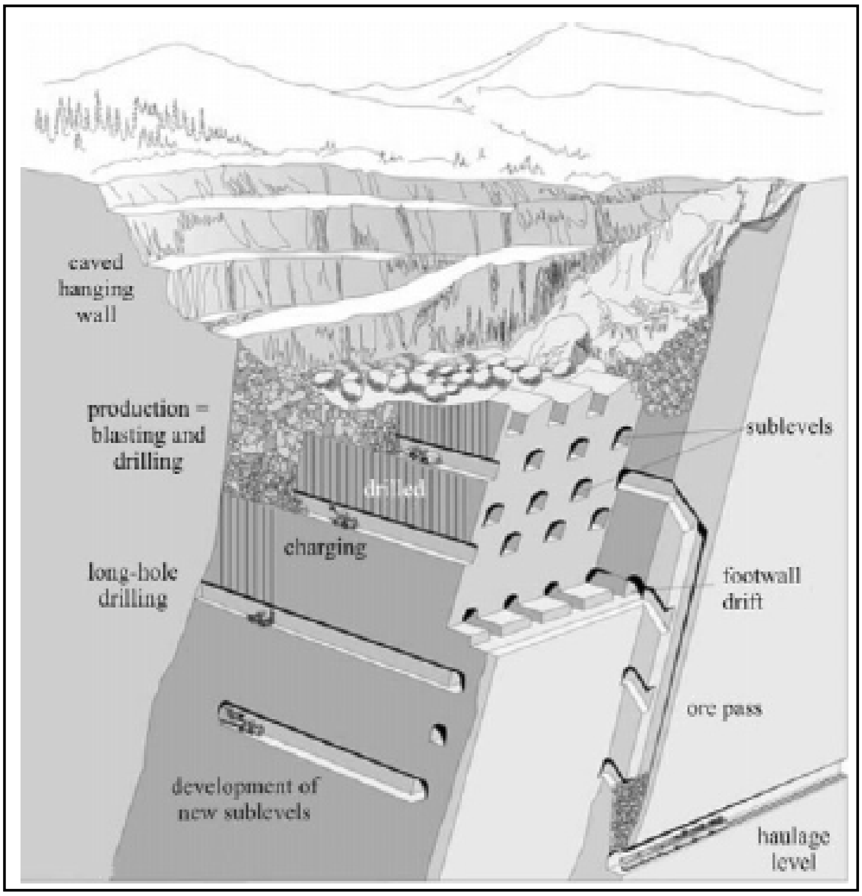

Figure 5: Sub-level caving (Brady & Brown, 2006)

Design criteria:

- Sub-level caving:
  - Size: Large enough to allow sublevels
  - Shape: Massive and / or Tabular and / or steep, uniform shape allows optimum extraction and limits dilution
  - Dip: When flat, orebody must be thick enough to allow various sub-levels
  - Structures: Affect face shape and lead between levels
  - Cavability: Hangingwall must cave
  - Competency: Good ore competency, footwall good for access, haningwall good to limit fragmentation
  - Stress: None if mining directin is haningwall to footwall, footwall to hangingwall mining direction could incraese stress if hang-ups occur

- Drift size: Limited by support
- Draw control: good control required
- Dilution: High dilution result, sensitive to fragmentiation
- Selectivity: Limited, selective loading.

Block caving methods

- Mining:
  - Access to mining blocks are usually via a vertical shaft and/or ramp situated within the waste host rock;
  - Ring drives are developed on both the loading and undercut levels, around the target ore body, situated within the competent waste host rock;
  - On the loading level, tunnels / drives with drawpoints are created throughout the target mining block or orebody;
  - On the undercut level, tunnels are developed though the target block or ore body;
  - On the undercut level the pillars between tunnels are dtilled and blasted whilst also createing a cone for future ore flow;
  - On the loading level, drawcones are drileld and blasted to allow ore to flow to the drawpoints for loading;
  - After the creation of the undercut and loading level, it is expected that the orebody will fail within the further need for blasting, loading the caved material on the loading level;
  - Once a block has been exctraced, the next block is extracted via loading and undercut levels created at a deeper elevation.

- Rock engineering:
  - High abutment stress can damage the loading level;
  - Incrrect caving sequences could induce execessive stress levels on the loading level excavations;
  - High induced stress levels and normaly weaker ore body material warrants the use of extensive support ssystems to maintain loading level stability;
  - Host rock must be competent enough to ensure access routes to loading and undercut levels are stable;
  - Sufficient geotechnical data is required to ensure cavability of the ore body, placement of acecss tnnels and design of support systems within and outside ore body;
  - Water handling to remove ecessive water is critical in some ore bodies and must be addressed to assist tunnel stability;
  - High stresses in the back may assist caving (through spalling) and fragmentation.

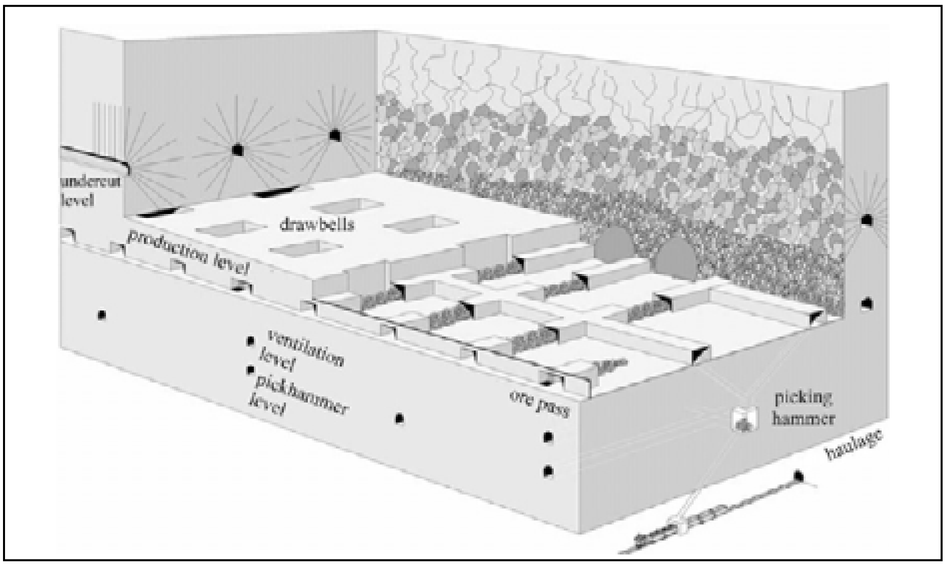

Figure 6: Block caving (Brady & Brown, 2006)

Design criteria:

- Block caving:
  - Size: Large enough to meet production requirements and to ensure ore b ody and hangingwall cave
  - Shape: Not important, uniform shape reduces dilution
  - Dip: Sufficient draw height with flat dip
  - Structures: Can cause collapses in ore, impact on cave zone
  - Competency: Affcets cavability, fragmentation and support design
  - Stress: Affects cavability and abutment stresses
  - Drawpoint size: Depend on competency and support
  - Drawpoint spacing: Draw zone interaction determines
  - Draw control: Essential
  - Dilution: Influenced by draw interaction, fragmentation
  - Selectivity: None

Cut and fill

- Mining:
  - Access is estalished via a footwall ramp, placed a sufficient distance from the orebody to ensure its long term stability;
  - Access to each lift is gained from the ramp and commenc-es with the development of a declined tunnel to the bottom of the cut to be taken between levels. Once the bottom cut has been extracetd and backfilled, the roof of the dcecline is drilled and blasted, compacting the blasted material on the footwall and creating the access to the next lift elevation. This process of "dropping" the roof in the access is repeated until the complete cut is mined out;
  - Mining is mostly in an upward direction to allow work on top of fill, but can also be done in a downward direction; howev-er, working underneath the backfill places restrictions on the backfill types and strengths required to ensure fill stability;
  - Typically, the mining is done by a cut (usually 3-4m) from the ore body above the previously filled cut, making sure safe and

back

support installation is followed by the loading and removal of the ore and filling of the created void, allowing work to progress on top of the fill to take the next cut.

- Personnel entry in the stopes requires critical issues such as making safe and support installation;
- Can be used in veins, massive and inclined tabular ore bodies;
- Dilution is possible and must be catered for;

- Rock engineering:
  - When ore body and host rock are weak/incompetent, this method allows for the reducton of mining spans to maintain stability;
  - Ore body dips 35-90°;
  - Deep and shallow deposits;
  - High grade required due to the high labour costs;
  - Unable to manage partial extraction within a selected block, but can follow lenses, curved rebodies due to flexible development of the ore body.

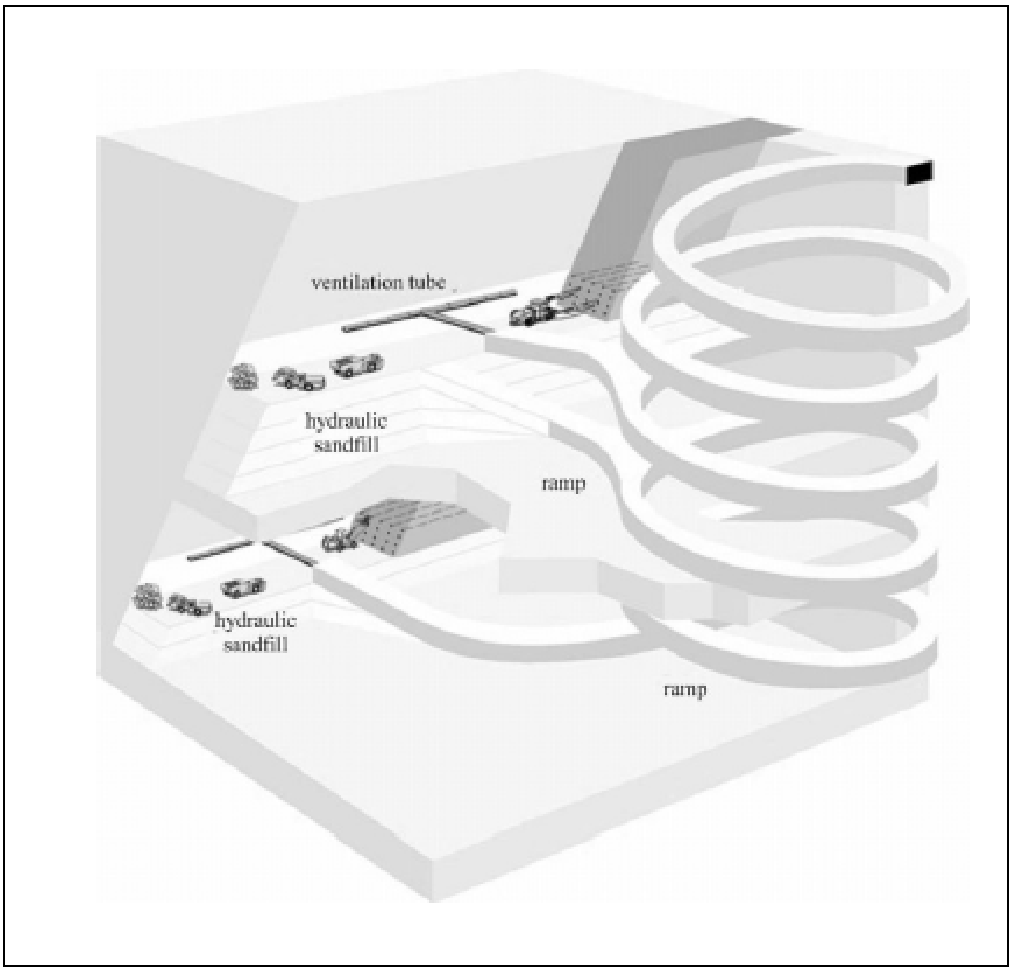

Figure 7: Cut and fill (Brady & Brown, 2006)

A variation on the standard cut and fill method include "post pillar cut and fill", where pillars are left in-situ to assist the stability of the mining excavation where the orebody roof mayerial is not strong / competent enough to remain stable over large spans. Backfill is placed around these pillars once the broken ore of each lift is removed. The backfill confines pillars as sucessive lifts are taken and maintains stable pillars until the full cut between levels has been taken.

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Design criteria:

- Post pillar cut and fill:
  - Size: Not important, usually not used in narrwo orebodies
  - Shape: Not important
  - Dip: Not important
  - Structures: Affect stability of pillars and hangingwall
  - Stope spans: Based on pillar size, spacing and support capability
  - Pillar size: Regular spacings and based on rock mass strength and local support requirement, not recoverablem, ensure stability
  - Selectivity: Very high selectivity and control of dilution

- Overhand cut and fill:
  - Size: Narrow to medium sized
  - Shape: Tabular
  - Dip: Steep
  - Structures: Affect stability, can be left in pillars
  - Stope spans: Depend on hangingwall competency
  - Pillar size: Size depend on support requirement, spacing depend on local or regional requirement
  - Selectivity: High

- Underhand cut and fill:
  - Size: Narrow deposits
  - Shape: Irregular hangingwall concern if material is weak
  - Dip: Steep but can be adapted for flatter dips
  - Stope spans: Based on fill material
  - Selectivity: High

Drift and fill

- Mining:
  - Mining is conducted in drifts, typically 4 to 5m wide and high;
  - The layout may be similar to that used for cut-and-fill with the exception that the 'cut' is taken in drifts, limiting the open spans;
  - This method is very suitable to shallow dipping orebodies of limited thickness where drifts are extraceted on strike and the ore body is extracted in a single inclined plane;
  - Mining process allows extraction of a drift, filling of the drift and mining of the adjacent drift, either exposing the filled drift or leaving a pillar between for later extraction (primary and secondary extractions);

- Rock engineering:
  - Since the extraction of the next drift exposes the backfill placed in the previsouly mined and filled drift in the new drift sidewall, fill with at least self-standing height strengths over the exposed heights is required;

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- Applied in areas where ore body and waste rock are weak/ incompetent and where small spans must be maintained;
- Design of sill pillars and/or backfill material/strength is critical.

NOTE

The diagram in Figure 8 illustrates a combination of drift and fill (extraction in drifts) and cut and fill (final layout). Conventional drift and fill where all drifts are on strike extrcats the ore body in drifts on a single plane.

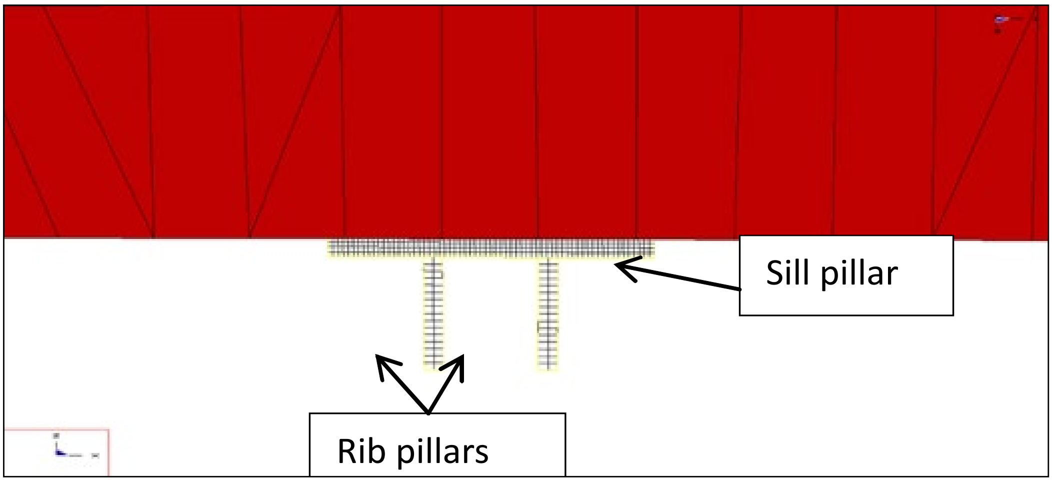

Figure 8: Drift and fill in a dipping orebody

Room and pillar

- Mining:
    - Ore produced from bords/rooms that also form the access and escape routes;
    - Ore is left as pillars in a regular array to simplify planning, mining and the use of mechanised equipment;
    - Thick ore bodies can be mined in multiple slices;
    - More appropriate for flat dip ore bodies;
    - Usually applied to shallow areas as pillars become too large, but large pillars can be extracted on retreat;
    - Close control of grade is possible, making selective mining and application to a variable grade ore body possible;
- Rock engineering:
    - Pillars cntrol on-reef displacements and therefore the overburden;
    - Spans are limited to stable widths depending on the quality of the exposed hangingwall material;
    - Ore body must be strong/competent enough to allow stable pillars;
    - Competent hangingwall material required to allow increased bord widths to achieve good extraction ratios.

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NOTE

Pillars must remain stable after benching and thus require that final dimensions must be utilised to indicate final state of stability (factor of safety). However, the possibility of pillars confinement using backfill should be recognised and considered.

Footwall, as well as hangingwall benching is possible.

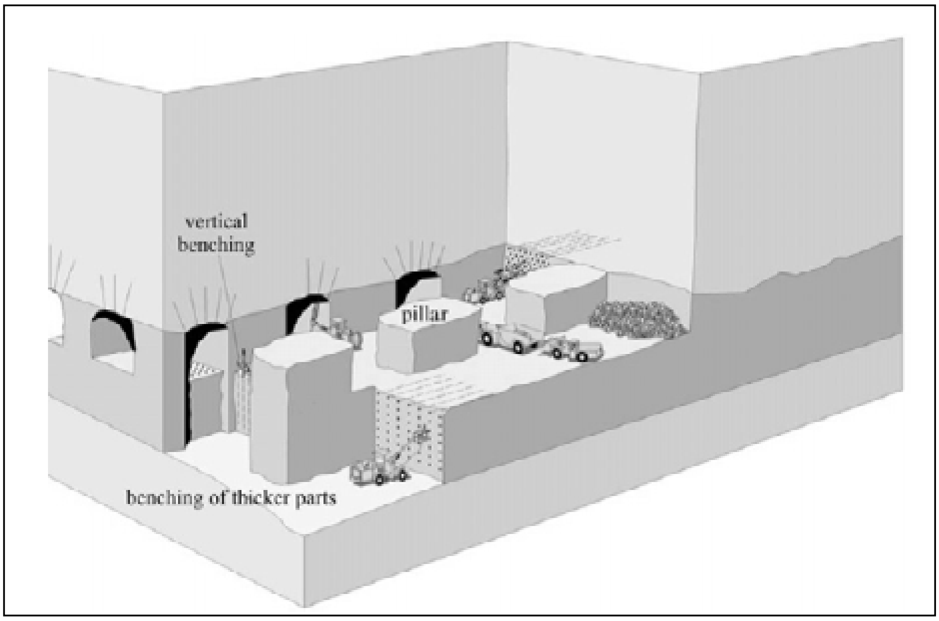

Figure 9: Bord and pillar (Brady & Brown, 2006)

CONNECTION 1

Paper 3.1, Outcome 4.1.1
Paper 3.2, Outcome 3.1, 4.1.1

Design criteria:

- Bord and pillar:
  - Size: Thick orebody can be accomodated
  - Shape: Tabular
  - Dip: Flat < 30°
  - Structures: Affects pillar and hangingwall strength
  - Stope spans: Room spans to ensure that roof / back / hangwall remain styable with support if required
  - Pillar size: Depends on mining depth, shallow - support overburden, deep – support tensile zone
  - Selectivity: Fairly selective

Summary of methods (Courtesy of Mining Engineering Handbook)

NOTE

IMPORTANT

It is suggested that the documents provided as links are all scrutinised to ascertain better understanding of the listed methods.

REFERENCE

- All mining methods and equipment
- Underground mining methods Hamrin
- Mining method selection Turner
- Mining engineering handbook Chapter 21.2
- Mining engineering handbook Chapter 23.4 Selection of mining methods

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- Guides to underground mining Atlas Copco
- Decisions based on analytical hierarchical process Kluge
- AHP process of selecting underground mining method Gupta
- A New method for mining method selection Shariar

LEARNING OUTCOME 4.1.1.2

CONNECTION 2

Refer to "LEARNING OUTCOME 4.1.1.1" on page 223

SKETCH, DESCRIBE, EXPLAIN AND DISCUSS THE LAYOUT OF THE ABOVE MINING METHODS

LEARNING OUTCOME 4.1.1.3

CONNECTION 3

Refer to "LEARNING OUTCOME 4.1.1.1" on page 223

SKETCH, DESCRIBE, EXPLAIN AND DISCUSS THE LOCATION AND STABILITY OF ACCESS TUNNELS AND SERVICE EXCAVATIONS

LEARNING OUTCOME 4.1.1.4

SKETCH, DESCRIBE, EXPLAIN AND DISCUSS HOW THE FOLLOWING ORE BODY GEOMETRIES AND/OR COMBINATIONS OF ORE BODY GEOMETRIES MAY BE MINED:

- Thick ore horizons,
- Multiple ore horizons
- Steeply dipping ore horizons,
- Flat dipping ore horizons
- Shallow ore horizon,
- Deep ore horizon,
- Irregular ore blocks,
- Scattered ore blocks.

Apply the knowledge gained in Outcomes 4.1.1.1 to 4.1.1.3 for each of the conditions listed above, following the process suggested below:

- List the rock engineering/geotechnical characteristics of the condition provided;
- By exclusion, select the methods that will not be feasible due to the condition sketched and remove them from the list;
- Evaluate the remaining methods at the hand of the conditions and select the most appropriate method.

EXAMPLE

Flat dipping, thin, tabular ore body:

The following methods will not be suitable:

- Sub-level open stoping;
- Block caving;

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- VCR;
- Sub-level caving;
- Cut-and-fill (not standard application of method, but with some changes, could be possible).

The following methods could be suitable:

1. Drift and fill;
    - Applicable in layouts where pillars are also used to protect accesses;
    - Limits spans in drifts to maintain stability;
    - Allows high extraction ratios;
    - In shallow areas, subsidence is possible;
    - Partially flexible in lower grade areas;
    - Can be used if ore is weak and cannot remain stable in pillars.

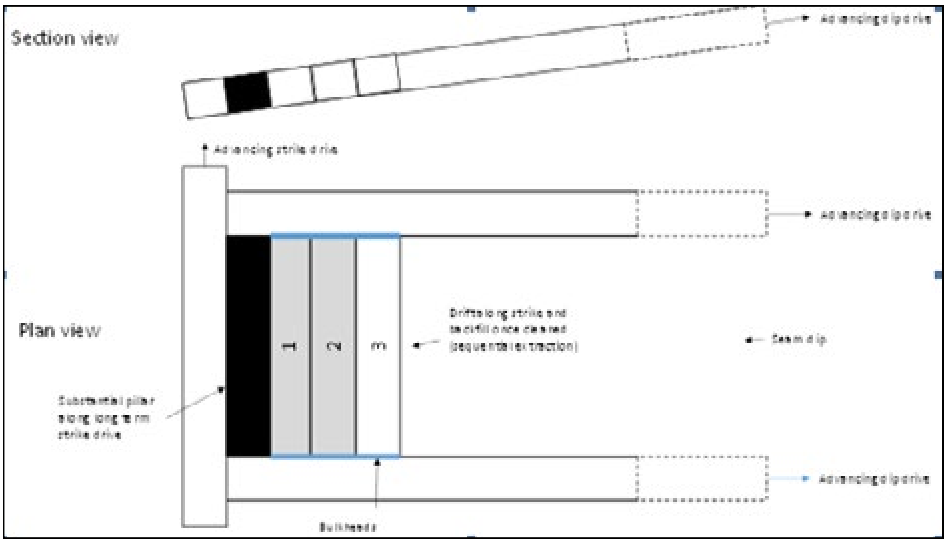

Figure 10: Possible drift and fill layout in flat dipping thin ore body

2. Bord-and-pillar:
    - Limiting spans between pillars (bord width) to maintain stability;
    - Prevents subsidence in shallow areas,
    - Extraction ratio low and even lower at greater depths;
    - Flexible in variable grade areas;
    - Requires competent ore to allow pillars to remain stable;

3. Draft additional information that will assist with final selection:
    - i.e. shallow ore body, variable grade, competent ore and
    - apply to the lists created above for the methods that are possibly suitable
    - Select bord-and-pillar.

NOTE

Deciding on the most appropriate mining method for a given ore body/ situation is not a simple exercise and requires significantly more data than that provided above. Make sure, when doing this in practice, that all possible data is gathered and evaluated.

Instead of just providing possible solutions to the situations given above, it is suggested that methods to evaluate conditions to allow for the

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selection of the most appropriate mining methods is more important and that the learner must apply these methods to the conditions provided to be competent in this outcome.

It is suggested that the materials linked below are all read and applied.

REFERENCE

- Page 368, Brady and Brown, Rock mechanics for underground mining, 2006

INTERESTING INFO

Several documented methods to assist with the selection of an appropriate mining method exist (detail can be found in the links provided below):
- Boshkov and Wright;
- Hartman;
- Morrosin;
- Laubscher
- Nicholas;
- Analytical hierarchy process.
- Application of mining methods (Courtesy of Mining Engineering Handbook)

Table 2:Boshkov and Wright method (Boshkov and Wright, 1973)

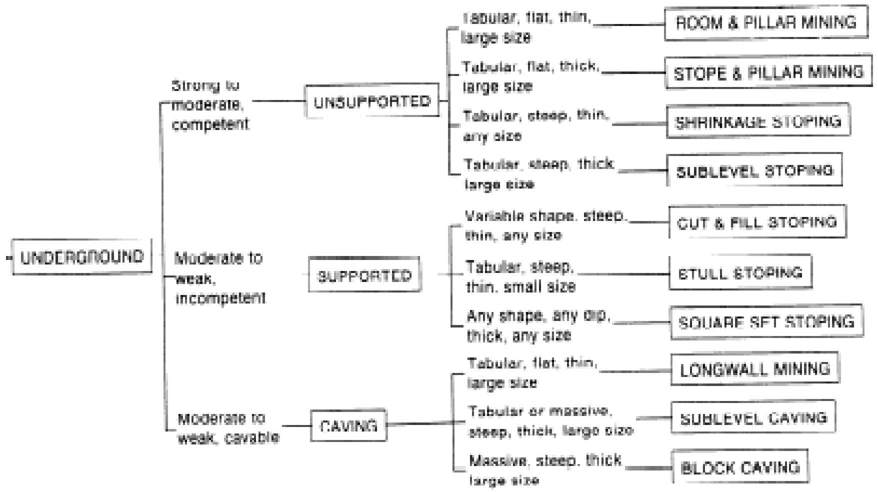

Fig. 23.4.1. Hartman's chart for selection of a mining method (Hartman, 1987).

Figure 11: Hartman's method (Hartman, 1987)

LINK

- Page 2096, Mining engineering handbook Chapter 23.4 Selection of mining methods
- Mining method selection Turner
- Mining engineering handbook Chapter 23.4 Selection of mining

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methods

INTERESTING INFO

The analytical hierarchy process (AHP) is a process whereby a number of parameters can be assessed comparatively, i.e. where the parameters are finally listed in order of their importance to or relative impact on the issue being investigated, whether this is the role certain issues play in a final outcome (weighting, as used in stope risk rating systems) or whether it is comparing a number of mining methods to a given condition. The process entails the following:

- Description of the 'issue' to be investigated;
- Listing of all parameters that impact on the issue investigated;
- Inclusion of the parameters into a set matrix format;
- Rating of each parameter's 'importance' in affecting the 'issue' against each other parameter, using the rating system suggested as shown below, completing the ratings in the set matrix format;
- Calculation of the competitive 'weighting' of the parameters applying the process with the data in the created matrix;
- Listing of the parameters in order of importance based on the 'comparative weighting'.

Table 3: Rating system for AHP process

Rating system			
Equally Important	1		Equally Important
Moderately Important	3	1/3	Moderately Important
Strongly more Important	5	1/5	Strongly more Important
Very strongly more Important	7	1/7	Very strongly more Important
Extremely strongly more Important	9	1/9	Extremely strongly more Important

REFERENCE

- Decisions based on analytical hierarchical process Kluge
- AHP process of selecting underground mining method Gupta
- A New method for mining method selection Shariar
- AHP calc weighting and consistency 10 criteria

4.1.2. CONDITIONS REQUIRED FOR MINING METHODS

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

LEARNING OUTCOME 4.1.2.1

CONNECTION 4

Refer to "LEARNING OUTCOME 4.1.1.1" on page 223

DESCRIBE, EXPLAIN AND DISCUSS THE CONDITIONS REQUIRED TO SUCCESSFULLY APPLY THE FOLLOWING MINING METHODS IN TERMS OF ROCKMASS CHARACTERISTICS AND STRESS REGIME CHARACTERISTICS:

- Open stoping methods
- Vertical crater retreat
- Sub-level caving methods
- Block caving methods
- Cut and fill methods
- Drift and fill methods
- Bord and pillar methods.

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CONNECTION 5

Refer to
Paper 3.1, Outcome 4.2
Paper 3.2, Outcome 4.2

Page 370, Brady and Brown, Underground mining methods, 2006

4.2. REGIONAL STABILITY STRATEGIES

4.2.1. REGIONAL PILLARS

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

LEARNING OUTCOME 4.2.1.1

CONNECTION 6

Refer to
Paper 3.1, Outcome 4.2.1.1.1

SKETCH, DESCRIBE, EXPLAIN AND DISCUSS THE FUNCTIONS OF REGIONAL STABILITY PILLARS FOR THE VARIETY OF MASSIVE MINING METHODS AT SHALLOW, INTERMEDIATE AND GREAT DEPTH

Function:
Regional stability pillars in a massive mine must limit spans to ensure stability of the stopes and/or the overburden.

NOTE

The function of these pillars implies that they are therefore not used in any of the caving mining methods as the creation of large spans is critical to the caving of the rockmass, while pillars will inhibit the caving.

Application:

1. Bord-and-pillar:

Regional pillars are usually not applied when the pillar design is for solid pillars and the competency of the material in the pillars is such that long-term strength reduction is not expected. If, as in coal mines, the long-term stability of the pillars cannot be ensured, it is practice to leave regional pillars between mining blocks to limit the extent of the overburden collapse should the pillars yield/fail in the long term;

2. Open stopes:

Pillars are left intact to limit stope spans and are called sill (along the strike of the ore body) and rib (along the dip of the ore body) pillars. Although these are not called regional, they perform the function of the regional pillars in that they ensure stability of the stopes and overburden;

3. Backfill-based methods:

Since stopes are backfilled, the use of regional pillars is often only considered if the closure into the mined-out area is expected to affect surface structures. If not, the backfill forms the regional support method that prevents significant regional overburden movement.

LEARNING OUTCOME 4.2.1.2

CONNECTION 7

Refer to
Paper 3.1, Outcome 4.2.1.1.1

SKETCH, DESCRIBE, EXPLAIN AND DISCUSS THE DIMENSIONS AND LAYOUT OF REGIONAL STABILITY PILLARS FOR THE VARIETY OF MASSIVE MINING METHODS AT SHALLOW, INTERMEDIATE AND GREAT DEPTH

Regional pillar dimensions and spacing can be determined in a number of methods, which include the following:

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INTERESTING INFO

An empirical strength/stress ratio was developed on deep level gold mines, which suggests the following:

- Stress fracturing present when the field stress = 33% of UCS;
- Fracturing is intense and requires substantial support when the field stress = 50% of UCS.

1. Empirical:

Pillar loading vs pillar strength:
Apply the appropriate method (e.g. Hedley and Grant for hard rocks) to determine the pillar strength and utilise TAT (taking notice of the dip of the ore body) to determine the pillar loading. Factor of safety (pillar strength/pillar stress) or fracturing potential (empirical rock strength/field stress ratio) provide the estimation of the ability of the pillar to remain stable and ensure regional stability.

NOTE

The impact of this dip on pillar stress levels can be estimated using the following (Hedley and Grant, 1972):

Pillar stress$=(\gamma H\cos^2\alpha + \sigma_h \sin^2\alpha) / (1-R)$

where γ is the unit weight of the overburden (MN/m^3),

H is the mining depth (m),

R is the extraction ratio (e),

α is the dip of the seam from horizontal (°) and

σ_h is the horizontal component of the in situ stress equal to $k.\sigma_v$ where k is the 'k-ratio' and

σ_v is the vertical component of the in situ stress.

A large number of methodologies to determine the pillar strengths are available. The learner should take cognisance of all methods appropriate to the specific environment before applying the methods.

Spacings between regional pillars
Spacings between regional pillars = 0.5 x depth or even 0.25 x depth were developed to prevent collapses between the pillars from affecting surface structures but are limited to flat dipping ore bodies;

LINK

- GAP334 Design of pillar systems

2. Numerical modelling:

Pillar loading vs pillar strength: Using numerical models to determine the pillar loading more accurately in the generally more complex geometrical layouts found in massive mines has found acceptance. The rest of the design assumes some pillar strength to achieve a method of pillar stability estimation and could include:

- Factor of safety;
- Potential for failure using Hoek-Brown failure criterion.

EXAMPLE

If it assumed that the Hedley and Grant pillar strength and loading formulae are appropriate for this environment, then the pillar strengths can be estimated by:

$\sigma_{strength} = kw^{0.5}/h^{0.75}$

while the pillar loading is estimated by

$\sigma_{average}=[\gamma H\cos^2(\alpha)+\sigma H\sin^2(\alpha)]/1-R$

where γ is the material unit weight,

H is the depth ,

α is the dip of the ore body,

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σ_H is the horizontal component of the stress field (k-ratio of 1.0 is assumed) and
R is the extraction ratio.

Using the relationship RMR = 9lnQ + 44 to convert the Q Index values to RMR (Bienawski), values of 60 and 61 were obtained for the ore bodies and were utilised to estimate the rock strength parameter (k) in the Hedley and Grant pillar strength formula. Applying the following general layout dimensions, the approximate pillar widths to ensure that stress-induced failure does not occur (Factor of Safety > 1.5) could be derived as shown in Table 4:

- Ore body widths of 5m, 15m and 30m;
- Open stope dip spans as derived from a 40m vertical span;
- Open stope strike span of 30m.

Table 4: Sill and rib pillar dimensions based on overburden loading

Ore body width	Body 1	Body 2
5m	4.0m	4.0m
10m	6.0m	6.0m
15m	8.0m	7.5m
20m	9.0m	9.0m
25m	10.5m	10.5m
30m	12m	11.5m

EXAMPLE

The purpose of pillars is to:

- Rib pillars: limit mining spans on strike and maintain stable stope walls at least during the period of time during which the stopes on both sides of the pillars are blasted and cleaned;
- Sill pillars: limit mining spans on dip and remain stable at least for the period of time during which the stope is blasted and cleaned (a top-down mining sequence is assumed).

An initial empirical approach for the determination of both the sill and rib pillar dimensions yielded the following results:

- Rib pillars: 10m width up to 15m ore body thickness;
- Sill pillars: At least equal to the ore body width (assuming post pillars are not used to limit spans).

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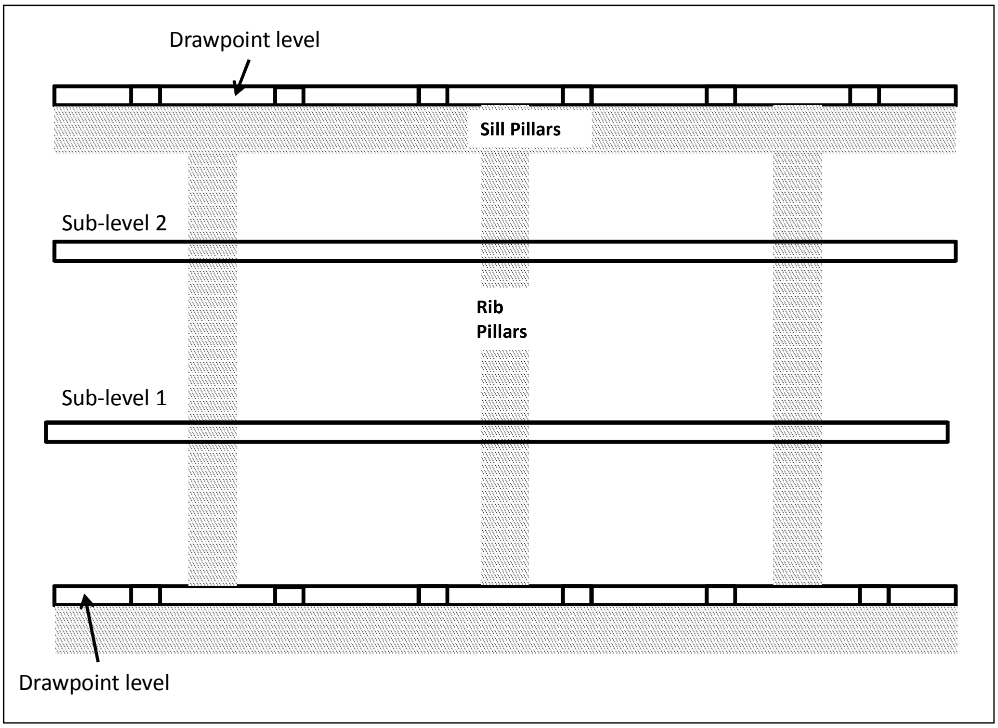


Figure 12: SLOS with pillars layout

Numerical modelling was conducted to evaluate the empirical pillar dimensions and indicate possible risks associated with the current rib and sill pillar dimensions. Map3D, a fully three-dimensional boundary element code, was utilised as it allows the construction of a fully three-dimensional ore body of substantial strike and dip dimensions with variable thickness and dip. A total of 36 models were created to represent ore body dips of 60°, 75° and 90° dips (Table 5), at various mining depths (360m to 560m) and with pillar dimensions of 10m, 15m, 25m and 30m (Table 6.

Table 5: Ore body dip variations simulated

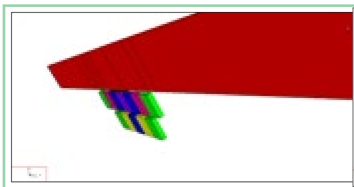
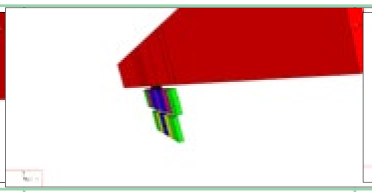
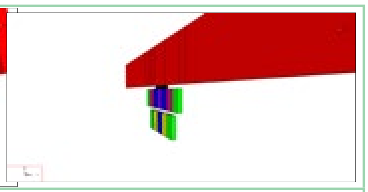
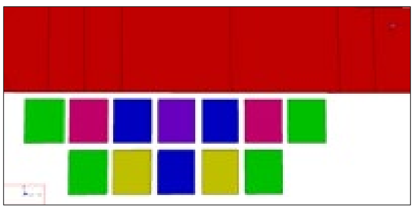
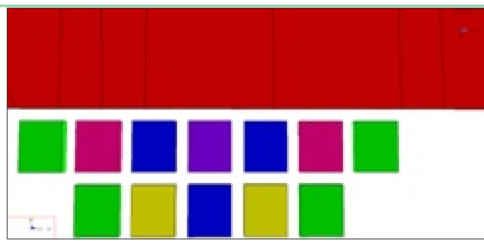
		
60° ore body dip	75° ore body dip	90° ore body dip

Table 6: Pillar thickness variations simulated

	
10m pillars	15m pillars

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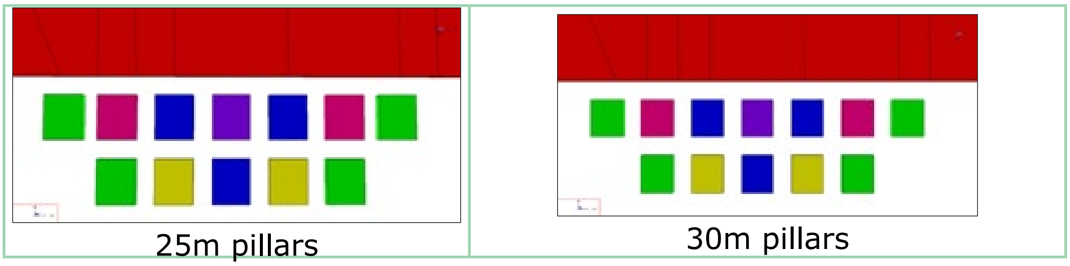


In agreement with the conditions for the change in the mining layout, the longitudinal SLOS stopes constructed are 50m long, 60m high and 15m thick (a critical limitation for the application of this method was the maximum ore body width of 15m) with rib and sill pillars placed around these stopes.

Grids were placed along the sill and rib pillars (Figure 13) adjacent to the mined-out shallower levels to allow the measurement of the impact of long-term mining on the pillar stress levels.

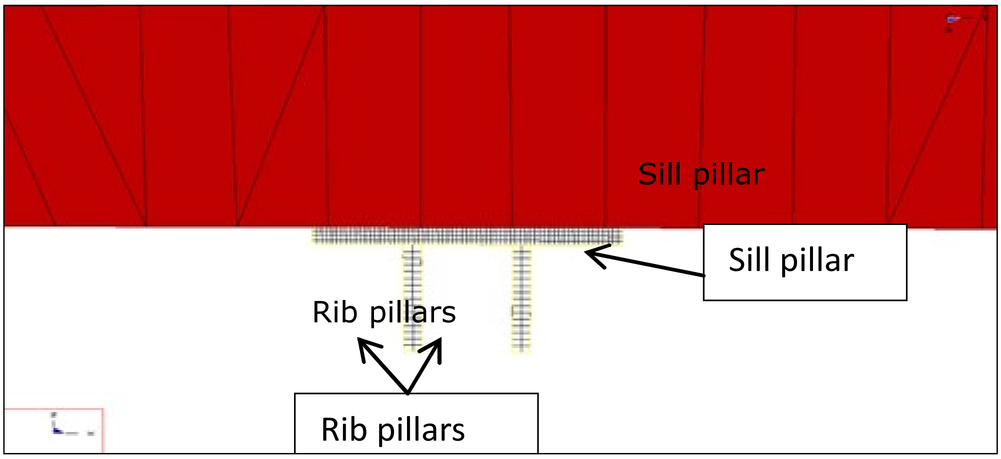
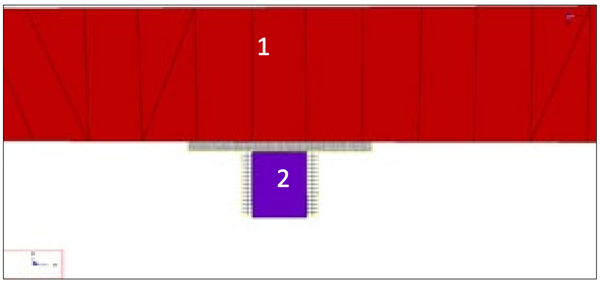
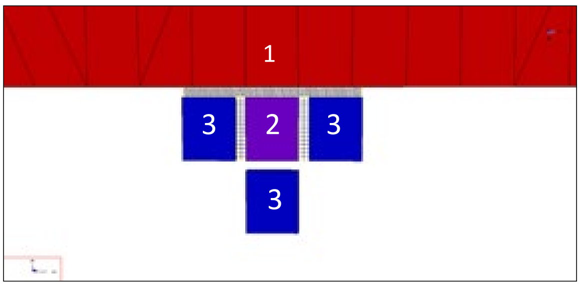


Figure 13: Grids placed along rib and sill pillar positions

Table 7: Mining sequence

	Step 1: Shallow areas mined out. Step 2: 1st stope extracted creating the sill pillar.
	Step 3: Stopes extracted completing the sill pillar of interest and extend downdip.

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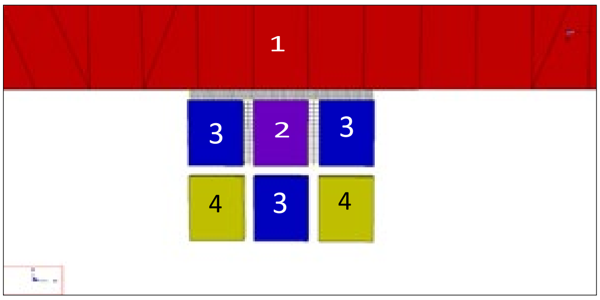
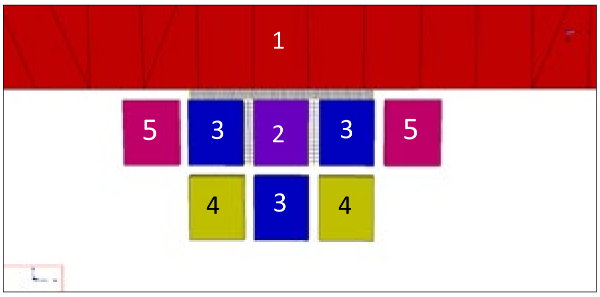
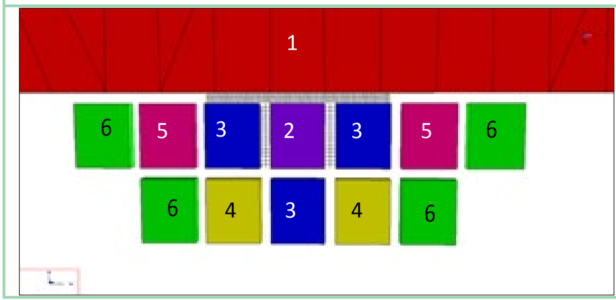


Step 4: Mining continues, completely forming all the pillars of interest.



Step 5: Mining continues on strike.



Step 5: Mining continues on strike.

Input parameters for the assessment included the following:

Table 8: Input parameters

Parameter	Value	Parameter	Value
Young's modulus (host rock)	50GPa	UCS (ore)	151 MPa
Poisson's ratio (host rock)	0.18	Vertical stress variation	-0.028 MPa/m
K-ratio (horizontal to vertical parallel to ore body)	1.5	Horizontal stress variation (parallel to ore body)	-0.042 MPa/m
K-ratio (horizontal to vertical perpendicular to ore body)	1.0	Horizontal stress variation (perpendicular to ore body)	-0.028 MPa/m
Hoek-Brown strength parameters (mi, m)	5.09 , 0.16	Hoek-Brown strength parameters (s,a.)	0.0003 , 0.505

Pillar stress levels were drafted from Map3D into spreadsheets and manipulated into the graphs shown in this section. Results for rib and sill pillars are provided separately.

Rib pillars
It was noted that ore body dip had little impact on average rib pillar stress values (Figure 14) and that all further analyses could be done using one of the ore body dip models. This managed to reduce the number of variables to be catered for in the implementation of the final pillar dimensions.

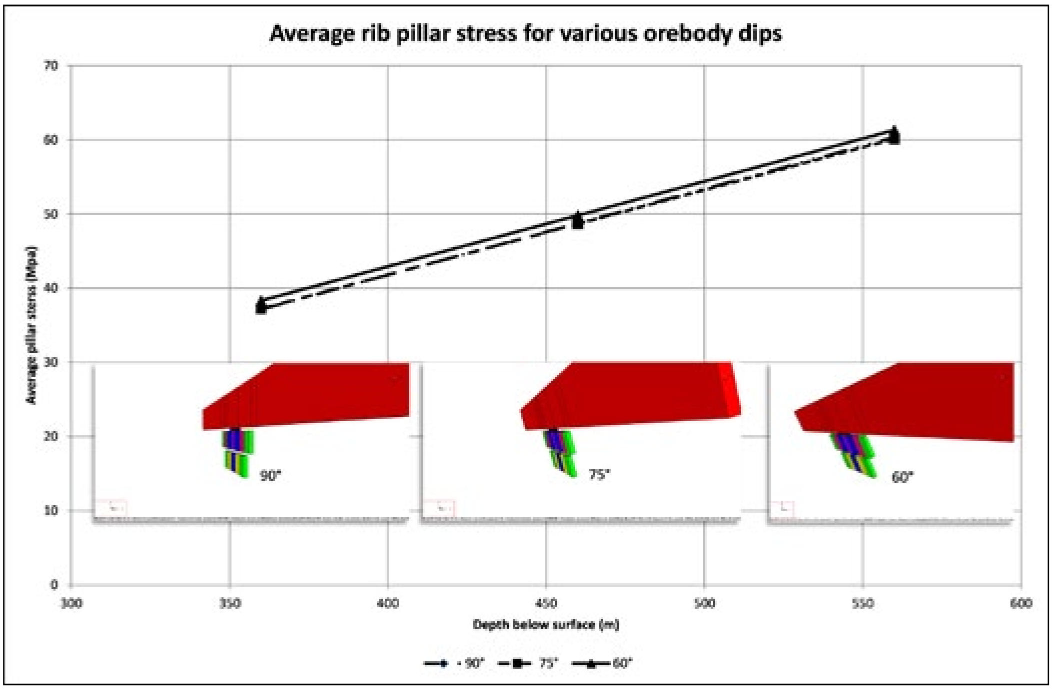


Figure 14: Average rib pillar stress for varying ore body dips

Since the rib pillars are essential to remain stable (i.e. no stress fracturing and scaling) during extraction of both stopes around the rib pillar, the critical stress levels to interrogate are those experienced during mining step 3. Applying an empirical stress-strength ratio of 0.33 (33%), the mining depths at which the potential for stress fracturing throughout each of the different pillar widths could occur were estimated and plotted in Figure 15.

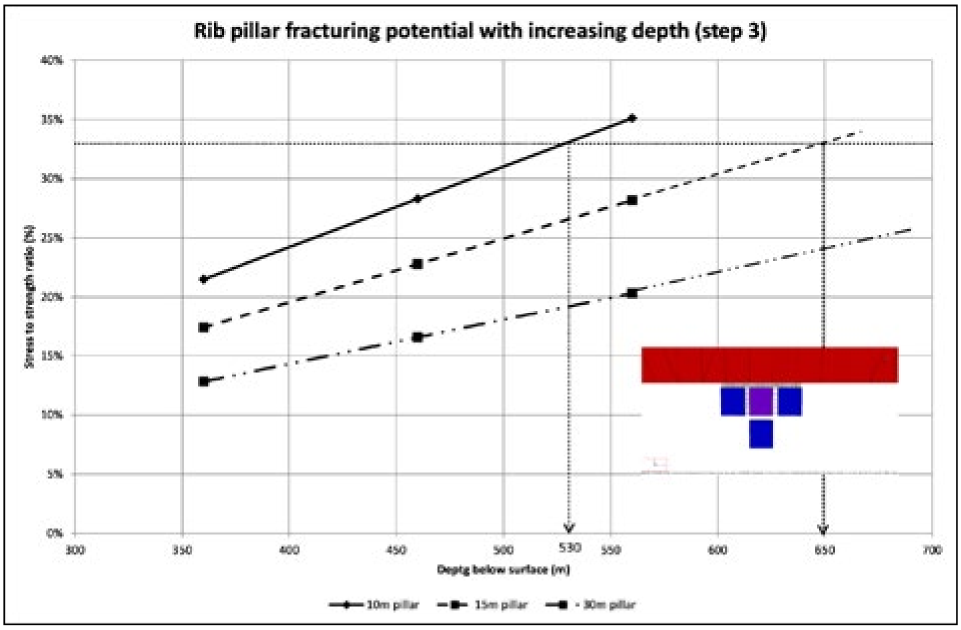


Figure 15: Fracture potential with increasing depth (step 3)

This gives the mining depths to which rib pillars of certain dimensions can be utilised without high levels of risk that stress-induced failures of the rib pillars could occur, resulting in premature scaling of the pillars and possibly the loss of stope wall integrity due to increasing mining spans. The results indicate that:

- 10m rib pillars to a depth of 530m below surface;
- 15m rib pillars to a depth of 650m below surface;
- 20m rib pillars to a depth of 740m below surface.

Sill pillars
The sill pillars were subjected to the same analysis as that explained above for rib pillars and indicated, as shown in Figure 16, that 15m sill pillars, as currently included into the medium-term mining plan, will be of sufficient dimension to prevent stress-induced fracturing to a depth of 525m below surface.

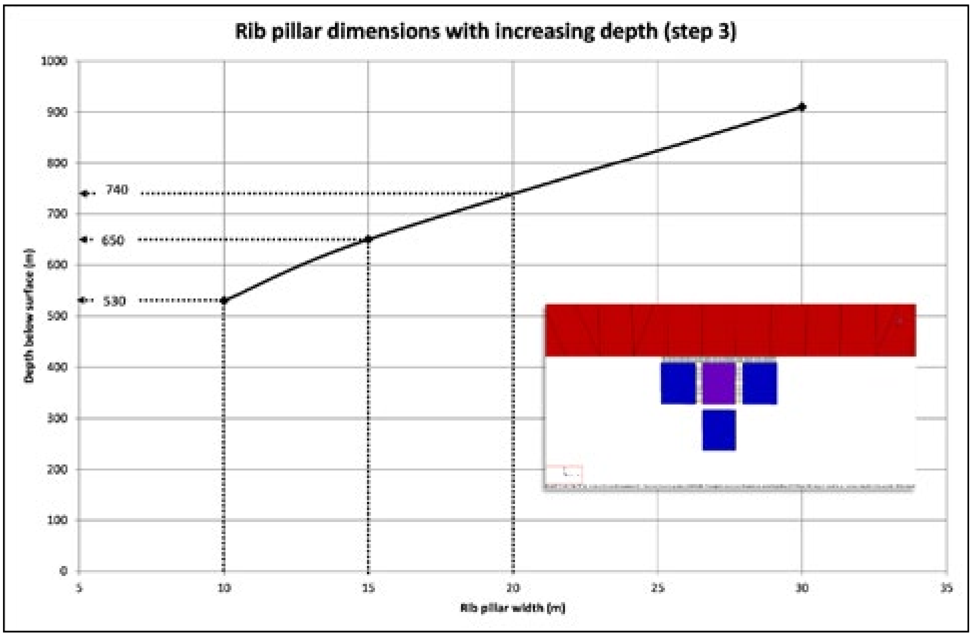


Figure 16: Mining step 3 limiting depth for rib pillars

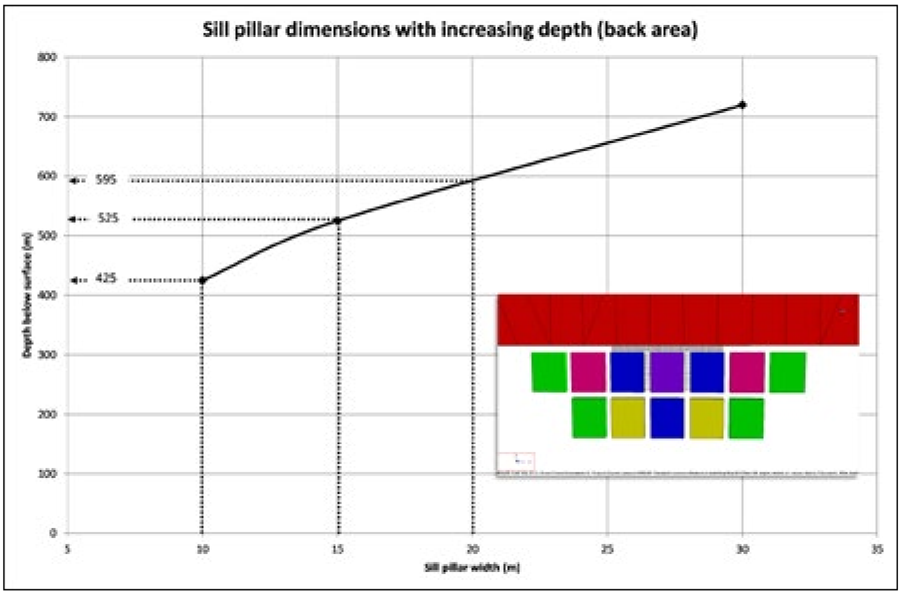


Figure 17: Step 6 limiting depth for sill pillars

EXAMPLE

Within a sub-level open stope layout where limited stope spans are required, the stopes are separated by sill pillars along strike and rib pillars along the dip of the ore body. These pillars have the function to ensure spans remain limited and backs remain stable, suggesting that the pillars

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PAPER 3.3: CHAPTER 4

must maintain integrity over long periods of time. Since both ore bodies dip > 45°, it is suggested that these pillars will not only be subjected to overburden loading, but will also be subjected to 'flexural' (sill pillar) or 'sliding' (rib pillar) failure under its own weight. In an attempt to determine the dimensions of sill and rib pillars, all three mechanisms were evaluated.

Failure under own weight
Based on work by Mitchell in 1991 (Pakalnis), backfill material beams created within open stopes are subjected to several failure mechanisms and these could be presented by a number of limit equilibrium equations (Figure 21). These failure criteria should also apply for rock material, which in essence is a very competent 'fill' material.

Figure 18: Limit equilibrium criteria for backfill sill pillar failure (Mitchell 1991) (Pakalnis)

Figures 19 and 20 show the results for ore body widths between 0m and 30m, as represented by factors of safety against failure. If it is assumed that a conservative factor of safety of 2 is appropriate, the resulting sill and rib pillar widths can be determined from these presentations, based on the applicable ore body width. In summary, the following pillar dimensions are suggested for an average 15m thick ore body:

- Sill pillar: 22.5m to prevent the pillar from failure by flexural deformation;
- Rib pillar: 4.5m to prevent the pillar from sliding along the ore body contact.

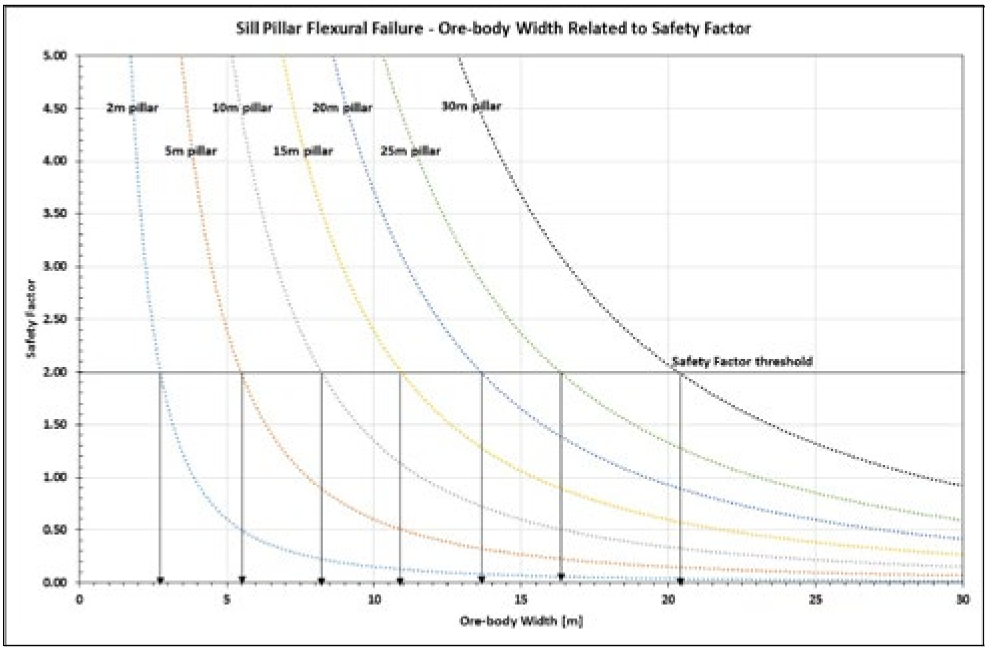

Figure 19: Sill pillar width

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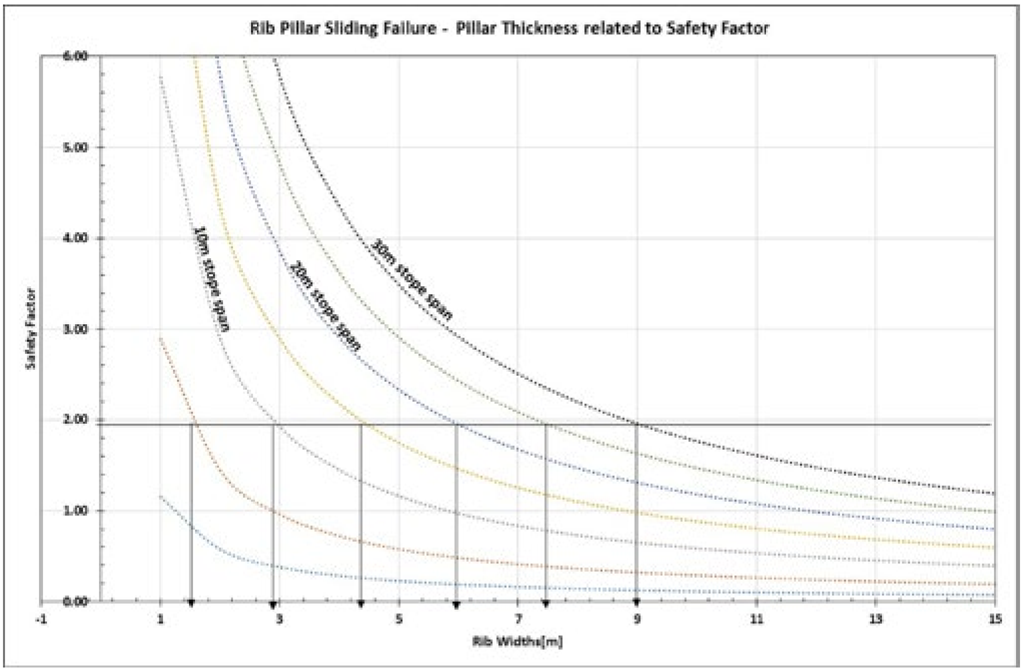

Figure 20: Rib pillar width

LEARNING OUTCOME 4.2.1.3

CONNECTION 8

Refer to Paper 3.1, Outcome 4.2.1.1.7 And "LEARNING OUTCOME 4.2.1.2" on page 239

DESIGN REGIONAL STABILITY PILLARS FOR WORKINGS AT SHALLOW DEPTHS

LEARNING OUTCOME 4.2.1.4

CONNECTION 9

Refer to Paper 3.1, Outcome 4.2.1.1.9

And "LEARNING OUTCOME 4.2.1.2" on page 239

DESIGN REGIONAL STABILITY PILLARS FOR WORKINGS AT INTERMEDIATE DEPTHS

LEARNING OUTCOME 4.2.1.5

CONNECTION 10

Refer to Paper 3.1, Outcome 4.2.1.1.9 And "LEARNING OUTCOME 4.2.1.2" on page 239

DESIGN REGIONAL STABILITY PILLARS FOR WORKINGS AT GREAT DEPTHS.

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4.2.2. CROWN PILLARS

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

LEARNING OUTCOME 4.2.2.1

SKETCH, DESCRIBE, EXPLAIN AND DISCUSS THE FUNCTIONS OF CROWN PILLARS FOR THE VARIETY OF MASSIVE MINING METHODS AT SHALLOW, INTERMEDIATE AND GREAT DEPTH

Crown pillar functions include:

- Protect underground mining of a massive ore body against any influences from mining conducted/conditions that exist above the crown pillar or
- Protect surface structures against the impact of the underground mining.

The application of crown pillars is known to be:

- Below the floor of an open pit when mining progresses underground;
- Between underground mining and surface;
- Between different levels of mining in an underground mine (often also called sill pillars).

LEARNING OUTCOME 4.2.2.2

SKETCH, DESCRIBE, EXPLAIN AND DISCUSS THE D7IMENSIONS AND LAYOUT OF CROWN PILLARS FOR THE VARIETY OF MASSIVE MINING METHODS AT SHALLOW, INTERMEDIATE AND GREAT DEPTH

The best known method to design crown pillars is an empirical method that makes use of a relationship between the crown pillar thickness, the rockmass quality and the ore body dip and thickness beneath the pits.

NOTE

Minimum stable crown pillar dimensions can be determined using the critical span model developed by Barton (1974) and modified by Golder Associates (1990).

Values for the rockmass rating Q and the scaled crown span Cs are related to the critical span, Sc.
For a given value of Q, crown pillars are potentially unstable if the scaled crown span, Cs, is greater than the critical span, Sc.
The critical span, Sc, is defined by the following relationship, where Q is the rockmass rating index (Barton):

$$S_c=3.3q^{0.3}(\text{Sinh})^{0.0016}\ Q$$

The scaled crown span, Cs, is defined by:

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$C_s = S[\gamma/t(1+S_R)(1-0.4Cos\theta)]^{0.5}$
Where S is the span of the pillar,
γ the specific gravity of the rock (e.g. 3.0 t/m³),
t the crown pillar vertical thickness,
S_R, the span to strike ratio and
θ the dip of the ore body.

EXAMPLE

Sections through the ore body at regular intervals below the pit allowed the detailed measurement of ore body thickness and dips at 50m intervals along the pit. The data indicated that the ore body at the shallow depths below the pit dips at variable angles and has a thickness that varies between 5 and 18.5m.

Based on this data, the minimum crown pillar thickness could be calculated to be between 7 and 80m, depending on the ore body thickness and dip and assuming that a rockmass quality of Q=1.5 is appropriate to this area.

Table 9: Ore body thickness, dip and crown pillar thickness

Slice	Y Coordinate	Av. Orebody Thickness in meters	Dip	**Crown pillar thickness with Q**
Slice 1	8550	10.4	54.0	30
Slice 2	8500	5.0	63.0	7
Slice 3	8450	10.0	81.0	22
Slice 4	8400	10.4	69.0	27
Slice 5	8350	14.1	73.0	46
Slice 6	8300	10.4	78.0	24
Slice 7	8250	10.4	68.0	27
Slice 8	8200	6.7	71.0	11
Slice 9	8150	19.3	67.0	80
Slice 10	8100	5.0	82.0	6
Slice 11	8050	11.9	86.0	30
Slice 12	8000	14.1	80.0	41
Slice 13	7950	9.6	75.0	20
Slice 14	7900	18.5	75.0	68
Slice 15	7850	9.6	53.0	26
Slice 16	7800	14.1	56.0	51

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Slice	Y Coordinate	Av. Orebody Thick-ness in meters	Dip	**Crown pillar thickness with Q**
Slice 17	7750	14.1	47.0	51
Slice 18	7700	7.4	68.0	14
Slice 19	7650	10.4	67.0	27
Slice 20	7600	11.1	50.0	35
Slice 21	7550	5.9	47.0	11
Slice 22	7500	9.6	65.0	23
Slice 23	7450	6.7	29.0	14
Slice 24	7400	5.9	85.0	8
Slice 25	7350	5.2	75.0	7
Slice 26	7300	6.7	61.0	11
Slice 27	7250	7.4	52.0	16
Slice 28	7200	10.4	65.0	27
Slice 29	7150	8.9	29.0	24
Slice 30	7100	8.9	69.0	20
Slice 31	7050	6.7	80.0	10
Slice 32	7000	9.6	88.0	20
Slice 33	6950	8.2	89.0	16
Slice 34	6900	8.6	88.0	17
Slice 35	6850	10.4	88.0	24
Slice 36	6800	12.6	84.0	34
Slice 37	6750	4.7	89.0	5

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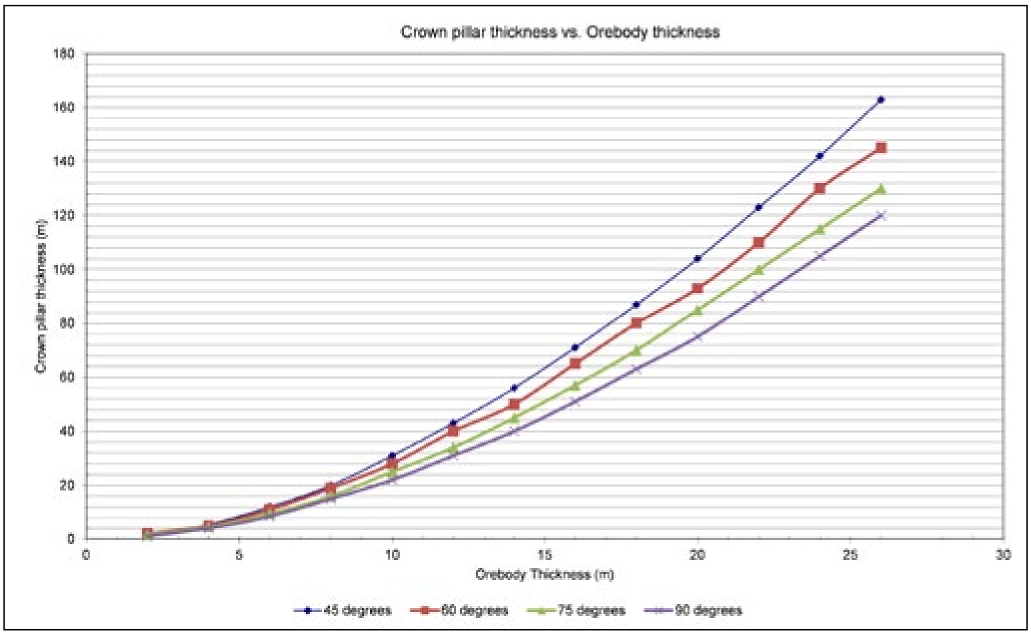

Figure 21: Crown pillar thickness at varying ore body dips

The following crown pillar thicknesses are suggested:

- 46m between y coordinates 8250 and 8550;
- 80m between y coordinates 8000 and 8250;
- 68m between y coordinates 7500 and 8000;
- 27m between y coordinates 7100 and 7500.

EXAMPLE

The critical span model developed by Barton (1974) and modified by Golder Associates (1990) has been used to estimate the minimum stable crown/sill pillar dimensions.

The following parameters were used to define the crown/sill pillar thickness:

A mean q value of 8.00 ;
A mass density of 2800 kg/m3
An ore body width of 30m and a strike length of 500m, which equates to a span ratio of 0.06
An ore body dip of 70°

A critical span of 8m (Figure 22) was derived, which was used to determine the vertical crown/sill pillar thickness at which the scaled span is not exceeded.

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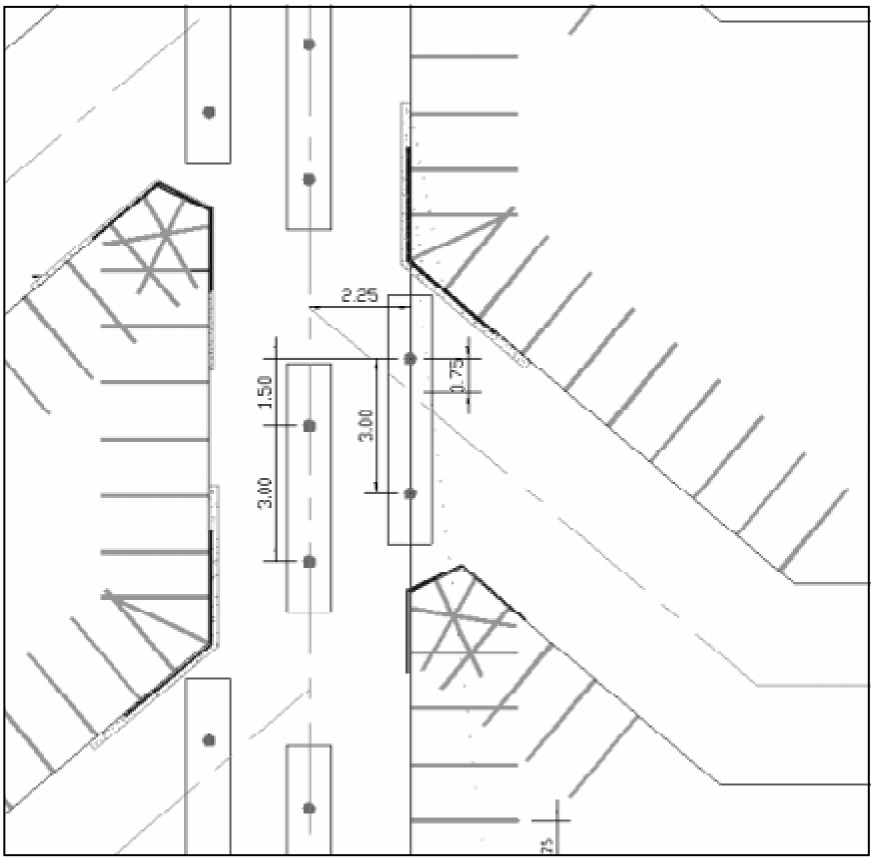

Figure 22: Critical crown/sill pillar span

The critical span was, in turn, related to the scaled span to derive the minimum crown/sill pillar thickness (Figure 23). A minimum thickness of 8.5m is suggested.

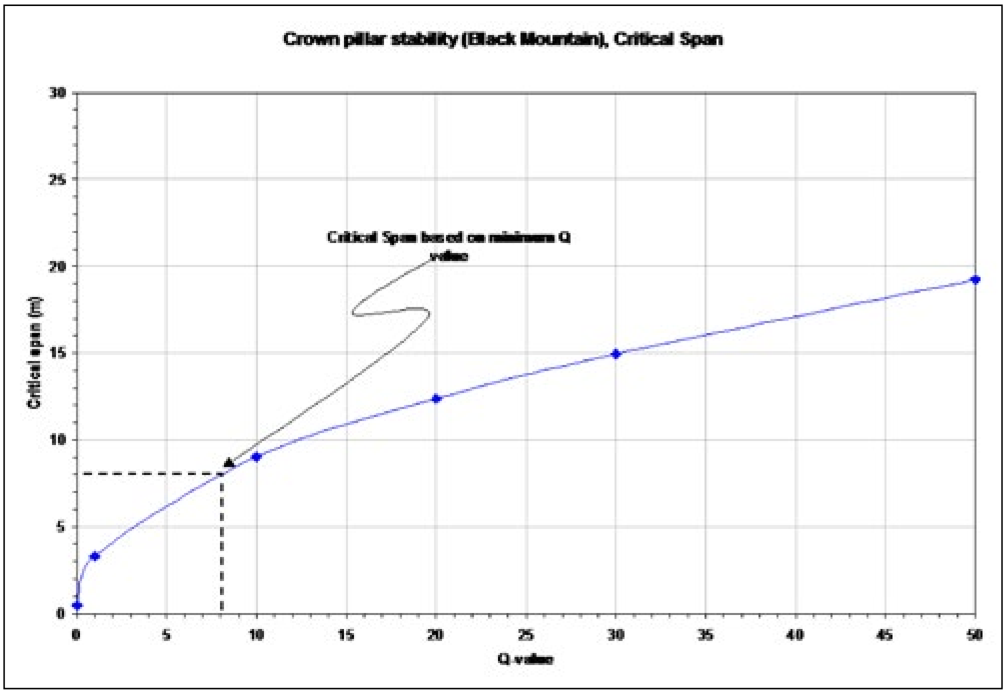

Figure 23: Minimum thickness of crown/sill pillar based on scaled span

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LEARNING OUTCOME 4.2.2.3

CONNECTION 11

Refer to
"LEARNING OUTCOME 4.2.2.2" on page 249

DESIGN CROWN PILLARS FOR MASSIVE ORE BODY WORKINGS

LEARNING OUTCOME 4.2.2.4

DESCRIBE, EXPLAIN AND DISCUSS THE PROBLEMS ASSOCIATED WITH DESIGNING ADEQUATE CROWN PILLARS

The following concerns must be noted:

- The design method currently utilised is empirical, which means that its founding is based on a specific environment where the pillar behaviour may or may not be appropriate to all other environments;
- The method is based on the quality of the material that forms the pillar, which has the following shortcomings:
    - The Q Index applied may not be appropriate as it is normally an 'average' value of the material quality over some area and is not completely representative of the pillar quality;
    - The average value applied means that parts of the pillar will consist of poorer material that could behave differently;
    - It assumes the process of determining the Q Index values was sound and appropriate;
    - Method is based on the ore body thickness and assumes a constant thickness in any specific area, which is normally not the case, and could lead to over or under estimation of the crown pillar thickness across an area;
    - Method assumes a constant ore body dip, which in most cases is not appropriate. This requires different analyses for different areas to be appropriate;
    - It ignores any other failure mechanism other than unravelling of the pillar due to its quality. Other mechanisms could include shear along the ore body contacts, beam failure, stress-driven failure, etc.

NOTE

Refer to the example in Outcome 4.2.1.2 as the method suggested by Mitchell for backfill/rock beam stability for other methods to determine crown pillar stability.

4.2.3. BACKFILL AS REGIONAL SUPPORT

CONNECTION 12

Refer to
Paper 2, Outcome 5.3
Paper 3.1, Outcome 4.2.2

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

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LEARNING OUTCOME 4.2.3.1

CONNECTION 13

Refer to Paper 2, Outcome 5.3.1 and 5.3.2

DESCRIBE, EXPLAIN AND DISCUSS THE FUNCTIONS OF BACKFILL AS REGIONAL SUPPORT FOR THE VARIETY OF MASSIVE MINING METHODS AT SHALLOW, INTERMEDIATE AND GREAT DEPTH

Backfill is intended to control stope wall behaviour and requires a particular stoping geometry and sequence to allow ore extraction to proceed and the backfill to fulfil its support potential.

Three mechanisms to demonstrate the support potential of mine backfill are provided by Brady and Brown (2006) and are shown in Figure 24:

- Preventing the displacement of key pieces in a stope boundary, backfill can prevent the progressive disintegration of the near-field rockmass, in low stress settings.
- The displacement of stope wall rock into the filled void mobilises the resistance of the fill.
- If the backfill is properly confined, it may assist to reduce the state of stress throughout the mine.

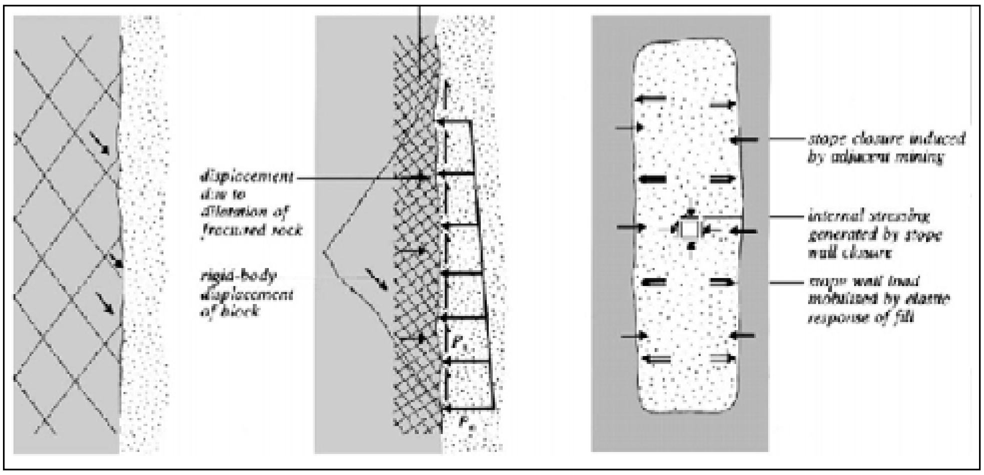

Figure 24: Backfill support mechanisms (Brady and Brown, 2006)

Backfill can be used as a support medium in mining practice:

- Conventional cut-and-fill stoping (by overhand or underhand methods): Backfill is placed regularly in successive lifts. The operational effectiveness is related to its capacity to provide a stable working surface after its placement in the stope.
- Open stoping operation: Backfill placement is delayed until production from the stope is complete. The performance of the backfill ensures that free-standing backfill walls (after pillar or secondary stope extraction) are stable.

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REFERENCE

- Page 408, Brady and Brown, Underground mining methods, 2006

LEARNING OUTCOME 4.2.3.2

CONNECTION 14

Refer to
Paper 3.1, Outcomes 4.2.2.1, 4.2.2.2, 4.2.2.3, 4.2.2.4

SKETCH, DESCRIBE, EXPLAIN AND DISCUSS TYPICAL PARTICLE SIZE DISTRIBUTION, FOR:

- Classified tailings backfill
- Full plant tailings
- Paste fill
- Rock fill
- Combinations of fill materials

LEARNING OUTCOME 4.2.3.3

CONNECTION 15

Refer to
Paper 3.1, Outcomes 4.2.2.5

SKETCH, DESCRIBE, EXPLAIN AND DISCUSS TYPICAL STRESS-STRAIN CURVES FOR:

- Classified tailings backfill
- Full plant tailings
- Paste fill
- Rock fill
- Combinations of fill materials

LEARNING OUTCOME 4.2.3.4

CONNECTION 16

Refer to
Paper 3.1, Outcome 4.2.2.6

SKETCH, DESCRIBE, EXPLAIN AND DISCUSS TYPICAL SGS FOR:

- Classified tailings backfill
- Full plant tailings
- Paste fill
- Rock fill
- Combinations of fill materials

LEARNING OUTCOME 4.2.3.5

CONNECTION 17

Refer to
Paper 3.1, Outcome 4.2.2.7

SKETCH, DESCRIBE, EXPLAIN AND DISCUSS TYPICAL DRAINAGE CHARACTERISTICS OF:

- Classified tailings backfill
- Full plant tailings
- Paste fill
- Rock fill
- Combinations of fill materials

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LEARNING OUTCOME 4.2.3.6

CONNECTION 18
Refer to
Paper 3.1, Outcome 4.2.2.8

SKETCH, DESCRIBE, EXPLAIN AND DISCUSS TYPICAL PUMPING CHARACTERISTICS OF:

- Classified tailings backfill
- Full plant tailings
- Paste fill
- Rock fill
- Combinations of fill materials

CONNECTION 19
Refer to
Paper 3.1, Outcome 4.2.2.9

Paste fill

REFERENCE

- Page 416, Brady and Brown, Underground mining methods, 2006

CONNECTION 20
Refer to
Paper 2, Outcome 5.3.3

Rock fill

REFERENCE

- Page 413, Brady and Brown, Underground mining methods, 2006

LEARNING OUTCOME 4.2.3.7

CONNECTION 21
Refer to
Paper 3.1, Outcome 4.2.2.1.10
Paper 2, Outcome 5.3.4

SKETCH, DESCRIBE, EXPLAIN AND DISCUSS THE USE AND EFFECTS OF CEMENTITIOUS BINDERS WITH THE ABOVE TYPES OF FILL

REFERENCE

- Page 412, Brady and Brown, Underground mining methods, 2006

LEARNING OUTCOME 4.2.3.8

CONNECTION 22
Refer to
Paper 3.1, Outcome 4.2.2.1.11

SPECIFY PERFORMANCE CRITERIA, BACKFILL TYPE AND PLACEMENT METHOD TO ENSURE SUCCESSFUL BACKFILLING ON A MINE.

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Backfill design for open stopes
Backfill requirements

To facilitate the selected mining methods, the required backfill characteristics have been listed in Table 8.

Table 10: Backfill requirements

Mining method	Method assumptions	Backfill characteristics and functions	Possible backfill types
Cut-and-fill	Complete ore body width is extracted in a single cut. No pillars are left intact. Fill exposed as working platform only. Fill will be undercut from next levels.	Working platform. Sill pillar. Undercut stability over cut width span (2.4-12.6m). Backfill supports the hanging and footwalls and limits closure, thereby reducing stress in the back (or roof).	If cemented sill is placed along stope bottom, any type can be used in cuts higher up in the stope.
Drift-and-fill	Small width drifts are mined next to each other until complete ore body width is extracted. No pillars are left intact. Fill exposed over height of drift. Fill will be undercut from next levels.	Working platform. Free-standing height (5m). Sill pillar. Undercut stability over drift width span (5m).	Cemented fill for free standing height.
Post pillar cut-and-fill	Complete ore body width is extracted in a single cut, leaving pillars of adequate dimension to be confined by the fill. Fill exposed as working platform only. Fill will be undercut from next levels.	Working platform. Sill pillar. Undercut stability over drift width span (12.6m).	If cemented sill is placed along stope bottom, any type can be used in cuts higher up in the stope.
Open stope with fill	Open stopes are created at suggested stable spans in primary and secondary extraction sequences and filled or fill is allowed to cure before mining continues in a single mining direction. No pillars are left intact. Fill exposed along vertical height of the open stopes. Fill will be undercut from next levels.	Working platform. Sill pillar. Free-standing height (20-40m). Undercut stability over stope width (2.4-12.6m).	Cemented fill for free standing height and undercutting stability.

Based on the analysis above, the critical backfill requirements for this operation would consist of free-standing heights (drift-and-fill and open stope with fill) to allow working adjacent to the fill and undercutting of the backfill sill pillar when mining occurs below a backfilled column.

Free standing height

Free standing backfill heights are analysed kinematically using an approach by Mitchell (Brady & Brown, 2006) and provide the maximum height a backfill column could be exposed to while ensuring that the fill will not collapse under its own weight. This concept indicates that non-cemented backfill types are not conducive to vertical stability if unconfined and therefore suggests that uncemented hydraulic and/or rockfill types can only be used when the free-standing requirement is not critical, as in the cut-and-fill and post-pillar cut-and-fill methods.

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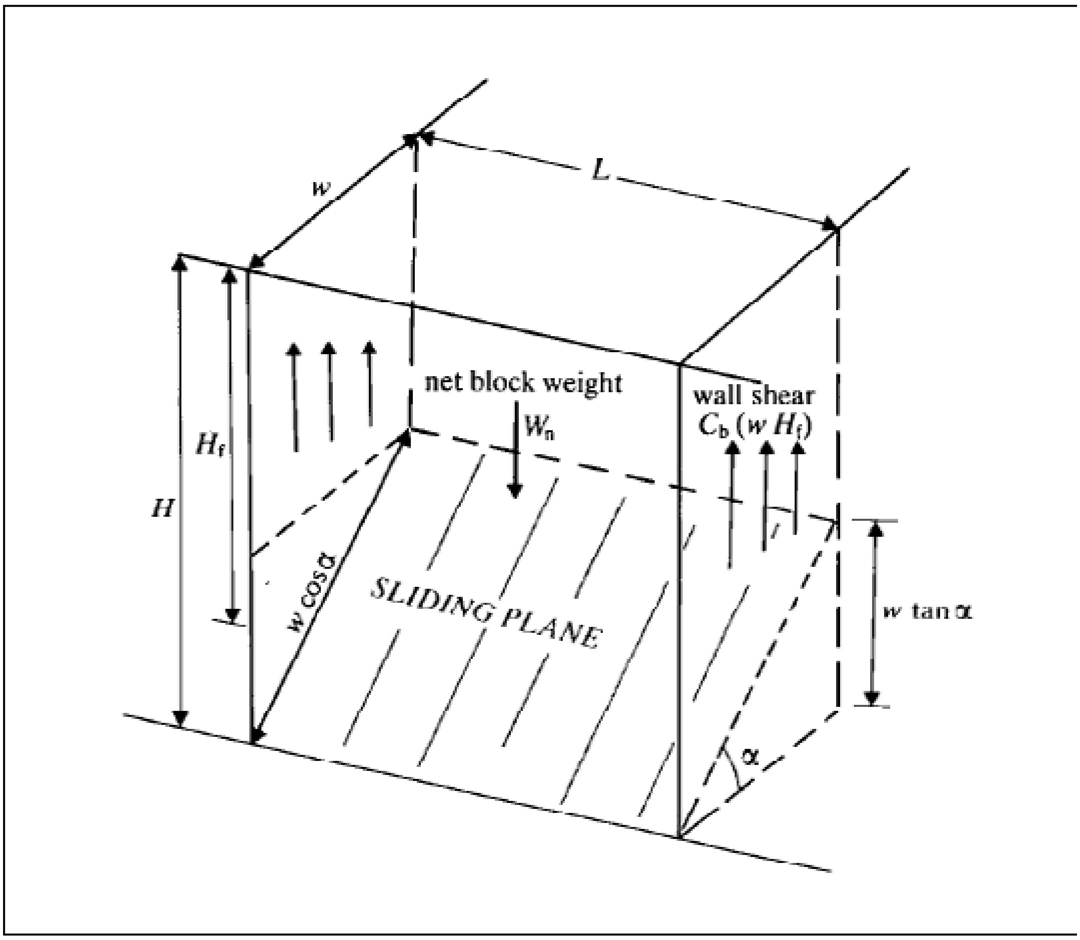
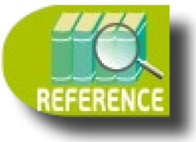


Figure 25: Free standing heght design (Brady & Brown, 2006)



- Page 418, Brady and Brown, Rock Mechanics for underground mines, 2006

Undercutting backfill columns

Undercutting of the fill, as required in all the methods listed in Table 10, assumes that the backfill sill pillar has the ability to remain stable across a certain unsupported, exposed span. This ability is a function of the fill material and strength, but also of the width (span) of backfill exposure in the hangingwall.

A literature study on undercut-and-fill operations and backfill designs implemented on these operations was conducted and relevant issues were summarised into Table 11. The study shows that the undercutting of backfill requires high backfill strengths (> 4MPa) and often also the use of additional support within the placed fill to ensure safe and stable conditions, especially in environments where seismic activity is expected.

Table 11: Literature study

Source	Span	Backfill type	Backfill strength	Additional support	Other comments

Underhand cut-and-fill at Lucky Friday Mine. C Peppin et al. Underground mining methods, WA Hustrulid, 2001	Average 1.2m to 1.8m spans. For areas where the ore body width > 5m, additional support is installed.	Paste fill with additives to improve flow. Sand made from tailings. Mix 2558kg sand, 304kg cement and 360 litres water (9% cement by weight).	1.4MPa to 1.9MPa after 3 days, 2.6MPa after 14 days and 3.8MPa after 28 days.	2.0m length bolts installed at 1.2m spacing in backfill, mesh, timber support and cables also utilised.	High stress and seismic conditions on the mine. Fill with strength < 1.5MPa does not stay up over 2.5m span, fill with strength > 4MPa fails under seismic conditions. Re-entry underneath fill in 3-5 days.
Underhand cut-and-fill mining at Murray mine, Jerrit Canyon Joint Venture, CE Brechtel et al. Underground mining methods, WA Hustrulid, 2001	6.4m to 10.7m during secondary extraction beneath backfill.	Rock fill using limestone aggregate that is crushed on-site. 6.5% cement.	4.96MPa	Bolts, shotcrete during primary extraction in ore body. Secondary extraction underneath backfill.	21 days curing but in practice more than 90 days before re-entering. Poor ore body quality and strength. Drift-and-fill method, jamming fill into place with LHDs.
Underhand cut-and-fill at the Barrick Bullfrog Mine, D Kump et al. Underground mining methods, WA Hustrulid, 2001	4m wide drifts.	Cement with footwall waste aggregate with 7% fill in high strength and 3-4% cement in low strength fill areas.	3.5MPa after 30 days curing.	Shotcrete when ore forms the hanging-wall.	Poor strength ore. Transport fill in trucks and shove into place.
Evolution of cut-and-fill at SMJ's Jouac Mine, France, R le Roy. Underground mining methods, WA Hustrulid, 2001	Small span drives. Actual span not specified.	Cemented fill.	12MPa.	Only when in original ore, bolting.	Backfill pillars are created. Not 100% of mined area filled.

Designing narrow undercut and fill exposures, T Grice, AMC Consultants, 2004.	Backfill failure mechanisms include flexural bending, caving, crushing, sliding, shearing and rotation. Caving appears to be dominant in steep dip areas with backfill width > 5m, For caving failure, the tensile strength of the material is critical. Thickness of the backfill 'sill pillar' has been shown to be in the region of the 0.5 x fill width, but is normally increased to equal to the fill width to cater for fill inconsistencies.

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Design spans in underhand cut-and-fill mining, R Pakalnis, 107th CIM-AGM Toronto, 2005.	The development of paste fill during the 1990s has changed underhand cut-and-fill to the application of backfill, without additional support, as roofs in workings. This is the result of engineered conditions and a high degree of constant supply of backfill in terms of the strength and behaviour of the backfill. Factors affecting the backfill design include vertical loading, stope geometry, failure modes, strength properties, stope closure etc. Numerical modelling could be repeated using analytical methods for caving, flexural, sliding and rotational failures. Of these, the flexural mode is viewed as most critical. Crushing, sliding and caving modes can be prevented by ensuring the high quality fill thickness exceeds 0.5 x span, the backfill UCS > 1.5Mpa and stope closure is absent. Rotational failure is suggested to be analysed separately. Paste fill properties are suggested in terms of friction angle of 30°, stiffness modulus of between 3 and 600MPa (10% binder).
An overview on the use of paste fill technology as a ground support method in cut-and-fill mines, T Belem et al. Ground support in mining and underground construction, 2004.	Analytical methods to establish backfill strengths in free-standing environments are shown and discussed based on work performed by Mitchell.

Backfill design

The selection of possible modes of failure of a backfill column is critical in establishing backfill strengths that will prevent the occurrence of backfill column failure while applying the fill to perform the allocated functions.

The possible backfill column failure modes include:

- Shear failure through fill: Collapse of a large exposed surface of backfill (free-standing height) due to the creation of a shear plane within the fill column, driven by the weight of the backfill column only;
- Sliding failure along fill/rock contact: Shear of the backfill column (sill) along the ore body surfaces and collapses into the mined-out void below the backfill;
- Flexural failure: Bending of the backfill column (sill) into the mined-out void below the column and the creation of tensile stresses in the centre of the mined-out span;
- Crushing failure: Crushing of the backfill due to excessive side-wall closure;
- Caving: Collapse of the backfill due to caving of the backfill material into the mined-out void below;
- Rotational failure: Rotation of the fill column due to the orientations of the stope boundaries and the dimension of fill column;
- Seismic or dynamic failure: Excluded until appropriate seismic data is available.

Shear failure through fill: Free-standing ability

The ability of backfill to remain stable at excessive heights and widths in a filled stope is critical to the successful mining of a pillar during secondary extraction. Mitchell (Brady & Brown, 2006) developed an approach that has been successful for more than 20 years of application in industry. Application of the Mitchell approach (frictionless contacts with rock boundaries) to open stope and cut-and-fill/drift-and-fill layouts is provided in Figures 26 and 27 and yielded approximate backfill strengths that

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should ensure free-standing stability of the appropriate backfill columns.

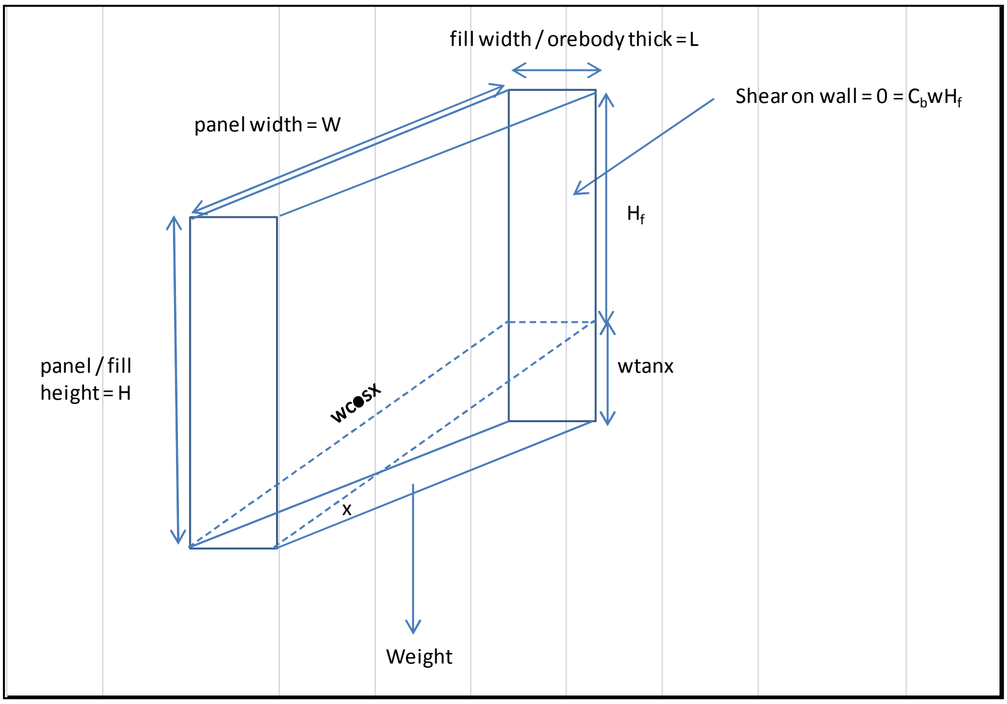

Figure 26: Open stope

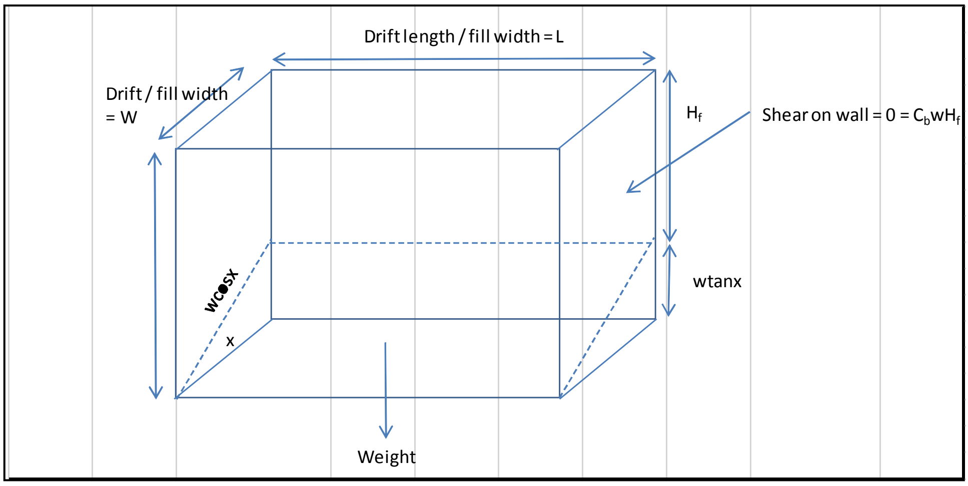

Figure 27: Drift-and-fill

Input parameters are shown in Table 12 below with the resultant backfill strengths calculated in Table 13.

Table 12: Input parameters (free standing height)

Dimensions		Properties
Open stope	Drift-and-fill	
H = 40m (2 levels of 20m)	H = 5m	Friction angle = 30°
L = 40m (maximum supported back span)	L = 50m	Density = 1800kg/m3
W = 13m (maximum ore body width)	W = 5m	Gravitational acceleration = 9.81m.s-2

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Table 13: Free-standing height backfill strengths

Mining layout	Standing height	UCS (MPa)	Factor of safety
Open stope	40m	0.5	1.2
Drift and fill	5m	0.037	1.2

Sill pillar: Rotational failure (Kinematic analysis)

Since the possibility of encountering rotational failure is a function of the orientation of the fill column boundaries, a two-dimensional kinematic analysis was conducted. The expected backfill sill pillar has a parallelogram shape (section on dip) and will kinematically not be able to rotate (Figure 28) when:

- the sum of the angles 'e' and 'B' in Figure 28 is greater than 90°;
- the ratio of the sill pillar thickness to its length > 0.5 (Table 14) and
- the ratio of the diagonal length (R2) to ore body width (R3) > 1.0 (Table 14).

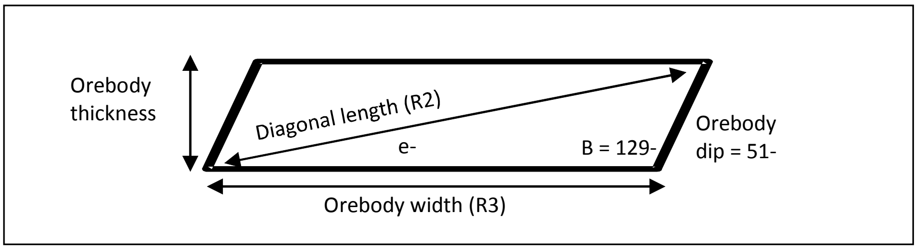

Figure 28: Kinematic rotation

Table 14: Kinematic rotation indicators

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Orebody width – R3 (m)	Still thickness (m)			Still thickness (m)		
	5	10	20	5	10	20
	Sill thickness to orebody width ratio			Diagonal length (R2) to Ore-body thickness (R3) ratio		
2	2.50	5.00	10.00	3.92	7.11	13.52
3	1.67	3.33	6.67	2.88	4.98	9.24
4	1.25	2.50	5.00	2.37	3.92	7.11
5	1.00	2.00	4.00	2.07	3.30	5.83
6	0.83	1.67	3.33	1.87	2.88	4.98
7	0.71	1.43	2.86	1.73	2.59	4.38
8	0.63	1.25	2.50	1.63	2.37	3.92
9	0.56	1.11	2.22	1.55	2.20	3.57
10	0.50	1.00	2.00	1.49	2.07	3.30
11	0.45	0.91	1.82	1.44	1.96	3.07
12	0.42	0.83	1.67	1.40	1.87	2.88

Results in Table 12 indicate that only one of the kinematic criteria for rotation to occur is exceeded, and then for a 5m sill pillar thickness at an ore body width that exceeds 10m. Since all the other criteria are still met, it is concluded that rotational failure of a backfill sill pillar is highly unlikely. However, this evaluation assumes that the fill will act as a 'solid beam' and that crushing of the pillar edges will not occur.

REFERENCE

- Design spans for undercutting fill Pakalnis

Sill pillar: Caving, Rotational and flexural failure

Applying the limit equilibrium analyses suggested by Mitchell (Pakalnis, 2005), the potential of backfill column failure when applied as a sill pillar was analysed and is reported in Figures 29, 30 and 31 below.

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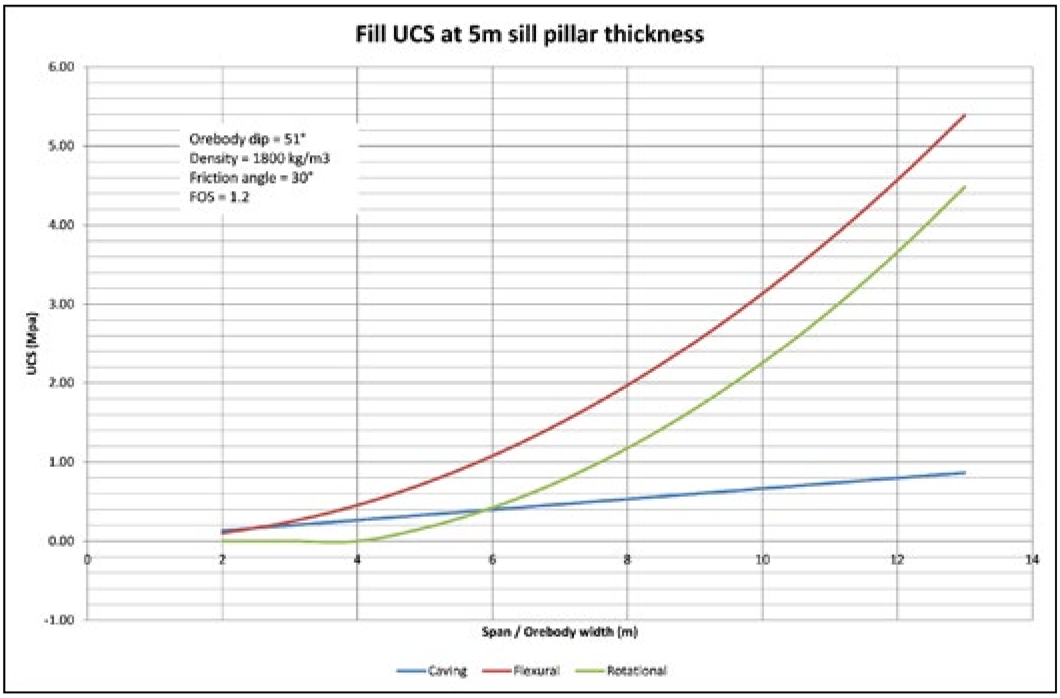

Figure 29: UCS/span relationship at 5m backfill sill pillar thickness

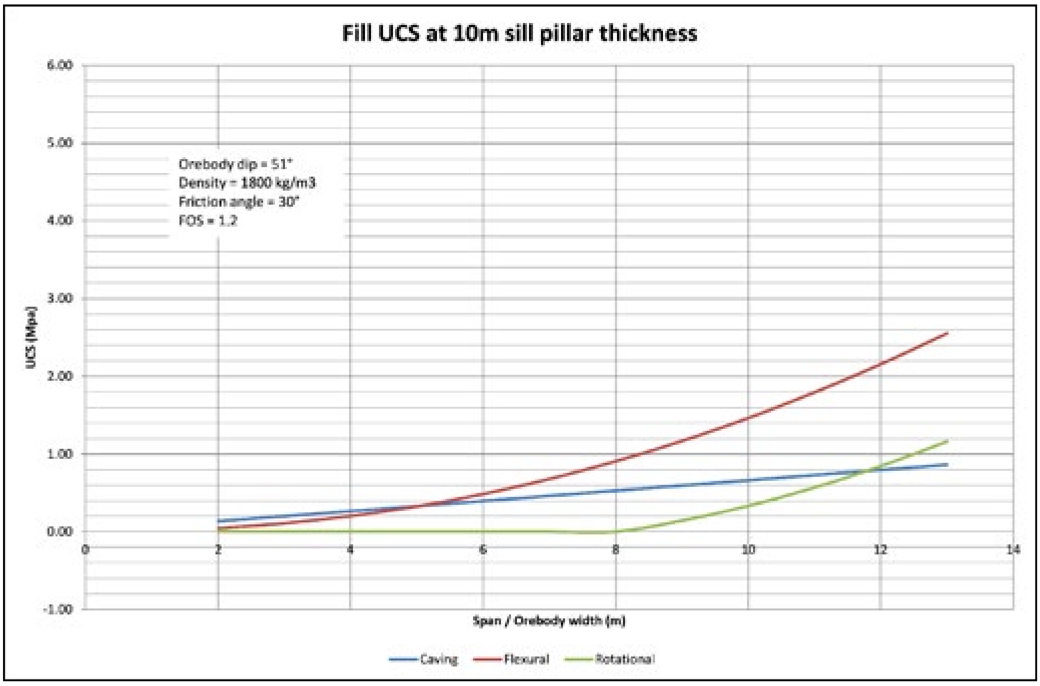

Figure 30: UCS/span relationship at 10m backfill sill pillar thickness

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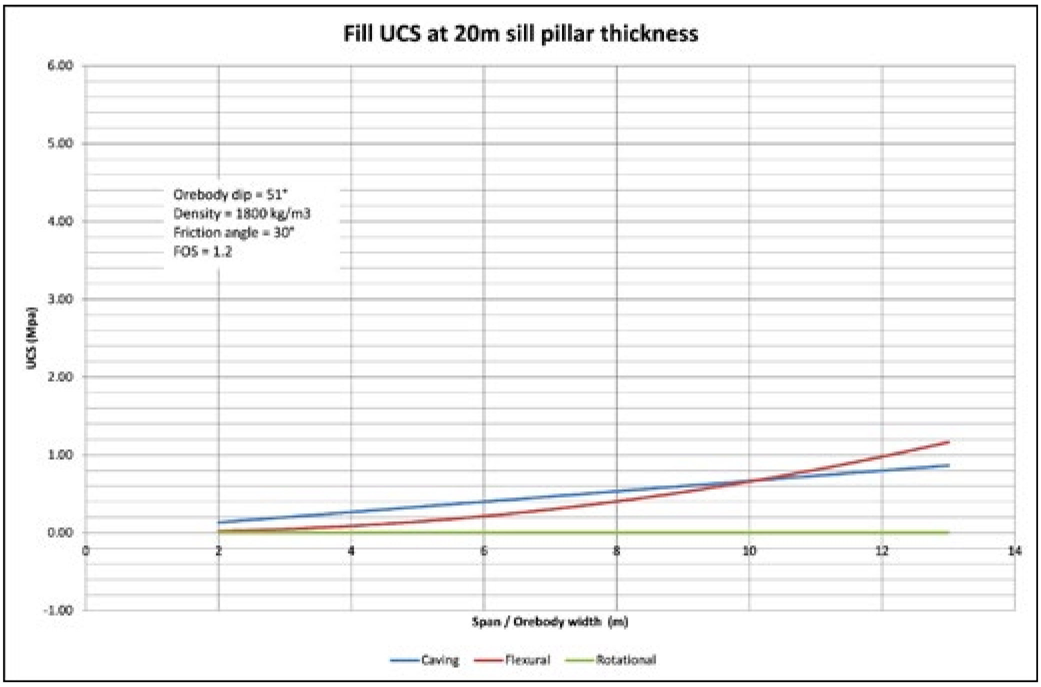

Figure 31: UCS/span relationship at 20m backfill sill pillar thickness

- Design spans for undercutting fill Pakalnis

Sill pillar: Crushing and shear (sliding) along stope boundaries
These failure modes will be addressed by assuming that the conclusions made by Pakalnis et al. (2005) and listed below are appropriate to the final backfill design:

- UCS of fill will exceed 1.5MPa;
- No wall closure occurs;
- That the backfill sill pillar thickness will exceed at least 0.5 x the ore body width.

Table 15: Suggested backfill sill pillar UCS based on failure mechanisms

		Sill pillar thickness (m)									
		5	10	20	5	10	20	5	10	20	**All**
		Backfill UCS (Mpa)									
		Caving			Flexural			Rotational			Shear
Orebody width (m)	5	0.33	0.33	0.33	0.73	0.33	0.14	0.17	0.00	0.00	1.50
	10	0.67	0.67	0.67	3.14	1.47	0.60	2.25	0.34	0.00	1.50
	13	0.86	0.86	0.86	5.39	2.55	1.16	4.48	1.16	0.00	1.50

- Design spans for undercutting fill Pakalnis

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Conclusions

For the current operations, at an average 51° ore body dip and maximum 13m ore body thickness, the following is suggested:

- For the drift-and-fill and open stope mining methods:
    - Cement addition to rockfill material for free-standing height to a 0.037MPa strength for the 'drift-and-fill' and 0.5MPa strength for the 'open stoping with backfill' methods;
- For backfill application as a sill pillar in all mining methods:
    - the analysis of the failure modes suggests that rotational failure would be kinematically impossible to occur, leaving the flexural and shear mechanisms as the major contributors to the required sill pillar strength.
    - The design methods utilised in this document were indicated to be appropriate to cemented rockfill and are therefore appropriate for this operation;
    - The use of uncemented rockfill in mining methods and areas within a stope where free-standing height and/or undercutting of the fill is not applicable, is acceptable;
    - The use of cemented rockfill as the sill pillar material to allow undercutting increases the requirement for the following practices:
- It is assumed that sufficient curing of the cemented rockfill sill pillar will occur prior to being undermined;
- Execution of a detailed fill design in terms of the use of local rock types, rock particle sizes, rock conditions (fresh/weathered), cement addition and control of mixing during preparation and possible segregation during placement, especially in open stopes;
- Investigation into the need for additional support practices to ensure a stable hangingwall in terms of mesh and bolting once the final backfill material design has been completed;
- Quality control on the strength of the placed fill material to allow alternative measures in areas where strength is suspect.
- Based on the results indicated in Table 15, it is suggested the backfill sill pillars will consist of:
    - Cement addition to rockfill material for application as sill pillars in all mining methods to a minimum strength of 1.5MPa for at least 10m thick sill pillars in ore body widths < 10m and
    - Cement addition to rockfill material for application as sill pillars in all mining methods where the ore body width exceeds 10m to:
- a minimum strength of 2.6MPa for 10m thick sill pillars or
- a minimum strength of 1.5MPa for 20m thick sill pillars.

REFERENCE

- Backfill and open stopes Doerner
- Paste fill design Karim
- Mining Eng Handbook Chapter 19.1 cut and fill stoping

CONNECTION 23

Refer to
Paper 3.1, Outcome 4.2.3

4.2.4. MINE LAYOUT FOR REGIONAL SUPPORT

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

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LEARNING OUTCOME 4.2.4.1

CONNECTION 24
Refer to Paper 3.1, Outcome 4.2.3.1

SKETCH, DESCRIBE, EXPLAIN AND DISCUSS MINING LAYOUT STRATEGIES TO ENSURE REGIONAL STABILITY OF MASSIVE WORKINGS AT SHALLOW, INTERMEDIATE AND GREAT DEPTH

LEARNING OUTCOME 4.2.4.2

CONNECTION 25
Refer to Paper 3.1, Outcome 4.2.3.2

SKETCH, DESCRIBE, EXPLAIN AND DISCUSS MINING LAYOUTS TO ENSURE REGIONAL STABILITY OF WORKINGS AT SHALLOW, INTERMEDIATE AND GREAT DEPTH

LEARNING OUTCOME 4.2.4.3

CONNECTION 26
Refer to Paper 3.1, Outcome 4.2.3.3

SKETCH, DESCRIBE, EXPLAIN AND DISCUSS MINING SEQUENCES TO ENSURE REGIONAL STABILITY OF WORKINGS AT SHALLOW, INTERMEDIATE AND GREAT DEPTH

LEARNING OUTCOME 4.2.4.4

CONNECTION 27
Refer to Paper 3.1, Outcome 4.2.3.4

DESIGN MINING STRATEGIES, LAYOUTS AND SEQUENCES TO ENSURE THE REGIONAL STABILITY OF MASSIVE WORKINGS AT SHALLOW, INTERMEDIATE AND GREAT DEPTH.

4.3. LOCAL STABILITY STRATEGIES

4.3.1. MINING LAYOUTS AND SEQUENCES

CONNECTION 28
Refer to Paper 3.1, Outcome 4.3

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

LEARNING OUTCOME 4.3.1.1

CONNECTION 29
Refer to "LEARNING OUTCOME 4.1.1.1" on page 223

SKETCH, DESCRIBE, EXPLAIN AND DISCUSS MINING STRATEGIES, LAYOUTS AND SEQUENCES TO ENSURE LOCAL STABILITY OF MASSIVE WORKINGS AT SHALLOW, INTERMEDIATE AND GREAT DEPTH

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LEARNING OUTCOME 4.3.1.2

CONNECTION 30

Refer to "LEARNING OUTCOME 4.1.1.1" on page 223 and "LEARNING OUTCOME 4.1.1.2" on page 235

SKETCH, DESCRIBE, EXPLAIN AND DISCUSS LAYOUTS AND SEQUENCES FOR THE FOLLOWING MASSIVE MINING METHODS:

- Open stoping methods
- Vertical crater retreat
- Sub-level caving methods
- Block caving methods
- Cut and fill methods
- Drift and fill methods
- Bord and pillar methods

Sequencing attempts to satisfy the following issues:

- the expected intermediate and final states of stress in a mining layout;
- the geomechanical constraints;
- specific mining requirements (i.e. production rate) and
- a tolerable level of risk.

The following general principles for optimum sequencing are provided by Brady and Brown (2006):

- To assure recovery of the highest grade blocks to maximise mineral yield;
- To ensure predictable behaviour of the mine layout:
- early extraction of blocks with little support potential;
- avoid scattered pillars/remnants;
- bracket pillars to control displacements on major structural features (where and if required) and
- retreat mining towards stable or solid ground.

NOTE

In the massive ore body environment, high pre-mining stresses are mostly not the driver for an appropriate extraction sequence (as in deep level gold mines in RSA) as it is still possible to lose a proportion of the ore reserve through a poor extraction sequence if field stress levels are low.

A good extraction sequence is one where:

- the structural geology is included in the evaluation;
- the changing stress distributions in the extraction layout are known;
- an initial point of attack in the ore body is identified that would allow a logical execution of the stope and pillar layout (including pillar extraction where appropriate).
- major installations are protected against mining-induced damage for as long as possible (i.e. ventilation shafts and airways, workshops, haulages, drives, ore passes and holes used for placement of backfill);
- a thorough geomechanical assessment has been conducted that

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includes the estimation of rockmass response, the analyses of stress changes, instability of large wedges or slip and separation on planes of weakness;

- seismic responses to the selected sequence have been evaluated (where appropriate);
- the amount of work to be conducted in areas subject to high stress or potential instability has limited exposure to mine personnel. This is usually achieved by avoiding the creation of narrow remnants/pillars.

NOTE

Sequencing is often simple in a single ore body, while for multiple ore bodies or a large ore body mined in a three-dimensional structure of stopes and pillars, it can become complex.

It is often necessary to evaluate several extraction sequences that satisfy production scheduling, engineering requirements (e.g. access, ventilation) and also maintain an acceptable direction of retreat mining of stoping towards stable ground.

In some instances, the generation of mining-induced seismicity has also started to become part of the list of factors to be evaluated when considering an extraction sequence. The stope and pillar layout illustrated in Figure 32a is based on primary stoping of every second ore block and recovery of the 'pillars' as secondary mining stopes have shown to have advantages in terms of controlling ground displacements in the initial stages of mining, but when field stress levels are high, could induce seismic activity during secondary extraction. Under those conditions, a pillar-less, outwards stoping sequence, shown in Figure 32b, usually using cemented backfill to limit the unsupported spans of stope walls, is implemented to reduce the seismic signature of the rockmass.

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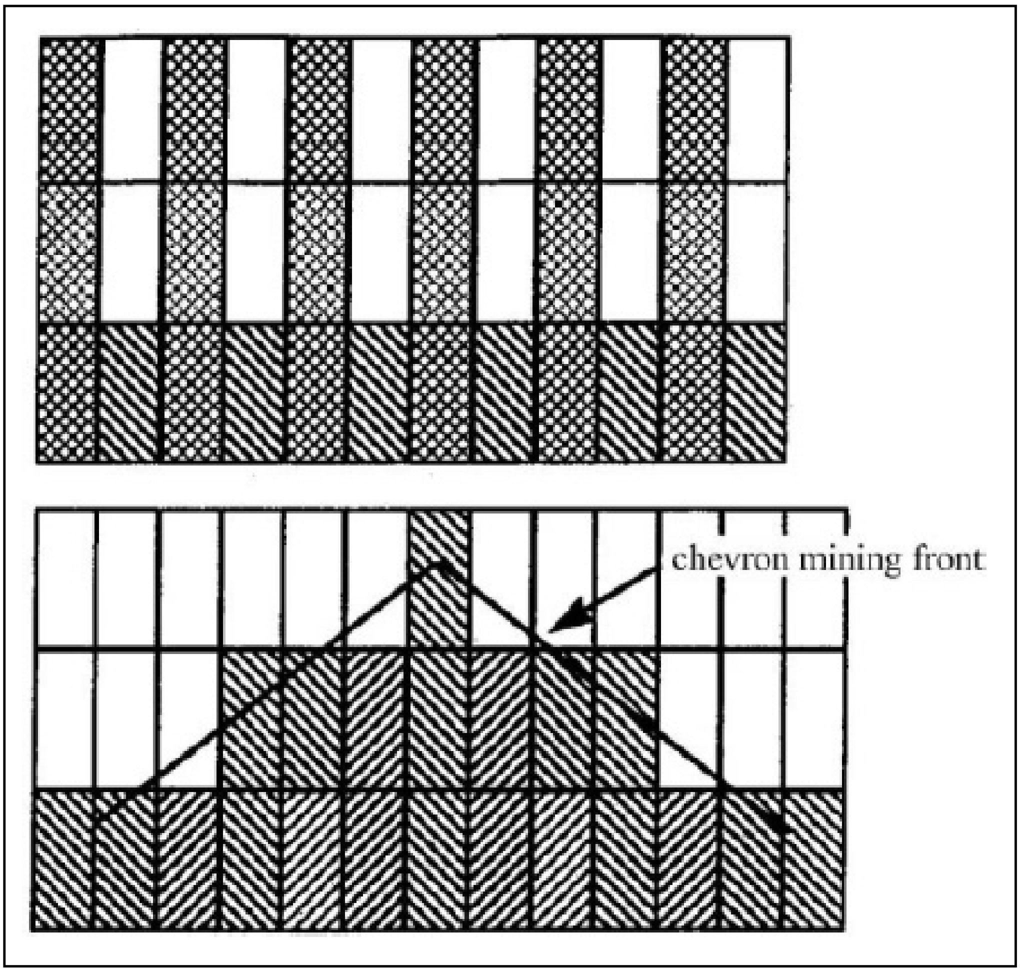

Figure 32: (a) Primary –secondary extraction sequence (b) outwards extractin sequence (Brady & Brown, 2006)

REFERENCE

- Global extraction sequence Villaescusa
- Page 400, Brady and Brown, Rock mechanics for underground mines, 2006
- Page 108, Updated cave mining handbook Laubscher

LEARNING OUTCOME 4.3.1.3

CONNECTION 31

Refer to "LEARNING OUTCOME 4.1.1.1" on page 223 and "LEARNING OUTCOME 2.3.1.2" on page 60, and "LEARNING OUTCOME 2.3.1.6" on page 62

EVALUATE GEOTECHNICAL DATA TO SELECT APPROPRIATE MASSIVE MINING METHODS BASED ON STABILITY, DILUTION AND EXTRACTION CONSIDERATIONS

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LEARNING OUTCOME 4.3.1.4

DESIGN APPROPRIATE MASSIVE MINING LAYOUTS USING ROCKMASS CLASSIFICATION TECHNIQUES

CONNECTION 32

Refer to "LEARNING OUTCOME 4.1.1.1" on page 223 and "LEARNING OUTCOME 2.3.1.2" on page 60, and "LEARNING OUTCOME 2.3.1.6" on page 62

LEARNING OUTCOME 4.3.1.5

DESIGN THE LAYOUT OF PRODUCTION TUNNELS, DRAWPOINTS AND DRILLING TUNNELS FOR OPEN STOPING AND SUB-LEVEL CAVING OPERATIONS FOR GIVEN GEOLOGICAL, GEOTECHNICAL AND STRESS CONDITIONS

CONNECTION 34

Refer to
Refer to "LEARNING OUTCOME 4.1.1.1" on page 223 and "LEARNING OUTCOME 4.1.1.2" on page 235

And "LEARNING OUTCOME 3.2.2.4" on page 107

LEARNING OUTCOME 4.3.1.6

DESIGN THE LAYOUT OF PRODUCTION TUNNELS AND THE UNDERCUTS FOR BLOCK CAVING OPERATIONS FOR GIVEN GEOLOGICAL, GEOTECHNICAL AND STRESS CONDITIONS

CONNECTION 35

Refer to "UNDERCUT LAYOUT AND DESIGN" on page 273 and "LEARNING OUTCOME 3.2.2.4" on page 107

LEARNING OUTCOME 4.3.1.7

DESIGN PRODUCTION EXCAVATION SEQUENCES FOR THE VARIOUS MASSIVE MINING METHODS FOR GIVEN GEOLOGICAL, GEOTECHNICAL AND STRESS CONDITIONS.

CONNECTION 36

Refer to "4.3.3 SERVICE EXCAVATION LAYOUTS" on page 273
Refer to examples in "LEARNING OUTCOME 4.2.1.2" on page 239 and "LEARNING OUTCOME 3.2.2.4" on page 107

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4.3.2. UNDERCUT LAYOUT AND DESIGN

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

LEARNING OUTCOME 4.3.2.1

CONNECTION 36

Refer to "LEARNING OUTCOME 2.3.1.9" on page 70, "LEARNING OUTCOME 3.3.1.3" on page 163, and "LEARNING OUTCOME 3.3.1.4" on page 169

DETERMINE AND DESIGN APPROPRIATE MASSIVE MINING UNDERCUT DIMENSIONS AND LAYOUTS USING ROCKMASS CLASSIFICATION TECHNIQUES

4.4. SERVICE EXCAVATION LAYOUTS

4.4.1. TUNNEL LAYOUTS

CONNECTION 37

Refer to
Paper 3.1, Outcome 4.4.1
Paper 3.2, Outcome 4.4.1

LEARNING OUTCOME 4.4.1.1

IMPORTANT

It is suggested that some situations specific to the massive mining environment will not be directly addressed in the outcomes referenced above but that sufficient background knowledge is provided to allow application of the knowledge to the massive mining environment.

A short list of conditions that may be specific to the massive mining environment are briefly listed below:

1. Access ramps:
   - Access ramps are placed within the waste rock mass, mostly in the orebody footwall (unless the shape of the orebody is similar to a kimberlite pipe) so that access tunnels can be developed towards the orebody;
   - In many ramps, passing bays are required to allow for two-way travelling of vehicles up and down the ramp;
   - In some instances, a ventilation hole is provided close to the ramp to allow its development in well ventilated conditions. This necessitates the development of drives towards the ventilation hole at regular spacings;
   - A ramp is usually of a circular or "figure 8" shape, resulting in the excavation intersecting the inherent joint sets at all possible angles, making the control of joint driven failures more difficult;

NOTE

In limited cases accesses have been placed in the orebody hangingwall but then only if the footwall material is so incompetent that its long term stability cannot be ensured

2. Large excavations:
   - In most cases, due to the ore transport system utilised, large underground excavations are required for services such as ore storage, crushing of ore, workshops etc;

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3. Loading bay areas:
   - Transfer of ore between transport modes or vehicles require larger excavation dimensions to allow free movement / loading and unloading. The positions of these areas is critical in the mining plan but must be placed such that favourable ground control conditions are created;

LEARNING OUTCOME 4.4.1.2

CONNECTION 38

Refer to Paper 3.1, Outcome 4.4.1.1

DESCRIBE, EXPLAIN AND DISCUSS THE ROCK ENGINEERING CRITERIA USED TO DETERMINE THE OPTIMUM SIZE, SHAPE AND ORIENTATION OF SERVICE TUNNELS

IMPORTANT

It is suggested that some situations specific to the massive mining environment will not be directly addressed in the outcomes referenced above but that sufficient background knowledge is provided to allow application of the knowledge to the massive mining environment.

The following is deemed in addition to the above and should be viewed in terms of the knowledge gained from the Outcome referenced above:

1. Size:
   - In most instances, due to the use of mechanical equipment in a massive mining environment, the size of the excavations are governed by the size of the equipment required to ensure optimum production profiles;
   - However, the input from the rock engineer on stable excavation dimensions and the need and cost for support to maintain open excavations should be considered by the mining engineers when deciding on the optimum excavation dimensions;
   - Most current massive mines are relatively shallow depth (compared to gold mines as discussed in paper 3.1), the stress impacts on excavations only become critical in incompetent rock mass, weak rock material, or higher stressed areas created by the mining method / sequence followed;
   - Until stress impacts realise, the main impact on excavation dimensions would be the creation of unstable wedges or unravelling of rock mass due to the exposure of joints;
2. Shape:
   - As for the size, the excavation shape is often impacted n by the mechanical equipment selected for the operation, making the interaction with mining engineers critical for optimum shape selection;
   - Shapes of excavations are critical when stress driven failure is expected, suggesting low tunnel heights in high vertical and low tunnel widths in high horizontal stress fields to limit the effect of the stress driven failure;
   - In the absence of high stress levels, the shape of tunnels are affected by the exposure of geological structures and should be handled based on the expected failure mechanisms and required support practices to prevent the failures;
   - In some cases, some excavation shapes are affected by the position of the tunnels. When an ore drive is developed along the hangingwall contact of a flat dipping orebody, the use of "shanty

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back" shaped tunnels allow the tunnel roof to be carried along the orebody contact, creating a specific excavation shape. This would also be allowed if a dominating bedding plane or joint set is intersected byt eh tunnel.

3. Orientation:

- Excavation orientation are in many instances in the massive mine layout not changeable.

LEARNING OUTCOME 4.4.1.3

DESCRIBE, EXPLAIN AND DISCUSS THE ROCK ENGINEERING CRITERIA USED TO DETERMINE THE OPTIMUM LAYOUT, POSITIONING, ORIENTATION AND SPACING OF SERVICE TUNNELS RELATIVE TO:

- Stopes, pillars
- Geological features
- Other excavations
- Apply these criteria to design service tunnel layouts.

CONNECTION 39

Refer to "LEARNING OUTCOME 3.2.2.4" on page 107

Stopes, pillars

CONNECTION 40

Refer to Paper 3.1, Outcome 4.4.1.2

Geological features

CONNECTION 41

Refer to Paper 3.1, Outcome 4.4.1

Other excavations

4.4.2. DRAWPOINT AND LOADING LAYOUTS

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

LEARNING OUTCOME 4.4.2.1

CONNECTION 42

Refer to "LEARNING OUTCOME 3.2.2.2" on page 98

DESCRIBE, EXPLAIN AND DISCUSS THE ROCK ENGINEERING CRITERIA USED TO DETERMINE THE OPTIMUM LAYOUT, POSITIONING, ORIENTATION AND SPACING OF DRAWPOINTS AND LOADING BAYS RELATIVE TO:

- Stopes, pillars
- Geological features

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- Other excavations

A drawpoint is defined as a place where ore can be loaded and removed, located beneath the stoping area and where gravity transfers the ore to the loading place
For the purpose of these Outcomes, a loading bay / loading tunnel / loading crosscut is viewed as a drive towards the orebody where ore is loaded, similar to the definition provided for drawpoints.

- Underground mining methods Hamrin

LEARNING OUTCOME 4.4.2.2

CONNECTION 43

Refer to "LEARNING OUTCOME 3.2.2.2" on page 98

APPLY THESE CRITERIA TO DESIGN DRAWPOINT, LOADING BAY AND LOADING TUNNEL LAYOUTS

A drawpoint is defined as a place where ore can be loaded and removed, located beneath the stoping area and where gravity transfers the ore to the loading place
For the purpose of these Outcomes, a loading bay / loading tunnel / loading crosscut is viewed as a drive towards the orebody where ore is loaded, similar to the definition provided for drawpoints.

- Underground mining methods Hamrin

LEARNING OUTCOME 4.4.2.3

DESCRIBE, EXPLAIN AND DISCUSS THE PRACTICAL ASPECTS OF DRAWPOINT AND LOADING BAY LAYOUTS AND SIZES WITH RESPECT TO ASPECT SUCH AS MACHINE SIZE, HAUL DISTANCE, ETC.

Drawpoint spacings reflect the spacing of the drawzones, but in the case of LHD layouts with a nominal drawpoint spacing of 15m, the drawzone spacing can vary across the major apex (pillar) from 18 to 24m, depending on the length of the drawbell (Figure 33). This occurs when the length of the drawpoint crosscut is increased to ensure that the LHD is straight before it loads.

In this case, optimum ore recovery is prejudiced by incorrect usage of equipment.

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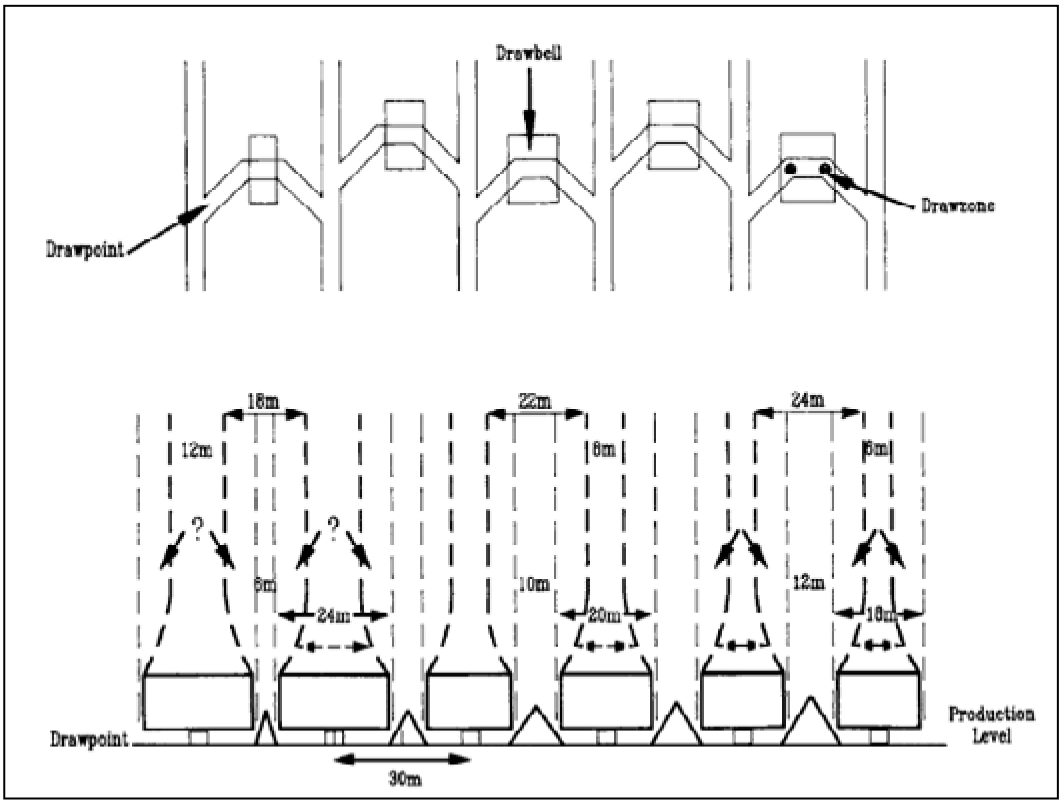

Figure 33: Drawzone spacing (Laubscher, 1994)

In LHD layouts, poor draw control (drawing of fine material at the expense of coarse material) is a major concern and it is suggested that blasting of coarse material is conducted in the same shift it reports to the drawpoint.

LHD layouts should allow the maximum amount of manoeuvring space while maintaining optimum drawzone spacings.
LHD equipment should be based on the required drawzone rather than just increasing the equipment size as the losses due to higher dilution exceed the lower operating costs associated with larger machines.

REFERENCE

- State of the art cave mining Laubscher
- Drawpoint design Castro
- Page 45, 59, Updated cave mining handbook Laubscher

CONNECTION 44
Refer to
Paper 3.1, Outcome 4.4.2

4.4.3. LARGE CHAMBER LAYOUTS

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

LEARNING OUTCOME 4.4.3.1

CONNECTION 45
Refer to
Paper 3.1, Outcome 4.4.2.1

DESCRIBE, EXPLAIN AND DISCUSS THE ROCK ENGINEERING CRITERIA USED TO DETERMINE THE OPTIMUM SIZE, SHAPE AND ORIENTATION OF LARGE SERVICE CHAMBERS

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LEARNING OUTCOME 4.4.3.2

CONNECTION 46
Refer to
Paper 3.1, Outcome 4.4.2.2

DESCRIBE, EXPLAIN AND DISCUSS THE ROCK ENGINEERING CRITERIA USED TO DETERMINE THE OPTIMUM EXCAVATION METHODOLOGY AND SEQUENCE FOR LARGE SERVICE CHAMBERS

LEARNING OUTCOME 4.4.3.3

CONNECTION 47
Refer to
Paper 3.1, Outcome 4.4.2.3

DESCRIBE, EXPLAIN AND DISCUSS THE ROCK ENGINEERING CRITERIA USED TO DETERMINE THE OPTIMUM SUPPORT METHODOLOGY AND SEQUENCE FOR LARGE SERVICE CHAMBERS

LEARNING OUTCOME 4.4.3.4

CONNECTION 48
Refer to
Paper 3.1, Outcome 4.4.2.4

DESCRIBE, EXPLAIN AND DISCUSS THE ROCK ENGINEERING CRITERIA USED TO DETERMINE THE OPTIMUM LAYOUT, POSITIONING, ORIENTATION AND SPACING OF LARGE SERVICE CHAMBERS RELATIVE TO:

- **Stopes, pillars**
- **Geological features**
- **Other excavations**

LEARNING OUTCOME 4.4.3.5

CONNECTION 49
Refer to
Paper 3.1, Outcome 4.4.2.5

APPLY THESE CRITERIA TO DESIGN LARGE SERVICE CHAMBERS LAYOUTS

4.4.4. ROCK PASS LAYOUTS

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

CONNECTION 50
Refer to
Paper 3.1, Outcome 4.4.3

LEARNING OUTCOME 4.4.4.1

CONNECTION 51
Refer to
Paper 3.1, Outcome 4.4.3.1

DESCRIBE, EXPLAIN AND DISCUSS THE CAUSES OF INSTABILITY IN ROCK PASSES

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LEARNING OUTCOME 4.4.4.2

CONNECTION 52

Refer to Paper 3.1, Outcome 4.4.3.2

DESCRIBE, EXPLAIN AND DISCUSS THE EFFECTS OF INSTABILITY IN ROCK PASSES

LEARNING OUTCOME 4.4.4.3

CONNECTION 53

Refer to Paper 3.1, Outcome 4.4.3.4

DESCRIBE, EXPLAIN AND DISCUSS THE OPTIONS AVAILABLE TO STABILISE ROCK PASSES

LEARNING OUTCOME 4.4.4.4

CONNECTION 54

Refer to Paper 3.1, Outcome 4.4.3.5

DESCRIBE, EXPLAIN AND DISCUSS THE OPTIONS AVAILABLE TO SUPPORT ROCK PASSES

LEARNING OUTCOME 4.4.4.5

CONNECTION 55

Refer to Paper 3.1, Outcome 4.4.3.6

DETERMINE APPROPRIATE ROCK PASS LAYOUT, ORIENTATION AND SUPPORT MEASURES USING ROCKMASS AND STRESS CHARACTERISATION TECHNIQUES

LEARNING OUTCOME 4.4.4.6

CONNECTION 56

Refer to Paper 3.1, Outcome 4.4.3.7 and Outcome 4.4.3.8

ASSESS THE SUITABILITY OF ROCK PASS LAYOUT, ORIENTATION AND SUPPORT MEASURES FOR GIVEN SETS OF ROCKMASS AND STRESS CONDITIONS

LEARNING OUTCOME 4.4.4.7

CONNECTION 57

Refer to Paper 3.1, Outcome 4.4.3.9

DESCRIBE, EXPLAIN AND DISCUSS THE ROCK ENGINEERING CRITERIA USED TO DETERMINE THE OPTIMUM SIZE, SHAPE AND ORIENTATION OF ROCK PASSES

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LEARNING OUTCOME 4.4.4.8

CONNECTION 58
Refer to
Paper 3.1, Outcome 4.4.3.10
and Outcome 4.4.3.11

DESCRIBE, EXPLAIN AND DISCUSS THE ROCK ENGINEERING CRITERIA USED TO DETERMINE THE OPTIMUM LAYOUT, POSITIONING AND SPACING OF ROCK PASSES RELATIVE TO:

- Stopes, pillars
- Geological features
- Each other

LEARNING OUTCOME 4.4.4.9

CONNECTION 59
Refer to
Paper 3.1, Outcome 4.4.3.11

APPLY THESE CRITERIA TO DESIGN ROCK PASSES

LEARNING OUTCOME 4.4.4.10

CONNECTION 60
Refer to
Paper 3.1, Outcome 4.4.3.12

DESCRIBE, EXPLAIN AND DISCUSS THE ROCK ENGINEERING CRITERIA USED TO REHABILITATE DAMAGED ROCK PASSES

CONNECTION 61
Refer to
Paper 2, Outcome 4.2.2
Paper 3.1, Outcome 4.4.4

4.5. SHAFT LAYOUTS
4.5.1. SHAFT PROTECTION PILLARS

LEARNING OUTCOME 4.5.1.1

THE CANDIDATE MUST BE ABLE TO DEMONSTRATE KNOWLEDGE AND UNDERSTANDING OF THE ABOVE SUBJECT AREA BY BEING ABLE TO:

LEARNING OUTCOME 4.5.1.2

CONNECTION 62
Refer to
Paper 3.1, Outcome 4.4.4.1

SKETCH, DESCRIBE, EXPLAIN AND DISCUSS THE ROCK ENGINEERING CRITERIA USED TO DESIGN INCLINE SHAFT PROTECTION PILLARS AT SHALLOW, INTERMEDIATE AND GREAT DEPTH

LEARNING OUTCOME 4.5.1.3

CONNECTION 63
Refer to
Paper 3.1, Outcome 4.4.4.2

SKETCH, DESCRIBE, EXPLAIN AND DISCUSS THE ROCK ENGINEERING CRITERIA USED TO DESIGN VERTICAL SHAFT PROTECTION PILLARS AT SHALLOW, INTERMEDIATE AND GREAT DEPTH

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LEARNING OUTCOME 4.5.1.4

CONNECTION 64
Refer to
Paper 3.1, Outcome 4.4.4.3

CALCULATE STRESS, STRAIN AND TILT IN VERTICAL SHAFTS USING EQUATIONS FOR CIRCULAR SHAFT PILLARS IN AN ELASTIC MEDIUM

LEARNING OUTCOME 4.5.1.5

CONNECTION 65
Refer to
Paper 3.1, Outcome 4.4.4.4

APPLY THE RESULTS FROM THE ABOVE CALCULATIONS TO DESIGN APPROPRIATE SHAFT PILLAR DIMENSIONS

LEARNING OUTCOME 4.5.1.6

CONNECTION 66
Refer to
Paper 3.1, Outcome 4.4.4.5

DESIGN SHAFT LAYOUTS FOR GIVEN DEPTH, GEOLOGY AND ROCKMASS CONDITIONS

LEARNING OUTCOME 4.5.1.7

CONNECTION 67
Refer to
Paper 3.1, Outcome 4.4.4.6

EXPLAIN THE LIMITATIONS THAT CONCRETE LININGS AND STEELWORK PLACE ON ALLOWABLE MOVEMENTS IN SHAFTS.

4.5.2. LAYOUT DESIGN CRITERIA

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

LEARNING OUTCOME 4.5.2.1

CONNECTION 68
Refer to
Paper 3.1, Outcome 4.5
Paper 3.2, Outcome 4.4.2,
"LEARNING OUTCOME 3.1.1.9" on page 96
"LEARNING OUTCOME 3.2.2.4" on page 107
"LEARNING OUTCOME 2.3.1.6" on page 62, and
"LEARNING OUTCOME 2.3.1.8" on page 68

DESCRIBE, EXPLAIN AND DISCUSS THE DERIVATION AND APPLICATION OF LAYOUT AND SUPPORT DESIGN CRITERIA FOR MASSIVE MINING OPERATIONS AT SHALLOW, INTERMEDIATE AND GREAT DEPTH

LEARNING OUTCOME 4.5.2.2

CONNECTION 69
Refer to
Paper 3.1, Outcome 4.5
Paper 3.2, Outcome 4.4.2

DESCRIBE, EXPLAIN, DISCUSS AND APPLY LAYOUT DESIGN CRITERIA APPLICABLE TO ALL OF THE MASSIVE MINING SITUATIONS SET OUT ABOVE

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CHAPTER

5

PAPER 3.3: MASSIVE UNDERGROUND MINING (HARD AND SOFT ROCK)

LEARNING OUTCOMES

5. MINING SUPPORT STRATEGIES
5.1. STOPE SUPPORT STRATEGIES

5.1.1. ROCKWALL REINFORCEMENT

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

- Describe, explain and discuss the variation in the behaviour of rock surrounding stoping excavations at shallow, intermediate and great depth;
- Describe, explain and discuss how this affects the support requirements at shallow, intermediate and great depth;
- Evaluate and determine appropriate support requirements in given circumstances in terms of the following factors:
- Rock conditions, field stresses, geological features;
- Describe, explain and discuss appropriate support strategies to control stope hangingwall under static conditions at shallow, intermediate and great depth;
- Determine the required support reaction for static conditions given typical fall of ground dimensions and loading conditions;
- Describe, explain and discuss appropriate support strategies to control stope hangingwall under dynamic conditions at shallow, intermediate and great depth;
- Determine the required support reaction for dynamic (rockburst) conditions given typical fall of ground dimensions and loading conditions;
- Sketch, describe, explain and discuss the function, use and installation of cable bolting to stabilise open stopes;
- Sketch, describe, explain and discuss the function, use and installation of cable bolting to stabilise cut and fill stopes;
- Sketch, describe, explain and discuss the function, use and installation of cable bolting to stabilise pillars;
- Sketch, describe, explain and discuss methods of installation of cable bolts in the above situations;
- Determine and design cable bolt layouts, strengths and performance characteristics for given open stoping situations;
- Determine and design cable bolt layouts, strengths and performance characteristics for given cut and fill stoping situations;

- Determine and design cable bolt layouts, strengths and performance characteristics for given pillar extraction situations;
- Evaluate rock conditions, field stresses and mining layout circumstances to determine bolting requirements for the above stoping situations;
- Describe, explain and discuss the special problems of supporting wide ore bodies at shallow, intermediate and great depths;
- Describe, explain and discuss appropriate strategies to ensure stability under the above conditions;
- Describe, explain and discuss the special problems of supporting multiple, closely-spaced ore bodies at shallow, intermediate and great depths; and
- Describe, explain and discuss appropriate strategies to ensure stability under the above conditions.

5.1.2. PILLARS AS STOPE SUPPORT

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

- Describe, explain and discuss the behavioural characteristics and applications of the following types of pillar:
- Crush pillars, yield pillars, rigid pillars,
- Crown pillars, rib pillars;
- Estimate the strength of hard rock pillars by using rock classification techniques and pillar strength equations;
- Apply Ryder and Ozbay’s procedure to derive the strength of hard rock pillars;
- Describe, explain and discuss the stress-strain behaviour of square pillars at various width-to-height ratios;
- Describe, explain and discuss the post-peak behaviour in the above situations;
- Describe, explain and discuss tributary area theory in respect of pillar design;
- Describe, explain and discuss the limitations of tributary area theory in respect of pillar design;
- Estimate pillar stresses using tributary theory;
- Describe, explain and discuss the effect of joints and other weaknesses on pillar strength;
- Describe, explain and discuss the role of loading system stiffness in the stability of pillar systems once pillars have exceeded their peak strength;
- Describe, explain and discuss the effect of loading system stiffness on the mode of failure of pillars;
- Describe, explain and discuss the mechanism of foundation failure of pillars;
- Describe, explain and discuss the factors that affect foundation failure of pillars;
- Describe, explain and discuss the factors that are taken into consideration when designing pillar support systems;
- Design in-stope pillar support layouts for the variety of massive mining methods;
- Describe, explain and discuss the functions and use of pillars as a means of ground control in steeply-dipping massive open stopes;
- Describe, explain and discuss the functions and use of pillars as a

means of ground control in shallow-dipping massive ore bodies;

- Describe, explain and discuss the functions and use of post-pillars as a means of ground control in cut and fill operations;
- Describe, explain and discuss how the stability of sill or rib pillars may be assessed; and
- Design appropriate pillar systems for given rockmass characteristics, stress environments and mining situations.

5.1.3. BACKFILL AS STOPE SUPPORT

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

- Describe, explain and discuss the backfill characteristics and types of backfill required for filling in the following mining situations:
- Cut and fill operations,
- Drift and fill operations,
- Vertical crater retreat operations,
- Shallow wide-reef operations;
- Describe, explain and discuss how backfill fits into the mining cycle in the above situations;
- Describe, explain and discuss how backfill assists in improving the stability of rockmasses;
- Determine the support resistance offered by backfill given the backfill performance characteristics;
- Evaluate given rockmass and mining situations and note the potential benefits of backfilling in these situations; and
- Evaluate given rockmass and mining situations and recommend appropriate backfill type and placement procedures.

5.2. SERVICE EXCAVATION SUPPORT STRATEGIES

5.2.1. TUNNEL SUPPORT STRATEGIES

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

- Sketch, describe, explain and discuss the various behaviour of rock around excavations at shallow, intermediate and great depth;
- Describe, explain and discuss how this affects tunnel support requirements at shallow, intermediate and great depth;
- Describe, explain and discuss the objective primary support in tunnels;
- Describe, explain and discuss the objective secondary support in tunnels;
- Describe, explain and discuss the concept of integrated support in tunnels;
- Describe, explain and discuss various support strategies for:
    - Return airways,
    - Roadways, haulages, crosscuts,
    - Travelling ways,
    - Winzes, raises,
    - Tunnel intersections,

- Tunnels traversing faults, tunnels traversing dykes,
- Undercuts, drawpoints,
- Gathering drives, drilling drives;

- Describe, explain and discuss the strategies to support tunnels under the following conditions:
- Low stress with joint controlled behaviour,
- High stress where stress fractures dominate stability,
- Seismic and rockburst conditions; and
- Determine appropriate support strategies for tunnels for given layouts, rockmass conditions, stress regimes and geological circumstances.

5.2.2. SUPPORT OF DRAWPOINTS

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

- Sketch, describe, explain and discuss support measures for drawpoints, loading bays and similar excavations at shallow, intermediate and great depth;
- Sketch, describe, explain and discuss support measures for drawpoints, loading bays and similar excavations where the stress-to-strength ratio of the rockmass approximates unity; and
- Sketch, describe, explain and discuss special support measures for high wear areas of drawpoints, loading bays and similar excavations.

5.2.3. SUPPORT OF ROCKPASSES

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

- Sketch, describe, explain and discuss support measures for rock passes and similar excavations at shallow, intermediate and great depth.

5.2.4. SUPPORT OF LARGE CHAMBERS

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

- Sketch, describe, explain and discuss the various behaviours of rock around large excavations at shallow, intermediate and great depth;
- Describe, explain and discuss how this affects large excavation support requirements at shallow, intermediate and great depth;
- Describe, explain and discuss rockmass classification schemes for the determination of support strategies for large chambers;
- Describe, explain and discuss the application of the rockwall condition factor methodology to design support for large excavations at depth;
- Apply the rockwall condition factor (RCF) methodology to design support for large excavations at depth;
- Describe, explain and discuss the effects of excavation sequence on support installation;
- Determine appropriate excavation sequences to facilitate support installation;

- Describe, explain and discuss the effects of excavation sequence on support effectiveness;
- Determine appropriate excavation sequences to facilitate support effectiveness;
- Describe, explain and discuss the rules of thumb to determine the length and spacing of tendons in large excavations;
- Apply these rules of thumb to the design of support in large excavations;
- Describe, explain and discuss the effects of overstoping and understoping on the stability and support requirements of off-reef excavations; and
- Determine appropriate support strategies for large chambers for given mining and shaft layouts, rockmass conditions, stress regimes and geological circumstances.

5.2.5. SUPPORT OF SHAFTS

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

- Describe, explain and discuss rockmass classification schemes for the determination of support strategies for shafts;
- Determine appropriate support strategies for vertical shafts for given mining and shaft layouts, rockmass conditions, stress regimes and geological circumstances;
- Determine appropriate support strategies for inclined shafts for given mining and shaft layouts, rockmass conditions, stress regimes and geological circumstances; and
- Determine appropriate support strategies for roadways for given mining and shaft layouts, rockmass conditions, stress regimes and geological circumstances.

5.3. SUPPORT DESIGN CRITERIA

5.3.1. STOPE SUPPORT DESIGN CRITERIA

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

- Describe, explain and discuss stope support requirements in terms of:
- Initial stiffness, yieldability,
- Areal coverage,
- Support resistance, energy absorption; and
- Describe, explain and discuss how support resistance varies for static and dynamic conditions.

5.3.2. TUNNEL, CHAMBER AND SHAFT SUPPORT DESIGN CRITERIA

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

- Describe, explain and discuss excavation support and rockwall reinforcement requirements in terms of:
- Initial stiffness, yieldability,
- Unit length, areal coverage,
- Support resistance, energy absorption;

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- Describe, explain and discuss how support resistance varies for static and dynamic conditions;
- Calculate required support resistances for static conditions;
- Calculate required support resistances for dynamic (rockburst) conditions;
- Calculate support loads for support systems made up of various units;
- Calculate energy absorption for support systems made up of various units; and
- Describe, explain and discuss support requirements for time-dependent (creep) conditions.

5.3.3. SUPPORT AND SUPPORT SYSTEM TYPES AND CHARACTERISTICS

- The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:
- Apply rockmass classification systems to select appropriate support for stopes, tunnels and chambers in massive mines;
- Apply the rock condition factor (RCF) to select appropriate support for tunnels in high stresses;
- Describe and discuss the following rockwall support types:
- Mechanically anchored bolts, cable bolts, friction bolts (split sets),
- Cement grouted bolts, resin bonded bolts,
- Full-column grouted/bonded bolts,
- Yielding bolts (cone bolts),
- Multiple rod bolts, multiple strand cable anchors,
- Pre-stressed tendons,
- Steel arches, massive concrete linings,
- Shotcrete, gunite, thin sprayed linings,
- Wire mesh, rope lacing, tendon straps;
- Characterise the following aspects of the above units:
- Their principles of operation,
- Their technical specifications,
- Their load-deformation characteristics,
- The methods of ensuring support unit quality,
- Their installation/application procedures,
- The methods of ensuring their installed quality; and
- Design and evaluate the use of appropriate support units, support systems, support patterns and installation procedures for given rockmass conditions, stress regimes and mining layouts.

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5. MINING SUPPORT STRATEGIES
5.1. STOPE SUPPORT STRATEGIES

5.1.1. ROCKWALL REINFORCEMENT

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

CONNECTION 1
Refer to
Paper 2, Outcome 5
Paper 3.1, Outcome 5.1, 5.4.4.4, 4.4.4.5

LEARNING OUTCOME 5.1.1.1

DESCRIBE, EXPLAIN AND DISCUSS THE VARIATION IN THE BEHAVIOUR OF ROCK SURROUNDING STOPING EXCAVATIONS AT SHALLOW, INTERMEDIATE AND GREAT DEPTH

CONNECTION 2
Refer to
Paper 3.1, Outcome 5.1.1

LEARNING OUTCOME 5.1.1.2

DESCRIBE, EXPLAIN AND DISCUSS HOW THIS AFFECTS THE SUPPORT REQUIREMENTS AT SHALLOW, INTERMEDIATE AND GREAT DEPTH

CONNECTION 3
Refer to
Paper 3.1, Outcome 5.1.2

LEARNING OUTCOME 5.1.1.3

EVALUATE AND DETERMINE APPROPRIATE SUPPORT REQUIREMENTS IN GIVEN CIRCUMSTANCES IN TERMS OF THE FOLLOWING FACTORS:

- Rock conditions,
- field stresses,
- geological features

CONNECTION 4
Refer to
Paper 3.1, Outcome 5.1.18

LEARNING OUTCOME 5.1.1.4

DESCRIBE, EXPLAIN AND DISCUSS APPROPRIATE SUPPORT STRATEGIES TO CONTROL STOPE HANGINGWALL UNDER STATIC CONDITIONS AT SHALLOW, INTERMEDIATE AND GREAT DEPTH

CONNECTION 5
Refer to
Paper 3.1, Outcome 5.1.7

LEARNING OUTCOME 5.1.1.5

DETERMINE THE REQUIRED SUPPORT REACTION FOR STATIC CONDITIONS GIVEN TYPICAL FALL OF GROUND DIMENSIONS AND LOADING CONDITIONS

CONNECTION 6
Refer to
Paper 3.1, Outcome 5.1.16

LEARNING OUTCOME 5.1.1.6

DESCRIBE, EXPLAIN AND DISCUSS APPROPRIATE SUPPORT STRATEGIES TO CONTROL STOPE HANGINGWALL UNDER DYNAMIC CONDITIONS AT SHALLOW, INTERMEDIATE AND GREAT DEPTH

CONNECTION 7
Refer to
Paper 3.1, Outcome 5.1.8

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LEARNING OUTCOME 5.1.1.7

CONNECTION 8

Refer to Paper 3.1, Outcomes 5.1.16 and 5.1.17

DETERMINE THE REQUIRED SUPPORT REACTION FOR DYNAMIC (ROCKBURST) CONDITIONS GIVEN TYPICAL FALL OF GROUND DIMENSIONS AND LOADING CONDITIONS

LEARNING OUTCOME 5.1.1.8

CONNECTION 9

Paper 2, Outcome 5.1.7
Paper 3.1, Outcome 5.4.4

SKETCH, DESCRIBE, EXPLAIN AND DISCUSS THE FUNCTION, USE AND INSTALLATION OF CABLE BOLTING TO STABILISE OPEN STOPES

Investigations have demonstrated the performance and benefit of cable reinforcement of stope boundaries. The relative effectiveness of various stope wall reinforcement designs was analysed through numerical modelling, as explained in Figure 1:
for a vertical open stope (Figure 1b);
and various cable bolt patterns (Figure 1c, d, e), where a constant 150m of tendon was included into each reinforcement pattern for option c, while option e required 200m of reinforcement;
the impact of the various patterns is indicated by the control exercised at the hangingwall surface by the reinforcement and is depicted by the deflection.

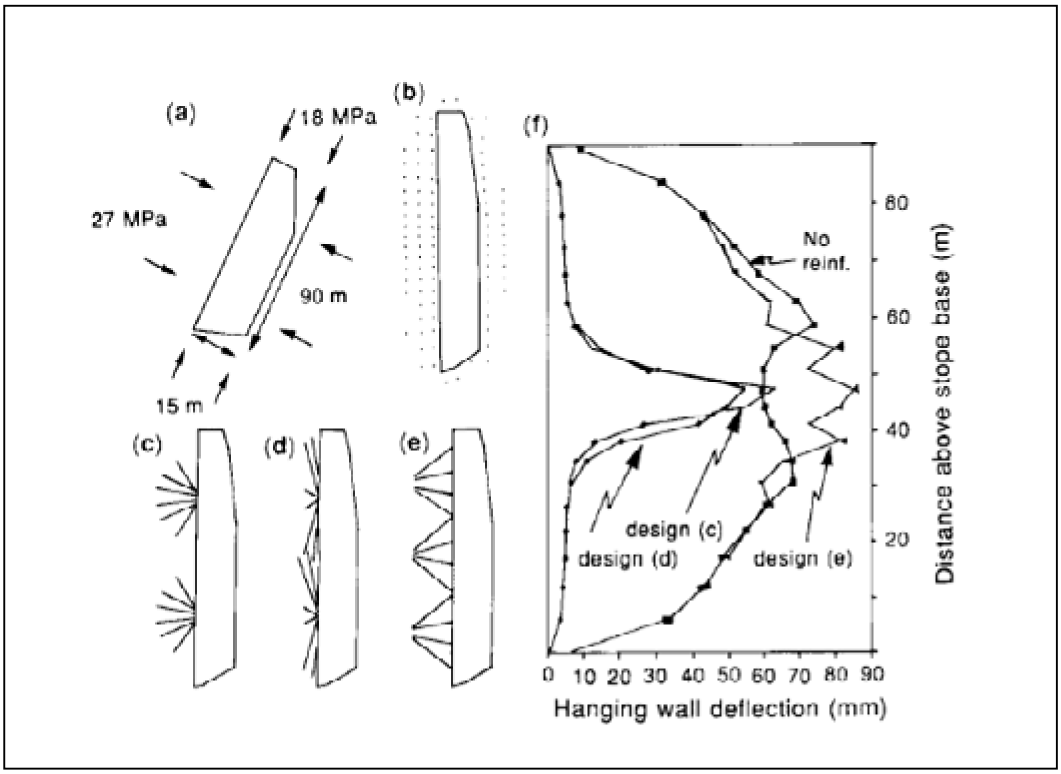

Figure 1: Impact of cable bolting on hangingwall deflections (Brady and Brown, 2006)

Results:
For the 'no reinforcement' hangingwall, the wall rock was de-stressed; the displacement distribution indicates wall failure (Note: The uneven shape of the wall deflection curve reflects the five-stage extraction sequence);
For options c and d: The hangingwall deflections are similar with substantial control of hangingwall displacement towards the edges of the

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stope, but possible instability near the middle of the stope (Note: this is consistent with the observed field performance);
For option e: The wall deflection plot for the uniform distribution of reinforcement indicates that this pattern is ineffective in controlling hangingwall displacements.

Conclusions:
The relatively uniform distribution of 'sparse' reinforcement is ineffective in achieving stope wall control (option e);
where reinforcement is necessarily sparse on average, concentration into appropriately located zones may enhance its impact on ground control (options c, d)

REFERENCE

- Page 428, Brady and Brown, Rock mechanics for underground mines, 2006
- Page 206-276, Cable bolting in underground mines Hutchinson

LEARNING OUTCOME 5.1.1.9

CONNECTION 10
Paper 2, Outcome 5.1.7
Paper 3.1, Outcome 5.4.4

SKETCH, DESCRIBE, EXPLAIN AND DISCUSS THE FUNCTION, USE AND INSTALLATION OF CABLE BOLTING TO STABILISE CUT AND FILL STOPES

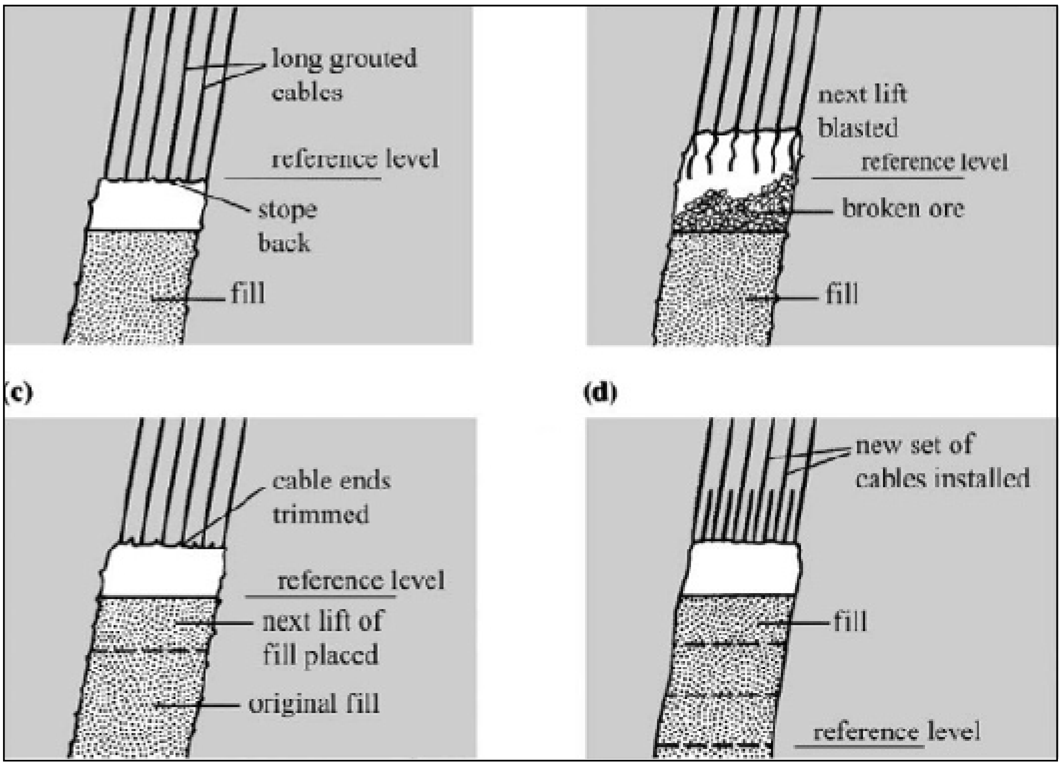

Figure 2: Crown support installation (Brady & Brown, 2006)

In cut and fill mining, stope stability is maintained by small open spans and the use of backfill, placed after each cut, has been made. This mining method requires personnel entrance into the production areas, and support installation to ensure the safety of personnel is therefore essential.
In general, two cut-and-fill methods exist, i.e. underhand or overhand (Figure 3), each of which requires a different approach to support installation:
Overhand:

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The ore body is cut and backfill is placed in an 'upwards' direction, resulting in the roof consisting of ore body material;
This material is of a certain quality and support is designed to address the condition of the exposed ore body;
Underhand:
The ore body is cut and backfill is placed in a 'downwards' direction, resulting in the roof consisting of backfill material;
This material has a certain strength and composition and support is designed to ensure that the fill column remains stable (Figure 4).

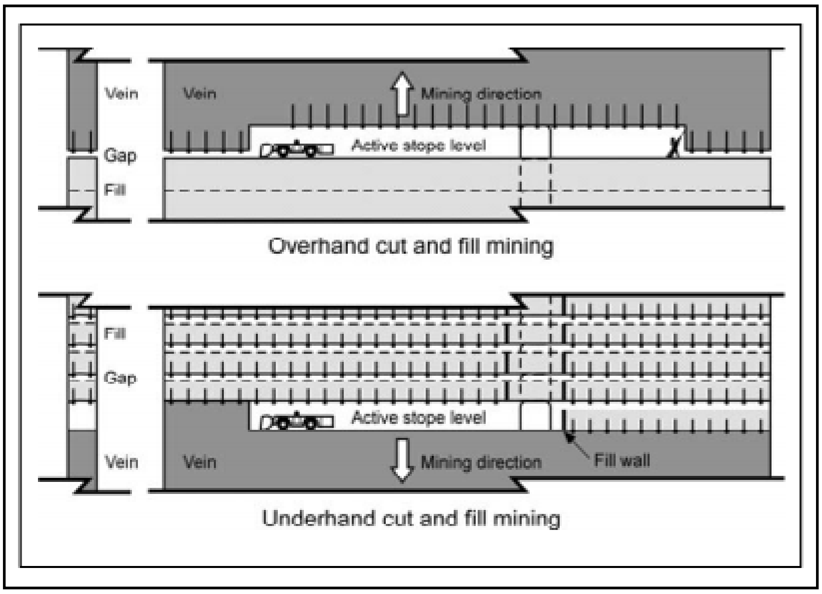

Figure 3: Over and underhand cut and fill stoping (Williams)

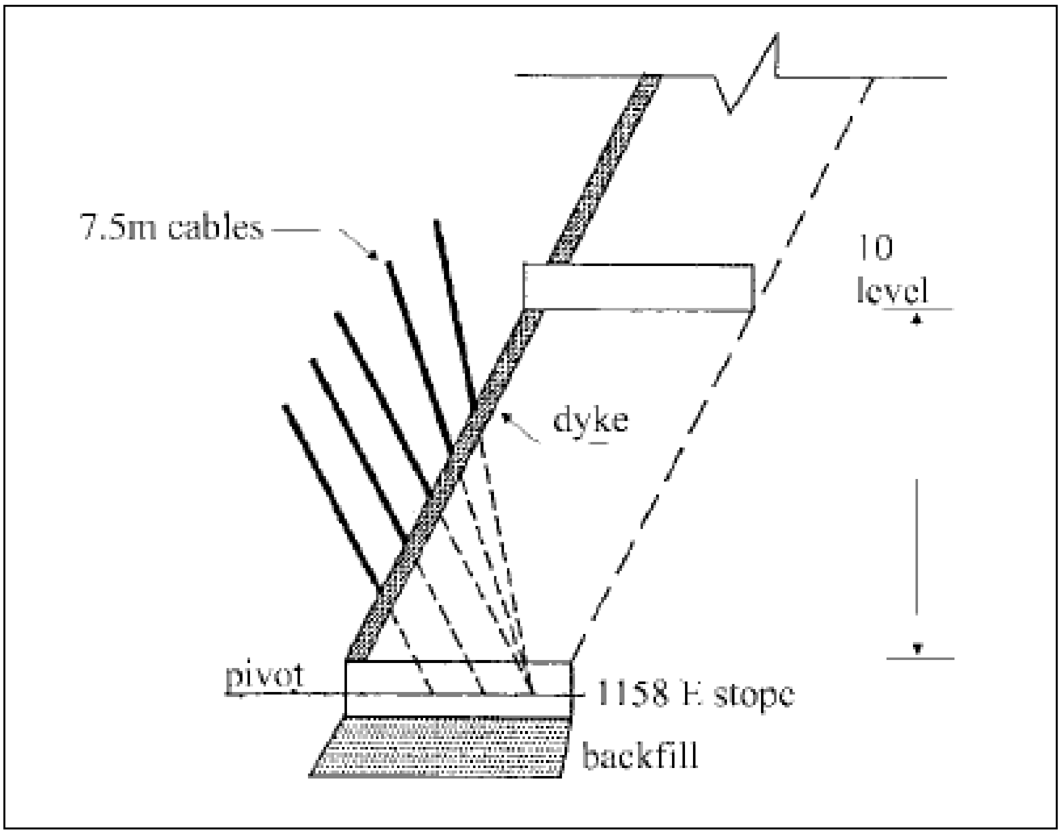

Figure 4: Support installation into backfill lift (Williams)

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REFERENCE

- Underhand cut and fill support Williams
- Page 206-276, Cable bolting in underground mines Hutchinson
- Page 324, Brady and Brown, Rock mechanics for underground mining, 2006

LEARNING OUTCOME 5.1.1.10

CONNECTION 11

Paper 2, Outcome 5.1.7
Paper 3.1, Outcome 5.4.4

SKETCH, DESCRIBE, EXPLAIN AND DISCUSS THE FUNCTION, USE AND INSTALLATION OF CABLE BOLTING TO STABILISE PILLARS

REFERENCE

- Page 206-276, Cable bolting in underground mines Hutchinson

LEARNING OUTCOME 5.1.1.11

CONNECTION 12

Paper 2, Outcome 5.1.7
Paper 3.1, Outcome 5.4.4

SKETCH, DESCRIBE, EXPLAIN AND DISCUSS METHODS OF INSTALLATION OF CABLE BOLTS IN THE ABOVE SITUATIONS

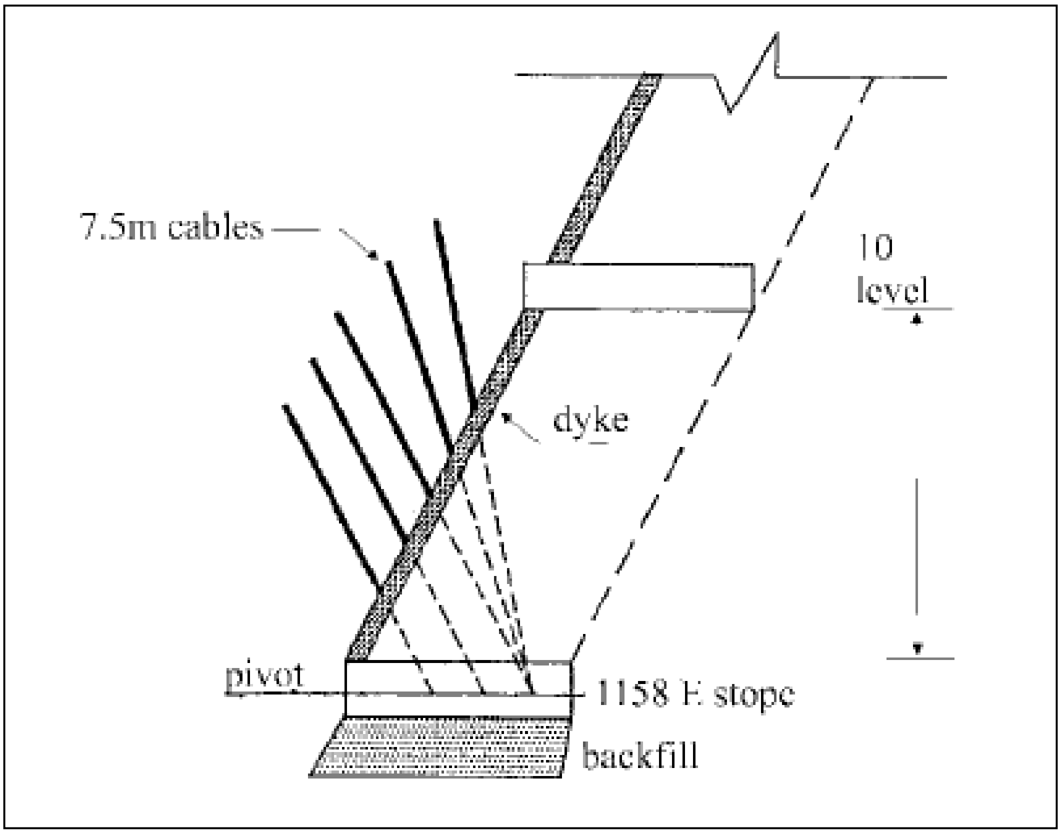

Figure 5: Support installation to support a dyke in hanngwall (Brady & Brown, 2006)

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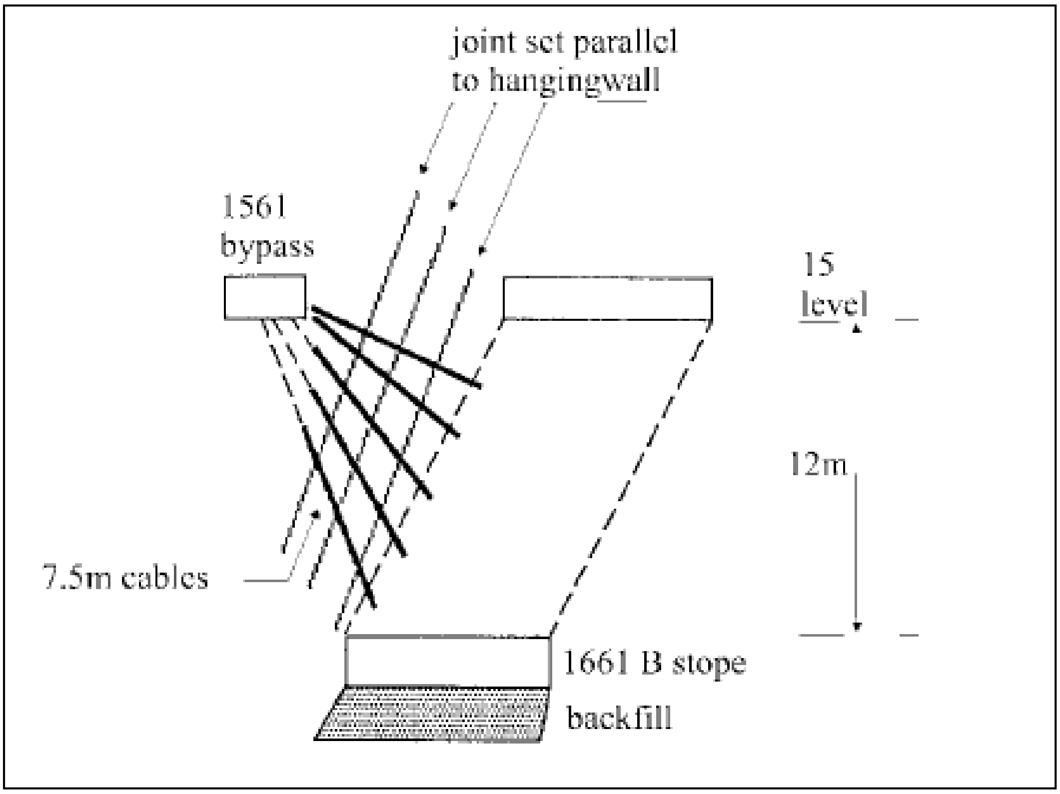

Figure 6: Support installation to support hangingwall parallel bedding / joints (Brady & Brown, 2006)

REFERENCE

- Page 1-19, 277-365, Cable bolting in underground mines Hutchinson
- Page 325, Brady and Brown, Rock mechanics for underground mining, 2006

LEARNING OUTCOME 5.1.1.12

CONNECTION 13

Paper 2, Outcome 5.1.7
Paper 3.1, Outcome 5.4.4

DETERMINE AND DESIGN CABLE BOLT LAYOUTS, STRENGTHS AND PERFORMANCE CHARACTERISTICS FOR GIVEN OPEN STOPING SITUATIONS

REFERENCE

- Page 23-175, and Page 206 - 276 Cable bolting in underground mines Hutchinson
- Example of empirical cable bolt design

  Refer to "LEARNING OUTCOME 3.1.1.9" on page 96
- Example of numerical modelling cable bolt design

  Refer to "LEARNING OUTCOME 3.1.1.9" on page 96

LEARNING OUTCOME 5.1.1.13

CONNECTION 14

Refer to ""LEARNING OUTCOME 5.1.1.12" on page 293

DETERMINE AND DESIGN CABLE BOLT LAYOUTS, STRENGTHS AND PERFORMANCE CHARACTERISTICS FOR GIVEN CUT AND FILL STOPING SITUATIONS

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LEARNING OUTCOME 5.1.1.14

CONNECTION 15

Refer to "LEARNING OUTCOME 5.1.1.12" on page 293

DETERMINE AND DESIGN CABLE BOLT LAYOUTS, STRENGTHS AND PERFORMANCE CHARACTERISTICS FOR GIVEN PILLAR EXTRACTION SITUATIONS

LEARNING OUTCOME 5.1.1.15

EVALUATE ROCK CONDITIONS, FIELD STRESSES AND MINING LAYOUT CIRCUMSTANCES TO DETERMINE BOLTING REQUIREMENTS FOR THE ABOVE STOPING SITUATIONS

REFERENCE

- Page 206-276, Cable bolting in underground mines Hutchinson

LEARNING OUTCOME 5.1.1.16

CONNECTION 16

Refer to Paper 3.1, Outcome 5.2

DESCRIBE, EXPLAIN AND DISCUSS THE SPECIAL PROBLEMS OF SUPPORTING WIDE ORE BODIES AT SHALLOW, INTERMEDIATE AND GREAT DEPTHS

NOTE

Support requirements for wide ore bodies are as discussed in Outcome 5.1.1 above.
Where the width of ore body drives increases due to the increase in ore body widths, the practices as applied to tunnels are applicable.

LEARNING OUTCOME 5.1.1.17

CONNECTION 17

Refer to Paper 3.1, Outcome 5.2

DESCRIBE, EXPLAIN AND DISCUSS APPROPRIATE STRATEGIES TO ENSURE STABILITY UNDER THE ABOVE CONDITIONS

NOTE

Support requirements for wide ore bodies are as discussed in Outcome 5.1.1 above.
Where the width of ore body drives increases due to the increase in ore body widths, the practices as applied to tunnels are applicable.

LEARNING OUTCOME 5.1.1.18

DESCRIBE, EXPLAIN AND DISCUSS THE SPECIAL PROBLEMS OF SUPPORTING MULTIPLE, CLOSELY-SPACED ORE BODIES AT SHALLOW, INTERMEDIATE AND GREAT DEPTHS

In a massive mining environment, closely-spaced ore bodies are mined as a unit, unless the resulting grade is found to be insufficient. In those cases, the orebody to be mined will be reduced until the extraction is feasible.

In limited cases, orebodies will be mined separately, leaving waste material between them intact; however, in these cases, the following is critical:

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CONNECTION 18
Refer to the example in "LEARNING OUTCOME 3.2.3.1" on page 121

- the placement of backfill is required in the extracted stopes before the next orebody is mined;
- dilution levels can become excessive if the waste middlings are unstable;
- Support installation into the middling between orebodies, depending on the actual middling, must be performed before the extraction of the next orebody;

Extraction sequence:
from the footwall side towards the hangingwall side of the orebody:

1. ensure that waste middlings are situated in the floor where the impact of gravity-induced failures is lower;
2. stope extraction is situated 'above' the previously extracted and filled stope, improving roof control in second stope;
3. access to the back ore bodies is problematic and has to be created through the backfilled stopes or from the hangingwall side of the ore bodies;

from the hangingwall side towards the footwall side of the the ore body:

1. access to the successive stopes is maintained from the normal footwall access tunnels;
2. the waste between ore bodies must be supported to maintain stability with second stope as gravity assist instability of the roof.

NOTE

Support requirements for wide ore bodies are discussed in Outcome 5.1.1 above and are applicable if all ore bodies are mined as a whole.

LEARNING OUTCOME 5.1.1.19

CONNECTION 19
Refer to "Outcome 1.1" on page 288

DESCRIBE, EXPLAIN AND DISCUSS APPROPRIATE STRATEGIES TO ENSURE STABILITY UNDER THE ABOVE CONDITIONS

5.1.2. PILLARS AS STOPE SUPPORT

CONNECTION 20
Refer to
Paper 3.1 Outcome 5.4.1
Paper 3.2, Outcome 3.1; 4.1.1 ; 5.1.1 ; 5.1.2

THE CANDIDATE MUST BE ABLE TO DEMONSTRATE KNOWLEDGE AND UNDERSTANDING OF THE ABOVE SUBJECT AREA BY BEING ABLE TO:

LEARNING OUTCOME 5.1.2.1

CONNECTION 21
Refer to
Paper 3.1 Outcome 5.4.1
Outcomes 4.2.1 and 4.2.2

DESCRIBE, EXPLAIN AND DISCUSS THE BEHAVIOURAL CHARACTERISTICS AND APPLICATIONS OF THE FOLLOWING TYPES OF PILLAR:

- Crush pillars,
- yield pillars,
- rigid pillars,

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- Crown pillars,
- rib pillars

LEARNING OUTCOME 5.1.2.2

CONNECTION 22

Refer to
Paper 3.1 Outcome
5.4.1.3 and 5.4.1.4
Outcomes 4.2.1 and 4.2.2

ESTIMATE THE STRENGTH OF HARD ROCK PILLARS BY USING ROCK CLASSIFICATION TECHNIQUES AND PILLAR STRENGTH EQUATIONS

LEARNING OUTCOME 5.1.2.3

CONNECTION 23

Refer to
Paper 3.1 Outcome
5.4.1.3 and 5.4.1.4
Outcomes 4.2.1 and 4.2.2

APPLY RYDER AND OZBAY'S PROCEDURE TO DERIVE THE STRENGTH OF HARD ROCK PILLARS

This method is based on the Hedley and Grant hard rock pillar strength formula, as discussed in the Outcomes listed below.

LEARNING OUTCOME 5.1.2.4

CONNECTION 24

Refer to
Paper 3.2, Outcome 3.1.3

DESCRIBE, EXPLAIN AND DISCUSS THE STRESS-STRAIN BEHAVIOUR OF SQUARE PILLARS AT VARIOUS WIDTH-TO-HEIGHT RATIOS

LEARNING OUTCOME 5.1.2.5

CONNECTION 25

Refer to
Paper 3.1 Outcome 5.4.1.3

DESCRIBE, EXPLAIN AND DISCUSS THE POST-PEAK BEHAVIOUR IN THE ABOVE SITUATIONS

Pillar failure is the process of fracture creation within the pillar, possible scaling of the fractured material and the subsequent reduction in ability of the remaining parts of the pillar to sustain high levels of stress while allowing the pillar to permanently deform. This failure initllay occurs along the edges of the pillar and then slowly develops towards the pillar centre (Figure 7).

REFERENCE

- Page 374, 395, Brady and Brown, Rock mechanics of underground mines, 2006

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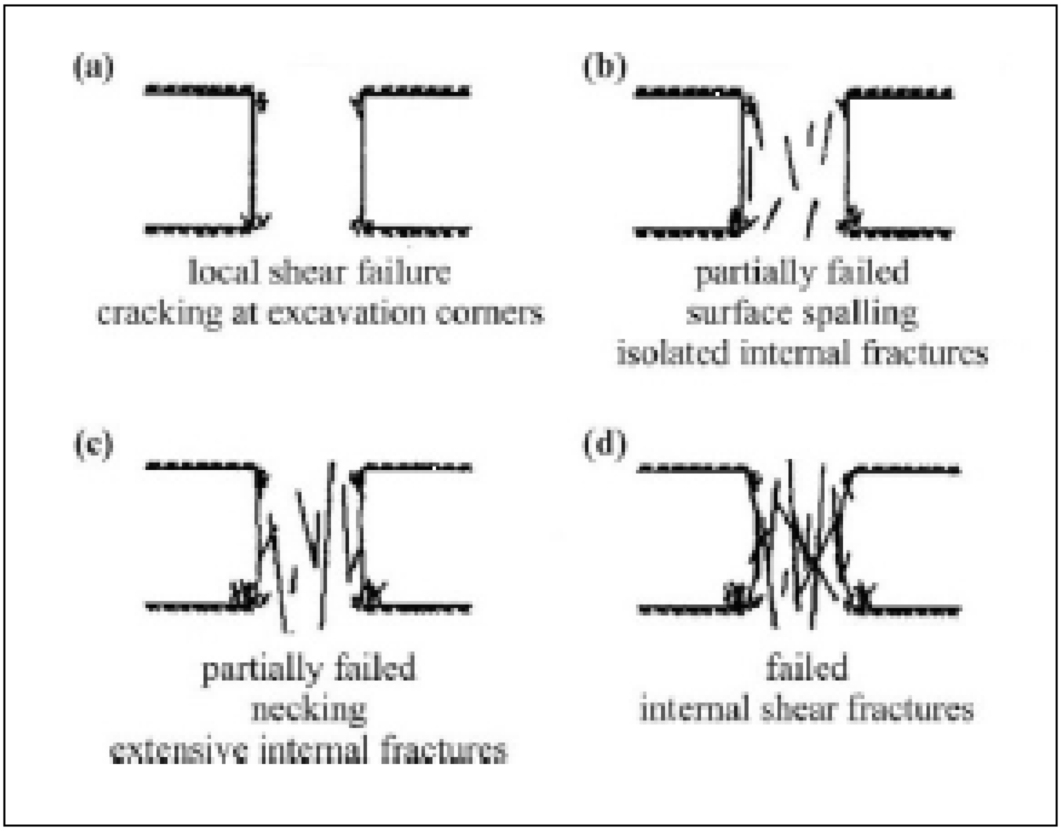

Figure 7: Pillar failure mechanism (Brady & Brown, 2006)

The apperance of a pillar undergoing failure is expressed in Figure 8 (Brady & Brown, 2006) and provides typical pillar shapes as failure progresses and as recorded at a specific mine.

Pillar Rating	Pillar condition	Appearance
1.	No indication of stress induced fracturing. Intact pillar.	
2.	Spalling on pillar corners, minor spalling of pillar walls. Fractures oriented sub-parallel to walls and are short relative to pillar height.	
3.	Increased corner spalling. Fractures on pillar walls more numerous and continuous. Fractures oriented sub-parallel to pillar walls and lengths are less than pillars height.	
4.	Continuous, sub-parallel, open fractures along pillar walls. Early development of diagonal fractures (start of hour glassing). Fracture lengths are greater than half of pillar height.	
5.	Continuous, sub-parallel, open fractures along pillar walls. Well-developed diagonal fractures (classic hour glassing). Fracture lengths are greater than half the pillar height.	
6.	Failed pillar may have minimal residual load carrying capacity and be providing local support to the slope back. Extreme hour-glassed shape or major blocks fallen out.	

Figure 8: Pillar failure progress (Brady & Brown, 2006)

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NOTE

Where factor of safety expresses the potential for pillar failure , the pillar width:height ratio expresses the behaviour of the pillar. High width:height ratios will exhibit ductile pillar behaviour (straining without losing its ability to sustain load), while low width:height ratios will allow sudden, brittle failures to occur.

LEARNING OUTCOME 5.1.2.6

CONNECTION 26

Refer to
Paper 3.2, Outcome 5.1.1.8

DESCRIBE, EXPLAIN AND DISCUSS TRIBUTARY AREA THEORY IN RESPECT OF PILLAR DESIGN

LEARNING OUTCOME 5.1.2.7

CONNECTION 27

Refer to
Paper 3.2, Outcome 5.1.1.9

DESCRIBE, EXPLAIN AND DISCUSS THE LIMITATIONS OF TRIBUTARY AREA THEORY IN RESPECT OF PILLAR DESIGN

LEARNING OUTCOME 5.1.2.8

CONNECTION 28

Refer to
Paper 3.2, Outcome 5.1.1.9

ESTIMATE PILLAR STRESSES USING TRIBUTARY THEORY

LEARNING OUTCOME 5.1.2.9

CONNECTION 29

Refer to
Paper 3.2, Outcome 3.1.1.1

DESCRIBE, EXPLAIN AND DISCUSS THE EFFECT OF JOINTS AND OTHER WEAKNESSES ON PILLAR STRENGTH

Modes of pillar response may be related to the structural geology of the pillar:

- a pillar with a set of joints dipping at some angle (Figure 9a) will 'yield' if the angle between the joint's angle and the pillar principal plane (plane perpendicular to the pillar axis) exceeds their effective angle of friction;
- a pillar with foliation or schistosity parallel to the principal axis of loading will fail in a buckling mode (Figure 6b).

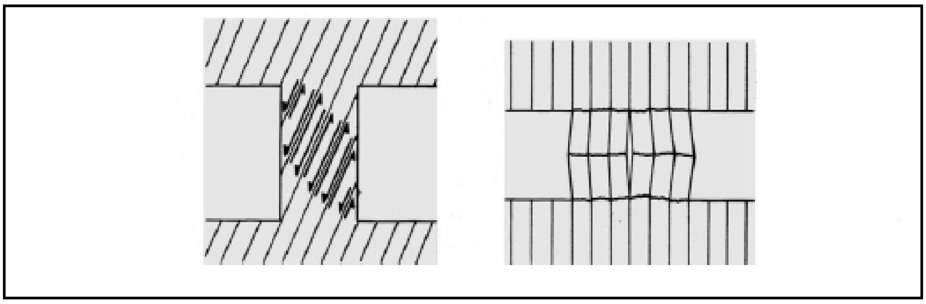

Figure 9: Impact of structures on pillar behaviour (Brady & Brown, 2006)

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REFERENCE

- Discontinuities and pillar strength Esterhuizen
- Page 374, Brady and Brown, Rock mechanics of underground mines, 2006

LEARNING OUTCOME 5.1.2.10

CONNECTION 30
Refer to
Paper 3.1 Outcome 5.4.1.5

DESCRIBE, EXPLAIN AND DISCUSS THE ROLE OF LOADING SYSTEM STIFFNESS IN THE STABILITY OF PILLAR SYSTEMS ONCE PILLARS HAVE EXCEEDED THEIR PEAK STRENGTH

LEARNING OUTCOME 5.1.2.11

CONNECTION 31
Refer to
Paper 3.1 Outcome 5.4.1.5

DESCRIBE, EXPLAIN AND DISCUSS THE EFFECT OF LOADING SYSTEM STIFFNESS ON THE MODE OF FAILURE OF PILLARS

LEARNING OUTCOME 5.1.2.12

CONNECTION 32
Refer to
Paper 3.1 Outcome 5.4.1.6

DESCRIBE, EXPLAIN AND DISCUSS THE MECHANISM OF FOUNDATION FAILURE OF PILLARS

When the orebody dip is such that loading of the orebody pillars occur between the orebody hangingwall and footwall material and where these materials are weak relative to the ore body rock, a post pillar may punch into the material that confines the pillar. This type of behaviour will be accompanied by floor heave next to the pillars or scaling and collapse of the roof around a pillar.

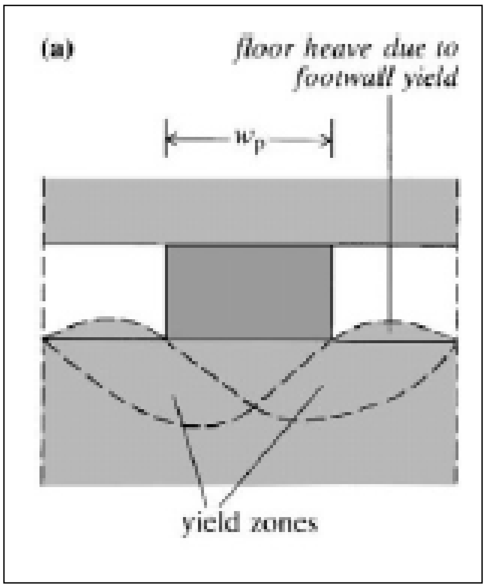

Figure 10: Pillar punching and footwall heave (Brady & Brown, 2006)

NOTE

The same can occur when the orebody pillar is loaded by orebody material only and where the pillar punches into locally weaker orebody in the floor.
The mode of failure is similar to the bearing capacity failure of a foundation.

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The load applied by a pillar to footwall or hangingwall rock in a stratiform ore body may be compared directly with a distributed load applied on the surface.

Useful methods of calculating the bearing capacity, q_b, of a cohesive, frictional material such as soft rock are provided by Brinch Hansen (1970).

NOTE

Bearing capacity is expressed in terms of stress.

For a uniform strip loading, bearing capacity is given by

$$q_b = 0.5.\ \gamma.w_p.N_\gamma + c.N_c$$

where
γ is the unit weight of the loaded medium
wp is the width of the pillar
c is the cohesion
and N_c and N_γ are bearing capacity factors.
The bearing capacity factors are given by
$N_c = (N_q - 1)\cot ø$
$N_\gamma = 1.5(N_q - 1)\tan ø$
where
ø is the angle of friction of the loaded medium, and
N_q is given by $N_q = e^{\pi \tan ø}.\tan^2[(\pi/4) + (ø/2)]$

The factor of safety against bearing capacity failure is given by

$F\ of\ S = q_b/\sigma_p$ (due to the assumptions made in this estimation, the factor of safety to prevent punching must be > 2.0)

Where
σ_p is the pillar stress level.

REFERENCE

- Page 390, Brady and Brown, Rock mechanics of underground mines, 2006

LEARNING OUTCOME 5.1.2.13

CONNECTION 33

Refer to
Paper 3.1 Outcome 5.4.1.7
and "LEARNING OUTCOME 5.1.1.12" on page 293

DESCRIBE, EXPLAIN AND DISCUSS THE FACTORS THAT AFFECT FOUNDATION FAILURE OF PILLARS

From the discussions in the listed outcomes, the following factors are suggested to affect the potential for foundation failure:
bearing capacity/strength against punching:
shear strength of the loaded material as given by the cohesion and friction angle of the material;
dimension of the area being loaded (pillar width);
pillar stress:
dimensions of the pillar (width, length and height);
overburden loading (depth, density, stiffness).

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LEARNING OUTCOME 5.1.2.14

CONNECTION 34

Refer to
Paper 3.1 Outcome 5.4.1.2

DESCRIBE, EXPLAIN AND DISCUSS THE FACTORS THAT ARE TAKEN INTO CONSIDERATION WHEN DESIGNING PILLAR SUPPORT SYSTEMS

LEARNING OUTCOME 5.1.2.15

CONNECTION 35

Refer to
“REGIONAL PILLARS” on page 239 and “CROWN PILLARS” on page 249

REFERENCE

DESIGN IN-STOPE PILLAR SUPPORT LAYOUTS FOR THE VARIETY OF MASSIVE MINING METHODS

- Geotechnical considerations Lipalile

LEARNING OUTCOME 5.1.2.16

CONNECTION 36

Refer to
“REGIONAL PILLARS” on page 239

DESCRIBE, EXPLAIN AND DISCUSS THE FUNCTIONS AND USE OF PILLARS AS A MEANS OF GROUND CONTROL IN STEEPLY-DIPPING MASSIVE OPEN STOPES

LEARNING OUTCOME 5.1.2.17

CONNECTION 37

Refer to
Outcome 4.2.1
Paper 3.1 Outcome 5.4.1.1
Paper 3.2, Outcome 3.1; 4.1.1 ; 5.1.1 ; 5.1.2

DESCRIBE, EXPLAIN AND DISCUSS THE FUNCTIONS AND USE OF PILLARS AS A MEANS OF GROUND CONTROL IN SHALLOW-DIPPING MASSIVE ORE BODIES

LEARNING OUTCOME 5.1.2.18

DESCRIBE, EXPLAIN AND DISCUSS THE FUNCTIONS AND USE OF POST-PILLARS AS A MEANS OF GROUND CONTROL IN CUT AND FILL OPERATIONS

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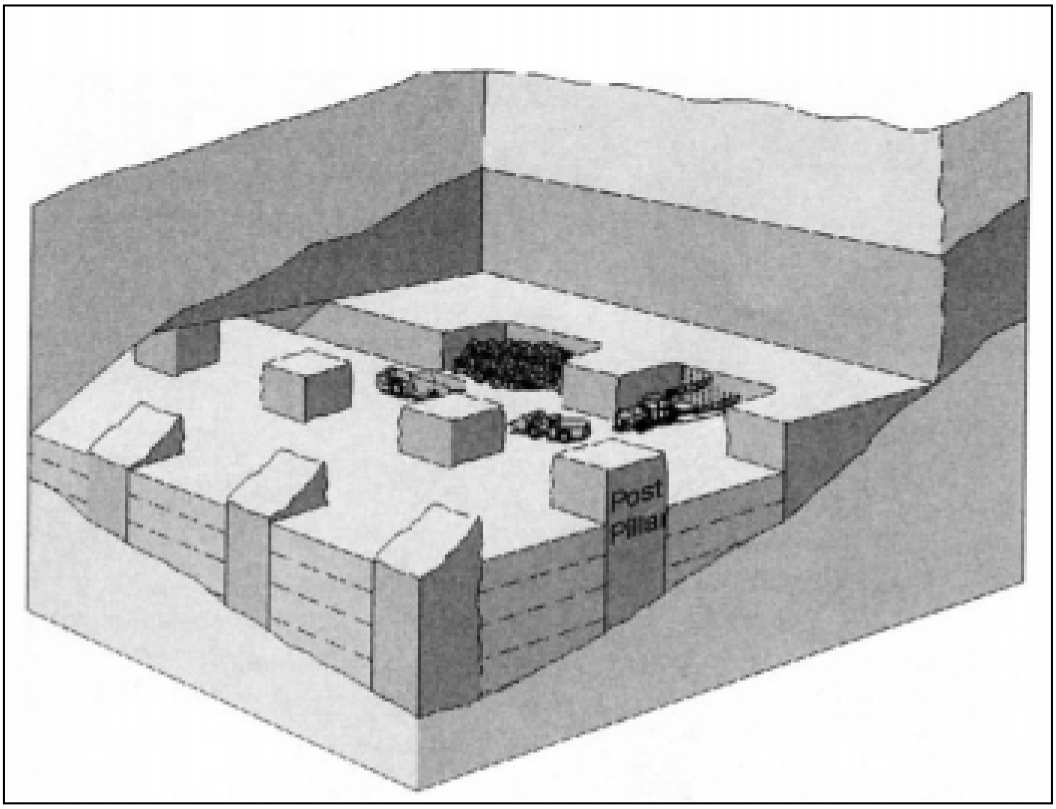

Figure 11: Post-pillar cut-and- fill mining on a room and pillar layout

Post-room and pillar mining is a combination of room-and-pillar and cut-and-fill stoping. With this method, ore is recovered in horizontal slices starting from the bottom and advancing upwards. Pillars are left inside the stope to support the roof, pillars are shaped as vertical beams across the ore body and continue through several layers of fill. Pillars and the fill contribute to the pillar's supporting ability. Mined-out stopes are backfilled and the next slice is mined by machines working from the fill surface.

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The use of post-pillars and backfill methods achieves the benefits from both methods and is usually applied when:
the ore body thickness is substantial;
the ore body material is not of sufficient quality to allow large unsupported or even large artificially supported spans;
backfill is available to confine and strengthen pillars, while changing their post-failure behaviour sufficiently to allow the taking of several cuts before the pillars become unstable.

This method means that the extraction can be executed in lifts similar to the cut-and fill method and where pillars can be utilised to maintain small spans. Pillars are left at the same position during each lift, finally creating pillars with very low width:height ratios (Figure 12 and 13), but pillars that are confined with backfill along their entire height.

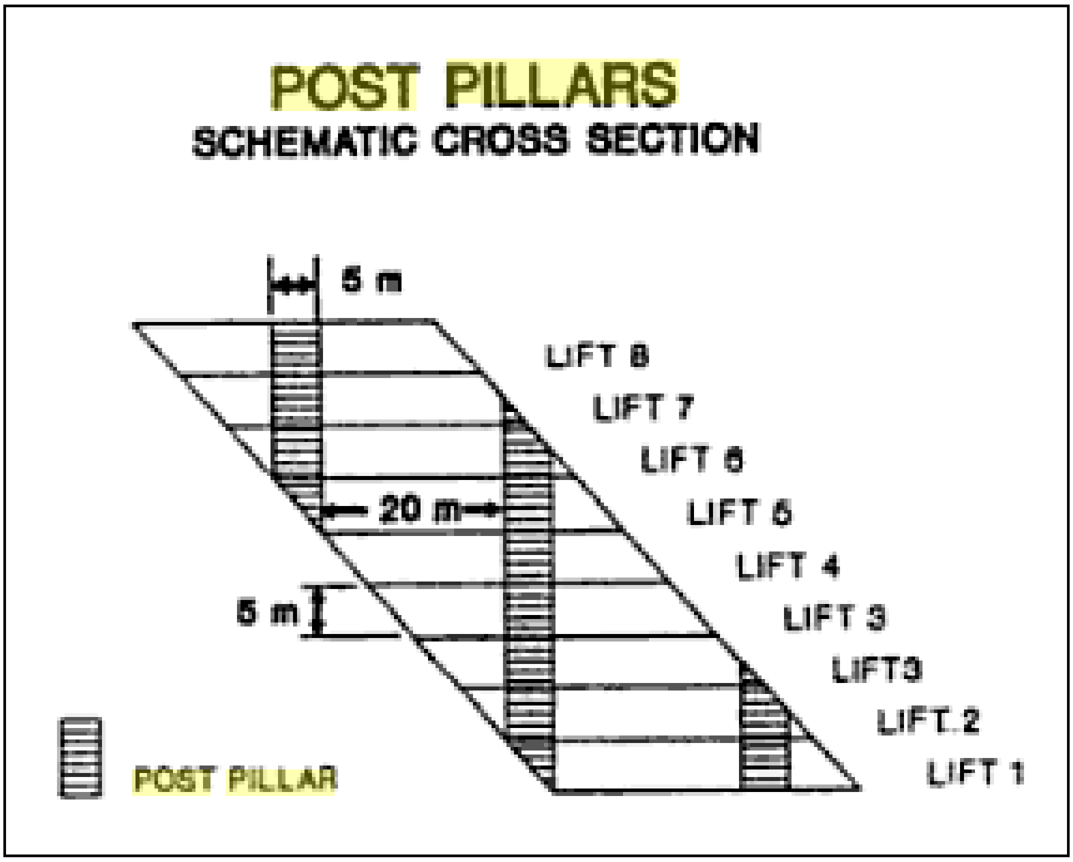

Figure 12: Post pillars left during eight lifts (Internet)

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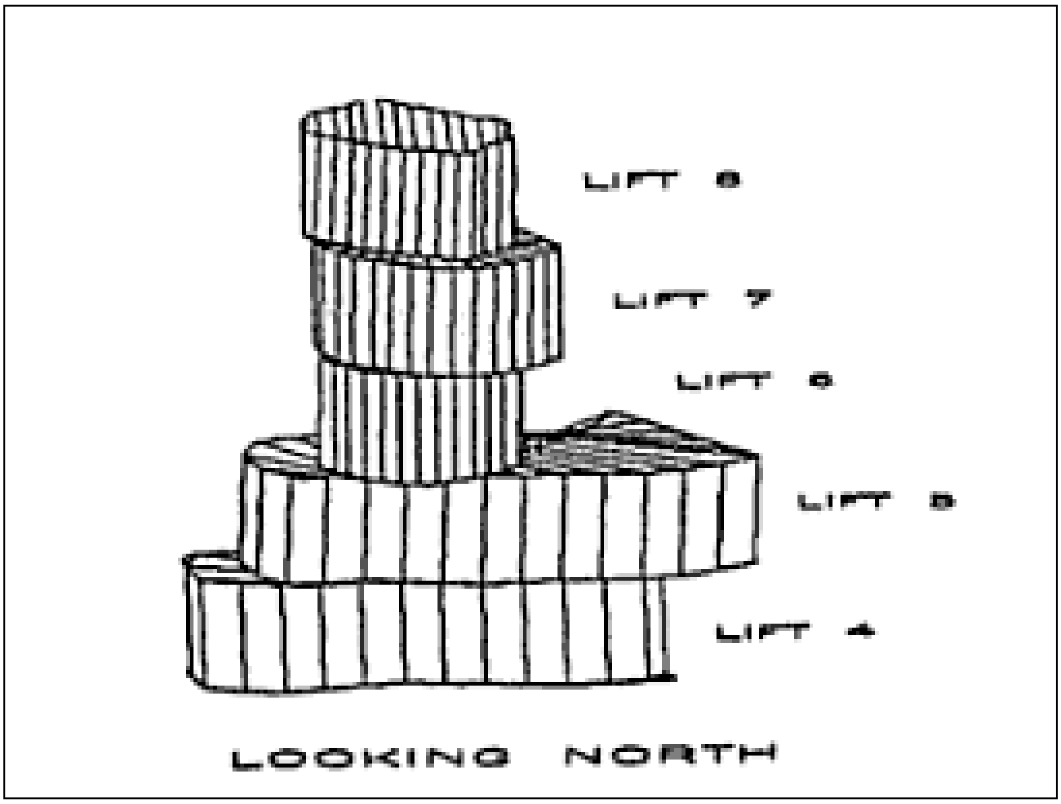

Figure 13: Isometric view of post-pillars (Internet)

Since the pillars left intact are initially of low height, they can be designed using the conventional pillar strength methods, but as the pillar height increases and backfill starts to confine the pillars, the estimation of pillar strengths using empirical methods seems to become ineffective and other approaches are required.

A number of sources suggest that numerical modelling becomes very appropriate to evaluate the pillar behaviour. In executing this modelling, the folowing is suggested:
pillar strength can be estimated using the rockmass strength (UCS, m and s parameters for the pillar material as defined by the Hoek-Brown failure criterion) allocated to the actual pillar dimensions in the model;
pillar stresses are predicted by using the appropriate software code and constitutive models for the overall rockmass;
backfill material is modelled as it is placed and its behaviour anticipated by using the appropriate backfill properties;
the impact of the loading system on pillar behaviour can be evaluated for irregular pillars, following the planned extraction and filling sequence, as the pillar dimensions change, but also as the confinement increases.

REFERENCE

- Post-pillar cut and fill Rangasamy

LEARNING OUTCOME 5.1.2.19

CONNECTION 38

Refer to "LEARNING OUTCOME 4.2.1.2" on page 239

DESCRIBE, EXPLAIN AND DISCUSS HOW THE STABILITY OF SILL OR RIB PILLARS MAY BE ASSESSED

LEARNING OUTCOME 5.1.2.20

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CONNECTION 39
Refer to
"LEARNING OUTCOME 4.2.1.2" on page 239

DESIGN APPROPRIATE PILLAR SYSTEMS FOR GIVEN ROCKMASS CHARACTERISTICS, STRESS ENVIRONMENTS AND MINING SITUATIONS

CONNECTION 40
Refer to
Paper 2, Outcome 5.3
Paper 3.1, Outcome 5.4.2
Paper 3.2, Outcome 5.5.2

5.1.3. BACKFILL AS STOPE SUPPORT

THE CANDIDATE MUST BE ABLE TO DEMONSTRATE KNOWLEDGE AND UNDERSTANDING OF THE ABOVE SUBJECT AREA BY BEING ABLE TO:

LEARNING OUTCOME 5.1.3.1

CONNECTION 41
Refer to
Paper 2, Outcome 5.3.3

DESCRIBE, EXPLAIN AND DISCUSS THE BACKFILL CHARACTERISTICS AND TYPES OF BACKFILL REQUIRED FOR FILLING IN THE FOLLOWING MINING SITUATIONS:

- Cut and fill operations
- Drift and fill operations
- Vertical crater retreat operations
- Shallow wide-reef operations

Cut and fill operations

1. Working platform:

Backfill material must be strong enough to be able to allow movement of equipment on top of fill. This can be achieved with a layer of cemented fill on top of any other fill material, but has also been achieved previously with a high density, uncemented fill;

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2. Dilution:

Fill must limit dilution during loading of the blasted ore. The use of a cemented fill will allow reduced dilution, but loading procedures with other fill materials can also achieve this goal.

3. Stabilise stope walls:

The fill must limit stope closure as the vertical height of the stope increases. The fill must have sufficient resistance against compression. This can be achieved with cement addition, but also with natural compaction of the fill under its own weight, increasing the placed density and consequently its load generation ability. Note that this is only possible if sufficient drainage of water from the backfill column is ensured if hydraulic fill is used. The use of rockfill is also sufficient to limit rockwall closure, provided a good fill of the open stope can be achieved.

4. Free-standing height:

If no pillars are left in the mine layout, the fill must be able to remain stable and not collapse into the next stope, requiring some strength that is only possible with the addition of cement. The amount of cement to be added is determined by the free-standing height required.

5. Undercutting previously filled stopes:

High strength is required to ensure that the fill remains stable in the hangingwall across the cut width when undercut. Paste fill allows the best control of fill quality and strength for this purpose, but in narrow ore bodies other types can be considered if artificial support (mesh, bolts, etc.) is also included in the design.

Drift and fill operations

1. Working platform:

Backfill material must be strong enough to be able to allow movement of equipment on top of fill. This can be achieved with a layer of cemented fill on top of any other fill material, but has also been achieved previously with a high density, uncemented fill;

2. Dilution:

Fill must limit dilution during loading of the blasted ore. The use of a cemented fill will allow reduced dilution, but loading procedures with other fill materials can also achieve this goal.

3. Stabilise stope walls:

The fill must limit stope closure as the vertical height of the stope increases. The fill must have sufficient resistance against compression. This can be achieved with cement addition, but also with natural compaction of the fill under its own weight, increasing the placed density and consequently its load generation ability. Note that this is only possible if sufficient drainage of water from the backfill column is ensured if hydraulic fill is used. The use of rockfill is also sufficient to limit rockwall closure, provided a good fill of the open stope can be achieved.

4. Free-standing height:

Within each drift, the fill must be able to remain stable and not collapse into the next drift, requiring some strength that is only possible with the addition of cement. The amount of cement to be added is determined by the free-standing height required. Mining of the next stope in drifts will

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only expose the fill over the height of a drift, suggesting that full stope high free standing ability is not required.

5. Undercutting previously filled drifts:

High strength is required to ensure that the fill remains stable in the hangingwall across the drift width when undercut. Paste fill allows the best control of fill quality and strength for this purpose.

Vertical crater retreat operations

1. Stabilise stope walls:

The fill must limit stope closure. The fill must have sufficient resistance against compression, which can be achieved with cement addition.

2. Dilution:

Collapse into the next stope (or blast damage) will increase dilution and must be prevented by ensuring sufficient backfill strength with the addition of cement.

3. Free-standing height:

Within the exposed stope height, the fill must be able to remain stable and not collapse into the next drift, requiring some strength that is only possible with the addition of cement. The amount of cement to be added is determined by the free-standing height required.

4. Undercutting previously filled stopes:

High strength is required to ensure that the fill remains stable in the hangingwall across the drift width when undercut. Paste fill allows the best control of fill quality and strength for this purpose.

Shallow Wide-reef operations (post cut and fill)

1. Working platform:

Backfill material must be strong enough to be able to allow movement of equipment on top of fill. This can be achieved with a layer of cemented fill on top of any other fill material, but has also been achieved previously with a high density, uncemented fill.

2. Dilution:

Fill must limit dilution during loading of the blasted ore. The use of a cemented fill will allow reduced dilution, but loading procedures with other fill materials can also achieve this goal.

3. Confine pillars:

Pillars left intact to limit spans and control the overburden must be confined to increase their strength as their height increases. Confinement forces required are not substantial and any well-placed backfill could achieve the required confinement.

LEARNING OUTCOME 5.1.3.2

CONNECTION 41

Refer to
Paper 3.1 Outcome 5.4.2.11

DESCRIBE, EXPLAIN AND DISCUSS HOW BACKFILL FITS INTO THE MINING CYCLE IN THE ABOVE SITUATIONS

The sequencing of filling in massive mining is often complex and are also governed by factors that must be considered during the design of the mining system. These factors may include, but will not be limited to the following:

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- filling requirements of stopes either side of a pillar before extracting the pillar;
- careful consideration not to sterilize ore in any lift before filling;
- curing times required when cement based fill is used and drainage time to ensure a stable working platform or exposed fill wall;
- maximum mining height that will remain stable when exposed;

LEARNING OUTCOME 5.1.3.3

CONNECTION 42
Refer to
Paper 3.1 Outcome 5.4.2

DESCRIBE, EXPLAIN AND DISCUSS HOW BACKFILL ASSISTS IN IMPROVING THE STABILITY OF ROCKMASSES

In massive ore bodies, the functions of the backfill to affect and improve rockmass stability can be summarised as:

1. Pillar confinement:

Pillar confinement increases pillar strengths and assists in maintaining overall rockmass stability by limiting rockmass displacement towards the mined-out areas.

2. Stope wall displacement:

Placement of fill, depending on the backfill properties, resists inelastic stope wall movement into the mined-out stopes by generating resisting forces in the fill. By limiting the near field rockmass displacements, overall rockmass displacement is reduced. This process prevents surface impacts in shallower dipping or very wide orebodies, but in other steep dipping, wide orebodies, the reduction in rock mass displacements due to the placement of sufficient backfill (creating fill pillars) in stopes allow the mining of remnants / pillars / secondary stopes at large spans.

NOTE

The impacts discussed above are very dependent on the type and quality of backfill placed and selection of fill types / designs should be carefully evaluated to ensure that the required rock mass impact is possible with the selected fill type.

LEARNING OUTCOME 5.1.3.4

CONNECTION 43
Refer to
Paper 3.1 Outcome 5.4.12

DETERMINE THE SUPPORT RESISTANCE OFFERED BY BACKFILL GIVEN THE BACKFILL PERFORMANCE CHARACTERISTICS

LEARNING OUTCOME 5.1.3.5

CONNECTION 44
Refer to
"LEARNING OUTCOME 5.1.3.1" on page 305 to 5.1.3.4

EVALUATE GIVEN ROCKMASS AND MINING SITUATIONS AND NOTE THE POTENTIAL BENEFITS OF BACKFILLING IN THESE SITUATIONS

Table 1: Backfill benefits

Mining method	Method assumptions	Backfill characteristics and functions	Possible backfill types

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Cut-and-fill	Complete ore body width is extracted in a single cut. No pillars are left intact. Fill exposed as working platform only. Fill will be undercut from next levels.	Working platform. Sill pillar. Undercut stability over cut width span (2.4-12.6m).	If cemented sill is placed along stope bottom, any type can be used in cuts higher up in the stope.
Drift-and-fill	Small width drifts are mined next to each other until complete ore body width is extracted. No pillars are left intact. Fill exposed over height of drift. Fill will be undercut from next levels.	Working platform. Free-standing height (5m). Sill pillar. Undercut stability over drift width span (5m).	Cemented fill for free-standing height.
Post-pillar cut-and-fill	Complete ore body width is extracted in a single cut, leaving pillars of adequate dimension to be confined by the fill. Fill exposed as working platform only. Fill will be undercut from next levels.	Working platform. Sill pillar. Undercut stability over drift width span (12.6m).	If cemented sill is placed along stope bottom, any type can be used in cuts higher up in the stope.
Open stope with fill	Open stopes are created at suggested stable spans in primary and secondary extraction sequences and filled or fill is allowed to cure before mining continues in a single mining direction. No pillars are left intact. Fill exposed along vertical height of the open stopes. Fill will be undercut from next levels.	Working platform. Sill pillar. Free-standing height (20-40m). Undercut stability over stope width (2.4-12.6m).	Cemented fill for free standing height and undercutting stability.

LEARNING OUTCOME 5.1.3.6

CONNECTION 45

Refer to "LEARNING OUTCOME 5.1.3.1" on page 305 to 5.1.3.4
Paper 3.1 Outcome 5.4.2

EVALUATE GIVEN ROCKMASS AND MINING SITUATIONS AND RECOMMEND APPROPRIATE BACKFILL TYPE AND PLACEMENT PROCEDURES

5.2. SERVICE EXCAVATION SUPPORT STRATEGIES

5.2.1. TUNNEL SUPPORT STRATEGIES

CONNECTION 46

Refer to
Paper 3.1, Outcome 5.2.1
Paper 3.2, Outcome 5.4

THE CANDIDATE MUST BE ABLE TO DEMONSTRATE KNOWLEDGE AND UNDERSTANDING OF THE ABOVE SUBJECT AREA BY BEING ABLE TO:

LEARNING OUTCOME 5.2.1.1

CONNECTION 47

Refer to
Paper 3.1, Outcomes 3.2 ; 3.3; 5.2.1.1

SKETCH, DESCRIBE, EXPLAIN AND DISCUSS THE VARIOUS BEHAVIOUR OF ROCK AROUND EXCAVATIONS AT SHALLOW, INTERMEDIATE AND GREAT DEPTH

LEARNING OUTCOME 5.2.1.2

CONNECTION 48

Paper 3.1, Outcome 5.2.1.12

DESCRIBE, EXPLAIN AND DISCUSS HOW THIS AFFECTS TUNNEL SUPPORT REQUIREMENTS AT SHALLOW, INTERMEDIATE AND GREAT DEPTH

LEARNING OUTCOME 5.2.1.3

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CONNECTION 49
Refer to
Paper 3.1, Outcome 5.2.1.4

DESCRIBE, EXPLAIN AND DISCUSS THE OBJECTIVE PRIMARY SUPPORT IN TUNNELS

LEARNING OUTCOME 5.2.1.4

CONNECTION 50
Refer to
Paper 3.1, Outcome 5.2.1.5

DESCRIBE, EXPLAIN AND DISCUSS THE OBJECTIVE SECONDARY SUPPORT IN TUNNELS

LEARNING OUTCOME 5.2.1.5

CONNECTION 51
Refer to
Paper 3.1, Outcome 5.2.1.6

DESCRIBE, EXPLAIN AND DISCUSS THE CONCEPT OF INTEGRATED SUPPORT IN TUNNELS

LEARNING OUTCOME 5.2.1.6

CONNECTION 52
Refer to
Paper 3.1, Outcome 5.2.1.7

DESCRIBE, EXPLAIN AND DISCUSS VARIOUS SUPPORT STRATEGIES FOR:

- **Return airways**
- **Roadways, haulages, crosscuts**
- **Travelling ways**
- **Winzes, raises**
- **Tunnel intersections**
- **Tunnels traversing faults, tunnels traversing dykes**
- **Undercuts, drawpoints**
- **Gathering drives, drilling drives**

Support is the application of a reactive force at the face of an excavation. It includes techniques and devices such as timber, fill, shotcrete, mesh, lacing, sprayed liners and steel or concrete sets or liners.

Reinforcement is a means of improving the overall rockmass properties from within the rockmass by devices such as rock bolts, cable bolts and ground anchors.

Laubscher lists support practices for block cave tunnels situated in low and high stress conditions (Figure 13 and Table 2) that include both 'support' and 'reinforcement' techniques. Figure 13 allows the user to estimate support characteristics required based on the relationship between the RMR and MRMR values determined for the rockmass to be supported. Figure 13 shows that for:

- RMR <<< MRMR: Rigid/stiff support systems are required;
- RMR >>> MRMR: Yielding support systems are required.

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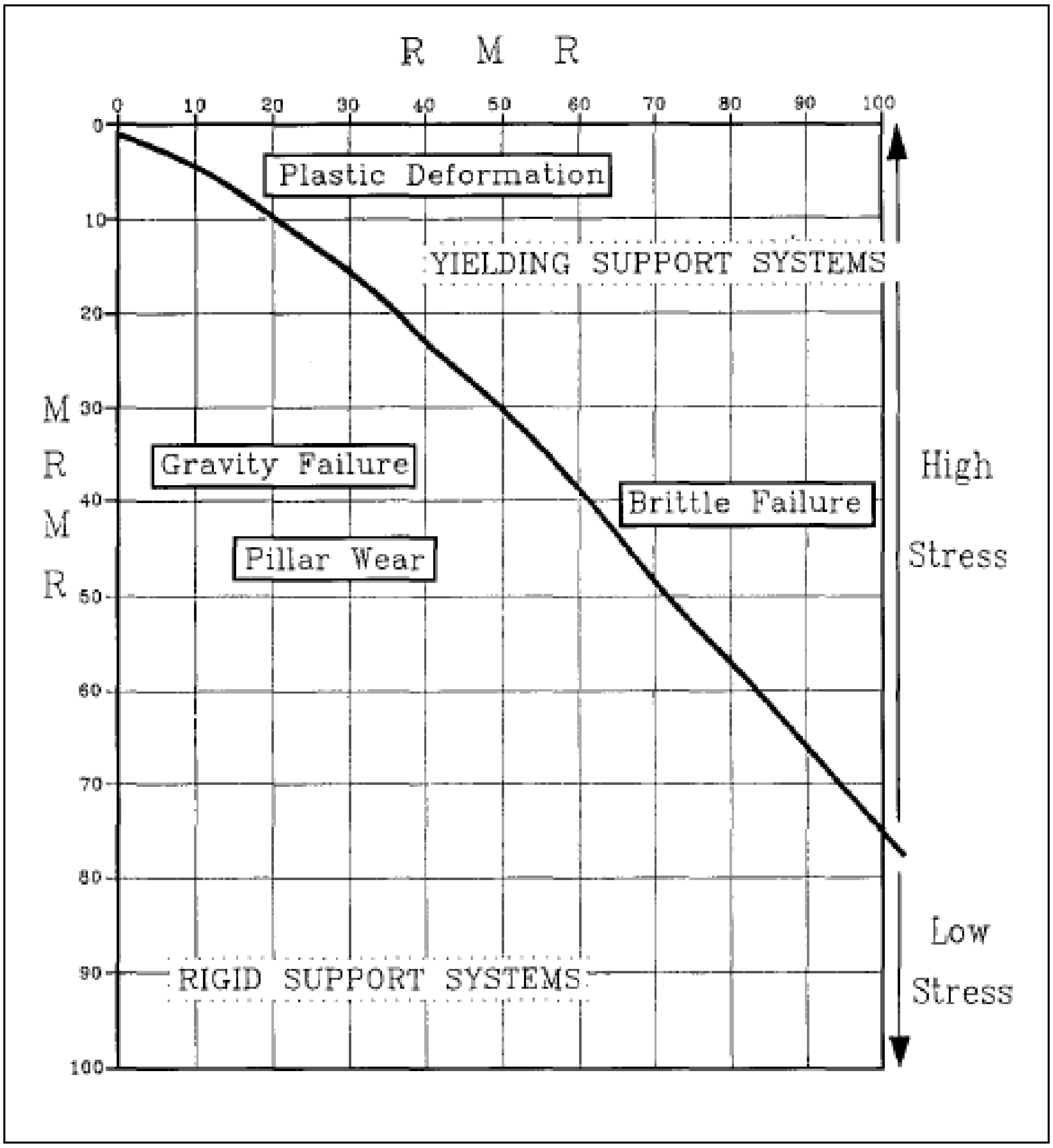

Figure 14: Support characteristics required based on RMR and MRMR (Laubscher, 1994)

Table 2: Drawpoint support technicques (Laubscher, 1994)

Support Techniques		
Support element	Low stress	**High stress**
Bolts - Length - Spacing - Type	= 1 m + (0,33 *W* × *F*) = 1 m = Rigid rebar	= 1 m + (0,5 *W* × *F*) = 1 m = Yielding, e.g. cones
Mesh	0.5 mm × 100 mm aperture	0.5 mm × 75 mm or 50 mm aperture
Deep-seated support	Cables = 1 m + 1.5 *W*	Steel ropes = 1 m + 1.5 *W* Long cone bolts
Shotcrete linings	Mesh-reinforced shotcrete	Mesh-reinforced shotcrete
Arches	Rigid steel arches Massive concrete	Yielding steel arches Reinforced concrete

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Support Techniques		
Support element	Low stress	**High stress**
Surface restraint	Large washers (triangles) Tendon straps	Large plate washers Yielding-tendon straps
Corners	25 mm rope-cable slings	25 mm rope-cable slings
Brows	Birdcage cables from undercut level Inclined pipes	Birdcage cables from undercut level Inclined pipes
Repairs	Grouting Extra bolts and cables Plates, straps, and arches	Grouting Extra bolts, ropes, plates, straps, and yielding arches

W is the span of the tunnel and *F* is based on

MRMR = 0-20 : F = 1,4 MRMR = 21-30 : F = 1.30 MRMR = 31-40 : F = 1,2

MRMR = 41-50 : F = 1.1 MRMR = 51-60 : F = 1,05 MRMR > 61 : F = 1,0

Figure 15: Steel arches in drawpoints (Courtesy of Petra Diamonds)

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Figure 16: Mesh, anchors and Steel strapping in drawpoints (Courtesy of Petra Diamonds)

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back

Figure 17: Steel arches and timber lagging in drawpoints (Courtesy of Petra Diamonds)

REFERENCE

- Block caving overview Petra
- State of the art cave mining Laubscher
- Inclined caving as massive mining method Munro
- Stress analysis of block cave layouts Esterhuizen
- Page 48, 81, 98, 104 Updated cave mining handbook Laubscher

LEARNING OUTCOME 5.2.1.7

CONNECTION 53

Refer to Paper 3.1, Outcome 5.2.1.8

DESCRIBE, EXPLAIN AND DISCUSS THE STRATEGIES TO SUPPORT TUNNELS UNDER THE FOLLOWING CONDITIONS:

- **Low stress with joint controlled behaviour**
- **High stress where stress fractures dominate stability**
- **Seismic and rockburst conditions**

LEARNING OUTCOME 5.2.1.8

CONNECTION 54

Refer to Paper 3.1, Outcome 5.2.1.9

DETERMINE APPROPRIATE SUPPORT STRATEGIES FOR TUNNELS FOR GIVEN LAYOUTS, ROCKMASS CONDITIONS, STRESS REGIMES AND GEOLOGICAL CIRCUMSTANCES.

5.2.2. SUPPORT OF DRAWPOINTS

THE CANDIDATE MUST BE ABLE TO DEMONSTRATE KNOWLEDGE AND UNDERSTANDING OF THE ABOVE SUBJECT AREA BY BEING ABLE TO:

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LEARNING OUTCOME 5.2.2.1

CONNECTION 55

Refer to "LEARNING OUTCOME 5.2.1.6" on page 310

SKETCH, DESCRIBE, EXPLAIN AND DISCUSS SUPPORT MEASURES FOR DRAWPOINTS, LOADING BAYS AND SIMILAR EXCAVATIONS AT SHALLOW, INTERMEDIATE AND GREAT DEPTH

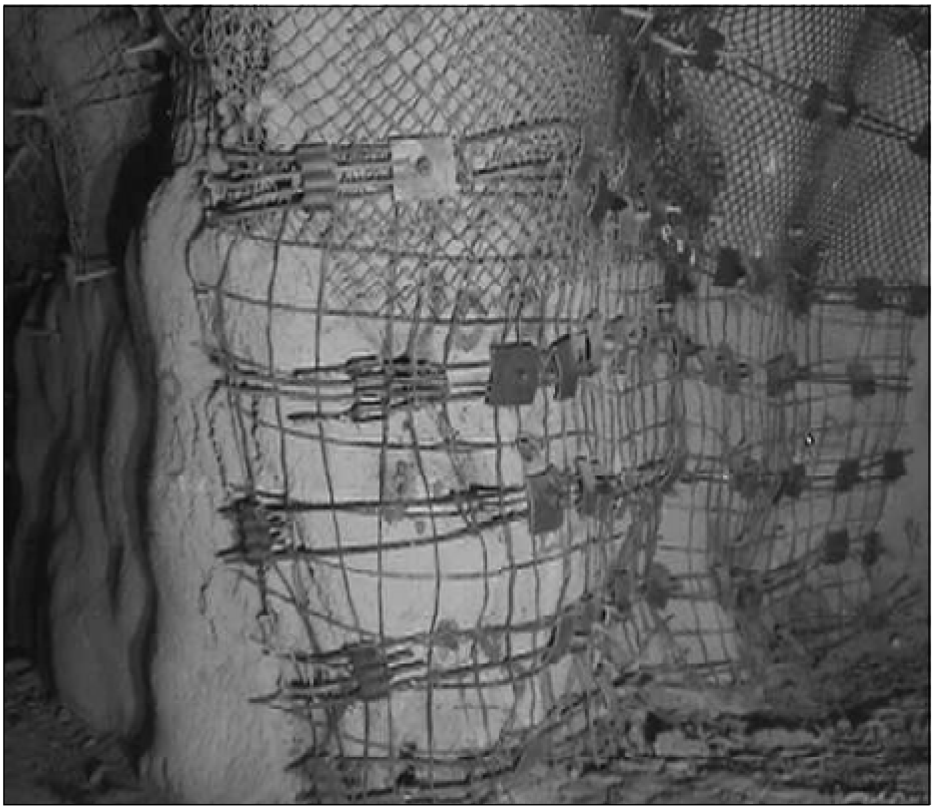

Figure 18: Drawpoint stub support (Wilson)

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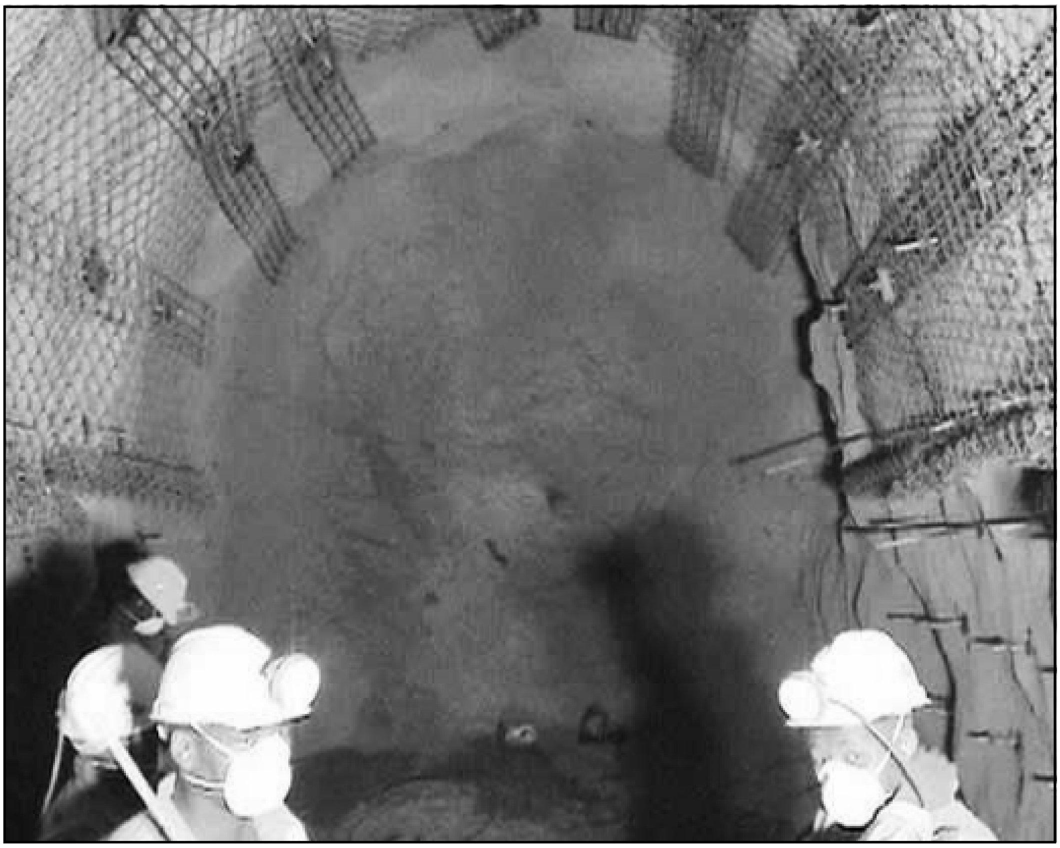

Figure 19: Drawpoint support (Wilson)

Figure 20: Stiff brow support (Wilson)

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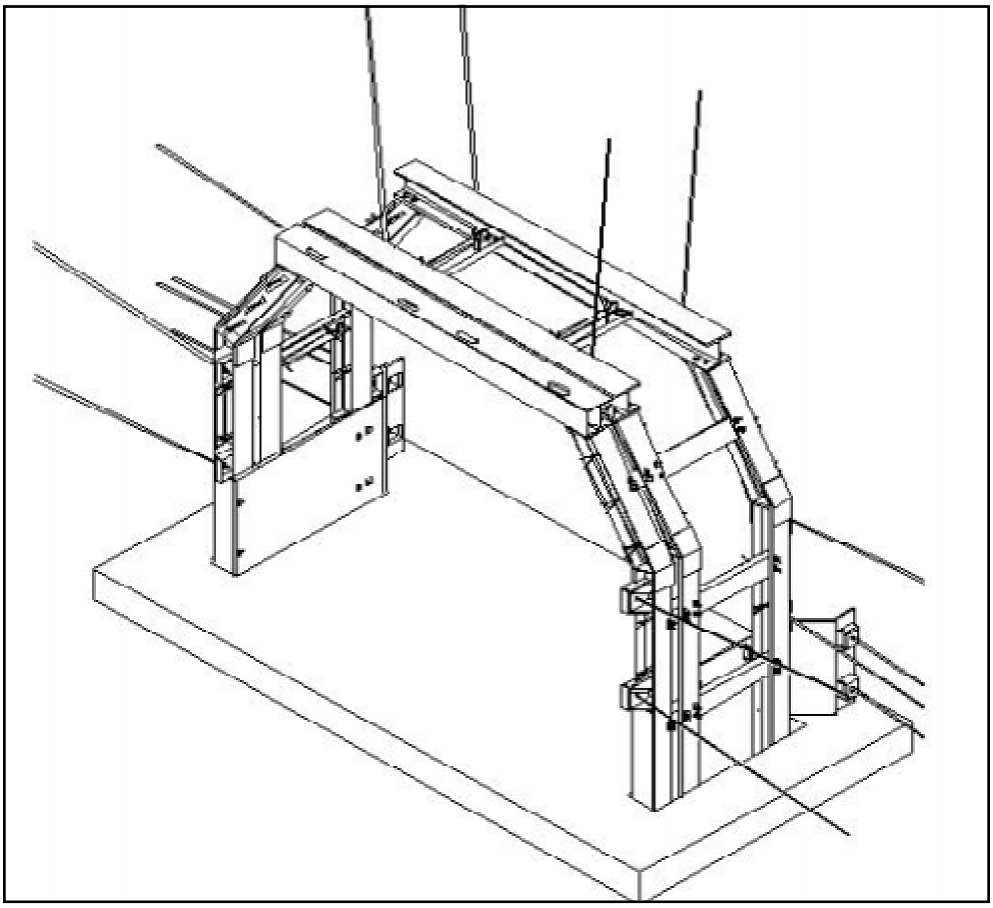

Figure 21: Stiff brow support (Jaggard)

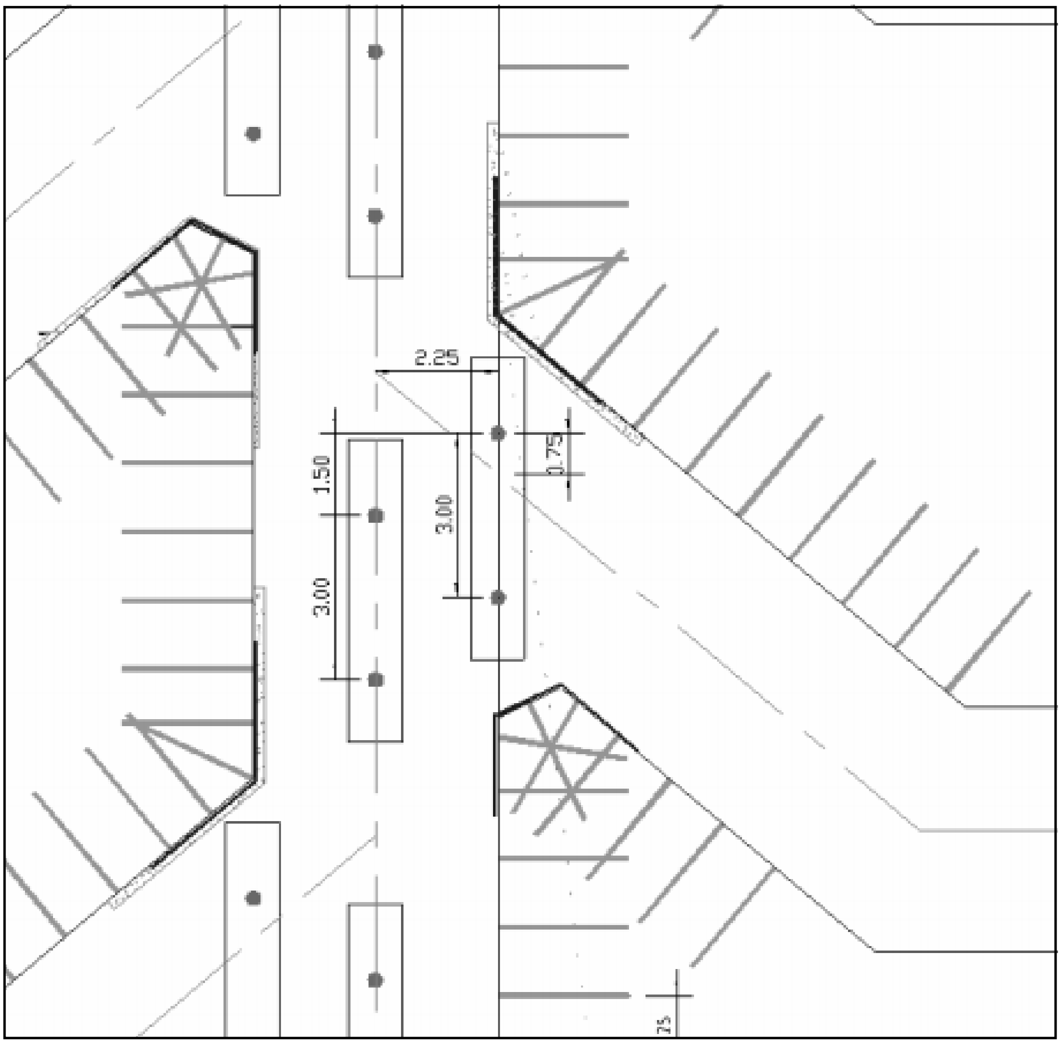

Figure 22: Support layout for production crosscuts and drawpoint entrance (Jaggard)

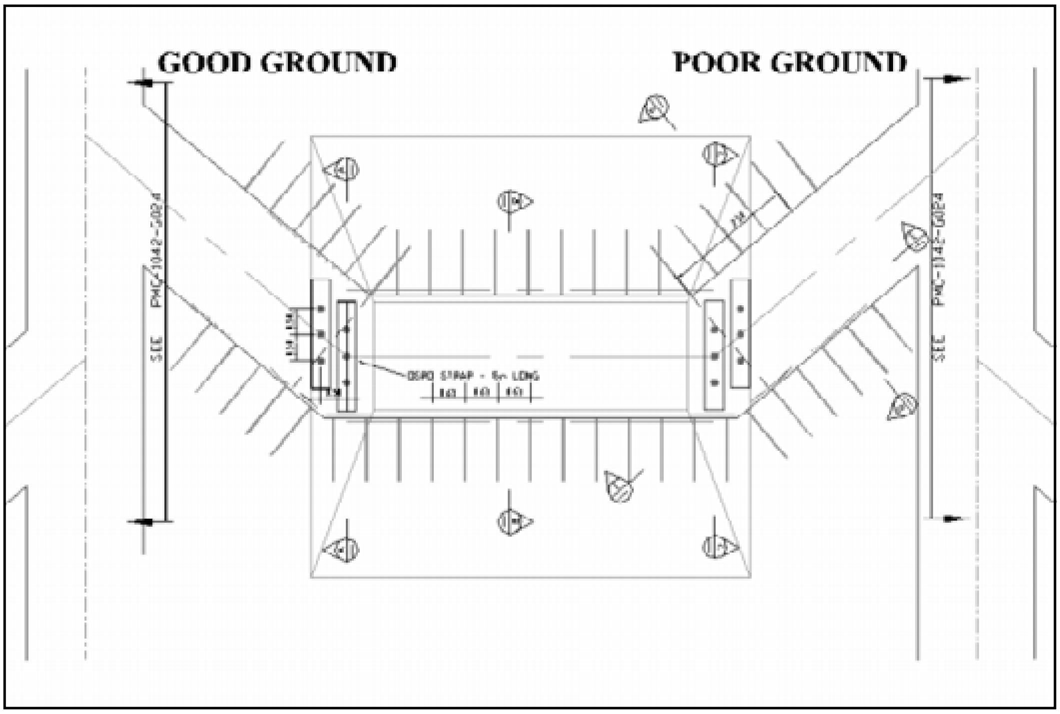

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Figure 23: Support for drawpoint and bell area (Jaggard)

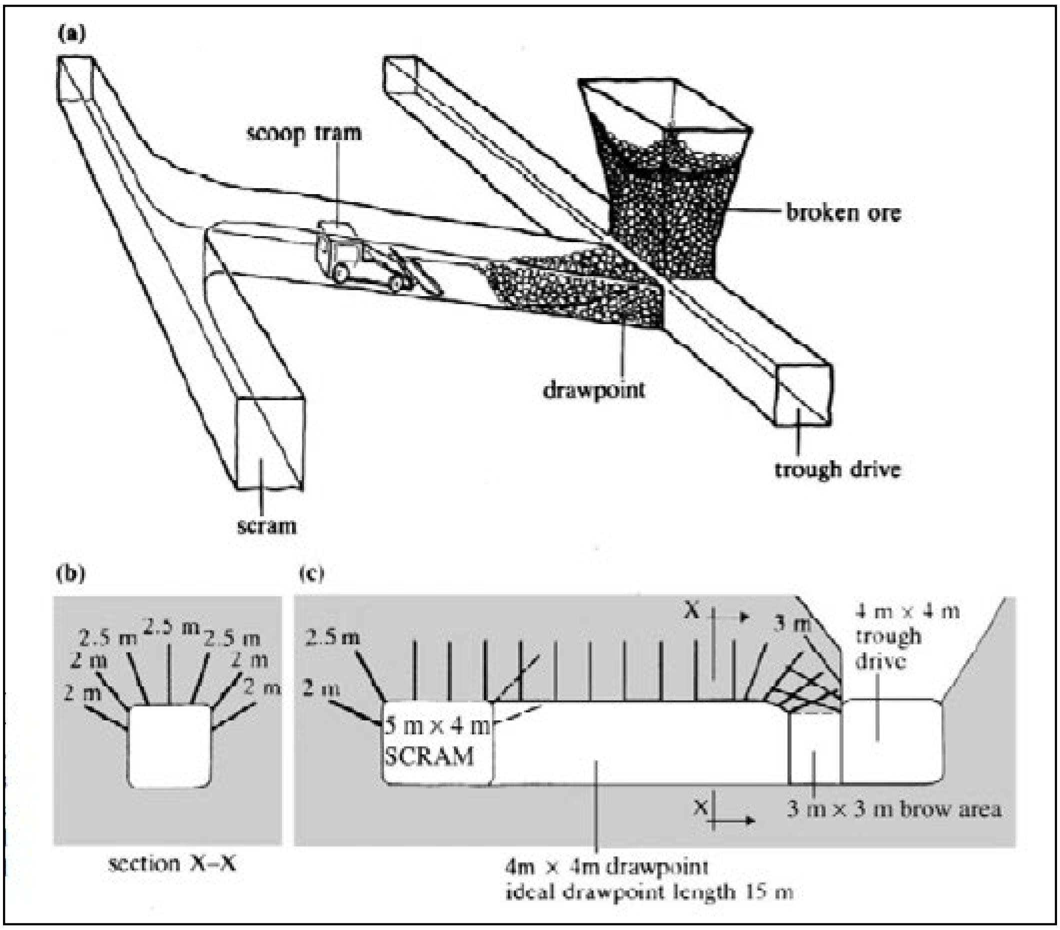

Figure 24: Support (Brady & Brown, 2006)

REFERENCE

- Drawpoint support Wilson
- Support challenges Jaggard
- Page 326, Brady and Brown, Rock Mechanics for underground mining, 2006

LEARNING OUTCOME 5.2.2.2

CONNECTION 56

Refer to
"LEARNING OUTCOME 5.2.1.6" on page 310
and
"LEARNING OUTCOME 5.2.2.1" on page 315

SKETCH, DESCRIBE, EXPLAIN AND DISCUSS SUPPORT MEASURES FOR DRAWPOINTS, LOADING BAYS AND SIMILAR EXCAVATIONS WHERE THE STRESS-TO-STRENGTH RATIO OF THE ROCKMASS APPROXIMATES UNITY

LEARNING OUTCOME 5.2.2.3

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CONNECTION 56

Refer to "LEARNING OUTCOME 5.2.1.6" on page 310 and "LEARNING OUTCOME 5.2.2.1" on page 315

SKETCH, DESCRIBE, EXPLAIN AND DISCUSS SPECIAL SUPPORT MEASURES FOR HIGH WEAR AREAS OF DRAWPOINTS, LOADING BAYS AND SIMILAR EXCAVATIONS

5.2.3. SUPPORT OF ROCKPASSES

THE CANDIDATE MUST BE ABLE TO DEMONSTRATE KNOWLEDGE AND UNDERSTANDING OF THE ABOVE SUBJECT AREA BY BEING ABLE TO:

LEARNING OUTCOME 5.2.3.1

CONNECTION 58

Refer to Paper 3.1, Outcomes 5.2.2.3 and 4.4.3

SKETCH, DESCRIBE, EXPLAIN AND DISCUSS SUPPORT MEASURES FOR ROCK PASSES AND SIMILAR EXCAVATIONS AT SHALLOW, INTERMEDIATE AND GREAT DEPTH.

5.2.4. SUPPORT OF LARGE CHAMBERS

THE CANDIDATE MUST BE ABLE TO DEMONSTRATE KNOWLEDGE AND UNDERSTANDING OF THE ABOVE SUBJECT AREA BY BEING ABLE TO:

LEARNING OUTCOME 5.2.4.1

CONNECTION 59

Refer to Paper 3.1, Outcomes 3.2, 3.3, 5.2.2.4.1

SKETCH, DESCRIBE, EXPLAIN AND DISCUSS THE VARIOUS BEHAVIOUR OF ROCK AROUND LARGE EXCAVATIONS AT SHALLOW, INTERMEDIATE AND GREAT DEPTH

LEARNING OUTCOME 5.2.4.2

CONNECTION 60

Refer to Paper 3.1, Outcomes 3.3.2, 3.3.20, 3.3.27, 5.2.2.4.2

DESCRIBE, EXPLAIN AND DISCUSS HOW THIS AFFECTS LARGE EXCAVATION SUPPORT REQUIREMENTS AT SHALLOW, INTERMEDIATE AND GREAT DEPTH

LEARNING OUTCOME 5.2.4.3

CONNECTION 61

Refer to Paper 3.1, Outcomes 3.3.2, 5.2.2.4.3

DESCRIBE, EXPLAIN AND DISCUSS ROCKMASS CLASSIFICATION SCHEMES FOR THE DETERMINATION OF SUPPORT STRATEGIES FOR LARGE CHAMBERS

LEARNING OUTCOME 5.2.4.4

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CONNECTION 62

Refer to
Paper 3.1, Outcomes
2.3.1.3, 5.2.2.4.4

DESCRIBE, EXPLAIN AND DISCUSS THE APPLICATION OF THE ROCKWALL CONDITION FACTOR METHODOLOGY TO DESIGN SUPPORT FOR LARGE EXCAVATIONS AT DEPTH

LEARNING OUTCOME 5.2.4.5

CONNECTION 63

Refer to
Paper 3.1, Outcomes
2.3.1.3, 5.2.2.4.5

APPLY THE ROCKWALL CONDITION FACTOR (RCF) METHODOLOGY TO DESIGN SUPPORT FOR LARGE EXCAVATIONS AT DEPTH

LEARNING OUTCOME 5.2.4.6

CONNECTION 64

Refer to
Paper 3.1, Outcomes
3.3.19, 3.3.20, 5.2.2.4.7

DESCRIBE, EXPLAIN AND DISCUSS THE EFFECTS OF EXCAVATION SEQUENCE ON SUPPORT INSTALLATION

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LEARNING OUTCOME 5.2.4.7

CONNECTION 65

Refer to Paper 3.1, Outcome 5.2.2.4.8

DETERMINE APPROPRIATE EXCAVATION SEQUENCES TO FACILITATE SUPPORT INSTALLATION

LEARNING OUTCOME 5.2.4.8

CONNECTION 66

Refer to Paper 3.1, Outcome 5.2.2.4.9

DESCRIBE, EXPLAIN AND DISCUSS THE EFFECTS OF EXCAVATION SEQUENCE ON SUPPORT EFFECTIVENESS

LEARNING OUTCOME 5.2.4.9

CONNECTION 67

Refer to Paper 3.1, Outcome 5.2.2.4.10

DETERMINE APPROPRIATE EXCAVATION SEQUENCES TO FACILITATE SUPPORT EFFECTIVENESS

LEARNING OUTCOME 5.2.4.10

CONNECTION 68

Refer to Paper 3.1, Outcome 5.2.2.4.11

DESCRIBE, EXPLAIN AND DISCUSS THE RULES OF THUMB TO DETERMINE THE LENGTH AND SPACING OF TENDONS IN LARGE EXCAVATIONS

LEARNING OUTCOME 5.2.4.11

CONNECTION 69

Refer to Paper 3.1, Outcome 5.2.2.4.12

APPLY THESE RULES OF THUMB TO THE DESIGN OF SUPPORT IN LARGE EXCAVATIONS

LEARNING OUTCOME 5.2.4.12

CONNECTION 70

Refer to Paper 3.1, Outcomes 5.2.2.4.13, 3.1.17

DESCRIBE, EXPLAIN AND DISCUSS THE EFFECTS OF OVERSTOPING AND UNDERSTOPING ON THE STABILITY AND SUPPORT REQUIREMENTS OF OFF-REEF EXCAVATIONS

LEARNING OUTCOME 5.2.4.13

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CONNECTION 71

Refer to
Paper 3.1, Outcomes
5.2.2.4.14, 3.3

DETERMINE APPROPRIATE SUPPORT STRATEGIES FOR LARGE CHAMBERS FOR GIVEN MINING AND SHAFT LAYOUTS, ROCKMASS CONDITIONS, STRESS REGIMES AND GEOLOGICAL CIRCUMSTANCES

5.2.5. SUPPORT OF SHAFTS

CONNECTION 72

Refer to
Paper 3.1, Outcomes
5.2.2.1, 5.2.2.2

THE CANDIDATE MUST BE ABLE TO DEMONSTRATE KNOWLEDGE AND UNDERSTANDING OF THE ABOVE SUBJECT AREA BY BEING ABLE TO:

LEARNING OUTCOME 5.2.5.1

CONNECTION 73

Refer to
Paper 3.1, Outcome 5.2.2.1

DESCRIBE, EXPLAIN AND DISCUSS ROCKMASS CLASSIFICATION SCHEMES FOR THE DETERMINATION OF SUPPORT STRATEGIES FOR SHAFTS

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LEARNING OUTCOME 5.2.5.2

CONNECTION 74

Refer to Paper 3.1, Outcomes 5.2.2.1, 4.4.4

DETERMINE APPROPRIATE SUPPORT STRATEGIES FOR VERTICAL SHAFTS FOR GIVEN MINING AND SHAFT LAYOUTS, ROCKMASS CONDITIONS, STRESS REGIMES AND GEOLOGICAL CIRCUMSTANCES

LEARNING OUTCOME 5.2.5.3

CONNECTION 75

Refer to Paper 3.1, Outcome 5.2.2.2

DETERMINE APPROPRIATE SUPPORT STRATEGIES FOR INCLINED SHAFTS FOR GIVEN MINING AND SHAFT LAYOUTS, ROCKMASS CONDITIONS, STRESS REGIMES AND GEOLOGICAL CIRCUMSTANCES

LEARNING OUTCOME 5.2.5.4

CONNECTION 76

Apply "support design criteria" on page 324

DETERMINE APPROPRIATE SUPPORT STRATEGIES FOR ROADWAYS FOR GIVEN MINING AND SHAFT LAYOUTS, ROCKMASS CONDITIONS, STRESS REGIMES AND GEOLOGICAL CIRCUMSTANCES.

REFERENCE

- Drawpoint support Wilson
- Block caving overview Petra
- State of the art cave mining Laubscher
- Inclined caving as massive mining method Munro
- Stress analysis of block cave layouts Esterhuizen
- Page 48, 81, 98, 104 Updated cave mining handbook Laubscher

5.3. SUPPORT DESIGN CRITERIA

5.3.1. STOPE SUPPORT DESIGN CRITERIA

CONNECTION 77

Refer to Paper 2, Outcome 5.1.5.2
Paper 3.1, Outcome 5.3

THE CANDIDATE MUST BE ABLE TO DEMONSTRATE KNOWLEDGE AND UNDERSTANDING OF THE ABOVE SUBJECT AREA BY BEING ABLE TO:

LEARNING OUTCOME 5.3.1.1

CONNECTION 78

Refer to Paper 3.1, Outcomes 5.3.1.1, 5.3.1.2

DESCRIBE, EXPLAIN AND DISCUSS STOPE SUPPORT REQUIREMENTS IN TERMS OF:

- Initial stiffness, yieldability
- Areal coverage
- Support resistance, energy absorption

LEARNING OUTCOME 5.3.1.2

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CONNECTION 79

Refer to
Paper 3.1, Outcome 5.1.16, 5.1.17

DESCRIBE, EXPLAIN AND DISCUSS HOW SUPPORT RESISTANCE VARIES FOR STATIC AND DYNAMIC CONDITIONS

5.3.2. TUNNEL, CHAMBER AND SHAFT SUPPORT DESIGN CRITERIA

CONNECTION 80

Refer to
Paper 2, Outcome 5.1, 5.2
Paper 3.1, Outcome 5.3.2

THE CANDIDATE MUST BE ABLE TO DEMONSTRATE KNOWLEDGE AND UNDERSTANDING OF THE ABOVE SUBJECT AREA BY BEING ABLE TO:

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LEARNING OUTCOME 5.3.2.1

CONNECTION 81

Refer to
Paper 3.1, Outcome 5.3.2.1

DESCRIBE, EXPLAIN AND DISCUSS EXCAVATION SUPPORT AND ROCKWALL REINFORCEMENT REQUIREMENTS IN TERMS OF:

- Initial stiffness, yieldability
- Unit length, areal coverage
- Support resistance, energy absorption

LEARNING OUTCOME 5.3.2.2

CONNECTION 82

Refer to
Paper 3.1, Outcome 5.3.2.2

DESCRIBE, EXPLAIN AND DISCUSS HOW SUPPORT RESISTANCE VARIES FOR STATIC AND DYNAMIC CONDITIONS

LEARNING OUTCOME 5.3.2.3

CONNECTION 83

Refer to
Paper 3.1, Outcome 5.3.2.3

CALCULATE REQUIRED SUPPORT RESISTANCES FOR STATIC CONDITIONS

LEARNING OUTCOME 5.3.2.4

CONNECTION 84

Refer to
Paper 3.1, Outcome 5.3.2.4

CALCULATE REQUIRED SUPPORT RESISTANCES FOR DYNAMIC (ROCKBURST) CONDITIONS

LEARNING OUTCOME 5.3.2.5

CONNECTION 85

Refer to
Paper 3.1, Outcome 5.3.2.5

CALCULATE SUPPORT LOADS FOR SUPPORT SYSTEMS MADE UP OF VARIOUS UNITS

LEARNING OUTCOME 5.3.2.6

CONNECTION 86

Refer to
Paper 3.1, Outcome 5.3.2.6

CALCULATE ENERGY ABSORPTION FOR SUPPORT SYSTEMS MADE UP OF VARIOUS UNITS

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LEARNING OUTCOME 5.3.2.7

DESCRIBE, EXPLAIN AND DISCUSS SUPPORT REQUIREMENTS FOR TIME-DEPENDENT (CREEP) CONDITIONS

CONNECTION 87

Refer to
Paper 3.1, Outcomes 3.2.7, 3.2.8, 3.3.114, 3.3.15, 5.2.1.10

REFERENCE

- Hoek and Marinos – predicting squeezing material

5.4. SUPPORT AND SUPPORT SYSTEM TYPES AND CHARACTERISTICS

THE CANDIDATE MUST BE ABLE TO DEMONSTRATE KNOWLEDGE AND UNDERSTANDING OF THE ABOVE SUBJECT AREA BY BEING ABLE TO:

CONNECTION 88

Refer to
Paper 2, Outcome 5.1
Paper 3.1, Outcome 5.5.1

LEARNING OUTCOME 5.4.1

APPLY ROCKMASS CLASSIFICATION SYSTEMS TO SELECT APPROPRIATE SUPPORT FOR STOPES, TUNNELS AND CHAMBERS IN MASSIVE MINES

CONNECTION 89

Refer to
Paper 3.1, Outcome 5.4.4.1

LEARNING OUTCOME 5.4.2

APPLY THE ROCK CONDITION FACTOR (RCF) TO SELECT APPROPRIATE SUPPORT FOR TUNNELS IN HIGH STRESSES

CONNECTION 90

Refer to
Paper 3.1, Outcome 5.4.4.2

LEARNING OUTCOME 5.4.3

DESCRIBE AND DISCUSS THE FOLLOWING ROCKWALL SUPPORT TYPES:

CONNECTION 91

Refer to
Paper 2, Outcome 5.1.7
Paper 3.1, Outcome 5.4.4.4

- Mechanically anchored bolts, cable bolts, friction bolts (split sets,
- Cement grouted bolts, resin bonded bolts
- Full-column grouted/bonded bolts
- Yielding bolts (cone bolts)
- Multiple rod bolts, multiple strand cable anchors
- Pre-stressed tendons
- Steel arches, massive concrete linings
- Shotcrete, gunite, thin sprayed linings
- Wire mesh, rope lacing, tendon straps

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LEARNING OUTCOME 5.4.4

CONNECTION 92

Refer to
Paper 2, Outcome 5.1.8
Paper 3.1, Outcome 5.4.4.5

CHARACTERISE THE FOLLOWING ASPECTS OF THE ABOVE UNITS:

- Their principles of operation
- Their technical specifications
- Their load-deformation characteristics
- The methods of ensuring support unit quality
- Their installation/application procedures
- The methods of ensuring their installed quality

LEARNING OUTCOME 5.4.5

CONNECTION 93

Refer to
Paper 2, Outcome 5.1.9
Paper 3.1, Outcome 5.4.4.6

DESIGN AND EVALUATE THE USE OF APPROPRIATE SUPPORT UNITS, SUPPORT SYSTEMS, SUPPORT PATTERNS AND INSTALLATION PROCEDURES FOR GIVEN ROCKMASS CONDITIONS, STRESS REGIMES AND MINING LAYOUTS

CHAPTER

6

PAPER 3.3 :MASSIVE UNDERGROUND MINING (HARD AND SOFT ROCK)

LEARNING OUTCOMES

6. INVESTIGATION TECHNIQUES
6.1. ROCK TESTING

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

- Describe, explain and discuss various rock testing procedures; and
- Interpret and incorporate test results in analyses and design.

6.2. MONITORING
6.2.1. SUBSIDENCE MONITORING

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

- Sketch, describe, explain and discuss the techniques used to measure surface subsidence;
- Describe, explain and discuss the equipment used to measure surface subsidence;
- Describe, explain and discuss how vertical and horizontal displacements are determined;
- Describe, explain and discuss how strains and tilts may be derived from these determinations;
- Calculate strain and tilt from given sets of measurements;
- For given sets of underground mining and surface infrastructure conditions:
  - State, describe, explain and discuss what types of measurements need to be made and monitored,
  - Describe, explain and discuss required monitoring station layouts,
  - Describe, explain and discuss appropriate monitoring programmes,
  - Describe, explain and discuss the frequency of measurements,
  - Interpret, explain and discuss given subsidence data in terms of likely surface behaviour.

6.2.2. EXCAVATION DEFORMATION MONITORING

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

- Sketch, describe, explain and discuss the techniques used to measure deformations in underground excavations;

- Describe, explain and discuss the equipment used to measure deformations in underground excavations;
- Describe, explain and discuss how displacements in the rockmass are determined;
- Describe, explain and discuss how strains and tilts may be derived from these determinations;
- Calculate strain and tilt from given sets of measurements;
- For given sets of underground mining conditions:
    - State, describe, explain and discuss what types of measurements need to be made and monitored,
    - Describe, explain and discuss required monitoring station layouts,
    - Describe, explain and discuss appropriate monitoring programmes,
    - Describe, explain and discuss the frequency of measurements,
    - Interpret, explain and discuss given monitoring data in terms of likely excavation behaviour.

6.2.3. IN SITU STRESS MEASUREMENT AND MONITORING

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

- Sketch, describe, explain and discuss the techniques used to measure in situ stress in the underground rockmass;
- Describe, explain and discuss the equipment used to measure in situ stress in the rockmass; and
- Interpret, explain and discuss given stress measurement data in terms of likely rockmass, pillar or excavation behaviour.

6.2.4. SEISMIC MONITORING SYSTEMS

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

- Describe, discuss and explain how to determine objectives for a seismic monitoring system in massive mining operations;
- Explain and discuss key performance criteria for seismic networks;
- Explain and discuss the evaluation of historic seismicity for network design;
- Explain and discuss the requirements of spatial coverage;
- Explain and discuss the needs of various stakeholders for seismic information with respect to monitoring objectives;
- Describe, discuss and explain how to derive a compromise between ideal network layout and the following constraints:
    - Accessibility constraints,
    - Budgetary constraints;
- Describe, discuss and explain the factors that need to be considered in placing seismometers;
- Explain and discuss how seismometer placement could affect the event location accuracy of a seismic system;
- Explain and discuss how seismometer placement could affect the sensitivity of a seismic system;
- Describe, discuss and explain relevant procedures to ensure

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that seismic monitoring information is distributed effectively to stakeholders;

- Describe, discuss and explain methods of quality assurance in seismic system management;
- Describe and discuss relevant aspects of seismic system audits;
- Describe and discuss methods to assess stakeholder needs; and
- Describe and discuss methods to identify possible shortfalls between seismic monitoring objectives and seismic system performance.

6.3. MODELLING

6.3.1. NUMERICAL MODELLING

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

- Describe, explain and discuss the selection of appropriate codes to tackle various problems;
- Describe, explain and discuss the input of appropriate parameters to investigate various problems; and
- Describe, explain and discuss the interpretation of output in the investigation of various problems.

6.4. AUDITING

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

- Describe, explain and discuss the concept of monitoring for understanding, prediction and design.

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6. INVESTIGATION TECHNIQUES
6.1. ROCK TESTING

CONNECTION 1

Refer to
Paper 2, Outcome 6.1

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

LEARNING OUTCOME 6.1.1

DESCRIBE, EXPLAIN AND DISCUSS VARIOUS ROCK TESTING PROCEDURES

CONNECTION 2

Refer to
Paper 2, Outcome 6.1.1
Paper 3.1, Outcome 6.1.1

LEARNING OUTCOME 6.1.2

INTERPRET AND INCORPORATE TEST RESULTS IN ANALYSIS AND DESIGN

CONNECTION 3

Refer to
Paper 2, Outcome 6.1.1
Paper 3.1, Outcome 6.1.1

6.2. MONITORING
6.2.1. SUBSIDENCE MONITORING

CONNECTION 4

Refer to
Paper 2, Outcome 6.2
Paper 3.1, Outcome 6.2.3
Paper 3.2, Outcome 6.2.1

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

LEARNING OUTCOME 6.2.1.1

SKETCH, DESCRIBE, EXPLAIN AND DISCUSS THE TECHNIQUES USED TO MEASURE SURFACE SUBSIDENCE

CONNECTION 5

Refer to
Paper 3.2, Outcome 6.2.1.1

LEARNING OUTCOME 6.2.1.2

DESCRIBE, EXPLAIN AND DISCUSS THE EQUIPMENT USED TO MEASURE SURFACE SUBSIDENCE

CONNECTION 6

Refer to
Paper 3.2, Outcome 6.2.1.2

LEARNING OUTCOME 6.2.1.3

DESCRIBE, EXPLAIN AND DISCUSS HOW VERTICAL AND HORIZONTAL DISPLACEMENTS ARE DETERMINED

CONNECTION 7

Refer to
Paper 3.2, Outcome 6.2.1.3

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LEARNING OUTCOME 6.2.1.4

DESCRIBE, EXPLAIN AND DISCUSS HOW STRAINS AND TILTS MAY BE DERIVED FROM THESE DETERMINATIONS

CONNECTION 8

Refer to Paper 3.2, Outcome 6.2.1.4

LEARNING OUTCOME 6.2.1.5

CALCULATE STRAIN AND TILT FROM GIVEN SETS OF MEASUREMENTS

CONNECTION 9

Refer to Paper 3.2, Outcome 6.2.1.5

LEARNING OUTCOME 6.2.1.6

FOR GIVEN SETS OF UNDERGROUND MINING AND SURFACE INFRASTRUCTURE CONDITIONS:

- State, describe, explain and discuss what types of measurements need to be made and monitored,
- Describe, explain and discuss required monitoring station layouts,
- Describe, explain and discuss appropriate monitoring programmes,
- Describe, explain and discuss the frequency of measurements,
- Interpret, explain and discuss given subsidence data in terms of likely surface behaviour.

CONNECTION 10

Refer to Paper 3.2, Outcome 6.2.1.6

6.2.2. EXCAVATION DEFORMATION MONITORING

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

CONNECTION 11

Refer to Paper 2, Outcome 6.2

LEARNING OUTCOME 6.2.2.1

SKETCH, DESCRIBE, EXPLAIN AND DISCUSS THE TECHNIQUES USED TO MEASURE DEFORMATIONS IN UNDERGROUND EXCAVATIONS

CONNECTION 12

Refer to Paper 2, Outcome 6.2.3

LEARNING OUTCOME 6.2.2.2

DESCRIBE, EXPLAIN AND DISCUSS THE EQUIPMENT USED TO MEASURE DEFORMATIONS IN UNDERGROUND EXCAVATIONS

CONNECTION 13

Refer to Paper 2, Outcome 6.2.3

LEARNING OUTCOME 6.2.2.3

DESCRIBE, EXPLAIN AND DISCUSS HOW DISPLACEMENTS IN THE ROCKMASS ARE DETERMINED

CONNECTION 14

Refer to Paper 2, Outcome 6.2.3

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LEARNING OUTCOME 6.2.2.4

CONNECTION 15
Refer to
Paper 2, Outcome 6.2.3

DESCRIBE, EXPLAIN AND DISCUSS HOW STRAINS AND TILTS MAY BE DERIVED FROM THESE DETERMINATIONS

LEARNING OUTCOME 6.2.2.5

CONNECTION 16
Refer to
Paper 3.1, Outcome 4.4.4.1.3

CALCULATE STRAIN AND TILT FROM GIVEN SETS OF MEASUREMENTS

LEARNING OUTCOME 6.2.2.6

CONNECTION 17
Refer to
Paper 2, Outcome 6.2
Paper 3.1, Outcome 6.2.1 and 6.2.3

FOR GIVEN SETS OF UNDERGROUND MINING CONDITIONS:

- State, describe, explain and discuss what types of measurements need to be made and monitored,
- Describe, explain and discuss required monitoring station layouts,
- Describe, explain and discuss appropriate monitoring programmes,
- Describe, explain and discuss the frequency of measurements,
- Interpret, explain and discuss given monitoring data in terms of likely excavation behaviour.

CONNECTION 18
Refer to
Paper 2, Outcome 6.2
Paper 3.1, Outcome 6.2.1
Paper 3.2, Outcome 6.2.2

6.2.3. IN SITU STRESS MEASUREMENT AND MONITORING

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

LEARNING OUTCOME 6.2.3.1

CONNECTION 19
Refer to
Paper 1, Outcome 5.1.5
Paper 3.2, Outcome 6.2.1.1

SKETCH, DESCRIBE, EXPLAIN AND DISCUSS THE TECHNIQUES USED TO MEASURE IN SITU STRESS IN THE UNDERGROUND ROCKMASS

LEARNING OUTCOME 6.2.3.2

CONNECTION 20
Refer to
Paper 1, Outcome 5.1.7
Paper 3.2, Outcome 6.2.1.2

DESCRIBE, EXPLAIN AND DISCUSS THE EQUIPMENT USED TO MEASURE IN SITU STRESS IN THE ROCKMASS

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LEARNING OUTCOME 6.2.3.3

CONNECTION 21
Refer to
Paper 3.2, Outcome 6.2.1.3

INTERPRET, EXPLAIN AND DISCUSS GIVEN STRESS MEASUREMENT DATA IN TERMS OF LIKELY ROCKMASS, PILLAR OR EXCAVATION BEHAVIOUR.

CONNECTION 22
Refer to
Paper 2, Outcome 6.3
Paper 3.1, Outcome 6.2.2
Paper 3.4 Outcome 3.1.7

6.2.4. SEISMIC MONITORING SYSTEMS

Conceptual approach to Outcomes 6.2.4.1-6.2.4.6:

Explain basis of system and network design; key performance measures are sensitivity and accuracy; tools to model performance; how to define 'required sensitivity'; impact of limitations to ideal network layout.

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

LEARNING OUTCOME 6.2.4.1

CONNECTION 23
Refer to
Paper 3.1, Outcome 6.2.2.5

DESCRIBE, DISCUSS AND EXPLAIN HOW TO DETERMINE OBJECTIVES FOR A SEISMIC MONITORING SYSTEM IN MASSIVE MINING OPERATIONS

LEARNING OUTCOME 6.2.4.2

CONNECTION 24
Refer to
Paper 3.1, Outcome 6.2.2.6

EXPLAIN AND DISCUSS KEY PERFORMANCE CRITERIA FOR SEISMIC NETWORKS

LEARNING OUTCOME 6.2.4.3

CONNECTION 25
Refer to
Paper 3.1, Outcome 6.2.2.7

EXPLAIN AND DISCUSS THE EVALUATION OF HISTORIC SEISMICITY FOR NETWORK DESIGN

LEARNING OUTCOME 6.2.4.4

CONNECTION 26
Refer to
Paper 3.1, Outcome 6.2.2.8

EXPLAIN AND DISCUSS THE REQUIREMENTS OF SPATIAL COVERAGE

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LEARNING OUTCOME 6.2.4.5

CONNECTION 27
Refer to
Paper 3.1, Outcome 6.2.2.10

EXPLAIN AND DISCUSS THE NEEDS OF VARIOUS STAKEHOLDERS FOR SEISMIC INFORMATION WITH RESPECT TO MONITORING OBJECTIVES

LEARNING OUTCOME 6.2.4.6

CONNECTION 28
Refer to
Paper 3.1, Outcome 6.2.2.12

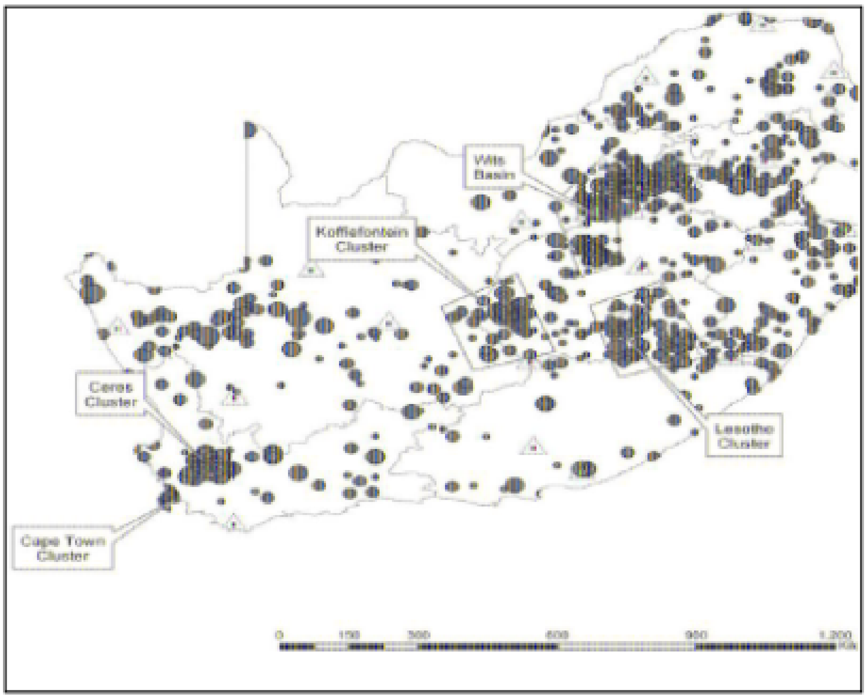
DESCRIBE, DISCUSS AND EXPLAIN HOW TO DERIVE A COMPROMISE BETWEEN IDEAL NETWORK LAYOUT AND THE FOLLOWING CONSTRAINTS:

- Accessibility constraints
- Budgetary constraints

Introduction

Many of the open pit and shallow massive mines in South Africa now operate seismic systems, collect data and analyse these regularly. The standard for seismic system technology is a combination of tri-axial geophone sets delivering seismograms to nearby seismographs where signals are digitised, stored and subsequently transmitted to a central system computer, provided they pass validity tests.

In addition to mine networks, the Council for Geosciences in Pretoria operates sensor sites near or directly within mining districts, as in the West Rand and Vaal River regions. Over 95% of seismicity recorded by the South African National Seismic Network originates in mining areas and the cluster of recorded events geographically linked to the deep mines follows the arch-shaped outline of the Wits basin, from Virginia in the South West to Boksburg in the North East (Figure 1).



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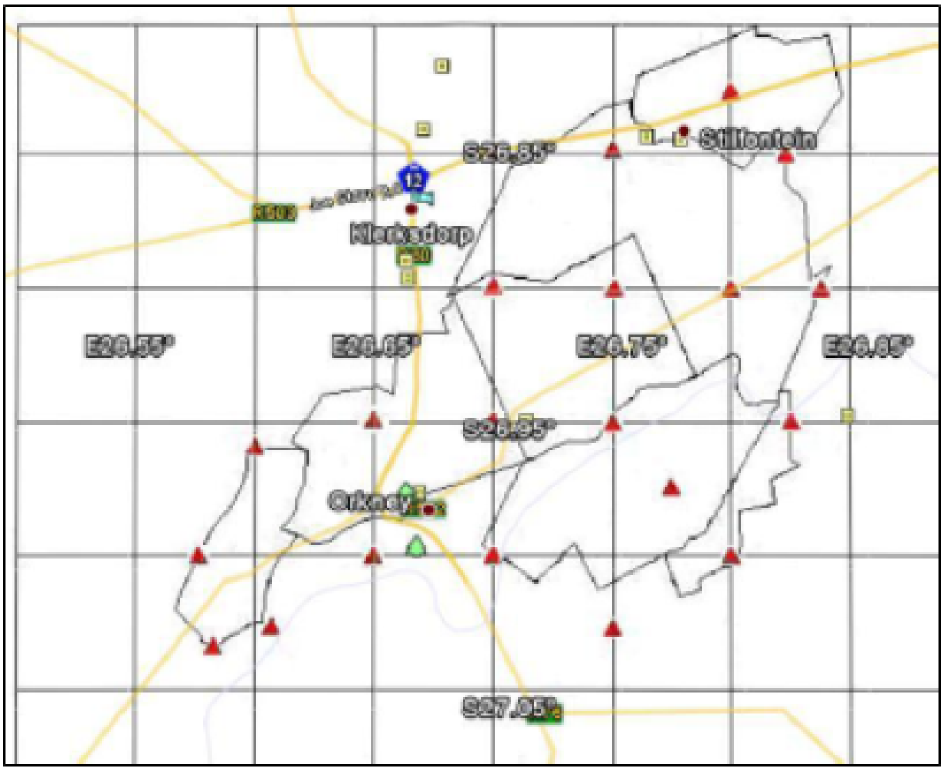

Figure 1: Left: Witwatersrand Basin Cluster (SANSN data from 1620 to 2008); Right: Planned network layout in the KOSH region (with permission from CGS)

If current plans are implemented, the SANSN will include 12 stations in the Johannesburg area, 10 in the Far West Rand and a further 25 in the Klerksdorp region (Figure 1). The main purpose of the SANSN is to fill a 'monitoring gap' that exists in the large-event range due to the sensors deployed in mines, which are aimed at recording small magnitude sources down to M=-1 or even lower. These sensors are unable to record large magnitude events equally well as sensors are built for a specific, limited frequency range. The SANSN's broad-band technology is suitable for seismic events of roughly M>2.0, a range not well covered by mine sensors. The other objective is to bring information pertaining to mining-induced seismicity into the public domain. The public, as the Stilfontein quake and several tremors in the Free State have shown, may be negatively affected by this phenomenon.

- Council for Geoscience for more information

The following chapters cover the concepts applied to seismic system layout and the key performance characteristics of such networks. The purpose is to provide a practical guide to the planning of a seismic monitoring strategy based on needs and explains required steps up to the design of a network, which will then be able to cover those needs.

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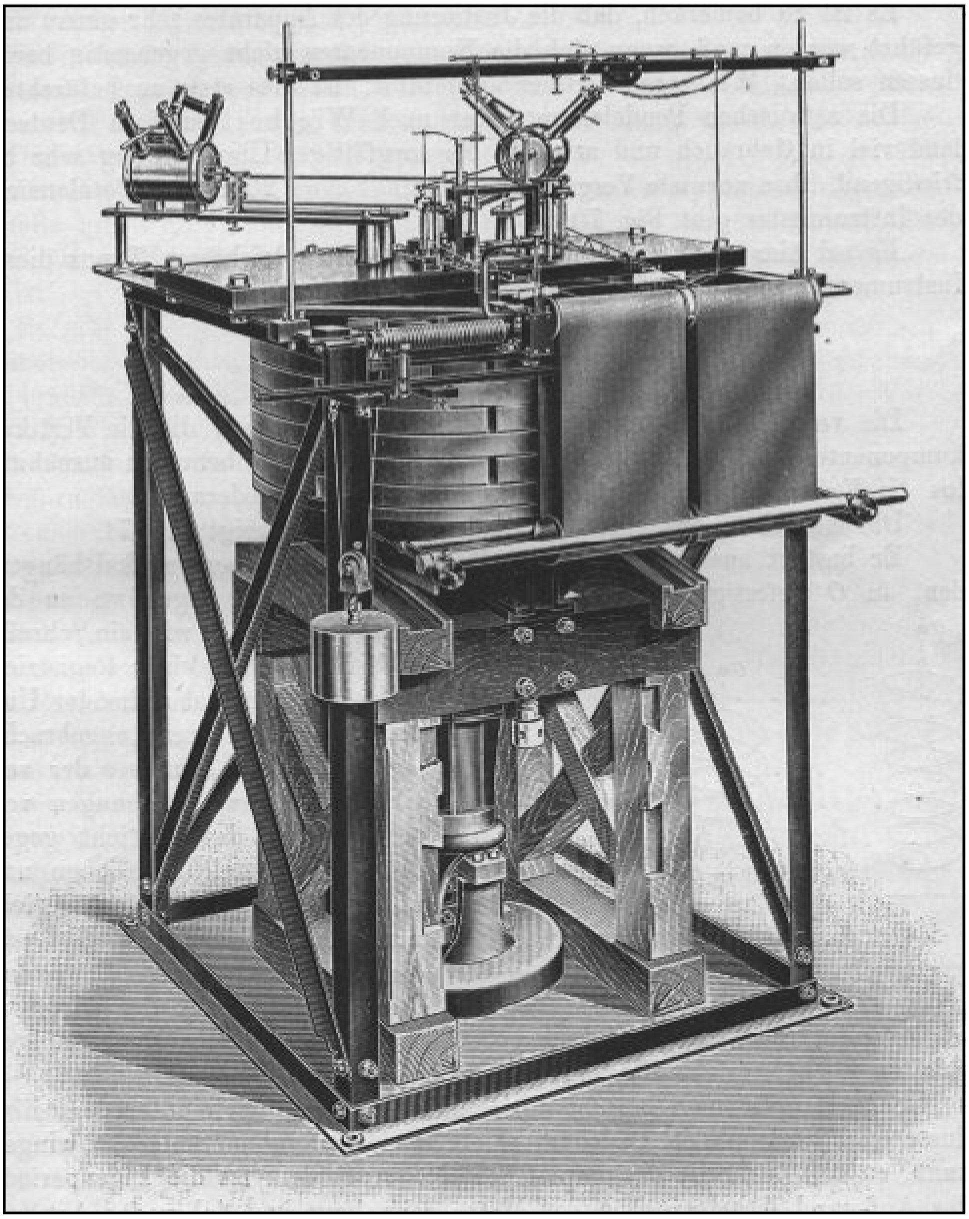

Figure 2: Seismograph installed at the Union Observatory in Johannesburg (1910, courtesy Africana Museum; Durrheim, 2010)

Figure 3: SANSN broad-band seismometer (surface mounted)

Figure 4: Tri-axial sensor set used in mines (grouted into bore hole, courtesy IMS)

Seismic system design

Quote from "Minimising the rockburst risk (Phase 2, output 3)", SIM050302, Durrheim et al., 2007:

There is often a fundamental difference between civil and mining engineering projects (Stacey et al., 2006). The design requirements for a civil engineering project (e.g. a road tunnel or dam) are usually clearly defined, and usually require a conservative design as the projects are usually of a long-term nature and open to the public. In contrast, many mining excavations are temporary or short term, not open to the public, and the viability of a project is usually critically sensitive to costs.

The legal requirements formulated in the MHSA stipulate the obligation by mines to monitor and mitigate seismic risk, but there are also operational needs that should be considered when planning the installation of a seismic system. A stakeholder survey should be conducted initially

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when setting up seismic information distribution procedures. Stakeholder requirements are central to the setting up of a seismology service as they determine the information type and reporting procedures seen to add value to the operation. A seismic system design and operation should be oriented on these needs.

The process of installing a seismic system on a mine with seismic rockmass response begins with the formulation of seismic monitoring objectives: What is the purpose of seismic system operation and what kind of seismic network would be adequate for a given mine? In answering these questions, one will come across the notion that not all parts of a mine require equal coverage, usually because the mine consists of active production areas with high worker exposure, back areas with little or no exposure, and areas where vertical, inclined or horizontal access ways are concentrated. As a result, the required network coverage varies spatially.

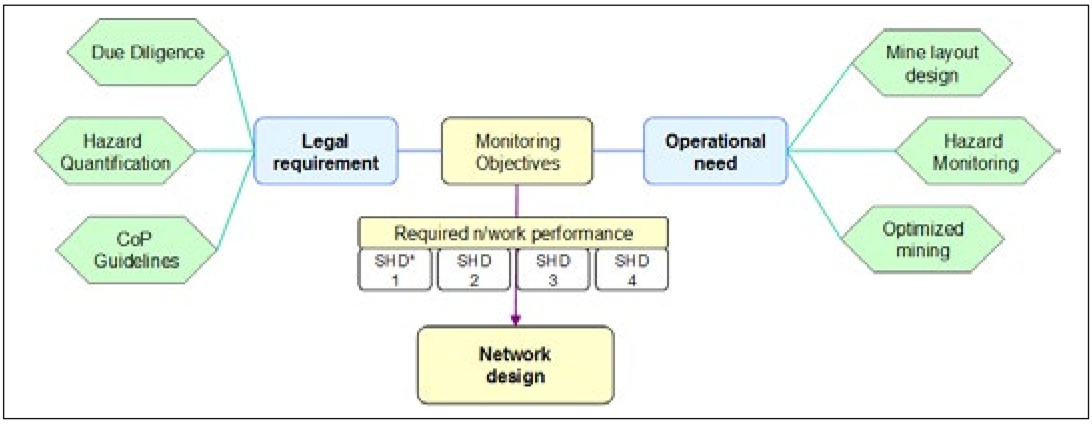

Figure 5: Process of defining monitoring objectives for each seismic hazard district (SHD), based on legal requirements and stakeholder needs

NOTE

How to evaluate historic seismicity:

- *rock engineering reports of wall instability and FOG incidents;*
- *rock-related injuries;*
- *eye-witness accounts of shaking, rock-talk, side-wall ejection, footwall heave, crushing, and bursting;*
- *seismic information gathered by neighbouring mines;*
- *seismic informat*ion available from the SANSN.

The study materials for REC paper 2 introduced a set of standard monitoring objectives widely accepted for hard-rock tabular mines, in line with the guide to routine seismic monitoring in mines (Mendecki et al., 1999) in the Handbook for Rock Engineering Practice (Jager & Ryder, 1999):

- 'Location': Hypocentral co-ordinates and magnitude to pinpoint the affected area and assist with rescue operations, if necessary. This often refers to a location obtained through automatic processing of a large event.
- 'Prevention': Study of the relationship between (seismic) failure and design; feed back into the mine's planning process to optimise sequence and layouts
- 'Control': Search for medium-term changes in seismicity, e.g. clustering in space or increases in activity rate over days; check

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for compliance and sub-standard conditions; possible reaction by reducing mining rates or temporarily suspending shifts.

- 'Warning': Detection of short-term changes in seismic source parameters that constitute precursory behaviour; analysis of seismic data in terms of rockmass instability and imminent failure
- 'Back-analysis': Detailed analysis of single, important events in terms of mining, geology and damage caused.

CONNECTION 28

Refer to Paper2, Outcome 6.3

These are the basic objectives and others may be added, such as the monitoring of known seismically active faults and dykes, for instance in the crown pillar; the monitoring of specific high-grade ore zones earmarked for extraction; or the detection of discontinuities ahead of the stope in caving applications. The seismic network design has to make provision for these monitoring objectives in the designated parts of the operation in which they have to be met (see flow chart in Figure 5 above).

In order to ensure adequate network location accuracy and sensitivity, an initial sensor layout and data communication plan is drafted, which is then tested against the performance benchmarks derived from the above process. Location error in the reef plane and in the vertical direction is modelled using suitable software (Figure 6). Where necessary, seismic sensors can be added or relocated until the design matches the requirements.

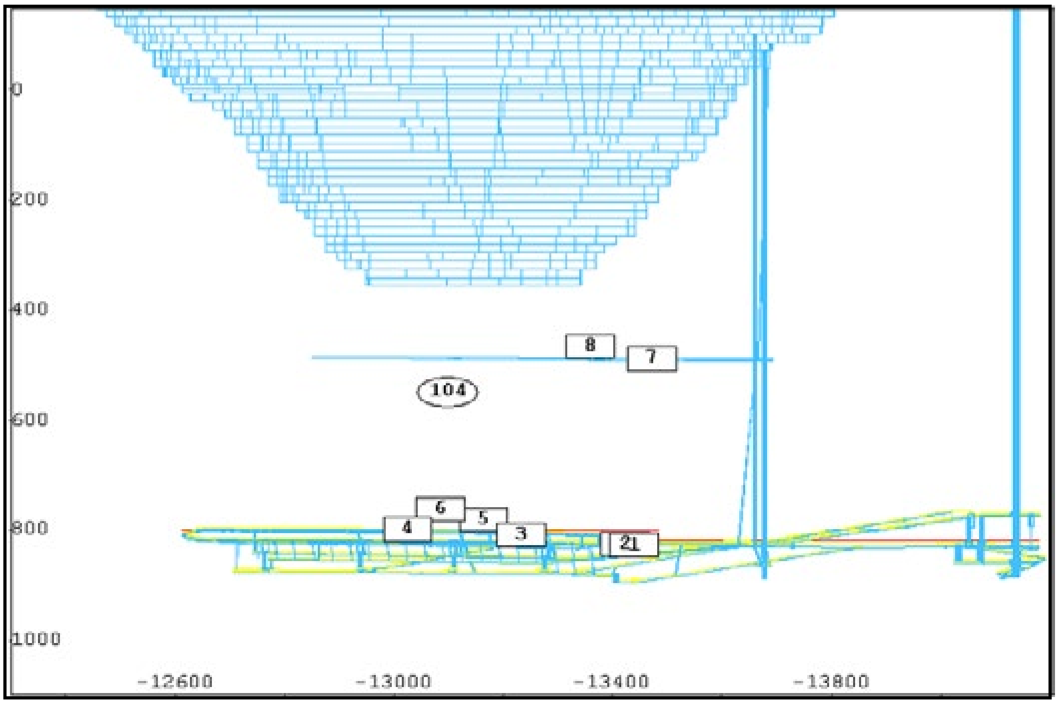

Figure 6: Seismic stations monitoring an underground operation beneath an open pit

At this stage of the network planning, two possible constraints need to be considered: Access to the planned sensor sites, and budgetary constraints, should they exist. Both of these may affect the design and both can change with time. Seismic networks do not only need to be purchased and installed, but also maintained for the duration of the monitoring period. Furthermore, roadways, shafts and tunnels currently maintained and safe may not be safe or accessible in a few years' time. It is essential to consider both these aspects during the planning phase.

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NOTE

- *sensitivity (...is the magnitude threshold above which all events are reliably recorded)*
- *location error (...measure of the network's ability to locate a seismic source accurately)*
- *source parameter quantification (is reliable and accurate where events are recorded by a large array of fully functional tri-axial sensors)*
- *station down-time (...reflects on the overall state of health of a network and the quality of upgrades and upkeep)*
- *% rejected events (...is an indicator of network health and data processing standards)*

CONCEPTUAL APPROACH TO OUTCOMES 6.2.4.7- 6.2.4.9:
Refer to other papers as shown below.

LEARNING OUTCOME 6.2.4.7

CONNECTION 29
Refer to
Paper 3.1, Outcome 6.2.2.13

DESCRIBE, DISCUSS AND EXPLAIN THE FACTORS THAT NEED TO BE CONSIDERED IN PLACING SEISMOMETERS

LEARNING OUTCOME 6.2.4.8

CONNECTION 30
Refer to
Paper 3.1, Outcome 6.2.2.14

EXPLAIN AND DISCUSS HOW SEISMOMETER PLACEMENT COULD AFFECT THE EVENT LOCATION ACCURACY OF A SEISMIC SYSTEM

LEARNING OUTCOME 6.2.4.9

CONNECTION 31
Refer to
Paper 3.1, Outcome 6.2.2.15

EXPLAIN AND DISCUSS HOW SEISMOMETER PLACEMENT COULD AFFECT THE SENSITIVITY OF A SEISMIC SYSTEM

OUTCOMES 6.2.4.10-6.2.4.14
Conceptual approach to SO39-43:

Principles of seismic system audits: needs-based data collection, analysis and reporting; quality management principles to be an effective service.

LEARNING OUTCOME 6.2.4.10

CONNECTION 32
Refer to
Paper 3.1, Outcome 6.2.2.16

DESCRIBE, DISCUSS AND EXPLAIN RELEVANT PROCEDURES TO ENSURE THAT SEISMIC MONITORING INFORMATION IS DISTRIBUTED EFFECTIVELY TO STAKEHOLDERS

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LEARNING OUTCOME 6.2.4.11

CONNECTION 33
Refer to
Paper 3.1, Outcome 6.2.2.17

DESCRIBE, DISCUSS AND EXPLAIN METHODS OF QUALITY ASSURANCE IN SEISMIC SYSTEM MANAGEMENT

LEARNING OUTCOME 6.2.4.12

CONNECTION 34
Refer to
Paper 3.1, Outcome 6.2.2.18

DESCRIBE AND DISCUSS RELEVANT ASPECTS OF SEISMIC SYSTEM AUDITS

LEARNING OUTCOME 6.2.4.13

CONNECTION 35
Refer to
Paper 3.1, Outcome 6.2.2.19

DESCRIBE AND DISCUSS METHODS TO ASSESS STAKEHOLDER NEEDS

LEARNING OUTCOME 6.2.4.14

CONNECTION 36
Refer to
Paper 3.1, Outcome 6.2.2.20

DESCRIBE AND DISCUSS METHODS TO IDENTIFY POSSIBLE SHORTFALLS BETWEEN SEISMIC MONITORING OBJECTIVES AND SEISMIC SYSTEM PERFORMANCE.

System auditing

NOTE

"Quality has to be managed – it does not just happen." [John S Oakland, 2003]

Contrary to common perception, the quality of a product is not defined in absolute terms. Rather, the needs and expectations of the user determine whether a product is seen to be of high or low quality.

DEFINITION

Quality: Totality of features and characteristics of a service or product that bear on its ability to satisfy stated or implied needs [ISO 8402, 1986]

To ensure that a seismic system is managed at high quality standards is equivalent to ensuring that the declared objectives of seismic monitoring are being met.
Principles of seismic system audits are based on quality management procedures developed in the 1980s and now well established in manufacturing and service industries worldwide. Some of the first comprehensive texts were authored by WE Deming (1982), PB Crosby (1979) and JM Juram (1988). Over the past twenty years, comprehensive total quality management (TQM) systems have been devised, which integrate stakeholders, customers, suppliers and processes into fully-managed systems with the aim to achieve optimal results for all parties involved.

The generation of seismic data and the flow of analysis results on mines usually affect three parties: The system manufacturer, the rock engineering department, and the main customer groups on the mine and in head office:

- The system supplier usually designs, installs and commissions the seismic system and often takes responsibility for maintenance and data analysis as well;
- the rock engineering department manages the process of seismic monitoring and reporting;

- and the customers receive reports and other updates on various aspects of seismicity within their respective areas of responsibility.

A review of seismic systems should therefore be widened to include all product and service providers, management processes and all stakeholders. The central process under scrutiny is that of data interpretation (Figure 7). This process receives, among other equipment, raw data and methodologies on the input side and produces information, documents, knowledge and compliance on the output side. Two important feedback functions connect customers to the core process (quality of service received) and the interpretation process to the input side (optimisation of inputs).

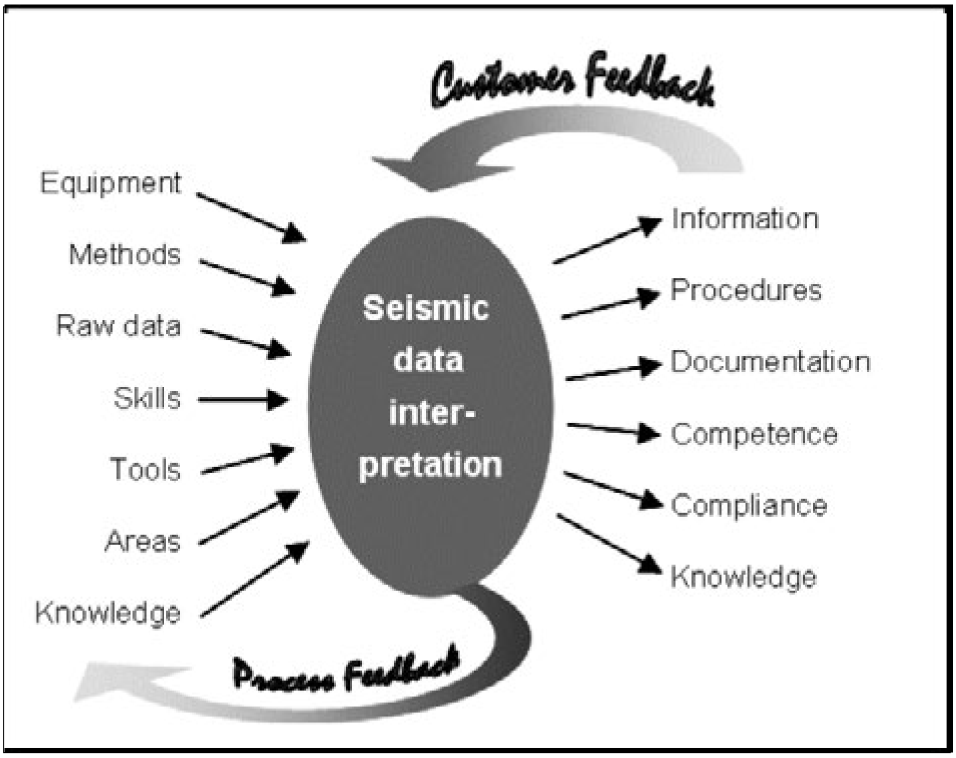

Figure 7: Seismic data interpretation process (Essrich, 2006): Inputs (left), outputs (right) and relevant feedback functions from the customer group and from the process back to the inputs

Initially, when setting up the seismology service, stakeholders should have been informed of standard seismic data analysis techniques and they should be enabled to interpret the results of such analyses. The main stakeholder groups are mine management, service departments such as rock engineering, geology and mineral resource management, and production personnel. Among the service departments, the primary customers of a seismology service are rock engineering personnel who should be trained in matters related to mining-induced seismicity. Common and most suitable means of seismic information distribution to various interested groups include:

- Monthly and quarterly planning/review meetings; safety and other meetings;
- Placing of seismicity reports on intranet websites or servers accessible to all employees;
- Notification via phone, e-mail and SMS;
- Displays on notice boards, e.g. at the u/g working place;
- Printed and bound report collections on a specific subject, and circulars;
- Informal verbal communications.

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To ensure effective information distribution, stakeholders should be awarded an opportunity to provide feedback on the perceived service level. Corrections can then be made to procedures where necessary.

A generally accepted principle in achieving optimum quality targets it that of 'never-ending improvement'. No process is expected to perform optimally right from the start. Instead, a continuous cycle of improvements is the way to achieve highest performance, and the steps within each cycle are:

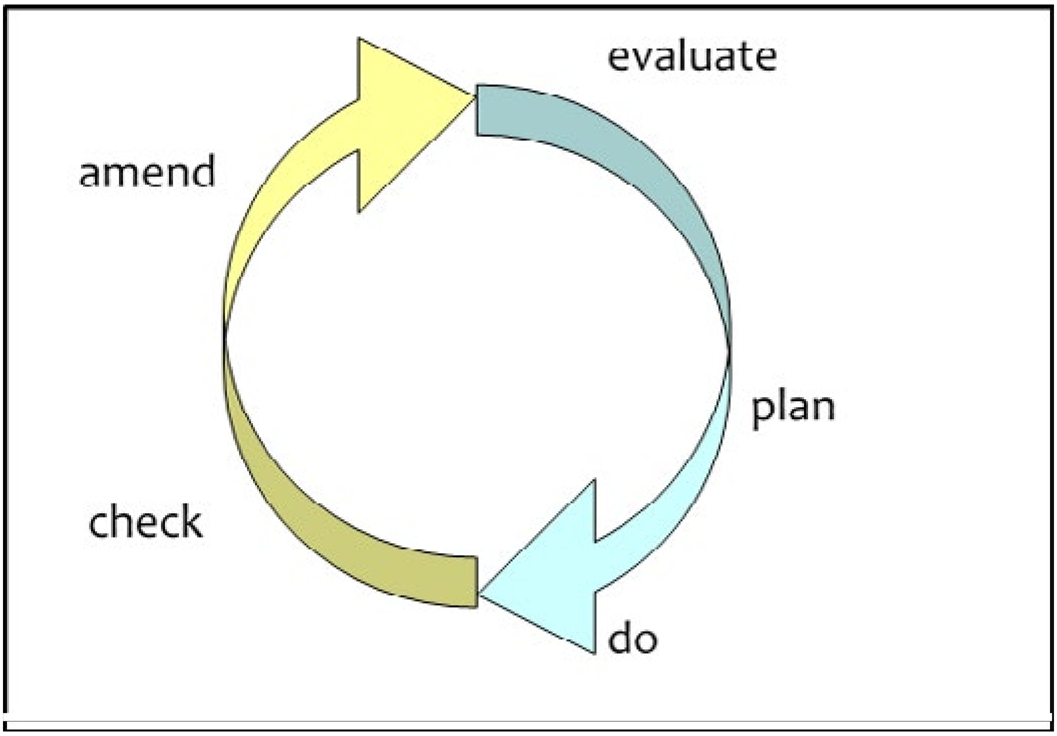
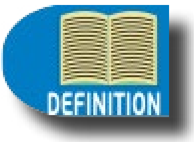


Figure 8: Cycle of improvement

In other words, analyse the current performance (evaluate), consider appropriate improvements (plan), implement improvements (do), check the effect the changes had and finally amend the process. After some time, re-evaluate the system and start over (sketch). This is seen as the proven way to achieve an efficient and effective process flow that satisfies the customer's needs.



Efficient = Maximum impact with minimum effort
Effective = Achieves the desired result

In the case of a mine seismic service review, it is suggested to consider and evaluate three main parts:

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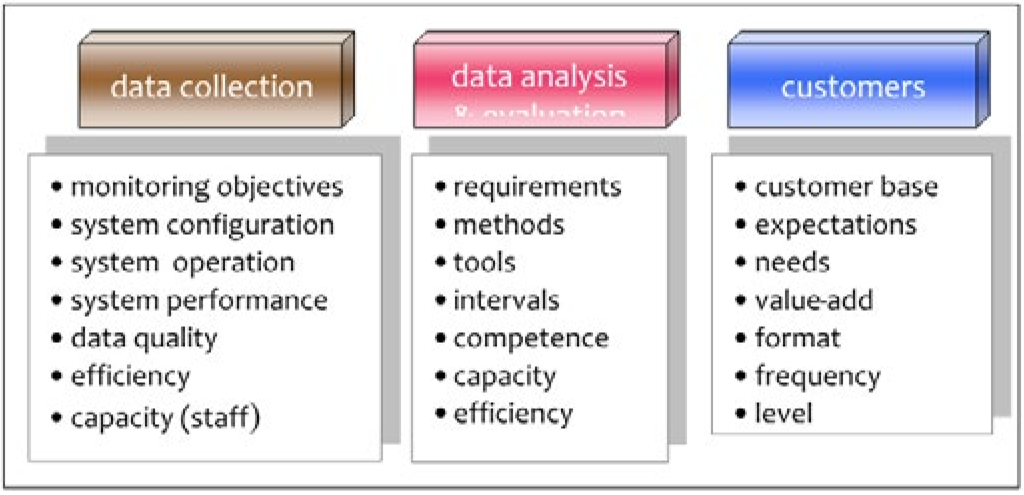

Figure 9: Main aspects of a seismic service review

It is evident that all three parts are interdependent in that customer needs determine the type and format of information that should be produced by the seismic department, which, in turn, determines the type of raw data needed from the network. Similarly, should the network or its components be impaired, this would negatively impact on the reports delivered to customers.

A customer questionnaire is a good way to quantify the satisfaction levels achieved by the seismology service. Questions could be formulated such that customers agree/disagree whether seismic information is useful and whether it is provided in the appropriate format. Figure 10 shows the results of such an exercise.

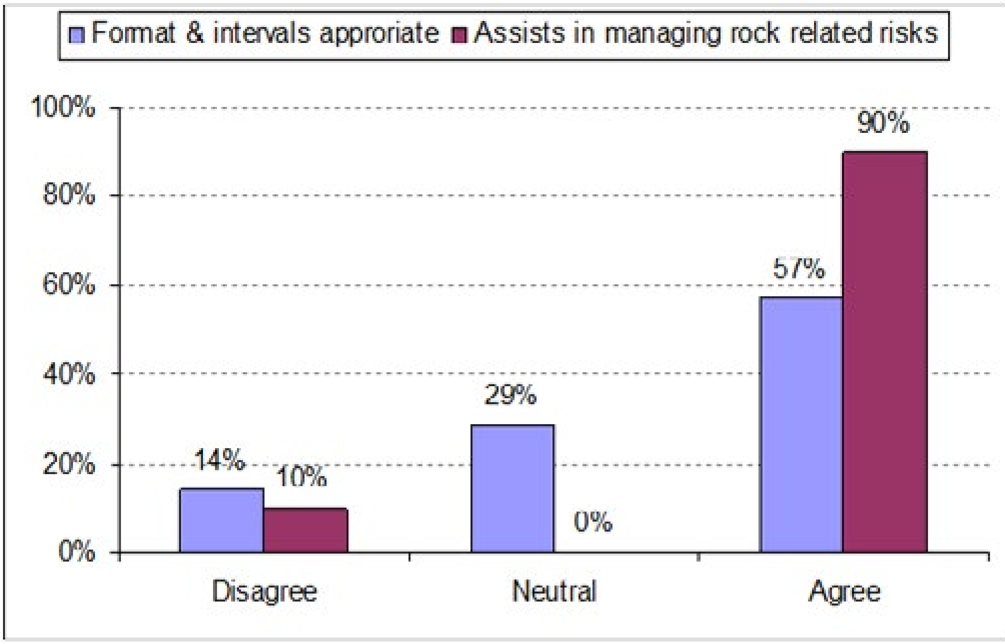

Figure 10: Customer response to two questions related to the format and usefulness of seismic information

For the data analysis and interpretation part of the audit, reviewers need to evaluate the appropriateness of analysis methods, the tools used to conduct analyses, the complement and competence of staff and other capacity-related issues. As audit findings show, routine data analysis is not always performed by adequately qualified staff and the results are not always understood by the recipients (training needs!).

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The same might be true for the technical support and maintenance function on mines, which can negatively affect data quality. Mines often do not have trained seismic network technicians and rather outsource this function to the network supplier or an associated agency. As a result, the maintenance function is not adequately managed and is subject to budget constraints; and one chosen way to achieve savings is to reduce technician time on the shaft.

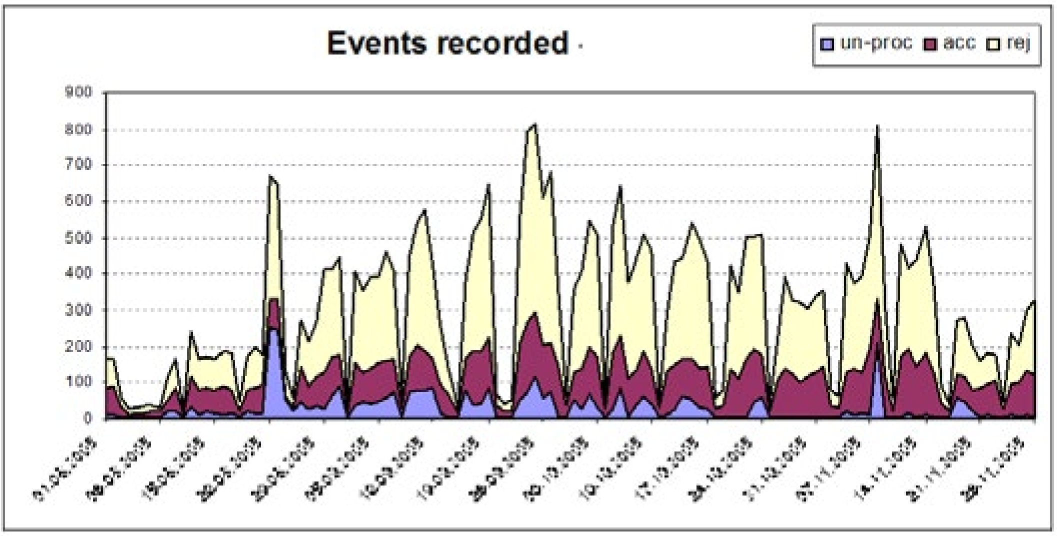

Figure 11: Daily percentage of unprocessed, accepted and rejected events (minimum levels correspond to weekends)

Common symptoms of poor hard- and software maintenance are faulty sensors, poor data communication between seismic stations and surface, sub-optimal settings of system parameters and an overall inadequate level of data quality. In some cases, the percentage of recorded but rejected events exceeds the percentage of accepted events in the database, an issue that may have a number of interlinked causes. In the example in Figure 11, 50 to 70% of all events recorded during the review period were rejected.

As an example of possible causes underlying levels of poor seismic data, the functionality of seismic sensors installed underground needs to be considered. In an example from seven mines in the various operations of a large ore producer, 20 to 50% of sensors may be impaired or faulty, which negatively impacts on the value the mines receive from their respective seismic systems. Sensor health is a crucial element of quality data collection, on par with optimal system settings and sound data processing procedures.

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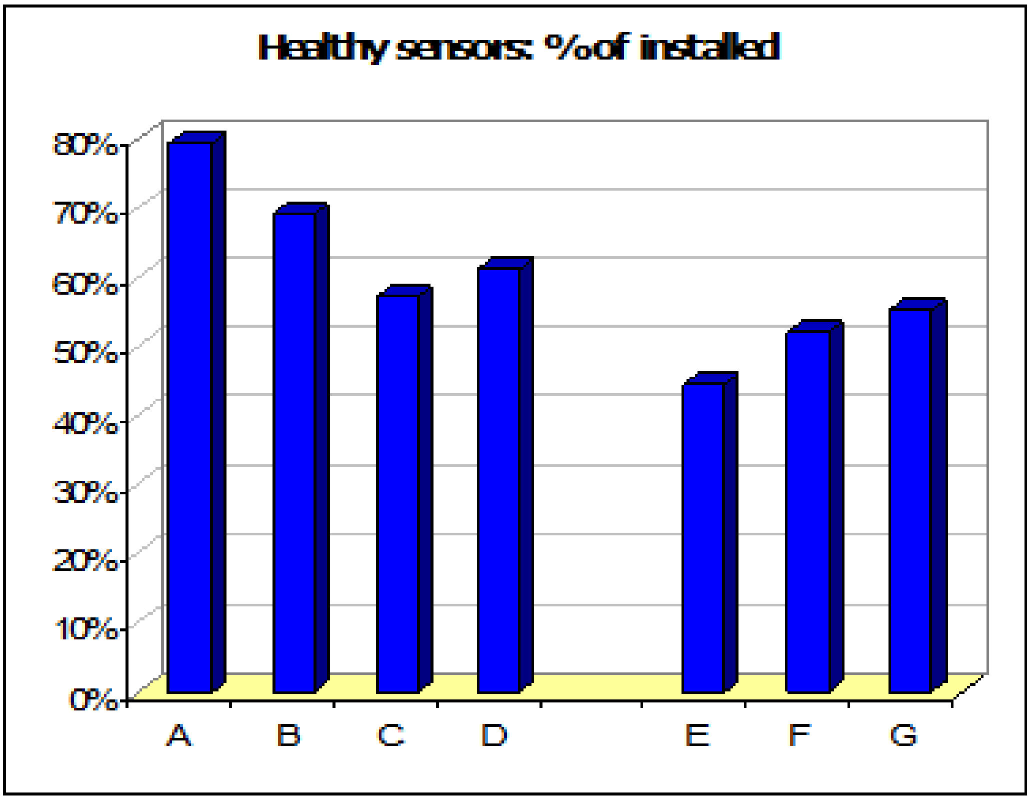

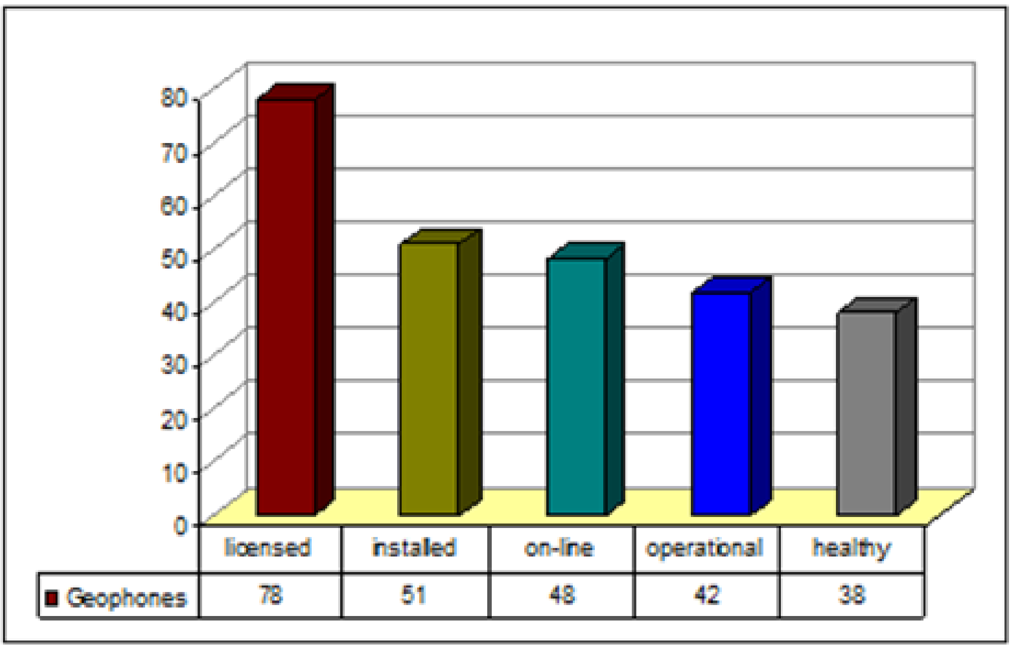

Figure 12: (a) Percentage of installed sensors considered fully functional (7 mines), (b) Number of geophones licensed, installed, on-line, operational and healthy

On some mines, the number of healthy sensors in a network can be less than half the number of licensed sensors (Figure 12b). Licensed sensors are those for which the mine purchased operating licenses and pays technical support for. Of course, once a sensor site was approved and hard- and software for its operation was purchased, it should have been installed and commissioned. Only a functional, healthy sensor adds value to the operation and generates a return on investment.

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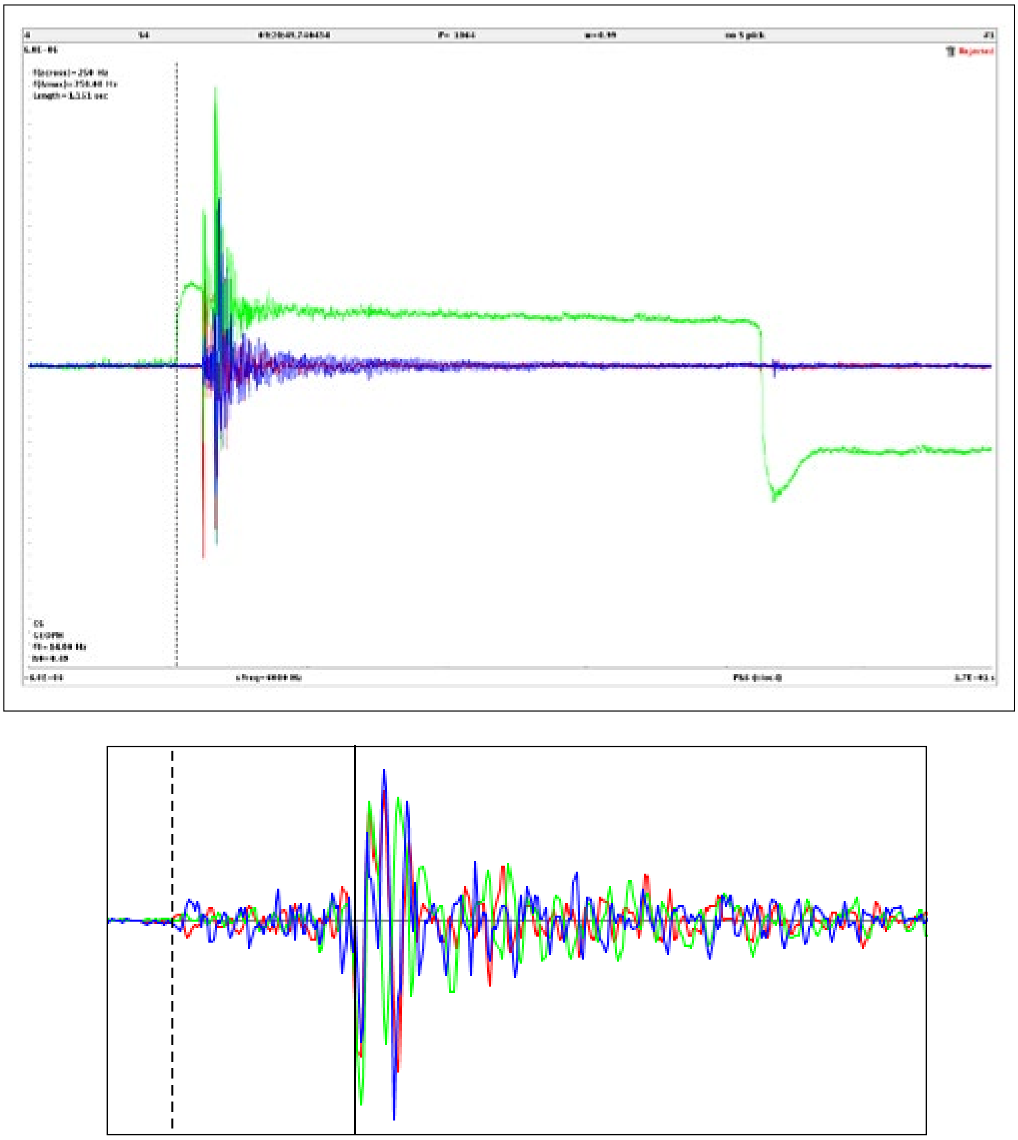

Figure 13: (a) Faulty green component in a tri-axial sensor set, (b) Fully functional tri-axial set

Within the context of value creation, it should be noted that, according to surveys conducted over the past ten years, the average cost per recorded, processed and accepted seismic event lies in the range of R0.10 to R40 for some of the deep level gold mines. It is therefore important that seismic system operations are optimised to the full extent of their technical possibilities and that all parts of a mine seismology service, from data collection through analysis to reporting, are effective, efficient and relevant.

6.3. MODELLING

6.3.1. NUMERICAL MODELLING

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

CONNECTION 37
Refer to
Paper 2, Outcome 6.4
Paper 3.1, Outcome 6.3
Paper 3.2, Outcome 6.3
Paper 3.4, Outcome 6.5.1

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LEARNING OUTCOME 6.3.1.1

DESCRIBE, EXPLAIN AND DISCUSS THE SELECTION OF APPROPRIATE CODES TO TACKLE VARIOUS PROBLEMS

CONNECTION 38
Refer to
Paper 3.1, Outcome 6.3.3

LEARNING OUTCOME 6.3.1.2

DESCRIBE, EXPLAIN AND DISCUSS THE INPUT OF APPROPRIATE PARAMETERS TO INVESTIGATE VARIOUS PROBLEMS

CONNECTION 39
Refer to
Paper 3.1, Outcome 6.3.4

LEARNING OUTCOME 6.3.1.3

DESCRIBE, EXPLAIN AND DISCUSS THE INTERPRETATION OF OUTPUT IN THE INVESTIGATION OF VARIOUS PROBLEMS

CONNECTION 40
Refer to
Paper 3.1, Outcome 6.3.5

6.4. AUDITING

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

LEARNING OUTCOME 6.4.1

DESCRIBE, EXPLAIN AND DISCUSS THE CONCEPT OF MONITORING FOR UNDERSTANDING, PREDICTION AND DESIGN

CONNECTION 41
Refer to
Paper 2, Outcome 4.4
Paper 3.1, Outcome 6.4
Paper 3.2, Outcome 6.4
Paper 3.4, Outcome 6.7

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CHAPTER

7

PAPER 3.3: MASSIVE UNDERGROUND MINING (HARD AND SOFT ROCK)

LEARNING OUTCOMES

7. ROCKBREAKING IN MASSIVE MINES
7.1. CUTTING TECHNIQUES

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

- Describe, explain and discuss geotechnical aspects associated with various non-explosive rock breaking procedures that include:
  - Tunnel boring, raise boring;
- Describe, explain and discuss the methodologies and applications of these techniques; and
- List and discuss the advantages and disadvantages of these techniques.

7.2. DRILLING TECHNIQUES

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

- Sketch, describe, explain and discuss the different rounds used in shaft sinking;
- Describe, explain and discuss the different cuts used in shaft sinking;
- Describe, explain and discuss the types of initiation used in the above rounds;
- Describe, explain and discuss the sequence of initiation of blast holes used in the above rounds;
- Sketch, describe, explain and discuss the different rounds used in tunnel development;
- Describe, explain and discuss the different cuts used in tunnel development;
- Describe, explain and discuss the types of initiation used in the above rounds;
- Describe, explain and discuss the sequence of initiation of blast holes used in the above rounds;
- Sketch, describe, explain and discuss blast hole layouts in massive stopes;
- Describe, explain and discuss the drilling of blast holes in from drilling drives in massive sub-level open stopes;
- Describe, explain and discuss the drilling of blast holes in from drilling drives in massive vertical crater retreat stopes;

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- Describe, explain and discuss the direction of drilling of blast holes in massive stopes;
- Describe, explain and discuss the sequence of initiation of blast holes in massive stopes;
- Describe, explain and discuss the importance of blast-hole drilling accuracy in the following applications:
  - Shaft sinking, chamber excavation, tunnel development
  - Massive stoping operations,
  - Cushion blasting, smooth blasting.

7.3. BLASTING PRACTICES

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

- List and discuss the advantages and disadvantages of these techniques;
- Evaluate and determine blasting requirements for tunnels;
- Describe, explain and discuss the effect of the following parameters on blast damage:
  - Explosive type, initiation method,
  - Initiation sequence, hole orientation;
- Describe, explain and discuss the objectives and effects of de-coupling explosives;
- Describe, explain and discuss the methods by which de-coupling of explosives is achieved;
- Describe, explain and discuss the following excavation cushion blasting and smooth blasting techniques:
  - Pre-splitting,
  - Concurrent smooth blasting,
  - Post-splitting;
- Describe, explain and discuss the methodologies and typical applications of each technique;
- Evaluate and determine appropriate blasting rounds to suit given conditions in tunnels;
- Evaluate and determine appropriate explosive types to suit given conditions in tunnels;
- Evaluate and determine blasting requirements for stopes making use of knowledge of explosives;
- Evaluate and determine appropriate blasting rounds to suit given conditions in stopes; and
- Evaluate and determine appropriate explosive types to suit given conditions.

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CONNECTION 1
Refer to
Paper 2, Outcome 7
Paper 3.1, Outcome 7.1
Paper 3.2, Outcome 7.1
Paper 3.4, Outcome 7.1

7. ROCK BREAKING IN MASSIVE MINES
7.1. CUTTING TECHNIQUES

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

LEARNING OUTCOME 7.1.1

CONNECTION 2
Refer to
Paper 3.1, Outcome 7.1.1

DESCRIBE, EXPLAIN AND DISCUSS GEOTECHNICAL ASPECTS ASSOCIATED WITH VARIOUS NON-EXPLOSIVE ROCK BREAKING PROCEDURES THAT INCLUDE:

- **Tunnel boring;**
- **Raise boring**

LEARNING OUTCOME 7.1.2

CONNECTION 3
Refer to
Paper 3.1, Outcome 7.1.2

DESCRIBE, EXPLAIN AND DISCUSS THE METHODOLOGIES AND APPLICATIONS OF THESE TECHNIQUES

LEARNING OUTCOME 7.1.3

CONNECTION 4
Refer to
Paper 3.1, Outcome 7.1.3

LIST AND DISCUSS THE ADVANTAGES AND DISADVANTAGES OF THESE TECHNIQUES

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7.2. DRILLING TECHNIQUES

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

CONNECTION 5
Refer to
Paper 2, Outcome 7.2
Paper 3.1, Outcome 7.2
Paper 3.2, Outcome 7.2
Paper 3.4, Outcome 7.2

LEARNING OUTCOME 7.2.1

SKETCH, DESCRIBE, EXPLAIN AND DISCUSS THE DIFFERENT ROUNDS USED IN SHAFT SINKING

CONNECTION 6
Refer to
Paper 3.1, Outcome 7.2.1

LEARNING OUTCOME 7.2.2

DESCRIBE, EXPLAIN AND DISCUSS THE DIFFERENT CUTS USED IN SHAFT SINKING

CONNECTION 7
Refer to
Paper 3.1, Outcome 7.2.2

LEARNING OUTCOME 7.2.3

DESCRIBE, EXPLAIN AND DISCUSS THE TYPES OF INITIATION USED IN THE ABOVE ROUNDS

CONNECTION 8
Refer to
Paper 3.1, Outcome 7.1.3

LEARNING OUTCOME 7.2.4

DESCRIBE, EXPLAIN AND DISCUSS THE SEQUENCE OF INITIATION OF BLAST HOLES USED IN THE ABOVE ROUNDS

CONNECTION 9
Refer to
Paper 3.1, Outcome 7.2.4

LEARNING OUTCOME 7.2.5

SKETCH, DESCRIBE, EXPLAIN AND DISCUSS THE DIFFERENT ROUNDS USED IN TUNNEL DEVELOPMENT

CONNECTION 10
Refer to
Paper 3.1, Outcome 7.2.5

LEARNING OUTCOME 7.2.6

DESCRIBE, EXPLAIN AND DISCUSS THE DIFFERENT CUTS USED IN TUNNEL DEVELOPMENT

CONNECTION 11
Refer to
Paper 3.1, Outcome 7.2.6

LEARNING OUTCOME 7.2.7

DESCRIBE, EXPLAIN AND DISCUSS THE TYPES OF INITIATION USED IN THE ABOVE ROUNDS

CONNECTION 12
Refer to
Paper 3.1, Outcome 7.2.7

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LEARNING OUTCOME 7.2.8

CONNECTION 13

Refer to
Paper 3.1, Outcome 7.2.8

Describe, explain and discuss the sequence of initiation of blast holes used in the above rounds

LEARNING OUTCOME 7.2.9

CONNECTION 14

Refer to
Paper 3.1, Outcome 7.2.9

SKETCH, DESCRIBE, EXPLAIN AND DISCUSS BLAST HOLE LAYOUTS IN MASSIVE STOPES

LEARNING OUTCOME 7.2.10

DESCRIBE, EXPLAIN AND DISCUSS THE DRILLING OF BLAST HOLES IN FROM DRILLING DRIVES IN MASSIVE SUB-LEVEL OPEN STOPES

Drill patterns are essential in ensuring:

- The required fragmentation;
- Limiting damage to the host rockmass and therefore dilution;
- Position of the final blasted material for loading practices;
- Complete extraction of the ore body etc.

NOTE

A drill pattern design is normally approved by different operations/in-divuduals so that all impacts of the blast can be evaluated prior to the round beng drilled.

Critical issues in the drilling process include:

- The shape of the ore body: Can an appropriate drill pattern (end of hole burdens, etc.) be drilled from the existing drill drives to reach the final required stope shape?
- Equipment: Can the equipment provide the required length and diameter as well as accuracy required?
- Drill control: Is the drill pattern finally created as was planned?

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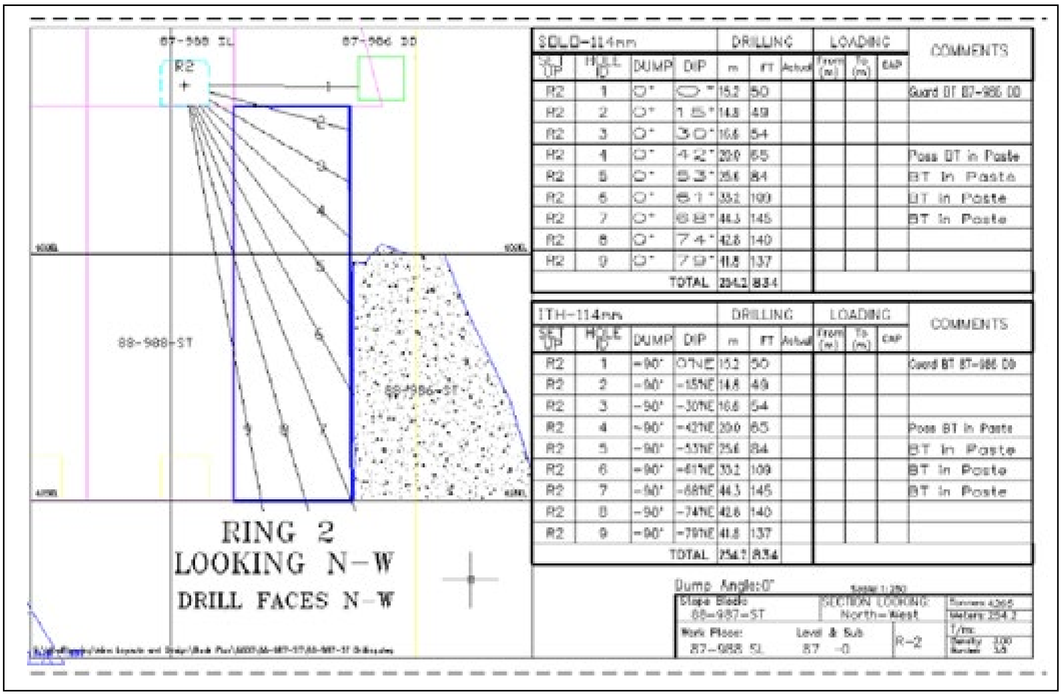

Figure 1: Drill control sheet (Courtesy of University of Saskatchewan)

- Page 518, Brady and Brown, Rock mechanics for underground mines, 2006

The drill patterns required for massive ore body stopes are a factor of the ore body properties and should therefore be designed for each specific environment.
However, Tables 1 through 6 show general drill patterns for the different mining methods.

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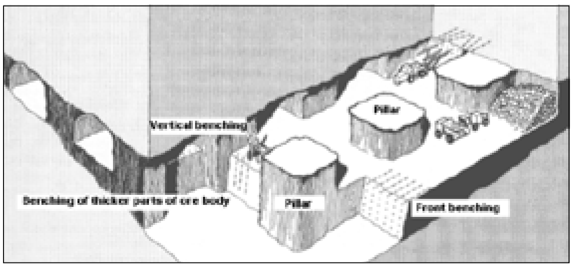
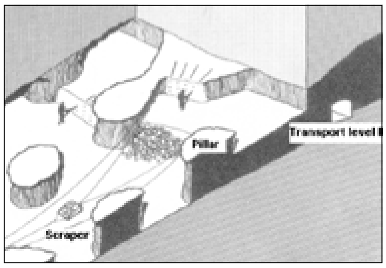
Table 1: Bord-and pillar drill patterns (Courtesy of University of Saskatchewan)





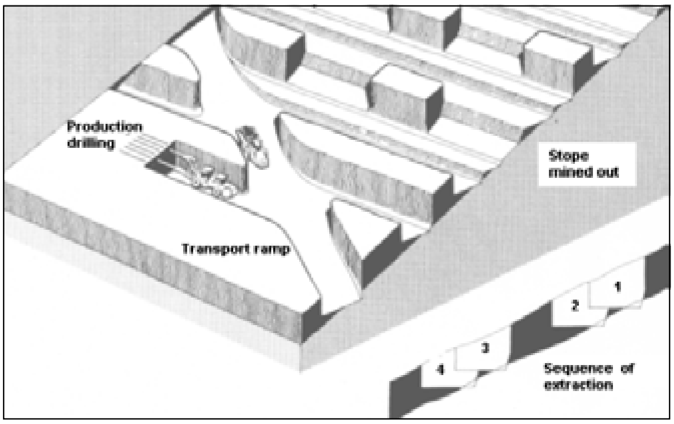
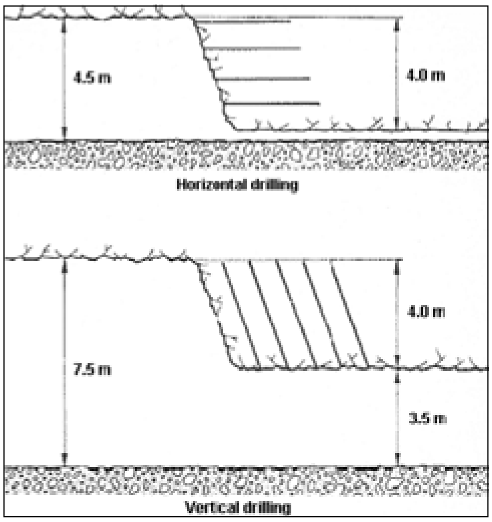
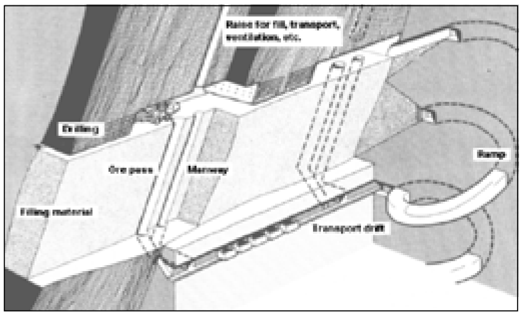


Table 2: Cut-and-fill drill patterns (Courtesy of University of Saskatchewan)





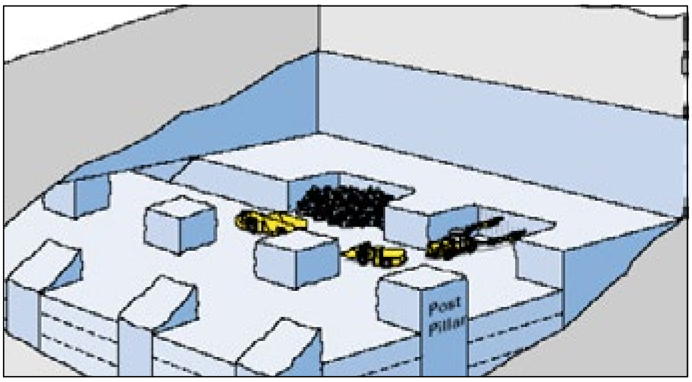


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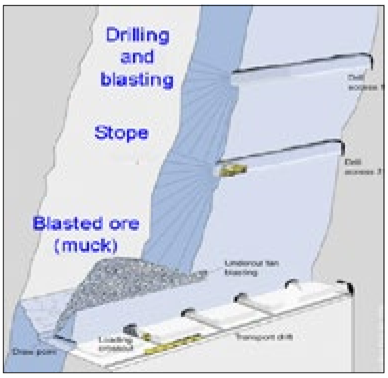
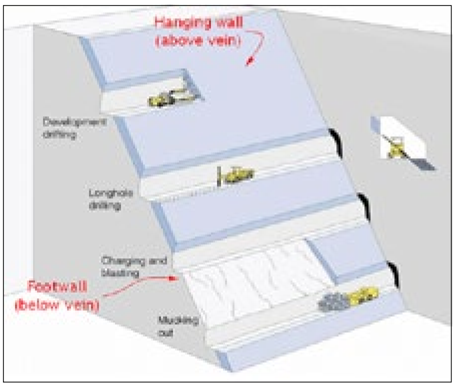
Table 3: Open stope drill patterns (Courtesy of University of Saskatchewan)





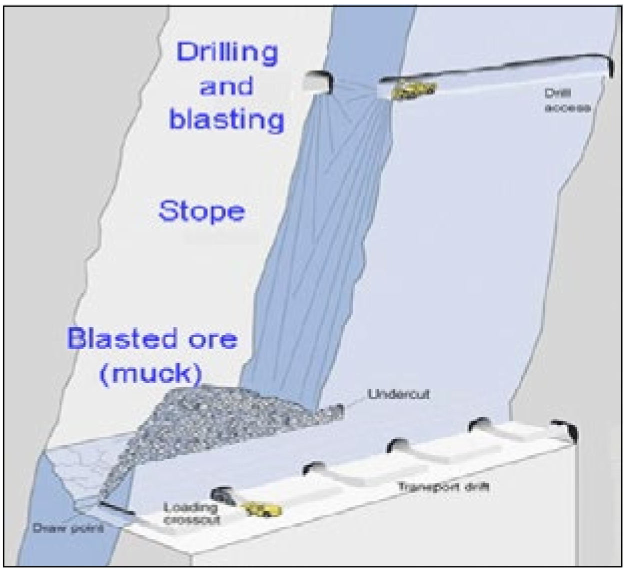


Table 4: VCR drill patterns (Courtesy of University of Saskatchewan)

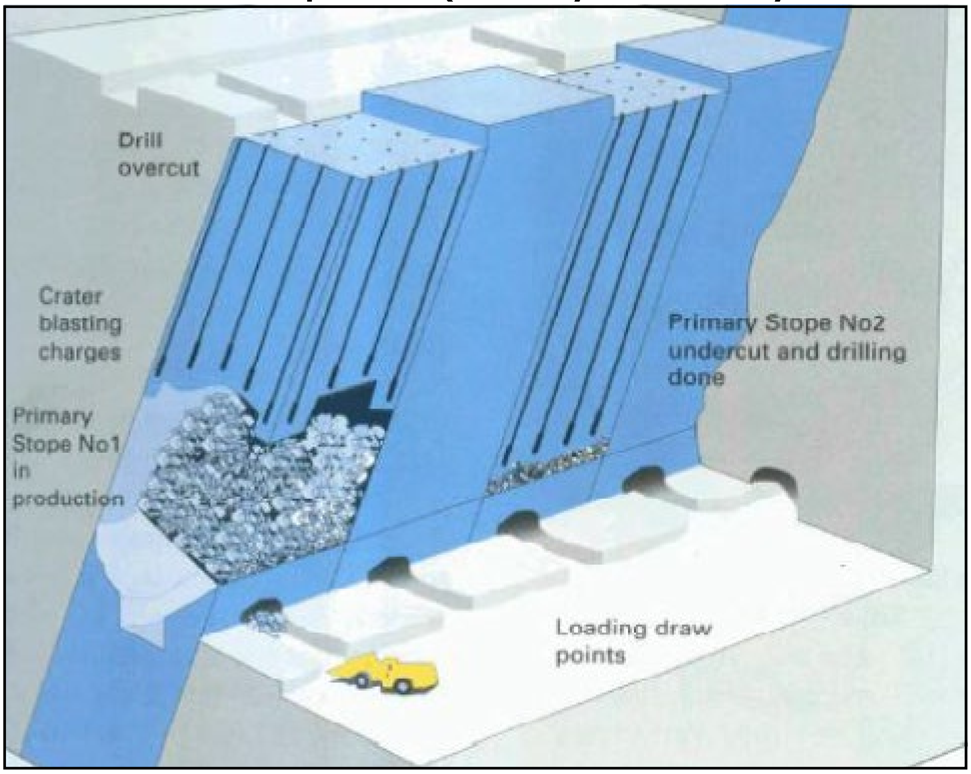


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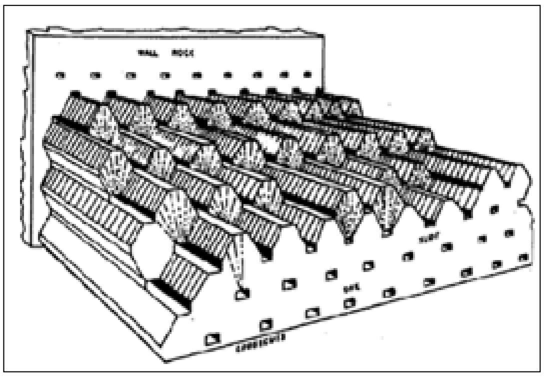
Table 5: Sub-level caving drill patterns (Courtesy of University of Saskatchewan)



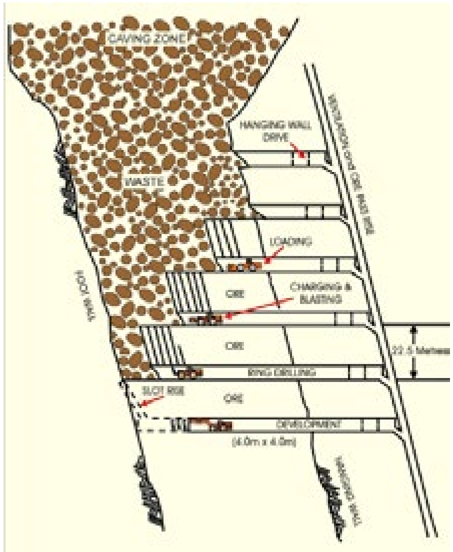
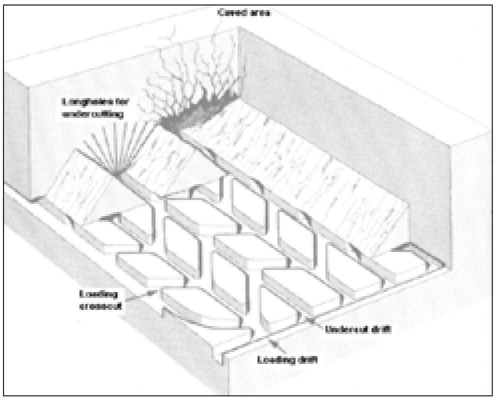
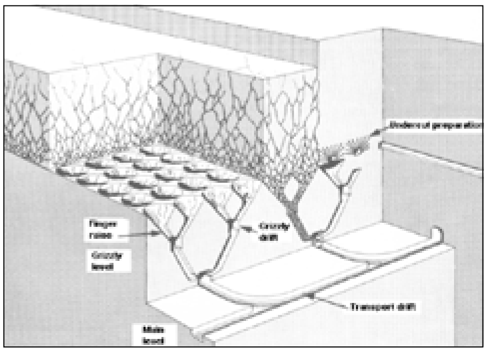
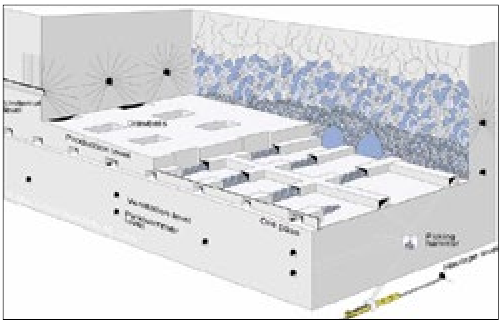
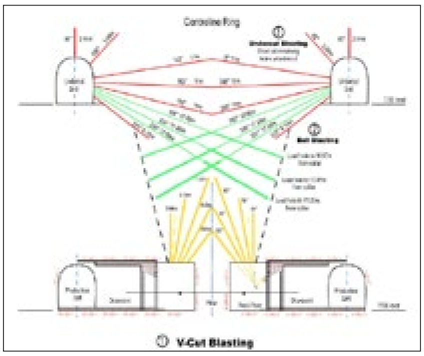


Table 6: Block cave drill patterns (Courtesy of University of Saskatchewan and Petra Diamonds)









- Fundamentals of blast design Bender
- Sub-level open-stoping design Sloane
- Underground mining methods Hamrin
- Underground blasting
- Drilling systems in open stopes Oraee

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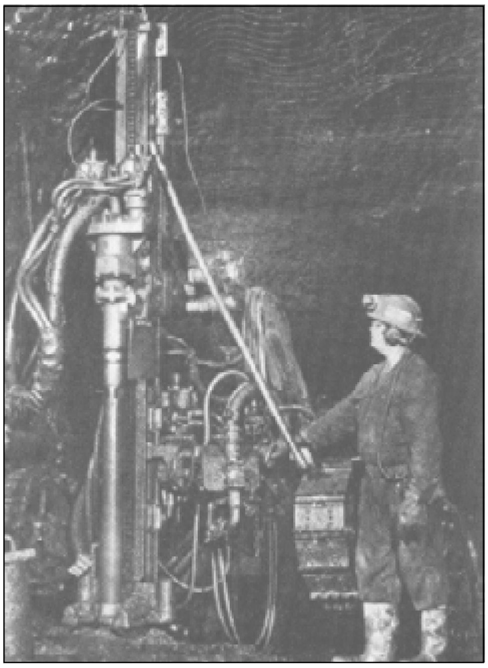
- Mining engineering handbook Chapter 19.2 Excl. techniques
- All mining methods and equipment
- Block caving overview Petra
- Improved technology for long hole blast hole Ferreira
- Keeping blast holes open Connor
- Long-hole stoping in Turkey Gonen

INTERESTING INFO

Drilling equipment

See Mining equipment drilling slide (p98) in All mining methods and equipment presentation

Table 8: Drilling equipment (Courtesy of University of Saskatchewan)

 Jumbo - Development	 Jumbo - Development
 Long hole drill	 Long hole drill

REFERENCE

- Page Brady and Brown, Rock mechanics for underground mines, 2006

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LEARNING OUTCOME 7.2.11

CONNECTION 15

Refer to "Learning Outcome 7.2.10" on page 355

DESCRIBE, EXPLAIN AND DISCUSS THE DRILLING OF BLAST HOLES IN FROM DRILLING DRIVES IN MASSIVE VERTICAL CRATER RETREAT STOPES

LEARNING OUTCOME 7.2.12

CONNECTION 16

Refer to "Learning Outcome 7.2.10" on page 355

DESCRIBE, EXPLAIN AND DISCUSS THE DIRECTION OF DRILLING OF BLAST HOLES IN MASSIVE STOPES

LEARNING OUTCOME 7.2.13

DESCRIBE, EXPLAIN AND DISCUSS THE SEQUENCE OF INITIATION OF BLAST HOLES IN MASSIVE STOPES

REFERENCE

- Fundamentals of blast design Bender

LEARNING OUTCOME 7.2.14

CONNECTION 17

Refer to Paper 3.1, Outcome 7.2.15

DESCRIBE, EXPLAIN AND DISCUSS THE IMPORTANCE OF BLAST-HOLE DRILLING ACCURACY IN THE FOLLOWING APPLICATIONS:

- Shaft sinking, chamber excavation, tunnel development
- Massive stoping operations
- Cushion blasting, smooth blasting

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CONNECTION 18

Refer to
Paper 2, Outcome 7.3
Paper 3.1, Outcome 7.3
Paper 3.2, Outcome 7.3
Paper 3.4, Outcome 7.3

7.3. BLASTING PRACTICES

Explosive charges are placed in blast holes and detonated. Rock surrounding the charges is fragmented and displaced by the impulsive loading, generated by the sudden release of the explosive's potential energy.

NOTE

Underground mining in hard-rock settings is highly dependent on the successful execution of blasting procedures. Although shaft borers and raise borers are often used in developing vertical access to and within an ore body, the majority of development is still undertaken with methods of drilling and blasting.

An explosive is any material or device that can produce a sudden outburst of gas, applying a high impulsive loading to the surrounding medium.

INTERESTING INFO

Chemical explosives are of two types:

- deflagrating explosives: burn (rather than explode) relatively slowly and produce relatively low pressures;
- detonating (high) explosives: very fast reaction rates and produce high blast hole pressures.

The detonating explosives are considered in three categories based on their respective sensitivities to ease of detonation:

- Primary explosives: Are initiated by spark or impact, are highly unstable and are used only in initiating devices such as blasting caps;
- Secondary explosives: Require the use of a blasting cap for initiation and in some cases may require an additional booster charge;

Tertiary explosives:
- Are insensitive to initiation by a standard (No. 6) strength blasting cap.

The response of the rock to explosives is determined by the explosive properties and the rock properties:

Explosive property:

Its strength;

Rock properties:

The capacity of the rockmass to transmit energy is related to the Young's modulus (stiffness) of the rock; the ease of generating fractures is a function of the strength of the rock (uniaxial compressive strength C0); while the unit weight of the rock affects the energy required to displace the fragmented rock.

Figure 2 (Brady & Brown, 2006) indicates that:

- For very low average fracture spacing (very jointed) or very low UCS, the rockmass may be ripped or dug mechanically, rather than blasted;
- ANFO is a suitable explosive for use in a wide range of rockmass conditions;

The application of high VOD explosives is justified only in the strongest, more massive rock formations.

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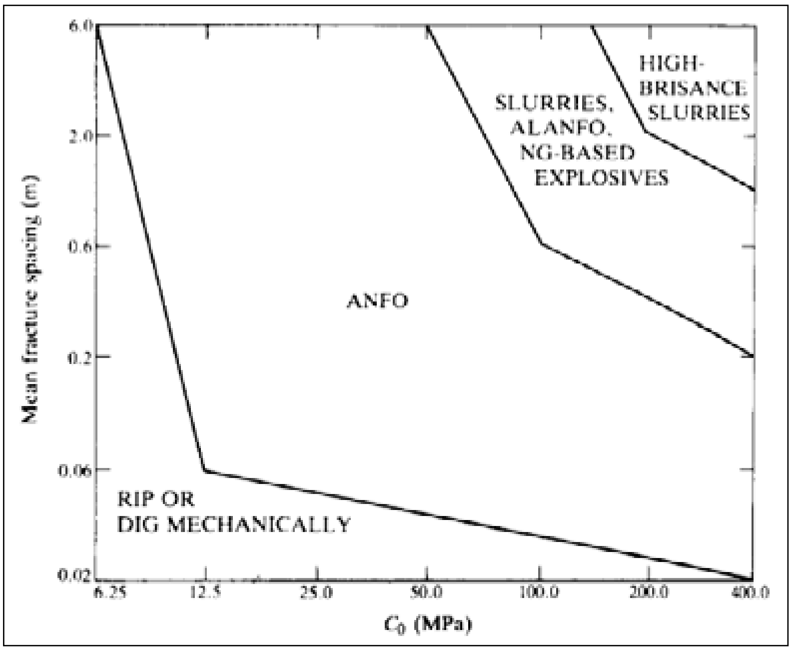

Figure 2: Rock properties and explosive types (Brady & Brown, 2006)

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

LEARNING OUTCOME 7.3.1

LIST AND DISCUSS THE ADVANTAGES AND DISADVANTAGES OF THESE TECHNIQUES

Syllabus incomplete

LEARNING OUTCOME 7.3.2

CONNECTION 19

Refer to Paper 3.1, Outcome 7.2 & 7.3

EVALUATE AND DETERMINE BLASTING REQUIREMENTS FOR TUNNEL MAKING

REFERENCE

Underground blasting

LEARNING OUTCOME 7.3.3

CONNECTION 20

Refer to Paper 3.1, Outcome 7.3.4

DESCRIBE, EXPLAIN AND DISCUSS THE EFFECT OF THE FOLLOWING PARAMETERS ON BLAST DAMAGE:

- explosive type;
- initiation method;,
- Initiation sequence;
- hole orientation

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LEARNING OUTCOME 7.3.4

CONNECTION 21
Refer to
Paper 3.1, Outcome 73.5

DESCRIBE, EXPLAIN AND DISCUSS THE OBJECTIVES AND EFFECTS OF DE-COUPLING EXPLOSIVES

LEARNING OUTCOME 7.3.5

CONNECTION 22
Refer to
Paper 3.1, Outcome 7.3.6

DESCRIBE, EXPLAIN AND DISCUSS THE METHODS BY WHICH DE-COUPLING OF EXPLOSIVES IS ACHIEVED

LEARNING OUTCOME 7.3.6

CONNECTION 23
Refer to
Paper 3.1, Outcome 7.3.7

DESCRIBE, EXPLAIN AND DISCUSS THE FOLLOWING EXCAVATION CUSHION BLASTING AND SMOOTH BLASTING TECHNIQUES:

- **Pre-splitting**
- **Concurrent smooth blasting**
- **Post-splitting**

LEARNING OUTCOME 7.3.7

CONNECTION 24
Refer to
Paper 3.1, Outcome 7.3.8

DESCRIBE, EXPLAIN AND DISCUSS THE METHODOLOGIES AND TYPICAL APPLICATIONS OF EACH TECHNIQUE

LEARNING OUTCOME 7.3.8

CONNECTION 25
Refer to
Paper 3.1, Outcome 7.3.11

EVALUATE AND DETERMINE APPROPRIATE BLASTING ROUNDS TO SUIT GIVEN CONDITIONS IN TUNNELS

LEARNING OUTCOME 7.3.9

CONNECTION 26
Refer to
Paper 3.1, Outcome 7.3.2

EVALUATE AND DETERMINE APPROPRIATE EXPLOSIVE TYPES TO SUIT GIVEN CONDITIONS IN TUNNELS

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LEARNING OUTCOME 7.3.10

CONNECTION 27

Refer to
"Learning Outcome 7: Rock breaking in massive mines" on page 353

EVALUATE AND DETERMINE BLASTING REQUIREMENTS FOR STOPES MAKING USE OF KNOWLEDGE OF EXPLOSIVES

LEARNING OUTCOME 7.3.11

CONNECTION 28

Refer to
"Learning Outcome 7: Rock breaking in massive mines" on page 353

EVALUATE AND DETERMINE APPROPRIATE BLASTING ROUNDS TO SUIT GIVEN CONDITIONS IN STOPES

LEARNING OUTCOME 7.3.12

CONNECTION 29

Refer to
"Learning Outcome 7: Rock breaking in massive mines" on page 353

EVALUATE AND DETERMINE APPROPRIATE EXPLOSIVE TYPES TO SUIT GIVEN CONDITIONS IN STOPES

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CHAPTER
8

PAPER 3.3: MASSIVE UNDERGROUND MINING (HARD AND SOFT ROCK)

LEARNING OUTCOMES

8. SURFACE & ENVIRONMENTAL EFFECTS

8.1. SURFACE EFFECTS

8.1.1. SUBSIDENCE ENGINEERING

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

- Describe, explain and discuss the following terms within the context of surface subsidence:
    - Angle of draw, curvature, tilt, critical span,
    - Horizontal strain, vertical subsidence, differential subsidence;
- Describe, explain and discuss the following surface expressions of subsidence:
    - Tension cracks, compression humps, ridges, thrusts;
- Describe, explain and discuss how mining height-to-depth ratio affects the type and severity of surface subsidence; and
- Describe, explain and discuss the effects of geological structures on surface subsidence.

8.1.2. SURFACE PROTECTION

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

- Describe, explain and discuss how the following surface features are affected by subsidence:
    - Roads, buildings, pylons, lands, streams, pans;
- Describe, explain and discuss possible remedial measures that may be applied to surface structures to limit subsidence damage;
- Describe, explain and discuss possible changes that may be made to underground mining layouts to reduce subsidence damage; and
- Determine potential subsidence damage to the following types of structure for given mining depths and mining heights using published damage tables:
    - Roads, buildings, pylons

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8.2. ENVIRONMENTAL EFFECTS

8.2.1. LONG-TERM STABILITY AND THE ENVIRONMENT

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

- Describe, explain and discuss the possible effects and consequences of given mining methods on the following issues:
  - Long-term stability of the ground surface,
  - Groundwater,
  - Ultimate closure of the mine; and
- Describe, explain and discuss the possible effects and consequences of given factors of safety on the following issues:
  - Long-term stability of the ground surface,
  - Groundwater,
  - Ultimate closure of the mine.

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8. SURFACE & ENVIRONMENTAL EFFECTS
8.1. SURFACE EFFECTS
8.1.1. SUBSIDENCE ENGINEERING

CONNECTION 1
Refer to
Paper 3.1, Outcome 8.1
Paper 3.2, Outcome 8.1.1

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

LEARNING OUTCOME 8.1.1.1

CONNECTION 2
Refer to
Paper 3.1, Outcome 8.1.1
Paper 3.2, Outcome 8.1.11

DESCRIBE, EXPLAIN AND DISCUSS THE FOLLOWING TERMS WITHIN THE CONTEXT OF SURFACE SUBSIDENCE:

- **Angle of draw, curvature, tilt, critical span**
- **Horizontal strain, vertical subsidence, differential subsidence**

LEARNING OUTCOME 8.1.1.2

CONNECTION 3
Refer to
Paper 3.2, Outcome 8.1.1.2

DESCRIBE, EXPLAIN AND DISCUSS THE FOLLOWING SURFACE EXPRESSIONS OF SUBSIDENCE:

- Tension cracks,
- Compression humps,
- Ridges, thrusts

LEARNING OUTCOME 8.1.1.3

CONNECTION 4
Refer to
Paper 3.2, Outcome 8.1.1.3

DESCRIBE, EXPLAIN AND DISCUSS HOW MINING HEIGHT-TO-DEPTH RATIO AFFECTS THE TYPE AND SEVERITY OF SURFACE SUBSIDENCE

LEARNING OUTCOME 8.1.1.4

CONNECTION 5
Refer to
Paper 3.2, Outcome 8.1.1.4

DESCRIBE, EXPLAIN AND DISCUSS THE EFFECTS OF GEOLOGICAL STRUCTURES ON SURFACE SUBSIDENCE.

CONNECTION 6
Refer to
Paper 3.1, Outcome 8.2
Paper 3.2, Outcome 8.1.3

8.1.2. SURFACE PROTECTION

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

LEARNING OUTCOME 8.1.2.1

DESCRIBE, EXPLAIN AND DISCUSS HOW THE FOLLOWING SURFACE FEATURES ARE AFFECTED BY SUBSIDENCE:

- Roads,
- Buildings,
- Pylons,

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- Lands,
- Streams,
- Pans

Refer to
Paper 3.2, Outcome 8.1.3.1

LEARNING OUTCOME 8.1.2.2

CONNECTION 7

Refer to
Paper 3.2, Outcome 8.1.3.2

DESCRIBE, EXPLAIN AND DISCUSS POSSIBLE REMEDIAL MEASURES THAT MAY BE APPLIED TO SURFACE STRUCTURES TO LIMIT SUBSIDENCE DAMAGE

LEARNING OUTCOME 8.1.2.3

CONNECTION 8

Refer to
Paper 3.2, Outcome 8.1.3.3

DESCRIBE, EXPLAIN AND DISCUSS POSSIBLE CHANGES THAT MAY BE MADE TO UNDERGROUND MINING LAYOUTS TO REDUCE SUBSIDENCE DAMAGE

LEARNING OUTCOME 8.1.2.4

CONNECTION 9

Refer to
Paper 3.2, Outcome 8.1.3.4

DETERMINE POTENTIAL SUBSIDENCE DAMAGE TO THE FOLLOWING TYPES OF STRUCTURE FOR GIVEN MINING DEPTHS AND MINING HEIGHTS USING PUBLISHED DAMAGE TABLES:

- Roads,
- Buildings,
- Pylons

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CONNECTION 10
Refer to
Paper 3.2, Outcome 8..2.1

8.2. ENVIRONMENTAL EFFECTS

8.2.1. LONG-TERM STABILITY AND THE ENVIRONMENT

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

LEARNING OUTCOME 8.2.1.1

CONNECTION 11
Refer to
Paper 3.2, Outcome 8.2.1.1

DESCRIBE, EXPLAIN AND DISCUSS THE POSSIBLE EFFECTS AND CONSEQUENCES OF GIVEN MINING METHODS ON THE FOLLOWING ISSUES:

- Long-term stability of the ground surface
- Groundwater
- Ultimate closure of the mine

LEARNING OUTCOME 8.2.1.2

CONNECTION 12
Refer to
Paper 3.2, Outcome 8.2.1.2

DESCRIBE, EXPLAIN AND DISCUSS THE POSSIBLE EFFECTS AND CONSEQUENCES OF GIVEN FACTORS OF SAFETY ON THE FOLLOWING ISSUES:

- Long-term stability of the ground surface
- Groundwater
- Ultimate closure of the mine.

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CHAPTER

9

PAPER 3.3: MASSIVE UNDERGROUND MINING (HARD AND SOFT ROCK)

LEARNING OUTCOMES

9. MINING STRATEGIES IN DIFFICULT CIRCUMSTANCES

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

- Describe, explain and discuss the geotechnical aspects of dealing with the following difficult circumstances:
  - Mining through geological structures or disturbances,
  - Mining in localised disturbed, weak or poor ground conditions,
  - Mining in localised high or anomalous stress situations,
  - Mining in areas of high water pressures and/or large water inflows,
  - Remnant mining,
  - Pillar mining,
  - Rehabilitation of previously mined areas,
  - Dealing with excessive overbreak situations, and
  - Dealing with runaway caving situations.

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9. MINING STRATEGIES IN DIFFICULT CIRCUMSTANCES

The candidate must be able to demonstrate knowledge and understanding of the above subject area by being able to:

LEARNING OUTCOME 9.1

DESCRIBE, EXPLAIN AND DISCUSS THE GEOTECHNICAL ASPECTS OF DEALING WITH THE FOLLOWING DIFFICULT CIRCUMSTANCES:

- Mining through geological structures or disturbances
- Mining in localised disturbed, weak or poor ground conditions
- Mining in localised high or anomalous stress situations
- Mining in areas of high water pressures and/or large water inflows
- Remnant mining
- Pillar mining
- Rehabilitation of previously mined areas.
- Dealing with excessive overbreak situations
- Dealing with runaway caving situations

Mining through geological structures or disturbances

The correct angle of attack of a tunnel or development to a fault dyke or shear zone would be at right angles (Figure 1), unless no flexibility to change the tunnel direction exists, at which time the installation of adequate support measures to ensure that any potentially unstable blocks formed by the structure are well supported (wedge failures). In most cases, the intersection with major structures is also associated with increased joint densities and therefore poorer ground conditions. Precautions that could assist in maintaining stability include the reductions of spans and/or the increase in support densities and/or the use of areal support practices such as shotcrete, thin skin liners or mesh.

Mining directions in a bord and pillar stope are pre-set and cannot be changed as geological structures are encountered, suggesting that the precautions as for development ends (above) would have to suffice.

In massive stopes where personnel do not enter, the intersection of structures mostly creates only a dilution concern. Wedges/poorer conditions created by the intersection of structures with a stope can be addressed by creating smaller spans, leaving well-planned pillars and/or the use of cable anchors.

In stopes where personnel entry is essential, such as cut-and-fill methods, the structures and associated conditions can be addressed by precautions similar to that applied to development ends, i.e. increased support densities, areal coverage support.

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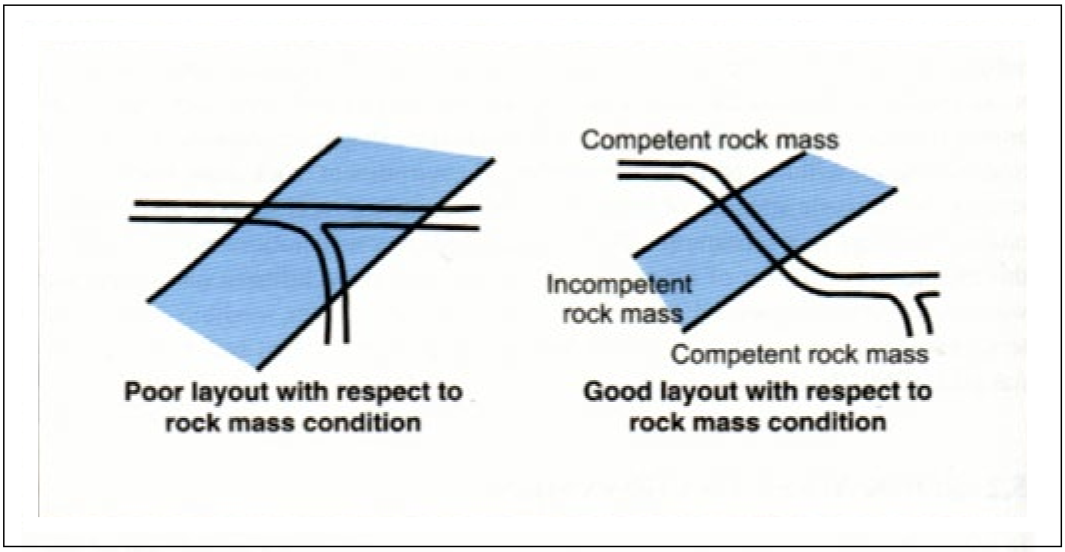

Figure 1: Development through geological structures

Mining in localised high or anomalous stress situations

Highly stressed conditions in development ends should be handled as follows:

- Re-route the end if possible to avoid the high induced stress levels;
- Remove the zone of high induced stress by mining (de-stressing); or
- Upgrade support practices to address the potential for increased fracturing around the ends, increased wall closure and possibly the increased risk of seismic damage.

For stopes, the avoidance of highly stressed zones is mostly only possible if a planned mining block is left intact and not mined. This means that mining highly stressed mining blocks could occur and that the mining practice must take cognisance of the increased risk of:

- Stress fracturing reducing the rockmass quality and thereby increasing the potential for scaling, dilution and stope collapse;
- Seismic activity during the extraction of the block, increasing the potential for damage, scaling and stope collapse.

Precautions could include:

- Reduce mining spans by leaving more or larger pillars;
- Reduce spans and/or affect rockmass behaviour by including backfill placement into stopes;
- Install cable bolting in stope backs/crowns to assist in controlling the potential fracturing.

Mining in areas of high water pressures and/or large water inflows

Water inflow has a number of potential impacts on the stability of development ends and stopes, which increase as the water flow/pressure increases:

back

- Water inflow and pressure increase pore pressure and thereby reduce the shear strength of structures/loose material;
- Reduces overall rockmass strength and therefore stability over set spans;
- Allows shear movement on structures due to reduced shear strength of such structures, increasing potential for failures on the structures;
- Increases potential for mud rushes of blasted material in stopes.

Remnant mining/pillar mining

Remnant or pillar mining in massive mines should be limited to the following situations:

- Secondary extraction of stopes between primary stope blocks;
- Post extraction of sill/rib pillars between backfilled stopes;
- Blocks left intact to protect poorly placed major access and service excavations.

Precautions within these conditions are limited to:

- The use of backfill to stabilise the stopes around the remnant and allow extraction of the remnant. Backfill can also be utilised to extract the remnant in stages by implementing a backfill mining methods or to extract the remnant in a series of smaller dimension stopes;
- Limiting access to remnant extraction areas reduce the risk of exposure of personnel to possible hazardous conditions and can include the use of remote control equipment;
- Additional support installation within accesses or even stope back support could assist stability;
- Retreat mining is a logical method to avoid remnant creation but can also be utilised during the extraction of a remnant. In combination with backfill and additional support to accesses, this is a very good solution to extracting remnant areas;
- Monitoring of conditions and stability is a useful tool to assist safe mining practices during remnant extraction. Instruments such as Borehole cameras, extensometers, laser scans etc are all usefull methods.

CONNECTION 1

Refer to
Paper 3.1, Outcome 9.1

Rehabilitation of previously mined areas

The extraction of mining blocks that have collapsed in an unplanned manner will mostly not be rehabilitated, but will require additional redevelopment to access, drill, blast and load the ore, which is addressed on a case-by-case basis.

Rehabilitation is limited to development ends/service excavations.

CONNECTION 1

Refer to
"7. ROCK BREAKING IN MASSIVE MINES" on page 353

Dealing with excessive overbreak situations

Overbreak is the result of the blasting process and the rockmass being blasted.

back

Dealing with runaway caving situations

There are three aspects that can be considered 'runaway caving' situations:

- Collapse of undercut and extraction level excavations, leading to inability to draw
- Mudrushes
- Development of air gaps followed by collapse and a resulting air-blast

Large-scale excavation collapse

Collapse of undercut and extraction level excavations may occur ahead of the undercut advance when the rockmass is too weak, and the pillars between excavations are too small, to support the level of stress concentration. Pillars crush out, excavations collapse, and may result in large sections of the overlying rockmass sitting down and collapsing into these excavations. This large-scale settlement of the rockmass may particularly occur where there are faults present, or weak contact partings (e.g. the boundary of a kimberlite pipe). An example illustrating the problem is shown in the figures below.

Experience on several mines has shown that the only way to recover from this sort of situation is to create a new series of undercut or extraction tunnels, generally on a level below the collapsed area, using improved designs, excavation spacings and support.

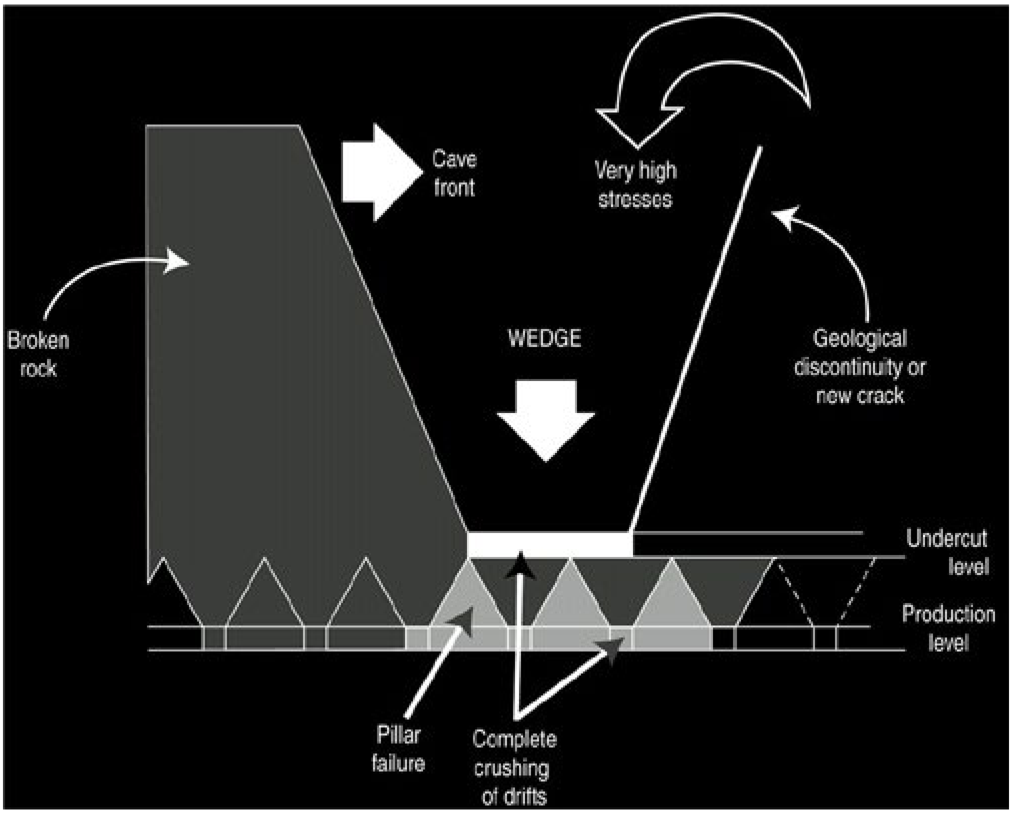

Figure 2: Collapse of large fault-bound wedge resulting in crushing of development

back

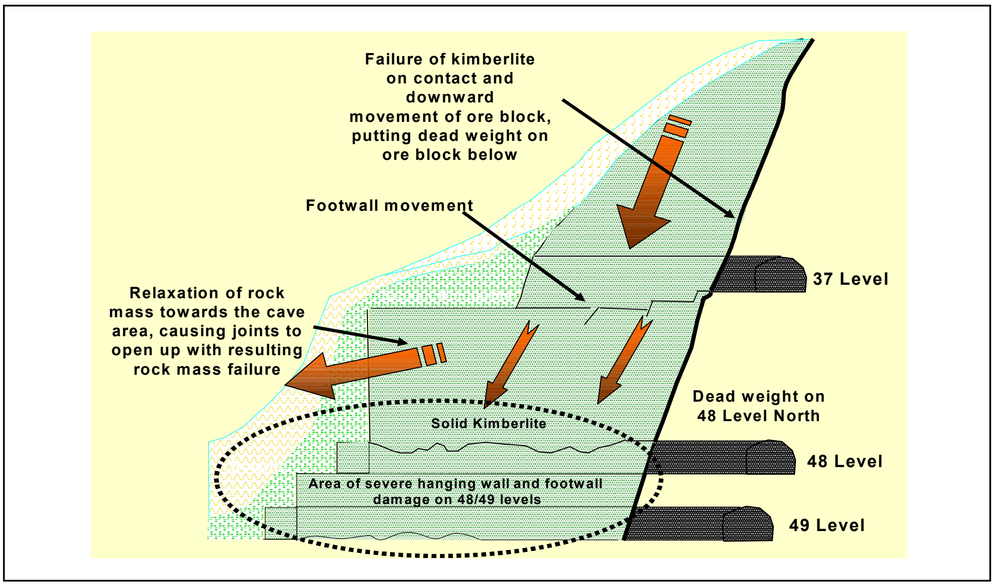

Figure 3: Example of uncontrolled collapse ahead of the cave front in a kimberlite pipe, where slip occurred on the pipe contacts

To prevent, or ameliorate the risks of major collapses, the following points apply:

- Ensure strict draw control;
- Review the angle of approach of the undercut and cave line to any major geological structures or contacts;
-
- Maintain adequate broken ore cover over the drawpoints;
- Ensure that any opening identified as having a significant risk of exposure to inrush or air blasts is safeguarded appropriately;
- Use an undercut strategy that will limit extraction and undercut stresses; and
- Protect extraction level excavations with suitable support and reinforcement

Mud rushes
Mud rushes have resulted in the inundation of several caving operations. Mud rushes are the sudden inflow, usually via drawpoints, of saturated fines from the cave. Wet muck flows are similar phenomena in which there is a sudden collapse and rapid run-out of wet particulate material, including coarser fragments.

Occasionally, mud rushes are the result of external sources (tailings dams, backfill, pit slopes); however, the mud is mostly produced by comminution of weak material to produce fines in the cave – most likely when the rock in the cave contains shale, or other clay-forming country rocks.

The mud rush hazard is illustrated in the figure below. The conditions required for a mud rush include:

- Mud-forming minerals present in the cave.
- Water entry into the cave (this could include surface run-off, storm water, ground water inflow, unsealed boreholes, leakage from underground dams or pipes, hydraulic fill placed elsewhere in the mine, water collected in old abandoned workings, etc.)

back

- A trigger mechanism (e.g. draw, blasting, seismicity)
- Discharge points into the mine workings (draw points, old tunnels that intersect the cave, etc.)

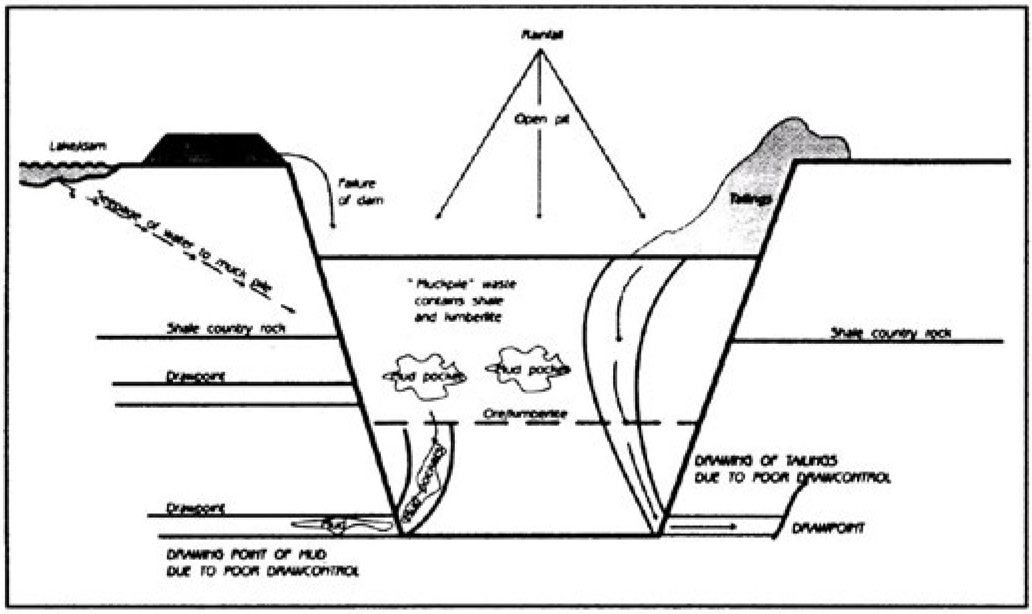

Figure 4: Mud rush hazards in a block cave operation (after Butcher, 2000)

To prevent or ameliorate the effect of water entry into the cave and any consequential mud rushes, the following points should be noted (see figure below):

- Tailings dams and surface water storages should be positioned and designed such that water cannot run into areas of surface subsidence, caving or open pits.
- If the cave is below an open pit, the pit slopes should be designed to remain stable and pit drainage methods put in place to limit water entry.
- Systems put in place for the handling of water from old abandoned mine areas, and water from areas of the mine where backfill is placed.
- Systems in place for groundwater control outside the mine (boreholes, drainage drifts, etc.) to reduce water inflow into the cave.
- Surface control of streams and storm water using appropriate channelling around the mine area.
- Ensure proper draw control (uneven draw enhances the mud rush potential).

Methods of water control are illustrated in the figure below.

back

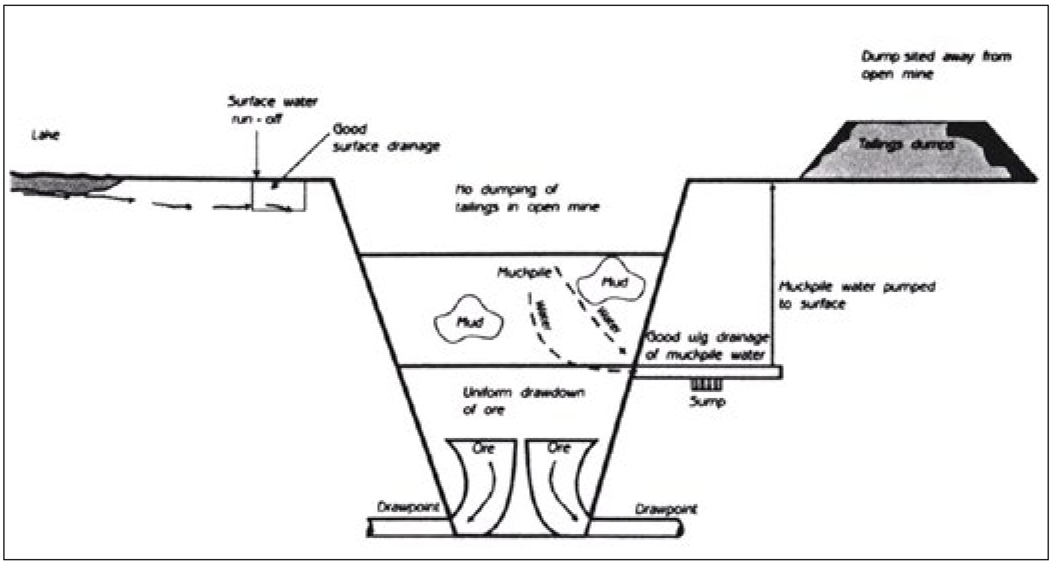

Figure 5: Factors to consider to reduce mud rush hazards

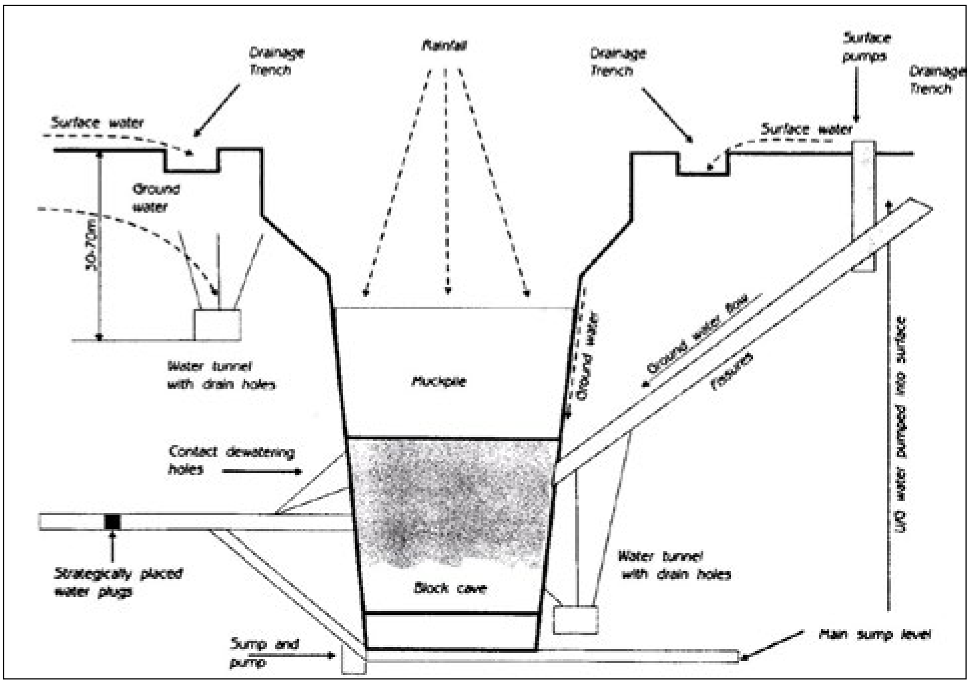

Figure 6: Ground water control methods for the prevention of mud rushes and general water entry into a block cave

REFERENCE

- Inclined caving as massive mining method Munro
- Block caving overview Petra
- Mud rushes in block caves Holder
- OTH003 accidents in other mines
- Mud rush risk SRK

Air blasts

An air blast may occur when an air gap develops between the cave back and caved material below. In this situation, a large collapse from the cave back may result in the air being forced rapidly from the air gap into any tunnels that may daylight into the void. Very often, these are old abandoned levels, but may connect to ramps or other active areas.

back

Air blasts into such areas can result in considerable damage, including fatalities.

Preventative measures include ensuring that all old levels that intersect the cave are sealed off with appropriate walls or doors.

On active levels where drawing is occurring, it is essential to have adequate cover of broken rock over drawpoints at all times.

REFERENCE

- Butcher R, Joughin W and Stacey T R, 2000. Methods of combating mudrushes in diamond and base metal mines, SIMRAC booklet.
- Flores G E, 1993. A new mining sequence to minimize stability problems in block caving mines. MSc Thesis, Colorado School of Mines.
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- Brown E T, 2003, Block caving geomechanics. JKMRC Monograph Series in Mining and Mineral Processing 3. Published by Julius Kruttschnitt Mineral Research Centre, The University of Queensland, Australia. Available at: http://www.jkmrc.uq.edu.au/Publications/BlockCavingGeomechanicsSecondEdition.aspx